BESRA

BAU PROJECT

STAGE 1

FEASIBILITY STUDY

December 2013

Signatures

Signed: 15th December 2013

Graeme W. Fulton, FAusIMM

General Manager - Malaysia,

North Borneo Gold Sdn Bhd/Besra Gold Inc.

Effective Date of this Report: 15th December 2013

Certificates of Qualified Persons

To Accompany the Report entitled

"Bau Project – Feasibility Study,

Bau, Sarawak, East Malaysia"

Dated 15th December, 2013

GRAEME WHITELAW FULTON

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CERTIFICATE of AUTHOR

I, Graeme Whitelaw Fulton, B.Sc. (Hons), Mining and Petroleum Engineering, FAusIMM, do hereby certify that:

1. I am a qualified mining engineer working and currently I am the General Manager on the Bau Project for North Borneo Gold Sdn Bhd/Besra Gold Inc.

2. This certificate relates to the technical report entitled, "***Bau Project – Feasibility Study, Bau, Sarawak, East Malaysia***" dated 15th December 2013.

3. I graduated with the degree of Bachelor of Science with Honours in Mining and Petroleum Engineering, in 1986 from the University of Strathclyde, Glasgow, Scotland.

4. I am a Fellow of the Australasian Institute of Mining and Metallurgy and have been since 2000. My AusIMM membership number is 208430.

5. I have practiced my profession continuously for a total of 28 years since my graduation from the University of Strathclyde, and have 29.5 years experience including pre-university work.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be an "qualified person" for the purposes of NI 43-101.

7. I am responsible for the overall preparation of the technical report.

8. I have had prior involvement with the property that is the subject of the Technical Report having been based on site or previously visited the site as a consultant prior to employment.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

10. I am not independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101, as I am an employee of North Borneo Gold Sdn Bhd/Besra Gold Inc.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 15th Day of December, 2013

G Fulton

Graeme W Fulton, B.Sc. (Hons), FAusIMM

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1. Executive Summary

1.1. Introduction

This executive summary lists some of the highlights and important aspects of the Feasibility Study and is intended as a brief summary of the project and study.

1.2. Property Description & Location

The Bau Goldfield projects are located on the Island of Borneo in Sarawak, Federation of Malaysia. The project area is centered on the township of Bau some 40 km WSW of the state capital of Kuching (population ~640,000); see *Figure 1-1 - Property Location Plan* below.

Besra's Bau Project is a brown-field project comprising Mining and Exploration tenements that cover more than 1,340km^2 of the most highly-prospective ground within the historic Bau Goldfield, spread over 3 regions in Sarawak. The main focus of Besra's activities are Blocks A and B which relate to the Bau district. The other two regions, known as Block C and Gunong Rawan lie east of Bau nearer to the Sarawak/Kalimantan border and these are earlier stage exploration projects.

Figure 1-1 - Property Location Plan

1.3. Ownership & Tenure

1.3.1. Ownership

Besra Gold Inc. (formally Olympus Pacific Minerals Inc. (“Olympus”)) ("Besra") is a Canadian incorporated public company listed on the Toronto Stock Exchange, on the Australian Securities Exchange and trades on the OTCQX Bulletin Board in the United States. Its head office is located in Toronto.

On 17th December 2009, shareholders of Zedex Minerals Limited (“Zedex”) amalgamated with Olympus Pacific Minerals. Following the amalgamation Olympus commenced trading on the ASX on 5th February 2010.

One of the assets acquired by Olympus through the amalgamation was Zedex’s interest in the Bau Gold Field in Sarawak, Malaysia.

In November 2012 Olympus Pacific Minerals Inc. was renamed Besra Gold Inc.

The Bau Project JV is managed by Besra through its majority owned subsidiary, North Borneo Gold Sdn Bhd, (“NBG”) a Malaysian incorporated company. The other shareholders are a Malaysian Mining Group, Gladioli Enterprises Sdn Bhd (“Gladioli”) and Golden Celesta Sdn Bhd.

The Bau Project is currently 85.61% owned and controlled by Besra Inc. with the minority of the project owned by a local partner. Besra has an agreement to acquire a further 7.94% from the partner for payments from December 2013 to September 2015 totalling $7.85m.

1.3.2. Tenure

All mineral resources in Malaysia are state owned. Exploration and mining rights are issued subject to the recently gazetted Minerals Ordinance 2004 which has an effective commencement date of 1 July, 2010, and Mining Rules (1995). The prospecting, exploration and mining tenure is in the form of a GPL (general prospecting licence), EPL (exclusive prospecting licence), MC (mining certificate) and ML (mining lease). These tenure types and associated information are outlined in the body of the report.

Besra Inc. through its JV Company and associated partner hold a number of granted licenses and/or application renewals on existing licences. These are listed in the body of this report. The licences for the Bau area (Block A & B) are shown in *Figure 1-2: Tenement Location Map for Bau Showing Mining Leases, Mining Certificates and EPL & GPL Applications Subject to Joint Venture* below.

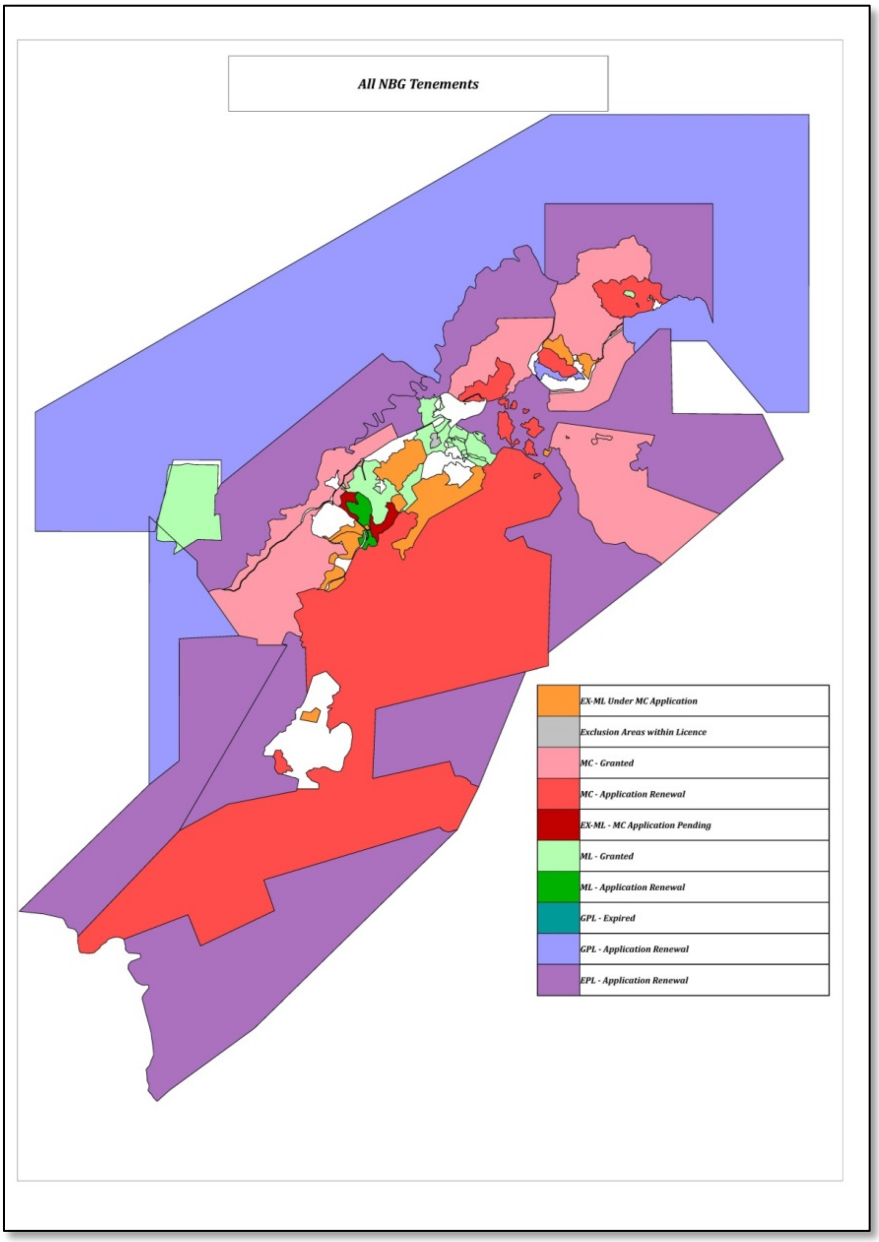


Figure 1-2: Tenement Location Map for Bau Showing Mining Leases, Mining Certificates and EPL & GPL Applications Subject to Joint Venture

1.4. Resources & Infrastructure

The Kuching District, (including Bau) has a population of approx. 640,000 people. At Bau the main population groupings are Bidayuh, from the Dyak ethnic group, and Chinese who are mainly descendants of early miners who came to the area in the mid to late 19th Century to exploit the gold and antimony deposits at Bau. Sarawak has a per capita GDP of US$1,400. Mining represents about 20% of Sarawak's GDP.

The main industries in the Bau district are limestone quarrying, fish farming, rice farming, palm oil and rubber production, and now mineral exploration.

The Bau Project generally has good infrastructural aspects both within Bau Township and in Kuching. The main infrastructural features are:

- Regular and reliable international air services to Kuching from Kuala Lumpur, Singapore, Hong Kong and Indonesia. Airport is only 35-40 minute drive from the project area;
- Two ports with good dock and storage facilities;
- Two main sealed trunk roads from Kuching for delivery of supplies, heavy plant and equipment to the plant site;
- Excellent labour and engineering support services;
- Easy Accessibility – project extremities are less than a 20 minute drive from the exploration base, and all important mines and gold prospects are linked by road;
- Area is serviced with power and water;
- The official language in Sarawak is Bahasa Malaysia, but most local communities speak English as a second language and have their own local dialects;
- Well educated workforce;
- An active quarrying industry focused mainly on limestone and marble for roading aggregates and agricultural purposes;
- Ready supply of earthmoving equipment that supports the quarrying industry;

A local labour source with mining experience gained from the quarrying industry and past gold mining activity.

1.5. History

The Bau Goldfield has been intermittently mined since the mid 19th Century. Historic production is estimated at > 3M oz gold. The most recent was the Tai Parit mine, which closed in 1996 after producing 1.2M oz gold from a single open-pit averaging 7 g/t Au. It is important to note that this pit was closed at the time due to fall in gold price to below $300/oz making mining at the time uneconomic, the pit was not mined out at the point of closure.

1.6. Geology & Mineralisation

1.6.1. Regional Setting

The Bau Goldfield lies within the Borneo metalliferous belt, which contains several other important gold mining areas, including: Kelian, Mamut (gold-copper) and Mt Muro. This belt and the associated gold mining areas are shown in *Figure 1-3 - Borneo Metalliferous Belt Showing Bau & Other Important Mining Areas* below.

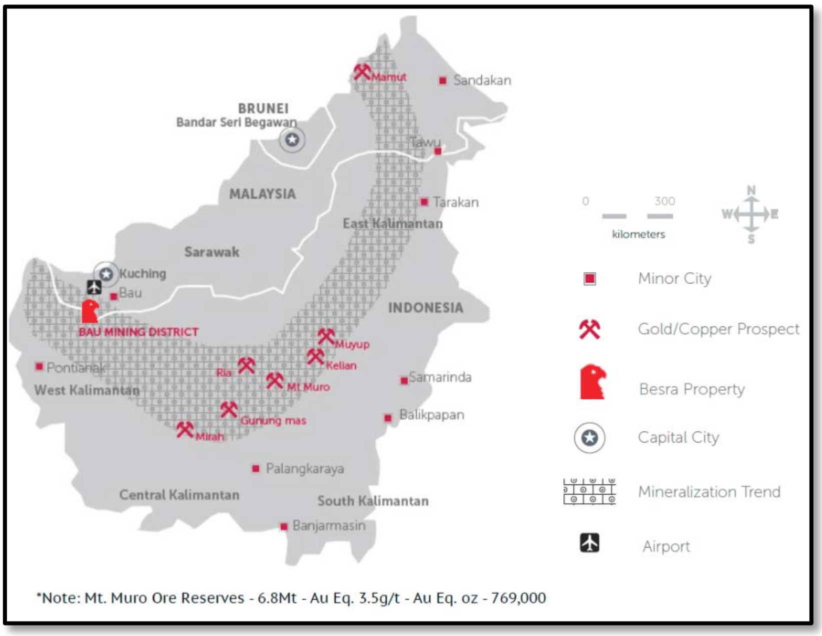


Figure 1-3 - Borneo Metalliferous Belt Showing Bau & Other Important Mining Areas

The geology and mineralization of the Bau Goldfield have been compared with that of the Carlin District of Nevada, USA (cumulative production > 60M oz) and there are a number of similarities. These are listed in *Figure 1-4 – Comparative Diagram & List between North Carlin Trend & the Bau Central Trend* below.

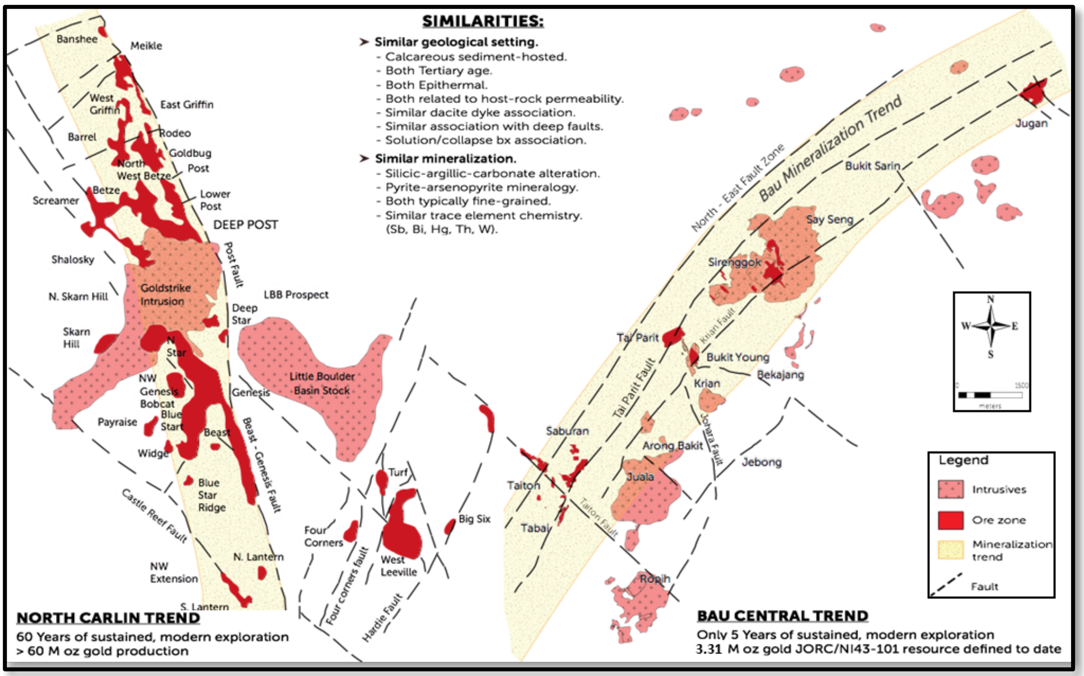


Figure 1-4 – Comparative Diagram & List between North Carlin Trend & the Bau Central Trend

1.6.2. Geological Setting

The exposed rocks in the Bau district are dominated by a sequence of late Jurassic to early Cretaceous aged marine sediments. These comprise a lower limestone formation, the Bau

Limestone, estimated to be 500 metre thick that is unconformably overlain by a 1,500 metre thick flysch sequence, known as the Pedawan Formation. The Pedawan Formation is dominated by shale but more arenaceous and conglomeratic units are reasonably widespread through the sequence.

The oldest rocks known in the Bau Goldfield are the Triassic-aged Serian andesitic volcanics. These do not crop out but have been intersected in drill holes at Bau, beneath the Bau limestone. An intrusive known as the Jagoi Granodiorite is thought to be co-eval with the Serian Volcanics and it crops out 15 km SW of Bau on the Indonesian border.

Figure 1-5 - Generalised Stratigraphy of the Bau District (after Schuh, 1993) below diagrammatically depicts the stratigraphic relationships for rocks of the Bau District.

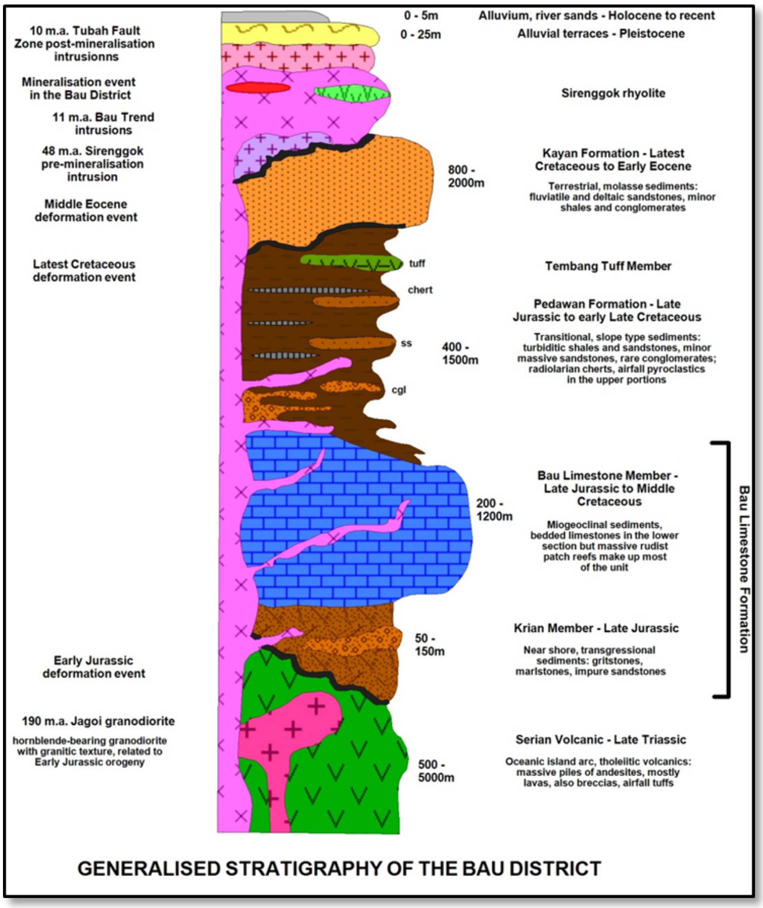


Figure 1-5 - Generalised Stratigraphy of the Bau District (after Schuh, 1993)

The Bau Limestone has a lowermost ~100 metre thick arenaceous unit, (the Krian Member), which also contains basal conglomerate beds. The Krian sandstones rest unconformably on the Serian Volcanics. The principle rock types and structures of the Bau Goldfield are shown in *Figure 1-6 - Generalised Geology of the Bau Goldfield* below.

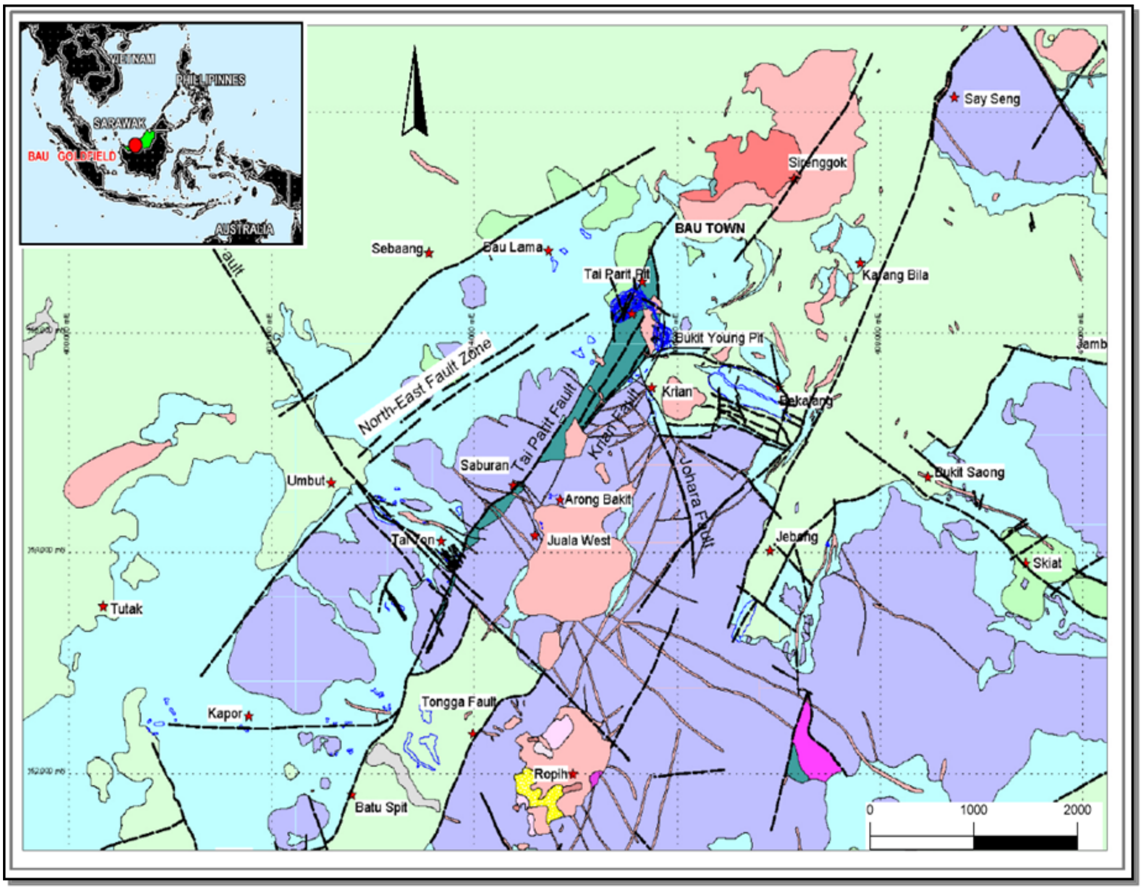


Figure 1-6 - Generalised Geology of the Bau Goldfield

A striking feature of the Bau District is a series of uplifted horst blocks of Bau Limestone juxtaposing the generally stratigraphically higher Pedawan formation. Throws on the NNE and SE trending controlling graben faults are in the order of 300 metres. Surrounding the horsts of limestone is a peneplane of Bau limestone with typical karst features and the overlying Pedawan formation.

The Pedawan Formation and Bau Limestone represent fore-arc shelf and slope deposits developed to the north of a Cretaceous magmatic arc, remnants of the arc are preserved as a granite belt in the Schwaner Ranges in Central Kalimantan.

1.6.3. Deposit Type/Mineralisation

There are four distinct mineralisation or deposit styles in the Bau goldfield. The deposit/mineralisation types are:

- Disseminated sediment hosted
- Silica replacement and breccias
- Magno-calcite quartz veining
- Porphyry-skarns

Each of the 34 deposits or prospects contains one or more of these styles of mineralisation covering an extent of 15km NE-SW by 7-8km NW-SE. The goldfield extents are shown in *Figure*

1-7 - Bau Goldfield Extents with Sectors & Deposits/Prospects below, along with the sectors (yellow text) and the deposits/prospects (red text).

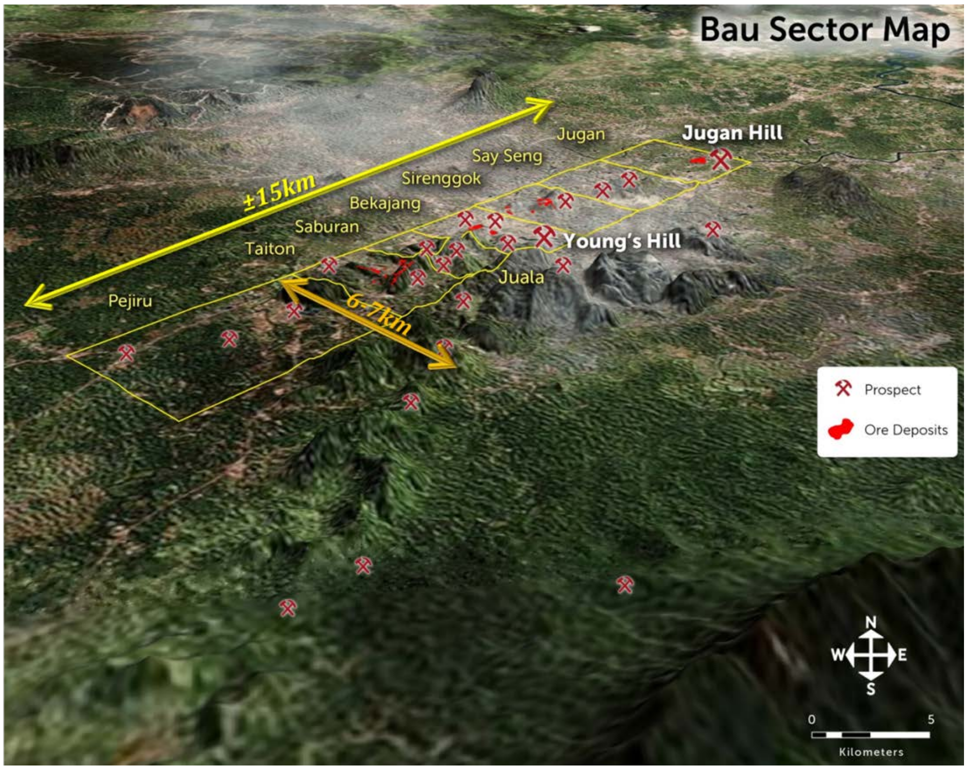


Figure 1-7 - Bau Goldfield Extents with Sectors & Deposits/Prospects

1.7. Mineral Resources

The mineral resources listed below are a combination of the 2010 resources as published in August 2010 and included in the 43-101 report ("*Technical Report on Bau Project in Bau, Sarawak, East Malaysia*") at that time, a small resource update to some deposits in 2011 (published February 2012) and a small resource update in November 2012.

A summary of the resource totals by category is shown in *Table 1-1 - Resource Update Summary by Category (November 2012)* below.

Resource Category	Tonnes (t)	Grade (g/t)
Measured	3,405,600	1.52
Indicated	17,879,700	1.67
Measured + Indicated	21,285,300	1.64
Inferred	51,329,000	1.32

Table 1-1 - Resource Update Summary by Category (November 2012)

For the 2010 resource definition Terra Mining Consultants/Stevens & Associates have classified the defined mineralization according to the definitions of National Instrument 43-101, CIMM

Definitions and the Australasian Institute of Mining & Metallurgy's JORC Code 2004. Similarly, the 2011 and 2012 resource updates have been classified in the same manner by Besra/NBG.

1.8. Exploration

1.8.1. Exploration Concept

Jugan has been well defined through drilling to the current depths, the depth extent is less well defined and there remains open-ended potential to increase the resource beyond the current depth. Geophysical surveys and soil sampling campaigns have identified some nearby anomalies which will require further work and exploration/resource drilling.

The nearby (±1.5km) small resource (±45,000 ozs) at Bukit Sarin (Jugan West) has a significant anomaly on and surrounding the deposit. Mapping and scout drilling is underway and gridded soil sampling has been conducted, but the extents of the deposit need to be tested with full exploration or resource drilling.

The BYG-Krian deposit requires further drilling to upgrade the resource beneath the current Indicated Resource in order to upgrade this Inferred zone. Additionally, there are further Inferred extensions along strike that need to be drilled also.

The remainder of the Bau goldfield contains 34 prospects or known deposits which are at the Inferred level or have suitable geological potential requiring an extensive amount of follow up work and exploration or resource drilling.

The goldfield also needs to be tested at depth below these deposits/prospects to fully understand the significant depth potential in line with the Carlin similarity model and concept. Below is a sequence of slides showing the strike extent of current resources and the depth (and strike) potential to be tested, over and above that shown by Jugan itself.

Figure 1-8 - Bau Strike Extent Showing Sectors, Deposits & Depth Potential (First 5km of 15km)

Figure 1-9 - Bau Strike Extent Showing Sectors, Deposits & Depth Potential (Second 5km of 15km)

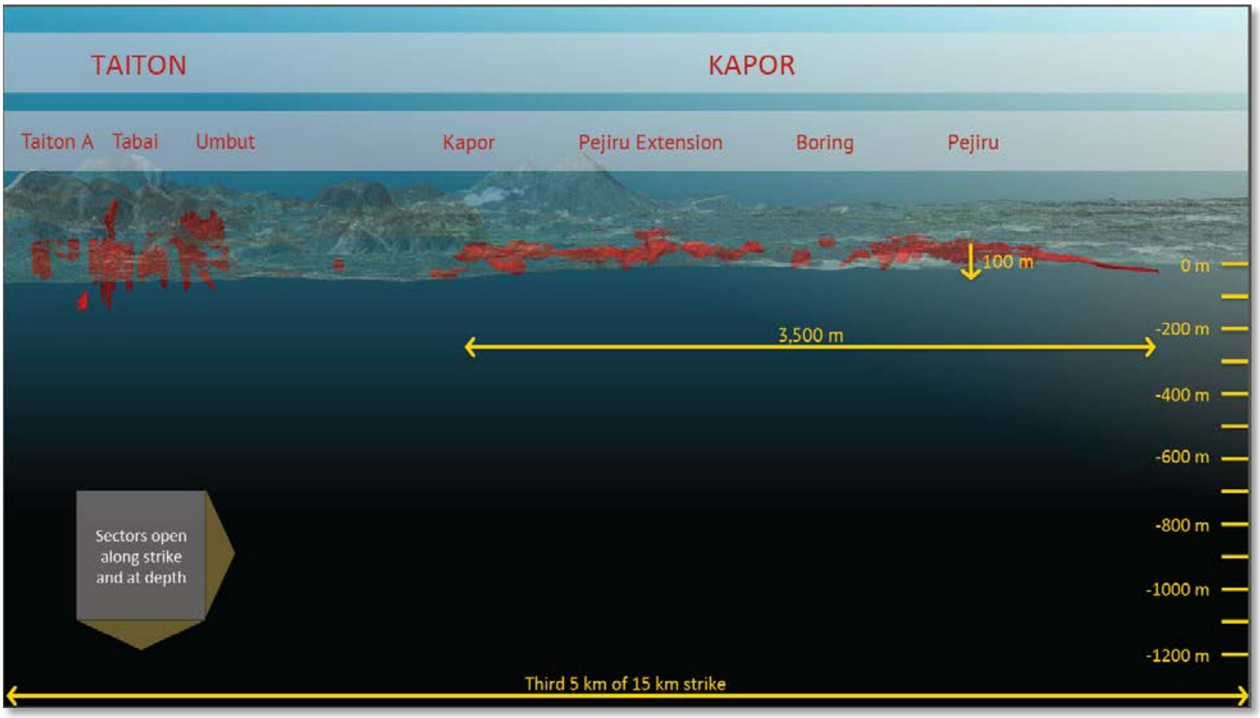


Figure 1-10 - Bau Strike Extent Showing Sectors, Deposits & Depth Potential (Last 5km of 15km)

1.8.2. Status of Exploration

The feasibility study level of resource drilling has been concluded and there is currently no further exploration being conducted at either site. Follow-up ground truthing, mapping and grid auger soil sampling is ongoing along with some scout drilling in the surrounding anomalies identified. Further resource drilling has been planned and is awaiting funding.

1.9. Mineral/Ore Reserves

A summary of reserve totals, for the contract mining base case, by Reserve Category is shown in *Table 1-2 - Reserve Summary by Category (November 2013)* and these reserves by area/sector and deposit are also shown in *Table 1-3 - Reserve Summary by Sector/Area & Deposit (November 2013)* below. Note that Mineral/Ore Reserves above are contained within Mineral Resources.

Reserve Category	Tonnes (t)	Grade (g/t)
Proven	3,418,650	1.47
Probable	7,243,920	1.81
Proven + Probable	10,662,570	1.70

Table 1-2 - Reserve Summary by Category (November 2013)

Sector	Reserve Category	Tonnes (t)	Grade (g/t)
Jugan	Proven	3,418,650	1.47
	Probable	6,368,190	1.61
	Proven + Probable	9,786,840	1.56
Bukit Young	Proven	0	0
	Probable	875,730	3.31
	Proven + Probable	875,730	3.31

Table 1-3 - Reserve Summary by Sector/Area & Deposit (November 2013)

For the reserve definition work found in this report, Besra/NBG have classified the ore/mineral reserves according to the definitions in the National Instrument 43-101, CIMM Definitions and the Australasian Institute of Mining & Metallurgy's JORC Code 2012.

The economic pit limit evaluations, open pit development sequence plans, and reserve estimates are based on a gold price of $1,500/oz. This is the gold price used in the optimisation to define the ultimate pit, with the optimal pit used being within this limit. Differing gold prices have been used in the cost models and sensitivity optimisations were done at a range from $1,200 to $2,000 per ounce gold prices.

Process recoveries used are an effective recovery of 77 % for the base case concentrate option. The concentrate recovery option is based on a flotation recovery, recovery for contract processing facility and their percentage of metal content (current offers under negotiation but conservative value applied). For the optimisations using the other metallurgical processes, the following recoveries were used - 85 % (POX), 80 % (BIOX & ALBION).

Base mining cost used for 8,000tpd and concentrate base case was $1.74/t and this relates to overburden removal, with mine cost adjustment factors (MCAF) of 1.52 and 1.34 for ore mining and waste mining respectively. Processing cost for the base case concentrate option was $7.19/t with the G&A's and other selling costs as $0.16/g. Mining, processing and other costs for other processes are detailed in *Chapter 21* of this report.

For the Jugan pit, comparing the designed total pit reserves for the owner operator option (9.94 Mt ore at 1.56 g/t Au) and that generated via the optimisation software (10.16 Mt ore at 1.55 g/t Au), for the same scenario, the reserves (Proven + Probable) are comparable and show 2.2 % difference in tonnage and 0.5 % difference in grade.

This difference is negligible in relation to the orebody modelling and design resolution. Therefore, the optimised schedules can be accepted as reasonable level for reserve generation for the open pits.

The comparison for the other base case (flotation concentrate, 8,000 tpd and contract-mining) is 9.79 Mt at 1.56 g/t Au for design and 9.92 Mt at 1.56 g/t Au for optimisation, which is a difference of 1.4 % in tonnage and 0.4 % difference in grade. The contract mining basecase reserves are detailed in *Section 15*.

Based on the optimisation runs and the applied parameters a cut-off grade of 0.39 to 0.44 g/t Au is realised for the Jugan reserves, with a strip ratio of 1.60 and 1.47 for owner-operator and contract-mining options, respectively.

For open pit inventory, the resource block model estimation methodology incorporates dilution and provides a reasonable estimate of mined tonnage and grades. However, an additional 5 % dilution is added with a 95 % mining recovery have been included as an additional factor in the pit optimisation process and in the reserves.

Based on the optimisation runs and the applied parameters a cut-off grade of 0.58 to 0.65 g/t Au is realised for the BYG-Krian reserves, with a strip ratio of 4.41 and 3.94 for owner-operator and contract-mining options, respectively.

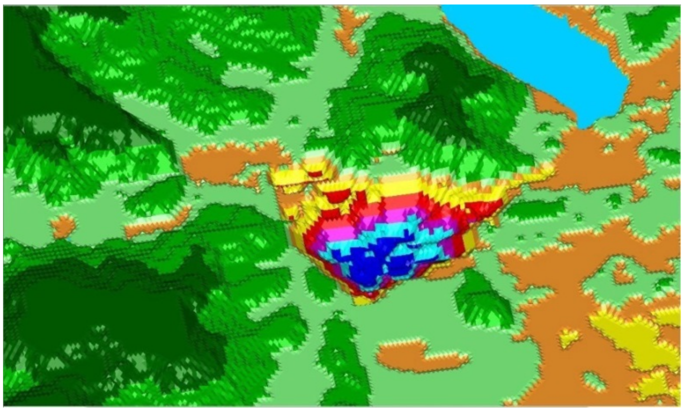
Although the BYG-Krian pit is small when considering the Indicated only, it has additional potential in terms of the inferred both under the pit and indicated zone but also in shallow extensions around. As the resource is Inferred, in this case it cannot be considered in the reserves, the potential for pit expansion is significant in terms of the current reserves.

This can easily be upgraded with some additional resource drilling and conversion to indicated. Listed below in *Table 1-4 - Comparison of Potential between Indicted Only & Indicated-Inferred Resources at BYG-Krian* is a comparison of the resources if Inferred was available as Indicated.

Description	Using Indicated Only		Using Indicated & Inferred	
	Tonnes	Au (g/t)	Tonnes	Au (g/t)
Ultimate Pit (Shell 65)	1,026,890	3.13	2,808,890	2.09
Optimal Pit (Shell 50)	1,007,380	3.11	2,696,450	2.11
Designed Pit	875,730	3.31	2,093,510	2.35

Table 1-4 - Comparison of Potential between Indicted Only & Indicated-Inferred Resources at BYG-Krian

Figure 1-11 - 3D Comparison of Indicated Only (top) & Indicated-Inferred (bottom) Pits below shows this impact visually.



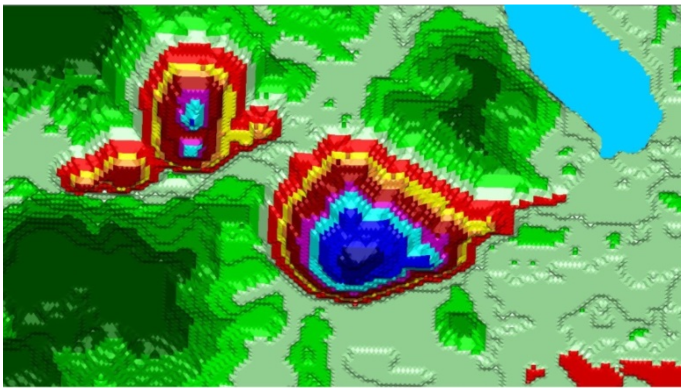


Figure 1-11 - 3D Comparison of Indicated Only (top) & Indicated-Inferred (bottom) Pits

It should be noted the above is only included for comparative purposes and should not be considered as reserves.

1.10. Mining

Due to the orebody outcropping as a hill and having significant resources at a shallow depth the initial method of extraction of ore is by open pit methods. The orebody does extend down to a depth of approximately 400m and is anticipated to carry on to further depths.

Due to the orebody nature the initial mining will be free digging, with the use of dozers and ripping where harder material encountered, with loading of haul trucks by hydraulic excavators. As the pit progresses deeper the open pit mining will be done by conventional drill and blast. The open cut design was guided by the results of NPVS Scheduler pit optimisation – for both the Jugan and BYG-Krian deposits.

A number of pit optimisations runs were undertaken for a combination of all key options. These were production (4,000 tpd to 12,000 tpd in 2,000 tpd increments), contract mining vs. owner operator mining, and by process method (POX, BIOX, Albion & Concentrate).

Gold price ranges were tested from $1,200/oz to $2,000/oz with a $1,500/oz finally selected. The difference between the $1,200/oz and $1,500/oz was negligible in terms of tonnes and grade so the $1,500/oz was suitable to use. This defined the ultimate pit with a practical and lesser optimal pit selected based on set criteria. The appropriate or latest gold price is used in the cost model and project schedule.

Costs and parameters used in the pit optimisation were defined from first principles, developed iteratively or based on known costs at Besra's other operations. Detailed costing, optimisation parameters (ultimate pit, pushbacks and schedules), and other parameters are detailed in *Section 16* of this report.

Shown below in *Figure 1-12 - Jugan Optimised Pit - Initial & Final* and *Figure 1-13 - BYG-Krian Optimised Pit - Initial & Final*, are examples (base case – 8,000 tpd) of the initial and final optimised pits for Jugan and BYG-Krian.

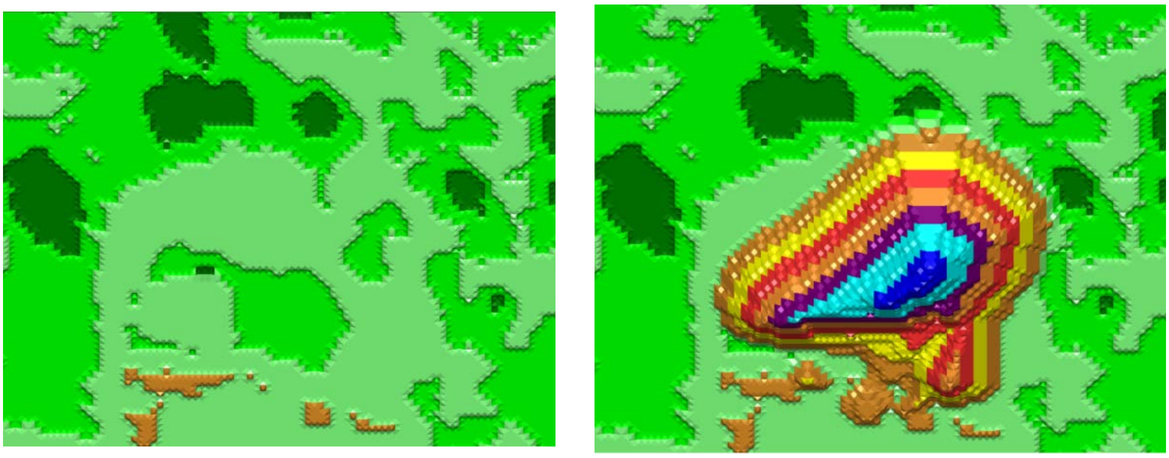


Figure 1-12 - Jugan Optimised Pit - Initial & Final

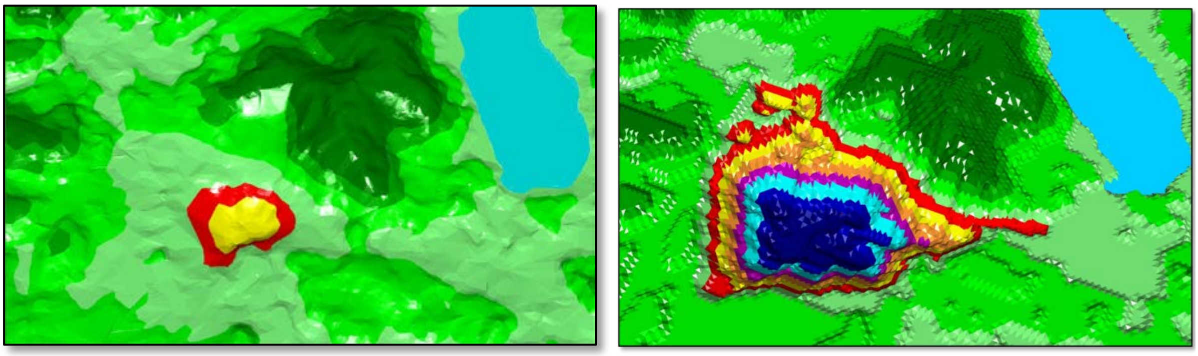


Figure 1-13 - BYG-Krian Optimised Pit - Initial & Final

Geotechnical parameters were provided by the geotechnical team. These were based on detailed geomechanical logging of resource and metallurgical drillholes throughout the full extent of the orebody and including surface mapping of the outcrops. This data was used to model geomechanical values into a 3D geomechanical model used to either define area/depth slope angles by geomechanical zones or the actual model point values used in the design process.

Detailed pit designs were done for the two base cases only – 8,000 tpd contract and owner operator mining. The reserves from these designs were comparable to the applicable pit optimisations. Therefore, pit optimisation reserves were deemed suitable for the other alternate options. The Jugan and BYG pit designs are shown in plan and 3D views below in *Figure 1-14 - Jugan Detailed Pit Design - Plan & 3D View* and *Figure 1-15 - BYG-Krian Detailed Pit Design - Plan & 3D View* respectively.

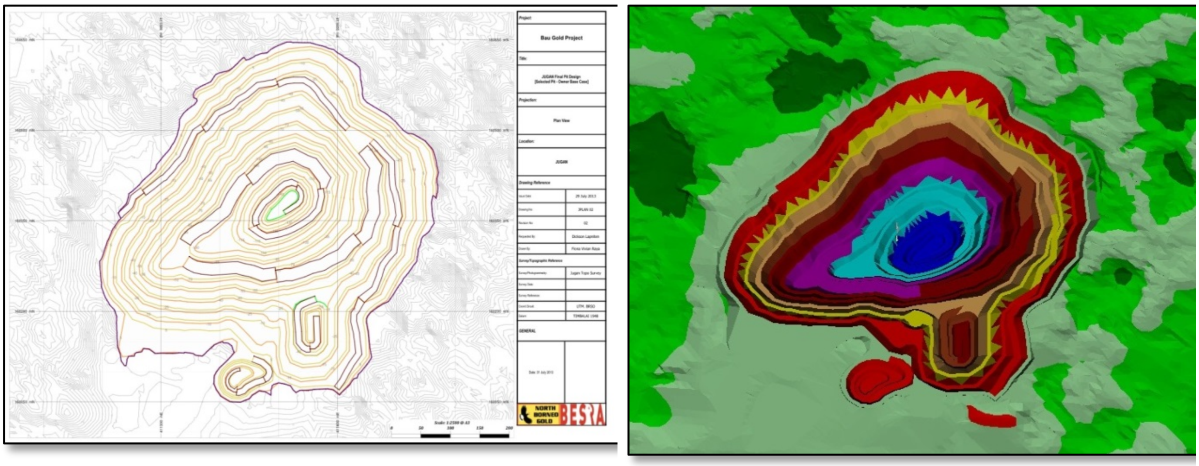


Figure 1-14 - Jugan Detailed Pit Design - Plan & 3D View

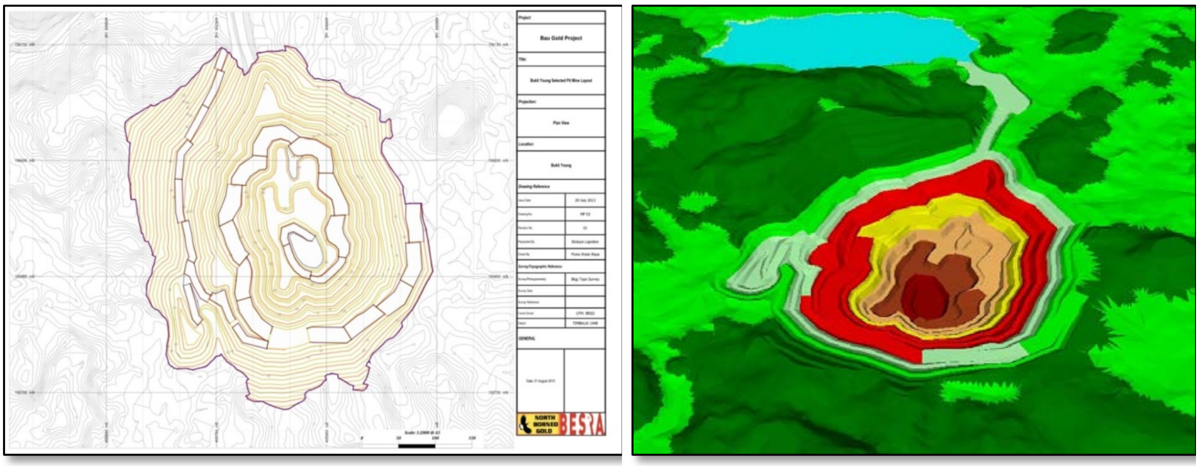


Figure 1-15 - BYG-Krian Detailed Pit Design - Plan & 3D View

The mine layouts are shown in *Section 1-12 Infrastructure & Ancilliary* below. These incorporate the open pit, TSF, waste landform, offices and plant and the roading and water infrastructure.

Detailed mining equipment lists (owner-operator only), and mining and associated labour are detailed in the body of the report.

1.11. Metallurgy & Ore Processing

1.11.1. Metallurgy

Both the historical and recent Besra metallurgical testwork on the Jugan ore deposit have demonstrated that the majority of the gold is associated with arsenopyrite and pyrite with the remaining gold present in silicious gangue material. The recovery of gold from the ore requires a gold pre-concentration step in a treatment flowsheet comprising crushing, grinding, desliming and flotation to produce a high gold grade concentrate. For the base case and preferred option the flotation concentrate will be filtered to about 10 % moisture, packaged and sent to an outside smelting or gold refining operation. The sale of a flotation concentrate offers the lowest capital expenditure and the lowest operating expenditure as well as the the highest return on investement compared with treating concentrate on site.

Additional options which have been considered in the testwork include further treatment of the flotation concentrate in one of three oxidation processes (Albion, POX or BIOX). The oxidized concentrate is then treated by conventional carbon-in-pulp cyanide leaching (CIL), elution, gold electrowinning and gold doré melting. The CIL tailings are detoxified by the copper catalyzed SO_2/Air process and the eluted carbon regenerated for recycle to the CIL.

POX delivers the highest gold extraction (98%) at the lowest cyanide consumption rate (6kg/t). Gold extractions for both the BIOX and Albion are substantially lower at around 90 % with higher cyanide consumptions up to 15 kg/t. The unit cost of cyanide has a large impact on the operating cost. In addition to higher OPEX, the Albion process has the highest risk with only one commercial plant in operation at the Las Lagunas project in the Dominican Republic. The advantages of the POX are in part offset by a higher CAPEX than for BIOX and the Albion.

1.11.2. Flotation Concentrate (Base Case Option)

- The Jugan ore exhibits a very low abrasion index and moderate bond ball mill work index (12.3 kWh/t).
- The assay data for the Jugan ore zones indicate that there is very little difference with respect to mineral distributions in the ore zones apart from minor variations in arsenic and gold contents. The increases in arsenic coincide with increases in gold showing an evident correlation. Based on sulphide sulphur and arsenic assays the ore is estimated to contain between 2 and 2.5 wt % arsenopyrite and 4.5 to 5 wt % pyrite with a combined arsenopyrite-pyrite in the feed in the range 6.5 to 7.5 wt %.
- The mineral assemblage is identical for all the Jugan ore zones tested across the deposit. The bulk of the Jugan ore feeds comprise non-sulphide gangue which is dominated by very fine grained Illite (mica) and silica. This results in production of excessive slimes after fine grinding.
- Gold deportment testing showed that very little gold is leached in whole ore cyanidation (0.6 to 2 %). About 70 % of the gold is associated with the arsenopyrite, 25 % with the pyrite and 5 % with silica.
- In excess of 95 % of the gold can be recovered in rougher – scavenging flotation. Due to varying slime entrainment the mass pull varied between 17 and 33 wt%. To mitigate the effect of feed slimes the flotation feed will be first deslimed by cyclone or a continuous gravity concentration. Flotation feed desliming test work is still in progress.
- Bulk rougher-scavenger followed by cleaner flotation without prior desliming has shown that 90 % of the gold can be recovered in a mass pull of 10 wt %. This corresponds to a gold upgrading ratio of 9:1 with respect to the feed grade. Mineralogical composition of a cleaner concentrate showed that the arsenopyrite and pyrite account for 67.4 wt% of the cleaner flotation concentrate. The remaining was comprised of 17 wt% mica (Muscovite), 6 wt% quartz, 6 wt% K-Feldspar, 3 wt% dolomite and minor rutile, sphalerite and siderite.

- The results indicate that inclusive of a desliming step, the flotation gold upgrade factor in the rougher circuit will be approximately 9 and in the cleaner stage greater than 2, giving an anticipated concentrate grade of +30g/tAu.

1.11.3. Ore Processing

In relation to the basecase option – namely 8,000 tpd (2,920,000 tpa) and flotation concentrate process option – the process plant will likely have the following configuration:

- Crushing
- Grinding/Primary Cyclone
- Cyclone or Continuous Knelson Desliming
- Rougher/Scavenger Flotation
- Regrinding/Secondary Cyclone
- Cleaner Flotation
- Concentrate Filter feed Thickener
- Filter Press
- Reagent mixing, storage and distribution
- Services.
- Control room & Facilities
- Support Facilities.

Concentrate will be dried and packed into bulk bags and transported by road to Kuching port facilities for export to Asian smelters.

The process flow sheet is shown below in *Figure 1-16 - 8,000 tpd Flotation Concentrate Process Flow Sheet*.

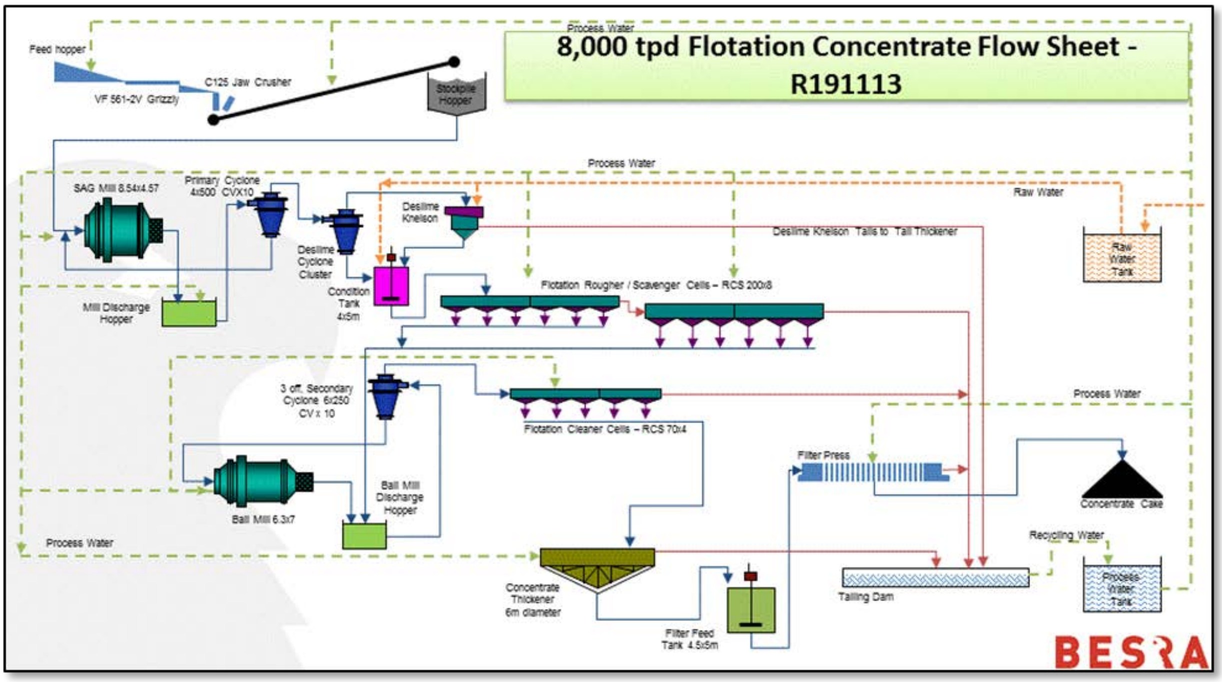


Figure 1-16 - 8,000 tpd Flotation Concentrate Process Flow Sheet

1.12. Infrastructure & Ancilliary

1.12.1. Infrastructure - Jugan

Two mine site layouts have been developed for Jugan. One is the base layout with an alternate should the condemnation drilling identify an ore deposit to the SW of Jugan where a geochemical and geophysical anomaly has been identified. The mine layout plan is shown in *Figure 1-17 - Jugan - Infrastructure Layout Plan* with the alternate below that in *Figure 1-18 - Jugan – Alternate Infrastructure Layout Plan*. These are also shown in 3D views in *Figure 1-19 - Jugan - Infrastructure Layout (3D)* and *Figure 1-20 - Jugan – Alternate Infrastructure Layout (3D)*.

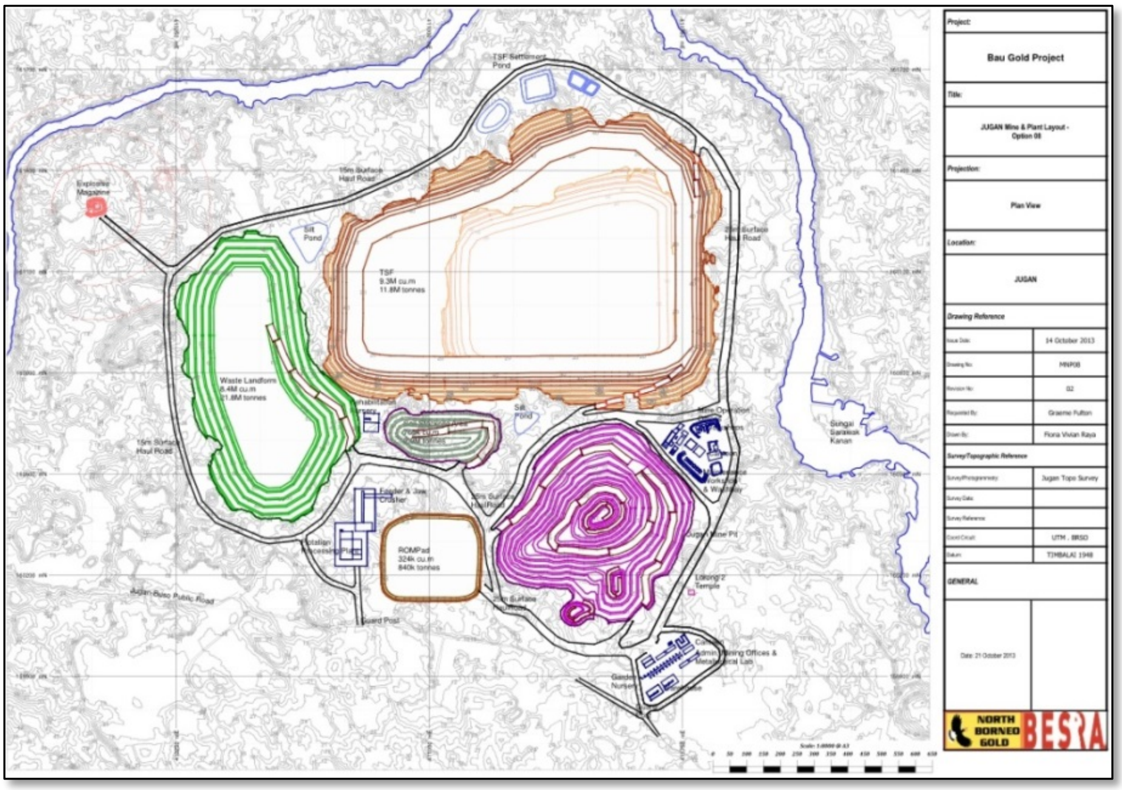


Figure 1-17 - Jugan - Infrastructure Layout Plan

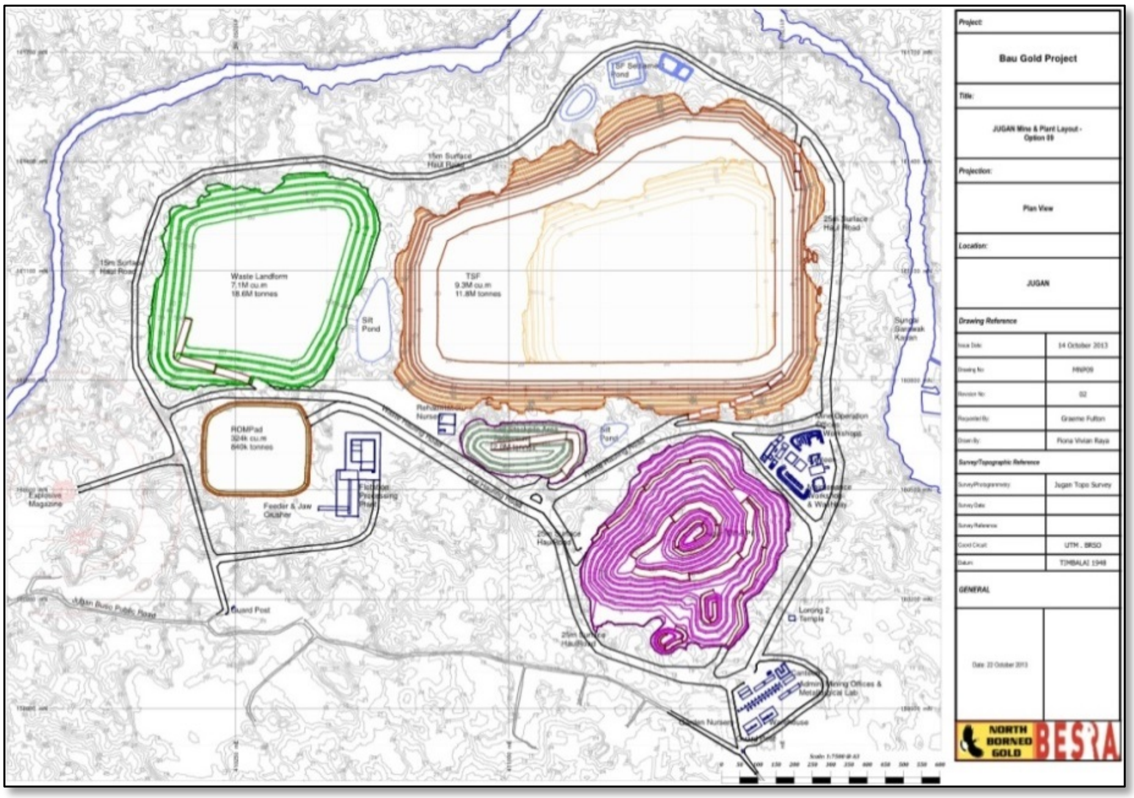


Figure 1-18 - Jugan – Alternate Infrastructure Layout Plan

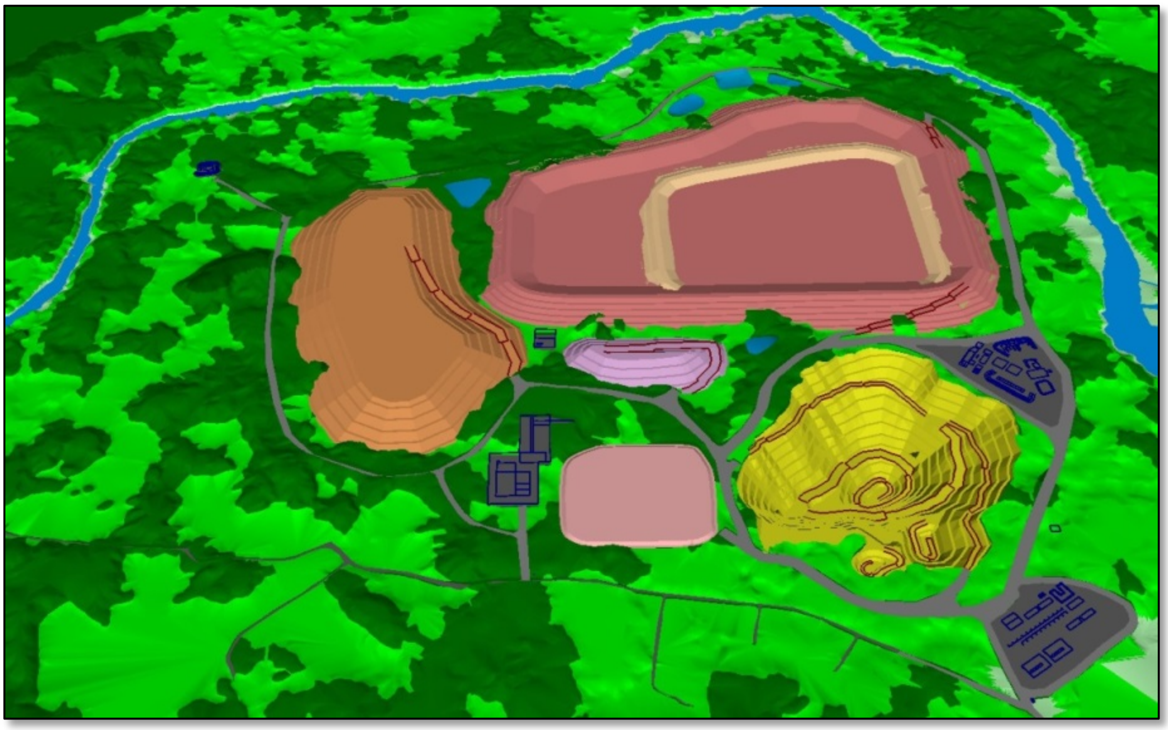


Figure 1-19 - Jugan - Infrastructure Layout (3D)

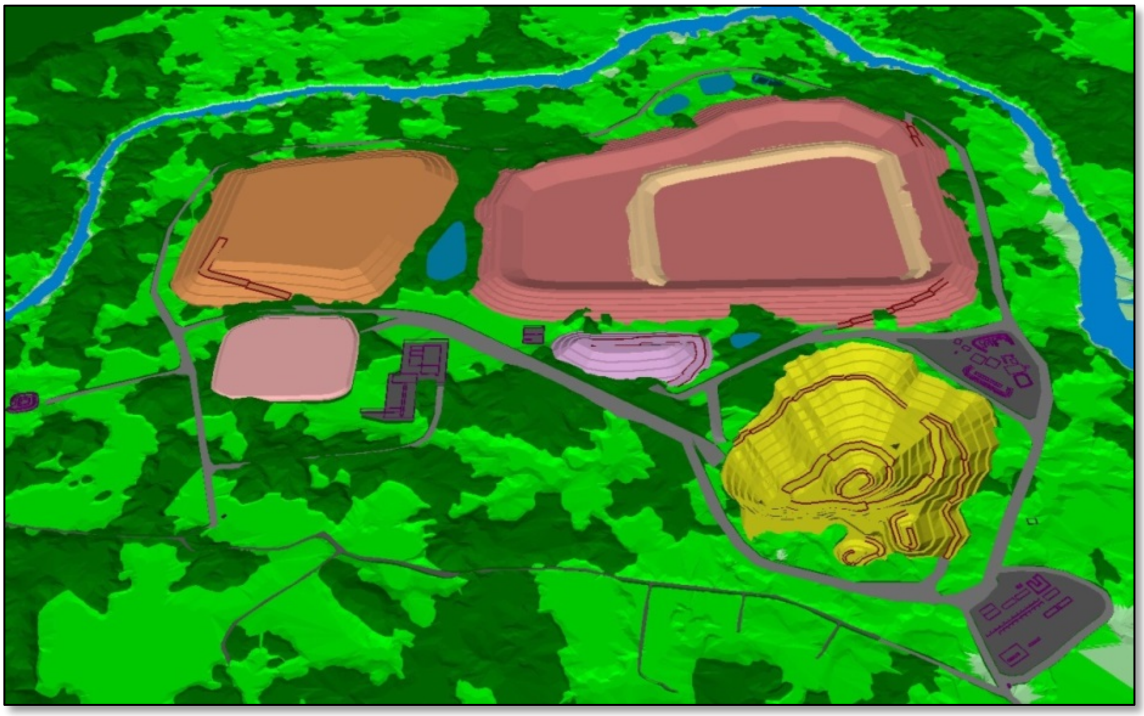


Figure 1-20 - Jugan – Alternate Infrastructure Layout (3D)

1.12.2. Infrastructure – BYG-Krian

The infrastructure at BYG-Krian is minimal and will be treated as a satellite operation with only the pit and waste dump plus haul roads required as the current mineable portion has a short time frame of operation. Should the Inferrred zones be drilled and resources upgraded then these can be converted into reserves which may extend the life of this pit.

Depending on this the infrastructure may remain the similar (in terms of pit, waste landform and haul roads) or may generate the need for other infrastructure requirements. This is a number of years further on in the schedule with a suitable decision made nearer the time.

The mine layout and 3D view of the current BYG-Krian pit are shown in *Figure 1-21 - BYG-Krian: - Infrastructure Layout Plan* and *Figure 1-22 - BYG-Krian: - Infrastructure Layout (3D)* below.

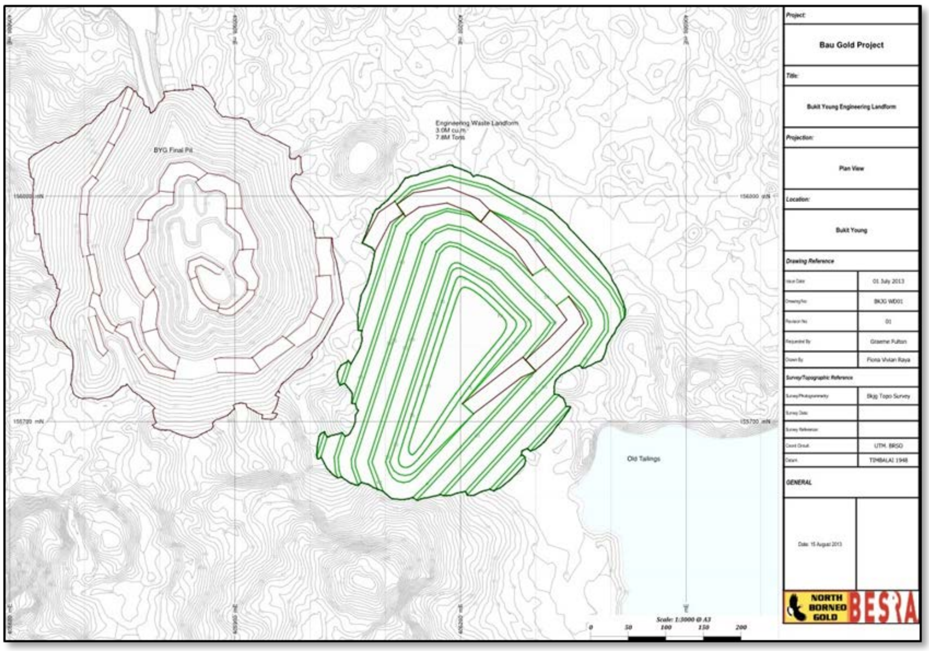


Figure 1-21 - BYG-Krian: - Infrastructure Layout Plan

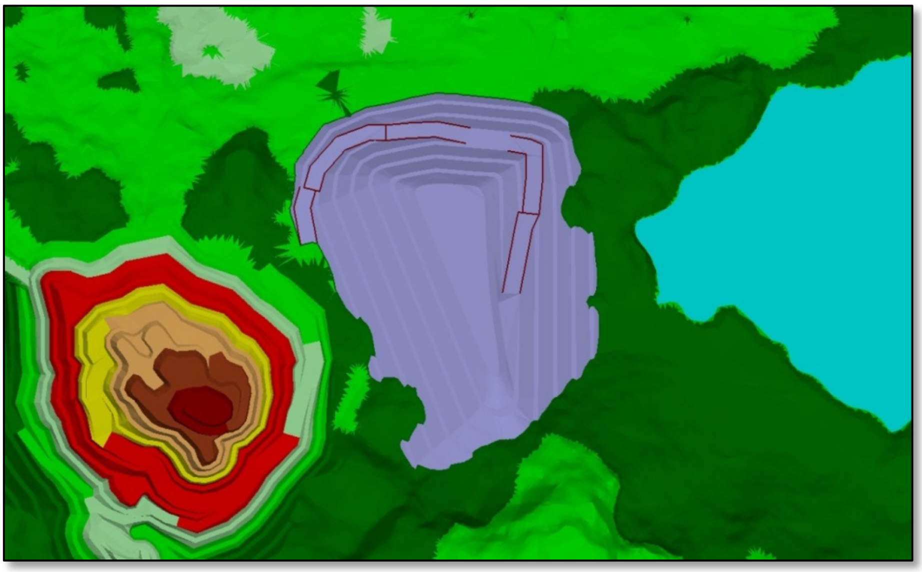


Figure 1-22 - BYG-Krian: - Infrastructure Layout (3D)

1.13. Environment

Besra is actively pursuing the development of economical mineral resources in the Bau goldfield and at the same time, the company's is committed to undertake an environmental impact assessment to identify issues and data gaps to develop an environmental management and rehabilitation plan in compliance to international standard and local regulatory requirement, in order to create a sustainable environment during post-mining operation and closure. The aim is to create a post-mining environment that is at least equal in environmental value or better.

Regulatory framework and compliance plays an important role to guarantee the basis for environmental impact assessment is being addressed appropriately to ensure sustainable development of mineral resource and at the same time, promote environmental stewardship to ensure the mining project is being developed in an environmentally sound, responsible and sustainable manner.

Socio-economic undertaking such as stakeholder engagement, local community development, public relation and liaison present a unique opportunity for the company to encourage local community participation and interest for dialog to identify possible concerns and expectation relating to socio-economic development and environmental awareness. This will foster relations and create credibility which will further elevate the company's public image and reputation. As such, some of the key environmental aspects, which required attention as the project progresses into a viable mining operation, are:

- Identification of environmental impacts and constraints associated with exploration and mining activities to ecology, socio-economic including historical / cultural sites in the area;
- The ecological impacts due to alteration of pre-existing environment such as land clearing and landscape modification effecting indigenous wild life species in the area
- The concern of Acid Mine Drainage (AMD) generation due to the oxidation of sulphide minerals from ore and waste and mitigation measure associated with the rehabilitation and regulatory compliance;
- Socio-economic effects (both positive and negative) on local communities associated with or affected by the mining development, and the costs involved in maximising positive and minimising negative socio-economic effects; and
- The scale and nature of rehabilitation scope for the eventual mine closure and post closure monitoring required for all deactivated mined-out areas and associated auxiliary structures and facilities.
- Post mining environmental monitoring to ensure the success of rehabilitation and the preservation of a sustainable ecosystem.

In order to achieve the task of creating a sustainable environment post mining, control measures and mitigation methods need to be in place to counter potential environmental impact. Hence, mine planning isincorporating the following management framework for the integration of site specific mitigation design. These are:

- Environmental Impact Assessment (EIA)
- Environmental Management Plan (EMP)
- Mine Rehabilitation Plan (MRP)
- Monitoring Program
- Alternatives Land Use Planning

A conceptual MRP with two updates has already been submitted to the authorities and accepted. An update to the MRP based on the Feasibility Study is nearly complete and ready for submission shortly. The majority of the baseline studies and monitoring have been undertaken over the last few years as a basis for the EIA. Some minor studies and ground water modelling are currently underway. Once complete the formal EIA submittal will begin.

Through proper incorporation of the above mentioned management framework, a progressive rehabilitation process can commence to deal with potential long-term environmental impacts due to mining. The objectives of rehabilitation schemes are to develop achievable goals at various stages as mine planning evolves by converting an area of concern to a safe and stable condition, restoring the site to a pre-mining condition as closely as possible in order to ensure sustainability development.

Mine rehabilitation is essentially a process whereby the development of mineral resources is being conducted in accordance with the principles of leading sustainable practice. Rehabilitation should be part of an effective integrated program coexisting with mine operation and mine development in all phases.

In summary, mining is a temporary use of the land, the successes of a mining venture lies with the notion that the mining operator has successfully integrated mining best practice with the development of sustainable mining operation and integrated the best mine closure standards by ensuring the future of the land is not compromised but rather in a sustainable manner.

The temporary use of the land and the current and future aspects are demonstrated in *Figure 1-23 - Jugan - Current Land Situation* to *Figure 1-25 - Jugan - Possible Land Use Options after Mining* below.

Figure 1-23 - Jugan - Current Land Situation

Figure 1-24 - Jugan - Mining Operations for Short Time

Figure 1-25 - Jugan - Possible Land Use Options after Mining

1.14. Capital & Operating Costs

1.14.1. Capital Costs

Capital costs for the Feasibility Study are detailed in *Section 21* of this report with further detail listed in the *Appendices*. Listed in *Table 1-5 - Initial Capital Costs* below is a summary of the initial capital costs grouped by major elements, for the base case – 8,000tpd, gold concentrate process, and contract mining option.

Capital Cost Group	*Total Cost (US$)*
Mining – Mobile Equipment	0
Mining – Fixed Equipment	196,150
Mining – Construction	3,771,680
Mining - Other	336,700
Total – Contract-Mining	**4,304,530**
Process – Main Plant Items	24,372,000
Process – Other Plant Items	7,400,000
Process – Ancilliary (incl. EPCM)	26,775,850
Total – Process Plant	**58,547,850**
Other – TSF Stage 1	8,122,880
Other – Infrastructure	8,345,970
Other – General	12,798,485
Total – Other	**29,267,335**
Total Initial Capital	***92,119,715***

Table 1-5 - Initial Capital Costs

The capital for the owner-operator option for the same scenario is $20.2M more due to the cost of the mining equipment. An amount of is also set aside for sustaining capital. With major ongoing capital costs costed separately during the life of the mine. These are TSF stage 2 & 3 and water infrastructure additions ($19.76M); rehabilitation for pre-closure, closure and post-closure ($7.16M); and other general items ($10.32M). This gives a total of $42.76M in ongoing capital.

1.14.2. Operating Costs

Total operating costs for the 8,000 tpd base case (contract-mining) are $31.38/tonne, with the equivalent owner operator option being $38.64/tonne. The contract-mining base case is broken down in the following *Table 1-6 - Operating Cost Breakdown (8,000 tpd Base Case - Contract-Mining)* below.

Operating Cost Group	*Cost (US$/t)*
Mining	9.59
Processing (incl. conc. transport & processing)	21.24
General & Admin	0.55
Total Operating Cost/Tonne	**31.38**

Table 1-6 - Operating Cost Breakdown (8,000 tpd Base Case - Contract-Mining)

1.14.3. Royalties, Taxes and Incentives

In Malaysia the corporate income tax is 24 % of net taxable profits. Other taxes are GST (10 %) and where applicable a service tax (6 %) – where services only are provided.

Import duties are applicable at a rate of bewtween 20-30 % for most standard goods; however, drilling and mining equipment are subject to nil import tariffs based on the individual item and related part numbers.

There is no royalty (0 %) on gold produced in Sarawak, and there is no export duty or tariff for gold concentrate. Exploration and prospecting costs are eligible for special tax allowances.

1.15. Project Economics

1.15.1. Economic Analysis

The economic evaluation of the Bau Gold Project was developed in a detailed cost model using a discounted cash flow on a pre-tax basis. The Net Present Value (NPV) and the Internal Rate of Return (IRR), based on a discount rate, was calculated for each project scenario option, and in particular the two base case options.

Case	NPV (Millions $)	IRR (%)
Contract Mining – 8,000tpd Flotation Concentrate	91.41	38.0
Owner Operator – 8,000tpd Flotation Concentrate	97.29	34.3

Table 1-7 - NPV & IRR – Base Case Scenarios

The following assumptions were made for the analysis:

- Gold price of $1,300/oz – being a conservative value below the 2013 average ($1,415.48)
- Discount rate applied – 8%
- Pre mining occurs – production build-up 6 months – process build-up over 6 months with 1 quarter offset from production – pre-mining & construction 1 year from EIA and other approvals in place;
- No escalation or inflation factor applied (constant 2013 $)

For the contract mining/owner operator case the cost per ounce is $1,030.61/oz and $1,010.50/oz respectively. Including land resale and salvage the cost per ounce drops to $973.14/oz and $945.52.

To determine the after-tax discounted cash flow the following assumptions are applicable:

- Zero percent (0%) royalty on gold produced;
- No export duty or tariffs applicable to gold concentrate exports;
- Corporate income tax is 24 % of net taxable profits.

In the case of the after-tax situation the Net Present Value (NPV) and the Internal Rate of Return (IRR), based on a discount rate, was calculated for each project scenario option, and in particular the two base case options.

Case	NPV (Millions $)	IRR (%)
Contract Mining – 8,000tpd Flotation Concentrate	71.98	32.6
Owner Operator – 8,000tpd Flotation Concentrate	76.11	29.4

Table 1-8 - After-Tax NPV & IRR – Base Case Scenarios

1.15.2. Sensitivity Analysis

A sensitivity analysis was conducted on the parameters deemed to have the biggest impact on the project financial performance: gold price, CAPEX, mining OPEX, process OPEX, grade & recovery. A range of gold prices form $1,100/oz to $2,000/oz and ±10% and ±20% variations on the other parameters were used around both base cases above. The NPV and IRR for the project were found to be more sensitive to gold price, grade and recovery, and less sensitive to CAPEX and OPEX.

1.15.3. Risk Analysis

A detailed risk analysis list has been developed and scored based on "consequence" and "probability" and the impact assessed with each risk having associated mitigation measures identified. Additionally, a risk matrix developed and the identification of the the keys risks for further work using the mitigation measures. Other lower level risks will be addressed thereafter. This is a live document and will be updated as the project progresses.

1.16. Conclusions & Recommendations

In conclusion, BESRA finds the first stage of the plan to develop and put the Bau Goldfield into production is a lean business case and economically viable strategy with manageable risks.

The region has significant opportunity for growth and by moving into detailed engineering and construction now BESRA can best be setup for a return to higher gold prices and for developing long term partnerships with the smelter customers. Strategically, the concentrate option offers advantages as fuel source for the smelters while leaving BESRA the opportunity for secondary processing on site should a more robust gold market return.

By moving into production now BESRA is able to generate significant cash flow to further improve the gold field resources and reserves as well to take advantage of the opportunity for growth with the site infrastructure built up to then. BESRA has become a stronger operator every year of its existence and the management team are fully aware of the lessons learned from the past while being cautiously optimistic about the next step in our future in Malaysia.

2. Introduction & Scope

2.1. Introduction

Besra Gold Inc. (formally Olympus Pacific Minerals Inc. ("Olympus")) ("Besra") is a Canadian incorporated public company listed on the Toronto Stock Exchange, under the trading symbol BEZ and on the Australian Securities Exchange under the symbol BEZ and trades on the OTCQX Bulletin Board ("OTCQX"), an over-the-counter market in the United States under the symbol BSRAF. Its head office is located in Toronto.

On 17th December 2009, shareholders of Zedex Minerals Limited ("Zedex"), a public company that was incorporated in New Zealand and listed on the Australian Stock Exchange (ASX), approved the amalgamation of Zedex with Olympus Pacific Minerals. Following the amalgamation Olympus commenced trading on the ASX on 5th February 2010.

One of the assets acquired by Olympus through the amalgamation was Zedex's interest in the Bau Gold Field in Sarawak, Malaysia.

In November 2012 Olympus Pacific Minerals Inc. was renamed Besra Gold Inc.

The Bau Project JV is managed by Besra through its majority owned subsidiary, North Borneo Gold Sdn Bhd, ("NBG") a Malaysian incorporated company. The other shareholders are a Malaysian Mining Group, Gladioli Enterprises Sdn Bhd ("Gladioli") and Golden Celesta Sdn Bhd, (a consortium of private interests).

On September 30, 2010 (as amended on May 20, 2011, January 20, 2012 and May 15, 2013), the Company entered into an agreement to acquire a further 43.50% interest in NBG. The settlement is to be paid in several tranches as set out below in *Table 2-1 - Revised Share Tranche Payment Schedule* and will bring the Company's effective interest to 93.55% by September 2015.

Amount	Purchase Date	Total per Annum	Effective Holdings
US$600,000	06/14/2013		85.61%
US$800,000	09/02/2013*		86.36%
US$800,000	12/02/2013*	US$2,200,000	87.10%
US$900,000	03/03/2014		87.95%
US$900,000	06/02/2014		88.80%
US$1,000,00	09/01/2014		89.75%
US$1,000,00	12/01/2014	US$3,800,000	90.70%
US$1,000,00	03/02/2015		91.65%
US$1,000,00	06/01/2015		92.60%
US$1,000,00	09/01/2015	US$3,000,000	93.55%

Deferred until February 2014

Table 2-1 - Revised Share Tranche Payment Schedule

2.2. Terms of Reference

Stevens & Associates and Terra Mining Consultants Ltd ("TMCSA") were previously retained by Olympus Pacific Minerals Inc. (Olympus) to carry out an independent technical review of the Bau Gold Project in Sarawak, Republic of Malaysia (the "Property") following the merger between Olympus and Zedex Minerals Ltd (Zedex) in late 2009. In particular, that review upgraded the then existing Mineral Resource estimates for the project to NI43-101 standards and produce new Mineral Resource estimates for several additional areas of the project where applicable.

That report set out the results of:

- A review and update of all available project data, including historic mining and exploration data and recent data from NBG's exploration since 2007;
- Several site visits to the Bau office and project areas at Bau and the surrounding district by Stevens and Associates and Terra Mining Consultants between 2nd December 2009 and 8th June 2010;
- Updated Mineral Resource estimates for the key gold deposits at Bau, including Jugan, Pejiru, Sirenggok and Bukit Young Tailings as well as new estimates for the additional areas of Taiton and Bekajang-Krian where there is sufficient data to support NI43-101 compliant resources;
- Review other relevant data including metallurgical factors and the environmental framework of operating in Sarawak.

This report sets out the results of the Company's resource and exploration programmes at Bau since August, 2010 until 31 October, 2012 where upgraded and expanded resource estimates were announced on February, 2012 and November 2012.

The report also sets out the work conducted for the Feasibility Study, particularly on the Jugan and BYG-Krian deposits within the Bau Project.

The report has been prepared in compliance with the standards of National Instrument 43-101 ("NI43-101") in terms of structure and content and the Mineral Resource estimates were carried out in accordance with the provisions of NI43-101 guidelines and the Council of the Canadian Institute of Mining, Metallurgy and Petroleum definitions ("C.I.M.M. Standards") and in compliance with the Australasian Institute of Mining and Metallurgy code for reporting mineral resources, (JORC).

2.3. Sources of Information & Data

Previously, Stevens and Associates and Terra Mining Consultants (TMCSA) have relied on reports and information prepared by and/or for Zedex Minerals Ltd and supplied by Olympus, historic and past reports prepared by Menzies Gold NL, Gencor, Renison Goldfields, Bukit Young Gold Mines Sdn Bhd, Wolf Schuh's PhD thesis, original paper assay and geological data records, soft copy data and observations made by TMCSA. Portions of the descriptive material used in this report have been taken from all of the above.

In addition, both Stevens and Associates and Terra Mining Consultants have been retained by North Borneo Gold Sdn Bhd to manage the Exploration and Resource Development programmes, and the Feasibility Study at Bau, and have relied on the results of work compiled and supervised by them. A full list of documents used in this report is listed in *Section 27 - References*.

2.4. Site Inspection

In relation to the August, 2010 NI 43-101 report, several site inspections were carried out by Murray Stevens, Consulting Geologist to Stevens and Associates. These took place on 5th to 22nd December 2009, 13th January to 9th February 2010, 1st to 30th March 2010, 30th April to 8th June 2010. Mr. Graeme Fulton, Consulting Mining Engineer/Director of Terra Mining Consultants, visited the site on several occasions, between 2nd December 2009 and 17th December 2009, 10th January to 10th February 2010, between 16th March and 10th April 2010 with a final site visit from 30th April to 8th June 2010.

Discussions were held with Olympus management and technical personnel on site at Bau and in Olympus's office's in Auckland, New Zealand.

Representative samples of drill core were examined from drill holes at all the deposits modelled.

Both Mr. Fulton and Mr. Stevens conducted their evaluation of the data and resource modelling on site at Bau, and in the offices of TMCSA in Auckland, New Zealand.

Mr. Stevens reviewed quality control procedures, core and sample handling procedures, core logging procedures and security procedures on site. In addition, a representative number of samples were selected and tracked through the QAQC procedures to confirm data integrity.

In relation to the current report, both Mr. Fulton and Mr. Stevens are involved with the project on a daily basis (and have done since September 2010) reviewing, planning and executing the exploration and resource programmes and undertaking the project feasibility study and compiling the associated report.

2.5. Units & Currency

Metric units are used throughout this report unless noted otherwise. Currency is United States dollars ("US$" or "$"), Canadian dollars ("C$"), New Zealand dollars ("NZ$") or Malaysian Ringgit, (MYR). In early August, 2013 the currency exchange rates were approximately 3.10MYR equals US$1.00. For converting grams of gold to ounces of gold, a factor of 31.1035 grams per troy ounce is used.

2.6. Naming Conventions

A full list of naming conventions, abbreviations and technical nomenclature can be found in *Appendix A2-1*.

3. Reliance on Other Experts

This report has been compiled by NBG/Besra staff under the supervision of Mr. Graeme Fulton, General Manager at NBG. The NBG personnel are experienced technical professionals in their respective areas of expertise

Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report.

The following people have contributed to this report, and have done so under the supervision of a Graeme Fulton a Qualified Person (QP), who has also written, edited and reviewed sections of this report and has provided a QP certificate. Their areas of expertise and section contributions are listed below:

Graeme Fulton

Qualifications: BSc. (Hons) Mining & Petroleum Engineering
Affiliations: Fellow of AusIMM
Experience: 29 years
Position: General Manager – Malaysia
Sections: 1, 2, 3, 6, 12, 14, 15, 16, 19, 21, 22, 24

Murray Stevens

Qualifications: BSc. & MSc. (Hons) Geology; Dip. Geol.Sci.
Affiliations: Member of AusIMM
Experience: 35 years
Position: Consulting Geologist
Sections: 4, 5, 7, 8, 9, 10, 11, 23

Erik Devyust

Qualifications: BSc. Mining Engineering, MSc. & PhD in Hydrometallurgy
Affiliations: CIMM
Experience: 40 years
Position: Technical Services Director (Metallurgy)
Sections: 13, 17, 21

Jayakumar Pillai Balakrishna

Qualifications: BSc Engineering (Mechanical)
Affiliations:
Experience: 31 years
Position: Technical Services Manager (Engineering)
Sections: 17, 18, 21

Dickson Lapniten

Qualifications: Bsc Mining Engineering
Affiliations: PIMMGE
Experience: 33 years
Position: Senior Planning Manager
Sections: 16, 21

Jose Sanchez

Qualifications: B.S. Mining Engineering; M.S. Geotechnical Engineering (in progress)
Affiliations:
Experience: 16 years
Position: Senior Geotechnical Engineer
Sections: 16, 18, 24

Brando Pang Tze Chiang

Qualifications: Diploma in Mineralogy; Advanced Geological Courses & Mineralogy from the University of Alberta
Affiliations:
Experience: 14 years
Position: Senior Environmental Engineer
Sections: 20

Other staff within NBG also contributed to limited aspects of this report, in terms of diagrams, tables, figures, designs, etc. but did not contribute directly to the writing of the report.

NBG/Besra has not relied on any other experts for legal or technical matters.

4. Property Description & Location

4.1. Location

Figure 4-1 - Property Location Plan

The Bau Goldfield projects are located on the Island of Borneo in Sarawak, Federation of Malaysia. The project area is centered on the township of Bau some 40 km WSW of the state capital of Kuching (population ~600,000); see *Figure 4-1 - Property Location Plan* above.

4.2. Regulation of Mining Industry & Foreign Investment in Malaysia

The two main legal instruments that govern activities relating to minerals are the Mineral Development Act, 1994 and the State Mineral Enactment. The Mineral Development Act, 1994 came into force in August 1998. The State Mineral Enactment for Sarawak, where the Bau Gold Project is located, is entitled the "Minerals Ordinance, 2004" and was proclaimed into effect on July 1, 2010.

The Mineral Development Act 525 of 1994 defines the powers of the Federal Government for inspection and regulation of mineral exploration and mining and other related issues. The State Mineral Enactment provides the States with the powers and rights to issue mineral prospecting and exploration licenses and mining leases and other related matters. The Governor of the state of Sarawak, in which the Bau Project is located, has statutory rights to forfeit or cancel the mining tenements if there is a breach of, or default in the observance of any of the covenants or conditions attached to the relevant Mining Tenement.

Parties may apply for a General Prospecting License or an Exclusive Prospecting License for an initial term of two years (with one renewal period for a further two years). Mining operations require a Mining Lease, or in the case of a Mining Lease where the boundary survey of the area has not been completed, a Mining Certificate. In either case, the maximum term is 21 years. The mineral tenure regime in Sarawak is explained in more detail in the next section.

Malaysia has been a member of the World Trade Organisation ("WTO") since 1 January 1995 and has made various commitments pursuant to the General Agreement on Trade in Services ("GATS") including setting out the transactions relating to investment in Malaysia which would require approval. Since Malaysia is a member of the WTO, foreign companies under the terms of the WTO membership are expected to be treated on an equal basis as Malaysian Companies.

No restrictions are imposed on foreign companies investing in Malaysia with regard to repatriation of capital, interest, profits and dividends. No gold royalties are payable to the Federal Government.

4.3. Mineral Tenure Regime

All mineral resources in Malaysia are state owned. Exploration and mining rights are issued subject to the recently gazetted Minerals Ordinance 2004 which has an effective commencement date of 1 July, 2010, and Mining Rules (1995).

The following *Table 4-1: Sarawak Mining Tenure Types - General Prospecting Licence (GPL)* to *Table 4-4: Sarawak Mining Tenure Types - Mining Lease (ML)* summarises the exploration and mining tenure types that are applicable in Sarawak, and to the Bau Project.

Licence Type	Parameters	Parameter Description
General Prospecting Licence (GPL)	Max Size	200 km^2 (50,000 acres) Pre 1991 tenements may be larger
	Term	2 years standard Renewable to maximum 6 years (3 x 2yrs) Convert to EPL after 1st 2 year term
	Rental	RM 0.50/Ha/year payable at start of term
	Obligations	No minimum expenditure 6 monthly report within 30 days Final report within 3 months of term expiry date
	Notes	Renewal application with final report 50% compulsory relinquishment end of 1st 2 year term Additional 10% relinquishment after 2nd 2 year term

Table 4-1: Sarawak Mining Tenure Types - General Prospecting Licence (GPL)

Licence Type	Parameters	Parameter Description
Exclusive Prospecting Licence (EPL)	Max Size	20 km^2 (5,000 acres) Pre 1991 tenements may be larger Multiple EPL's allowed up to max.
	Term	4 years standard Renewable for subsequent 4 years
	Rental	RM 1.50/Ha/year (or part thereof) payable at start of term
	Obligations	Minimum expenditure of RM 75,000 over EPL term (4yrs)

Licence Type	Parameters	Parameter Description
		6 monthly report within 30 days Final report within 3 months of term expiry date
	Notes	Renewal application with final report No compulsory reduction for 2nd term

Table 4-2: Sarawak Mining Tenure Types - Exclusive Prospecting Licence (EPL)

Licence Type	Parameters	Parameter Description
Mining Certificate (MC)	Max Size	2,000 hectares Pre 1991 tenements may be larger
	Term	21 year maximum Renewal 1 year before expiry
	Rental	RM 10/Ha/year (or part thereof) paid annually 10% penalty for any arrears
	Obligations	No minimum expenditure Final report within 3 months of new calendar year (March)
	Notes	Does not extinguish any previously existing land titles and allows mining in unalienated land with the permission of the owner and requires negotiation of compensation and royalty

Table 4-3: Sarawak Mining Tenure Types - Mining Certificate (MC)

Licence Type	Parameters	Parameter Description
Mining Licence (ML)	Max Size	2,000 hectares
	Term	21 year maximum Renewal 1 year before expiry
	Rental	RM 10/Ha/year (or part thereof) paid annually 10% penalty for any arrears
	Obligations	No minimum expenditure Final report within 3 months of new calendar year (March)
	Notes	In the case of unalienated land, all land issues such as Native Customary Rights must be recorded by Lands & Surveys Department prior to the issuance of ML If no renewal, the land reverts to 'State land' irrespective of what other titles may have pre-existed

Table 4-4: Sarawak Mining Tenure Types - Mining Lease (ML*)*

Aspects of Sarawak tax law encourages new investment with an investment tax allowance (ITA) that provides for 60% ITA on qualifying capital expenditure incurred for 5 years, subject to a maximum income tax exemption on 70% of statutory income for a year of assessment. Unutilized allowances can be carried forward to subsequent years along with an exemption from import duty and sales tax on machinery/equipment.

The current Sarawak mining ordinance sets mineral royalties at 5% ad valorem on all minerals except gold for which the royalty rate is zero (0%).

4.4. Property & Tenement Description

The current exploration and mining tenements that cover the property and comprise the Bau Project Joint Venture and their status are outlined in *Table 4-5: Granted Mining Leases (ML) Subject to Joint Venture* to *Table 4-14: General Prospecting Licenses under New Application* below and shown in *Figure 4-1 - Property Location Plan, Figure 4-2: Tenement Location Map for Bau Showing Mining Leases, Mining Certificates and EPL & GPL Applications Subject to Joint Venture* shows the tenure of the more advanced projects in more detail.

The tenements subject to the joint venture cover three regions in Sarawak. Blocks A and B relate to the Bau District. The other two regions known as Block C (Serian area) and Gunong Rawan lie east of Bau and near the Sarawak/Kalimantan Border. These are still at early stage exploration or under application.

Company/Applicant	Ex-ML/MC No.	New ML No.	Area (Ha)	Minerals	Expiry Date
Bukit Lintang Enterprises Sdn. Bhd.	ML 102	1D/134/ML/2008	40.50	Au	11/06/2025
Priority Trading Sdn. Bhd.	ML 108	ML 136	139.6	Sb/Ag/Au/Ca	18/01/2023
Bukit Lintang Enterprises Sdn. Bhd.	ML 109	ML/01/2012/1D	12.74	Sb/Au	18/01/2023
Carino Sdn. Bhd.	ML 115	ML/03/2012/1D	49.4	Au	04/03/2024
Gladioli Enterprises Sdn. Bhd.	ML 117 (A) & (B)	ML/04/2012/1D	52.1	Sb/Ag/Au/Ca	09/01/2025
Gladioli Enterprises Sdn Bhd	ML 119	ML/05/2012/1D	5.28	Sb/Ag/Au/Ca	09/01/2025
Bukit Lintang Enterprises Sdn. Bhd.	ML 121	ML 142	38.40	Sb/Au	11/06/2025
Bukit Lintang Enterprises Sdn. Bhd.	ML 122	ML/02/2012/1D	49.81	Sb/Au	22/06/2024
Priority Trading Sdn. Bhd.	ML 123	1D/137/ML/2008	2.6	Sb/Au	22/06/2024
Buroi Mining Sdn. Bhd.	ML 125	ML 138	409.5	Sb/Ag/Au/Ca	19/11/2025
	MC No. 1D/1/1987	ML 01/2013/1D	380.2	Sb/Au	22/01/2033
	Total Area (Ha)		**1,180.13**		

Table 4-5: Granted Mining Leases (ML) Subject to Joint Venture

Company/Applicant	Lease No.	Area (Ha)	Minerals	Expiry Date	Application Date
Gunong Wang Mining Sdn. Bhd.	ML 101	48.16	Au/Sb	30/10/1999	31/10/1998
	Total Area (Ha)	**48.16**			

Table 4-6: Mining Lease under Renewal Application

Company/Applicant	Certificate No	Area(Ha)	Minerals	Expiry Date
Gladioli Enterprises Sdn. Bhd.	MC No. KD/01/1994	1,694.86	Sb/Ag/Au	26/10/2014
	Total Area (Ha)	**1,694.86**		

Table 4-7: Granted Mining Certificates (MC) Subject to Joint Venture

Company/Applicant	Certificate No	Area(Ha)	Minerals	Expiry Date	Application Date
Gladioli Enterprises Sdn. Bhd.	MC No. 1D/2/1987 (A)	82	Not specified	12/07/2008	15/03/2008
Gladioli Enterprises Sdn. Bhd.	MC No. 1D/2/1987 (B)	3,237	Not specified	12/07/2008	15/03/2008
Gladioli Enterprises Sdn. Bhd.	MC No. 1D/3/1987	7,240	Not specified	31/07/2008	15/03/2008
Gladioli Enterprises Sdn. Bhd.	MC No. SD/1/1987	1,379	Sb/Ag/Au	12/07/2008	15/03/2008
	Total Area (Ha)	**11,938.00**			

Table 4-8: Mining Certificate under Renewal Application

** Expiry date of MC 1D/1/1987 not specified in original MC document. The expiry date of MC 1D/1/1987 has been assumed to expire on 12/07/2008, same expiry date for MC 1D/2/1987 & MC SD/1/1987 since they were issued at the same time.*

Company/Applicant	Expired ML No. [1]	Area (Ha)	Minerals	Expiry Date	Application Date [2]
Gladioli Enterprises Sdn. Bhd.	Ex-ML 93	17.10	Au/Ag/Base Metals	28/08/2001	22/09/2006
Gladioli Enterprises Sdn. Bhd.	Ex-ML 129	263	Au/Ag/Base Metals	26/02/2002	22/09/2006
Gladioli Enterprises Sdn. Bhd.	Ex-ML 132	126	Au/Ag/Base Metals	01/04/2003	22/09/2006
	Total Area (Ha)	**406.10**			

Table 4-9: Mining Certificate Applications over Expired Mining Leases of Other Companies

[1] Ex-ML 93: Syarikat Tabai Sdn Bhd
Ex-ML 129: Syarikat Kalimantan Enterprise Sdn Bhd
Ex-ML 132: Southern Gold Mining Development Sdn Bhd

[2] Presentation to the authority carried out on 19/09/2009
Application Forms submitted on 21st September 2007

Company/Applicant	License No.	Area (Ha)	Minerals	Expiry Date	Application Date
Gladioli Enterprises Sdn. Bhd.	EPL 326-329 (Lot 1)	7,163	Au/Ag/Hg/Ca	11/05/1990	05/01/1990
Gladioli Enterprises Sdn. Bhd.	EPL 316-325 (Lot 2)	1,210	Au/Ag/Hg/Ca	12/05/1990	05/01/1990
Gladioli Enterprises Sdn. Bhd.	EPL 308-311 (Lot 3a)	1,070	Au/Ag/Hg/Ca	15/05/1990	05/01/1990
Gladioli Enterprises Sdn. Bhd.	EPL 308-311 (Lot 3b)	3,785	Au/Ag/Hg/Ca	15/05/1990	05/01/1990
Gladioli Enterprises Sdn. Bhd.	EPL 312-315 (Lot 4)	8,373	Au/Ag/Hg/Ca	15/05/1990	05/01/1990
Gladioli Enterprises Sdn. Bhd.	EPL 337 [Lot 5A]	1,817	Au/Ag/Base Metals	14/12/1997	06/03/1998
Gladioli Enterprises Sdn. Bhd.	EPL 337 [Lot 5B (1)]	1,897	Au/Ag/Base Metals	14/12/1997	06/03/1998
Gladioli Enterprises Sdn. Bhd.	EPL 338 [Lot 6]	763.53	Au/Ag/Base Metals	14/12/1997	06/03/1998
Gladioli Enterprises Sdn. Bhd.	EPL 339 [Lot 9]	1,710	Au/Ag/Base Metals	14/12/1997	06/03/1998
Gladioli Enterprises Sdn. Bhd.	EPL 340 [Lot 7]	927	Au/Ag/Base Metals	27/09/1996	20/09/1998
	Total Area (Ha)	**28,715.53**			

Table 4-10: Exclusive Prospecting Licenses (EPL) under Renewal Application

Company/Applicant	License No.	Area (Ha)	Minerals	Application Date
Gladioli Enterprises Sdn. Bhd.	EPL [Lot 8] [#]	2,000	Au/Ag/Base Metals	09/08/1994
	Total Area (Ha)	**2,000.00**		

Table 4-11: Exclusive Prospecting Licences under New Application

[#] Gladioli Enterprises Sdn Bhd applied:
(i) to renew 2 portions of the original area of GPL no. 3/1992;
(ii) for one EPL [Lot 8] to be issued from part of the original area of GPL No. 3/1992

Company/Applicant	License No.	Area (Ha)	Minerals	Expiry Date	Grant Date
Gladioli Enterprises Sdn. Bhd.	GPL 01/2008/1D	30.97	Au/Ag/Base Metals	13/04/2010	14/04/2008
	Total Area (Ha)	**30.97**			

Table 4-12: Granted General Prospecting License (GPL) Subject to Joint Venture

Company/Applicant	License No.	Area (Ha)	Minerals	Expiry Date	Application Date
Gladioli Enterprises Sdn. Bhd.	GPL No. 3/1992 a	2,800	Au/Ag/Base Metals	25/08/1994	09/08/1994
Gladioli Enterprises Sdn. Bhd.	GPL No. 3/1992 b	5,700	Au/Ag/Base Metals	25/08/1994	09/08/1994
Gladioli Enterprises Sdn. Bhd.	GPL No. 4/1992	4,061	Au/Ag/Base Metals	25/08/1994	09/08/1994
Gladioli Enterprises Sdn. Bhd.	GPL No. 7/1995	17,028	Au/Ag/Base Metals	09/11/1997	06/03/1998
Gladioli Enterprises Sdn. Bhd.	GPL 4/1996	492.90	Au/Ag/Base Metals	14/11/1998	30/10/1998
Gladioli Enterprises Sdn. Bhd.	GPL 39/1997	5726.50	Metal/Mineral other than Mineral Oil	21/08/1999	05/01/2000
	Total Area (Ha)	**35,808.40**			

Table 4-13: General Prospecting Licenses under Renewal Application

Company/Applicant	Licence No.	Area (Ha)	Minerals	Application Date
Gladioli Enterprises Sdn. Bhd.	SB1-SB6	77,500	Au/Hg/Cu/Sb/Coal/Industrial Minerals	27/03/1996
	Total Area (Ha)	**77,500**		

Table 4-14: General Prospecting Licenses under New Application

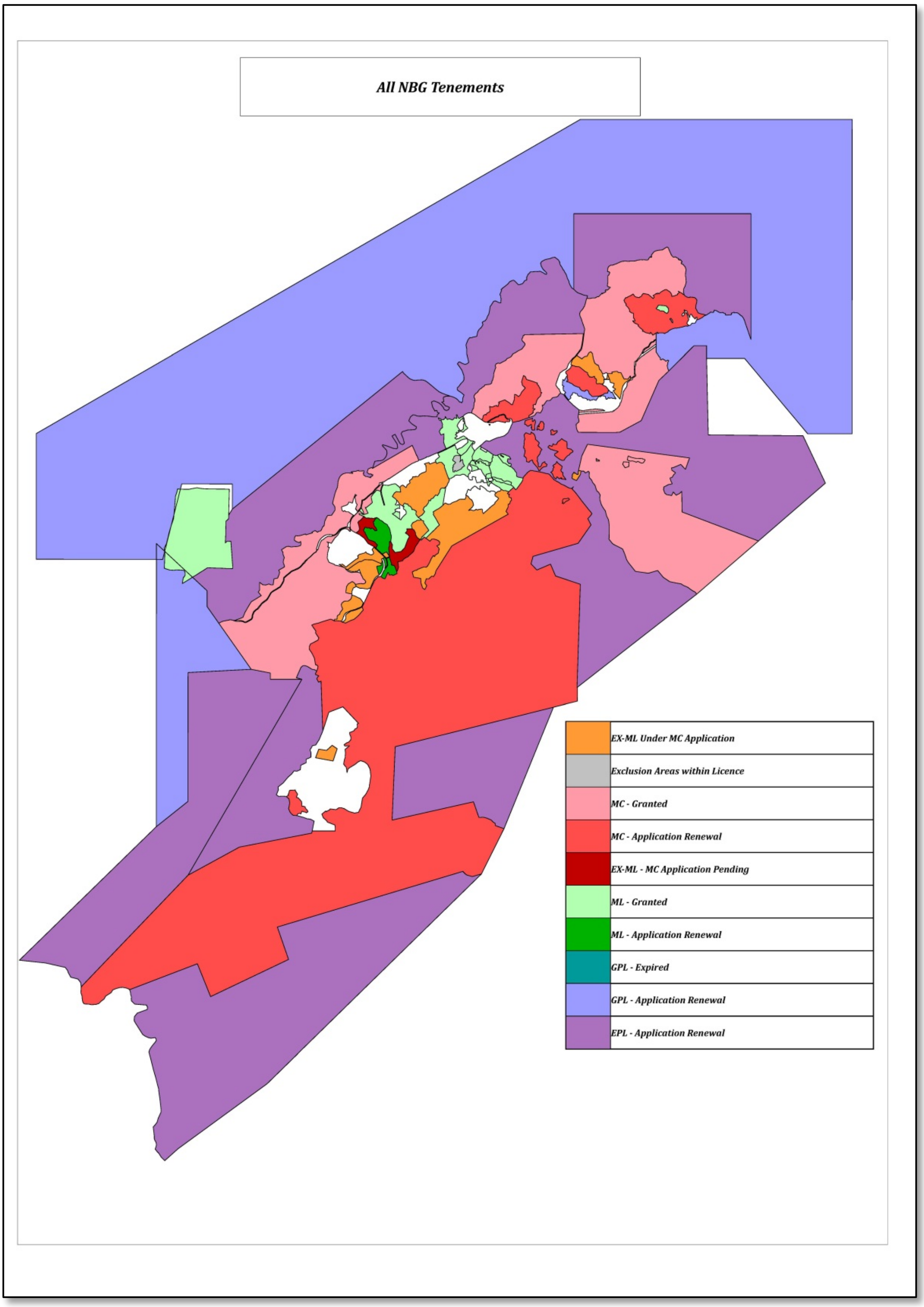


Figure 4-2: Tenement Location Map for Bau Showing Mining Leases, Mining Certificates and EPL & GPL Applications Subject to Joint Venture

4.5. Joint Venture Agreement with Gladioli Enterprises Sdn Bhd

4.5.1. Original Joint Venture Agreement

The rights and obligations of the joint venture between Zedex and Gladioli Enterprises are encumbent on Besra through its amalgamation with Zedex. Zedex (now Besra, formerly Olympus) and its then wholly owned subsidiary, North Borneo Gold Sdn Bhd entered into an earn-in agreement with Malaysian Mining Group, Gladioli Enterprises Sdn Bhd in November, 2006.

The principal terms of the agreement pursuant to which Zedex acquired a 50.05% interest in the Bau Gold Project are as follows:

- Zedex paid US$ 1 million to Gladioli. A further US$ 1 million will become payable to Gladioli as follows:
  - US$ 500,000 upon commencement of mining at Jugan deposit; and
  - US$ 500,000 payable six months after commencement of mining at Jugan deposit.
- Zedex (now Besra, formerly Olympus) is to fund exploration activities as operator (including all rents and licence fees and included US$ 230,000 in respect of existing rental payments) through to completion of a feasibility study, including meeting the following (cumulative) minimum expenditure requirements:
  - US$ 200,000 within 6 months of completion;
  - US$ 700,000 within 12 months of completion;
  - US$ 1 million within 18 months of completion.
- Zedex (now Besra, formerly Olympus) to be responsible for financing 100% of project development (upon a decision to mine). All exploration, development and capital to be treated as loans funds, which are to be recoverable from future production profits.
- If, upon completion of a positive feasibility study, Zedex (now Besra, formerly Olympus) does not use reasonable efforts to secure project finance, and project finance to develop the project is not secured within 12 months of completion of the study, Gladioli has right to require Zedex (now Besra, formerly Olympus) to transfer its interest in respect of the deposit the subject of the study to Gladioli.

4.5.2. Amended Joint Venture Agreement

On 30 September, 2010 the Company entered into an agreement, as amended on 20 May, 2011 and 20 January, 2012, to acquire up to 93.55% interest in North Borneo Gold Sdn Bhd by January, 2014 subject to payments to be made in several tranches.

The transaction is summarized in *Table 4-15: Amended Gladioli Joint Venture Payment Schedule* as follows:

Tranche	Purchase Price	Purchase Date	North Borneo Gold Class 'A' Shares	Company's Effective Holding
Tranche 1	$7,500,000	30/09/2010	31,250	62.55%
Tranche 2	$7,500,000	20/10/2010	31,250	75.05%
Tranche 3a	$6,000,000	20/05/2011	13,700	80.53%
Tranche 3b	$3,000,000	20/01/2012	6,800	83.25%

Tranche	Purchase Price	Purchase Date	North Borneo Gold Class 'A' Shares	Company's Effective Holding
Tranche 3c	$2,000,000	28/01/2013	4,500	85.05%
Tranche 4a	$3,000,000	13/09/2013	7,000	87.85%
Tranche 4b	$6,000,000	21/01/2014	14,250	93.55%
Total	**$35,000,000**		**108,750**	**93.55%**

Table 4-15: Amended Gladioli Joint Venture Payment Schedule

The agreement includes a condition subsequent that must be met before Tranche 3c payment is required to be settled. The condition subsequent requires the vendor to obtain:

- All renewals or grants (as applicable) of mining licenses and mining certificates relating to the Jugan Hill deposit (including, without limitation, the renewal of mining certificate MC 1D/1/1987 relating to Jugan Hill, Sirenggok and Jambusan areas) on terms acceptable to the Purchaser in all respects; and
- All ministerial, Governor and other regulatory approvals to ensure that the mining licences and certificates referred to at (a) above are valid and effective in all respects in accordance with applicable laws and regulations.

Conditions to be met before settlement of each tranche are as follows:

- Tranche 1 – has no conditions;
- Tranche 2 – amendment to the Joint Venture agreement to deal with a number of operational and governance matters; this condition was met on 30 October, 2010 and settlement of Tranche 2 occurred on that date;
- Tranche 3 – if the condition subsequent noted above has been met by 31 March, 2012 settlement of Tranche 3c payment occurs; if the condition subsequent has not been met then all remaining shares transfer to the purchaser at no additional cost; on completion of Tranche 3c the right of the vendor to appoint a director to the board of North Borneo Gold Sdn Bhd ceases;
- Tranche 4 – has no conditions.

The agreement was further amended on 15 May 2013, to acquire up to a 93.55% interest in North Borneo Gold Sdn Bhd (NBG) by September 2015, subject to payments to be made in several tranches:

Tranche Amount (US$)	Purchase Date	Yearly Amount (US$)	Effective Holdings (%)
600,000	14 June 2013		85.61
800,000	2 September 2013*		86.36
800,000	2 December 2013*	2,200,000	87.10
900,000	3 March 2014		87.95
900,000	2 June 2014		88.80
1,000,000	1 September 2014		89.75
1,000,000	1 December 2014	3,800,000	90.70
1,000,000	2 March 2015		91.65

Tranche Amount (US$)	Purchase Date	Yearly Amount (US$)	Effective Holdings (%)
1,000,000	1 June 2015		92.60
1,000,000	1 September 2015	3,000,000	93.55

**Deferred until February 2014*

Table 4-16 - Revised Share Tranche Payment Schedule as at 15th May 2013

During the quarter ended June 30, 2013 the Company reached an agreement to amend the payment schedule for the final tranches of the acquisition of NBG.

5. Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1. Accessibility

The project area is centered on the township of Bau, about 40 kms WSW from the port city and state capital of Kuching.

The project area is serviced by a network of sealed and gravel roads. Most of the main prospects and deposits can be accessed by vehicle tracks. The advanced prospects and deposits are all located within a 7 km radius of Bau Township. Foot access is required for some of the more rugged or outlying areas.

5.2. Climate

The Bau area is characterised by a typical monsoonal tropical climate with annual rainfall of around 3,500 to 4,000 mm. The highest rainfall usually occurs between December and January with significant rain possible all year round.

Mean temperatures range from a high of around 31°C to a low of 22°C, while humidity averages approximately 70%.

5.3. Local Resources

The Kuching District, (including Bau) has a population of approx. 640,000 people. At Bau the main population groupings are Bidayuh, from the Dyak ethnic group, and Chinese who are mainly descendants of early miners who came to the area in the mid to late 19th Century to exploit the gold and antimony deposits at Bau. Sarawak has a per capita GDP of US$1,400. Mining represents about 20% of Sarawak's GDP.

The main industries in the Bau district are limestone quarrying, fish farming, rice farming, palm oil and rubber production, and now mineral exploration.

5.4. Infrastructure

The Bau Project generally has good infrastructural aspects both within Bau Township and in Kuching. The main infrastructural features are:

- Regular and reliable international air services to Kuching from Kuala Lumpur, Singapore, Hong Kong and Indonesia. Airport is only a thirty-five to forty (35-40) minute drive from the project area;
- Two (2) ports with good dock and storage facilities (port has a capacity for vessels up to 17,000 tonnes);

- Two (2) main sealed trunk roads from Kuching for delivery of supplies, heavy plant and equipment to the plant site;
- Excellent labour and engineering support services;
- Easy Accessibility – project extremities are less than a twenty (20) minute drive from the exploration base, and all important mines and gold prospects are linked by road;
- Area is serviced with power and water;
- The official language in Sarawak is Bahasa Malaysia, but most local communities speak English as a second language and have their own local dialects;
- Well educated workforce (90% of population have received a secondary education);
- An active quarrying industry focused mainly on limestone and marble for roading aggregates and agricultural purposes;
- Ready supply of earthmoving equipment that supports the quarrying industry;
- A local labour source with mining experience gained from the quarrying industry and past gold mining activity.

5.5. Physiography

The Bau area has a striking physiography. Karst limestone blocks rise up to 350 m above a peripheral peneplain lowland of sediment of between 20 m to 50 m above sea level.

Much of the area is covered by severely modified tropical rain forests, with sporadic Kampung (village) style residential developments.

Numerous tributaries of the right hand branch of the Sarawak River dissect the region, which is generally, a slow flowing meandering river system especially toward the coast and prone to flash flooding during frequent rain storm events in the wet season (December-March).

6. History

6.1. General History

The occurrence of gold on the island of Borneo was known in China as early as the 4th Century, and gold was reported to have been exported from Bau from the 12th Century. Archaeological sites have been found near Kuching, and gold mining activities have been reported from the Indonesian southern extension of the Bau District from as early as 1760 (Van Bemmelen, 1949).

Mining in the Bau District dates from the 1820s, when Chinese prospectors were reportedly exploiting antimony ores and later, gold ores. Historical recorded gold production from the Bau area is 1.46 million ounces (Schuh, 1993). However, the true figure is thought to be in the vicinity of 3-4 million ounces when production prior to 1898, unreported production and recent production by Gladioli Group in the mid to late 1990's, is taken into account.

The district was also mined for antimony historically, mainly during the 18th Century. Total antimony production was 83,000 tonnes (Wolfenden, 1965; Hon, 1981). Mercury was mined also from 1868 through to 1909, and then again during the Japanese occupation from 1942-1945. Total historic production of mercury from the Bau District was 1,110 tonnes or 32,300 flasks (Roe, 1949).

6.1.1. Borneo Company

In the late 19th Century, the British owned, Borneo Company Ltd, established control of the mining operations in the district.

They introduced new metallurgical techniques, and claimed establishment of the world's first commercial cyanide treatment plant. By consolidating the various mines on the goldfield, the Borneo Company was able to maintain production until 1921 and produced approximately 980,000 oz of gold, mainly from the Tai Parit mine, close to Bau Township and the Bidi area approximately three (3) kilometres SW of the town.

6.1.2. Bukit Young Goldmine

In the late 1970's a prominent local family (the Ling family) consolidated the tenements into a holding covering most of the prospective ground in the Bau Goldfield. This resulted in the re-opening of the Tai Parit mine and the construction of a modern CIL plant at Bau.

Reported production from Tai Parit is 700,000 oz of gold, which included approximately 213,000 oz produced by Bukit Young Goldmine Sdn Bhd ("BYG") between 1991 and 1997. BYG is a member of the Gladioli Group.

6.1.3. Renison Goldfields Consolidated Ltd.

A joint venture was formed between BYG and Renison Goldfields (RGC) of Australia in 1985. RGC conducted regional work around Bau as well as drilling a number of deep diamond drillholes in and around the Tai Parit mine and some of the central intrusive contacts. Due to a policy change within the company, RGC withdrew from all of its offshore projects in 1993, at which time Menzies Gold NL (Australia) secured a joint venture with the Ling family.

6.1.4. Minsarco

Minsarco, the Australian subsidiary of the South African mining house GENCOR, carried out a feasibility study at Jugan in 1994. The study was based on the BIOX treatment process, a technology developed by GENCOR for the processing of refractory ore. Resource estimates were prepared by Resource Services Group ("RSG") of Western Australia. Minsarco concluded that "the operation would be moderately positive" but elected not to proceed. Menzies Gold NL was invited to replace Minsarco as they were already involved in the "Bau 1 Joint Venture" with BYG. Menzies eventually joint ventured into the Jugan deposit as part of the "Bau 2 Agreement" in 1996, on the basis that the resource could be treated at a central processing facility, possibly at Bau.

6.1.5. Menzies Gold NL

In 1993, Menzies through its Malaysian subsidiary BYGS entered into a farm-in agreement with Gladioli. The agreement, known as the "Bau 1 Agreement", gave BYGS the right to earn a 55% interest in certain exploration and mining tenements within the Bau Goldfield, covering an area of around 1,000 km^2. Tenements excluded from that agreement covered properties being exploited by Gladioli, and the Jugan deposit. In 1996, BYGS entered into a second agreement with Gladioli, the "Bau 2 Agreement", whereby BYGS acquired a 55% interest in all tenements held by Gladioli. This agreement required BYGS to deliver a bankable feasibility study by June 1999, subject to certain conditions. During this period Menzies was part funded through an exploration and development agreement with Cameco Gold from Canada.

In 1996, Menzies initiated a feasibility study based on four (4) deposits at Bau, namely Jugan, Pejiru, Kapor and Bekajang. The study was based on a treatment complex involving a concentrator, a BIOX leach plant and conventional CIP gold recovery.

Resource models for the advanced deposits, Jugan and Pejiru, were prepared for Menzies, and the subsequent resource estimates for Jugan were reported as significantly lower than the 1994 estimates. As a result, Menzies decided that the size and grade of the known resources would not support an economic operation with the then prevailing gold price (< $US300/oz, late 1997).

Menzies continued with an extensive exploration programme throughout the field of largely shallow RC drilling, but withdrew by 2001.

6.1.6. Zedex Minerals

Zedex entered into an earn-in agreement with Malaysian Mining Group, Gladioli Enterprises Sdn Bhd in November, 2006. Terms of this joint venture are outlined *in Section 3.4 Joint Venture Agreement with Gladioli Enterprises Sdn Bhd.*

Since commencement of the joint venture, NBG has conducted the following exploration programme:

- Geological mapping, surface sampling, drilling and resource modelling to validate and extend the inherited geological database, and formally define resources to JORC status within three near-surface deposits (Jugan, Pejiru and Sirenggok). An estimate of gold contained within historic mine tailings at the BYG Gold Mine site was also undertaken. These deposits cumulatively had a JORC status resource estimate of 1.612 Moz gold status;
- The first stage of a metallurgical programme was carried out by OMC (a subsidiary of Lycopodium Ltd of Western Australia) in order to identify the metallurgical test-work needed to specify the most cost-effective gold recovery process route and conceptual mining studies commenced;
- NBG exploration (geophysical modelling, geological mapping, surface and underground sampling and drilling) was conducted to define additional resources targets within the Central part of the Bau Goldfield (Tenement Block A). These results were reviewed and geological potential for a further, 3.3 – 4.5 Moz gold was identified in additional areas and extensions to known resources;
- Regional exploration (of tenement Blocks B and C) mainly consisted of a review of prior exploration, with some limited field work that confirmed the exploration potential of these blocks near the border with Indonesia.

The key events in the chronology of gold mining history in Bau, Sarawak are summarised below in *Table 16: Gold Mining History of Bau, Sarawak - Chronology of Events.*

Year/Period	Event
Early 19th Century	Elluvial and alluvial gold at Pangkalan Tebang and Bau itself was panned and sluiced by the Chinese miners from Sambas, Indonesia.
1870's	The Chinese miners started to process gold ore by hand crushing and panning in and around Bau area.
1882 – 1884	Borneo Company Limited assisted the local miners with mechanical equipment and a stamping mill to crush gold ore at Bau. Later, the company rented out pumping engines to the Chinese miners to work elluvial gold below water tables.
1896	The Borneo Company introduced the cyanide leaching method of gold extraction after successful experiments.
1898	The Borneo Company went into the gold mining industry actively, producing significant amount of gold at the Tai Parit Gold Mine using cyanidation process together with stamping battery and crushing mills.

Year/Period	Event
1900	Bidi Gold Mine, the second biggest gold mine of the Borneo Company commenced operation.
1910	Tai Ton elluvial gold ore was discovered and transported to Bau mill by rail for processing.
1911	Bidi Plant was closed down.
1912	Tai Parit (Tasik Biru- Bau Lake) became a large opencast mining pit.
1920	The Tai Parit Mine reached a depth of 200 feet at one point.
1921	Tai Parit gold mine was flooded and abandoned by The Borneo Company Limited.
1921 – 1941	Enterprising Chinese miners adopted cyanidation methods to work on small deposits and reworked on the tailings.
1942 – 1945	There was no gold production during the Second World War.
1946 – 1984	Small scale mining of gold by local miners with an average production of about 2,500 troy ounces annually.
1983 – 1985	The revival of the gold-mining industry in Bau. The restructured Bukit Young Goldmine Sdn. Bhd. consolidated small gold mines and set up a centralized heap-leaching plant in 1983 which became operational in 1984.
1986 – 1990	Bukit Young Goldmine Sdn. Bhd. introduced the carbon-in-leach method in 1986.
June 1990	Old tailings mostly worked out and development to re-mine Tai Parit (Tasik Biru - Bau Lake) commenced, including dewatering one million tons of water from the lake.
1991	Detailed exploration drilling by Bukit Young Goldmine Sdn. Bhd. and subsequent re-mining of Tai Parit proved that the report of the mine manager of Borneo Company Ltd. in 1921 that the end of ore reserve was in sight was not true. Bukit Young Goldmine Sdn. Bhd. introduced state-of-the-art computerised Pressure Zadra (Fastrip System) gold elution and electro-winning circuits.
August 1992	Bukit Young Goldmine Sdn. Bhd. introduced further progressive state-of-the-art gold processing technology to Bau; started the construction of Carbon-In-Pulp (CIP) circuit, including the use of 750 hp motor driven ball mill.
1993	A record of gold production - 1,547.33 kilograms valued at RM 46,350,200.00 (US$ 17,660,745) based on the 1993 gold price.
1997	Bukit Young Goldmine Sdn Bhd stop mining and processing at Bau

Table 6-1: Gold Mining History of Bau, Sarawak - Chronology of Events

6.2. Project History

6.2.1. 2010 Work

Resource drilling focused largely on the Taiton A Zone (Taiton Sector) and Tabai Zone (Taiton Sector). During the second half of 2010, exploration drilling of new geological and geophysical targets commenced. A separate resource drilling program, aimed at upgrading the bulk of the existing resource to Measured and Indicated categories and to test deeper and lateral extensions of mineralization also commenced around the same time.

Upgraded sample processing facilities and an on-site assay laboratory were put into place enabling processing times for samples of 48 to 72 hours.

The deposits, prospects and sectors are shown in *Figure 6-1 - 3D View of the Bau Project – Showing the Sectors, Deposits & Prospects* below.

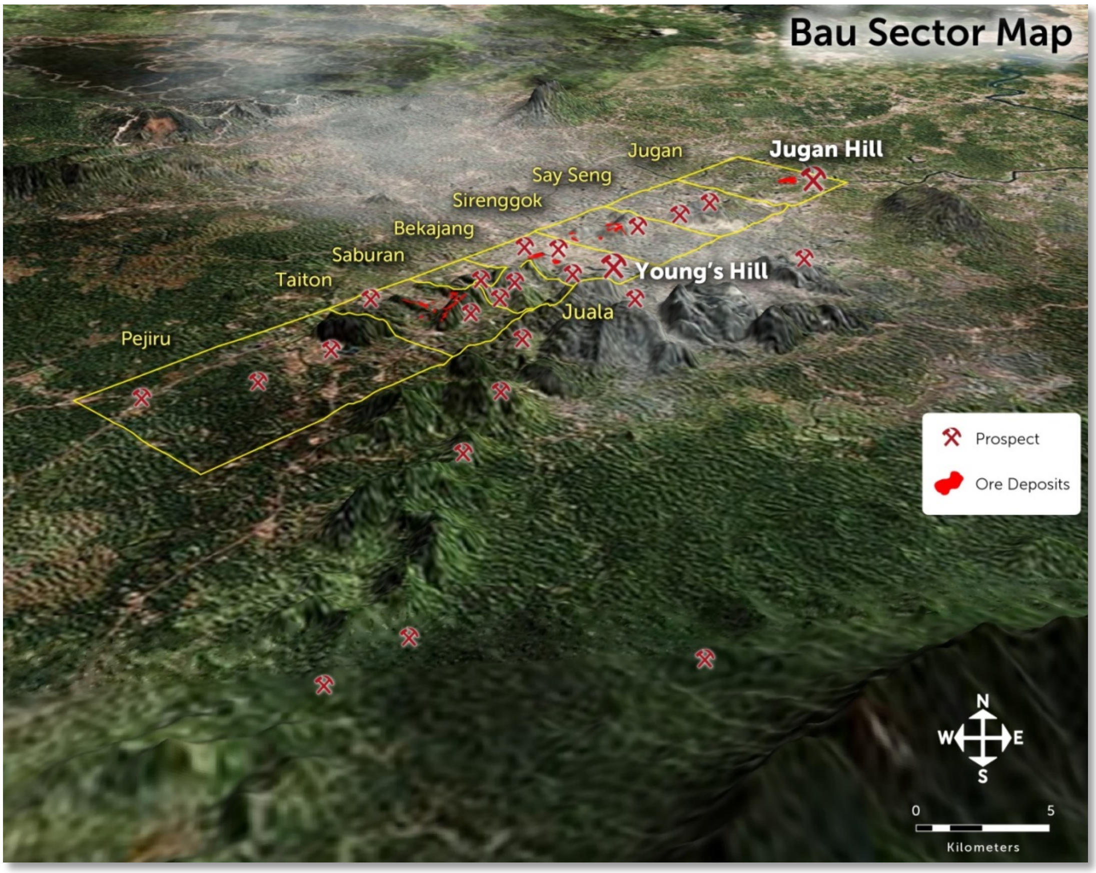


Figure 6-1 - 3D View of the Bau Project – Showing the Sectors, Deposits & Prospects

6.2.2. 2011 Work

Geological mapping and geophysical modeling continued as did the drilling program that was commenced during the second half of 2010.

The focus of the drilling program was the completion of the Taiton Sector drilling, drilling the Bukit Young Pit Zone (Bekajang Sector) and Jugan Hill (Jugan Sector), the immediate focus for upgrading resources and the feasibility study. Additional exploration drilling was undertaken to follow up on geological and geophysical targets in these areas as well.

In addition to drilling at Jugan Hill, the Company also completed 673 metres of trenching in 7 trenches, which confirmed a higher grade zone at south-western end, north-eastern end and locally extended the mineralization boundary. In addition, two metallurgical drill holes were completed and >400kg of sample was sent to SGS Lakefield in Perth and Core Resources in Brisbane for metallurgical test-work. A deep NE trending, steeply NW dipping (fault-bounded) structure has been interpreted as the main mineralization control or feeder structure.

The total drilling program culminated in an updated resource estimate that was announced during March 2012. The update indicated an approximate 23% gold resource increase at Bau to 913,500 oz Au in the Measured and Indicated categories and 2,107,000 oz Au in the Inferred category. Included in the resource estimate is an upgrade of 349,600 oz of resources from Inferred to Measured and Indicated resource categories. This increase and upgrade derive from the drilling of 19,817 metres in 122 drillholes during 2011.

During the third quarter of 2011, the Company moved the Bau project into feasibility phase with the objective of achieving a favorable development decision targeting stage one production of at least 100,000 oz of gold per annum by 2015. Exploration, mining feasibility, and environmental studies were planned to further expand the resource base, determine the best development route and examine the issues involved in developing multiple deposits in an optimal manner. Key development objectives include upgrade of the resource categories for a mining feasibility study. Key exploration objectives are to geologically, geophysically and geochemically define new targets for drilling in order of assessed priority.

6.2.3. Work for the Six-Month Transition Year Ending June 30, 2012

The Bau feasibility study is ongoing, initially focused on the Jugan deposit, which is the first of several Bau Central deposits listed for development. A feasibility study of resources within the Bekajang Sector is also in progress for planned development. Other sectors spread along the 17 km long Bau Central gold trend and are at varying stages of exploration and feasibility assessment. Ongoing drilling and related exploration activities during 2012 are expected to continue to expand and upgrade these for subsequent mining feasibility assessment. A further resource update is scheduled for the fourth quarter of 2012.

Activity during the 2012 financial year ended June 30, 2012 focused on Jugan Hill in connection with the feasibility study. 8,812 metres of resource drilling in 33 drillholes were completed as well as 502.8 metres of metallurgical drilling in 6 holes. Ongoing resource (in-fill and step-out) drilling is continuing to broaden the Jugan deposit boundaries and upgrade the resource categories.

The Jugan/Carlin comparison continues to be considered valid (similar host rock geological setting, mineralization age and pathfinder geochemistry) (eg: Arsenic, Mercury, Barium, Tungsten, Thallium).

A definitive C-horizon soil geochemical survey is now in progress. This comprises a close-spaced (25 metres) grid survey over the Jugan Hill deposit and immediate surrounds, coupled with extensive (sector-wide) peripheral C-horizon ridge and spur sampling at 50 metre sample interval. Comprehensive ICP multi-element assay (26 elements) and HyChip sampling will be conducted over both phases.

Evaluation work also continued at Pejiru where preliminary remote sensing analysis identified prospective target areas, which require further field work prior to drill target specification.

6.2.4. Work for Fiscal Year Ending June 30, 2013

In the second half of 2012, a 76-hole drill program totaling 17,395.4 metres at Jugan Hill delivered a 42% resource increase from 659,100 to 870,500 ounces Measured and Indicated and from 16,300 to 89,800 Inferred. This was a 9.4% overall increase at the Bau Goldfield for 1,124,900 ounces Measured and Indicated, and 2,181,600 ounces Inferred.

An IP survey was conducted during April & May 2013, with post-processing work being conducted in Perth. Thereafter, analysis was conducted by our geophysical consultant who identified ten (10) IP anomalies. These anomalies along with those identified in the ridge and spur soil sampling, were followed up by grid soil sampling and geological mapping (ground-truthing).

Feasibility for the Jugan Hill deposit, with metallurgy and process having been conceptually resolved and peer reviewed by independent engineering, procurement, and construction management. Besra is targeting public release of the results of the feasibility study in the quarter ending December 31, 2013.

7. Geological Setting & Mineralisation

7.1. Project/Deposit Geology

7.1.1. Stratigraphy

The exposed rocks in the Bau district are dominated by a sequence of late Jurassic to early Cretaceous aged marine sediments. These comprise a lower limestone formation, the Bau Limestone, estimated to be 500m thick that is unconformably overlain by a 1,500m thick flysch sequence, known as the Pedawan Formation. The Pedawan Formation is dominated by shale but more arenaceous and conglomeratic units are reasonably widespread through the sequence.

Figure 7-1: Generalised Stratigraphy of the Bau District (after Schuh, 1993) below diagrammatically depicts the stratigraphic relationships for rocks of the Bau District.

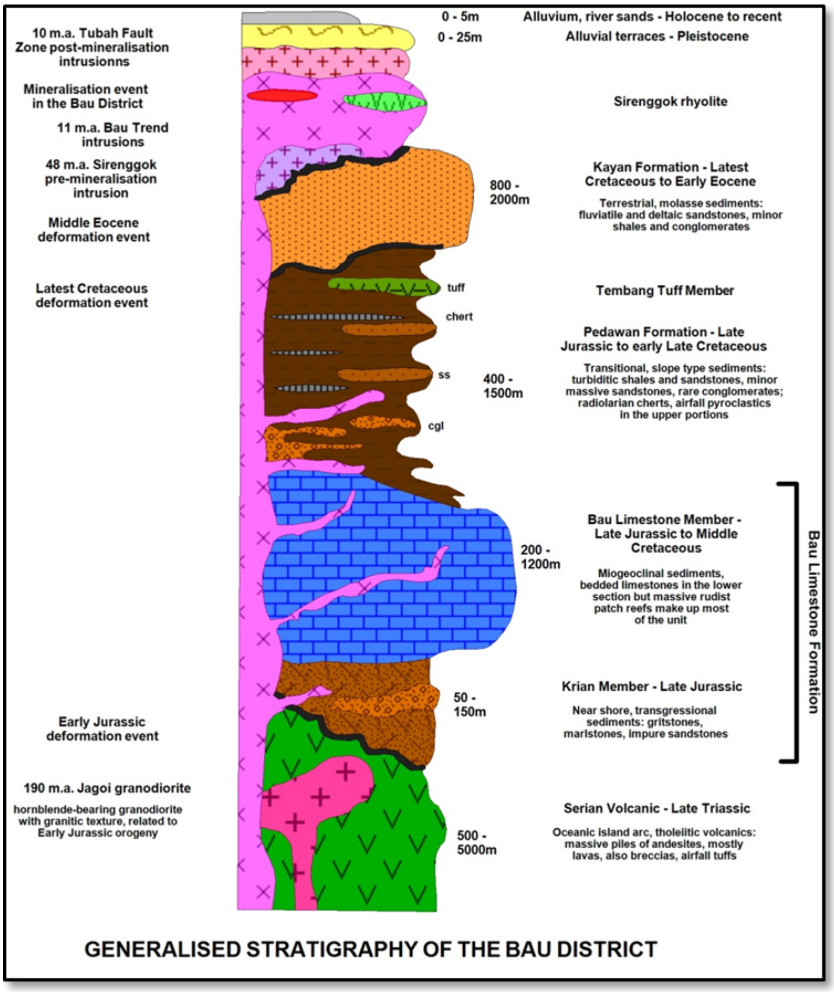


Figure 7-1: Generalised Stratigraphy of the Bau District (after Schuh, 1993)

The oldest rocks known in the Bau Goldfield are the Triassic-aged Serian andesitic volcanics. These do not crop out but have been intersected in drill holes at Bau, beneath the Bau limestone. An intrusive known as the Jagoi Granodiorite is thought to be co-eval with the Serian Volcanics and it crops out 15 km SW of Bau on the Indonesian border.

The Bau Limestone has a lowermost ~100 metre thick arenaceous unit, (the Krian Member), which also contains basal conglomerate beds. The Krian sandstones rest unconformably on the Serian Volcanics. The principle rock types and structures of the Bau Goldfield are shown in *Figure 7-2: Generalised Geology of the Bau Goldfield* below.

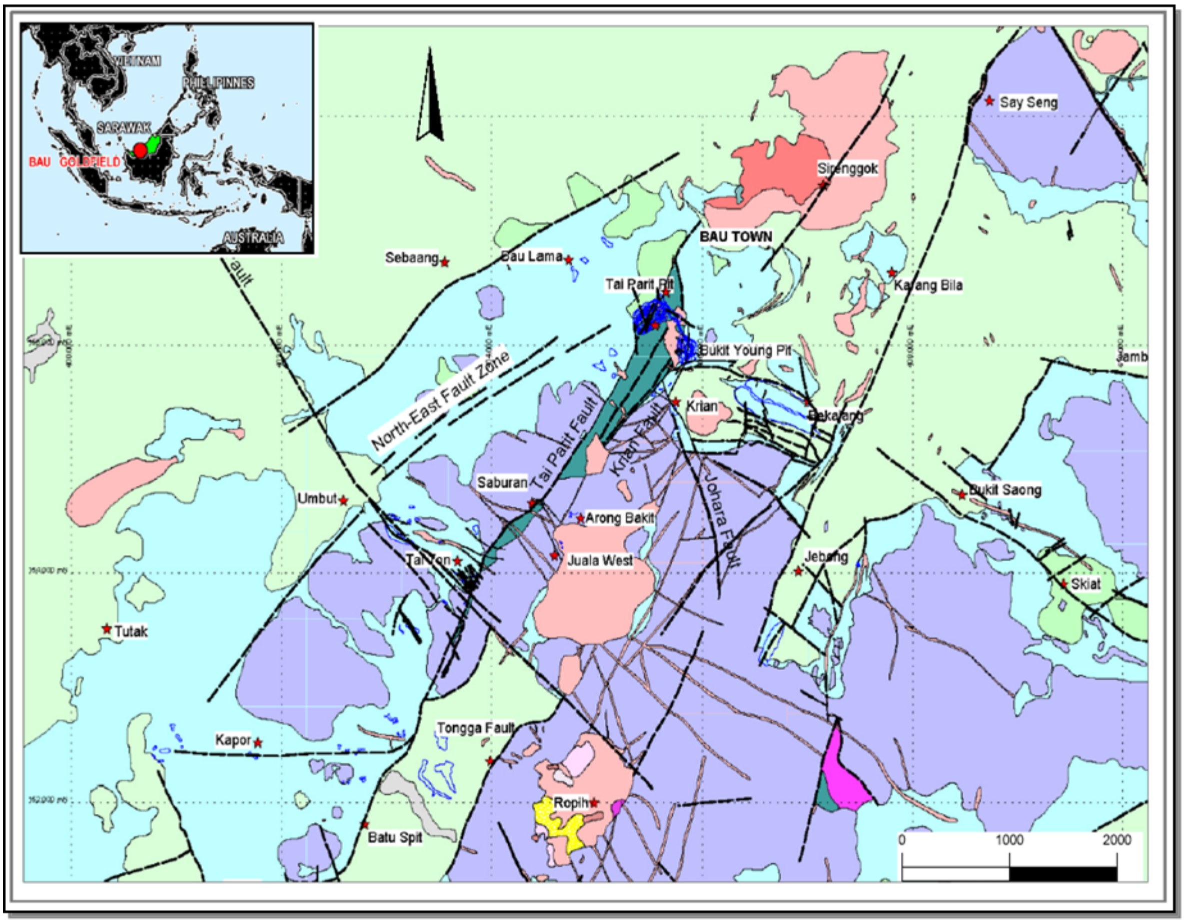


Figure 7-2: Generalised Geology of the Bau Goldfield

A striking feature of the Bau District is a series of uplifted horst blocks of Bau Limestone juxtaposing the generally stratigraphically higher Pedawan formation. Throws on the NNE and SE trending controlling graben faults are in the order of 300m. Surrounding the horsts of limestone is a peneplane of Bau limestone with typical karst features and the overlying Pedawan formation.

The Pedawan Formation and Bau Limestone represent fore-arc shelf and slope deposits developed to the north of a Cretaceous magmatic arc, remnants of the arc are preserved as a granite belt in the Schwaner Ranges in Central Kalimantan.

7.1.2. Tertiary Aged Intrusives

Miocene age sub-volcanic intrusives of acid-intermediate composition (predominantly granodiorite porphyry, micro-granodiorite and dacite); intrude the Jurassic-Cretaceous sediments at Bau. They form a narrow belt of small stocks (generally <2 km^2 in area), and associated dykes and sills trending NNE-SSW from the Indonesian border through the central Bau Goldfield (the Bau Trend).

The current level of exposure of the intrusives appears to be high-level, geophysical surveys indicate larger masses and unexposed bodies occur at shallow depth. Drilling has shown that at least one intrusive body near Bau (Seringgok porphyry), has the form of an upward-flaring funnel and therefore may be an endogenous dome.

The Bau Trend is correlated with a Late Oligocene (41my) to Late Miocene (8my) intrusive belt which occurs immediately south of the Sarawak border and extends across the entire width of Borneo (Sintang intrusives). This belt is thought to have formed due to a prolonged episode of crustal extension and increased thermal gradient across the Borneo microcontinent in the early to mid-Tertiary.

The age of the Sintang Volcanic Suite intrusives gradually decreases from West to East (Moss et al, 1998), which suggests extension and crustal thinning may have originated in the West and progressed in an Easterly direction.

The NNE-SSW Bau Trend of Miocene intrusives is readily apparent in *Figure 7-3: Filtered Aero-Magnetic Plot (analytic signal derivative) over the Bau Goldfield*.

Virtually all the magnetic features (yellow to red), are intrusive stocks, either outcropping or at shallow depth under cover.

Note, that the Bau Trend terminates to the North close to a separate belt of andesite plugs within an ENE-WSW striking fault.

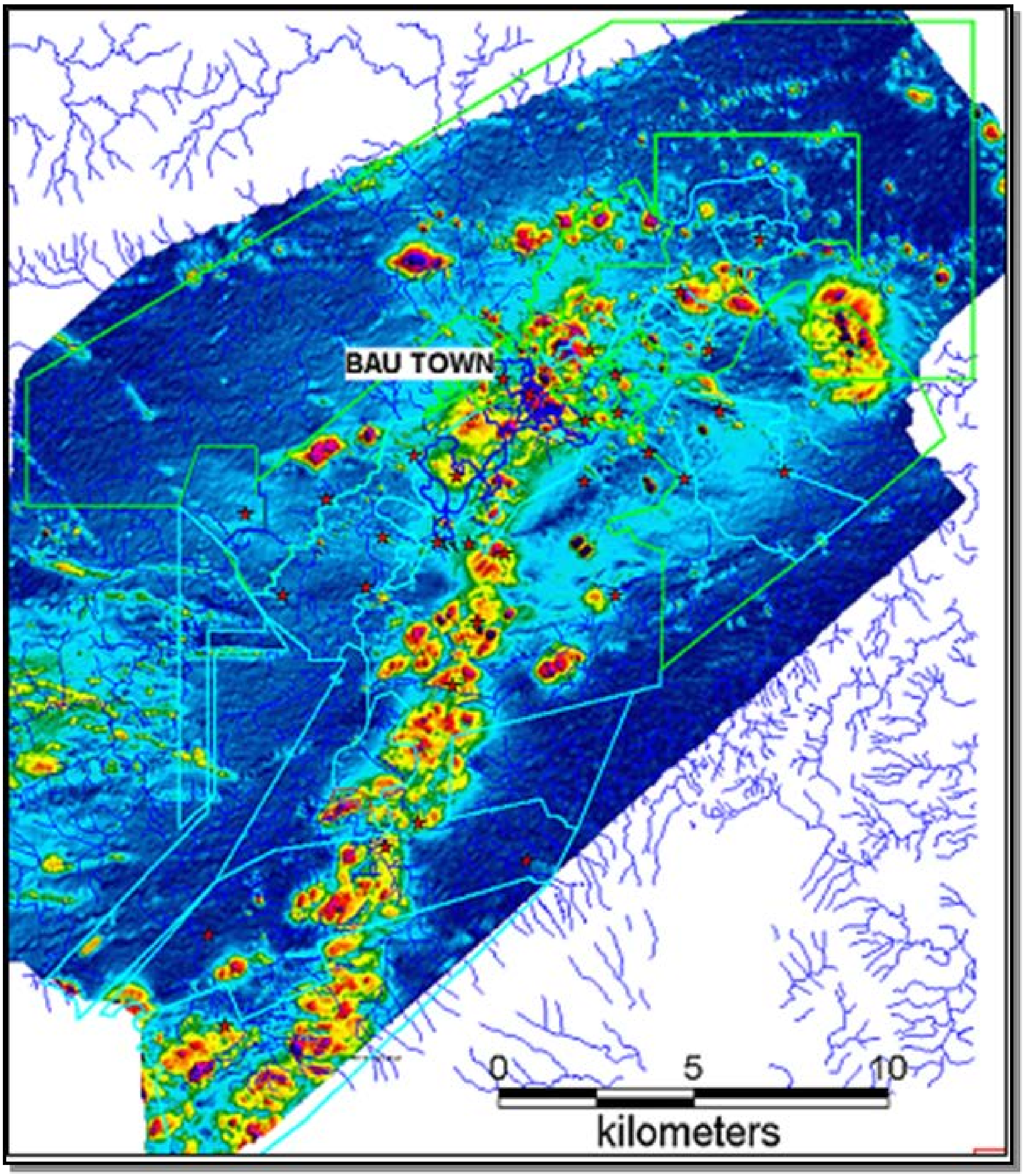


Figure 7-3: Filtered Aero-Magnetic Plot (analytic signal derivative) over the Bau Goldfield

7.1.3. Structural Settings

7.1.3.1. Regional Deformation Trends

Four (4) deformation/folding events have been recognised in West Sarawak (*Majoribanks 1989 and Schuh, 1993*), and appear to be evident at Bau (*Bobis et al, 1992*). *Figure 7-4: Mapped Faults, Deformation Events Overlain on Digital Elevation Model of Bau* shows the main mapped faults in the Bau District and the assignment of them to the major deformation events.

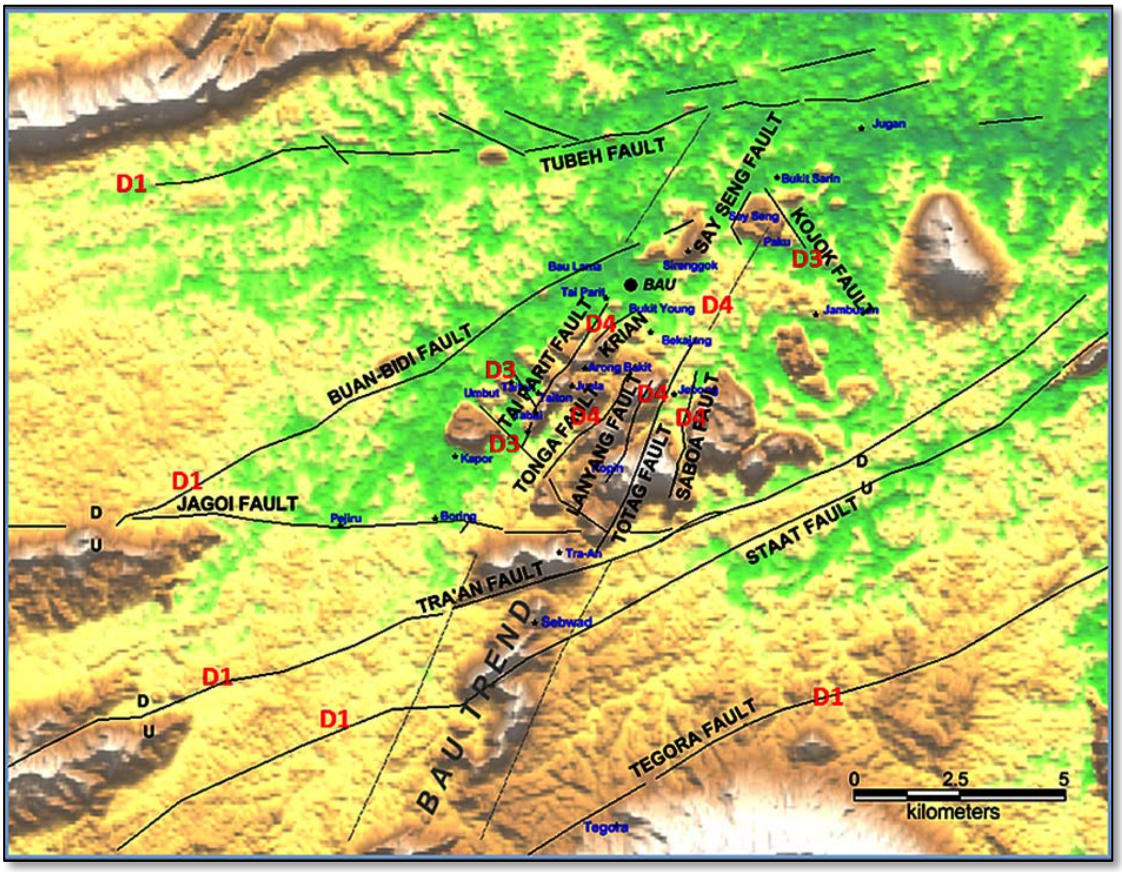


Figure 7-4: Mapped Faults, Deformation Events Overlain on Digital Elevation Model of Bau

These can be summarized as follows (*Banks, 2011*):

- ***D1, Late – Triassic - Early Jurassic:***

An early event that produced the ENE Jagoi Trend. NNW-SSE directed extension produced deep-seated ENE trending major faults which were active prior to the Early Jurassic and controlled the ascent of the Gunung Jagoi & Kisam intrusions. These granodiorite plutons are coeval with the Serian Volcanics. D1 faults include: Tubeh, Buan-Bidi, Tra'an, Staat and Tegora Faults.

- ***D2, Late Cretaceous:***

Compression directed SW-NE deformation resulted in tight upright folding, accretion and obduction of Pedawan Formation turbidites due to Late Cretaceous subduction. D2 structures include low angle thrusting within the Pedawan Formation.

- ***D3, Mid-Eocene:***

W-E to NW-SE compression produced gentle folding of Bungoh Range and Kayan Basin molasse basins to the south and NW of Bau and the Bau and Ropih anticlines. The extensive orthogonal fracture pattern in the Bau Limestone was a response to this deformation event. These D3 fractures were later intruded in the mid-Miocene by dacite porphyry dikes and were significant conduits for hydrothermal solutions. D3 faults include the Taiton, Johara, Gumbang and Kojok Faults.

- ***D4, Mid-Miocene:***

Right lateral transcurrent to trans-tensional faulting along the 200km long NNE Bau Trend controlled the intrusion of mid-Miocene microgranodiorite to dacite porphyry intrusions in central Bau. Intersection of the larger D1 faults with the D4 Bau Trend structures has controlled the loci of the larger intrusive centres such as Juala, Ropih, Tra'an, etc. *Schuh, 1993* contends that farther south, Bau Trend intrusions are dominantly controlled by D3 faults. D4 structures include the Tai Parit, Krian, Say Seng, Tonga, Lanyang and Totag Faults among others.

These concepts have been reviewed in light of more recent interpretation of the Bau Anticline as a Pop-Up structure, (*Mustard, 2001*).

7.1.3.2. Bau Anticline

The Pedawan shales are up-arched along an ENE-WSW axis running through the Bau region, and in the central crest of the arch Bau limestone is exposed. This structure, known as the Bau Anticline consists of up lifted horsts of Bau limestone which form steep karst ranges to the SW and East of Bau Township. Fault-bounded, dropped blocks (roofed by Pedawan shale), also occur within the anticline. The anticline is up-domed to the maximum extent along steep faults over a 4km long corridor running SW of Bau. This corridor forms the central zone of the Bau Goldfield and is intensively intruded at shallow depth by granodioritic plutons, some of which are unexposed (inferred from geophysical evidence and mapped contact metamorphic haloes).

Along the faulted domal axis, the basal Krian unit of the Bau limestone is exposed between the Tai Parit and Krian faults, while the older Serian volcanic formation is known at shallow depths from drilling.

7.1.3.3. Bau Trend

The Bau Trend line of acid-intermediate felsic intrusives that occur through the Bau Goldfield, are believed to be localised by a major NNE striking deep structural zone that was under tension during the Miocene. In Kalimantan the main Tertiary basins are elongate WNW parallel to the Lupar Line (a major fault system running through Western Borneo to Vietnam and regarded as a former subduction zone). The basins are intruded by the Sintang granitoids – correlatives of the Bau intrusives. Therefore, the Bau NNE intrusive trend may reflect an old basement transfer structure that was reactivated during extension and development of the Tertiary Basins.

7.1.3.4. Faulting

The main fault directions in the district are North-Northeast and Northwest. Vertical displacement of at least 300m has been determined for the North-Northeast striking structures (from drilling at Tai Parit). The block faulting has elevated horsts of limestone to form prominent scarp-bounded ridges, and dropped blocks of shale into the limestone. The sediment blocks are typically gently dipping but locally can be severely disrupted. Dissolution of the top of the limestones by acid groundwater (and possibly acidic hydrothermal fluids), has produced a karstic surface. This process has led to the development of collapse breccias at the limestone-shale contact.

7.1.3.5. Structural Model

From the mapped distribution of geological units and local structures that the central uplift within the Bau Anticline is better described as a block-faulted dome structure.

Pop-up structures can be described as strike-slip bounded pull-apart basins in reverse, as shown in *Figure 7-5: Strike-Slip Geometries at Bau*. In both cases the central area of vertical deformation occurs as a rhombic, or lozenge shaped block between sub-parallel strike-slip faults. In pull-apart basins the central block is under tensional strain due to the orientation of pre-existing tangential cross-faults being under tension, in the case of pop-ups, the tangential cross-faults are under compression.

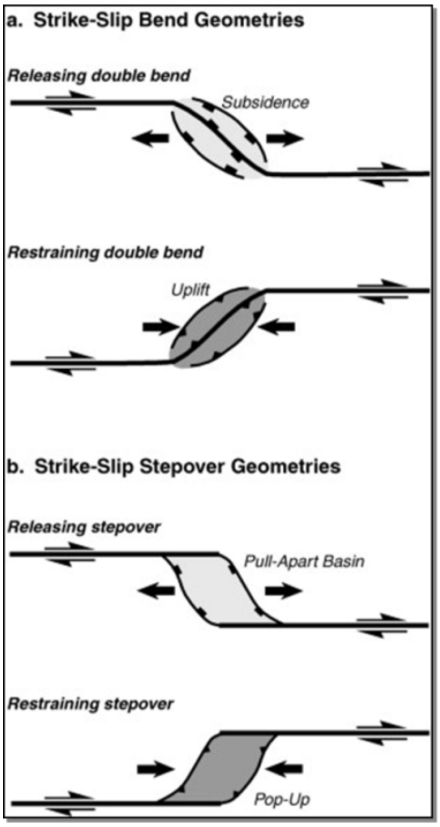


Figure 7-5: Strike-Slip Geometries at Bau

The central uplift block at Bau is an 8kms by 12 kms rhombic shaped block of limestone (pop-up) that lies at the intersection of the NE trending Bau Anticline and a NNE striking zone of intrusives and is located between two (2) major ENE striking structures, the Tubah and Staat Fault Zones.

McClay and Bonora, 2001 carried out experimental studies on the development of "pop-up" structures using sand box experimentation as depicted in *Figure 7-6: Surface Model Photograph (a), Upper Surface Model Structure Contours (b) & 3D Structure Model (c)*. The model bears a remarkable similarity to the morphology of the structure in the Bau Goldfield.

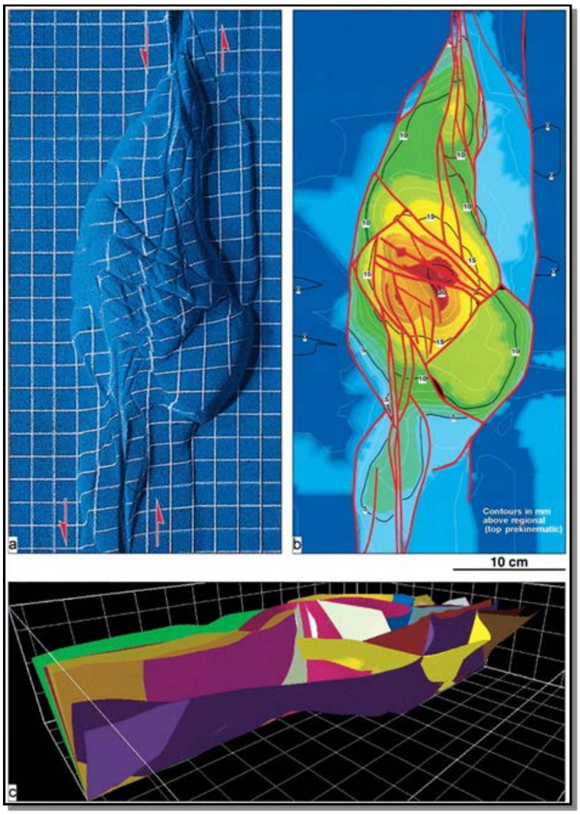


Figure 7-6: Surface Model Photograph (a), Upper Surface Model Structure Contours (b) & 3D Structure Model (c)

In McClay's experimental models and type examples the main through going faults lie sub-parallel (usually within 10° to 20°) to the strike of the principle displacement zone. This suggests that the main strike-slip fault direction in Bau is sub-parallel to the NNE striking structures and is probably the structure controlling the emplacement of the NNE striking line of intrusives that extends South of and passes through Bau. McClay's studies showed that the width and angle of the stepover along the strike-slip fault controls the shape of the pop-up. The model that most closely resembles the sigmoidal shape of the pop-up in Bau is one where the stepover is at an angle of 150° to the main strike-slip fault direction.

The Bau pop-up has been 'diced up' by NNE and NW striking faults into rectangular prisms. Many of the faults, particularly the NW striking faults, host intrusive dykes.

The highest point topographically and the greatest amount of vertical exposure of limestone at Bau, lies in the centre of the pop-up (Mt. Kawa). From the centre to the margins of the pop-up there is a gradual but stepped decrease in topography and relative uplift of limestone. The rhombic shape of the pop-up, stepped topography and fault controlled margins suggest faulting and not folding was the primary cause of uplift. The NNE striking faults are the dominant faults within the pop-up and appear to have had the most influence on the shapes of the limestone blocks and their uplift. The limestone cliff faces around the edge of the pop-up represent faults along which the limestone has been uplifted and may mark the edge to the pop-up.

The sets of NNE and NW striking faults are known to extend from the limestone into the surrounding shale (*Mustard, 2001*), but are difficult to identify because of the ductile style of deformation in the shale compared to the brittle open structures filled by dykes and sills in the limestone.

Some of the features unique to pop-up structures are:

- Curvilinear faults bound the area of doming or uplift;
- Dip of the bounding faults change along strike;
- In a plan view, the overall shape of the pop-up area is a lozenge or rhombic shaped dome with doubly plunging anticlines;
- Centre of the pop-up or area of most relative uplift occurs at the step-over;
- From the centre to the outside of the pop-up, there is typically a gradual stepped decrease in relative uplift.

All these features can be observed at Bau and in other natural examples (*Mustard, 2001*).

7.2. Regional Geology & Structure

There are two (2) regional exploration projects that form part of the Besra Gold & Gladioli Enterprises Sdn Bhd Joint Venture. These are known as Block C and the Rawan Area (Gunong Rawan). They are located in the Western corner of the State of Sarawak, East Malaysia, and 25 km to 80 km south and Southeast of Kuching City and accessible by the Kuching-Serian road.

The geology of both Block C and the Rawan Area is broadly similar to that of the Bau Goldfield. Both areas contain volcanic rocks and sediments (including limestone) intruded by belts of Miocene-age felsic intrusives. Gold mineralisation has been known to occur in Block C area for many decades, however, in the Rawan area (characterised by six [6] major intrusive complexes running in an East-West line parallel to the Indonesian border), gold has only recently been discovered.

7.3. Deposit/Project Mineralisation

7.3.1. General

The geology of the main deposits modelled in the resource estimation is described as well as other significant prospects that are at the pre resource stage. These generally have significant results and/or have been tested by NBG since the commencement of the joint venture. Additional information on these and other prospects/occurrences are described in *Section 9 - Exploration*.

7.3.2. Deposit/Prospect Mineralisation

7.3.2.1. Sirenggok

The Sirenggok deposit lies approximately 1.5km NE of Bau Township. The current resource modelling has outlined an Inferred Resource of 8.346 million tonnes at 1.14 g/t Au for 307,000 ounces at a lower cut-off of 0.5g/t Au.

The gold-arsenic-antimony mineralisation is hosted by veins, vein stockworks and as disseminations within quartz-sericite to propyllitic altered quartz-feldspar micro-quartz diorite porphyry. A younger phase of xenolithic quartz diorite porphyry intrudes the earlier porphyry. See *Figure 7-7: Geological Plan of the Sirenggok Deposit* which shows the surface distribution of the main mineralised zones and *Figure 7-8: NE-SW Section through Sirenggok Deposit,* which shows a section view of the mineralisation. The host porphyry appears to be a funnel shaped composite body with concentric phase's younging inward that intruded through the Bau Limestone and Pedawan Formation and flattened out at higher elevation. There is a number of breccia phases recognised.

The currently defined resource is open along strike and at depth. The main trend appears to be NW-SE and steeply dipping to the NE. There are two other areas of mineralization picked up to the NE in surface samples and several drillholes and surface mineralization in the SW.

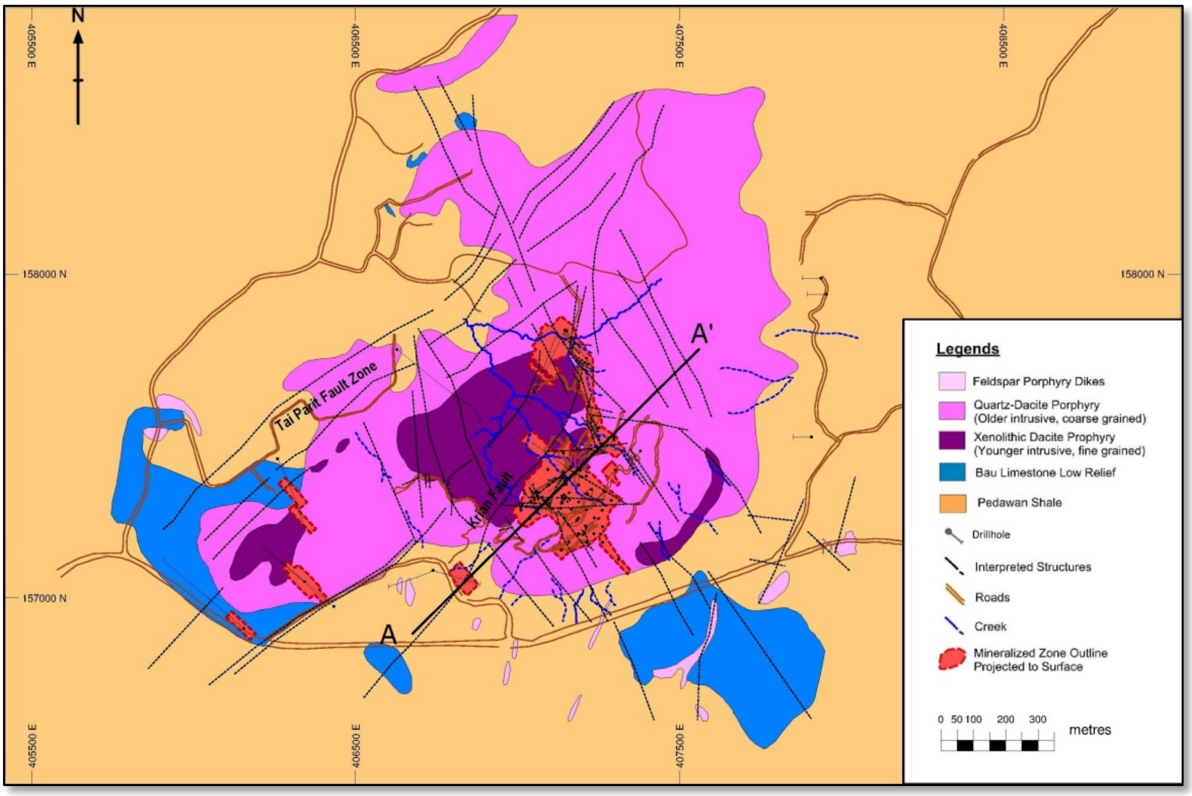


Figure 7-7: Geological Plan of the Sirenggok Deposit

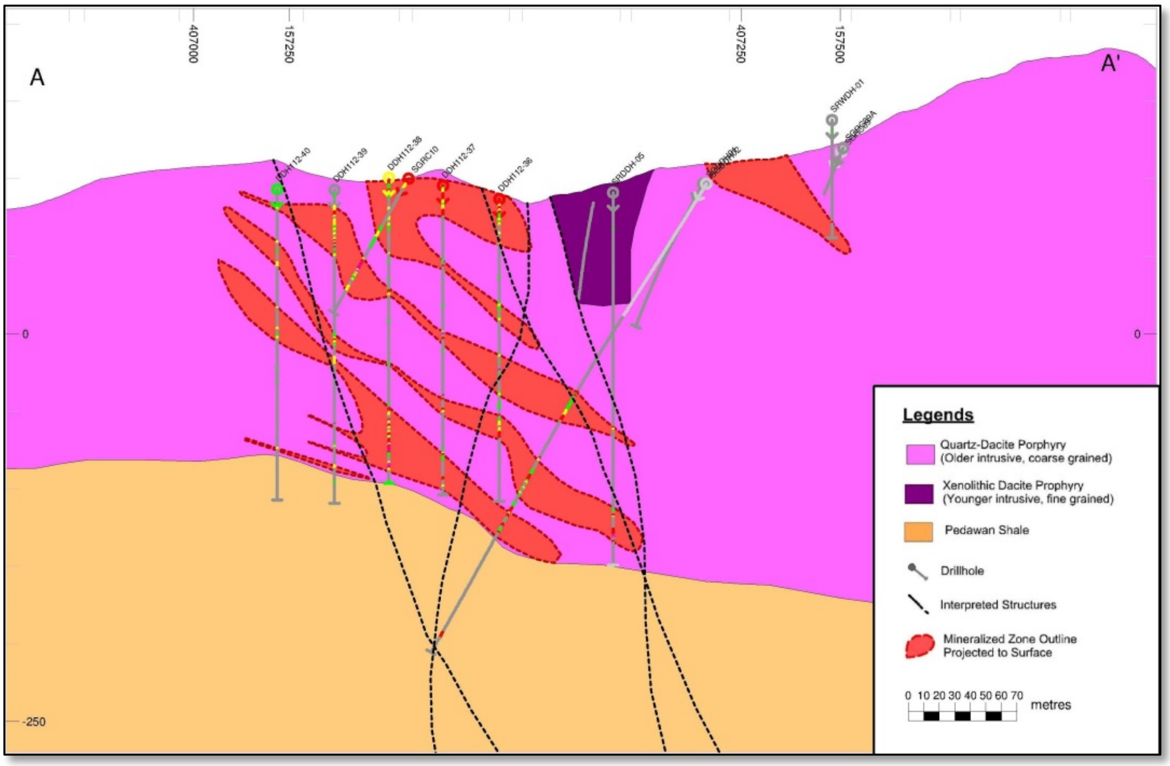


Figure 7-8: NE-SW Section through Sirenggok Deposit

7.3.2.2. Jugan

The Jugan Deposit is centred on Jugan Hill, approximately 7 km NE of Bau, within a kilometre of the Bau-Kuching Road. See *Figure 7-9: Surface Outline of Gold Mineralization at Jugan* below. The current measured and indicated resource modelled stands at 17.91 million tonnes at 1.51 g/t Au for 870,500 ounces gold. Inferred resources are 1.77 million tonnes at 1.57 g/t Au for 89,800 ounces gold.

The deposit is hosted within the Pedawan formation, predominantly in highly deformed and sheared carbonaceous shale, laminated shales, mudstones and interbeds of fine to medium grained sandstone. The shearing and fold axes are dominantly NE trending with the gold mineralisation forming within acicular arsenopyrite and arsenian pyrite disseminated throughout the sediments and within carbonate (ankeritic) veinlet stockworks.

Typically, the arsenopyrite content ranges between 1 % and 5 % and arsenian pyrite trace to 5 %. Overall sulphide content in the ore zone can be in the 5 % to 7 % range. Sulphide content and gold grade have a close correlation. The deposit has been drilled to approximaelty 350 m vertically without the limestone-shale contact being intersected. Several NW-trending dykes of post mineral microgranodiorite porphyry transect the ore zone and are invariably strongly hydrothermally altered.

The currently defined resource is largely constrained between hangingwall and footwall shears that strike NE-SW and dip between 55° and 75° NW. In addition a number of NW-SE trending shear zones have been identified some which appear to be post mineral although may have been developed prior to or during the mineralising event. There is an interpreted dextral sense of movement on these and opens the possiblility of offset extensions and repetitions of the deposit. A well developed NW-SE trending shear is interpreted to dip at approximately 70° to the NE and appears to cut off the ore body.

The drill programme has helped define a higher grade zone that plunges NE within the plane of the NW dipping ore body. This correlates with a slight increase in incipient silicification and sulphide content. Mineralisation remains open at depth and to the NE.

Jugan is the only known deposit to be hosted solely in the Pedawan formation.

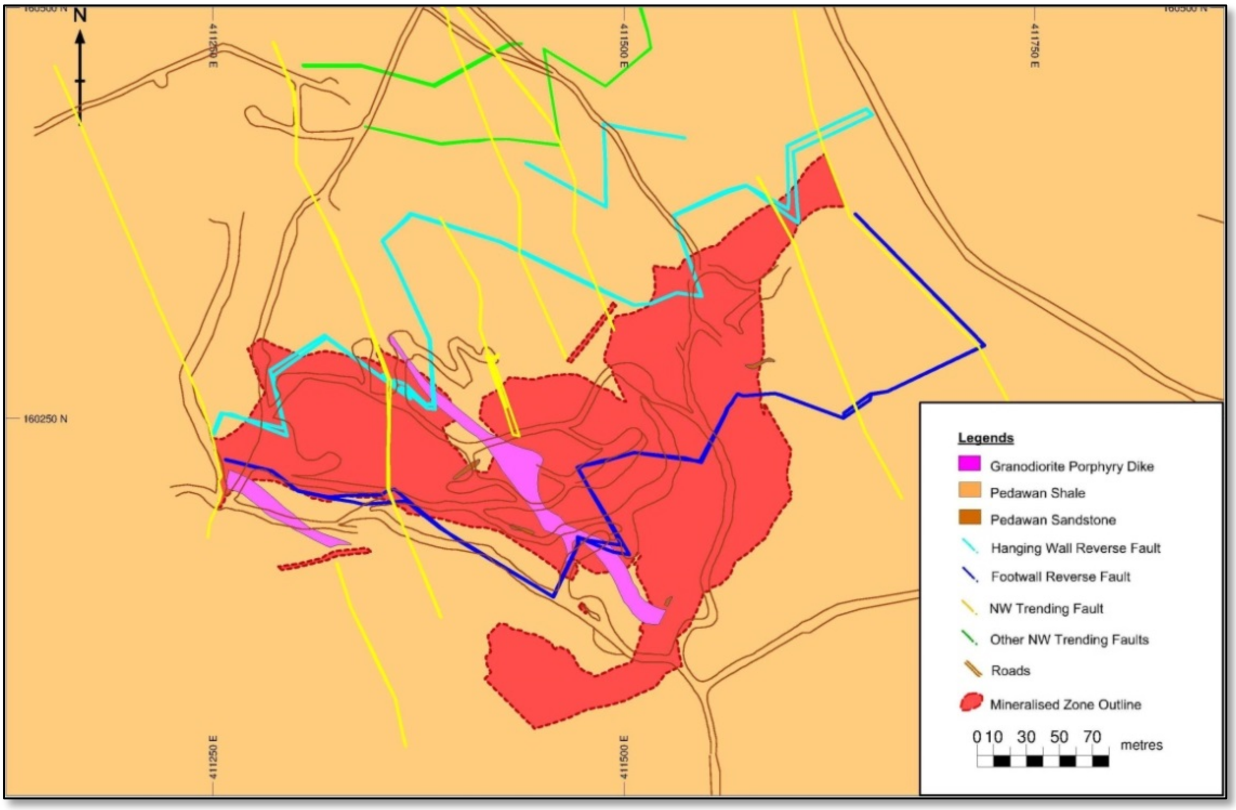


Figure 7-9: Surface Outline of Gold Mineralization at Jugan

7.3.2.3. Pejiru Sector

The Pejiru Sector between 5 km and 8 km SW of Bau, comprises four (4) deposits that have been modelled. These are the Pejiru-Bogag, Pejiru Extension, Kapor and Boring deposits. As a result of the current resource work these now have Inferred Resources of 25.8 million tonnes at a grade of 1.20 g/t Au for 997,800 ounces gold at a lower cut-off of 0.5 g/t Au. The full Pejiru resource outlines are shown in *Figure 7-10 - Surface Outline of Gold Mineralization at Pejiru* below.

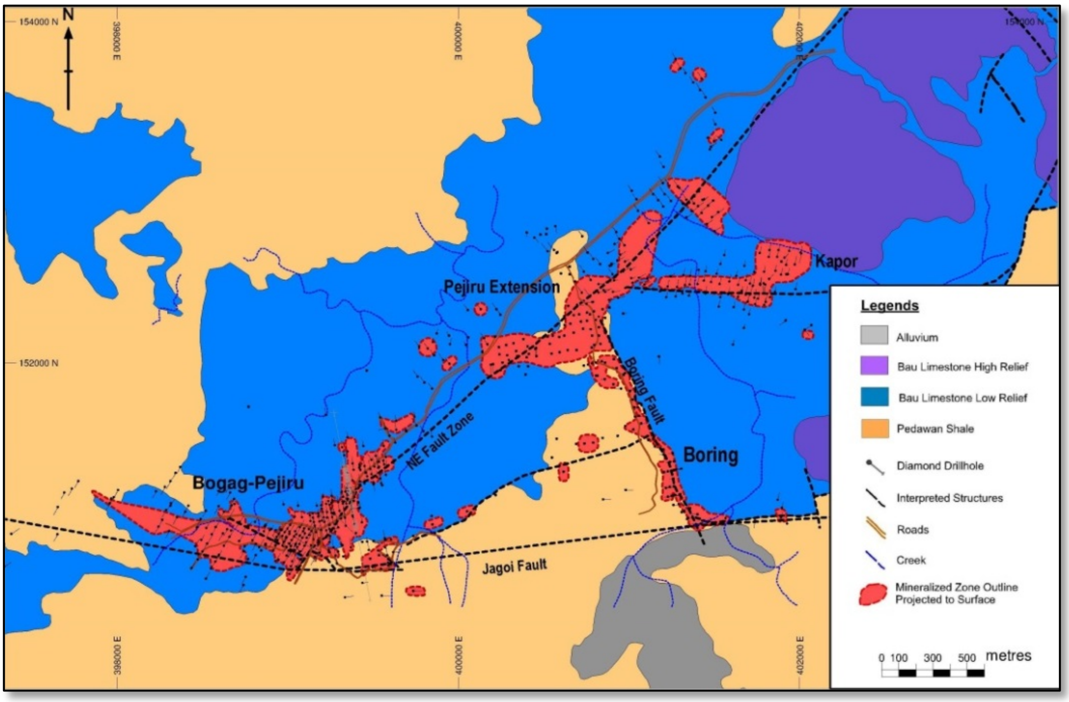


Figure 7-10 - Surface Outline of Gold Mineralization at Pejiru

7.3.2.3.1. Pejiru-Bogag & Pejiru Extension

The Pejiru-Bogag deposit has a main zone of mineralization that is essentially flat lying with a 1,500 m length, 50 m to 150 m wide and up to 80m thick, averaging 15 m to 20 m. It has a NE-SW trend and a NW-SE trend, see *Figure 7-11: Geological Plan of Pejiru Deposit Showing Surface Projection of Gold Mineralisation* giving a lobate V surface projection. Pejiru Extension lies to the NE and is essentially a continuation of the Pejiru-Bogag zone.

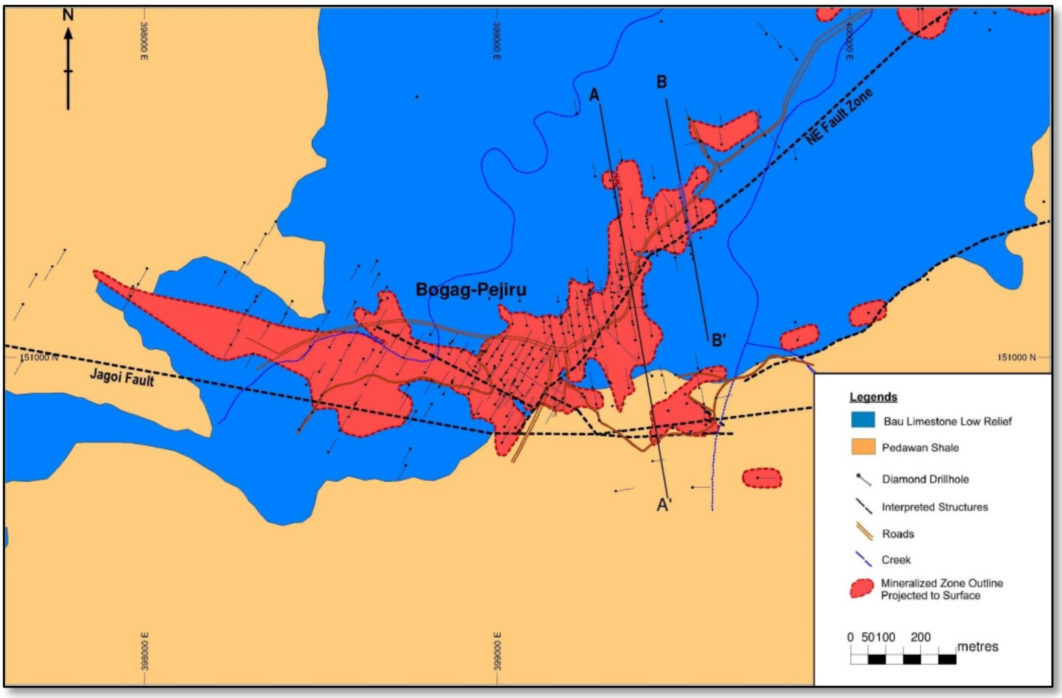


Figure 7-11: Geological Plan of Pejiru Deposit Showing Surface Projection of Gold Mineralisation

Previous workers have thought mineralisation controls relate to the so called Bau Anticlinal axis, however recent work by Besra has concluded it is more likely developed within a horst and graben style block faulting regime related to the Bau pop-up structural model. Mineralisation can occur at the limestone-shale contact but more commonly lies within limestone. The general outlines of the ore zones show that the most extensive mineralization occurs at the intersection of NW-SE and NE-SW structures.

The main ore zones lie at 20-30 m below surface being thickest at proposed fault intersections and where semicircular highly conductive zones in the DIGEM data have been interpreted as collapse features within limestone. These may in turn represent fluid upflow zones as described below.

The infiltration of mineralising acidic hydrothermal fluids has lead to the development of extensive karst dissolution features within the limestone and at or near the limestone shale contact. Gold mineralisation occurs as encapsulated gold in arsenopyrite needles or within arsenian pyrite in a sulphide rich zone, often brecciated and silicified that lies beneath a massive calcite zone. Beneath the thickest zone a stockwork of thin calcite veins occurs below the mineralisation and probably reflect the fluid conduit for the mineralisation.

Where karst development is greatest, collapse breccias are common with highly auriferous clay that has been produced from weathering of the primary ore. *Figure 7-12: Cross-Section through Pejiru Deposit Showing Mineralised Zone* gives a sectional view of the model of formation at Pejiru.

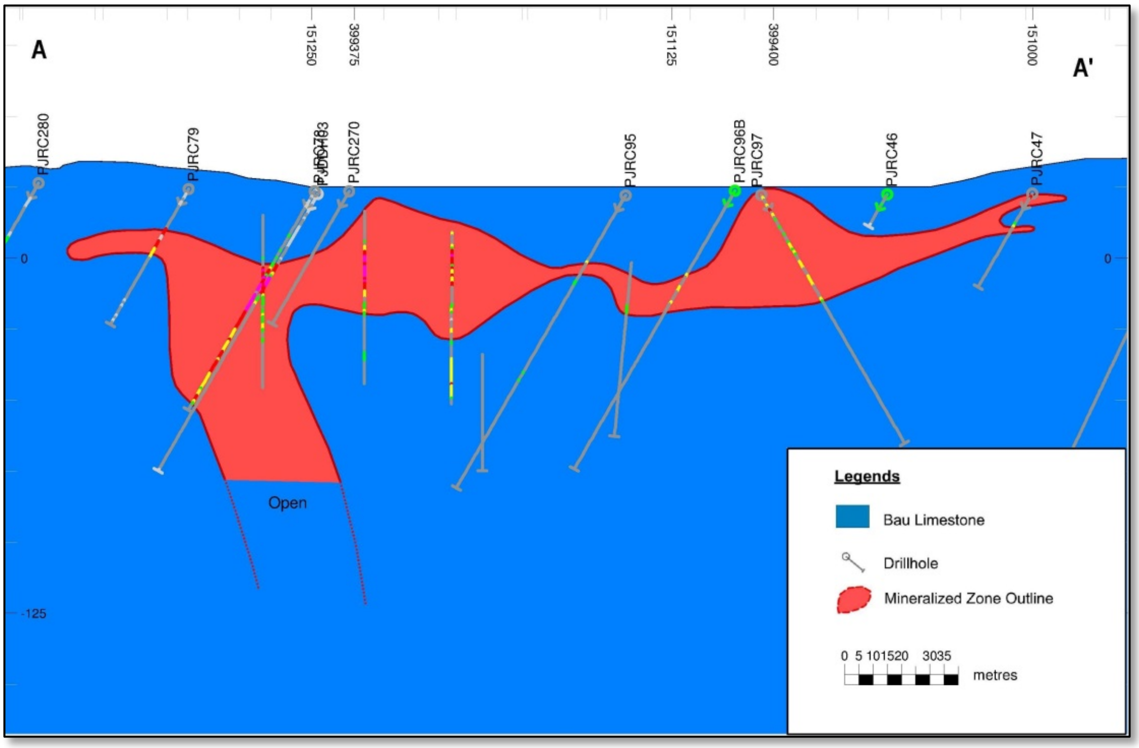


Figure 7-12: Cross-Section through Pejiru Deposit Showing Mineralised Zone

7.3.2.3.2. Boring

The Boring deposit comprises two areas that lie between 1.3 km and 2km NE of the centre of the Pejiru-Bogag deposit. The northern area covers around 150 m square and consists of several intersections of gold mineralization ranging from 3 m to 50 m in downhole intersections. All holes are vertical so it is difficult to know if mineralisation is flat lying or has some steep dipping structural control or a combination of both.

The southern area of the Boring Deposit consists of a NW-SE trending zone of mineralization as defined by drilling. The geology of the area is dominated by the SE trending Boring Fault against which a 1,500 m by 800m block of Pedawan formation, 40 m to 80 m thick, is down thrown against Bau Limestone. The mineralization is found within veins in the limestone and within sulphidic breccia along the karstic limestone shale contact. *Figure 7-13 - Geological Plan of Boring Deposit Showing Surface Projection of Gold Mineralisation* below outlines the gold mineralisation in the Boring area of the Pejiru Sector.

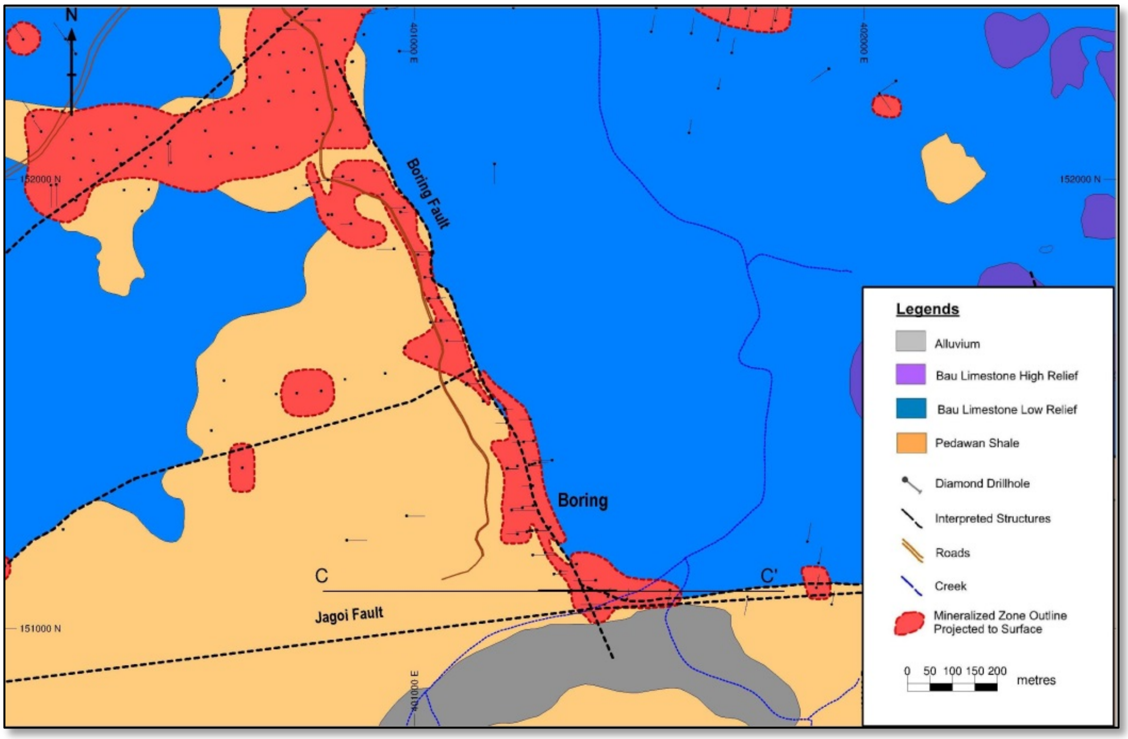


Figure 7-13 - Geological Plan of Boring Deposit Showing Surface Projection of Gold Mineralisation

7.3.2.3.3. Kapor

The Kapor deposit lies 5 km SW of Bau and is adjacent to the Fairy Cave National Park. Mineralisation can be traced almost continuously to Pejiru; 2.5 km along strike to the SW. Mineralisation is hosted in limestone as is the case at Pejiru but with much higher arsenic levels recorded. Again gold is associated with arsenopyrite and records show arsenic can reach to around 30 % in isolated samples, antimony is strongly anomalous with values in the 100's and 1000's of ppm. *Figure 7-14 - Geological Plan of Kapor Deposit Showing Surface Projection of*

Gold Mineralisation below outlines the gold mineralisation in the Kapor area of the Pejiru Sector.

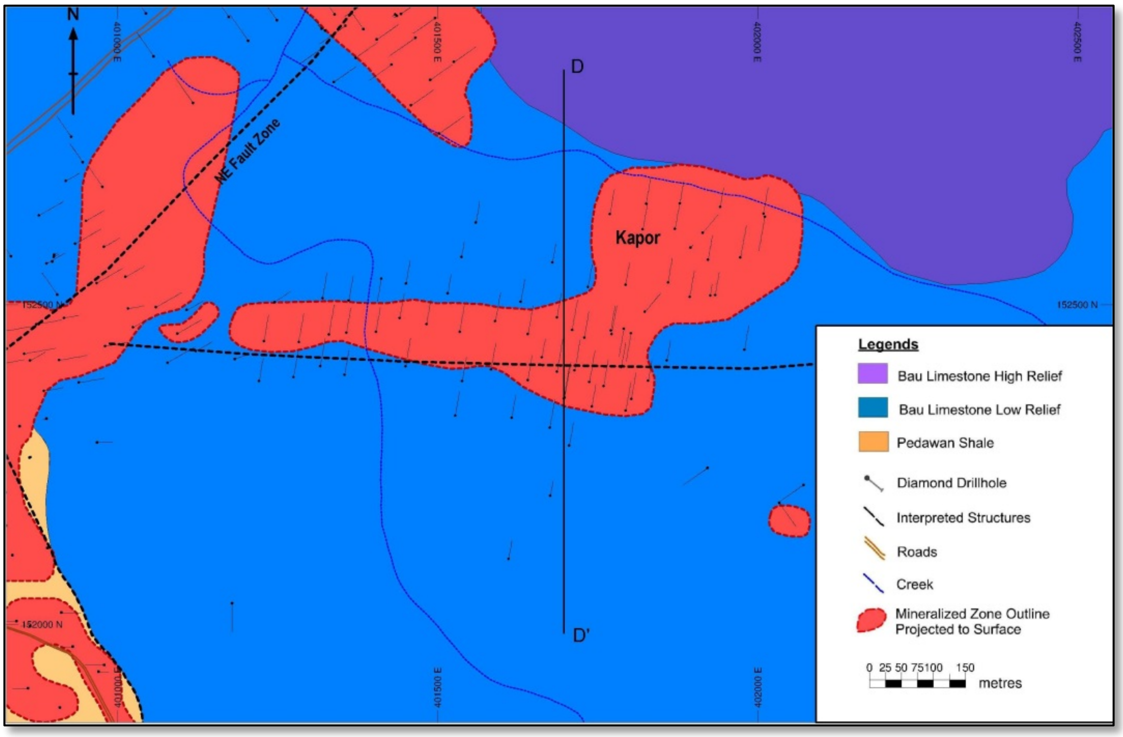


Figure 7-14 - Geological Plan of Kapor Deposit Showing Surface Projection of Gold Mineralisation

7.3.2.4. Taiton Sector

The Taiton Sector lies some 3 km to 4 km SW of Bau Township and is easily accessed via sealed and gravel road. The deposit types are dominantly mangano-calcite veins and breccias and remnants of the extensive elluvial auriferous clays that were mined historically by the Chinese miners of the late 19th Century and early 20th Century. Current modelling has defined Indicated Resources of 1.517 million tonnes at a grade of 2.75 g/t Au for 134,000 ounces and Inferred Resources of 3.419 million tonnes at a grade of 1.75 g/t Au for 192,000 ounces. This specifically excludes the mineralisation exposed in the underground workings at Taiton B as there has been insufficient work here to define a resource.

The main target areas aligned with the Tai Parit Fault are from South the North, Tabai, (which includes the former Rumoh Mine), Taiton A, Overhead Tunnel (over a strike length of ~1.2 km), Bungaat and Saburan, while those aligned with the NE Taiton Fault are Umbut, Taiton B and Taiton C. A geological plan showing the mineralisation outlines at the Taiton Sector are shown in *Figure 7-15 - Surface Outline of Gold Mineralization at Taiton Sector* below.

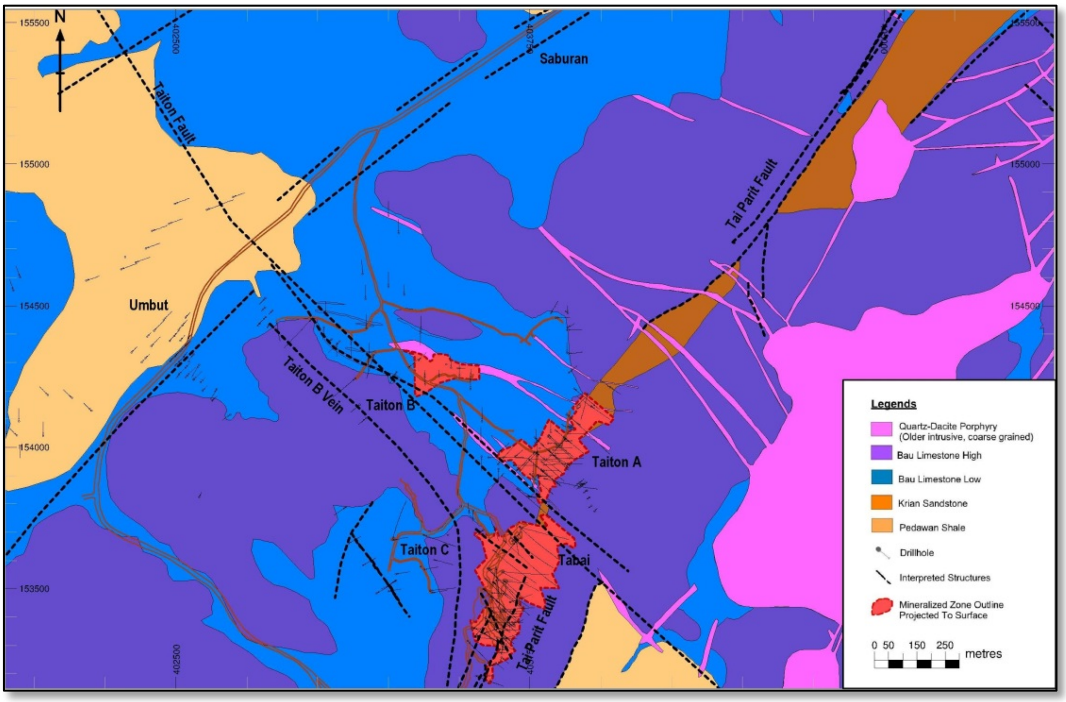


Figure 7-15 - Surface Outline of Gold Mineralization at Taiton Sector

7.3.2.4.1. Tabai

Tabai (including the former Rumoh mine) is developed on a vein system between 4m and up to 23m wide, (observed) mostly composed of brecciated mangano-calcite vein, frequently vuggy with drusy quartz infilling and overgrowths along with patchy silicification, auriferous clay, arsenopyrite, realgar, stibnite and native arsenic. The drill programmes in 2010 to 2011 and subsequent modeling has traced the mineralization essentially as far as Taiton A with a small (less than 100 m) separation between the two.

There are up to four (4) sub-parallel NE to NNW trending gold mineralised structures that persist to depths of 300 m in drillholes below surface. The vein system remains open at depth and along strike to the South. To the North it merges with Taiton A.

Mineralisation is largely confined to structures within the Bau Limestone; however, there are instances of gold being developed within vuggy drusy quartz veins along contacts with intrusive dacite porphyry dykes.

7.3.2.4.2. Taiton A

Taiton A is approximately 400 m further north along strike on the Tai Parit Fault Structure. It comprises the Taiton A open pit, the NW striking Overhead Tunnel Adit above Taiton A Pit and several adits that are located at the base of the limestone bluffs.

There are several mineralised NW fault structures trending toward, intersecting and cross cutting the Tai Parit fault zone. The main NW structure, the Overhead Tunnel Adit, lies vertically above the Taiton A pit and continues in a SE direction for several hundred metres.

Numerous old mine pits occur near the intersection of these prominent NW-SE trending structures with the NE trending Tai Parit Fault system.

Exposure at the Taiton A pit is limited; however the extensive drill core inventory shows mineralisation passes from an upper zone of auriferous secondary clay deposits into primary ore comprising mangano-calcite veining with abundant native arsenic, realgar, arsenopyrite, some silicification and drusy quartz veining, brecciation and massive white calcite veins.

A notable feature of Taiton A is that drilling has shown mineralization persists to at least 300m vertically and is open at depth. The mangano-calcite vein style is most common within the Bau Limestone but there is also a common association with the contact zones of NW trending dacite porphyry and andesite porphyry dykes and often weathering of auriferous clay to depths of 100m plus vertically below surface.

7.3.2.4.3. Bungaat

The Bungaat area lies ~400 m North of Taiton A. It comprises a NW-SE trending zone of mineralisation with native arsenic, realgar and coarse calcite vein material developed in a steep dipping structure with a well developed sub-horizontal set of mineralised calcite veins peripheral to the main structure.

7.3.2.4.4. Saburan

The Saburan Prospect lies on the Tai Parit Fault approximately 1 km north of Taiton A. The entrance to the former mine lies just outside the boundary with Gladioli's ML 108, but the workings extend into Gladioli's ground. The area outside the mining lease is under application by Gladioli. Saburan mineralisation is similar in character to Taiton A and Tabai with grades from underground rock samples collected by Zedex to 9 g/t Au recorded.

7.3.2.4.5. Taiton B & Taiton C

The Taiton-B massive mangano-calcite vein has now been mapped over a 1.5 km of strike length and includes the section known as Taiton C. It trends NW-SE and a 700 m section of this vein has historically been underground mined. Strike and depth extensions remain unexplored. From 74 vein rock chip samples, assays ranged from 0.16 to 62.0 g/t Au, with 48 % reporting above 1.0 g/t Au; the average being 7.85 g/t Au.

Mineralisation in detail is confined to mangano-calcite, quartz with bands and pods of realgar, arsenopyrite and stibnite. The vein is generally steep dipping to the NE with some flatter lying zones predominantly developed in the hanging wall that are composed of banded fine grained silica, native arsenic, arsenopyrite, realgar and rare stibnite.

7.3.2.4.6. Umbut

The Umbut area lies to the NW of Taiton B and partially straddles the Krokong Road. It is described by Bukit Young in internal memos from the mid 1990's as having mineralisation within quartz calcite ore and within the shale limestone contact.

7.3.2.5. Bekajang Sector

Revised modeling has increased the resources in the Bekajang Sector which now stand at Indicated Resources of 1.857 million tonnes at a grade of 2.02 g/t Au for 120,400 oz Au and Inferred Resources of 7.5 million tonnes at a grade of 1.76 g/t Au for 423,700 oz Au. A further 100,400 ounces of gold has been inferred for the BYG Tailings. *Figure 7-16: Representative Drill Intersections of the BYG-Krian-Johara Fault Trends* shows the main elements of the BYG pit-Krian Fault and Johara Fault mineralisation.

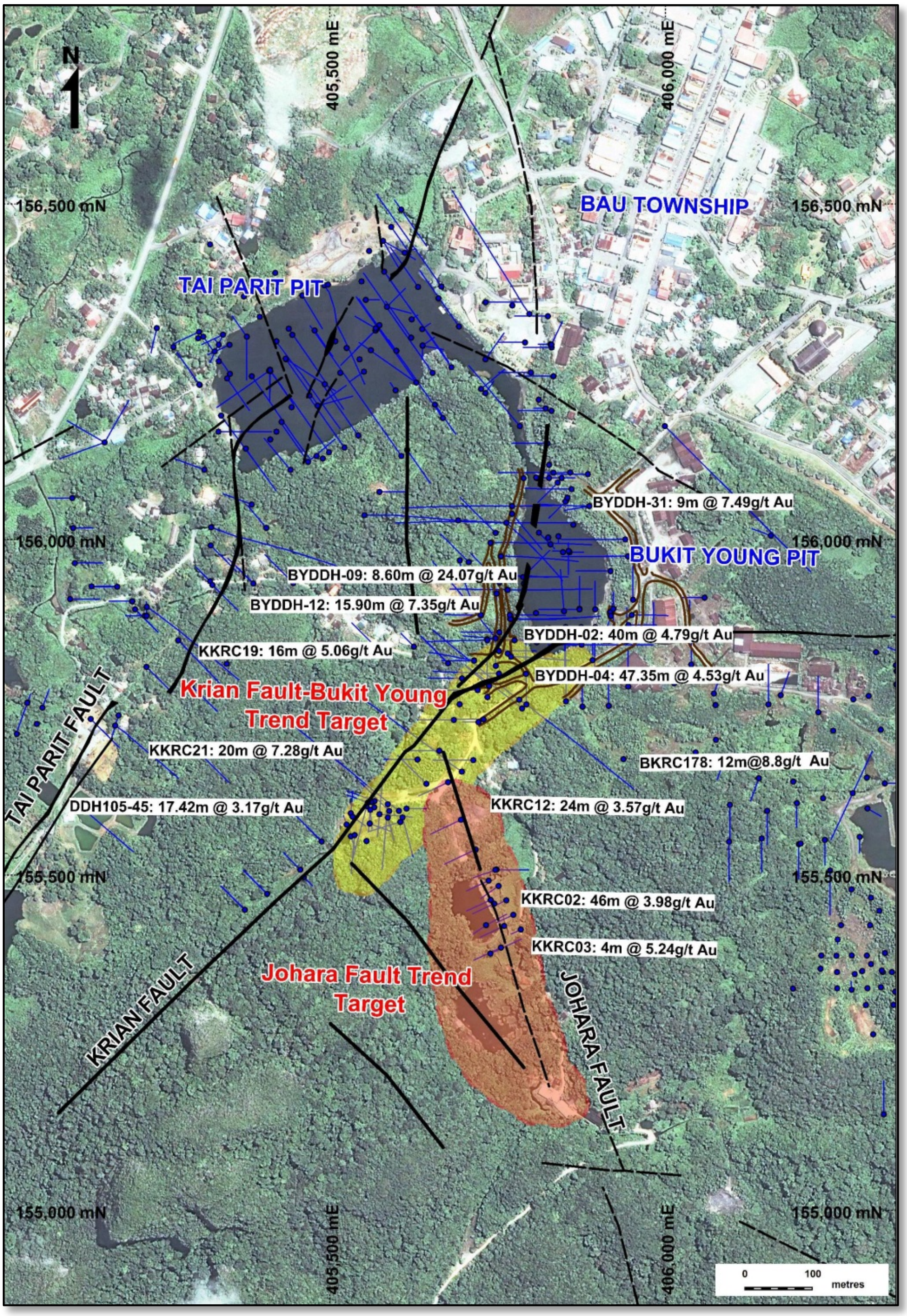


Figure 7-16: Representative Drill Intersections of the BYG-Krian-Johara Fault Trends

7.3.2.5.1. Gunong Krian

The Gunong Krian prospect is located on a steep up faulted block of Bau Limestone approximately 750m SW of the BYG plant site.

Essentially the target at Krian is based on surface and underground expressions of quartz calcsilicate and calcite veining historically mined for antimony (Lucky Hill mine) and gold and thought to be derived from a deep source.

The veins are generally NW-SE mineralised structures, frequently vuggy and with comb quartz infillings.

7.3.2.5.2. Bukit Young Pit

The Bukit Young Gold Pit (BYG Pit) is situated adjacent to the old mine office and plant site.

The deposit is developed in the eastern side of the NNE trending Krian Fault where it abuts on the western side against up thrown blocks of Krian Sandstone and adjoining felsic porphyry intrusives.

Gold mineralisation is associated with auriferous quartz-manganocarbonate-sulphide veins, vein stockworks, and tectonic/hydrothermal breccias developed principally within fault jogs and at major fault intersections in limestone. Ferruginous auriferous clays as cavity fill and microcrystalline silica as breccia matrix and limestone replacement show similarities to the ore mined at the adjoining Tai Parit Pit.

To the immediate west of the BYG Pit, a separate zone of disseminated and vein stockwork style gold mineralization occurs within an adjacent intrusive stock of dacitic composition and Krian coarse sandstone. Veins with classic bladed quartz pseudomorphing carbonate have been observed along with quanities of sphalerite indicative of deeper and hotter conditions during gold deposition, as compared to the deposits further south along the Tai Parit Fault.

7.3.2.5.3. Karang Bila

The area known as Karang Bila lies approximately 1km east of the BYG Plant site and 500m NE of the BYG Tailings dam. There has been a total of 6,806m of RC drilling in 54 drillholes recorded as having been drilled.

The mineralised zone appears to trend SE and is flat lying. There seem to be several zones of mineralisation and given the proximity of Bekajang are likely to be developed at the limestone shale contact and in parallel zones within the limestone.

7.3.2.5.4. Tai Parit

The Tai Parit deposit is immediately adjacent to Bau Township with the abandoned open pit now forming a recreational lake (Tasik Biru) for the town.

The Tai Parit Pit itself has recorded production of 700,000 ounces at an average grade of over 7 g/t Au from a body of silicified fault breccia aligned NNE-SSW on the Tai Parit Fault, the main controlling mineralising structure. Several ore types were recognised and mined at Tai Parit. These include, auriferous clay, siliceous breccia, jasperoidal silica and calcite veining

The deposit, while being apparently controlled by the Tai Parit Fault, is also in close proximity to high level felsic porphyry intrusives with typical quartz-sericite-pyrite alteration. Host rocks also include the Krian Sandstone, Bau Limestone and Pedawan Shale.

7.3.2.5.5. Bekajang

The Bekajang area lies immediately SE of the Bukit Young processing plant and has been traced for around 1,500m SE and approximately 700m across strike. Several deposits are known to occur at the shale/limestone contact and are generally shallow dipping features with mineralization developed in siliceous breccias within the shales on the contacts between shale and limestone.

During exploration in 2011 several holes were drilled to test beneath the lake within the Bekajang TSF. These holes intersected vuggy quartz veins developed in limestone with gold mineralization as well as a dacite porphyry dyke with strong quartz sericite alteration and disseminated arsenopyrite needles developed marginal to microfractures with a very similar paragenesis to the gold mineralisation at Sirenggok.

7.3.2.6. Say Seng Sector

7.3.2.6.1. Say Seng

The Say Seng Prospect is located between the west flank of Gunung Pangga and the Buso Road, about 3km NE of Bau. Exploration to date has been insufficient to define a resource here.

Mineralisation at Sey Seng appears to be controlled by steep structures within limestone, shallow dipping bedding plane parallel features, limestone shale contacts with the Sey Seng fault and intrusive contacts. The Borneo Geological Survey logs describe altered porphyry intrusives and calc-silicate alteration of wollastonite and garnet exoskarn. Mineralisation is associated with high sulphide contents.

7.3.2.6.2. Bukit Sarin

Bukit Sarin lies approximately 4.5km NE of Bau and is located near the intersection of the NW-SE Kojok Fault and the NE-SW trending Say Seng Fault. The area is described in Wolfenden as comprising quartzose Sb-Au ore in a quartz-shale breccia. There are similarities to Jugan in terms of geology and mineralisation style.

Significant gold mineralisation occurs in many of the previous holes drilled and consists of very fine, almost invisible needles of arsenopyrite hosted in shale, sandstone and to a lesser extent limestone. Better grade intersections are located in sandier and more deformed beds, adjacent to intrusive contacts.

7.3.2.7. Juala Sector

7.3.2.7.1. Juala West

The Juala West prospect is approx 700 m SSW on the same road that leads to Arong Bakit and is some 2.7 km SSW of Bau Township.

Surface sampling and trenching had located several areas of quartz veined stock worked porphyry and some boulders of highly siliceous skarn and breccia that locally had grades of 95 g/t Au.

The mineralisation can be classified as gold-copper+/- molybdenum in style with auriferous and copper bearing endo and exo-skarns on the contacts between limestone and the quartz stockworked porphyry.

The main ore minerals and associated alteration minerals observed in drill core and in sparse outcrop comprise bornite-chalcopyrite with minor sphalerite and molybdenite associated with wollastonite, diopsidic pyroxene, grossular garnet, calcite and quartz.

7.3.2.7.2. Arong Bakit

The Arong Bakit area lies approximately 2 km SSW of Bau Township. The site is currently being worked as a marble/limestone quarry and is proximal to the Juala intrusives.

The current quarrying operations have obscured much of the mineralisation at lower easily accessible elevations, however, there are a large number of boulders derived from the quarry that comprise crackle brecciated marble with the interstices between clasts infilled with arsenopyrite and pyrite. Galena, sphalerite, chalcopyrite and bornite was also observed in some pieces. This crackle breccia tends to average around 10 g/t Au.

Several holes were targeted at the marble/intrusive contact. High grade gold minerlaisation was successfully intersected in an exoskarn zone which upon petrographic examination showed intensely altered fine-grained probable radiolarian and bioclastic limestone. Veining of quartz-wollastonite-pyroxene, (diopside)–garnet, (grossularite) and quartz-arsenopyrite-sphalerite-wollastonite-garnet, (grossularite) dominates.

The altered radiolarian limestone marginal to the veining has rare radiolaria replaced by sphalerite with galena and occasional minute inclusions of gold (<10 μm); while banded quartz-arsenopyrite-sphalerite-wollastonite-garnet veins have intergrowths of arsenopyrite with quartz and lesser sphalerite. The arsenopyrite contains several gold grains (~25 μm in size).

8. Deposit Types

8.1. Summary of Deposit Types

The known deposits in the Bau Goldfield can be characterized by four (4) distinctive gold mineralisation styles that exhibit both lateral and vertical geochemical and mineralogical zonation with respect to the Bau Trend intrusives. In general these styles are:

- Sediment Rock-Hosted Disseminated Gold Deposits, e.g. Jugan; Bukit Sarin;
- Silica replacement (jasperoid) and open space siliceous breccias, e.g. Tai Parit; Bukit Young Pit, Bekajang;
- Mangano-calcite-quartz veins, e.g. Tai Ton; Pejiru, Kapor;
- Magmatic – Hydrothermal porphyry related deposits with/without calc-silicate skarn, e.g. Sirenggok, Sey Seng, Ropih, Arong Bakit, Juala West.

It is noted that a number of the individual deposits have elements of several of the recognized mineralisation styles.

These deposit styles have been likened to and share characteristics with the gold deposits of the Carlin Trend, Nevada, USA. They have a similar geological setting where deposits are commonly hosted in calcareous sediments, are Tertiary in age, host rock permeability important in focusing mineralization, associated with deep faults, Tertiary-aged dacitic intrusives, solution collapse breccias common and a probable epithermal association, at least in part in the case of Bau.

Similarities in mineralisation style include silicic-argillic-carbonate hydrothermal alteration, fine grained arsenopyrite-pyrite gold ore common and similar trace element geochemistry, (As, Sb, Hg, Tl). *Figure 8-1: Generalised Section of Bau Deposit Types with Jugan as the Hypothetical Example* shows a conceptualized sectional view of the relationship of the main deposit types in the Bau Goldfield.

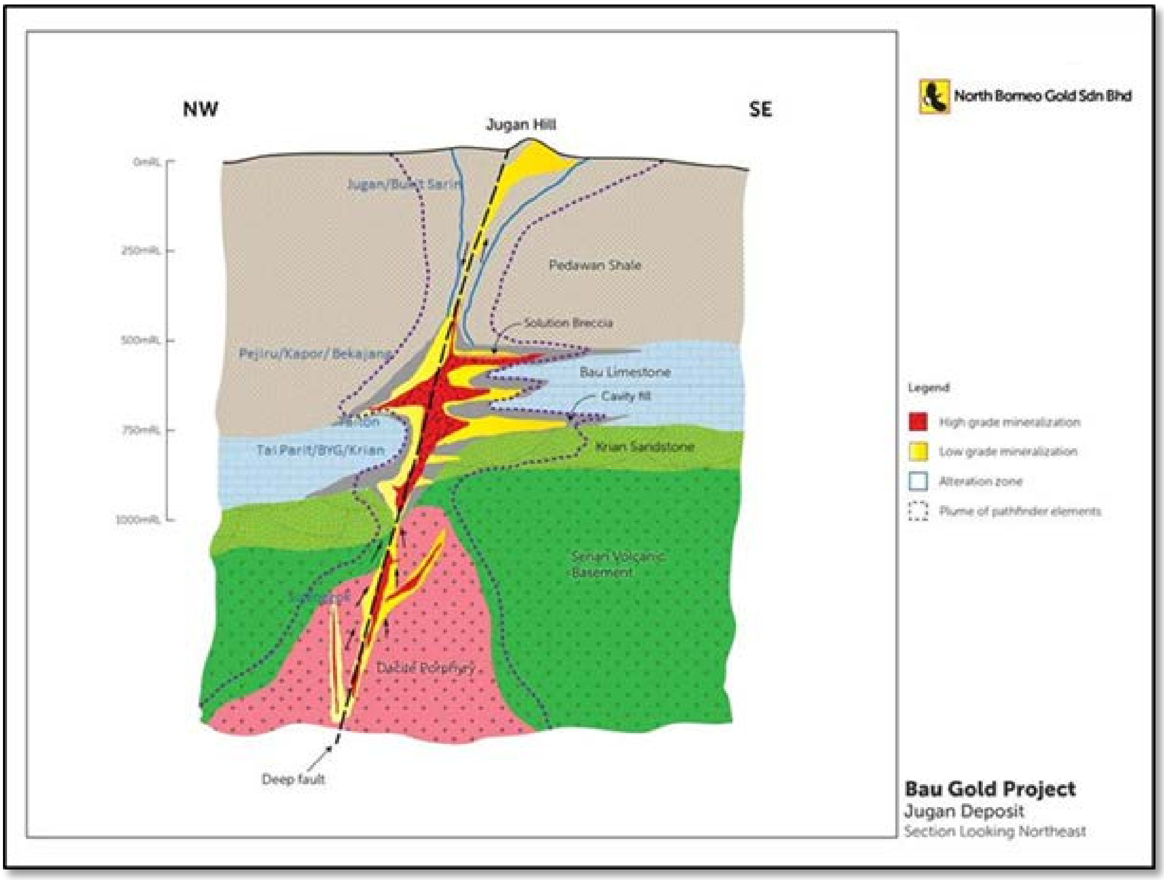


Figure 8-1: Generalised Section of Bau Deposit Types with Jugan as the Hypothetical Example

Lateral zoning observed is interpreted to be related to the proximity to the Bau Trend felsic intrusives where they crop out in the up domed portion of the Bau Limestone, (anticline axis of the Bau Anticline or Bau Pop-up?).

The general trend outward from intrusive centres is skarn/calc-silicate porphyry environment to silica rich mineralised breccias to silica replacement/calcite limestone contact to the more distal disseminated styles such as Jugan.

Similar zonation patterns have been observed vertically within deposits such as Tai Parit which is the only deposit to have been mined to any depth. This zoning pattern is exemplified in the fluid inclusion temperature of formation zoning pattern developed by *Schuh, 1993* and *Percival et al, 1990*, as shown in *Figure 8-2: Lateral Fluid Inclusion Temperature Zoning*.

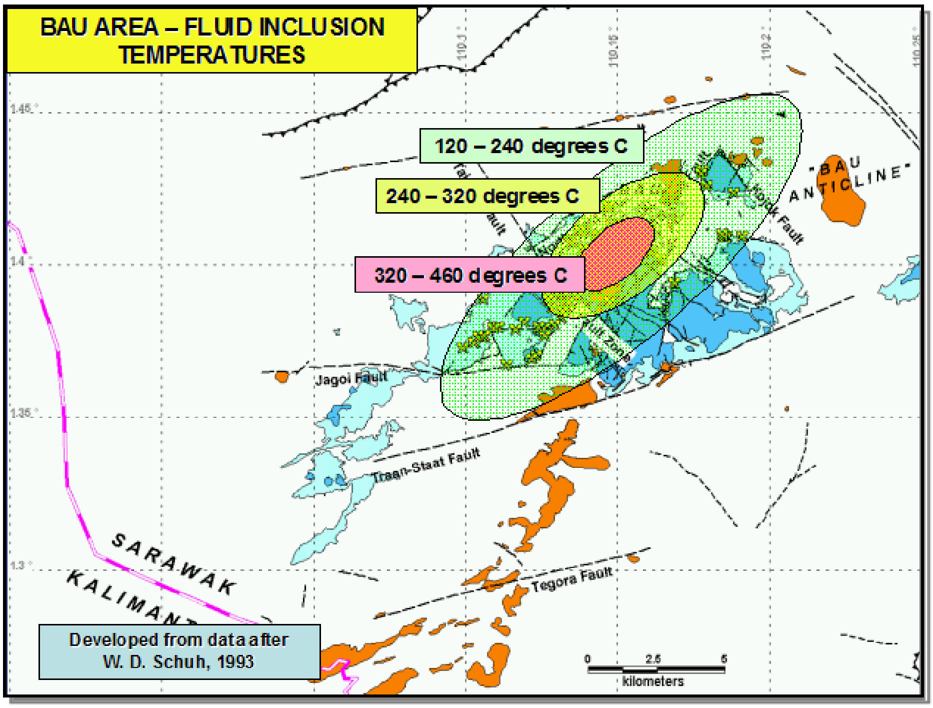


Figure 8-2: Lateral Fluid Inclusion Temperature Zoning

Previous exploration has to some degree focused on ideas that the deposits in the central part of the field are less refractory due to the general observation that the deposits become more arsenopyrite rich further away from the intrusive centres as shown in *Figure 8-3: Bau District Metal Zonation & Refractoriness*

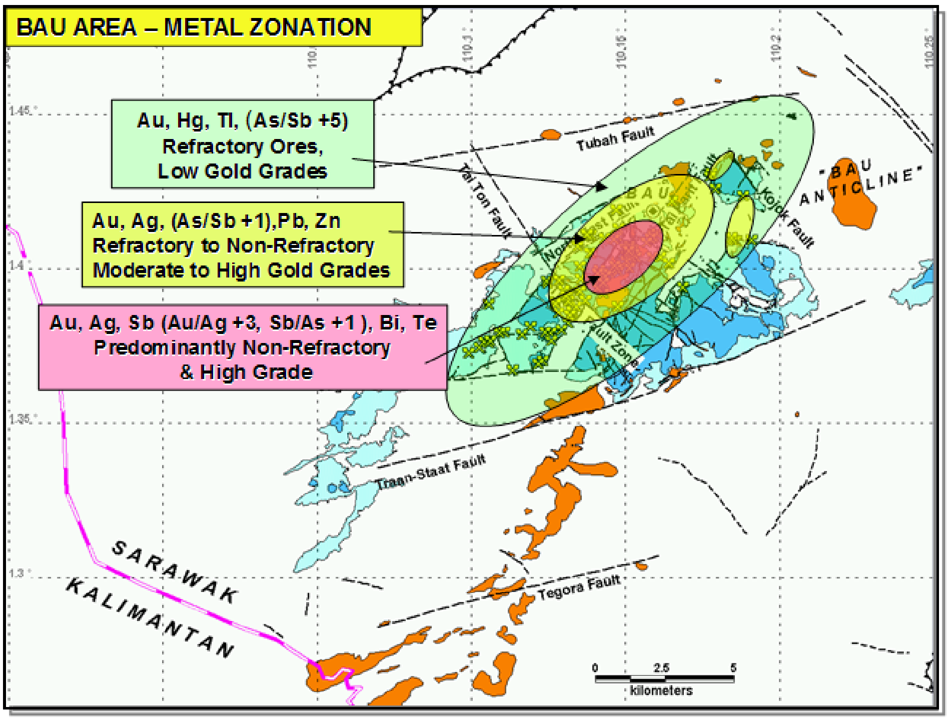


Figure 8-3: Bau District Metal Zonation & Refractoriness

It is believed that the current zonation is partly a function of the level of exposure and that seemingly more distal deposits such as Jugan, Taiton, and Pejiru have excellent potential for locating mineralisation similar to Tai Parit/Bekajang vertically beneath the current levels of exposure.

9. Exploration

9.1. General

Zedex Minerals became involved in the Bau Goldfield in late 2006 through its joint venture with Gladioli Enterprises Sdn Bhd and the formation of North Borneo Gold Sdn Bhd (NBG). NBG pursued a programme comprising drilling, geological mapping, database collation, evaluation and resource modeling that culminated in delineating combined resources of some 2.45 million ounces of gold in 2010.

With the merger between Besra Gold Inc., and Zedex Minerals in January, 2010 Besra took over as operator of the joint venture and the exploration and development programmes going forward at Bau.

NBG have carried out and completed an aggressive exploration campaign since 2010 that has seen an increase in the resource base, both in ounces and in category, to 1.125 Moz Au of Measured and Indicated Resources and 2.182 Moz Au of Inferred Resources.

The programmes consisted of data review, target generation and combined resource and exploratory drilling to maximize the Company's objective of having the Bau Project advance to feasibility in the shortest possible time.

The following is a description of the exploration projects which necessarily includes some of the information already presented in *Chapter 8 Deposit Types*.

9.2. Project Exploration Review

9.2.1. Jugan Sector

9.2.1.1. Jugan

9.2.1.1.1. Resource Upgrade

NBG have undertaken an intensive exploration and resource definition programme on the Jugan Hill Deposit since late 2011. This saw the completion of 17,395.4 metres of diamond core drilling in seventy-nine (79) drillholes including 678.8 metres in eight (8) drillholes for metallurgical purposes, and the excavation of eleven (11) trenches over a length of 746.1 metres.

The programme was designed to increase the resources at Jugan both in quantum of contained gold ounces and in resource category. It also augments previous extensive exploration by other companies including NBG before the Zedex-Besra merger in 2010 that included over 17,450 metres of drilling in one hundred and sixty-eight (168) drillholes and 1,133.5 metres of trenching in forty-four (44) trenches.

This has seen the resources increased from an Indicated Resource of 10.963 million tonnes at 1.60 g/t Au for 563,000 ounces to a Measured and Indicated Resource of 17.911 million tonnes at 1.51 g/t Au for 870,500 ounces and an Inferred Resource of 1.774 million tonnes at 1.57 g/t Au for 89,800 ounces.

Significant drill and trench intercepts are shown in *Table 9-1: Jugan Significant Drill Intersections* and *Table 9-2: Jugan Significant Trench Intersections* below.

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
JUDDH-07	4.00	6.00	2.00	0.79
JUDDH-08	0.00	2.00	2.00	0.77
JUDDH-09	0.80	9.00	8.20	1.26
JUDDH-10	22.70	25.00	2.30	0.55
JUDDH-10	233.00	236.00	3.00	1.04
JUDDH-10	243.00	244.00	1.00	0.55
JUDDH-10	264.00	265.00	1.00	0.70
JUDDH-10	287.00	295.00	8.00	1.21
JUDDH-10	320.00	322.00	2.00	0.75
JUDDH-10	328.00	333.00	5.00	1.07
JUDDH-10	352.00	353.00	1.00	0.66
JUDDH-10	360.00	361.00	1.00	1.10
JUDDH-10	367.00	370.00	3.00	0.65
JUDDH-10	450.00	451.00	1.00	0.54
JUDDH-10	457.00	458.35	1.35	1.06
JUDDH-11	0.00	78.30	78.30	2.01
JUDDH-11 - incl	0.00	11.00	11.00	4.40
JUDDH-11 - incl	17.00	28.00	11.00	3.90
JUDDH-11 - incl	39.00	43.00	4.00	4.05
JUDDH-12	18.00	21.00	3.00	0.84
JUDDH-12	28.60	36.00	7.40	0.84
JUDDH-12	40.00	51.80	11.80	1.23
JUDDH-12	58.00	82.00	24.00	2.32
JUDDH-12 - incl	65.00	74.00	9.00	3.87
JUDDH-13	0.00	16.50	16.50	3.08
JUDDH-13 - incl	3.00	9.00	6.00	5.75
JUDDH-13 - and	3.00	5.00	2.00	9,13
JUDDH-14	0.00	68.50	68.50	1.00
JUDDH-14 - incl	27.00	60.00	33.00	1.46
JUDDH-14 - and	51.00	58.00	7.00	3.21
JUDDH-15	0.00	2.60	2.60	0.91
JUDDH-15	0.00	8.00	8.00	0.62
JUDDH-15	14.00	39.00	25.00	1.77
JUDDH-15	41.00	64.00	23.00	1.62
JUDDH-15	71.00	81.50	10.50	1.99
JUDDH-16	2.00	9.70	7.70	0.93
JUDDH-16	13.00	15.00	2.00	0.77
JUDDH-16	19.00	36.00	17.00	0.72

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
JUDDH-16	39.00	60.35	21.35	1.74
JUDDH-16 - incl	41.00	45.85	4.85	4.67
JUDDH-17	0.00	3.50	3.50	0.95
JUDDH-17	26.90	28.00	1.10	1.00
JUDDH-17	32.00	36.00	4.00	2.65
JUDDH-17	62.00	92.00	30.00	1.53
JUDDH-18	22.00	87.40	65.40	1.79
JUDDH-19	0.00	4.40	4.40	0.51
JUDDH-19	26.60	27.20	0.60	1.09
JUDDH-19	66.70	76.70	10.00	0.67
JUDDH-20	1.30	7.30	6.00	2.21
JUDDH-20 - incl	4.30	6.30	2.00	4.07
JUDDH-20	13.30	62.15	48.85	1.98
JUDDH-20 - incl	16.30	18.30	2.00	3.93
JUDDH-20 - incl	22.30	26.50	4.20	6.30
JUDDH-20 - with	23.30	24.50	1.20	15.30
JUDDH-20 - incl	37.50	40.00	2.50	4.39
JUDDH-20 - incl	36.50	42.85	6.35	2.48
JUDDH-20 - incl	45.40	62.15	16.75	2.13
JUDDH-21	0.00	15.00	15.00	1.07
JUDDH-21	31.00	32.00	1.00	0.55
JUDDH-22	65.50	71.00	5.50	2.39
JUDDH-23	1.70	11.00	9.30	0.93
JUDDH-23	27.00	65.10	38.10	3.91
JUDDH-24	4.00	20.00	16.00	1.94
JUDDH-24	24.00	38.00	14.00	1.86
JUDDH-24	43.20	44.40	1.20	0.69
JUDDH-25	265.00	291.00	26.00	0.54
JUDDH-25 - incl	265.00	272.00	7.00	0.88
JUDDH-26	211.00	235.00	24.00	1.52
JUDDH-26	245.00	247.00	2.00	0.60
JUDDH-27	0.00	2.00	2.00	2.28
JUDDH-27	5.00	5.70	0.70	0.99
JUDDH-27	77.10	94.00	16.90	2.14
JUDDH-28	168.00	169.00	1.00	1.74
JUDDH-28	222.00	252.00	30.00	0.82
JUDDH-28 - incl	232.00	240.00	8.00	1.36
JUDDH-29	0.00	3.00	3.00	1.34
JUDDH-30	2.00	6.00	4.00	0.62
JUDDH-31	0.00	40.00	40.00	3.99
JUDDH-31 - incl	2.00	19.00	17.00	5.69
JUDDH-32	241.00	282.00	41.00	1.79
JUDDH-32 - incl	247.00	249.00	2.00	5.24
JUDDH-32 - incl	276.00	281.00	5.00	5.24
JUDDH-33	128.00	151.00	23.00	0.87
JUDDH-33 - incl	128.00	139.00	11.00	1.04

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
JUDDH-33 - incl	146.20	151.00	4.80	1.40
JUDDH-34	33.00	52.00	19.00	0.87
JUDDH-34 - incl	33.00	45.00	12.00	1.19
JUDDH-35	36.00	44.10	8.10	1.09
JUDDH-35	53.50	87.50	34.00	3.01
JUDDH-35 - incl	54.50	63.50	9.00	7.60
JUDDH-36	256.40	297.00	40.60	1.51
JUDDH-36 - incl	313.20	320.00	6.80	2.73
JUDDH-36 - with	317.00	320.00	3.00	5.22
JUDDH-37	163.00	169.00	6.00	0.78
JUDDH-37	182.00	186.00	4.00	0.93
JUDDH-37	190.00	234.00	44.00	1.10
JUDDH-38	9.00	93.00	84.00	0.78
JUDDH-38 - incl	9.00	12.00	3.00	1.22
JUDDH-38 - and	16.00	93.00	77.00	0.79
JUDDH-38 - and	25.10	36.00	10.90	1.02
JUDDH-38 - and	50.00	74.00	24.00	0.98
JUDDH-38 - and	80.00	93.00	13.00	1.31
JUDDH-39	276.00	291.20	15.20	1.32
JUDDH-40	44.00	106.00	62.00	2.02
JUDDH-40 - incl	56.00	71.00	15.00	3.65
JUDDH-41	157.00	163.00	6.00	0.67
JUDDH-41	168.00	184.00	16.00	1.08
JUDDH-41	195.30	200.00	4.70	0.46
JUDDH-42	416.40	417.00	0.60	0.35
JUDDH-43	7.00	14.00	7.00	0.89
JUDDH-43	33.00	73.00	40.00	1.18
JUDDH-44	2.70	55.40	52.70	4.64
JUDDH-44 - incl	6.00	27.00	21.00	6.80
JUDDH-44 - incl	23.00	27.00	4.00	11.97
JUDDH-45	4.00	10.00	6.00	0.66
JUDDH-45	26.00	65.50	39.50	1.91
JUDDH-45 - incl	47.00	65.50	18.50	3.39
JUDDH-46	181.00	184.00	3.00	0.73
JUDDH-46	194.00	195.00	1.00	0.53
JUDDH-46 - incl	199.00	232.00	33.00	0.88
JUDDH-46 - incl	199.00	202.10	3.10	1.73
JUDDH-46 - incl	215.00	231.00	16.00	1.00
JUDDH-47	0.00	1.50	1.50	0.85
JUDDH-47	14.30	25.00	10.70	1.02
JUDDH-47	32.00	74.95	42.95	1.36
JUDDH-48	0.00	29.30	29.30	2.94
JUDDH-48	36.00	37.00	1.00	0.62
JUDDH-48	42.70	96.00	53.30	1.41
JUDDH-48 - incl	19.00	22.00	3.00	10.05
JUDDH-49	313.00	324.00	11.00	2.58

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
JUDDH-49	338.00	343.00	5.00	1.36
JUDDH-49	348.00	352.30	4.30	1.72
JUDDH-50	0.00	5.00	5.00	1.00
JUDDH-50	16.70	77.80	61.10	2.40
JUDDH-50 - incl	19.00	21.00	2.00	5.93
JUDDH-50 - incl	29.00	36.00	7.00	5.36
JUDDH-50 - incl	57.00	62.70	5.70	3.53
JUDDH-50 - incl	67.00	71.00	4.00	3.90
JUDDH-51	187.00	199.00	12.00	0.65
JUDDH-51	203.00	237.00	34.00	2.47
JUDDH-51 - incl	225.00	233.80	8.80	3.52
JUDDH-51	240.00	260.30	20.30	1.13
JUDDH-51	269.00	286.00	17.00	1.66
JUDDH-52	205.00	218.00	13.00	1.10
JUDDH-52	231.00	243.00	12.00	0.94
JUDDH-53	285.00	321.00	36.00	1.15
JUDDH-53 - incl	287.00	299.00	12.00	1.43
JUDDH-53 - incl	305.00	316.00	11.00	1.35
JUDDH-54	139.00	141.00	2.00	0.72
JUDDH-54	149.00	150.00	1.00	0.50
JUDDH-54	153.00	154.00	1.00	0.90
JUDDH-54	206.00	210.00	4.00	1.19
JUDDH-54	243.00	248.20	5.20	0.85
JUDDH-55	175.00	177.00	2.00	1.19
JUDDH-55	182.00	189.00	7.00	0.75
JUDDH-55	201.00	224.00	23.00	0.92
JUDDH-55	227.00	230.00	3.00	1.14
JUDDH-56	331.80	341.00	9.20	1.88
JUDDH-57	192.00	196.00	4.00	1.75
JUDDH-57	201.00	234.00	33.00	1.48
JUDDH-57	237.00	243.00	6.00	2.26
JUDDH-57	248.00	248.90	0.90	0.67
JUDDH-58	263.00	287.00	24.00	1.76
JUDDH-58 - incl	267.00	274.00	7.00	3.27
JUDDH-59	202.00	203.90	1.90	0.79
JUDDH-59	209.10	209.50	0.40	14.10
JUDDH-59	224.00	233.00	9.00	0.95
JUDDH-59	237.00	239.20	2.20	0.75
JUDDH-59	242.00	274.40	32.40	1.92
JUDDH-59 - incl	246.00	274.00	28.00	2.21
JUDDH-60	221.15	263.00	41.85	1.55
JUDDH-60 - incl	221.15	235.00	13.85	1.76
JUDDH-60- incl	249.00	263.00	14.00	2.25
JUDDH-61	232.00	268.40	36.40	1.40
JUDDH-61 - incl	260.00	268.40	8.40	2.44
JUDDH-63	208.00	256.00	48.00	1.05

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
JUDDH-65	239.00	240.00	1.00	0.76
JUDDH-66	226.30	232.00	5.70	1.10
JUDDH-66	239.00	254.00	15.00	1.41
JUDDH-66 -incl	251.00	253.00	2.00	5.23
JUDDH-66	271.80	282.10	10.30	0.73
JUDDH-67A	285.00	291.20	6.20	0.86
JUDDH-68A	317.50	342.00	24.50	7.15
JUDDH-68A - incl	320.00	331.50	11.50	10.42
JUDDH-69A	215.00	216.00	1.00	0.58
JUDDH-69A	223.00	270.00	47.00	1.05
JUDDH-69A - incl	231.00	241.00	10.00	1.68
JUDDH-69A	275.00	279.00	4.00	0.89
JUDDH-70	87.00	94.00	7.00	1.13
JUDDH-70 - and	105.00	127.00	22.00	1.44
JUDDH-70 - incl	111.00	118.00	7.00	2.70
JUDDH-70 - and	134.40	174.65	40.25	0.86
JUDDH-70 - incl	151.00	157.00	6.00	1.55
JUDDH-70 - incl	165.00	174.00	9.00	1.27
JUDDH-71	98.00	161.00	63.00	1.58
JUDDH-71 - incl	110.00	119.00	9.00	2.28
JUDDH-71 - incl	136.00	146.00	10.00	3.17
JUDDH-72	28.70	49.00	20.30	1.24
JUDDH-72	53.00	91.00	38.00	2.08
JUDDH-72 - incl	60.00	65.00	5.00	3.00
JUDDH-72 - incl	86.00	90.00	4.00	3.74
JUDDH-73	210.00	213.00	3.00	0.99
JUDDH-73	215.00	218.00	3.00	0.6
JUDDH-73	224.00	243.00	19.00	0.89
JUDDH-73	250.00	265.00	15.00	1.69
JUDDH-74	288.00	294.00	6.00	1.57
JUDDH-74	323.00	324.00	1.00	2.27
JUDDH-75	227.00	234.00	7.00	1.11
JUDDH-75	244.00	290.00	46.00	1.24
JUDDH-75	295.30	316.30	21.00	1.25
JUDDH-76	260.00	279.20	19.20	3.14
JUDDH-76 - incl	267.35	279.20	11.85	4.65
JUDDH-76	291.00	298.00	7.00	0.62
JUDDH-76	302.35	311.00	8.65	3.09
JUDDH-76 - incl	308.00	309.00	1.00	16.00
JUDDH-77	216.00	219.00	3.00	1.63
JUDDH-77	279.00	289.00	10.00	2.94
JUDDH-77 - incl	282.00	289.00	7.00	3.84
JUDDH-78	249.00	250.00	1.00	0.83
JUDDH-78	266.00	268.00	2.00	0.68

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
JUDDH-78	293.00	317.00	24.00	0.97
JUDDH-78 - incl	293.00	304.00	11.00	1.77
JUDDH-78 - and	294.00	299.00	5.00	2.79
JUDDH-79	316.40	343.00	26.60	2.12
JUDDH-79 - incl	320.00	335.00	15.00	3.33
JUDDH-79	353.00	355.00	2.00	1.58
JUDDH-79	363.00	365.00	2.00	3.38
JUDDH-80	315.00	339.00	24.00	2.68
JUDDH-80 - incl	315.00	330.00	15.00	3.86
JUDDH-80	344.40	349.00	4.60	4.69
JUDDH-80 - incl	347.00	348.00	1.00	16.60
JUDDH-80	352.00	357.00	5.00	4.87
JUDDH-80	362.00	367.00	5.00	0.69
JUDDH-81	307.00	331.10	24.10	1.31
JUDDH-81 - incl	312.00	331.10	19.10	1.55

Table 9-1: Jugan Significant Drill Intersections

Trench No	From (m)	To (m)	Length (m)	Au (g/t)
JUT-01	46.00	48.00	2.00	0.67
JUT-01	24.00	32.00	8.00	1.08
JUT-02	12.20	14.20	2.00	0.55
JUT-02	16.20	18.20	2.00	1.04
JUT-02	26.20	32.90	6.70	1.61
JUT-02	29.20	30.90	1.70	4.18
JUT-02	26.20	34.95	8.75	1.30
JUT-02	39.95	81.30	41.35	3.47
JUT-02	89.55	105.80	16.25	4.18
JUT-03A	71.3	113.3	42.0	1.17
JUT-04	0.0	39.0	39.0	1.74
JUT-04	45.3	73.0	27.7	1.71
JUT-04	69.0	172.0	103.0	1.30
JUT-05A	0.00	10.00	10.00	1.83
JUT-05A	18.0	33.0	15.0	1.27
JUT-05A	37.6	40.6	3.0	0.58
JUT-05A	53.80	60.80	7.00	0.76
JUT-05A	72.8	78.8	6.0	1.30
JUT-05A	82.2	83.2	1.0	10.10
JUT-05B	4.7	5.7	1.0	1.45
JUT-05B	9.7	13.7	4.0	2.56
JUT-05B	16.7	50.3	33.6	1.87
JUT-06A	11.0	12.0	1.0	0.50
JUT-06B	0.0	1.1	1.1	4.08
JUT-06B	6.0	9.0	3.0	1.17
JUT-06B	11.0	19.0	8.0	1.00

Trench No	From (m)	To (m)	Length (m)	Au (g/t)
JUT-07	0.0	1.0	1.0	0.92
JUT-07	6.0	9.0	3.0	1.19

Table 9-2: Jugan Significant Trench Intersections

9.2.1.1.2. Geology, Mineralisation & Structural Modelling

The deposit is hosted within the Pedawan formation, predominantly in highly deformed and sheared carbonaceous shale, laminated shales, mudstones and interbeds of fine to medium grained sandstone. The shearing and fold axes are dominantly NE trending with the gold mineralisation forming within acicular arsenopyrite and arsenian pyrite disseminated throughout the sediments and within carbonate (ankeritic) veinlet stockworks. Typically, the arsenopyrite content ranges between 1 % and 5 % and arsenian pyrite from trace to 5 %. Overall sulphide content in the ore zone can be in the 5 % to 7 % range. Sulphide content and gold grade have a close correlation. The deposit has been drilled to approximately 350 metres vertically without the limestone-shale contact being intersected. Several NW trending dykes of post mineral microgranodiorite porphyry transect the ore zone and are invariably strongly hydrothermally altered. *Figure 9-1 - Surface Outline of Gold Mineralisation showing Alteration and Mapping at Jugan* below and in *Chapter 7* shows the geology of the deposit.

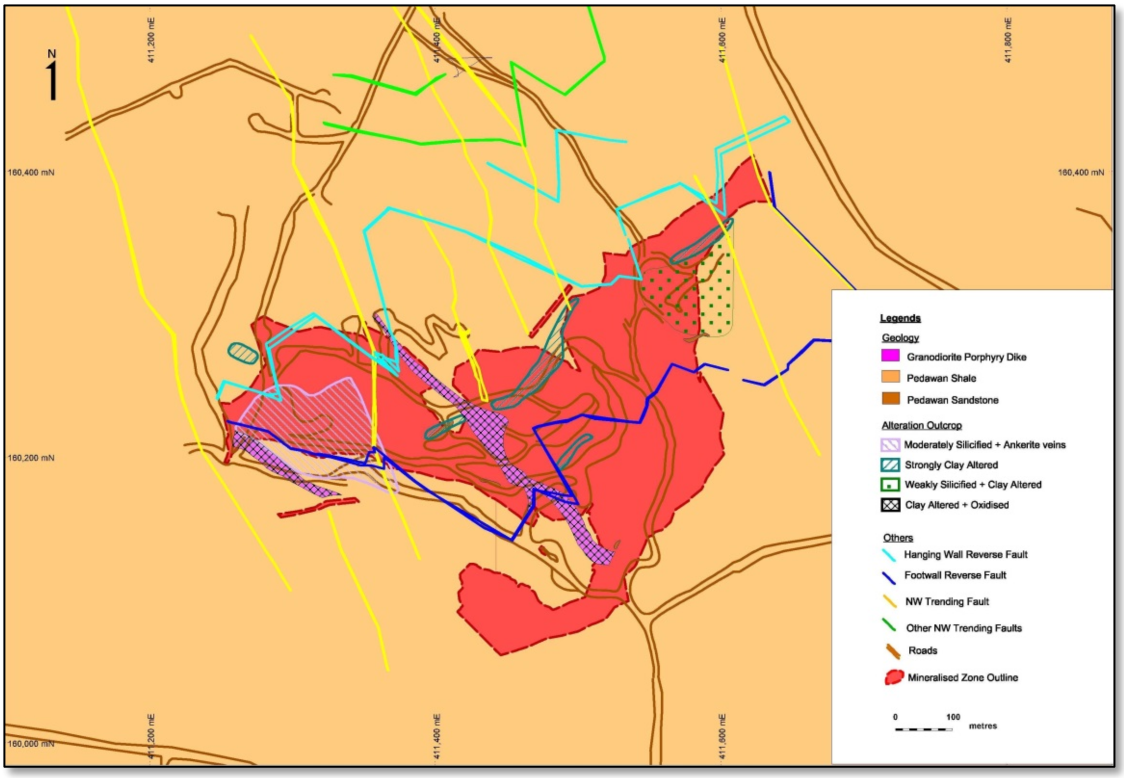


Figure 9-1 - Surface Outline of Gold Mineralisation showing Alteration and Mapping at Jugan

The currently defined resource is largely constrained between hanging wall and footwall shears that strike NE-SW and dip between 55° and 75° NW. In addition a number of NW-SE trending shear zones have been identified some which appear to be post mineral although may have been developed prior to or during the mineralizing event. There is an interpreted dextral sense of movement on these and opens the possibility of offset extensions and repetitions of the deposit. A well-developed NW-SE trending shear is interpreted to dip at approximately 70° to the NE and appears to cut of the ore body.

The there is a higher grade zone that plunges NE within the plane of the NW dipping ore body. This correlates with a slight increase in incipient silicification and sulphide content. Mineralisation remains open at depth and to the NE.

Structural analysis by NBG geologists has identified that in the eastern part of the ore body there may be a displacement to the ESE by dextral-movement of the traversing NW-fault. This is based on analysis of oriented drill core and interpretation but no direct evidence exists at this time however the hypothesis needs to be tested with further drilling. *Figure 9-2 - 3D View of Jugan Orebody & NE Faults looking NW* to *Figure 9-3 - 3D View of Jugan Orebody & NW Faults looking NE* show the geometry of the ore body and main structural elements in 3D.

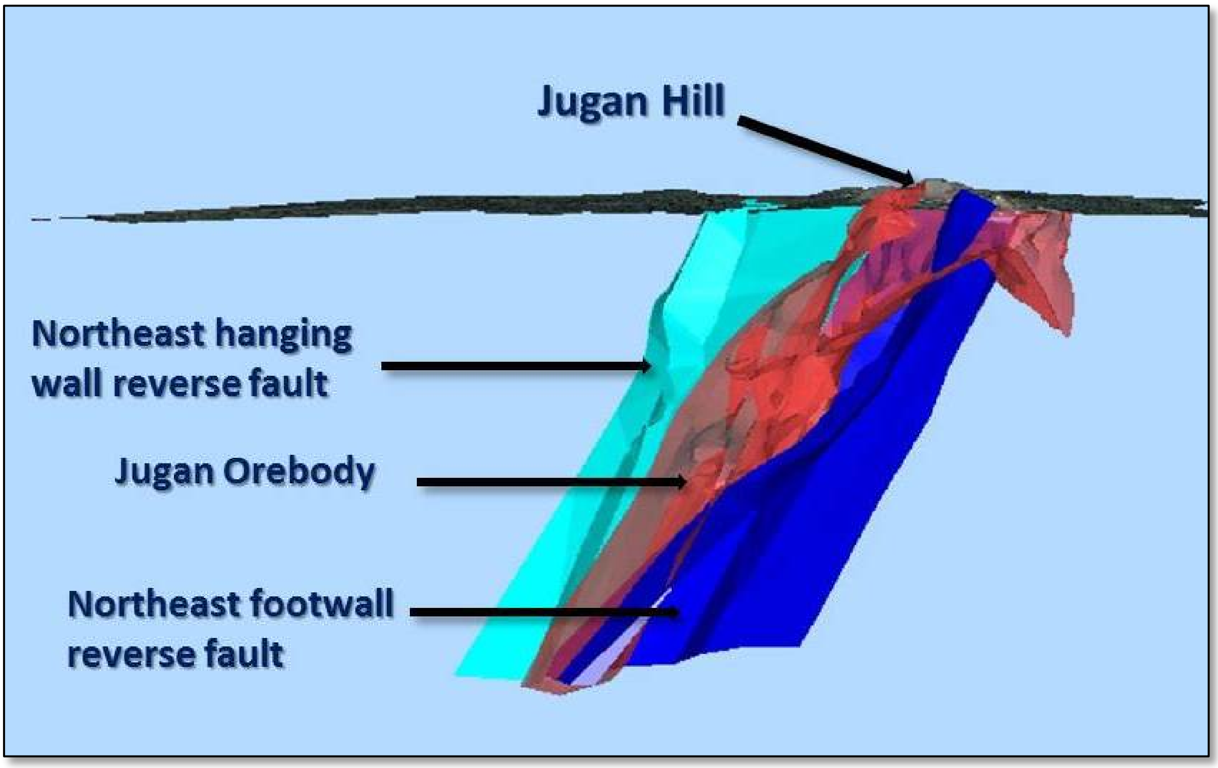


Figure 9-2 - 3D View of Jugan Orebody & NE Faults looking NW

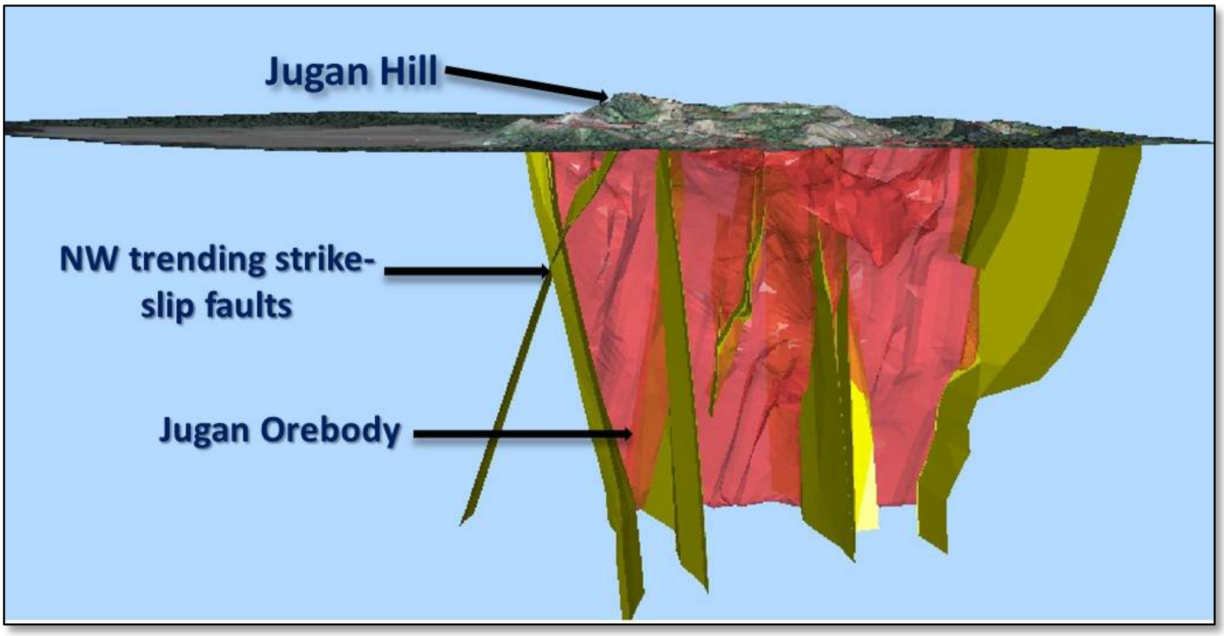


Figure 9-3 - 3D View of Jugan Orebody & NW Faults looking NE

9.2.1.2. Jugan Exploration Programme

9.2.1.2.1. General

In light of the positive results from the resource drilling programme and past reviews of the exploration potential of the Pedawan Formation surrounding Jugan NBG embarked on a combined programme of soil geochemical sampling both detailed and regional ridge and spur to provide a geochemical basis for follow-up geophysics particularly 3D Induced Polarisation (IP) along with collection of soil/subcrop for spectral analysis (Hychips) to delineate coincident geochemical anomalies and hydrothermal alteration minerals associated with "Carlin-style" or SHRGD deposits analogous to Jugan.

9.2.1.2.2. Soil Sampling

Molujin, 2013 has provided an analysis of the soil sampling programme. The programme had two (2) components, firstly a close spaced 25 metre by 25 metre grid based programme over the surface expression of the Jugan deposit and secondly, a more regional ridge and spur programme with samples collected at 50 metre centres and an approximate 200 metre separation between lines. The area covered for both surveys is seen on *Figure 9-4: Soil Sample Location Map* below.

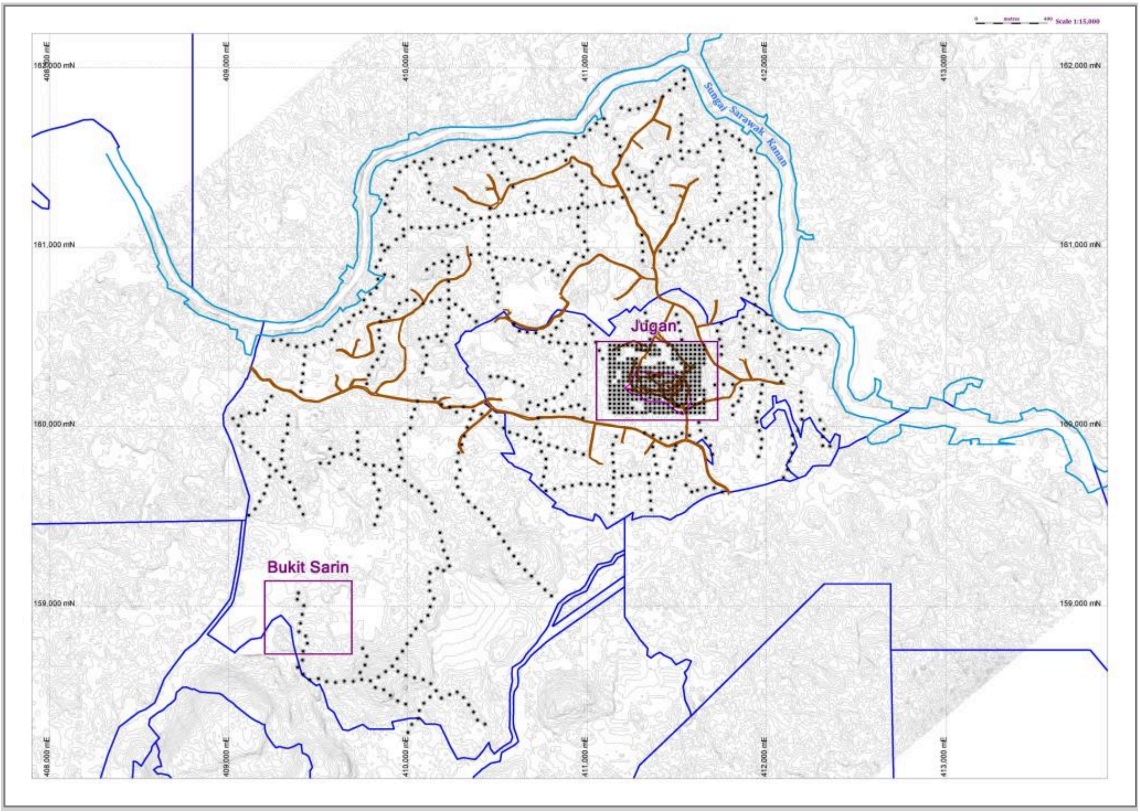


Figure 9-4: Soil Sample Location Map

Sampling was conducted using hand auger and most of the samples were collected from the C-horizon, however when the upper horizons are thick (more than 3 metres), the samples were then collected from the B horizon. There were some three hundred and three (303) samples collected on the Jugan Grid and a further six hundred and sixty-nine (669) samples collected during the ridge and spur phase of the programme.

All samples were sent to the SGS laboratory in Bau for the sample preparation and for the Au analysis (Fire assays, AAS, 50g). The ICP analysis of 26 pathfinder elements (Ag, As, Sb, Cu, Pb, Zn, Al, Ba, Bi, Ca, Cd, Co, Cr, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, Ti, V, S, W, and Tl) was carried out in SGS Port Klang, West Malaysia or SGS Perth, Australia.

The data was analysed statistically to determine threshold and anomalous values and correlation of elements associated with Jugan style mineralisation.

Five (5) elements have good correlation coefficients with gold (Au). They are arsenic (As), antimony (Sb), bismuth (Bi), sulphur (S) and thallium (Tl). Statistical parameters for these elements are tabulated in *Table 9-3: Statistical Parameters of Selected Elements in the Soil Samples*.

	Au	As	Sb	Bi	S	Tl
Minimum	0.000	1	1	2.5	15	0.05
1st Quartile (25%) – Q1	0.000	10	1	2.5	55	0.05
2nd Quartile (50%) – Q2	0.004	19	4	2.5	105	0.05
3rd Quartile (75%) – Q3	0.017	91	13	2.5	700	0.1

	Au	As	Sb	Bi	S	Tl
Interquartile Range – IQR	0.017	81	12	0	645	0.05
Q3+1.5IQR	0.043	212.5	31	2.5	1667.5	0.175
Q3+3IQR	0.068	334	49	2.5	2635	0.25
Maximum	11.528	25900.0	815	24.0	63500	7.50
Mean	0.155	1018	14	3.2	1631	0.19
Median	0.004	19	4	2.5	105	0.05
Mode	0.000	10	1	2.5	40	0.05
Standard Deviation	0.712	3198	44	2.4	4933	0.49
Sample Variance	0.507	10226072	1929	5.7	24338273	0.24
Kurtosis	109.567	22	214	19.0	44	84.18
Skewness	9.184	4	13	4.0	6	7.69
Range	11.528	25899	814	21.5	63485	7.45
Count	972	972	972	972	972	972

Table 9-3: Statistical Parameters of Selected Elements in the Soil Samples

From the EDA analysis the selected elements anomalies were classified in *Table 9-4: Geochemical Anomaly Classification* as follows:

Element	Background			Threshold			Possible			Probable			Highly		
Au (ppm)	0.000	-	0.004	0.004	-	0.017	0.017	-	0.043	0.043	-	0.068	0.068	-	12.528
As (ppm)	0.0	-	19.0	19.0	-	91.0	91.0	-	212.5	212.5	-	334.0	334.0	-	25901.0
Sb (ppm)	0	-	4	4	-	13	13	-	31	31	-	49	49	-	816
Bi (ppm)	0.0	-	2.5	2.5	-	5.0	5.0	-	9.0	9.0	-	11.0	11.0	-	25.0
S (ppm)	0.0	-	105.0	105.0	-	700.0	700.0	-	1667.5	1667.5	-	2635.0	2635.0	-	63501.0
Tl (ppm)	0.000	-	0.050	0.050	-	0.100	0.100	-	0.175	0.175	-	0.250	0.250	-	8.500

Table 9-4: Geochemical Anomaly Classification

The distribution of gold (Au), arsenic (As), antimony (Sb), bismuth (Bi), sulphur (S) and thallium (Tl) in the soil are shown respectively in the figures below. Jugan Hill (main Jugan deposit) shows good correlations of all the elements mentioned above. Bukit Sarin, the other known near surface gold mineralised area has the association of Au-As-Sb and Tl.

Figure 9-5: Au Anomaly in Jugan Grid Soil Sampling to *Figure 9-10: S Anomaly in Jugan Grid Soil Sampling* shows the geochemical distribution of anomalies of gold and elements with good positive correlation to mineralisation over the Jugan gridded area.

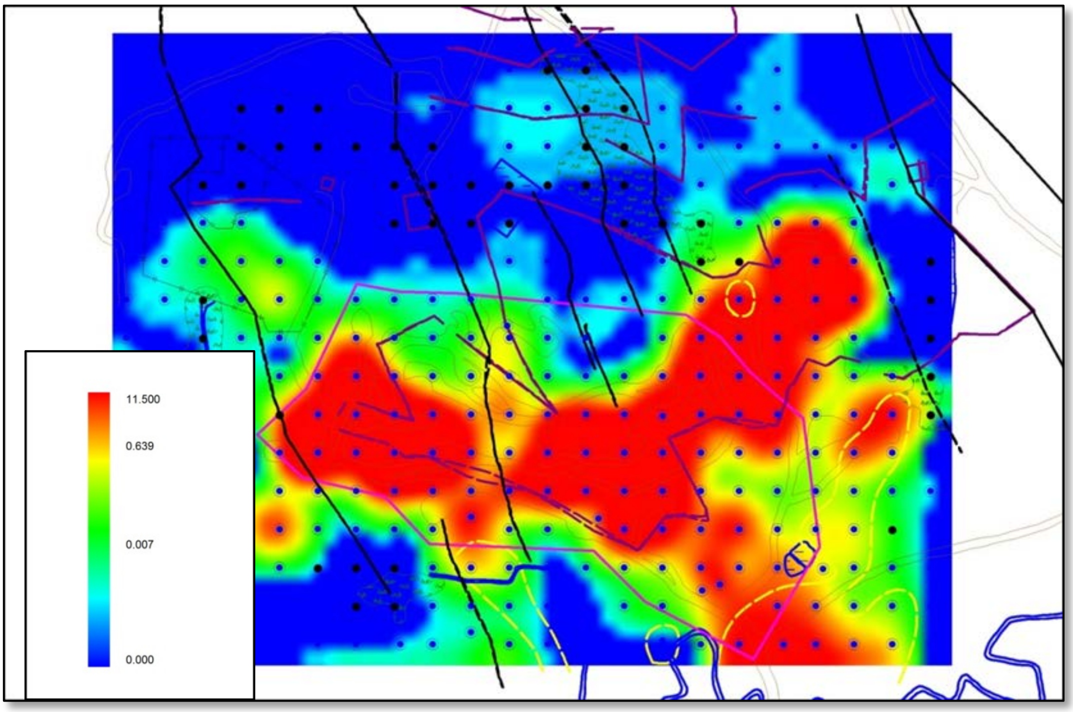


Figure 9-5: Au Anomaly in Jugan Grid Soil Sampling

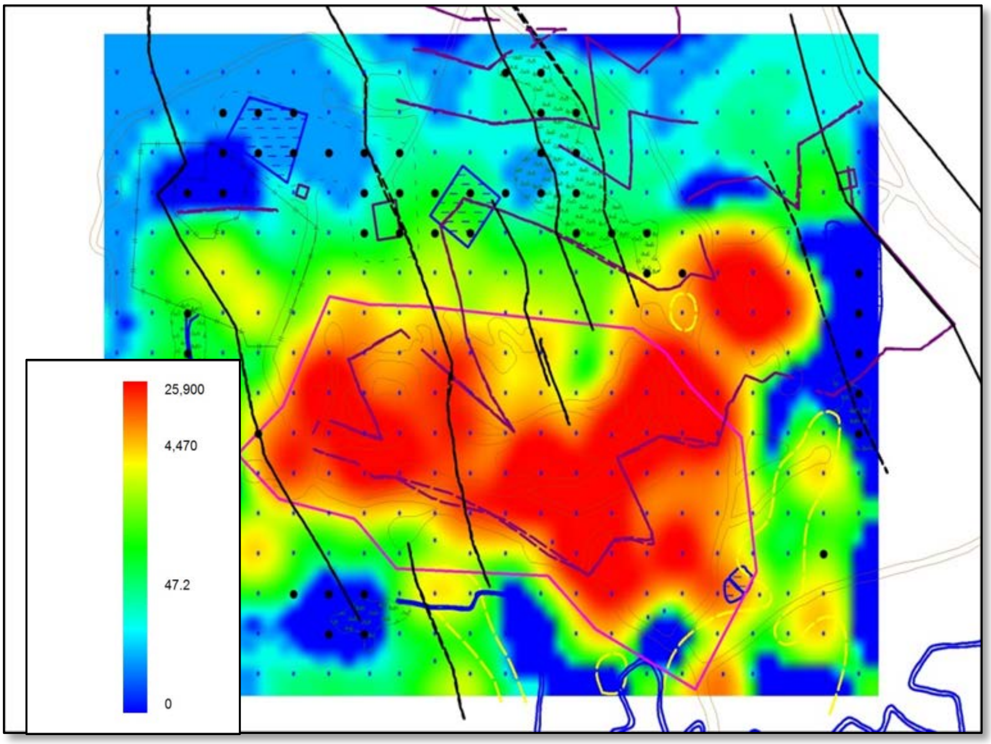


Figure 9-6: As Anomaly in Jugan Grid Soil Sampling

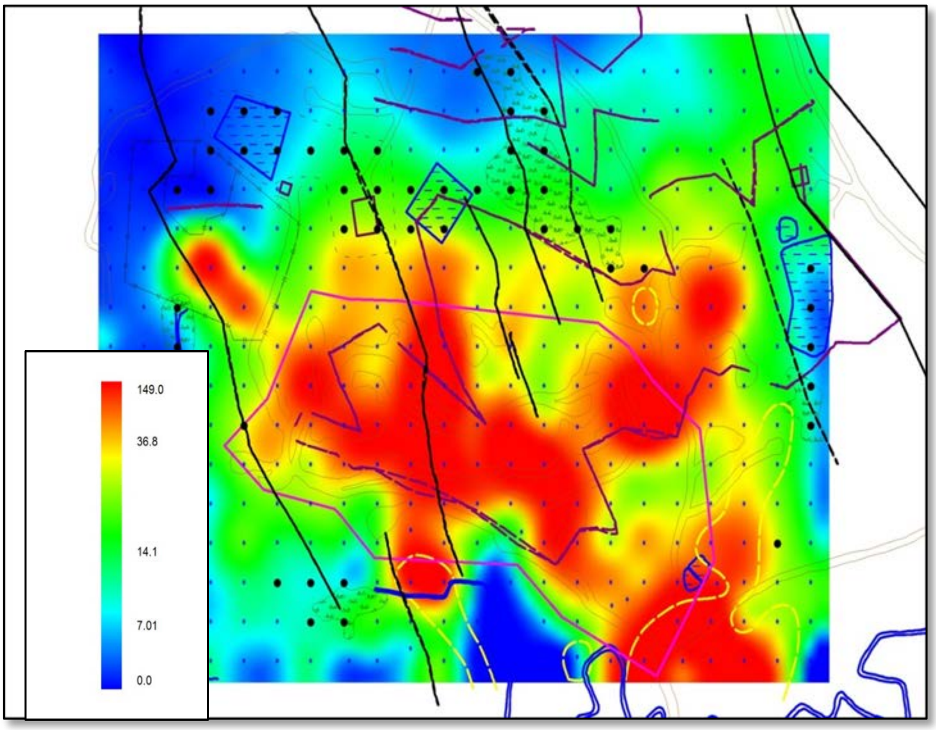


Figure 9-7: Sb Anomaly in Jugan Grid Soil Sampling

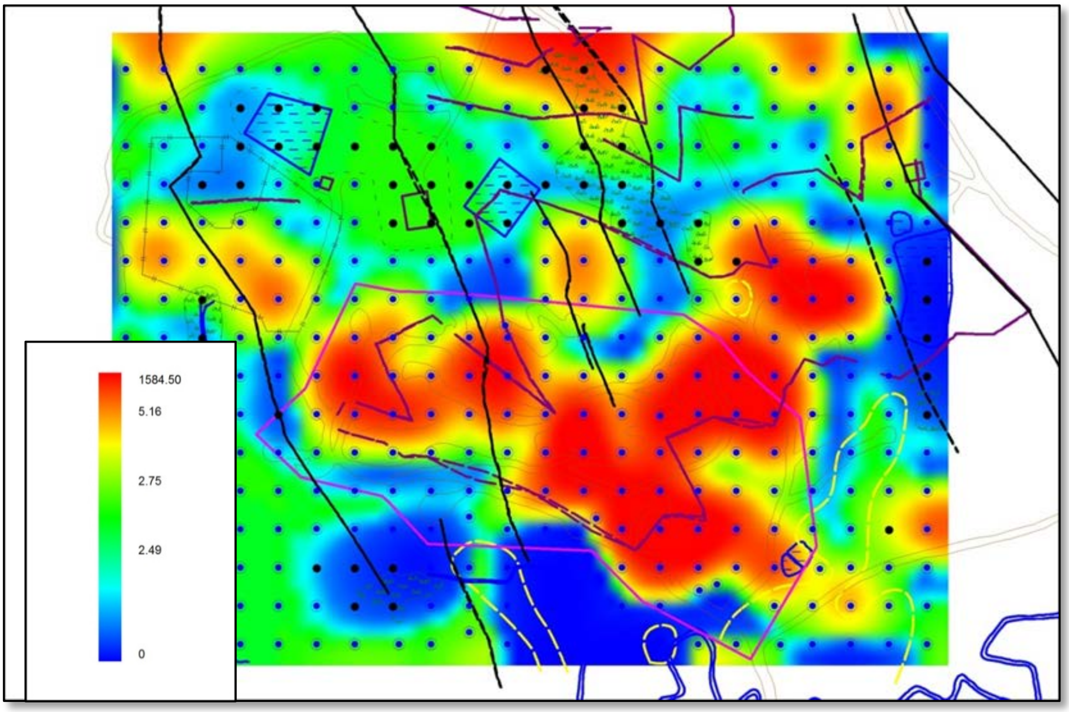


Figure 9-8: Bi Anomaly in Jugan Grid Soil Sampling

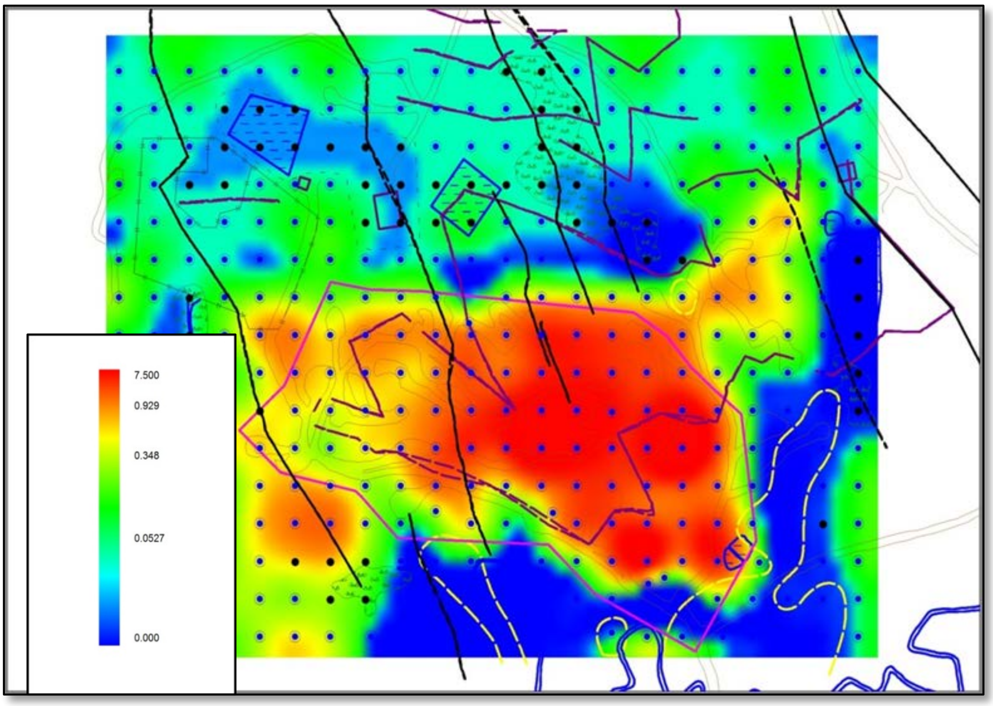


Figure 9-9: Tl Anomaly in Jugan Grid Soil Sampling

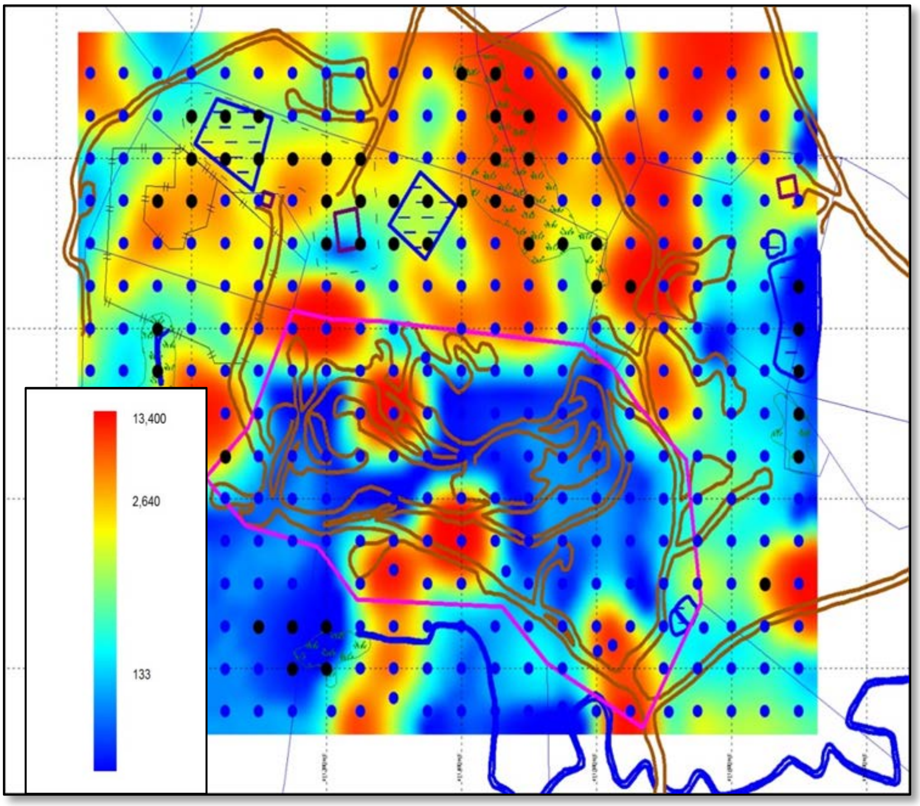


Figure 9-10: S Anomaly in Jugan Grid Soil Sampling

Data from the ridge and spur soil samples were amalgamated and analysed together with the Jugan grid soil samples. Some twenty-nine (29) gold anomalies were defined, including some small single point anomalies.

Some of the single point anomalies occur near the Sarawak River and don't appear to have other pathfinder elements associated. It is possible some of these maybe from an alluvial source but need to be ground truthed before they can be discounted.

However, of the twenty-nine (29) gold anomalies the majority have pathfinder element anomalies associated and subject to ground follow mapping can be regarded as representing geochemical halos associated with gold mineralisation.

The multi-element anomaly at Jugan Hill area was expected as it is a known gold mineralised area. Additionally, several larger geochemical targets have been identified. Firstly, at Bukit Sarin a far more extensive geochemical anomaly has been delineated to the South of the area indicated from the previously drilled area by Menzies, where a small gold resource has been identified. *Figure 9-11: Anomalous Gold in Soil Samples (red/Magenta) South of Bukit Sarin Drillholes* shows Bukit Sarin, the area drilled and the multi-element geochemical data.

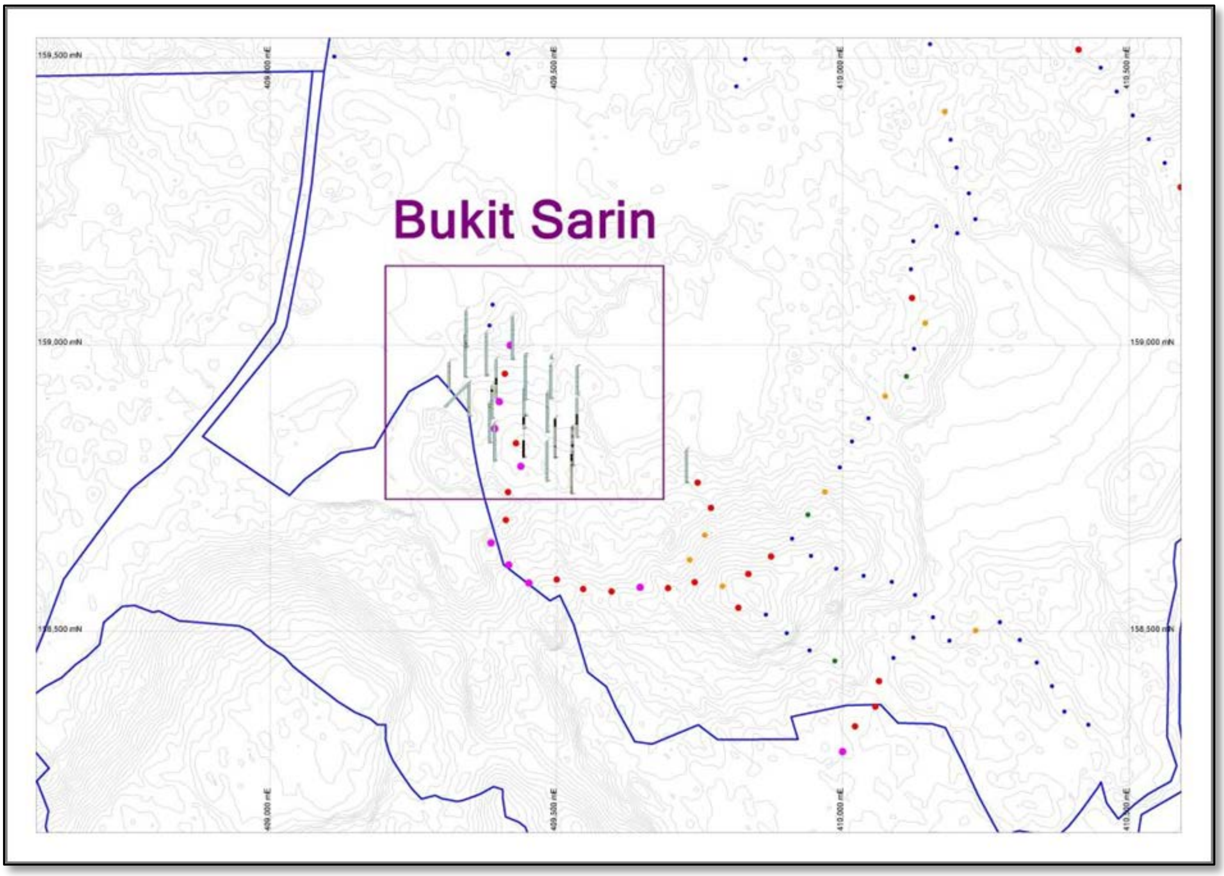


Figure 9-11: Anomalous Gold in Soil Samples (red/Magenta) South of Bukit Sarin Drillholes

Secondly, two other multi-element anomalous areas (Anomaly 1 and Anomaly 2), as shown in *Figure 9-12: Overlapping of Multi-Element Anomalies,* were also identified during this work programme. Anomaly 1 is located SW of Jugan midway to Bukit Sarin, where there is an association of Au-As-Sb and several smaller Au-Sb and Au-S anomalies.

Anomaly 2 is small area located ESE of the Jugan Hill area; it consists of small anomalies of Au-As-Sb-S and Au-Sb-Tl.

Anomalies 1, 2 and Bukit Sarin require in fill grid based soil sampling as a matter of priority while the other point sample anomalies need ground truthing and follow-up geochemical sampling to understand their significance.

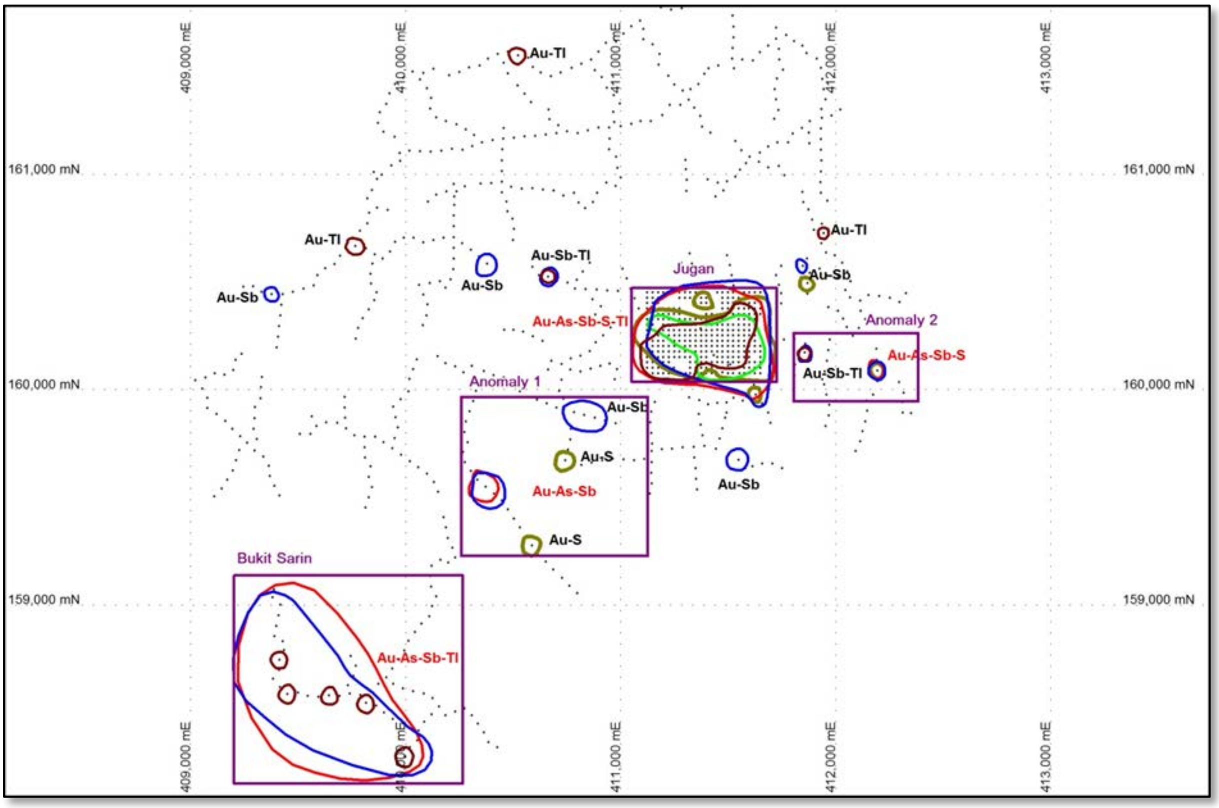


Figure 9-12: Overlapping of Multi-Element Anomalies

9.2.1.2.3. Geophysics – 3D IP Survey

The gold mineralisation at Jugan is associated with disseminated sulphides and weak silicification along with carbonate vein networks and stockworks. NBG and their consultants decided that a 3D offset pole-dipole induced polarisation (IP) survey over Jugan and surrounds was the best geophysical technique to characterize the chargeability and resistivity response of a known orebody, and determine any other areas either near surface or at depth with similar response to Jugan that could constitute extensions of Jugan or of new Jugan-style orebodies.

The survey was contracted to Planetary Geophysics from Toowoomba, Queensland, Australia and supervised by Consultant Geophysicist Paul Vidanovich of Auckland, New Zealand. The modelling, interpretation and reporting were completed by Paul Vidanovich with contributions from Resource Potentials and Tooronga Resources Pty Ltd of Western Australia.

Figure 9-13: Final Placement of Transmitter & Receiver Electrodes & 2D Pole-Dipoles shows the grid and electrode layout for the survey in relation to the Jugan Hill deposit. The green circles indicate the receiver electrode locations, whilst the red squares indicate the transmitter electrode locations. The four (4) lines of 2D pole-dipole are labelled in the diagram. The magenta outline indicates the surface expression of mineralisation at Jugan Hill.

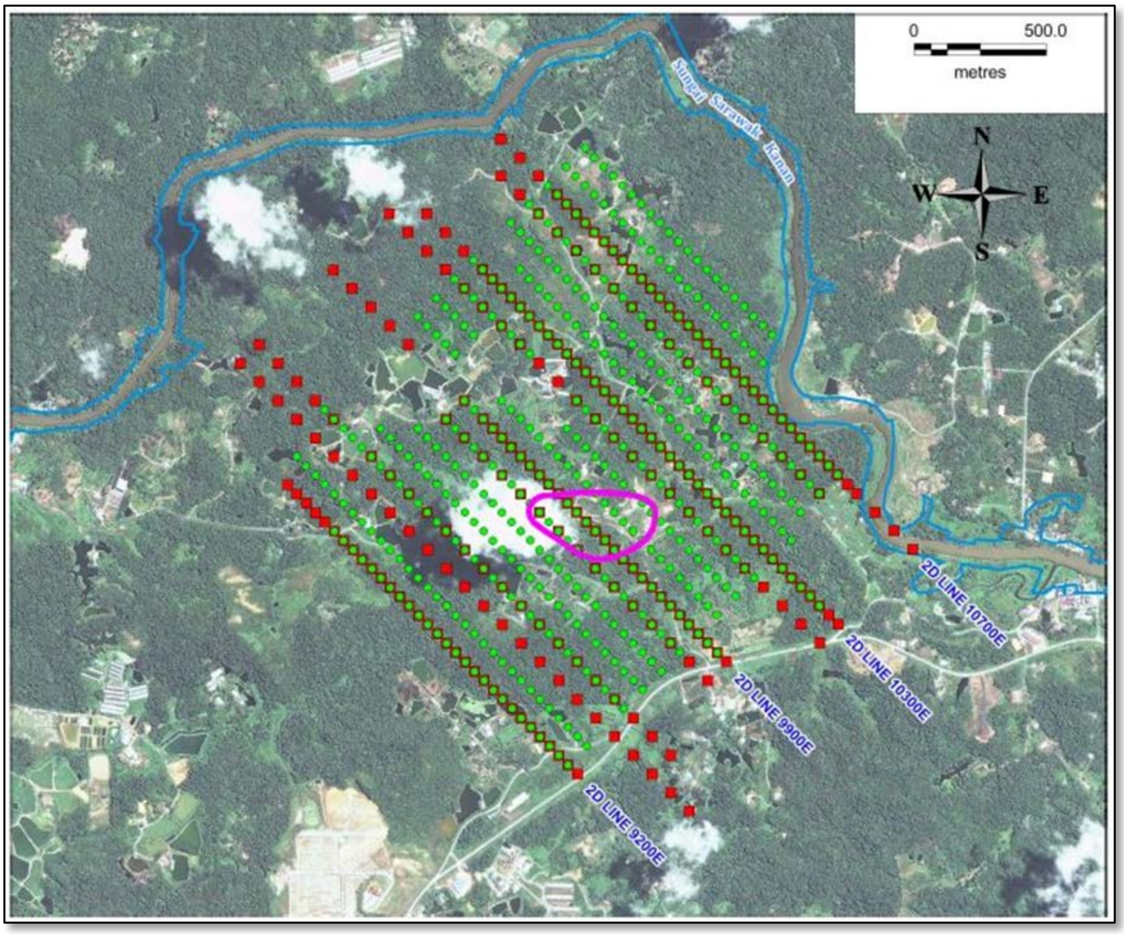


Figure 9-13: Final Placement of Transmitter & Receiver Electrodes & 2D Pole-Dipoles

As a check on the 3D-IP and to provide detail where access was limited for the 3D array, 2D pole-dipole profiles were also carried out.

The results were then interpolated with existing Dighem and aeromagnetic survey data from the 1990's to see if features in the IP data were resolved and if it showed potential targets outside the IP survey area.

The 3D IP survey has proved successful in delineating the known mineralisation at Jugan Hill down to at least 300 metres depth and identifying nine (9) new targets for further investigation.

Figure 9-14: Chargeability Polygons over the Magnetic Analytical Signal shows the surface projection of these overlain on analytical signal of the aerial magnetics. Intrusives stand out as areas with high magnetic gradient and are common.

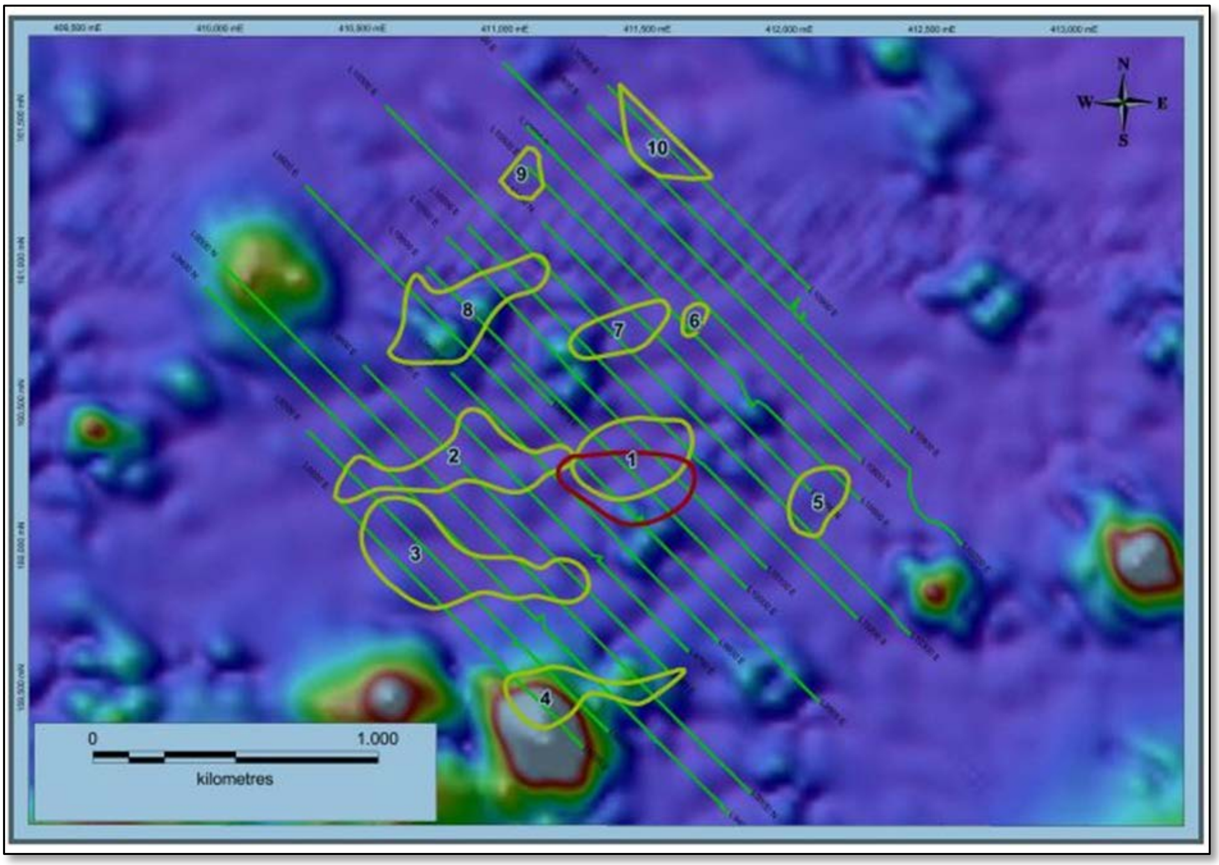


Figure 9-14: Chargeability Polygons over the Magnetic Analytical Signal

Inversion processing proved successful in providing consistent modelling in both the 3D and 2D data. Inversion from two (2) separate groups using different code yielded very similar and consistent results.

The ore zone at Jugan Hill was found to have a weak chargeability up to 3-4 mS, and this was associated with the disseminated mineralisation there. The resistivity too, proved to be a low 20-40 Ohm-m, but in the context of the very conductive shale this was more than sufficient to define the ore body. The raised resistivity is caused by weak silicification and carbonate (mainly ankerite) veining of the host shales.

Figure 9-15: Resistivity Response over Known Orebody at Jugan Hill shows the 20 to 40 Ohm-m surfaces at Jugan. It can be seen that the 20 Ohm-m surface closely coincides with the drill derived geological model. The yellow surface is 40 Ohm-m, lemon for the 30 Ohm-m and green is the 20 Ohm-m.

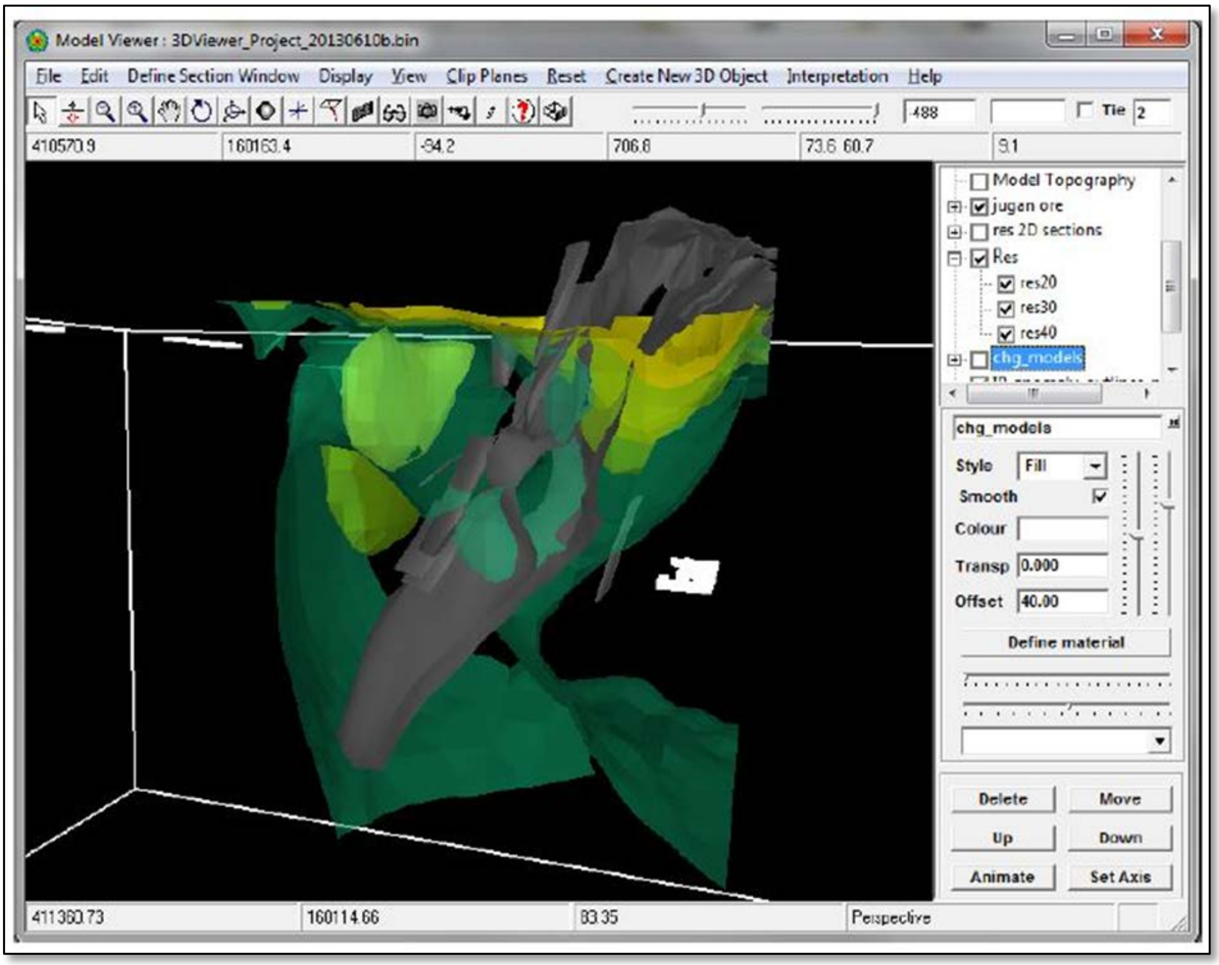


Figure 9-15: Resistivity Response over Known Orebody at Jugan Hill

Using the extent of the known ore body at Jugan Hill, reliable information down to approximately 300m below the surface was achieved with the inversion modelling.

There appear to be several sub-parallel East-West trending chargeability zones crossing the IP grid. Jugan Hill sits on one of these, with extensions to the East and West. Two (2) more zones lie to the South and at least one (1) more lies to the North. Where there is coverage these are supported generally by the soil geochemistry.

It is noted that mineralisation at Jugan Hill has a sigmoidal geometry; hence “en echelon” zones may be found in the footwall and/or along strike.

The nine (9) new target zones require follow up mapping, soil sampling and full integration with the existing data bases to generate ranked and robust drill targets.

In a more regional sense, *Figure 9-16: Structural Overview & Anomaly Map Showing EM Resistors* shows the observed chargeability anomalies in a broader structural context, and observed structural lineaments. Figure shows the EM resistors (in yellow), particularly those coincident and extending East from the Jugan Hill orebody, and the identified IP chargeability anomalies (in magenta). The cyan rectangle is the IP grid position and coverage.

Once the main lineaments were identified, the chargeability polygons were overlain to view the relationships. The East-West trends in the chargeability anomalies that are reflected in the soil geochemistry most closely follow the major fault just to the south of Jugan Hill, suggesting structures with this orientation (*D1 in Schuh*) have a role to play in creating foci for mineralisation and fluid flow.

Resistivity features that are coincident with the chargeability anomalies are outlined in yellow and these extend the possible silicified bodies a further 1-2km West of the IP grid. They also track the major fault to the South very closely. Similar resistive zones are observed on a sub-parallel structure to the North, starting on Anomalies 6, 7, and 8, and extending further West.

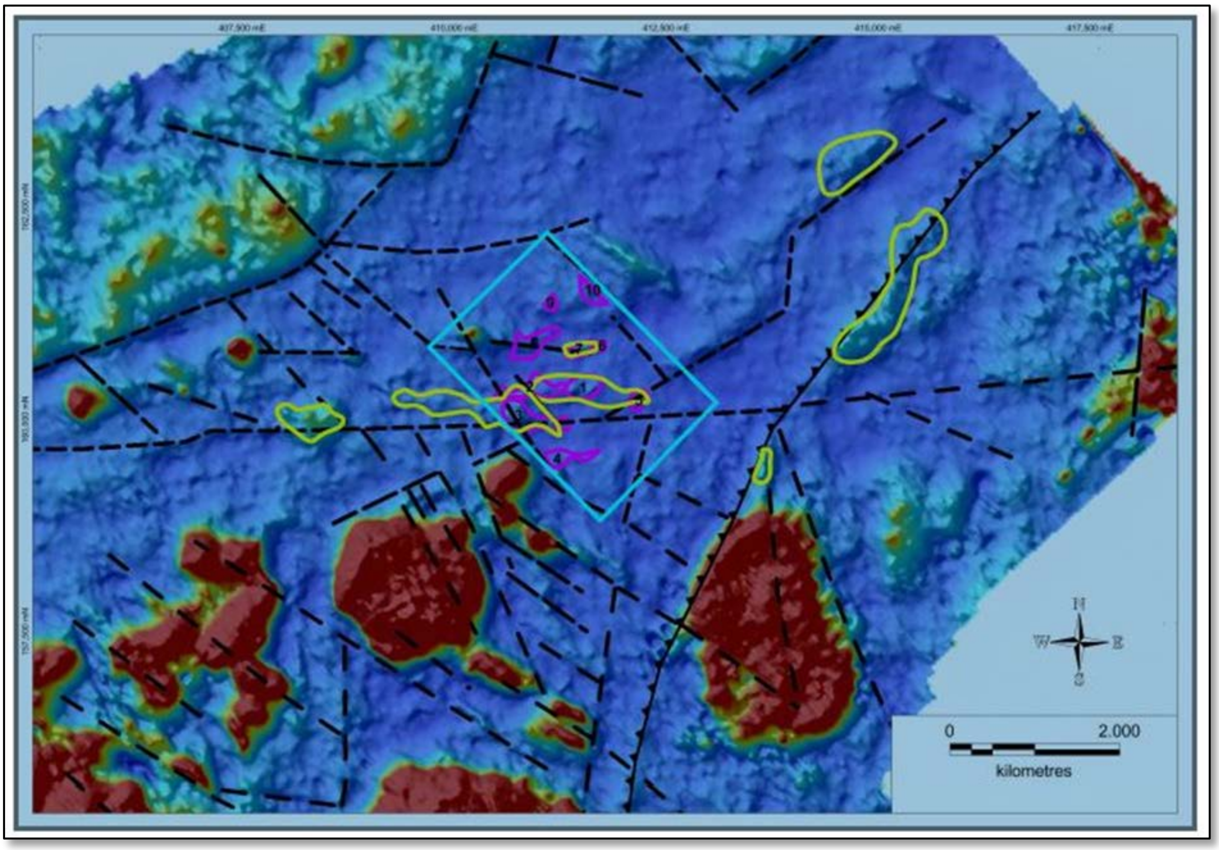


Figure 9-16: Structural Overview & Anomaly Map Showing EM Resistors

9.2.2. Bekajang Sector

9.2.2.1. Bukit Young Pit

The Bukit Young Gold Pit (BYG Pit) is adjacent to the old mine office and plant site. The pit was mined until September 1992, prior to the redevelopment of the Tai Parit deposit, and according to Bukit Young mine records had produced some 440,926 tonnes at a recovered grade of 4.51 g/t Au. They noted in their records (as at March 1995) that ore remains in the SW edge of the pit. The deepest level of mining was to 60 metre depth.

The deposit is developed in the Eastern side of the NNE trending Krian Fault where it abuts on the Western side against up thrown blocks of Krian Sandstone and adjoining felsic porphyry intrusives. Ore types are similar to Tai Parit, with siliceous jasperoidal breccias, ferruginous auriferous clay and mangano-calcite veining with sulphide coatings.

Evaluation of old drill sections and level plans confirms that several areas of ore were not mined. Several of these holes have ore grade mineralization. For example, BYG drillhole DDH102-36 intersected 26.95 metres from surface grading 6.51 g/t Au.

NBG drilled a shallow reconnaissance hole BYWDDH-01 next to this hole. This confirmed the presence of strong gold mineralization, (6m at 7.62 g/t Au, from 24m depth), although core recovery was poor and is an indication only of the tenor of gold grade.

This mineralised trend appears to plunge South and dip toward the Krian Fault. From the Bukit Young Pit and trending Southwest on the trace of the Krian Fault Zone for 500 metres there are a number of ore grade intersections observed in old drill hole data. Most of these are shallow and have had little follow up. For example, KRRC-21 intersected 19.8 metres @ 7.34 g/t Au from 12.2 metres depth.

Through 2011 to early 2012 NBG undertook infill drilling and extended drilling to depth and along strike in and around the BYG pit. Several of these holes also tested to the west of the Bukit Young Pit where outcrops of altered Krian Sandstone and dacite porphyry intrusives occur. Little historic drilling had been completed in this area and comprised only shallow Winkie holes. There was some indication of gold mineralization historically so the NBG resource drill programme explored this area as well. Significant results for the NBG drill programmes at BYG are tabulated below in *Table 9-5: BYG Pit Significant Intersections from 2011 Drill Programme*.

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
BYDDH-02	1.90	5.50	3.60	1.06
BYDDH-02	7.70	10.70	3.00	0.50
BYDDH-02	15.50	55.50	40.00	4.79
BYDDH-02 - incl	15.50	17.00	1.50	5.47
BYDDH-02 - and	28.80	31.00	2.20	16.51
BYDDH-02 - and	37.35	55.50	18.15	7.30
BYDDH-02 - with	44.00	47.00	3.00	15.20
BYDDH-03	20.00	21.00	1.00	0.64
BYDDH-03	27.00	28.00	1.00	1.27
BYDDH-03	31.00	34.00	3.00	1.24
BYDDH-04	0.00	0.70	0.70	1.52
BYDDH-04	4.70	8.80	4.10	1.68
BYDDH-04	34.65	82.00	47.35	4.53
BYDDH-04 - incl	55.00	69.00	14.00	7.54
BYDDH-04 - with	62.00	65.00	3.00	21.50
BYDDH-04	86.70	89.30	2.60	2.41
BYDDH-04	117.00	118.25	1.25	3.54
BYDDH-04	136.00	137.00	1.00	1.61

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
BYDDH-05	218.40	219.40	1.00	0.50
BYDDH-06	1.00	2.00	1.00	1.17
BYDDH-06	6.00	9.00	3.00	0.59
BYDDH-06	11.00	13.00	2.00	1.28
BYDDH-06	19.00	31.00	12.00	3.35
BYDDH-06 - incl	20.00	21.00	1.00	24.60
BYDDH-06	36.00	42.00	6.00	8.56
BYDDH-06 - incl	36.85	39.10	2.25	21.69
BYDDH-06	68.50	71.00	2.50	3.26
BYDDH-06	88.00	89.00	1.00	0.53
BYDDH-07	0.00	5.00	5.00	1.90
BYDDH-07	10.40	11.70	1.30	0.68
BYDDH-07	145.00	146.00	1.00	0.60
BYDDH-07	150.00	151.00	1.00	3.80
BYDDH-07	161.00	165.30	4.30	1.57
BYDDH-07	192.00	193.20	1.20	7.18
BYDDH-08	83.00	85.00	2.00	0.84
BYDDH-09	0.00	5.30	5.30	1.02
BYDDH-09	10.00	11.00	1.00	0.67
BYDDH-09	15.60	18.70	3.10	0.89
BYDDH-09	45.00	46.00	1.00	5.52
BYDDH-09	62.50	65.50	3.00	4.88
BYDDH-09 - incl	63.50	64.50	1.00	10.30
BYDDH-09	75.00	83.60	8.60	24.07
BYDDH-09 - incl	76.30	80.20	3.90	48.47
BYDDH-09	95.95	97	1.05	0.94
BYDDH-10	1.00	3.90	2.90	1.03
BYDDH-10	8.90	9.90	1.00	0.51
BYDDH-10	10.90	11.90	1.00	0.54
BYDDH-10	51.10	55.10	4.00	0.80
BYDDH-10	60.10	66.65	6.55	2.10
BYDDH-10 - incl	63.00	64.00	1.00	9.70
BYDDH-10	69.00	70.00	1.00	0.92
BYDDH-11	1.20	2.50	1.30	1.26
BYDDH-12	21.45	24.10	2.65	0.68
BYDDH-12	26.15	26.40	0.25	0.97
BYDDH-12	54.25	59.70	5.45	1.31
BYDDH-12	116.00	131.90	15.90	7.35
BYDDH-12 - incl	123.00	128.00	5.00	18.16
BYDDH-12	143.00	144.20	1.20	0.75
BYDDH-13	0.00	4.10	4.10	0.79
BYDDH-13	5.00	5.30	0.30	2.65
BYDDH-13	8.70	9.00	0.30	1.50
BYDDH-13	14.30	14.80	0.50	5.82
BYDDH-13	15.60	16.70	1.10	5.98

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
BYDDH-13	27.10	27.40	0.30	2.51
BYDDH-13	34.80	48.30	13.50	1.54
BYDDH-13	58.00	60.00	2.00	0.81
BYDDH-13	63.00	85.00	22.00	0.83
BYDDH-13	91.00	93.00	2.00	0.74
BYDDH-13	97.00	98.00	1.00	0.66
BYDDH-13	105.00	106.00	1.00	0.58
BYDDH-13	115.00	116.00	1.00	0.51
BYDDH-13	126.20	127.00	0.80	0.64
BYDDH-13	192.00	193.00	1.00	0.52
BYDDH-14	1.35	2.60	1.25	0.69
BYDDH-15	27.30	30.50	3.20	0.78
BYDDH-15	79.70	80.50	0.80	0.92
BYDDH-15	110.50	110.80	0.30	10.40
BYDDH-15	141.00	143.00	2.00	0.80
BYDDH-15	152.00	161.00	9.00	0.60
BYDDH-16	25.00	26.00	1.00	1.13
BYDDH-16	32.80	34.50	1.70	2.29
BYDDH-17	6.90	7.55	0.65	1.04
BYDDH-17	105.60	116.60	11.00	11.71
BYDDH-17	128.55	129.00	0.45	0.51
BYDDH-17	137.00	140.40	3.40	4.24
BYDDH-17	177.00	178.00	1.00	1.27
BYDDH-18	0.00	0.60	0.60	2.38
BYDDH-18	56.00	56.60	0.60	1.43
BYDDH-18	65.70	68.00	2.30	3.28
BYDDH-18	101.20	102.00	0.80	2.95
BYDDH-19	0.00	1.00	1.00	1.35
BYDDH-20	13.60	16.80	3.20	3.16
BYDDH-20	23.00	30.10	7.10	2.83
BYDDH-20	37.40	38.80	1.40	0.94
BYDDH-20	102.00	109.00	7.00	0.57
BYDDH-20	162.00	165.00	3.00	10.25
BYDDH-21	3.00	4.00	1.00	1.23
BYDDH-21	6.70	13.20	6.50	1.74
BYDDH-21	37.00	41.70	4.70	8.14
BYDDH-21	55.65	56.95	1.30	1.75
BYDDH-21	59.60	60.70	1.10	1.24
BYDDH-22	0.00	1.50	1.50	1.70
BYDDH-23	5.00	12.00	7.00	1.35
BYDDH-23 - incl	10.00	12.00	2.00	3.49
BYDDH-24	0.80	4.00	3.20	1.06
BYDDH-24	9.00	12.00	3.00	0.78
BYDDH-25	1.00	9.30	8.30	0.65
BYDDH-25	1.00	9.30	8.30	0.65

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
BYDDH-26	0.00	3.00	3.00	3.80
BYDDH-26	10.00	16.00	6.00	1.15
BYDDH-26	20.00	26.20	6.20	1.15
BYDDH-26	104.00	105.00	1.00	1.20
BYDDH-26	132.30	133.40	1.10	0.64
BYDDH-26	136.00	137.00	1.00	0.52
BYDDH-26	157.00	158.00	1.00	1.04
BYDDH-26	168.35	169.75	1.40	4.25
BYDDH-27	0.00	1.00	1.00	0.97
BYDDH-27	10.30	11.00	0.70	3.47
BYDDH-27	14.00	16.00	2.00	1.10
BYDDH-27	20.00	20.90	0.90	1.84
BYDDH-27	102.00	103.00	1.00	4.07
BYDDH-28	1.00	5.00	4.00	0.96
BYDDH-28	8.00	15.00	7.00	1.60
BYDDH-28 - incl	8.00	9.00	1.00	7.35
BYDDH-28	17.10	23.90	6.80	1.30
BYDDH-28	60.00	61.00	1.00	0.61
BYDDH-28	72.50	73.50	1.00	2.51
BYDDH-28	86.10	93.00	6.90	0.58
BYDDH-28	98.00	99.00	1.00	1.24
BYDDH-28	109.00	109.70	0.70	0.51
BYDDH-29	3.80	4.90	1.10	0.93
BYDDH-29	26.70	36.50	9.80	1.01
BYDDH-29 - incl	26.70	29.80	3.10	1.51
BYDDH-29 - and	32.70	36.50	3.80	1.33
BYDDH-30	0.00	3.00	3.00	1.09
BYDDH-30	8.80	13.00	4.20	0.80
BYDDH-30	93.00	95.00	2.00	0.65
BYDDH-31	2.00	3.20	1.20	1.02
BYDDH-31	6.20	15.00	8.80	0.72
BYDDH-31	21.00	23.20	2.20	0.70
BYDDH-31	26.20	35.20	9.00	7.49
BYDDH-31 - incl	28.20	30.20	2.00	29.56
BYDDH-33	0.00	14.30	14.30	3.59
BYDDH-33 -incl	9.20	11.40	2.20	19.91
BYDDH-33 - and	13.20	14.30	1.10	3.30
BYDDH-34	29.00	30.00	1.00	1.41
BYDDH-34	72.30	73.00	0.70	3.21
BYDDH-34	113.00	115.00	2.00	2.35
BYDDH-34	161.00	172.00	11.00	0.84
BYDDH-35	1.00	2.00	1.00	0.97
BYDDH-36	4.40	8.50	4.10	4.28
BYDDH-36	124.80	127.00	2.20	2.27
BYDDH-37	0.00	1.00	1.00	1.37

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
BYDDH-37	6.00	7.00	1.00	1.37
BYDDH-38	29.60	30.50	0.90	1.31
BYDDH-38	33.00	38.00	5.00	0.82
BYDDH-38	50.00	51.00	1.00	1.28
BYDDH-38	114.30	117.00	2.70	2.10
BYDDH-38	127.80	131.00	3.20	3.07
BYDDH-38	147.00	157.00	10.00	1.16
BYDDH-38	164.00	175.00	11.00	1.06
BYDDH-38	185.00	200.00	15.00	0.85
BYDDH-39	0.00	4.00	4.00	3.97
BYDDH-39	63.00	65.00	2.00	0.94
BYDDH-39	89.00	90.00	1.00	0.80
BYDDH-39	121.85	122.45	0.60	0.83
BYDDH-39	183.00	218.00	35.00	0.69
BYDDH-39 - incl	186.00	199.00	13.00	1.10
BYDDH-39	223.00	224.00	1.00	1.42
BYDDH-39	270.00	273.00	3.00	0.95
BYDDH-39	275.90	280.00	4.10	2.00
BYDDH-39 - incl	278.00	280.00	2.00	3.61
BYDDH-39	293.00	295.00	2.00	1.01
BYDDH-40	0.00	0.90	0.90	1.55
BYDDH-40	34.00	35.00	1.00	0.80
BYDDH-40	50.00	51.00	1.00	0.80
BYDDH-41	1.00	2.00	1.00	0.58
BYDDH-41	12.00	13.00	1.00	1.05
BYDDH-41	26.00	38.00	12.00	0.76
BYDDH-41	42.00	45.30	3.30	6.35
BYDDH-41- incl	43.00	44.00	1.00	18.10
BYDDH-41	89.80	90.80	1.00	7.30
BYDDH-41	98.60	99.20	0.60	14.00
BYDDH-42	3.00	6.25	3.25	0.58
BYDDH-42	56.95	59.80	2.85	6.34
BYDDH-42	90.20	92.00	1.80	7.27
BYDDH-42	125.80	127.00	1.20	0.82

Table 9-5: BYG Pit Significant Intersections from 2011 Drill Programme

The results of this drill programme were interpolated with the previous historic drilling and modeled to give a new resource figure that upgraded the category for portions of the resource and increased the total resource. The resource now stands at 1.857 million tonnes at 2.02 g/t Au for 120,400 ounces Indicated and 3.328 million tonnes at 1.51 g/t Au for 168,800 ounces Inferred.

Figure 9-17: Geology and Drill Location Plan BYG Pit below shows the current drill pattern and geology of the BYG Pit. It is notable that some of the better grades were obtained from within the Krian Sandstone unit which is a basal member to the Bau Limestone. This is also true for the adjoining Tai Parit Deposit. The Krian Fault cuts through the pit and is up-thrown on the NW side exposing intrusives and Krian Formation. Serian Volcanics have also been intersected in this area where they are altered and with some brecciation. On the SE side of the Krian Fault predominantly Bau Limestone is exposed. A prime exploration target exists here at depth in the Krian Sandstone and Serian Volcanics where it is postulated that the high grade mineralisation seen to the NW should be repeated.

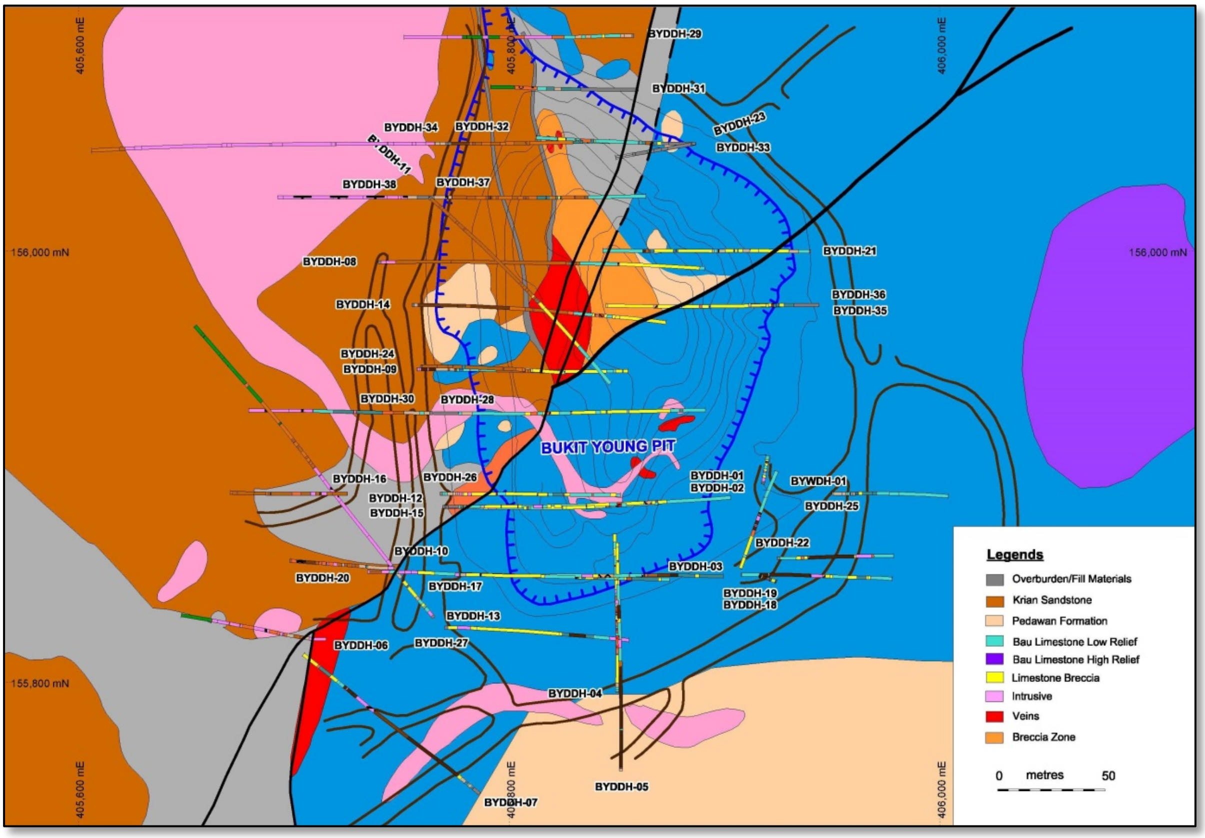


Figure 9-17: Geology and Drill Location Plan BYG Pit

9.2.2.2. Gunong Krian

The Gunong Krian prospect is located on a steep up faulted block of Bau Limestone approximately 750m SW of the BYG plant site.

Essentially the target at Krian is based on surface and underground expressions of quartz and calcite veining historically mined for antimony (Lucky Hill Mine) and gold and a deep source DIGHEM conductor representing a more massive mineralised vein/breccia zone at depth with the exposed mineralization representing the vertical expressions of the zone.

The veins are generally NW-SE mineralised structures, frequently vuggy and with comb quartz infillings. The resource potential here is discussed and included with that for the BYG Pit and the mineralised trends associated with the Krian and Johara Fault.

9.2.2.3. Karang Bila

The area known as Karang Bila lies approximately 1 kilometre East of the BYG Plant site and 500 metres NE of the BYG Tailings dam. There has been a total of 6,806 metres of RC drilling in fifty-four (54) drillholes recorded as having been drilled. There is a number of significant drill intersections recorded, including 4m at 14.6 g/t Au in drillhole KBRC48 from 52 metres depth. An Inferred Resource of 48,500 ounces gold has been delineated here.

The mineralised zone appears to trend SE and is flat lying. There seem to be several zones of mineralization and given the proximity of Bekajang are likely to be developed at the limestone shale contact and in parallel zones within the limestone.

The area is certainly prospective and should be evaluated further in conjunction with any work at Bekajang. One negative factor is the proximity of new housing estates, one of which has encroached on the SE corner of the mineralization.

9.2.2.4. Tai Parit

The Tai Parit deposit is immediately adjacent to Bau Township with the abandoned open pit now forming a recreational lake (Tasik Biru) for the town.

The Tai Parit Pit itself has recorded production of 700,000 troy ounces at an average grade of over 7 g/t Au from a body of silicified fault breccia aligned NNE-SSW on the Tai Parit Fault, the main controlling mineralised structure.

From examination of the extensive drillhole database for Tai Parit there is evidence that the mineralization continues at depth, particularly on a NW trending zone that intersected the Tai Parit Fault in the pit.

The deposit, while being apparently controlled by the Tai Parit Fault, is also in close proximity to high level felsic porphyry intrusives with typical quartz-sericite-pyrite alteration. Host rocks also include the Krian Sandstone, Bau Limestone and Pedawan Shale.

Extensions to the mineralization along strike on the Tai Parit Fault and on the NW trend are not well tested by drilling. Good potential exists for extensions to the Tai Parit gold deposit.

9.2.2.5. Bekajang

The Bekajang area lies immediately SE of the old Bukit Young processing plant and has been traced for around 1,500 metres SE and approximately 700 metres across strike. Several small deposits are known to occur at the shale/limestone contact and are generally shallow dipping features with mineralization developed in siliceous breccias within the shales on the contacts between shale and limestone. One of these, Gumbang, was mined by Gladioli in the 1990's to a limited degree and was located at the shale limestone contact.

In addition, there are a number of NW-SE faults mapped or interpreted with mineralization indicated from drillholes. These have been interpreted as possible feeders to the lateral mineralization and present targets themselves.

Surface exposure of the mineralization is scant. The bulk of the prospect is masked by the Bukit Young tailings impoundment. This is believed to have infilled early open pits for which there are no production records or survey plans.

An Inferred Resource in two (2) deposits at Bekajang has been outlined, called in this document Bekajang North and Bekajang South. Together they comprise 3.544 million tonnes at a grade of 1.86 g/t Au for 211,500 ounces.

Drilling by NBG in the SE corner of the Bekajang South resource area in 2007 intersected a mineralised fault zone with economic grades. This zone appears to be a relatively confined fault angle wedge.

Significant results from NBG's drilling at Bekajang up until 2008 are listed in *Table 9-6: Significant Drill Results for Bekajang to 2008* below.

Hole No	From (m)	To (m)	Interval (m)	Au Grade (g/t)
BKDDH-01	17.00	23.67	6.67	2.21
BKDDH-02	6.00	9.10	3.10	8.10
BKDDH-03	6.00	31.90	25.90	8.12
BKDDH-04	15.00	20.10	5.10	2.00
	38.00	40.85	2.85	2.44
BKDDH-06	6.00	25.75	19.75	10.46
BKDDH-08	13.00	21.40	8.40	16.90

Table 9-6: Significant Drill Results for Bekajang to 2008

There are few drill holes in the area of the tailings dam and only one (1) angle hole that attempted to drill beneath the dam.

The potential for discovering resources beneath the Bekajang TSF has been evaluated. The pits that are now in-filled with tailings were mined by the Borneo Company in the 19th and early 20th Centuries. They were constrained by the refractory nature of the ore and it is probable that any ore that was not free milling or oxidised was not mined.

In addition, there are a number of intrusives in contact with the limestone and shale adjacent to Bekajang. Some of the historic drillholes show low but potentially economic gold grades in the intrusives.

NBG has re-evaluated the potential here and in particular modeled the DIGHEM data flown by Menzies in the 1990's. This showed the presence of a large conductor beneath the Bekajang TSF that had not been tested by previous drilling. There is also an indication based on the geometry of the anomaly that is could be tied to some of the shallow mineralisation in the Bekajang North resource area.

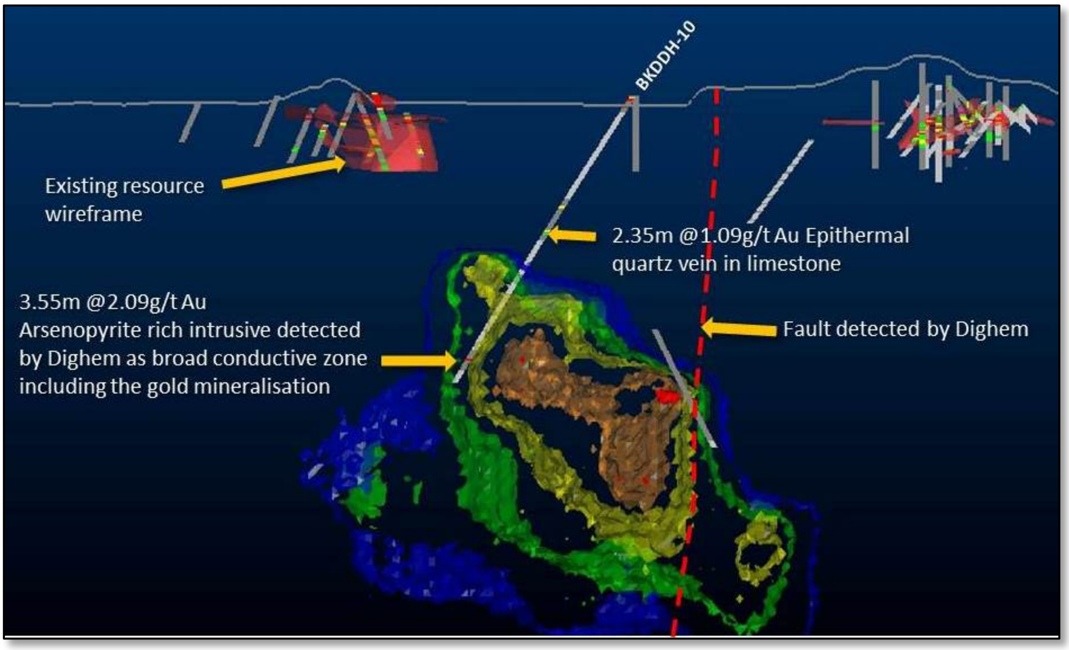


Figure 9-18 - DIGHEM Conductor Iso-Surfaces & Exploration Drillholes

Two (2) drillholes were drilled to test the DIGHEM conductor and its mineral potential. Both drillholes intersected gold bearing tailings in the upper 6 metres to 14 metres.

The first hole BKDDH-10 intersected a 2.35 metres wide quartz vein in limestone. This vein was vuggy with bladed quartz after carbonate indicative of boiling in an epithermal environment. Further from 297.05 metres downhole a 3.55 metre wide micro-quartz diorite porphyry dike was intersected. This exhibited at least two (2) phases of hydrothermal alteration with the second phase of fracture controlled alteration carrying arsenopyrite needles in fracture surfaces and in the altered wall rock selvedges, identical to the mineralization style at Sirenggok.

Significant intersections for the two additional drillholes at Bekajang are listed in *Table 9-7: Significant Intersections at Bekajang TSF* below.

Hole No	From (m)	To (m)	Length (m)	Au (g/t)
BKDDH-10	1.00	6.20	5.20	1.01
BKDDH-10	9.20	14.00	4.80	1.71
BKDDH-10	126.25	128.60	2.35	1.09
BKDDH-10	154.00	155.00	1.00	0.51
BKDDH-10	297.05	300.60	3.55	2.09
BKDDH-11	0.00	6.00	6.00	1.26
BKDDH-11	276.20	277.40	1.20	1.02

Table 9-7: Significant Intersections at Bekajang TSF

The discovery of this style of mineralization has increased the potential for the discovery of porphyry hosted mineralisation within the outcropping intrusives at Bekajang and at depth.

9.2.3. Sirenggok Sector

Exploration by BYG, Renison Goldfields and Menzies and NBG up to 2008 has outlined an Inferred Resource of 8.346 million tonnes at a grade of 1.14 g/t Au for 307,000 ounces using a 0.5g/t lower cut.

The gold-arsenic-antimony mineralization is hosted by veins, vein stockworks and as disseminations within quartz-sericite to propyllitic altered quartz-feldspar micro-quartz diorite porphyry. A younger phase of xenolithic quartz diorite porphyry intrudes the earlier porphyry and the overall morphology of the deposit is funnel shaped.

The currently defined resource is open along strike and at depth. The main trend appears to be NW-SE and steep to moderately dipping to the NE. There are two (2) other areas of mineralisation picked up to the Northeast in surface samples and several drill holes and surface mineralisation in the SW. Given that the current resource only covers around one third ($^1/_3$) of the surface and drilled mineralisation there is significant upside to increase the resource.

In addition, as shown in *Figure 9-19: DIGHEM Resistors & Gold Anomalous Soils at Sirenggok* there are several strong resistors shown in the DIGHEM data and coincident gold in soil geochemical anomalies that have had scant exploration in the past and that are prime targets for additional gold mineralisation and to expand the current resources.

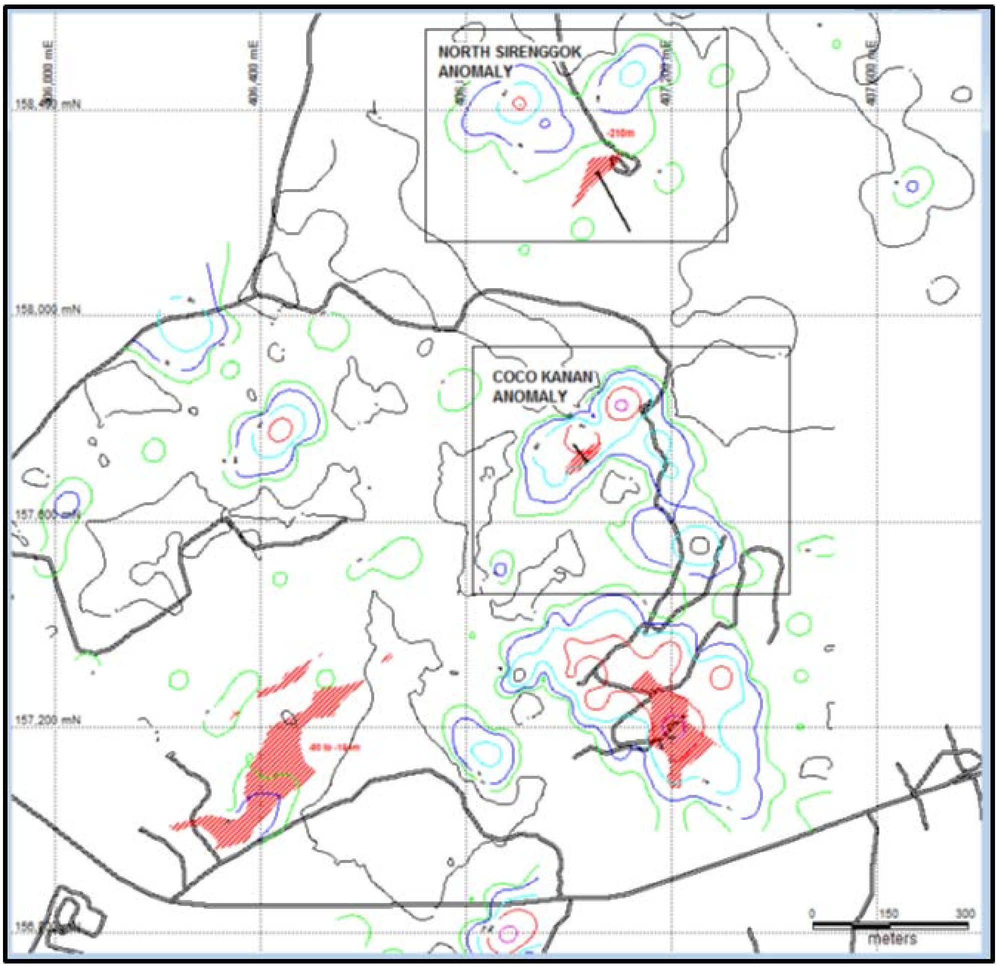


Figure 9-19: DIGHEM Resistors & Gold Anomalous Soils at Sirenggok

9.2.4. Pejiru Sector

The Pejiru Sector has been the focus of intensive exploration particularly through the Menzies era. A total of approximately seven hundred and four (704) drillholes (682 RC and 22 DDH drillholes) have been drilled at Pejiru including; two hundred and twenty-seven (227) drillholes (214 RC and 13 DDH drillholes) at Pejiru-Bogag, one hundred and two (102) drillholes (102 RC drillholes) at Pejiru Extension, fifty-four (54) RC drillholes at Boring and fifty-one (51) drillholes (50 RC and 1 DDH drillholes) at Kapor.

Pejiru has been subject to metallurgical studies and mine scoping studies by Menzies in the 1990's. NBG's work has mainly focused on updating the resource figures here using the existing data.

Pejiru has a substantial Inferred Resource outlined, however, there is potential to upgrade this by further drilling. Much of the past drilling has been near the road network and the limits to mineralisation are not that well defined. The main ore zone is not closed off so there is potential for lateral extensions as well as for extensions in structurally favourable sites, and at depth in areas of fluid upflow.

Recent evaluation of available data including geophysical data by Besra has highlighted areas of favourable structure associated with on-lapping shale over Bau Limestone and areas of high conductivity within the Bau Limestone related to pipe-like features, some of which correlate with known areas of gold mineralisation, but are largely untested as shown in *Figure 9-20 - Plan Showing Pejiru Resource Outlines & Areas of High Conductivity in Limestone* to *Figure 9-24 - Boring Cross-Section D-D Showing Steep Dipping Mineralisation Open at Depth.*

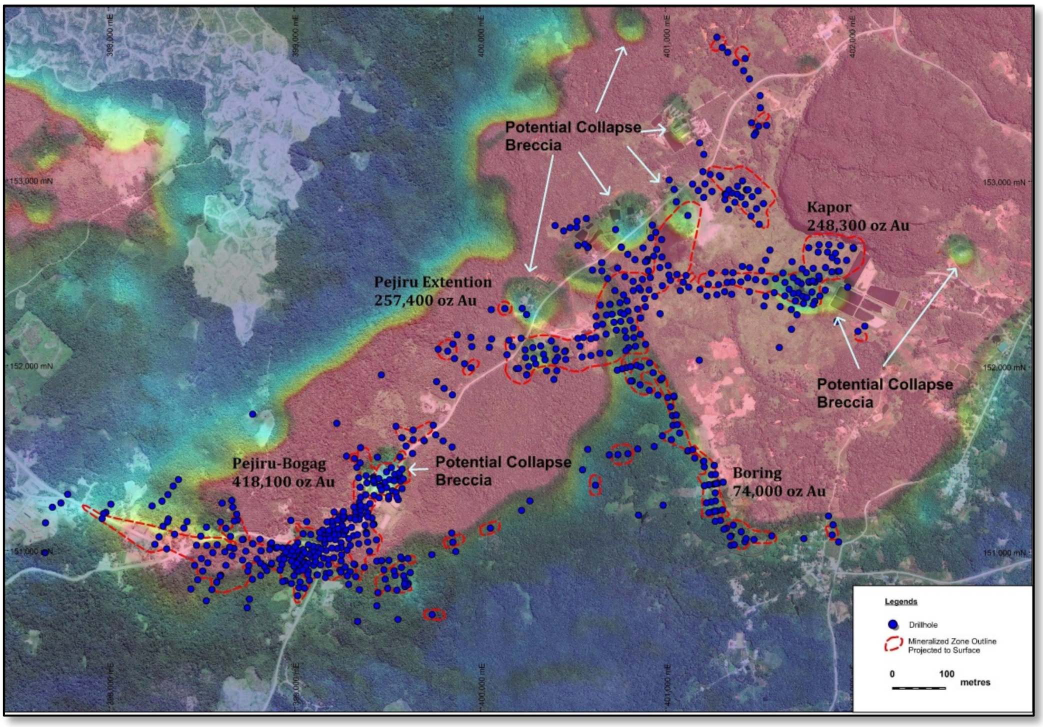


Figure 9-20 - Plan Showing Pejiru Resource Outlines & Areas of High Conductivity in Limestone

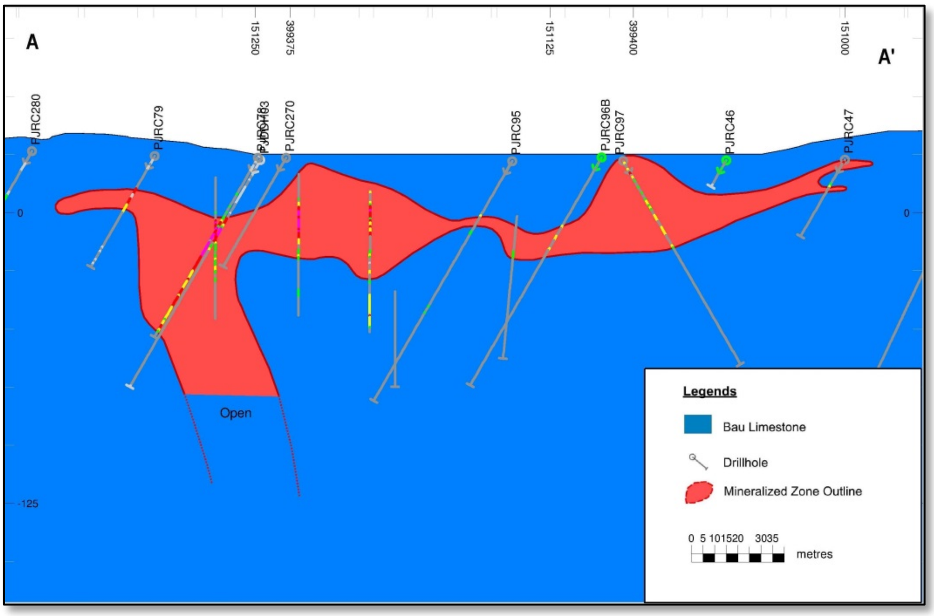


Figure 9-21 - Long Section A-A through Pejiru Showing Depth Potential (refer to figure 7-11)

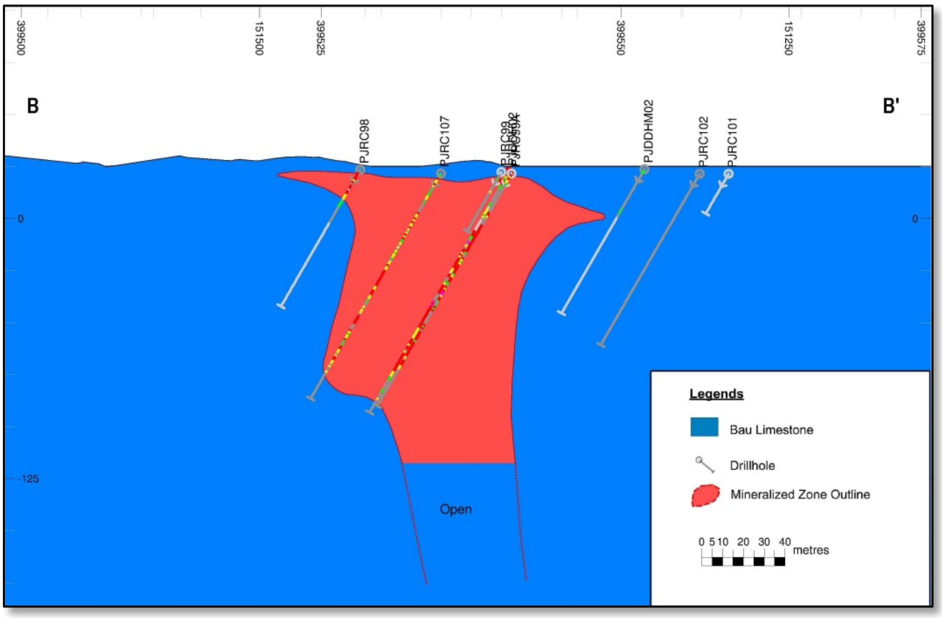


Figure 9-22 – Cross-Section B-B through Pejiru Showing Depth Potential Associated with Conductivity High (refer to figure 7-11)

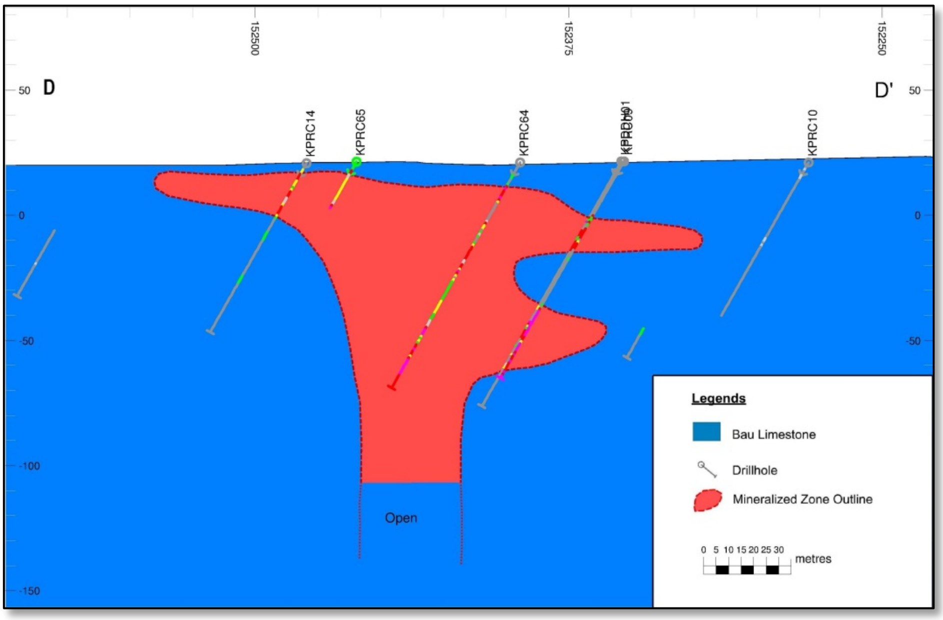


Figure 9-23 - Kapor Cross-Section C-C Showing Depth Potential Associated with Conductivity High (refer to figure 7-14)

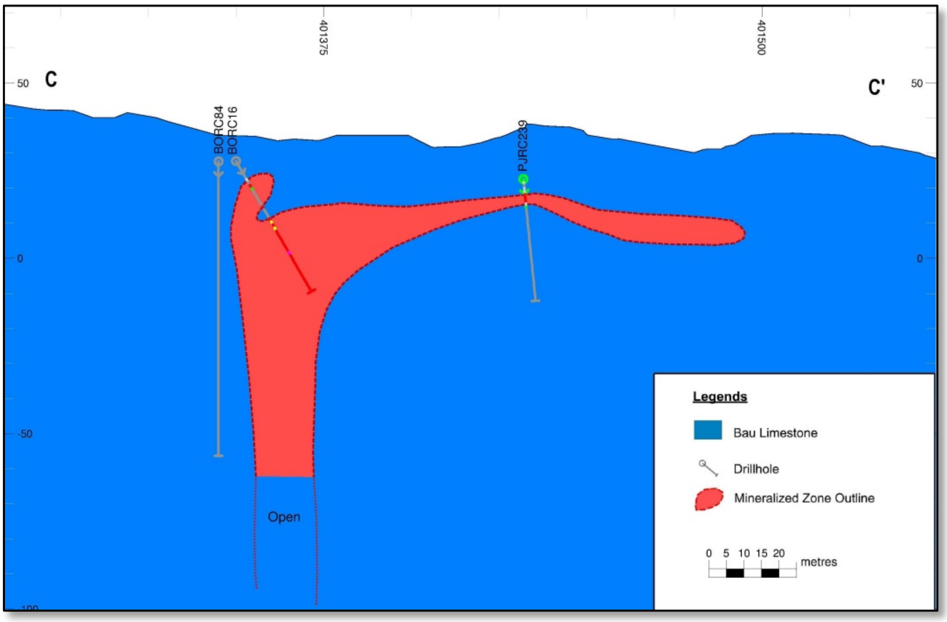


Figure 9-24 - Boring Cross-Section D-D Showing Steep Dipping Mineralisation Open at Depth (refer to figure 7-13)

9.2.5. Taiton Sector

The Taiton Sector deposit types are dominantly mangano-calcite veins and breccias and remnants of the extensive elluvial auriferous clays that were mined historically by the Chinese miners of the late 19th Century and early 20th Century. These were largely developed on the limestone shale contact. The current target areas are vein systems aligned on two (2) major fault systems, the NE-SW trending Tai Parit Fault zone and the NW-SE Taiton Fault zone.

The main target areas aligned with the Tai Parit Fault are from south the north, Tabai-Rumoh and Taiton A (over a strike length of ~1.2 kilometres) and Saburan, while those aligned with the Taiton Fault are Umbut and Taiton B.

NBG has completed geological mapping, trenching, channel sampling and both resource and exploration drilling mainly at Tabai and Taiton A. Some limited trenching and drilling were conducted over mineralised intrusives that lie close to Taiton B but outside the main vein structure there.

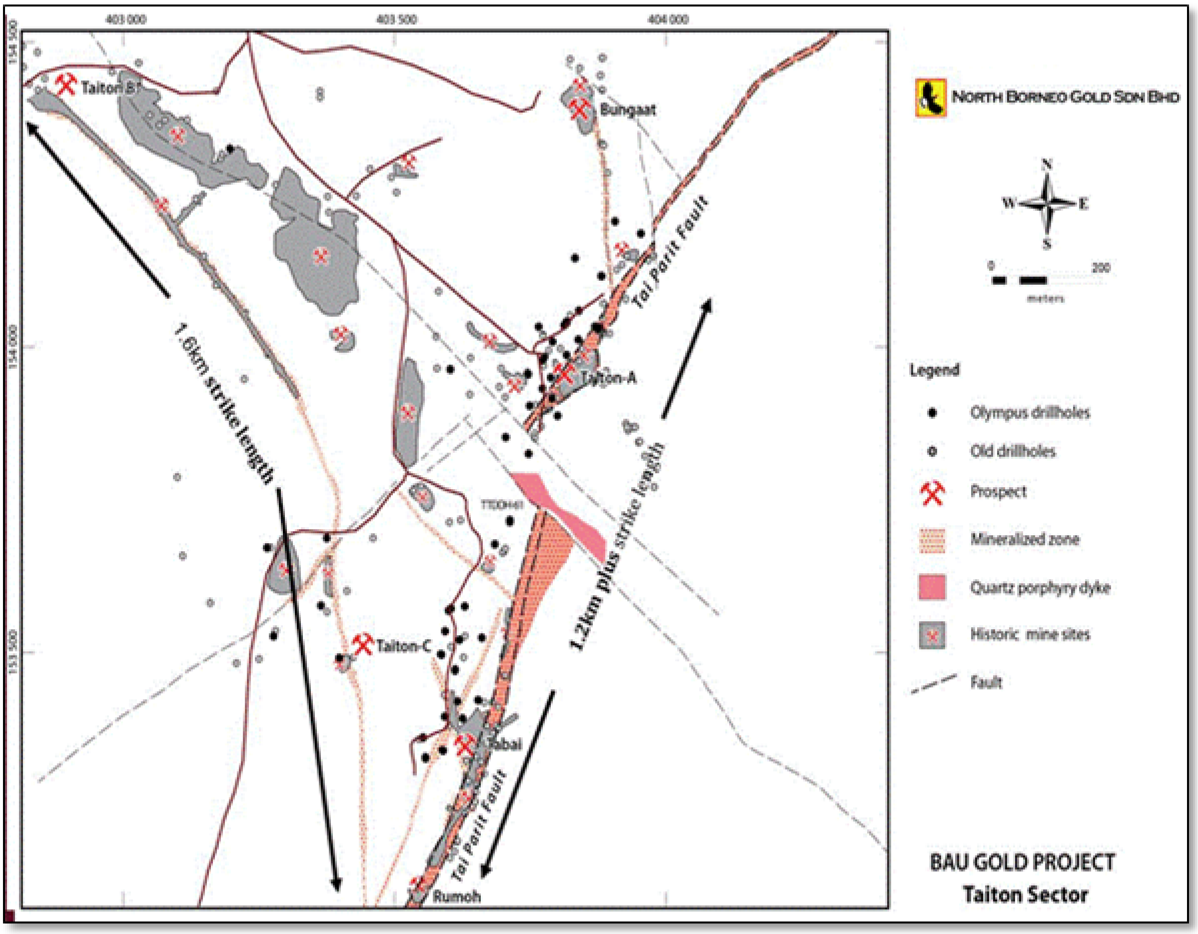


Figure 9-25: Taiton Sector Exploration Features

9.2.5.1. Tabai

Tabai (including the former Rumoh mine) is developed on a vein system between 4 metres and up to 23 metres wide, (observed) mostly composed of brecciated mangano-calcite vein, frequently vuggy with drusy quartz infilling and overgrowths along with patchy silicification, auriferous clay, arsenopyrite, realgar, stibnite and native arsenic. The drill programmes in 2010 to 2011 and subsequent modeling has traced the mineralisation essentially as far as Taiton A with a small (less than 100m) separation between the two.

There are up to four (4) sub-parallel NE to NNW trending gold mineralised structures that persist to depths of 300 metres in drillholes below surface. The vein system remains open at depth and along strike to the South. To the North it merges with Taiton A.

Mineralisation is largely confined to structures within the Bau Limestone; however, there are instances of gold being developed within vuggy drusy quartz veins along contacts with intrusive dacite porphyry dykes.

Mining by BYG in 1995 extracted a 2,340 tonne parcel that averaged 10.81 g/t Au. Underground rock sampling by NBG has returned grades that range from 0.14 to 115 g/t Au and average 8.46 g/t Au from within the excavation.

From 2010 to 2011 NBG completed a combined exploration and resource drill programme. Significant drill intercepts for holes drilled in 2010 through 2011 at Tabai are shown below in *Table 9-8: Significant Intersections for Tabai 2010 to 2011 Drill Programme.*

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area
TTDDH-05	13.80	14.20	0.40	1.77	Tabai
TTDDH-05	15.30	18.30	3.00	8.73	Tabai
TTDDH-05	19.30	21.80	2.50	18.64	Tabai
TTDDH-07	19.50	20.20	0.70	0.54	Tabai
TTDDH-07	42.00	43.00	1.00	5.87	Tabai
TTDDH-10	193.30	194.50	1.20	0.62	Tabai
TTDDH-13	41.40	42.55	1.15	0.67	Tabai
TTDDH-13	94.70	95.50	0.80	7.74	Tabai
TTDDH-13	130.30	134.00	3.70	0.95	Tabai
TTDDH-13	148.60	156.60	8.00	1.76	Tabai
TTDDH-13	159.00	161.00	2.00	0.87	Tabai
TTDDH-13	163.10	163.60	0.50	0.56	Tabai
TTDDH-13 - and	155.10	156.60	1.50	3.44	Tabai
TTDDH-13 - incl	149.60	152.10	2.50	2.52	Tabai
TTDDH-39	7.80	9.50	1.70	2.18	Tabai
TTDDH-39	48.40	48.95	0.55	0.53	Tabai
TTDDH-39	76.00	77.30	1.30	0.60	Tabai
TTDDH-39	117.80	132.00	14.20	1.50	Tabai
TTDDH-39	164.00	164.40	0.40	1.13	Tabai
TTDDH-39 - incl	123.60	126.00	2.40	5.97	Tabai
TTDDH-42	106.14	107.86	1.72	2.08	Tabai
TTDDH-42 - incl	106.14	107.10	0.96	3.18	Tabai
TTDDH-44	39.00	50.72	11.72	1.17	Tabai
TTDDH-44	49.46	50.72	1.26	4.63	Tabai
TTDDH-44 - incl	42.00	50.72	8.72	1.39	Tabai
TTDDH-45	61.25	69.45	8.20	1.21	Tabai
TTDDH-45 - incl	65.50	69.20	3.70	1.73	Tabai
TTDDH-46	67.50	67.90	0.40	0.52	Tabai
TTDDH-47	152.12	153.00	0.88	0.91	Tabai
TTDDH-47	158.00	159.00	1.00	1.80	Tabai
TTDDH-47	161.52	165.00	3.48	0.65	Tabai
TTDDH-47	169.55	175.00	5.45	0.80	Tabai
TTDDH-47	187.00	189.13	2.13	0.57	Tabai
TTDDH-47	198.41	201.52	3.11	0.90	Tabai
TTDDH-48	14.00	16.00	2.00	1.05	Tabai
TTDDH-48	72.40	72.65	0.25	2.00	Tabai
TTDDH-48	75.40	76.90	1.50	2.04	Tabai
TTDDH-48 -incl	14.00	14.75	0.75	2.09	Tabai
TTDDH-50A	126.00	127.60	1.60	2.17	Tabai
TTDDH-51	32.62	33.30	0.68	0.60	Tabai
TTDDH-51	38.42	39.00	0.58	1.61	Tabai

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area
TTDDH-51	121.44	123.30	1.86	0.66	Tabai
TTDDH-51	129.85	132.00	2.15	0.73	Tabai
TTDDH-51	138.10	138.33	0.23	1.98	Tabai
TTDDH-51	141.33	169.00	27.67	1.09	Tabai
TTDDH-51	145.00	145.67	0.67	3.80	Tabai
TTDDH-51	155.80	160.90	5.10	1.97	Tabai
TTDDH-51	160.28	160.90	0.62	5.55	Tabai
TTDDH-52	24.30	29.60	5.30	1.29	Tabai
TTDDH-52	30.30	30.85	0.55	2.70	Tabai
TTDDH-52	35.90	39.20	3.30	2.96	Tabai
TTDDH-52	42.30	42.75	0.45	1.30	Tabai
TTDDH-52	105.30	106.00	0.70	1.69	Tabai
TTDDH-52	108.30	116.30	8.00	1.42	Tabai
TTDDH-52	118.30	121.50	3.20	2.00	Tabai
TTDDH-52	146.60	147.60	1.00	1.14	Tabai
TTDDH-53	10.90	11.30	0.40	1.14	Tabai
TTDDH-53A	17.10	18.00	0.90	0.64	Tabai
TTDDH-53A	117.60	119.10	1.50	1.69	Tabai
TTDDH-54	28.70	29.60	0.90	11.00	Tabai
TTDDH-54	61.00	62.00	1.00	0.56	Tabai
TTDDH-54	66.60	67.00	0.40	0.75	Tabai
TTDDH-55	3.17	4.00	0.83	0.81	Tabai
TTDDH-55	18.90	19.90	1.00	2.50	Tabai
TTDDH-55	82.80	83.10	0.30	1.10	Tabai
TTDDH-55	94.05	98.08	4.03	4.33	Tabai
TTDDH-55	102.15	103.20	1.05	0.65	Tabai
TTDDH-55	163.00	163.70	0.70	0.53	Tabai
TTDDH-55	222.00	223.00	1.00	0.56	Tabai
TTDDH-55	232.80	234.00	1.20	1.18	Tabai
TTDDH-55	243.30	245.00	1.70	2.33	Tabai
TTDDH-55	244.30	245.00	0.70	4.70	Tabai
TTDDH-55	252.00	253.00	1.00	1.26	Tabai
TTDDH-55	257.00	258.00	1.00	8.22	Tabai
TTDDH-55	266.00	267.00	1.00	0.80	Tabai
TTDDH-55	274.00	275.00	1.00	0.64	Tabai
TTDDH-55	279.00	280.00	1.00	1.38	Tabai
TTDDH-55	295.64	295.90	0.26	1.54	Tabai
TTDDH-55	301.25	304.00	2.75	0.80	Tabai
TTDDH-55	307.00	308.00	1.00	4.05	Tabai
TTDDH-55	313.00	314.00	1.00	0.68	Tabai
TTDDH-55 -incl	95.00	97.00	2.00	6.67	Tabai
TTDDH-57	93.70	96.00	2.30	0.73	Tabai
TTDDH-57	107.40	107.70	0.30	1.06	Tabai
TTDDH-57	109.85	110.20	0.35	0.50	Tabai
TTDDH-59	32.00	32.50	0.50	1.14	Tabai

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area
TTDDH-61	151.30	161.40	10.10	0.84	Tabai
TTDDH-61	164.40	166.70	2.30	0.67	Tabai
TTDDH-61	171.20	172.20	1.00	0.54	Tabai
TTDDH-61	173.70	181.70	8.00	0.89	Tabai
TTDDH-61	192.00	193.00	1.00	0.55	Tabai
TTDDH-61 - incl	159.40	160.40	1.00	3.62	Tabai
TTDDH-62	101.30	102.30	1.00	0.65	Tabai
TTDDH-62	106.00	108.00	2.00	17.55	Tabai
TTDDH-62	210.55	211.35	0.80	1.58	Tabai
TTDDH-62 - incl	106.00	107.00	1.00	34.20	Tabai
TTDDH-63	124.40	125.40	1.00	0.77	Tabai
TTDDH-63	161.00	164.00	3.00	0.88	Tabai
TTDDH-63	175.00	176.00	1.00	1.46	Tabai
TTDDH-63	190.00	192.00	2.00	1.13	Tabai
TTDDH-63	216.80	217.80	1.00	1.46	Tabai
TTDDH-64	22.00	25.00	3.00	0.73	Tabai
TTDDH-64	30.20	35.15	4.95	1.30	Tabai
TTDDH-64 - and	34.00	35.15	1.15	2.60	Tabai
TTDDH-64 - incl	30.20	32.85	2.65	1.27	Tabai
TTDDH-65	212.00	213.00	1.00	1.04	Tabai
TTDDH-65	230.25	231.40	1.15	2.41	Tabai
TTDDH-65	246.85	249.30	2.45	0.81	Tabai
TTDDH-65	279.00	280.00	1.00	1.06	Tabai
TTDDH-65	279.00	282.00	3.00	0.61	Tabai
TTDDH-65	311.00	312.00	1.00	0.50	Tabai
TTDDH-66	40.90	42.50	1.60	0.68	Tabai
TTDDH-68	45.10	45.80	0.70	0.55	Tabai
TTDDH-68	46.00	46.70	0.70	0.52	Tabai
TTDDH-68	47.00	49.00	2.00	2.44	Tabai
TTDDH-68	52.20	52.90	0.70	0.64	Tabai
TTDDH-68	56.70	59.20	2.50	1.42	Tabai
TTDDH-68	61.00	64.20	3.20	1.25	Tabai
TTDDH-68	66.20	67.80	1.60	0.57	Tabai
TTDDH-68	77.80	78.40	0.60	1.40	Tabai
TTDDH-68	82.40	83.70	1.30	0.88	Tabai
TTDDH-68	87.70	88.30	0.60	0.92	Tabai
TTDDH-68	96.20	97.80	1.60	0.89	Tabai
TTDDH-69	15.40	16.60	1.20	0.50	Tabai
TTDDH-69	17.35	19.00	1.65	0.55	Tabai
TTDDH-69	22.10	23.20	1.10	1.92	Tabai
TTDDH-69	17.35	19.00	1.65	0.55	Tabai
TTDDH-69	22.10	23.20	1.10	1.92	Tabai
TTDDH-69	25.60	27.10	1.50	0.97	Tabai
TTDDH-69	30.00	35.00	5.00	0.86	Tabai
TTDDH-69	33.00	35.00	2.00	1.57	Tabai

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area
TTDDH-69	47.00	48.00	1.00	0.89	Tabai
TTDDH-69	61.60	62.10	0.50	1.23	Tabai
TTDDH-69	65.00	65.50	0.50	0.91	Tabai
TTDDH-69	68.80	70.90	2.10	0.65	Tabai
TTDDH-70	105.70	107.20	1.50	0.86	Tabai
TTDDH-71	132.50	135.25	2.75	0.41	Tabai
TTDDH-71	133.50	134.50	1.00	0.50	Tabai
TTDDH-72	71.70	72.50	0.80	0.59	Tabai

Table 9-8: Significant Intersections for Tabai 2010 to 2011 Drill Programme

The programme succeeded in defining a two small resource areas, one amenable to possible open cut mining with Indicated Resources of 12,100 oz Au and Inferred Resources of 4,200 oz Au. This estimated used a 0.5 g/t Au lower cutoff.

The second comprises an underground resource due to adverse topography rendering open cut mining impractical. This comprises an Indicated Resource of 39,700 oz Au and Inferred Resources of 6,000 oz based on a 2.0 g/t lower cutoff.

9.2.5.2. Taiton A

Taiton A is centred approximately 400 metres further North from Tabai along strike on the Tai Parit Fault Structure. It comprises the Taiton open pit, the Tunnel Adit above Taiton A Pit and several adits that are located at the base of the limestone bluffs and includes Bungaat several hundred metres further North again.

There are several mineralised NW fault structures trending toward, intersecting and cross cutting the Tai Parit fault zone. The main NW structure, the Overhead Tunnel Adit, lies vertically above the Taiton A pit and continues in a SE direction for several hundred metres. Numerous old mine pits occur near the intersection of these prominent NW-SE trending structures with the NE trending Tai Parit Fault system.

Exposure at the Taiton A pit is limited however from the extensive drill core inventory, mineralisation passes from an upper zone of auriferous secondary clay deposits into primary ore comprising mangano-calcite veining with abundant native arsenic, realgar, arsenopyrite, some silicification and drusy quartz veining, brecciation and massive white calcite veins.

A notable feature of Taiton A is that drilling has shown mineralization persists to at least 300 metres vertically and is open at depth. The mangano-calcite vein style is most common within the Bau Limestone but there is also a common association with the contact zones of NW trending dacite porphyry and andesite porphyry dykes, often weathering to auriferous clay to depths of 100m plus vertically below surface.

The Company completed programmes of geological mapping, trenching and both exploration and resource drilling. Significant intersections for the drill programme are listed below in *Table*

9-9: Significant Intersections from Taiton A Drill Programme 2010-2011. These include holes drilled near Taiton B based on trench results tracing mineralized intrusives NW from Taiton A.

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area
TTDDH-15A	131.00	132.57	1.57	0.54	Bunggat
TTDDH-15A	151.50	152.00	0.50	5.61	Bunggat
TTDDH-15A	163.23	163.82	0.59	0.72	Bunggat
TTDDH-06	10.80	14.00	3.20	0.69	Taiton A
TTDDH-06	141.55	142.90	1.35	1.70	Taiton A
TTDDH-08	75.00	76.00	1.00	1.58	Taiton A
TTDDH-08	79.50	80.20	0.70	0.92	Taiton A
TTDDH-08	89.60	90.60	1.00	0.56	Taiton A
TTDDH-11	43.68	56.00	12.32	2.54	Taiton A
TTDDH-11	59.00	63.00	4.00	1.29	Taiton A
TTDDH-11	93.20	93.48	0.28	6.48	Taiton A
TTDDH-11	126.53	127.28	0.75	0.55	Taiton A
TTDDH-12	33.30	33.80	0.50	0.48	Taiton A
TTDDH-12	126.50	127.40	0.90	1.39	Taiton A
TTDDH-12	150.80	154.00	3.20	1.38	Taiton A
TTDDH-12	189.90	193.00	3.10	1.45	Taiton A
TTDDH-12	199.30	200.20	0.90	4.50	Taiton A
TTDDH-16	32.70	33.70	1.00	1.23	Taiton A
TTDDH-16	52.80	55.10	2.30	1.89	Taiton A
TTDDH-18	94.00	97.30	3.30	0.50	Taiton A
TTDDH-18	95.75	97.30	1.55	0.78	Taiton A
TTDDH-18	108.00	109.00	1.00	0.51	Taiton A
TTDDH-18	128.00	129.00	1.00	0.48	Taiton A
TTDDH-18	140.35	141.20	0.85	0.47	Taiton A
TTDDH-18	146.00	147.00	1.00	2.17	Taiton A
TTDDH-18	146.00	148.45	2.45	1.05	Taiton A
TTDDH-18	153.60	156.00	2.40	0.58	Taiton A
TTDDH-18	171.80	173.50	1.70	0.71	Taiton A
TTDDH-19	30.50	31.60	1.10	0.87	Taiton A
TTDDH-19	93.30	94.10	0.80	0.69	Taiton A
TTDDH-19	99.65	100.55	0.90	0.52	Taiton A
TTDDH-20	172.00	176.00	4.00	1.69	Taiton A
TTDDH-20	173.00	176.00	3.00	2.05	Taiton A
TTDDH-21	15.60	16.27	0.67	1.43	Taiton A
TTDDH-21	31.84	35.00	3.16	3.09	Taiton A
TTDDH-22	42.85	43.60	0.75	0.68	Taiton A
TTDDH-24	20.47	22.00	1.53	0.48	Taiton A
TTDDH-24	25.05	28.10	3.05	3.78	Taiton A
TTDDH-24	51.45	60.25	8.80	0.93	Taiton A
TTDDH-24	68.80	74.77	5.97	1.80	Taiton A
TTDDH-24	83.53	84.84	1.31	2.60	Taiton A
TTDDH-24 - incl	58.50	60.25	1.75	3.08	Taiton A

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area
TTDDH-24 - incl	72.00	74.77	2.77	3.20	Taiton A
TTDDH-25	46.00	65.40	19.40	1.60	Taiton A
TTDDH-25	107.00	108.00	1.00	0.49	Taiton A
TTDDH-25 - and	52.00	57.80	5.80	2.37	Taiton A
TTDDH-25 - and	59.10	65.40	6.30	1.57	Taiton A
TTDDH-25 - incl	46.00	48.00	2.00	3.53	Taiton A
TTDDH-25 - with	56.00	57.00	1.00	7.07	Taiton A
TTDDH-25 - with	60.00	61.00	1.00	7.33	Taiton A
TTDDH-26	0.00	1.00	1.00	1.22	Taiton A
TTDDH-26	65.00	68.50	3.50	2.21	Taiton A
TTDDH-26 - incl	67.00	68.50	1.50	4.47	Taiton A
TTDDH-28	5.50	6.80	1.30	0.63	Taiton A
TTDDH-28	9.45	10.50	1.05	0.58	Taiton A
TTDDH-28	21.00	22.00	1.00	0.55	Taiton A
TTDDH-28	72.30	74.20	1.90	0.93	Taiton A
TTDDH-28	92.00	95.00	3.00	1.02	Taiton A
TTDDH-28	149.00	150.00	1.00	1.17	Taiton A
TTDDH-28	156.55	161.15	4.60	1.09	Taiton A
TTDDH-29	93.00	105.00	12.00	1.39	Taiton A
TTDDH-29 - incl	95.40	99.00	3.60	3.09	Taiton A
TTDDH-30	111.60	114.00	2.40	0.91	Taiton A
TTDDH-34	97.10	99.00	1.90	1.46	Taiton A
TTDDH-34	103.00	105.00	2.00	2.25	Taiton A
TTDDH-34	108.00	111.00	3.00	0.50	Taiton A
TTDDH-34 - incl	108.00	109.00	1.00	0.71	Taiton A
TTDDH-35	31.45	32.10	0.65	1.14	Taiton A
TTDDH-35	46.20	46.90	0.70	7.36	Taiton A
TTDDH-36	76.20	77.50	1.30	0.76	Taiton A
TTDDH-37	34.10	35.90	1.80	2.88	Taiton A
TTDDH-37	54.00	55.90	1.90	1.13	Taiton A
TTDDH-37 - incl	54.85	55.90	1.05	1.54	Taiton A
TTDDH-74	24.70	26.25	1.55	0.64	Taiton A
TTDDH-77	14.70	16.70	2.00	1.00	Taiton A/B
TTDDH-78	15.00	17.10	2.20	1.85	Taiton A/B
TTDDH-78	19.70	21.20	1.50	2.44	Taiton A/B
TTDDH-27	215.00	216.30	1.30	1.52	Taiton B West

Table 9-9: Significant Intersections from Taiton A Drill Programme 2010-2011

The results of the trenching programme at Taiton A and B that targeted gold mineralization in the contact zone of several intrusive dykes at surface are tabulated in *Table 9-10: Taiton A/B Trench Channel Significant Results* below.

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area	Notes
TATR-03	1.00	2.00	1.00	0.20	Taiton A/B	

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area	Notes
TATR-03	12.00	15.00	3.00	0.26	Taiton A/B	
TATR-03	21.00	27.00	6.00	0.19	Taiton A/B	
TATR-03	34.00	40.00	6.00	0.18	Taiton A/B	
TATR-04	0.00	1.00	1.00	0.11	Taiton A/B	
TATR-04	5.00	11.00	6.00	0.10	Taiton A/B	
TATR-04	24.00	25.00	1.00	0.10	Taiton A/B	
TATR-05	0.00	7.00	7.00	1.59	Taiton A/B	1g/t Au cut-off
TATR-05	0.00	8.00	8.00	1.46	Taiton A/B	0.5g/t Au cut-off
TATR-05	8.00	18.00	10.00	0.36	Taiton A/B	
TATR-05 - incl	8.00	13.00	5.00	0.20	Taiton A/B	
TATR-05 - incl	13.00	17.00	4.00	0.57	Taiton A/B	
TATR-05 - and	17.00	18.00	1.00	0.28	Taiton A/B	
TATR-05	27.00	30.00	3.00	0.20	Taiton A/B	
TATR-06	0.00	8.00	8.00	0.28	Taiton A/B	
TATR-06	11.00	17.00	6.00	1.30	Taiton A/B	0.5g/t Au cut-off
TATR-06 - incl	14.00	17.00	3.00	2.35	Taiton A/B	1g/t Au cut-off
TATR-06 - incl	14.00	16.00	2.00	3.18	Taiton A/B	
TATR-07	10.00	35.00	25.00	0.54	Taiton A/B	
TATR-07 - incl	15.00	18.00	3.00	1.16	Taiton A/B	
TATR-07	37.00	52.00	15.00	0.22	Taiton A/B	
TATR-07	67.30	72.00	4.70	0.26	Taiton A/B	
TATR-07	110.00	111.00	1.00	0.42	Taiton A/B	
TATR-08	2.00	3.00	1.00	0.13	Taiton A/B	
TATR-08	8.00	14.00	6.00	0.18	Taiton A/B	
TATR-09	0.00	14.00	14.00	0.16	Taiton A/B	
TATR-09	17.00	18.00	1.00	0.12	Taiton A/B	
TATR-09	21.00	26.00	5.00	0.32	Taiton A/B	
TATR-09	27.00	36.00	9.00	0.32	Taiton A/B	
TATR-09	53.00	59.00	6.00	2.13	Taiton A/B	1g/t cut-off
TATR-09	53.00	60.00	7.00	1.92	Taiton A/B	0.5g/t Au cut-off
TATR-10	1.00	2.00	1.00	0.61	Taiton A/B	
TATR-10	20.00	22.00	2.00	0.12	Taiton A/B	
TATR-10A	59.00	64.00	5.00	0.28	Taiton A/B	
TATR-10B	4.00	12.00	8.00	0.17	Taiton A/B	
TATR-11	0.00	1.00	1.00	0.15	Taiton A/B	
TATR-11	3.00	43.00	40.00	0.31	Taiton A/B	
TATR-11	23.00	25.00	2.00	0.73	Taiton A/B	
TATR-11	78.00	81.00	3.00	0.40	Taiton A/B	
TATR-11	81.00	87.00	6.00	2.19	Taiton A/B	
TATR-12	0.00	1.00	1.00	0.37	Taiton A/B	

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Area	Notes
TATR-12	0.00	18.00	18.00	0.15	Taiton A/B	
TATR-12	21.00	28.00	7.00	11.84	Taiton A/B	
TATR-12 -incl	21.00	22.00	1.00	48.80	Taiton A/B	
TATR-12 - and	24.50	26.00	1.50	17.20	Taiton A/B	
TATR-12	28.00	29.00	1.00	0.40	Taiton A/B	

Table 9-10: Taiton A/B Trench Channel Significant Results

There are a number of significant intersections including some high grade in Trench TATR-12 with 7 metres at 11.84 g/t Au. Most of the intersections are in auriferous clay on the contact zones between the intrusives and limestone. Several drillholes were drilled beneath these intersections and grade did not persist in the holes drilled. Elsewhere though, some of these clay deposits do persist to 100 metres plus depth.

9.2.5.3. Taiton B & Taiton C

The Taiton B vein is hosted within Bau Limestone and comprises a 2 metre to 6 metre wide vein and vein breccia of mangano-calcite, quartz with bands and pods of realgar, arsenopyrite and stibnite mineralisation. It trends NW-SE along the Taiton Fault and may ultimately intersect the Tai Parit Fault near Tabai. A large drive on vein is developed over distance of 700 metres with extensive stopes overhead. Mineralisation is exposed in the backs and underfoot. The deposit was mined by BYG during the 1990's with a recovered grade recorded at 3.7 g/t Au. This included dilution and with the metallurgical issues at the time suggests the head grade to be higher than 3.7 g/t Au.

A weighted average from historic channel samples by BYG in the Taiton B tunnel gives value of 4.7 g/t Au. This only covers around the first 400 metres of the current tunnel. NBG have completed reconnaissance rock sampling along the remainder of the Taiton B vein tunnel with grades ranging from 0.38 g/t Au to 22.9 g/t Au.

The Taiton-B massive mangano-calcite vein has now been mapped over a 1.5 kilometres of strike length and includes the section known as Taiton C. Strike and depth extensions remain unexplored. From seventy-four (74) vein rock chip samples, assays ranged from 0.16 to 62.0 g/t gold, with 48 % reporting above 1.0 g/t Au; the average being 7.85 g/t Au.

Prior to NBG's involvement there had been no drilling to test beneath the Taiton B vein even though ore is exposed underfoot in the main drive. NBG initially drilled three (3) drillholes at the NW end with inconclusive results. In mid-2008, after further geological modeling a fourth drillhole was drilled to test beneath the vein. This hole intersected a 4 metre wide calcite-quartz vein with grades up to 0.8 g/t Au. While not ore grade here it does prove that there is depth continuity to the vein and it is expected that higher grade gold will be encountered in shoots within the vein structure.

There is no mineral resource established for Taiton B underground portion, however, given the dimensions of the known vein underground, the known grade range and the surface mapped extensions, NBG are of the opinion that with the completion of a suitable drilling and sampling programme there is potential to delineate a 43-101/CIMM/JORC compliant resource.

9.2.5.4. Umbut

The Umbut area lies to the NW of Taiton B and partially straddles the Krokong Road. It is described by Bukit Young in internal memos from the mid 1990's as having resources of 56,088 tonnes at a grade of 2.84 g/t Au within quartz calcite ore and within the shale limestone contact. While the later drill results from BYG were disappointing, from examination of the drill data, the Umbut area is typical of the whole Taiton area where there are several generations of drill data with ore grade intersections that have not been followed up or evaluated in light of new interpretations. The current resource modeling has delineated and Inferred Resource of 47,600 oz of gold.

9.2.6. Tailings

9.2.6.1. Tailings Dam

At the BYG mine site near Bau Township, auriferous tailings derived from the mining and processing of ore from various deposits by BYG between 1983 and 1996 have been deposited in the Bekajang Tailings impoundment adjacent to the now disused BYG processing plant. A significant volume of the tailings are derived from ores mined at the high-grade Tai Parit gold deposit using the Carbon-In-Pulp (CIP) gold processing method during the period 1991 to 1996. A total of over 3.0 million tonnes of tailings have been deposited in the pond, based on BYG mine production records.

A further unknown quantity of tailings was contained here dating from the mining activity of the Borneo Gold Company in the early 20th Century. As well as these NBG have completed an extensive auger drill programme on a 25 metre by 25 metre grid pattern. Each auger hole was sampled in 1 metre splits and assayed by an internationally accredited laboratory.

The tailings impoundment has been modelled and an Inferred Resource of 100,400 oz gold estimated.

9.2.6.2. Other Tailings

There are a number of other areas of tailings within several kilometers of the former BYG plant site. These have not been assessed in recent times but could potential add to the tailings resource in the BYG tailings impoundment at Bekajang.

The main areas that could have economic potential straddle the Krokong Road Bypass and out toward Bau Lama. Some of this area has been mined and partially sterilised by the Bypass road; however, BYG records from 1995 refer to over 500,000 tonnes grading 1.85 g/t Au. BYG also outlined several other areas of tailings locally known as the Army Camp and Filipino Camp. Smaller tonnages are mentioned ranging from 29,000 to 60,000 tonnes grading approximately 1.2 to 2.0 g/t Au.

While the tonnages are not large, the grades are reasonably high; the tailings are within a short distance from the BYG plant site, (1-2 kilometres) and would be additional to the Bekajang Tailings resource. Further investigation is warranted.

9.2.7. Say Seng Sector

9.2.7.1. Say Seng

The Say Seng Prospect is located between the West flank of Gunung Pangga and the Buso Road, about 3km Northeast of Bau.

The area is currently operated as a large limestone quarry by the Gladioli Group and forms part of the joint venture area with NBG.

Historically stibnite was mined here in the 19th Century, while gold has been mined intermittently since the 1930's. Monthly production at times during the 1930's was as much as 1,000 ounces of gold. The total production and average grade are unknown.

Ore has been extracted from two (2) and possibly three (3) opencast workings along the Say Seng Fault and to a lesser extent, from underground workings. The Malaysian Geological Survey drilled two (2) diamond drillholes (BH-9 & BH-10) in 1964.

Both holes were drilled below a flooded opencast working. BH-9 intersected significant gold grades as indicated below. NBG have undertaken an eleven (11) drillhole programme here with some encouraging results. Key intersections are tabulated in *Table 9-11: Say Seng - Significant Drillhole Intersections*, along with the old Geological Survey drillhole.

Hole No	From (m)	To (m)	Interval (m)	Au Grade (g/t)
BH-09	53.19	53.34	0.15	25.82
	103.02	104.85	1.83	1.40
	106.68	108.81	2.13	1.09
	118.87	119.18	0.31	5.91
	121.92	123.14	1.22	255.03
SSDDH-03	107.00	110.00	3.00	2.95
	112.00	113.00	1.00	2.00
SSDDH-04	92.00	102.00	10.00	15.43
	108.40	109.40	1.00	2.27
	118.55	119.60	1.05	1.95
SSDDH-05	69.00	70.60	1.60	3.26
	123.50	124.50	1.00	1.39
SSDDH-07	150.00	151.00	1.00	2.62
	159.10	159.75	0.65	1.04

Hole No	From (m)	To (m)	Interval (m)	Au Grade (g/t)
SSDDH-08	16.80	17.40	0.60	10.80
	28.00	29.00	1.00	3.29
	30.50	31.60	1.10	1.60
	33.50	34.00	0.50	6.61
	67.70	68.20	0.50	6.00
	70.80	71.70	0.90	7.38
SSDDH-09	0.55	7.80	7.25	1.85
	86.46	87.30	0.84	18.80
	91.40	92.20	0.80	1.82
	101.00	102.60	1.60	4.80
	105.75	108.60	2.85	5.62
	126.70	128.65	1.95	5.84

Table 9-11: Say Seng - Significant Drillhole Intersections

Mineralisation at Sey Seng appears to be controlled by steep structures within limestone, shallow dipping bedding plane parallel features, limestone shale contacts with the Sey Seng fault and intrusive contacts. The Borneo Geological Survey logs describe altered porphyry intrusives and calc-silicate alteration of wollastonite and garnet exoskarn. Mineralisation is associated with high sulphide contents.

The major controlling influence on mineralization is the Say Seng Fault, a high angle reverse fault. Massive Bau Limestone, locally largely recrystallised to white marble, has been upfaulted against black shales of the Pedawan Formation. Porphyry dykes were intruded and mineralisation occurs along the fault zone, as steep feeder structures and locally as veins along bedding planes in the limestone/marble.

The major NNE trending porphyry dyke, that is marginal to the Sey Seng Fault, bifurcates to the South with one branch trending SE. This also is probably fault controlled.

To date, NBG have partially tested around 300 metres of strike with encouraging results. From NBG's observations there is a high probability that the mineralisation at Sey Seng is associated with the intrusives as evidenced by calc-silicate alteration in contact zones with the intrusives and limestone, with steep dipping feeder veins and lateral mineralised off-shoots controlled by bedding planes and/or dilation.

NBG have taken the existing drill data and modeled a small Inferred Resource of 244,000 tonnes at a grade of 3.24 g/t Au for 25,400 oz Au.

In NBG's opinion potential exists to outline high grade ore amenable to underground mining. A more conceptual target also exists related to the margins of the porphyry intrusives underlying the Bau Limestone.

9.2.7.2. Paku

The Paku area lies approximately 4 kilometres NE of the BYG plant site at Bau. The 1965 literature from the Malaysian Geological Survey describes the deposits here as being mainly elluvial Au and Sb, derived from erosion of primary deposits ascending on the shale limestone contact.

From NBG's observations during a brief field visit to the marble quarrying operations now sited here was that there were indications of strong vein controlled stibnite mineralization similar to Say Seng.

There appears to have been little modern systematic exploration here. With the proximity of the mineralisation to the LSC and major NW-SE trending faults (Kojok fault) as well as the intrusive bodies and dykes at Say Seng and NW toward Bukit Sarin, there is scope for Besra to develop some worthwhile exploration targets here.

9.2.7.3. Bukit Sarin

Bukit Sarin lies approximately 4.5 kilometres NE of Bau and is located near the intersection of the NW-SE Kojok Fault and the NE-SW trending Say Seng Fault. The area is described in Wolfenden as comprising quartzose Sb-Au ore in a quartz-shale breccia.

Menzies drilled twenty-five (25) RC drillholes for a total of 3,281 metres at Bukit Sarin. Significant results are highlighted in the *Table 9-12: Bukit Sarin – Significant Drillhole Results* below.

Hole No	From (m)	To (m)	Interval (m)	Au Grade (g/t)
SNRC03	13.0	20.0	7.0	1.13
SNRC04	99.0	104.0	5.0	0.96
	111.0	115.0	4.0	4.42
SNRC07	62.0	73.0	11.0	3.17
	85.0	88.0	3.0	1.25
SNRC09	1.0	3.0	2.0	1.27
	10.0	21.0	11.0	1.46
SNRC10	0.0	18.0	18.0	1.36
	81.0	134.0	53.0	1.10
SNRC13	0.0	20.0	20.0	3.14
	80.0	90.0	10.0	1.13
SNRC16	68.0	72.0	4.0	2.79
	78.0	84.0	6.0	1.09
SNRC19	67.0	68.0	1.0	3.59
SNRC20	1.0	20.0	19.0	3.82

Table 9-12: Bukit Sarin – Significant Drillhole Results

Interbedded shale and sandstone of the Pedawan Formation dominate the geology at Bukit Sarin. The sediments dip 40-50° to the North and have been down faulted against limestone of

Gunung Pangga to the South. A series of sub-parallel dykes up to 30 metres in thickness, that follow bedding planes in the sediments were intersected in drillholes. These dykes appear to connect to an intrusive body that is been exposed in road cuts 100 metres further South. Limestone was intersected at the base of drillholes SNRC01, 02, 10, 11 and 12.

Significant gold mineralisation occurs in many of the drillholes and consists of very fine, almost invisible needles of arsenopyrite hosted in shale, sandstone and to a lesser extent limestone. Better grade intersections are located in sandier and more deformed beds, adjacent to intrusive contacts.

NBG rate Bukit Sarin as having excellent potential to extend the current resource of 45,500 ounces of gold significantly based on its similarities to Jugan and the results of the ridge and spur sampling programme that extended the surface footprint of Au-As_Sb-Tl geochemistry. Further grid based soil sampling; 3D pole-dipole IP surveying is recommended to aid planning drilling to expand the current resource.

9.2.8. Juala Sector

9.2.8.1. Juala West

The Juala West prospect is a further 700 metres SSW on the same road that leads to Arong Bakit and is some 2.7 kilometres SSW of Bau township.

Juala West was the focus of a reasonably intensive exploration programme by NBG during 2007 which culminated in the drilling of ten (10) drillholes targeting the contact zone between intrusive porphyry to the east and Bau Limestone to the West.

Surface sampling and trenching had located several areas of quartz veined stock-worked porphyry and some boulders of highly siliceous skarn and breccia that locally had grades of 95 g/t Au.

Most of the holes intersected narrow vein zones with patchy grades ranging from 5 metres at 1.58 g/t Au in JWDDH-10 to 2.6 metres at 3.25 g/t Au in JWDDH01, with very narrow high grade zones of 0.15 metres at 29.8 g/t Au and 0.2 metres at 11.6 g/t Au in the same drillhole. See *Table 9-13: Juala West - Significant Drillhole Intersections* for list of significant drillhole intersections at Juala West.

Hole No.	From (m)	To (m)	Interval (m)	Au Grade (g/t)
JWDDH-01	156.50	159.10	2.60	3.25
JWDDH-01	168.70	170.20	1.50	3.68
JWDDH-01	254.00	254.50	0.50	1.79
JWDDH-05	49.00	52.52	3.52	2.09
JWDDH-10	23.00	28.00	5.00	1.58

Table 9-13: Juala West - Significant Drillhole Intersections

In 2010 and 2011 further work was completed on reviewing the geophysics and DIGHEM anomalies after NBG successfully intersected high grade skarn mineralisation at the nearby Arong Bakit prospect.

This work showed several large structures associated with the Krian Fault and the intrusive contacts that were thought targets for further high grade skarn and porphyry copper-gold mineralisation. Significant intersections in relation to this are listed in *Table 9-14: Significant Intersections for Juala & Arong Bakit Drilling 2010-2011* below.

Hole No	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Cu (%)	Area
ABDDH-02	65.75	67.00	1.25	1.35	2.17		Arong Bakit
ABDDH-02	76.30	78.80	2.50	12.13	1.67		Arong Bakit
ABDDH-02	112.00	117.70	5.70	1.08	9.25		Arong Bakit
JADDH-01	4.50	5.50	1.00	0.67	1.30	0.10	Arong Bakit
JADDH-01	8.70	9.30	0.60	1.08	0.80	0.08	Arong Bakit
JADDH-01	15.50	15.95	0.45	0.51	0.50	0.04	Arong Bakit
JADDH-01	72.00	74.00	2.00	1.39			Arong Bakit
JADDH-02	162.00	162.70	0.70	0.36	7.10	0.89	Juala
JADDH-02	166.00	169.00	3.00	0.11	1.43	0.18	Juala
JADDH-02	177.00	185.00	8.00	0.11	3.32	0.48	Juala
JADDH-02	276.70	280.00	3.30	0.14	2.33	0.39	Juala
JADDH-04	73.2	73.6	0.4	1.47			Juala

Table 9-14: Significant Intersections for Juala & Arong Bakit Drilling 2010-2011

Drillholes JADDH-02 to JADDH-05 were drilled at Juala West. Holes JADDH-02 and JADDH-04 intersected mineralisation. JADDH-02 was collared in microquartz diorite porphyry and drilled out through the contact into marble passing through a zone from 162 metres to 185 metres of strongly anomalous copper mineralisation, mainly chalcopyrite and bornite in garnet-wollastonite-diopside endo and exoskarn.

Besra Concludes that further assessment of the area is warranted. The main potential lies in the contact zone of the porphyry and marble/limestone. The current Mining Lease boundary limits access for further drill testing and it is recommended that some effort is directed to extend the Mining Lease boundary to allow effective testing.

9.2.8.2. Arong Bakit

The Arong Bakit area lies approximately 2 kilometres SSW of Bau Township. The site is currently being worked as a marble/limestone quarry. The area is proximal to the Juala Intrusives.

The prospect consists of large bluffs of Bau Limestone that have been contact metamorphosed to marble. Of economic interest a number of flat lying veins in the higher part of the bluffs that are auriferous. The Malaysian Geological Survey in 1965 recorded four (4) deposits here (numbered III, IV, V and VI) and steeper structures that have been observed in the current quarry face.

The current quarrying operations have obscured much of the mineralisation at lower easily accessible elevations however there are a large number of boulders derived from the quarry that comprise crackle brecciated marble with the interstices between clasts infilled with arsenopyrite and pyrite. Galena, sphalerite, chalcopyrite and bornite was also observed in some pieces. This tends to average around 10 g/t Au.

The mineralisation here has a strong association with calc-silicate skarn and is in close proximity to the boundary of a large intrusive body of quartz diorite porphyry. The Malaysian Geological Survey assayed twenty-three (23) rock samples from Arong Bakit that ranged from a low of 1.09 g/t Au to a high of 67.12 g/t Au.

NBG have taken twenty-three (23) rock samples of the breccia boulders and other mineralised float and limited outcrop, and consistently obtain anomalous gold values ranging from 0.04 g/t Au to 7.96 g/t Au. The Arong Bakit area is prospective for a substantial porphyry related skarn gold deposit.

In 2010 NBG drilled three (3) holes in the Arong Bakit area to test the contact zone of the Juala Intrusives that are quartz vein stockworked at surface and evaluate the source of the high grade skarn float coming out of the quarry. Drillhole ABDDH-02 intersected 2.5 metres at 12.13 g/t Au in a base metal mineralised garnet-wollastonite-diopside exoskarn. Further drilling of this zone could not be completed due to Mining Licence boundary constraints. Besra believes there is considerable potential here and access is required, and should be sought, to fully test the zone.

9.2.9. Bau Other

There are a number of other mineral occurrences, old mining areas and prospects that have had varying amounts of exploration conducted on them as well as regional geochemical surveys, ranging from sediment sampling, soil sampling, MMI sampling, rock, channel and trench sampling to drill sampling. The more notable occurrences are listed and briefly described below. While outside the known main resource areas it is likely that future deposits will be sourced from some of these prospects. It is important that an exploration strategy is developed to ensure timely evaluation of these and other as yet undiscovered areas are assessed to ensure the long term supply of ore. The following list is not in any particular order of priority or ranking.

9.2.9.1. Saburan

The Saburan Prospect lies on the Tai Parit Fault approximately 1 kilometre North of Taiton A. The entrance to the former mine lies just outside the boundary with Gladioli's ML 108, but the workings extend into Gladioli's ground. The area outside the mining lease is under application by Gladioli. Saburan mineralisation is similar in character to Taiton A and Tabai with grades from underground rock samples collected by NBG up to 9 g/t Au recorded. Exploration here is at an early stage but has similar potential to Taiton A and Tabai-Rumoh.

NBG regards this area highly with exploration potential similar to Taiton.

9.2.9.2. Jebong

The Jebong prospect is located 3kms Southeast of Bau. Exploration has included; construction of road access; re-establishment and infill of grid control to a 50 metre by 50 metre grid pattern; geological mapping of grid lines, road cuttings and creeks; infill auger soil sampling on a 50 metre by 50 metre grid pattern; channel sampling of road cuttings and trenches.

Exploration drilling by Menzies targeted the intrusives and shale limestone contact at Jebong. The best result was 4 metres at 3.21 g/t Au in drillhole JBRC07. EM and ground magnetic surveys have been conducted.

In 1999, further mapping identified high grade (up to 57.4 g/t Au) visible gold mineralisation associated with stibnite and silica localised at the limestone-shale contact adjacent to NW trending faults

Whilst Menzies concluded that the high grade mineralisation had a limited extent they were focussed on the LSC model as their preferred exploration model. NBG have established a number of mineralisation models in the Bau Goldfield that have economic potential and Jebong will be reviewed in light of these.

9.2.9.3. Skiat

The Skiat prospect consists of a shale ridge known as Bukit Punggu Dulang in the Kampung Skiat area 4 kilometres Southeast of Bau. Menzies excavated sixteen (16) trenches around the base of the ridge after locating high grade gold samples (7.5 g/t to 11.7 g/t Au) near the contact with the limestone.

The trenching programme delineated an extensive area of gold mineralisation with similar characteristics to other mineralised LSC deposits such as Pejiru and Bekajang and is associated with ENE striking faults.

A fourteen (14) drillhole RC programme did not identify the source of the high grade mineralisation found in the surface rock chip samples and NBG believes there is scope to locate the source and evaluate the economic potential here.

9.2.9.4. Jambusan

A series of pits lies immediately North of, and parallel to the Ah Onn Road, 4 kilometres East of Bau. The pits were excavated by Chinese in the mid 1800’s mining antimony. Menzies identified the area as a low priority drill target, after rock chip samples from mineralised outcrops exposed in the old mine pits, returned anomalous gold values.

Menzies drilled eighteen (18) RC drillholes (JMRC01-17) for 1,762 metres to test mineralisation adjacent to the old mine pits, developed at the limestone-shale contact (LSC) along a steeply dipping fault and the stratigraphic contact between the Pedawan Formation and the Bau Limestone Formation.

Menzies describes the mineralization as low-grade with a best intersection in drillhole JMRC07 (36-40 metres, 4metres @ 1.69 g/t Au; 112-116 metres, 4 metres @ 4.27 g/t Au).

NBG will review these results and develop exploration targets for testing as the results from previous exploration are encouraging in light of the improved knowledge of the the mineralisation controls for the Bau Goldfield.

9.2.9.5. Ropih

The Ropih prospect is 4 kilometres SW of Bau. Exploration at Gunung Ropih includes gridding, auger soil sampling, geological mapping and rock chip sampling.

Gunung Ropih (Ropih Hill) consists of a quartz-plagioclase porphyry that intrudes the Bau limestone. The intrusive has outcrop dimensions of ~1,000 metres by 600 metres. Precipitous limestone hills surround the intrusive.

The limestone/intrusive contact exhibits recrystallisation of the limestone to marble while the intrusive is intensely altered to sericite-pyrite and kaolinite. The margin of the intrusive is typically silicified with weak to intense skarn development. Skarn minerals that are present are brown andradite garnet, epidote and chlorite.

Massive magnetite mixed with other sulphide crops out at the intrusive's Western contact and near its South-Eastern margin. Patchy outcrop and large boulders of quartz stockworking cutting intrusive lie at the South-Western contact of the intrusive. Disseminated chalcopyrite, chalcocite and bornite have been observed where skarn minerals are present. The Southern margin of the intrusive contains the greatest abundance of copper sulphides.

'C' horizon auger soil samples outlined three (3) areas of anomalous geochemistry, one related to an area of disseminated sulphides and quartz veining in the centre of the intrusive, a second related to skarn mineralisation at the Southern margin of the intrusive and a third related to breccia and disseminated sulphides in the intrusive.

Rock chip sampling of float and outcrop returned gold values better than 0.5 ppm in six (6) samples. The Ropih prospect has potential to locate a porphyry style disseminated copper-gold and skarn related copper-gold mineralisation and in NBG's opinion warrants further testing.

9.2.9.6. Sebaang

Sebaang lies around 2 kilometres W of Bau near the Sawarak River. Work by Menzies culminated in the drilling of twelve (12) RC drillholes and the excavation of twenty-three (23) trenches. The target was the LSC and the NE trending Sebaang Fault zone.

Trenching intersected some higher grade zones at 6.55 g/t Au in Trench 8, for example. The drilling produced anomalous gold and arsenic with one of the drillholes (SERC05) intersecting, 12 metres at 1.98 g/t Au.

NBG consider that this prospect requires further testing to fully evaluate its economic potential.

9.2.9.7. Bau Lama

Bau Lama has been mentioned earlier with respect to its tailings potential; however a proportion of the mineralisation is described as primary elluvial auriferous clays. Potential for primary mineralisation remains untested as far as NBG have been able to determine.

9.2.9.8. Buroi

The Buroi area located approximately 9 kilometres WSW of Bau is described as a limestone hosted quartz-calcite-stibnite vein. BYG undertook soil sampling in 1980 from which nine hundred and twenty-three (923) soil samples were analyzed for Au. Results ranged from 0.01 g/t to 0.82 g/t Au. They drilled ten (10) drillholes but the results are not recorded. Menzies drilled eleven (11) RC drillholes but didn't hit any significant mineralisation.

NBG understands that the Menzies work did not target the outcropping mineralization but rather a geophysical anomaly.

They noted the presence of antimony mineralisation and old workings at the LSC, and the anomalous gold values from rock chip samples collected from trenches evidencing that mineralising fluids have passed through the area. NBG consider that the area is underexplored and more detailed work is warranted particularly in the area between Buroi and the nearby Pejiru Gold Deposit.

9.2.9.9. Batu Sepit

Batu Sepit is located 5 kilometres SW of Bau. Menzies soil sampled the area, with fifteen (15) of one hundred and sixty-one (161) samples giving values of between 5 and 87 ppb Au. Several rock float samples in creeks draining the area had anomalous gold. Further ridge and spur sampling expanded the area of anomalous gold between the Tai Parit and Tongga Faults with a highest value of 314 ppb Au. Menzies subsequently drilled eleven (11) RC drillholes here with encouraging results in several of the holes. For example, drillhole BSRC04 assayed 1.5 g/t Au over 24 metres from 20 metres downhole, while hole BSRC08 assayed 1.46 g/t Au over 4.0 metres from 48 metres downhole.

NBG believe there is considerable potential here to define further resources in close proximity to the other Pejiru deposits.

9.2.9.10. Traan

The Traan area lies ~6km SW of Bau. Gold mineralisation is associated with silicified and brecciated shale and intrusives exhibiting disseminated stibnite and pyrite.

BYG drilled ten (10) shallow Winkie diamond holes at the NE end of Traan with high grade gold values over narrow widths in two (2) of the holes (3 metres at 21.03 g/t Au; 3 metres at 11.49 g/t Au).

Surface rock chip and float samples have assayed up to 14.5 g/t Au and 150 g/t Ag.

Menzies in 1997 drilled five (5) shallow RC drillholes to test the intrusive contact zone with the shales and marble. One (1) drillhole TNRC04 returned significant assays of 3.11 g/t Au over 4 metres from 60 metres downhole.

Again the prospect is underexplored and NBG believes potential exists here to discover further economic gold mineralisation.

9.2.9.11. Sebwad

The Sebwad prospect lies 7 kilometres S of Bau. Creek float composed of silicified intrusive containing base metal sulphides and anomalous gold were traced to Sebwad in the early 1990's. Exploration since then has comprised gridding, road construction, soil sampling, mapping, trenching and RC drilling of twenty-seven (27) drillholes.

Mineralisation is associated with the dacite porphyry intrusives into limestone and shale, with silicification and quartz vein stockwork. The quartz veins are chalcedonic with fine sulphides.

Eight (8) veins composed of quartz, marcasite, arsenopyrite, sphalerite and minor galena and chalcopyrite exposed by the earthworks are hosted in the intrusive and they vary from 1 centimetre to 200 centimetres in thickness. The thicker veins are always flat lying and rarely exceed 30 metres in length. Samples of the vein material assayed up to 84.41 g/t Au. They were not considered significant enough by past explorers to warrant further investigation.

The highest assays values in drilling referred to in the Menzies data comprise 8 metres at 1.10 ppm gold, in drillhole SBRC01, across a quartz vein.

NBG notes the association of gold mineralisation with intrusives, chalcedonic silica, arsenopyrite and low temperature marcasite. This prospect requires assessment on the basis of its similarity to deposits/prospects like Sirenggok and may have considerable potential for economic mineralisation.

9.2.9.12. Seromah

The Seromah Prospect, 8 kms SE of Bau has undergone exploration programmes involving gridding, road construction, trenching, mapping, rock chip and channel sampling.

The geology at Seromah is dominated by Bau Limestone, shale of the Pedawan Formation and dacite porphyry dykes and stocks. Shale overlies the limestone and bedding is generally flat lying. Dacite porphyry has intruded the shale and limestone. The intrusives are variably replaced by silica, clay, chlorite and calcite.

The results of sampling show two (2) areas contain widespread anomalous gold values. One area at the East side of the grid consists of an East striking zone of silicified shale breccia that forms a ridge (Triangle Area). The second area is composed of brecciated and silicified shale and radiolarian chert lying above limestone close to the eastern edge of the dacite porphyry intrusive at Bukit Lidau.

Some fourteen (14) shallow RC drillholes have been drilled with a best intersection of 4 metres at 0.89 g/t Au in drillhole SMRC01.

The area is underexplored and NBG recommend further follow-up exploration to determine its economic gold potential.

9.3. Planned Exploration Programmes

Besra's main focus is on getting the Bau Project into production in the shortest possible time frame.

For the medium to long term development of the Bau Project it is essential that Besra also prioritise ongoing exploration to realize the full potential of the Bau Goldfield. Programme implementation subject to funding and available labour resources.

The programme is structured to:

- Develop high priority exploration targets to maximise resource development opportunities to grow Company's precious metal asset base;
- Carry out programmes of surface exploration, geophysics and drilling of prioritised targets;
- Select those sectors for drill testing that are likely to produce economically significant results in the shortest space of time;
- Contemporaneously undertake programmes of surface exploration to upgrade the level of knowledge leading to development of further high priority targets from the approx. 34 plus known areas, and new areas, to provide a pipeline of development projects.

Programme Summary	No of Drill Holes	Quantity or Rate	Unit
HIGH PRIORITY TARGET AREAS			
JUGAN			
Soil Sampling			
Detailed 25m centre programme over ridge and spur soil anomalies and over chargeability/resisivity anomalies		2000.00	samples

Programme Summary	No of Drill Holes	Quantity or Rate	Unit
identified in the 3D pole-dipole survey.			
Hychips analysis		2000.00	samples
Geophysics			
Interpolate all data sets and define drill targets. Extend 3-D IP survey to cover Bukit Sarin and extensions to Jugan		2	surveys
Exploration Drilling:			
Deep Holes at Jugan	2.00	1000.00	m
Bukit Sarin	5.00	1000.00	m
New Targets	5.00	1000.00	m
Total holes and metres	**12.00**	**3000.00**	**m**
SIRENGGOK			
Channel Sampling			
1m channels to expand 2008 programme and define drill targets		2,000.00	samples
Geophysical Target Evaluation			
Evaluate and review validity of geophysical targets.			
Geological Mapping			
Geological mapping of Sirenggok focussing on hydrothermal alteration, structure, lithology and mineralisation.			
Exploration Drilling:			
Chen's Breccia Resistor	2.00	600.00	m
Water Tank Resistor	2.00	600.00	m
Depth and orientation of main ore zone	2.00	600.00	m
Scout holes to follow up channel sampling	5.00	750.00	m
Total holes and metres	**11.00**	**2,550.00**	**m**
PEJIRU			
Remote sensing evaluation of structure/old mine pits			
Use available datasets and interpretations to generate new targets in Pejiru Structures			
Ground Confirmation and Geological Mapping			
Soil Sampling/Trenching			
Programmes of soils/trenching to test structure and ground truth geophysics		2,000.00	samples
Geophysics			
Review past geophysics and generate targets, look at least one IP survey here.			
Exploration Drilling:			
Drill test large conductor beneath Pejiru deposit	1.00	350.00	m
Review and target holes where current resource open ended	10.00	1,000.00	m
Drill test other targets as they are developed	10.00	1,500.00	m
Total holes and metres	**21.00**	**2,850.00**	**m**
TAITON B			

Programme Summary	No of Drill Holes	Quantity or Rate	Unit
Surveying			
Establish shape of workings and voids			
Underground mapping			
Mapping to show areas of vein widening and possible plunge to vector grade at depth.			
Channel Sampling			
Underground channel sampling,		1400	samples
Exploration Drilling:			
Holes to intersect 50m and 100m below vein exposure at 100m staggered centres	3.00	600.00	m
	3.00	800.00	m
Total holes and metres	**6.00**	**1,400.00**	**m**
BAU CENTRAL OTHER TARGETS			
SAY SENG			
Geological Review, Mapping and Sampling			
BEKAJANG			
Data Review and Target Generation			
Review in particular targets at Gumbang, Krian, BYG Pit, SW Tai Parit and Bekajang Lake			
Soil Sampling and Mapping			
Soils and mapping required over intrusives at Bekajang/Gumbang for Sirenggok look- a-likes		500.00	
Exploration Drilling:			
BKG North Exploration Drilling (including consumables and factors)	6.00	600.00	m
BYG-Krian Exploration Drilling (including consumables and factors)	3.00	600.00	m
SW Tai Parit Exploration Drilling (including consumables and factors)	6.00	600.00	m
Total holes and metres	**15.00**	**1,800.00**	**m**
BAU REGIONAL EXPLORATION PROGRAMME			
Data Compilation, Evaluation and target Ranking			
Ground follow up of Ranked Targets of Rawan Block and Area C			
Carry out mapping, sampling, trenching to generate drill targets.			
TOTAL EXPLORATION DRILL HOLES AND METRES FOR ONE YEAR PROGRAMME	**65.00**	**11,600.00**	**m**

Table 9-15: Summary Table of Proposed Exploration Programmes for Bau Project

The recommended programme if executed in a systematic manner with the appropriate ground work done prior to drilling will see the discovery of new resources and additional extensions to known resources.

A programme as recommended will see early drilling on those prospects that are sufficiently advanced, these being Jugan, Sirenggok and Pejiru.

Integration of all geophysical data sets focusing on the targets generated by the recent 3D-IP survey and the evaluation of the soil anomalies will bring several of these anomalies to drill status and drilling should proceed as soon as possible, as it is likely discoveries in the shale basin will be metallurgically similar to Jugan.

At Bukit Sarin (Jugan West) drilling is recommended to step out from the old Menzies holes and test the extensions indicated by the soil geochemistry.

At Sirenggok the basic infrastructure needs to be put in place; however, several targets can be tested relatively quickly once landowner access is agreed, drill positions finalised and roading established. The other targets are dependent on completion and interpretation of the channel sampling programme.

At Pejiru, preliminary remote sensing analysis has identified some prospective looking target areas; these require ground truthing before embarking on a major drill programme, however, several of the more compelling targets could be drilled earlier by relying on previous Menzies exploration data.

At Taiton B, ideally the underground mapping programme should be completed, however, holes could be drilled here as access is easy and it is on a granted ML.

Bau Central – Other targets include BYG Pit, Krian, Bekajang Lake, Say Seng and SW Tai Parit. These all require further ground work/data evaluation to firm up the targets prior to drilling.

The regional Bau programme to evaluate the remainder of the known prospects, advance them, and develop new targets requires a committed and continuous programme as outlined.

10. Drilling

10.1. General

The Bau Goldfield within the project area has had a number of drill programmes focused on the various deposits and prospects. The first modern drilling was carried out by the Borneo Geological Survey in the 1960's. After the Ling family gained control of the principal deposits and prospective ground, drilling campaigns were undertaken by companies associated with their interests and by a number of joint venture companies up until the involvement and formation of North Borneo Gold Joint Venture Company in late 2006. Most of the historic drilling was shallow, testing less than 100 m vertically below surface.

In September 2010 NBG commenced several programmes of resource drilling in the Taiton, Bekajang and Jugan Sectors as well as exploration drilling in the Taiton and Juala Sectors. These programmes were completed on 18 September, 2012.

10.2. Historic Drilling - Prior 2007

A total of more than 175,000 metres in two thousand, one hundred and fifty-six (2,156) drillholes is recorded in the historic drill database and from additional drillhole data located by TMCSA/NBG in archived records. A further two hundred and thirty-seven (237) shallow hand auger holes were drilled to define the tailings resource in the BYG tailings dam. Additional auger holes have also been drilled over other old tailings areas but have not been itemized separately.

Many of the early diamond holes by Bukit Young were drilled in BQ (some NQ) using Winkie rigs, NQ using a Longyear 28 and HQ/NQ using a Korean rig. Diamond drilling by RGC and Gencor was largely HQ and used more substantial diamond rigs such as Longyear 44's.

Renison Goldfields (RGC) was the only company to routinely take downhole surveys during this period and they were responsible for most of the deeper holes.

The Menzies/BYGS programmes used reverse circulation methods. Rigs used were a Schramm T4 and a G&K850. Samples were collected through cyclones and sampled using a spear when sample was dry. Initially air volumes were insufficient to keep samples dry below the water table, and samples were simply collected wet from base of cyclone. Fine material in suspension could not be captured in water overflow hence there are some inherent shortcomings in this drilling method in wet environments.

Table 10-1: Summary of Drilling Completed pre 2007 gives a summary of the drilling completed on the Bau Goldfield up to 2007.

Project	Companies	Drill Type	No. of Holes	Total Metres
Jugan	Renison Goldfields (RGC), BYGS, Gencor	Diamond; BQ, NQ and HQ	86	7,743.05
Jugan	BYGS, Menzies	Reverse Circulation	82	9,716.00
Sirenggok	RGC, BYG	Diamond	48	7,798.95
Sirenggok	BYGS/Menzies	Reverse Circulation/Diamond Tail	3	792.90
Sirenggok	BYGS/Menzies	Reverse Circulation	13	1,166.00
Bekajang-Krian	Geol Survey, RGC, BYG	Diamond	360	28,857.94
Bekajang-Krian	BYGS/Menzies	Reverse Circulation	310	28,935.00
Pejiru	Geol Survey, BYGS/ Menzies	Diamond	20	2,477.96
Pejiru	BYGS/Menzies	Reverse Circulation	682	49,380.50
Taiton	BYG	Diamond	177	8,752.43
Taiton	BYGS/Menzies	Reverse Circulation	120	9,841.00
Juala West - Arong Bakit	BYG	Diamond	21	844.14
Sey Seng	Geol Survey	Diamond	2	269.75
Other prospects	BYG/RGC/BYGS	Diamond	4	353.70
Other prospects	BYGS/Menzies	Reverse Circulation	228	18,410.00
TOTAL			**2,156**	**175,339.32**

Table 10-1: Summary of Drilling Completed pre 2007

10.3. Drilling by North Borneo Gold 2007 – 2008

Two contractors were used during this programme, Drillcorp Sdn Bhd, a Malaysian based company and CDSI from the Philippines. The Drillcorp rig was a Boyles BBS-10, while the CDSI rig was a Christensen-Boyles CS1000 skid mounted rig. All holes were drilled in HQ triple tube core size.

NBG in the later part of the programme purchased a Winkie Rig to drill AQ sized core for geochemical sampling purposes. A total of five (5) shallow holes were drilled with this machine. Two (2) at Sirenggok, two (2) at Pejiru and one (1) at the BYG pit.

Table 10-2: Summary of Drilling Completed by NBG to 2008 shows details of the drilling completed at each project area by NBG since the joint venture was established up until 2008.

Project	Drill Type	No. of Holes	Total Metres
Jugan	Diamond HQ	4	310.00
Sirenggok	Diamond HQ	6	1,250.30
Sirenggok	Diamond AQ	2	154.95
Bekajang-Krian	Diamond HQ (10 holes), AQ (1 hole)	11	669.90
Pejiru	Diamond (AQ)	2	126.85
Taiton	Diamond	4	532.15
Juala West	Diamond	10	1,018.40
Sey Seng	Diamond	11	1,719.45
TOTAL		**50**	**5,782.00**

Table 10-2: Summary of Drilling Completed by NBG to 2008

10.4. Drilling by North Borneo Gold 2010 – 2012

From September, 2010 to September, 2012 North Borneo Gold Sdn Bhd drilled a total of 40,031.05 metres of drilling in two hundred and eight (208) drillholes. These comprised resource, exploration and metallurgical holes. The drill contractor was DrillCorp Sdn Bhd a Malaysian based drilling contractor.

All holes were collared in PQ generally until good ground conditions were encountered and then the holes reduced to HQ. All drilling operations were triple tube with either 1.5 metre or 3 metre core barrels as drilling conditions dictate. Only rarely were holes reduced to NQ when drill conditions dictated.

Drillhole surveys are conducted routinely at 25 metre intervals down hole for resource holes and at 50 metre intervals for exploration holes. Initially a Chinese made HKCX single shot camera with conventional film was used. This was replaced by a Camteq 'ProShot' electronic multi-shot camera.

All drill core where geological conditions allowed were oriented at the end of each 3 metre run. Early in the programme this was achieved by an orientation spear and then progressed to the use of an electronic 'OriShot' orientation device. The drillers mark the base of the drill core at the end of the run and mark the base line of the core axis. This is checked by the NBG site geologist for accuracy and consistency.

Table 10-3: Summary of Drilling Completed by NBG to 18 September 2012 below lists the drilling by project/sector, category and year.

Project/Sector	Category	Drill Type	Year	No. of Holes	Drill Metres
Juala	Exploration	PQ/HQ Diamond Core	2010	2	370.40
Juala	Exploration	PQ/HQ Diamond Core	2011	5	1,559.90
Taiton	Exploration	PQ/HQ Diamond Core	2010	18	3,756.80
Taiton	Exploration	PQ/HQ Diamond Core	2011	6	882.70
Taiton	Resource	PQ/HQ Diamond Core	2010	13	2,241.30
Taiton	Resource	PQ/HQ Diamond Core	2011	41	6,688.65
Bekajang	Exploration	PQ/HQ Diamond Core	2011	2	713.50
Bukit Young	Exploration	PQ/HQ Diamond Core	2011	3	356.60
Bukit Young	Resource	PQ/HQ Diamond Core	2011	39	6,065.80
Jugan	Resource	PQ/HQ Diamond Core	2011	24	3,374.50
Jugan	Metallurgical	PQ Diamond Core	2011	2	176.00
Jugan	Resource	PQ/HQ Diamond Core	2012	47	13,342.10
Jugan	Metallurgical	PQ Diamond Core	2012	6	502.80
Total				***208***	***40,031.05***
Total by Year			***2010***	***33***	***6,368.50***
			2011	***122***	***19,817.65***
			2012	***53***	***13,844.90***
Total by Type	***Exploration***			***36***	***7,639.90***
	Resource			***164***	***31,712.35***
	Metallurgical			***8***	***678.80***

Table 10-3: Summary of Drilling Completed by NBG to 18 September 2012

11. Sample Preparation, Assaying & Security

11.1. General

There have been several companies involved with the Bau project since the 1980's whose data is incorporated and has been used in the compilation of databases used for the resource evaluation being reported herein.

These are principally, Bukit Young Gold Mines, who mined Tai Parit, BYG pit, Taiton, Umbut and a number of other deposits in the district. They had their own mine laboratory which they used for general assaying and grade control work.

Subsequent companies such as Gencor at Jugan, Renison Goldfields, BYGS/Menzies Gold all used this laboratory to varying degrees, but with rigorous check assaying and use of alternative laboratories in some instances.

Following the merger of Zedex Gold and Olympus Pacific Minerals (now Besra) in 2010, NBG decided to enter a contract with SGS Laboratories to set up an accredited sample preparation and fire assay facility at the Bau mine site to ensure that QAQC procedures for sample preparation, assaying and security were up to industry best practice. To that end all gold assays have since 2010 have been performed by SGS on site and all minor elements analysed by SGS at their Port Klang facility in Kuala Lumpur or at their facilities in Perth Australia.

11.2. Sampling Method & Approach

11.2.1. Prior to North Borneo Gold

11.2.1.1. General

Prior to the formation of NBG, exploration at Bau had mainly been carried out by Bukit Young Group, Gencor/Minsarco (Jugan), Renison Gold Fields (RGC) and Bukit Young Group Services (BYGS)/Menzies Gold.

Data relevant to the resources under consideration was reviewed. It was noted that there have been issues particularly with respect to the BYG mine assay laboratory, and these issues are addressed in *Section 12*.

11.2.1.2. Surface and Underground Sampling

With respect to surface and underground channel sampling, the many original sample maps and sections were reviewed and in general they have been found to be adequate for resource estimation purposes where positions and survey control could be verified. Where data could not be verified it was excluded from the database.

11.2.1.3. Historic Drill Core

Observations of historic drill core shows that all previous companies involved systematically geologically logged data onto paper logs with adequate geological descriptions, sample intervals marked, and correlated to assay data, to lead to the conclusion that systematic procedures were followed in most cases to the accepted standard at the time. It is noted that much of the early core drilling by BYG was BQ size and was split by core splitter. Since the late 1980's however all drill core was split by diamond saw. The majority of this drill core is still available.

RGC, Gencor and BYGS/Menzies predominantly used HQ core and examination shows that all drill core was logged and sampled systematically, captured on paper logs and transferred to digital format.

11.2.1.4. Reverse Circulation Drilling

BYGS/Menzies gold used reverse circulation drilling for the majority of their drilling at Bau.

The sampling procedure used by Menzies involved sample collection at 1 metre down hole intervals with rock samples collected through a cyclone into sample bags. Samples for assay were collected by using a "spear", which involves inserting a 4-inch diameter tube down the centre of a 1 metre sample bag until it reaches the bottom of the bag. This was then placed into the 1 metre sample bag. From this bag a second split was collected using the same procedure but with a 2-inch spear. These second splits were composited into 4 metre intervals for assay. When composites assayed greater than 0.5 g/t Au, the original 1 metre samples were then assayed.

All the Menzies RC holes were geologically logged and geological codes assigned on paper logs. Data was manually entered and for the most part was systematically and accurately done.

This data has been reviewed and it is concluded that the sampling method and approach used historically is adequate for the purpose that it is being used for in this report and that errors or discrepancies found have been rectified where possible.

11.2.2. North Borneo Gold Prior to 2010

11.2.2.1. Surface and Underground Sampling

NBG prior to 2010, before the merger of Zedex and Olympus Pacific Minerals, (now Besra) completed programmes of surface and underground rock chip outcrop and float sampling, and surface channel sampling on Sirenggok, Taiton, Krian, Sey Seng, Arong Bakit and Juala West.

A channel sampling programme at Sirenggok of available road outcrop was partially completed in 2008.

Samples were collected, surveyed with GPS and/or tape and compass and entered into an electronic sample register.

11.2.2.2. Drill Core Handling & Logging Protocol

North Borneo Gold drilled all holes as HQ triple tube since the inception of the joint venture with Gladioli. Drill core was placed by the contractor into metre long core trays with the runs marked by core blocks. Core barrels range from 1.5 metres to 6.0 metres depending upon ground conditions. The driller keeps a record of each drill run in a daily drill log sheet which is signed by the drill company's and NBG's representative each day.

1. The supervising geologist/junior geologist completed a skeleton log and measured core recovery on site before transport by 4WD vehicle back to the BYG sampling facilities. Drill core was covered and secured to minimise disruption of core during transport from the drill sites.
2. The core was received at the logging facility. The core was marked out, cleaned and photographed, core recoveries measured and geotechnically logged.
3. The junior geologist and supervising geologist geologically logged the core onto standard paper geological logging sheets, the data from which are then entered in the Company's computer database.
4. The geological staff selected the mineralized intercepts and marked out the intervals for sampling. Sample intervals were generally selected based on geological contacts and/or at 1m intervals, which ever were the lesser. General practice was to sample several metres either side of mineralized intercepts.
5. The drill core was then passed to the sample preparation staff.

In general terms the procedures followed for drill core handling and processing prior to 2010 were consistent with standard industry practice.

All drill hole collars drilled by NBG before 2010 have been now been surveyed using registered surveyors from Kuching.

It was noted that NBG did not carry out downhole surveys before 2010 however this has been rectified in drill programmes since 2010 and downhole surveys are routine on all drill holes.

11.2.3. North Borneo Gold 2010 to 2013

11.2.3.1. Drill Core and Logging Protocol

During 2010, the core handling and logging, rock sample and soil sample handling systems were reviewed and overhauled. Core and sample handling facilities were upgraded and expanded to accommodate the increased volumes for the programmes carried out from August 2010.

1. Under the supervising geologist's control, site geologists and field crew visit the rigs twice per day. There they record daily progress and produce skeleton logs for that day's drilling which are recorded and plotted in a daily progress report circulated to management. This ensures that the geology and mineralization of the hole is recorded so that drill targets are tested as to the drillhole design, modified if necessary and holes

are terminated in a timely manner. The site geologist also liaises with the drillers and any issues that arise are dealt with. The site geologist ensures that downhole surveys and orientations are being carried out in accordance with the agreed protocols. The field crew under the supervision of the site geologist then packages and secures the drill core on site and from where it is loaded and transported by 4WD vehicle to the Company's core processing facility in Bau.

2. Drillhole surveying and orientation readings. All drill holes are routinely surveyed using either single shot or multishot downhole cameras. In the early programmes notably at Arong Bakit and the early part of the Taiton programme single shot Eastman or HKCX (Chinese made) downhole cameras were used. For the most part however Camteq Proshot multishot electronic cameras were the norm. Drillhole surveys were taken every 25 metre downhole for all resource drill holes and at 50 metre downhole intervals for purely exploration holes. As most holes were dual purpose the majority of drilling has surveys at 25 metres. Each hole was also surveyed at its termination. Orientation data was collected electronically using an Orishot orientation device. This was routinely done at the end of each HQ drill run where the driller judged he would be able to appropriate to obtain usable information. Drill runs normally ran with the core barrel length of between 1.5 metres and 3.0 metres.

3. Once core is received at the core shed the hole is assigned to a geologist who is responsible for the processing of that particular hole. The hole is firstly measured on a run by run basis and marked out in 1 metre intervals. Core recoveries are completed and any issues with discrepancies between drill runs as recorded and as measured are rectified.

4. The drill core is then photographed with each tray clearly marked with drillhole identification and the interval from beginning of the tray to the end of the tray. Each tray is photographed wet and dry. All photos are collated electronically and indexed.

5. Logging: A variety of geological and related data is captured. All data is input directly into ruggedized laptops with standardized excel spreadsheets and dropdown menus linked to look up tables. This has been designed for later migration to the GeoMIMS platform that is currently being implemented. Data is transferred twice daily to the Company's server.

   - **Drillhole Lithology Log**

   Figure 11-1: Lithology Log Example below is extracted from the logging template and shows an example of the lithology information captured. The standard lithology codes and descriptions, formation codes and descriptions, colour codes and descriptions, colour intensity and description, oxidised percentage and description are listed in *Appendix 11.1*. These codes are common to all the project areas drilled. An additional column is included in the logging template where other lithological observations can be made by the geologist.

HOLE_ID	FROM	TO	L_CODE	L_DESC	FORMATN	FORMATN_DESC	COLOUR	COL_DESC	COL_INT	INT_DESC	OXD_PERC	OXD_DESC	L_OBS
JUDDH-80	199.3	200	FT	Fault	P	Pedawan Formation	GY	Gray	L	Light	1	Unoxidised	FAULT ZONE: Pale grey, intensely clay altered medium grained sandstone, brecciated towards the contact margin, cut by few ankerite stringers and few calcite veinlets. Traces of very fine grained pyrite disseminations (py 0.5%).
JUDDH-80	200	207.6	SH-SL	Interbedded Shale-Siltstone	P	Pedawan Formation	BK	Black	L	Light	1	Unoxidised	SHALE-SILTSTONE: Dark grey to black, partly laminated with flaser bedding in certain parts, cut by weak to moderate ankerite and weak calcite-dickite stringers mostly along sandstone beds. Localised coarse grained sandstone beds noted. Specks of very fine grained pyrite present (py 0.1-0.5%).
JUDDH-80	207.6	211.85	SH	Shale	P	Pedawan Formation	BK	Black	L	Light	1	Unoxidised	SHALE: Dark grey to black, silty, moderately fractured, cut by weak to moderate ankerite, minor calcite stringers and some fractures healed with dickite. Traces of very fine grained pyrite noted (py 0.1%).
JUDDH-80	211.85	213.45	SS	Sandstone	P	Pedawan Formation	GY	Gray	D	Dark	1	Unoxidised	SANDSTONE: Dark grey, coarse grained and laminated, cut by moderate ankerite and calcite stringers. Specks of very fine grained pyrite (py 0.5%).
JUDDH-80	213.45	222.15	SH-SL	Interbedded Shale-Siltstone	P	Pedawan Formation	BK	Black	M	Medium	1	Unoxidised	SHALE-SILTSTONE: Black, partly laminated with flaser bedding in certain parts, moderately fractured, cut by weak to moderate ankerite and weak calcite-dickite stringers mostly along sandstone beds. Localised coarse grained sandstone beds and narrow shear zones noted. Specks of very fine grained pyrite present (py 0.1-0.5%).
JUDDH-80	222.15	223.8	FT	Fault	P	Pedawan Formation	BK	Black	M	Medium	1	Unoxidised	FAULT BRECCIA: Black, brecciated shale-siltstone with clay alteration in certain parts, cut by few ankerite stringers. Traces of very fine grained pyrite disseminations (py 0.1%).

Figure 11-1: Lithology Log Example

- **Drillhole Alteration Log**

Figure 11-2: Drillhole Alteration Log - Part 1 to *Figure 11-4: Drillhole Alteration Log - Part 3* display an example of the Alteration log sheet in three (3) parts so as to fit into page width. This shows the logging template used to record hydrothermal alteration, mineralisation and sulphide types and content of the mineralised zones. The codes and descriptions for this logging template are shown in *Appendix 11.1*.

HOLE_ID	FROM	TO	ALT_INTVL	ALT_TYPE	ALT_TYPE_DESC	ALT_STYLE	ALT_STYLE_DESC	ALT_INT	ALT_INT_DESC	ALT_TYPE2	ALT_STYLE2	ALT_INT2
JUDDH-10	0	0.5	0.5	CLX	Clay altered-Oxidized	PV	Pervasive	5	Intense			
JUDDH-10	0.5	5.3	4.8	CLX	Clay altered-Oxidized	PV	Pervasive	4	Strong			
JUDDH-10	5.3	10.4	5.1	UN	Unaltered	SP	Semi-pervasive	3	Moderate	CLX	FD	2
JUDDH-10	10.4	17	6.6	UN	Unaltered	PV	Pervasive	4	Strong	CL	FD	1
JUDDH-10	17	26.15	9.15	CL	Clay altered	FD	Fracture controlled	3	Moderate	UN	PV	3
JUDDH-10	26.15	26.7	0.55	RCO	Sericite-Carbonate-Chlorite	PV	Pervasive	4	Strong			
JUDDH-10	26.7	36	9.3	RCO	Sericite-Carbonate-Chlorite	PV	Pervasive	4	Strong	X	SP	3
JUDDH-10	36	36.45	0.45	RCO	Sericite-Carbonate-Chlorite	PV	Pervasive	4	Strong			
JUDDH-10	36.45	41.7	5.25	CL	Clay altered	FD	Fracture controlled	3	Moderate	UN	PV	3
JUDDH-10	41.7	58.5	16.8	UN	Unaltered	PV	Pervasive	4	Strong	CL	FD	2
JUDDH-10	58.5	70	11.5	UN	Unaltered	PV	Pervasive	4	Strong	CL	FD	1
JUDDH-10	70	80.8	10.8	UN	Unaltered	PV	Pervasive	4	Strong	CL	FD	2

Figure 11-2: Drillhole Alteration Log - Part 1

HOLE_ID	FROM	TO	MIN_TYPE	MIN_TYPE_DESC	MIN_STYLE	MIN_STYLE_DESC	MIN_INT	MIN_INT_DESC	SULPHIDE	SULPHIDE_DESC	S_STYLE	S_STYLE_DESC	S_PERC
JUDDH-10	0	0.5											
JUDDH-10	0.5	5.3											
JUDDH-10	5.3	10.4	P	Pyrite	DI	Disseminated	2	Weak	P	Pyrite	FD	Fine grained disseminated	1
JUDDH-10	10.4	17	H	Pyrite-Arsenopyrite	DI	Disseminated	4	Strong	A	Arsenopyrite	FD	Fine grained disseminated	3
JUDDH-10	17	26.15	H	Pyrite-Arsenopyrite	DI	Disseminated	4	Strong	P	Pyrite	FD	Fine grained disseminated	3
JUDDH-10	26.15	26.7	P	Pyrite	SC	Specks	1	Incipient	P	Pyrite	SC	Specks	0.1
JUDDH-10	26.7	36	P	Pyrite	SO	Spots	1	Incipient	P	Pyrite	SO	Spots	0.1
JUDDH-10	36	36.45	P	Pyrite	SC	Specks	1	Incipient	P	Pyrite	SC	Specks	0.1
JUDDH-10	36.45	41.7	H	Pyrite-Arsenopyrite	DI	Disseminated	1	Incipient	P	Pyrite	SC	Specks	0.5
JUDDH-10	41.7	58.5	H	Pyrite-Arsenopyrite	DI	Disseminated	1	Incipient	P	Pyrite	FD	Fine grained disseminated	0.5
JUDDH-10	58.5	70	H	Pyrite-Arsenopyrite	DI	Disseminated	2	Weak	A	Arsenopyrite	FD	Fine grained disseminated	1.5
JUDDH-10	70	80.8	H	Pyrite-Arsenopyrite	DI	Disseminated	2	Weak	P	Pyrite	FD	Fine grained disseminated	1

Figure 11-3: Drillhole Alteration Log - Part 2

HOLE_ID	FROM	TO	VN_TYPE	VN_TYPE_DESC	VN_WIDTH	VN_PER_METRE	VN_VOL_PERC	BRC_TYPE	BRC_TYPE_DESC	BRC_WIDTH	BRC_INT	BR_INT_DESC	ALT_OBS
JUDDH-10	0	0.5											
JUDDH-10	0.5	5.3											
JUDDH-10	5.3	10.4	VR	Carbonate stringers	0.001	13	0.255						
JUDDH-10	10.4	17	VR	Carbonate stringers	0.003	19	0.864						
JUDDH-10	17	26.15	VR	Carbonate stringers	0.001	8	0.087						
JUDDH-10	26.15	26.7	VR	Carbonate stringers	0.001	1	0.182						
JUDDH-10	26.7	36	VR	Carbonate stringers	0.001	4	0.043						
JUDDH-10	36	36.45	VR	Carbonate stringers	0.001	1	0.222						
JUDDH-10	36.45	41.7	VR	Carbonate stringers	0.003	10	0.571						
JUDDH-10	41.7	58.5	VR	Carbonate stringers	0.001	18	0.107						
JUDDH-10	58.5	70	VS	Sulphidic	0.003	6	0.157						
JUDDH-10	70	80.8	VS	Sulphidic	0.015	2	0.278						

Figure 11-4: Drillhole Alteration Log - Part 3

- **Drillhole Structure and Veining Log**

The drillhole and structural logging records all structural orientation data and vein orientation data that can be captured and in particular related to the oriented core to enable calculation of dip and dip direction of veins, faults, joints and breccias. The program used to perform the calculations is Geocalc – developed by the University of Queensland in Brisbane Australia. An example of the logging sheet is shown below in *Figure 11-5: Drillhole Structure & Veining Log* with the related codes listed in *Appendix 11.1*.

HOLE_ID	S_DEPTH	DIP	DIP_DIRN	THCKNESS	ORIENTN	CONFIDENCE	DEFORM_INT	STRUCT	STRUCT_DESC	INFILL	INFILL_DESC	ROUGHNESS	ROUGH_DESC	ALPHA_ANG	BETA_ANG	DELTA_ANG	STRUCT_OBS
JUDDH-80	19.35				^	3	Weakly Deformed	CN	Contact	CL	Clay	5	Smooth, undulating	75		15	Contact between sandstone and shale-siltstone (no orishot)
JUDDH-80	28.6	54.8	18.1	9.25	^	4	Weakly Deformed	BD	Bedding	CL	Clay	4	Rough or irregular, undulating	70	40	20	Interbedded shale-siltstone
JUDDH-80	39	38.7	352.8	0.008	^	4	Weakly Deformed	VL	Veinlets	CA	Calcite	7	Rough or irregular, planar	60	50	30	Calcite veinlet
JUDDH-80	39.15	38.7	352.8	0.008	^	4	Weakly Deformed	VL	Veinlets	CA	Calcite	7	Rough or irregular, planar	60	50	30	Calcite veinlet
JUDDH-80	41.35	42.4	26.6	0.05	^	4	Strongly Deformed	VL	Veinlets	CA	Calcite	7	Rough or irregular, planar	50	85	40	Brecciated calcite veinlet
JUDDH-80	51.1	37.4	0.6	0.05	^	3	Weakly Deformed	SH	Shear zone	CL	Clay	4	Rough or irregular, undulating	60	60	30	Clay altered sandstone
JUDDH-80	52.6				^	3	Weakly Deformed	CN	Contact	CL	Clay	5	Smooth, undulating	45		45	Contact between sandstone and shale-siltstone (no orishot)
JUDDH-80	61			0.3	^	3	Weakly Deformed	SH	Shear zone	CL	Clay	4	Rough or irregular, undulating	60		30	Brecciated clay altered sandstone (no orishot)
JUDDH-80	76.75	70	347		^	4	Weakly Deformed	CN	Contact	CL	Clay	5	Smooth, undulating	30	35	60	Contact between shale-siltstone and interbedded shale-sandstone-siltstone
JUDDH-80	84.5	71	333		^	4	Weakly Deformed	CN	Contact	CA	Calcite	4	Rough or irregular, undulating	30	20	60	Contact between interbedded shale-sandstone-siltstone and shale
JUDDH-80	90.5	56.2	303.1		^	4	Undeformed	CN	Contact	CL	Clay	5	Smooth, undulating	45	345	45	Contact between interbedded sandstone-siltstone and shale-siltstone
JUDDH-80	121	66.2	12.8		^	3	Moderately Deformed	CN	Contact	CL	Clay	5	Smooth, undulating	30	60	60	Contact between fault zone and siltstone
JUDDH-80	125.3	81.4	327.2		^	3	Moderately Deformed	CN	Contact	CL	Clay	5	Smooth, undulating	20	10	70	Contact between fault zone and siltstone

Figure 11-5: Drillhole Structure & Veining Log

- **Drillhole Geotechnical log**

The data gathered and recorded in the standard geotechnical logs for all diamond drill holes includes; weathering, rock quality (RQD), discontinuity types and frequency per metre.

- **Drillhole Geomechanical Logs**

A comprehensive geomechanical logging system has also been developed to calculate RMR and other geomechanical factors for the deposits that have been drill tested by NBG.

6. Density determinations have been carried out routinely on drill core with 10 centimetre cylinders of whole core taken between 10 metres and 20 metres downhole or wherever there is a change in lithology. The method used is a displacement method with samples air dried, weighed, and then sprayed with polyurethane to seal them. They are then weighed again in air and then in water and the density determined using the standard

formula. This has enabled comprehensive density models to be developed for each deposit that NBG has drill tested.

7. Once all the logging has been completed the geologists select the sample intervals for assay. Sample intervals are determined by geology and mineralized features. Where there are no obvious contacts sample intervals were assigned on a metre by metre basis. These are marked out on the core and the intervals entered into the sample template. During this process sample numbers are assigned, standards and blanks inserted and the intervals selected for field and lab duplicate sampling.

8. The core is then delivered to the cutting room where the field technicians under the supervision of the geologist responsible for each drill hole cuts the core in half using one of the four Clipper core saws installed in 2010.

9. The sample numbers are written onto cloth bags along with the insertion of a numbered sample tag from the assay sample booklets.

10. The field technicians then place half (½) core from each sample interval into the numbered and labelled sample bags.

11. The geologists fill out standard instruction forms for SGS and the samples are delivered to the SGS lab sample reception area where they pass into the SGS sample preparation and processing system that will be discussed in *Chapter 12*.

12. Surveying – the collar positions of each drill hole have been surveyed using established survey control points by registered surveyors from Kuching, mine surveyors seconded from Besra's other projects or by tape and compass plus handheld GPS using known survey points if near to the holes being surveyed. A selection of these has also been check surveyed. All data was originally captured in BRSO and converted to UTM for x and y and using BRSO elevations for the z-axis. Now that the project is at an advanced stage, the Company has decided to use BRSO as the standard and is in the process of transferring all survey data to this system.

11.2.3.2. Channel Sampling, Rock and Soil Sampling

Rock float, rock outcrop, channel and soil samples are all handled in essentially the same way as drill core with the same or similar data recording protocols as for drill core. All samples are given an x, y and z coordinate. Channel samples and soil samples are recorded using the same survey formats as drill holes to ensure ease of data handling in geological and resource modelling software.

The Company's standard sampling procedures with insertion of standards, blanks and duplicates are applied in the same manner as for drill core.

NBG have made considerable efforts and set systems in place in terms of the procedures followed for drill core and other sample handling and processing since 2010 that are consistent with industry best practice.

11.3. Preparation, Analyses & Security – Prior to 2007

11.3.1. Sample Preparation

Early sample preparation was carried out by BYG. It is difficult to say what the precise procedures were at the time however; examination of the vast drill log database shows that samples of drill core were collected based on geology and mineralised intervals in the core. This core was split with a sample splitter up until the early 1990's after which diamond core saws were used. The authors have authenticated this from their own observations of remaining core.

RGC and Gencor are/were reputable international companies. RGC set up the current sample preparation facility and some of the current SGS staff were trained by them. The authors have no reason to believe that these companies did not use systematic and representative sampling methods.

Examination of Menzies Gold's records show that they had a rigorous and systematic sample collecting methodology in place for their largely RC drill programme. They prepared their samples at the sample preparation facility on site.

11.3.2. Assaying

Initially, BYG set up the mine site laboratory with an AAS facility only. This was later expanded to include classical fire assay and then fire assay with an AAS finish. The authors have reviewed many thousands of original assay records from the BYG drill holes. During the 1980's samples were generally reported in pennyweights.

Once the refractory nature of the gold at Bau was recognized, BYG routinely did a fire assay, followed by a repeat fire assay after roasting for either 0.5 hrs or 1 hr.

RGC and Gencor used commercial laboratories outside the BYG laboratory and had their own systems for QAQC that were to industry standards of the 1990's. The authors have reviewed the data captured from their work and viewed original assay records and have not seen any evidence to doubt the validity of the geochemical results.

BYGS/Menzies Gold initially assayed all their samples through the BYG laboratory but after becoming aware of contamination of their samples from grade control sampling at the Tai Parit mine, they used Assaycorp who were initially based in Australia but moved their laboratory to Kuching.

11.3.3. Quality Assurance & Quality Control

BYG operated the mine laboratory essentially for grade control and exploration assaying purposes. Issues arose with some of the early assay data. Other companies that used the laboratory, such as Renison and Menzies carried out their own QAQC of the laboratory and produced validated in their respective databases.

NBG have reviewed much if the original data and discuss this in *Chapter* ***Error! Reference source not found.***.

Gencor and RGC used their own protocols of duplicates, standards, blanks and umpires.

BYGS / Menzies Gold had a rigorous QAQC protocol. This included:

- Duplicate sampling to check sample preparation and precision;
- Repeat sampling by the primary laboratory to check lab precision;
- Comparison of the 4 metre composite sampling against the 1 metre sample average over the 4 metre interval;
- Umpire sampling at a laboratory independent of the main assay laboratory;
- Insertion of certified standards;
- Insertion of silica blanks to check on contamination and instrument drift.

Menzies had identified an issue with contamination of their RC samples in the BYG lab, especially at the lower range of assay values. Thereafter, Menzies used the BYG lab for their 4 metre composite samples only and sent any samples assaying more than 0.5 g/t Au to Assaycorp in Australia and later in Kuching. They used McPhar, Analabs and Inchape laboratories for umpire sampling and QAQC.

Issues were also raised with potential smearing of values in the RC drilling at Pejiru when comparative results between twinned diamond and RC holes were examined, especially below the water table. Mustard 1996 evaluated the issue and concluded that the amount of smearing was not significant.

The authors take the view that this issue remains unresolved and therefore is one reason why the resources at Pejiru have been categorized as Inferred.

11.3.4. Security

The sampling procedures and handling protocols were managed by the various companies operating at Bau. From the investigations made by the authors there is no reason to suspect that samples were systematically or deliberately tampered with.

11.4. Preparation, Analyses & Security – North Borneo Gold Since 2007–2009

11.4.1. General

North Borneo Gold prepared their samples at the former BYG sample preparation facility. NBG refurbished this facility and all samples from 2007 to 2010 were prepared there prior to shipment overseas for analysis.

11.4.2. Sample Preparation

The core to be sampled was selected using the protocols described in *Section 11.2*. The core was sawn by diamond saw or split (where too soft to cut) into approximately equal halves with one half sent for analysis and the remaining half labelled and retained in core boxes for future reference. In order to prevent bias, the geologist logging the core supervised core cutting and ensured that the core is cut along the apex of any veins or significant mineralized structure.

Each sample of core was assigned a unique sample number from the pre-printed sample tickets. Sample preparation consisted of essentially six (6) steps:

- Place core/rock sample into numbered metal trays
- Dry in a gas fired oven at 100 – 120 °C
- Primary Crush to approx 8 mm top size
- Secondary Crushing to approx 80% passing minus 2 mm
- Splitting to the required weight for pulverizing
- Pulverising using either an Essa LM1/B2000 mill combination or and Essa LM3 Mill to produce a pulp of approximately 90% passing 75 microns.

The required pulp specification was approximately 90% passing 75 microns, prepared from a sufficiently large and finely crushed sub samples as to be representative of the whole sample taking into consideration likely gold sizing and grades. The following flow sheet was the standard used but may be varied if for example there was likely to be visible gold. Not usually an issue at Bau.

<u>Standard Sample Preparation Flowsheet</u>

for

<u>Rocks & Drill Core (not containing visible gold)</u>

Total Sample

|

<u>Dryer</u>

|

<u>Jaques Crusher</u>

|

<u>Altec Crusher 1st Pass</u>

|

<u>Altec Crusher 2nd Pass</u>

|

<u>Sample Splitter - repeat splitting till</u>

| |

+750g approx. Remainder of crushings

| |

LM1/B2000 4 minutes. Return to original sample bag

| | |

150g approx. Remainder of pulp. Store till results known

to laboratory store for future use

One sample was sent for assay and the remainder of the pulp retained as a duplicate. The crushings not sent for assay were also retained on site for future reference and check assaying etc.

The third and final sample ticket remained in the sample ticket book with the drillhole number and metrages filled in.

Sealed sample bags were placed in durable plastic bags of around thirty (30) samples each for shipment to the laboratory. The geologist sending the sample shipment kept a record of all samples shipped. The samples were transported to Kuching by road and dispatched by DHL to Mineral Assay and Services Co. Limited's (MAS) laboratory in Bangkok, Thailand. Assay results were then electronically distributed to authorized personnel and a hard copy of Assay Certificates sent to NBG's office in Kuching.

11.4.3. Assaying

Samples were assayed at MAS Laboratory in Bangkok, Thailand. The Thailand Department of Industrial Works and Ministry of Industry certify the MAS laboratory. Upon receipt, samples were sorted, inspected, logged and dried (if necessary and/or requested).

Gold was assayed by fire assay using a 50 gram charge with an Atomic Absorption Spectophotometric (AAS) finish, (detection limit 0.02 g/t Au).

The Laboratory inserted their own certified control standard at random in each batch of approximately thirty (30) samples. In addition, the laboratory re-assayed every 10th sample.

A suite of seven (7) elements were generally determined by ICP analyses from selected mineralized intervals on a routine basis. These elements were: Ag, As, Cu, Mo, Pb, Sb and Zn.

11.5. Quality Assurance & Quality Control North Borneo Gold 2007-2009

11.5.1. Geochemical Standards

During the drill and sampling program since 2007 NBG introduced a "standard" from a homogenized mineralised sample for which they had a reasonable degree of confidence in its gold value, however, was not a certified standard. The assay results from this NBG "standard" are shown in *Figure 11-6: North Borneo Gold "Standard"* below.

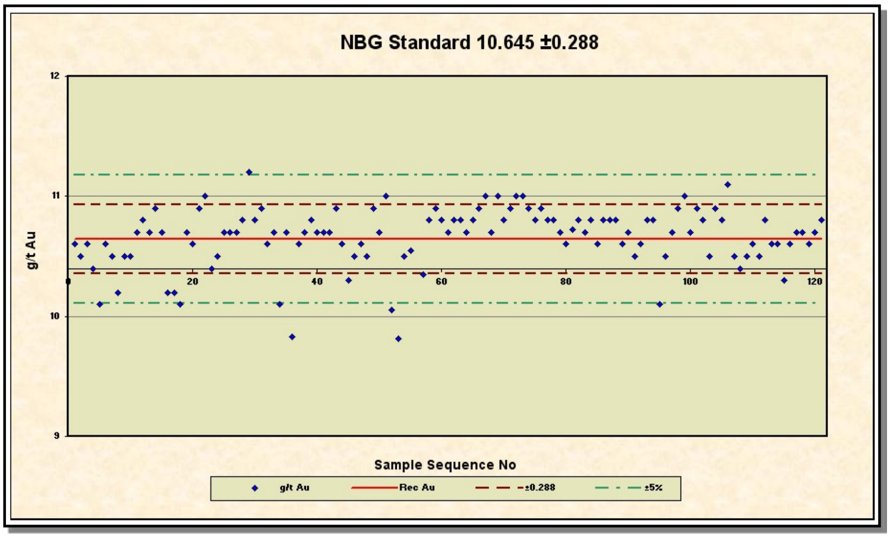


Figure 11-6: North Borneo Gold "Standard"

The 120 plus NBG standards analysed gave a mean of 10.645 g/t Au with a standard deviation of 0.288 g/t Au. Apart from four (4) samples all results lie within the 95th percentile.

Reliance on assay integrity was largely placed on the protocols adopted by MAS.

Figure 11-7: Assay Values for MAS Standard ST-04/6369 and *Figure 11-8: Assay Values for MAS Standard ST-04/9210* show the gold scatter plots of the standards used by MAS during 2007 to 2009.

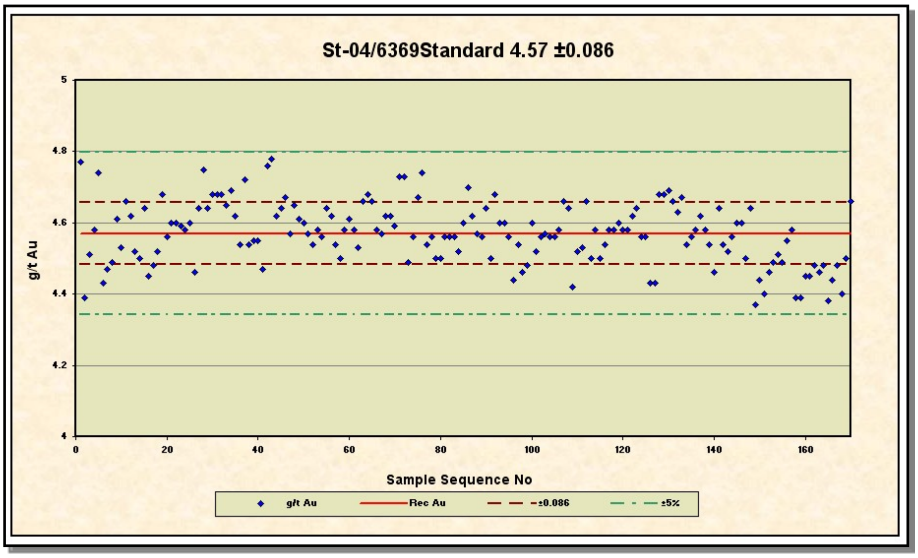


Figure 11-7: Assay Values for MAS Standard ST-04/6369

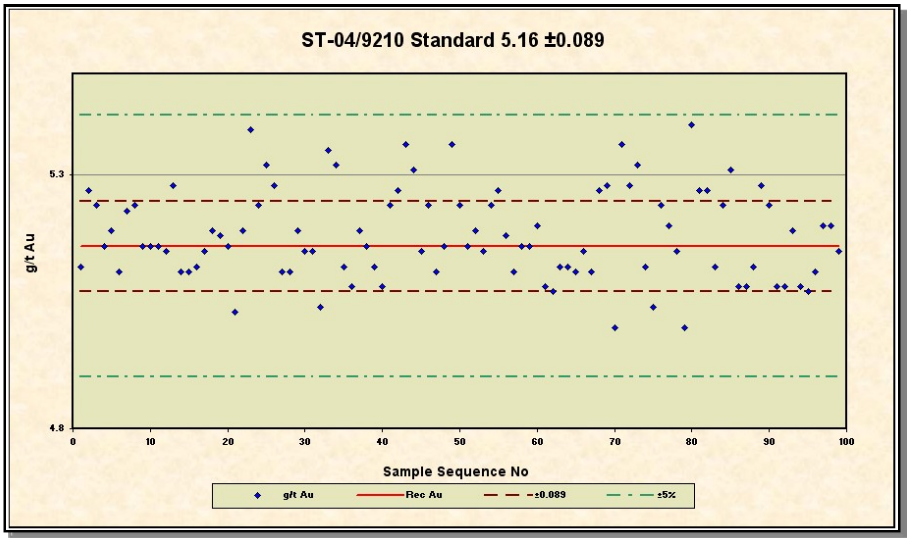


Figure 11-8: Assay Values for MAS Standard ST-04/9210

11.5.2. Duplicates

As part of NBG's quality control procedure, duplicates of the pulps were retrospectively analysed at intervals of every ten (10) samples from the NBG database. Each duplicate sample is assigned a unique number that can be related to the primary sample number and tracked.

The succeeding figure (*Figure 11-9: Logarithmic Correlation of Original and Laboratory Repeat Samples*), illustrates the logarithmic plots of the NBG duplicates verses the laboratory duplicates. Logarithmic plotting was used instead of linear correlation because of tight spacing among sample points making linear graph ineffective for interpretation and presentation. The red line shows the ideal trend line for a perfect original-duplicate sample result, derived from the equation y=mx+b where m is the slope which is equal to one (1) and b is the y-intercept equal to zero.

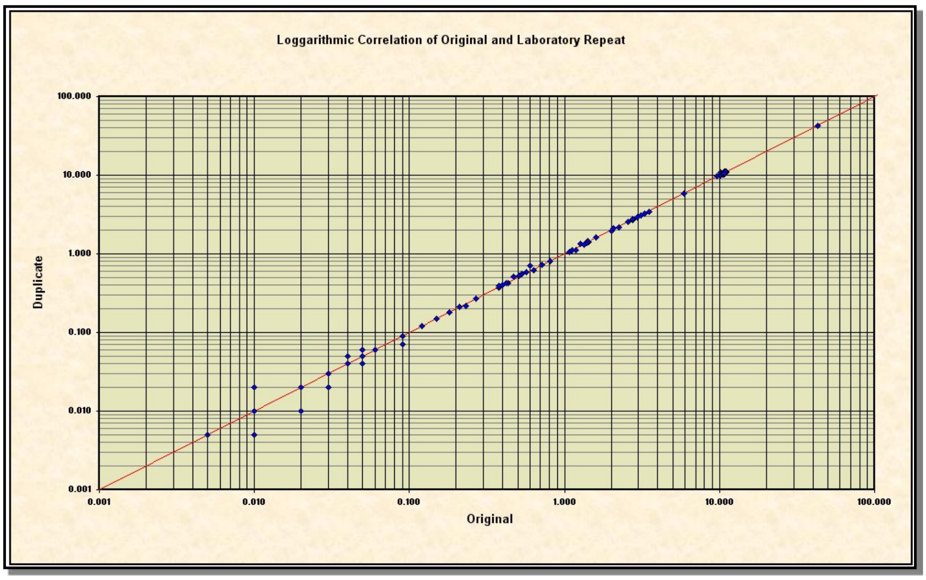


Figure 11-9: Logarithmic Correlation of Original and Laboratory Repeat Samples

Sample points for the duplicates show a good correlation between the original and replicate samples. The distribution is nearly patterned to the ideal linear trend line, with few sporadically scattered points but still close to the line. Grades in the lower limits, however, show more sample dispersion signifying lesser replication of grades of the original samples. The higher variation between the original and duplicate grades of samples within this zone can be considered normal, since this is already near and within the detection limit zone.

11.5.3. Blanks

NBG did not use blank samples and relied on the laboratory QAQC procedures.

11.5.4. Umpire Sampling

Umpire samples were not routinely collected through the programme however in the case of Jugan all holes drilled by NBG and assayed at MAS were reassayed by ALS in Orange, NSW, Australia which is an accredited laboratory and can be used as an umpire population to give a reasonable appreciation of any major issues with the precision and accuracy of MAS.

Figure 11-10: Logarithmic Plot of Correlation between MAS Original Samples & ALS Umpires shows reasonable correlation between MAS and ALS for NBG drillholes JUDDH-01 to JUDDH-05.

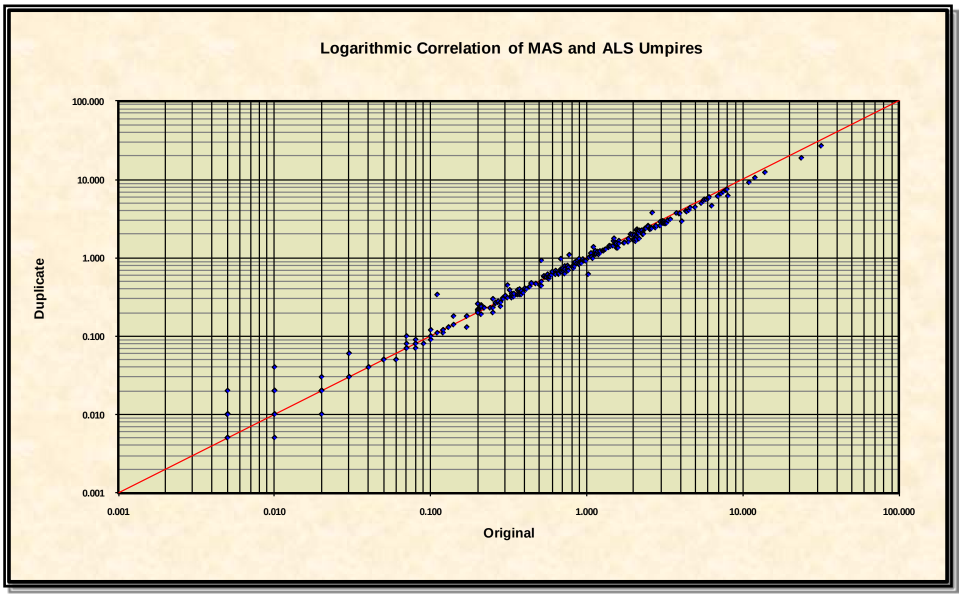


Figure 11-10: Logarithmic Plot of Correlation between MAS Original Samples & ALS Umpires

11.5.5. Security

During the diamond drilling program since 2007, all drillcore was moved from drilling sites to secure sample preparation facilities at the field office in Bau as soon as practical under the supervision of the site geological staff.

The core logging and sample preparation areas were manned during working hours and had security patrols at night. The sample preparation and logging area were under the supervision of the senior site geologist, junior geologist and senior sample preparation staff. The Company employed on site security personnel and only authorised persons could enter the compound.

All samples were packaged in sealed plastic bags. These sealed bags were then transported to Kuching, received by NBG staff in Kuching accompanied with sample dispatch sheets and bills of lading, copies of which were retained with the sample ledger. They were then air freighted using DHL to the MAS laboratory in Bangkok, Thailand or other laboratories as appropriate. The laboratory was required to notify NBG if the samples did not arrive with the NBG seals intact and to retain all seals so that a probable Chain of Custody would be available.

Opinion on the Adequacy of Sampling, Sample Preparation, Security and Analytical Procedures

The authors consider that the sampling, sample preparation, security and analytical procedures and results detailed in this report by and undertaken by NBG have been carried out in a systematic and secure manner. The internal QAQC carried out by the laboratories concerned show conformance with accepted industry standards. As such, the authors accept that the data is valid for the purposes being used in this report.

11.6. Preparation, Analyses & Security – North Borneo Gold 2010 to 2013

11.6.1. General

In 2010 NBG entered into a contract with SGS Laboratories to supply analytical services for the Bau project. This involved SGS taking over and upgrading the sample preparation facilities on site and setting up a fire assay facility. Minor element analyses are sent off shore and analysed either at SGS Perth or SGS Port Klang in Kuala Lumpur.

NBG decided to commit to this arrangement to ensure that all assaying and preparation are carried out to industry best practice by an accredited laboratory.

11.6.2. Sample Preparation

Figure 11-11: Sample Preparation Flow Chart for Bau shows the sample preparation procedure followed by SGS in the Bau sample preparation facility.

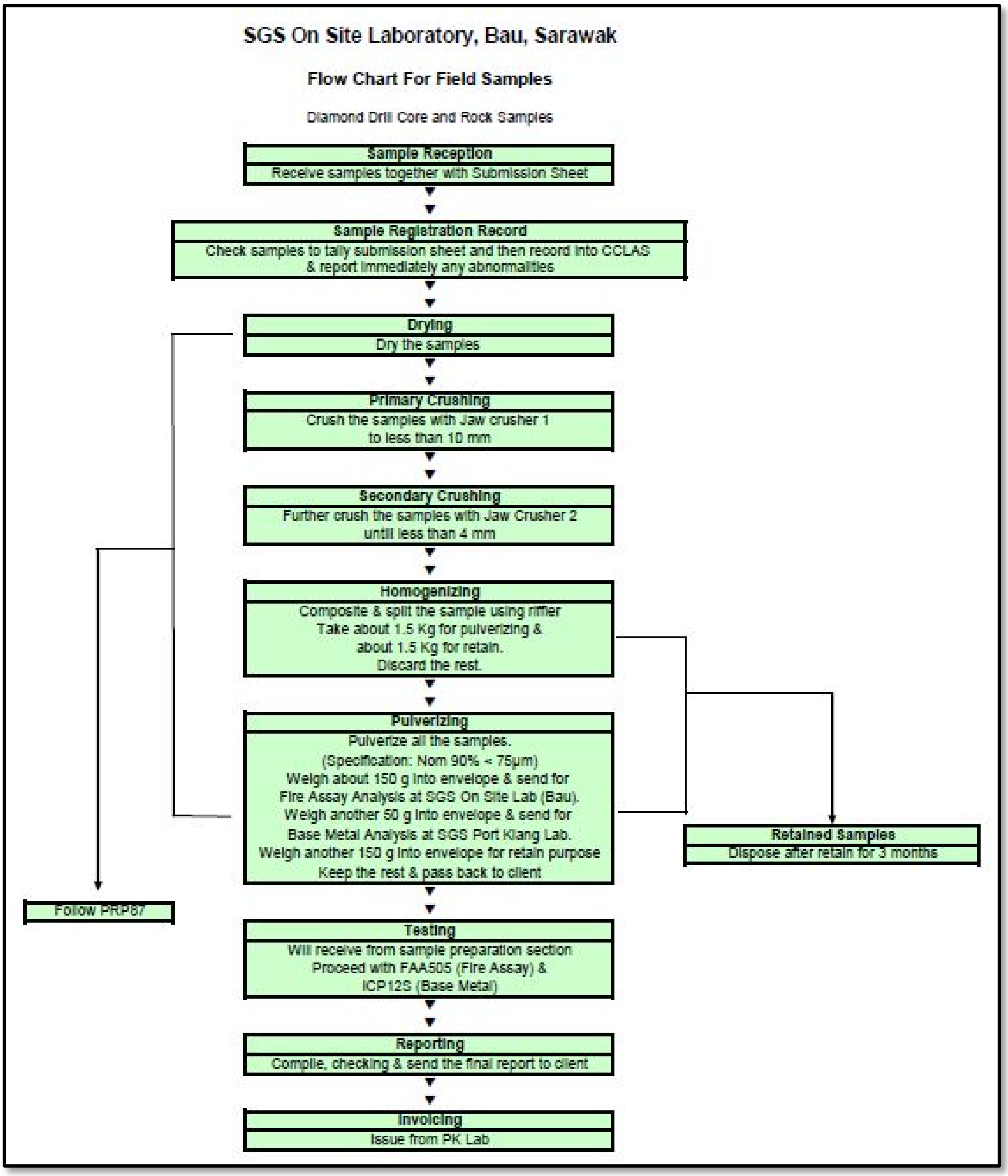


Figure 11-11: Sample Preparation Flow Chart for Bau

SGS have set up the laboratory at Bau to comply with ISO17025 certification. They routinely carry out screen sieve tests during the sample preparation process on the coarse -4 mm secondary crush and on the pulps following pulverizing in the LM2 ring mills. They sieve 5 % each of the -4 mm and the -75 micron fractions with the target of achieving 90 % plus passing in each case.

11.6.3. Assaying

Gold assaying is by Fire Assay using a 50 g charge with an AAS finish, using SGS method FAA505. Detection limit is 0.01 ppm.

A suite of twenty-three (23) other elements are analysed by SGS method ICP12S. This suite did not initially include sulphur which was added late in the Jugan programme to help provide additional geo-metallurgical information.

Additionally, tungsten and thallium were added to the suite for soil sample analyses. Tungsten and Thallium are analysed using ICP-MS to get the low detection limit required for soil sampling while total sulphur values above 2.5 % are determined by method CSA06V which utilizes high temperature combustion in a furnace with Infrared measurement. Arsenic values above 0.5 % are determined by AAS.

These are listed below in *Table 11-1: Minor Element Analyses Method and Detection Limits*.

Element	Method	Lower Detection	Upper Detection	Units
Ag	ICP12S	0.2	50	PPM
As	ICP12S	2	5000	PPM
Sb	ICP12S	2	2500	PPM
Cu	ICP12S	2	5000	PPM
Pb	ICP12S	3	5000	PPM
Zn	ICP12S	2	5000	PPM
Al	ICP12S	50	50000	PPM
Ba	ICP12S	2	5000	PPM
Bi	ICP12S	5	5000	PPM
Ca	ICP12S	20	200000	PPM
Cd	ICP12S	1	2500	PPM
Co	ICP12S	1	5000	PPM
Cr	ICP12S	3	10000	PPM
Fe	ICP12S	50	1000000	PPM
Hg	ICP12S	1	2500	PPM
K	ICP12S	50	200000	PPM
Mg	ICP12S	10	50000	PPM
Mn	ICP12S	5	25000	PPM
Mo	ICP12S	1	5000	PPM
Na	ICP12S	20	50000	PPM
Ni	ICP12S	1	5000	PPM
Ti	ICP12S	10	5000	PPM

Element	Method	Lower Detection	Upper Detection	Units
V	ICP12S	1	5000	PPM
S	ICP12S	10	25000	PPM
W	IMS12S	0.1	1000	PPM
Tl	IMS12S	0.1	2500	PPM
As	AAS12S	50	10000	PPM
S	CSA06V	0.005	40	%

Table 11-1: Minor Element Analyses Method and Detection Limits

SGS's flow chart for the fire assay facility in Bau is shown below as *Figure 11-12: Fire Assay Process Flow Chart for the SGS Facility at Bau.*

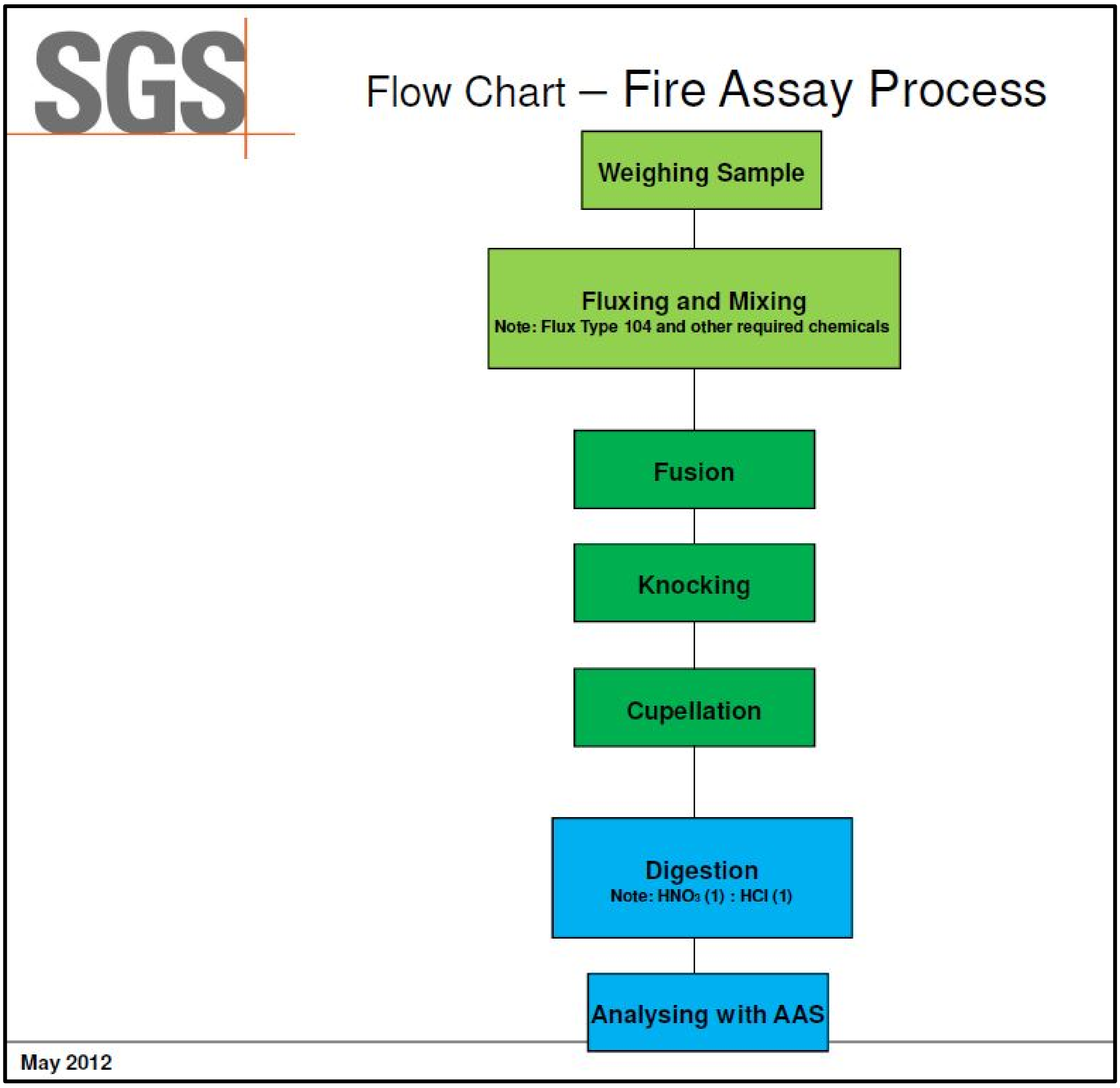


Figure 11-12: Fire Assay Process Flow Chart for the SGS Facility at Bau.

All retained pulps and coarse rejects are retained on site and stored by NBG.

11.7. Quality Control and Quality Assurance (QAQC) NBG 2010 to 2013

11.7.1. General

NBG have introduced industry standard protocols for QAQC procedures involving the insertion of certified standards, blank samples, umpire sampling, field duplicates from the coarse crushed material and preparation duplicates from the pulverized splits.

In addition SGS supplied NBG an analysis on a monthly basis of the laboratory's performance with respect to their own internal QAQC procedures.

11.7.2. Geochemical Standards

Certified geochemical standards are inserted into the sample stream at a ratio of 1:30. They are sourced from Rocklabs New Zealand one of the world's largest suppliers of certified reference materials for the gold mining industry. A variety of standards are used of different grades.

The standards used at Bau since 2010 are tabulated below in*Table 11-2: Certified Standards used throughout the Bau Programme 2010 to 2013*.

Rocklabs Standard	Expected Au Value (ppm)	95 Percentile (+/-)	Standard Deviation
SE58	0.607	0.006	0.190
SG56	1.027	0.011	0.033
SK52	4.107	0.029	0.088
SN60	8.595	0.073	0.223
SG40	0.976	0.009	0.022
SG50	8.685	0.062	0.180

Table 11-2: Certified Standards used throughout the Bau Programme 2010 to 2013

Figure 11-13: SGS Standard SE 58 to *Figure 11-18: SGS Standard SN 50* shows a graphical representation of the lab standards used by SGS.

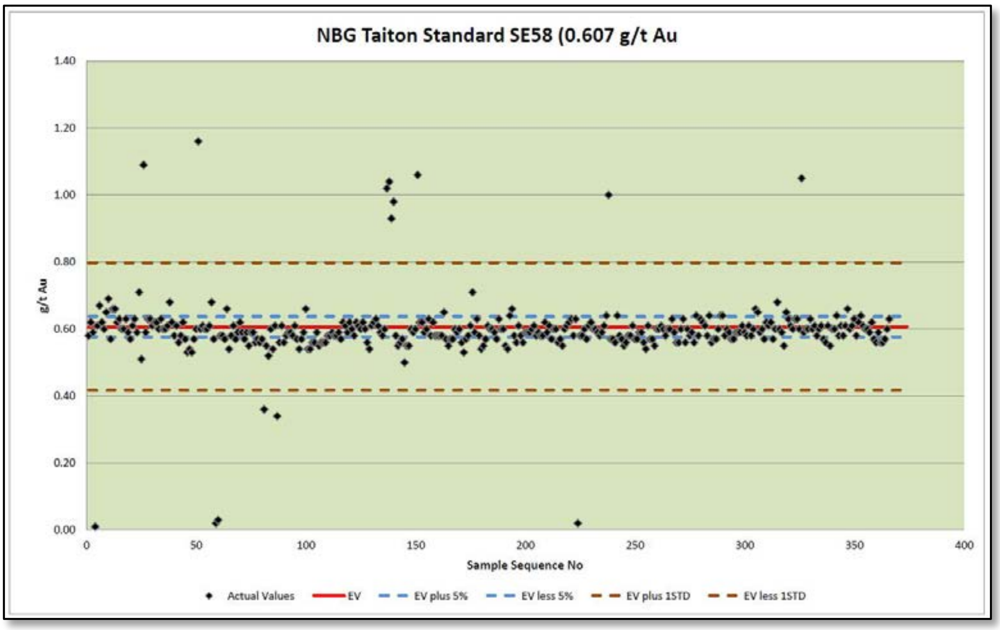


Figure 11-13: SGS Standard SE 58

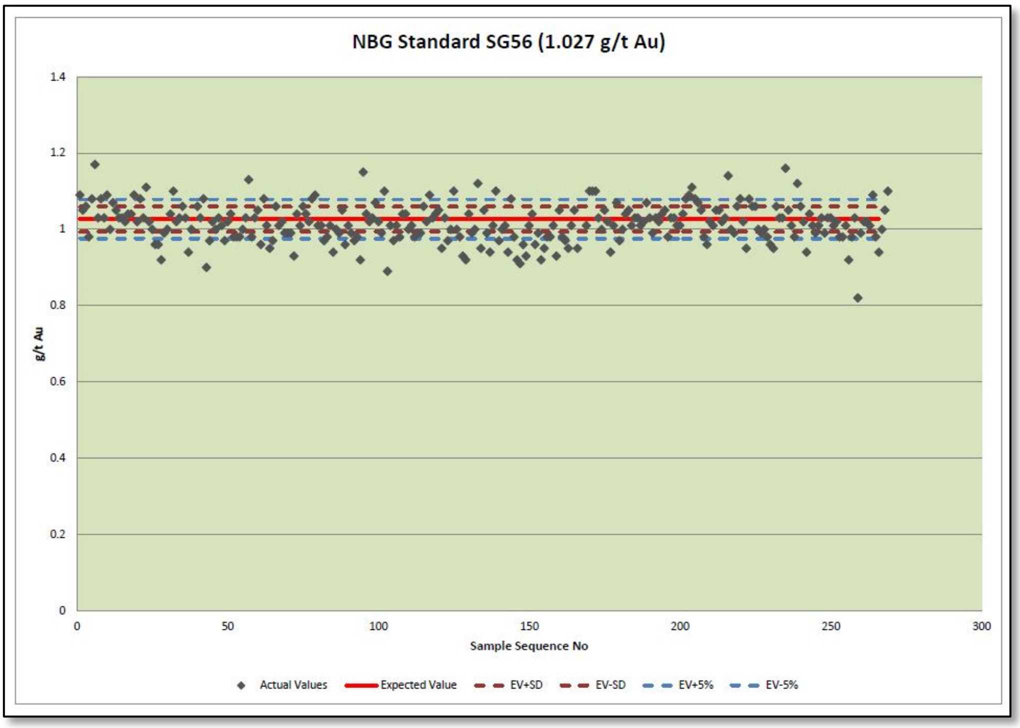


Figure 11-14: SGS Standard SG 56

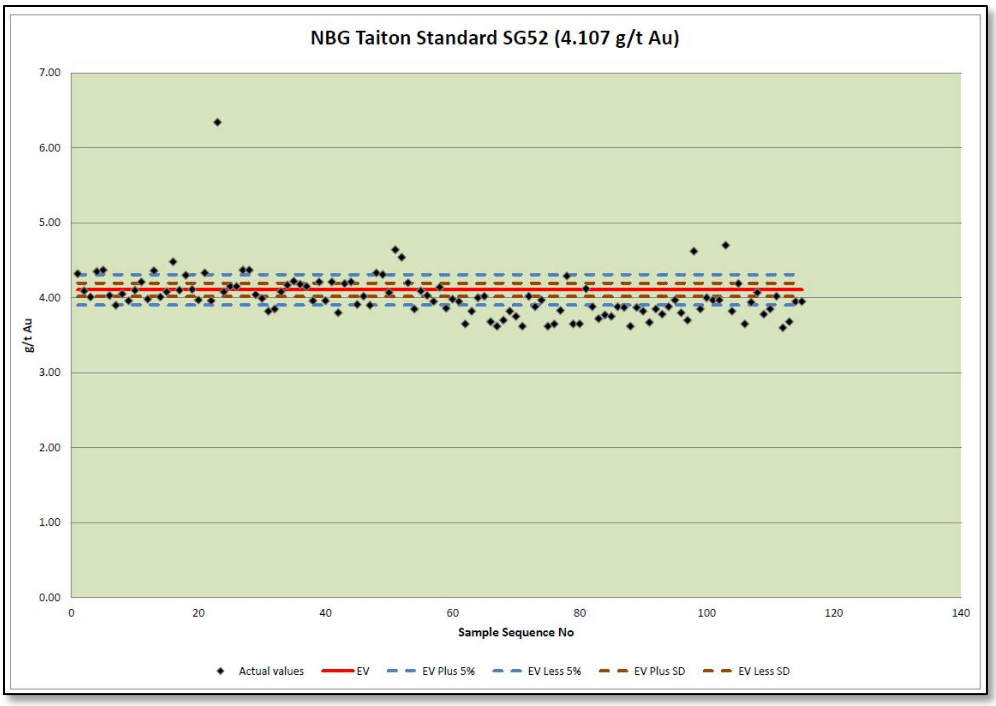


Figure 11-15: SGS Standard SG 52

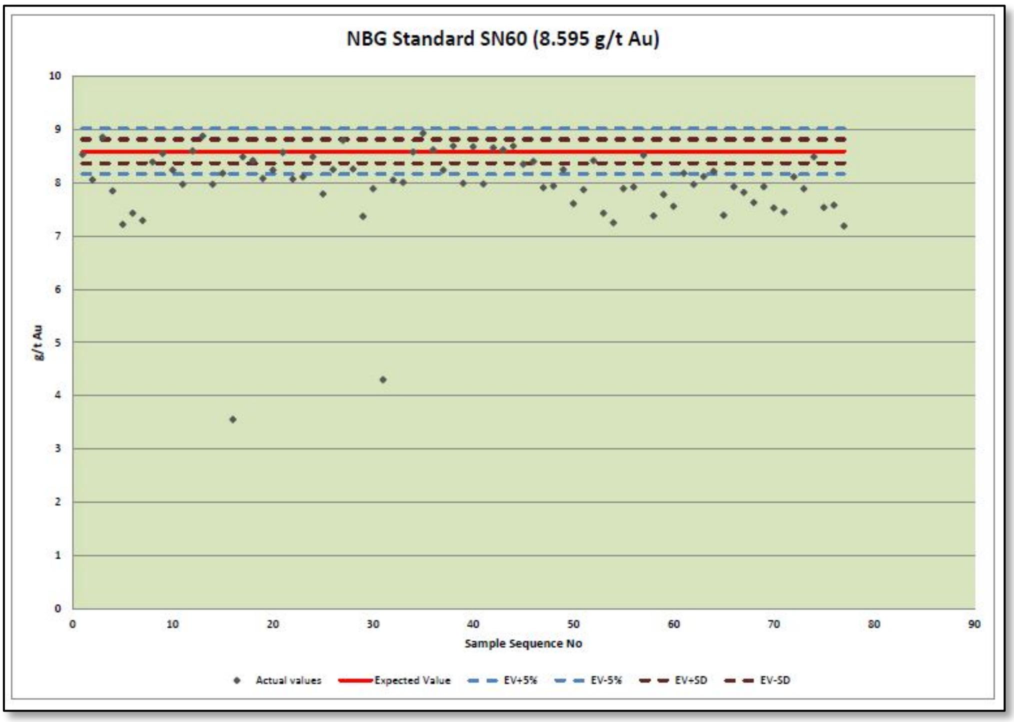


Figure 11-16: SGS Standard SN 60

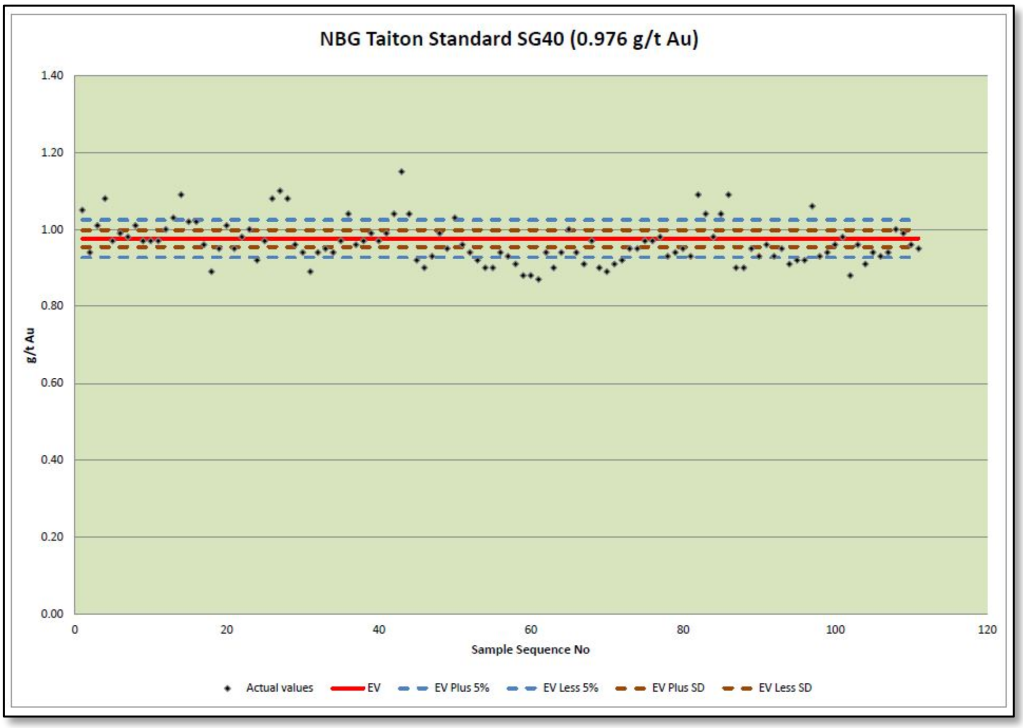


Figure 11-17: SGS Standard SG 40

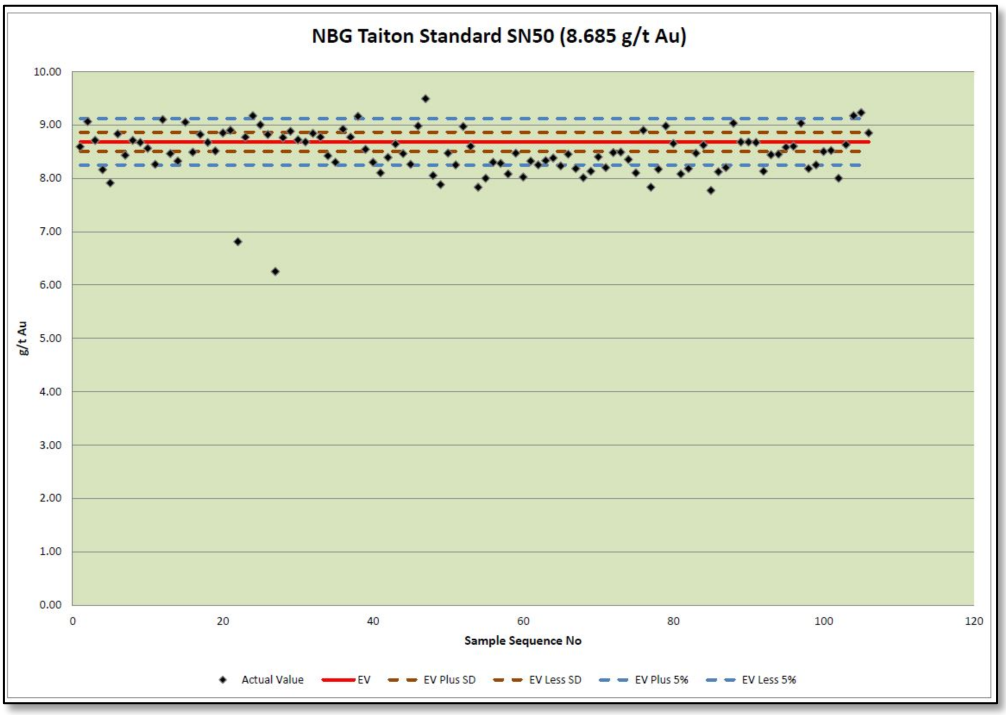


Figure 11-18: SGS Standard SN 50

As a general comment the majority of the standards have performed reasonably well with a slight tendency to report on the lower side of the expected value based on the 95 percentile values. Most fall within plus or minus 5% of the expected value. There are several instances of values well outside the +/-5% range and the +/-10% range of the expected value for the standard. These can be attributed to several causative factors, including mostly human error in transposing sample numbers, entering the standard identification into the sample sheet and or transposition to the database. The number of values in this category represents approximately 1.7% of the total number of standard analyses and is not statistically significant.

11.7.3. Field Duplicates

As part of the quality control system a duplicate from every 10th sample is taken from the split after the second crushing to the nominal P_{80} -4mm whole sample. This provides information on reproducibility, effectiveness of homogenization at the crushing stage and information on distribution of gold content in the sample. Each field duplicate is assigned a unique sample number in the sample stream for each batch.

The following graphs in *Figure 11-19: Log-log Plot, Field Duplicates, Jugan* to *Figure 11-22: Log-log Plot, Field Duplicates, Juala* show the field duplicate samples plotting as log-log plots for each of the four areas drilled since 2010, namely Jugan, Bekajang, Taiton and Juala.

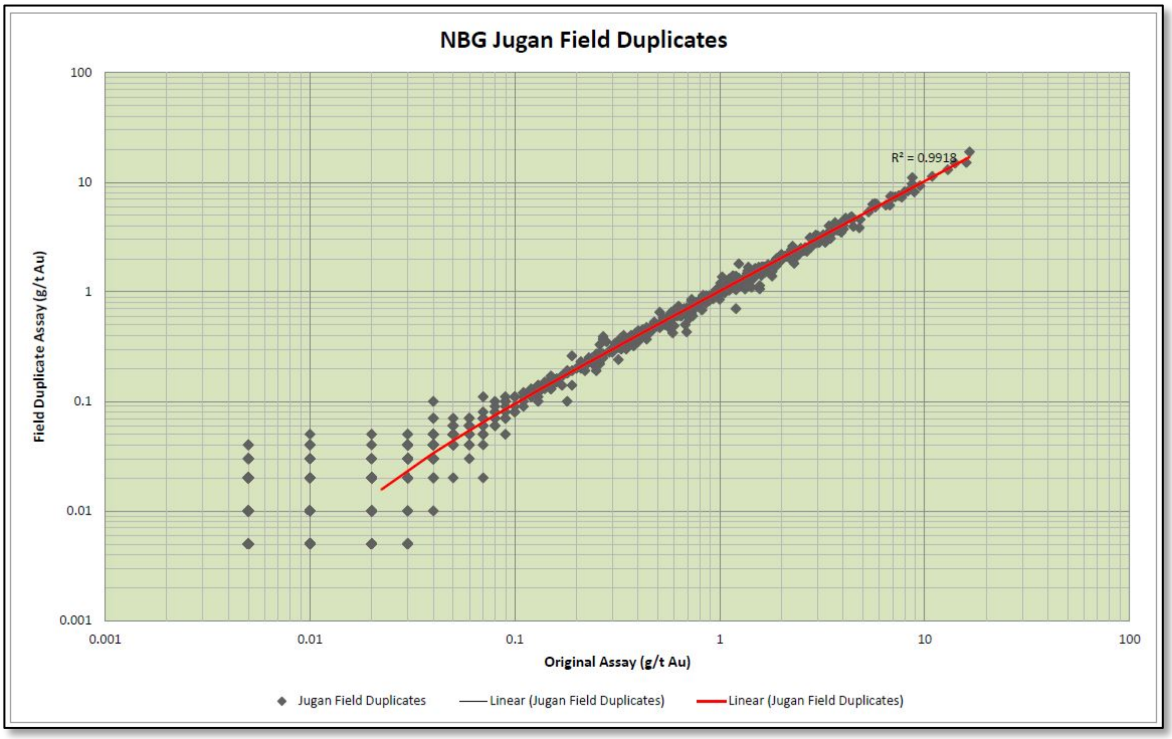


Figure 11-19: Log-log Plot, Field Duplicates, Jugan

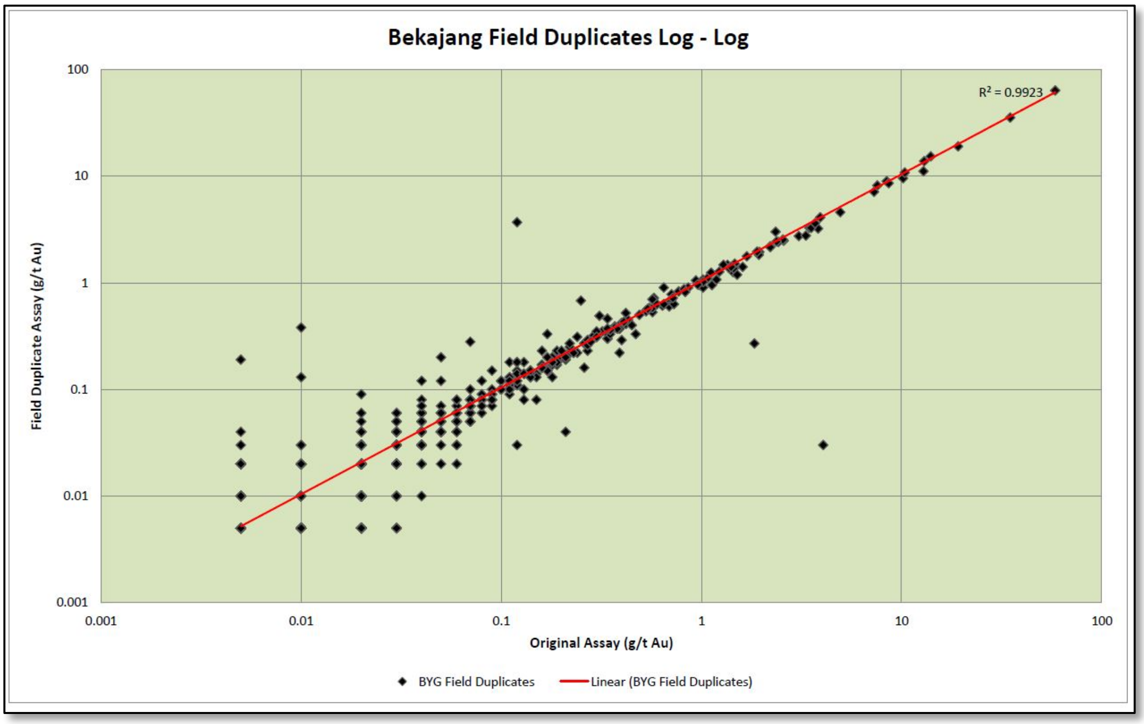


Figure 11-20: Log-log Plot, Field Duplicates, Bekajang

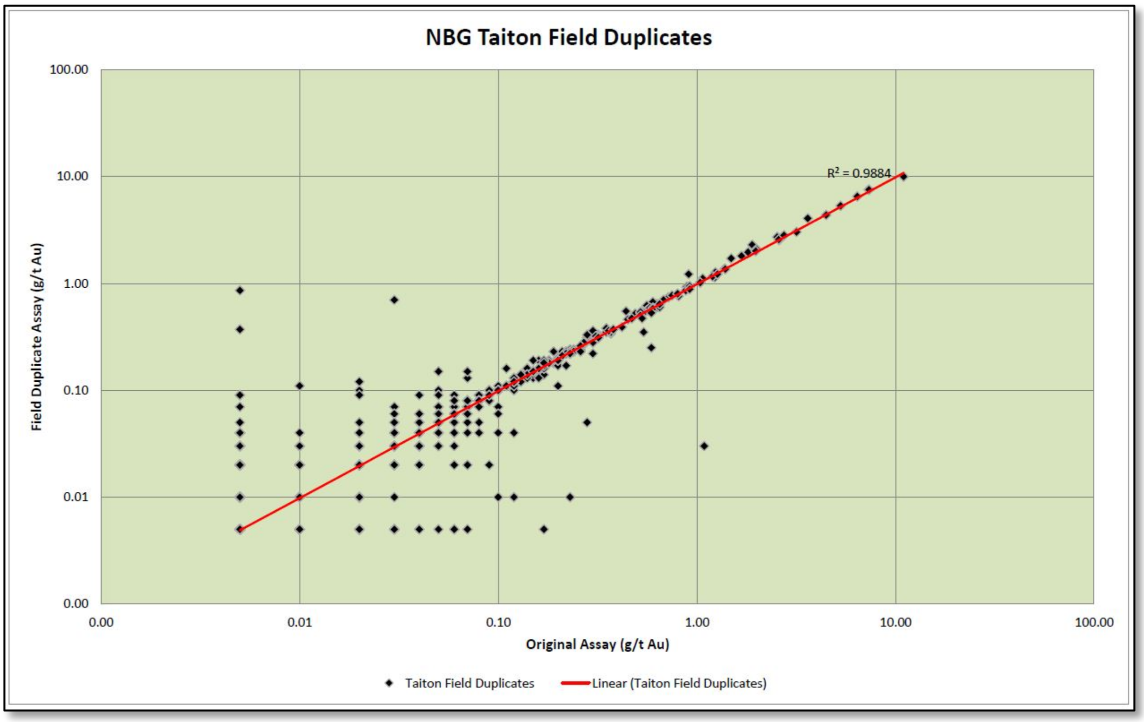


Figure 11-21: Log-log Plot, Field Duplicates, Taiton

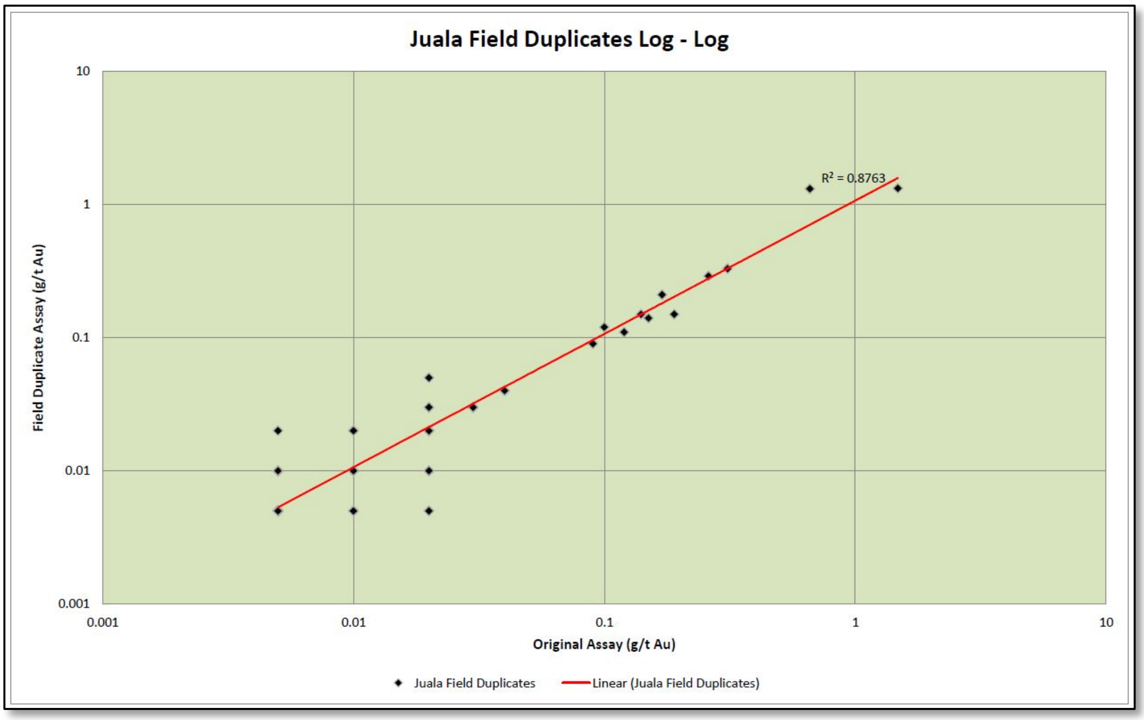


Figure 11-22: Log-log Plot, Field Duplicates, Juala

Comparison of the field duplicate plots shows that correlation coefficients for Taiton Jugan and Bekajang are close to one (1), ranging from 0.9884 to 0.9923. In the case of Juala the R2 value drops to 0.8763. This is possibly a reflection of a smaller data set and the number of samples that fall below detection of 0.01 ppm Au that are set to 0.005 ppm.

11.7.4. Preparation Duplicates

In addition to field duplicates a further duplicate from every 10^{th} sample is taken from the split after pulverizing in the ring mill to the nominal P_{80} -75 microns. This provides information on reproducibility, effectiveness of homogenization at the fine grinding stage and information on sampling for the fire assay by laboratory personnel and other factors for instance creating of false nugget effects by overgrinding etc. Each field duplicate is assigned a unique sample number in the sample stream for each batch.

The following graphs in *Figure 11-23: Log-log Plot, Preparation Duplicates, Jugan* to *Figure 11-26: Log-log Plot, Preparation Duplicates, Juala* show the preparation duplicate samples plotted as log-log plots for each of the four areas drilled since 2010, namely Jugan, Bekajang, Taiton and Juala.

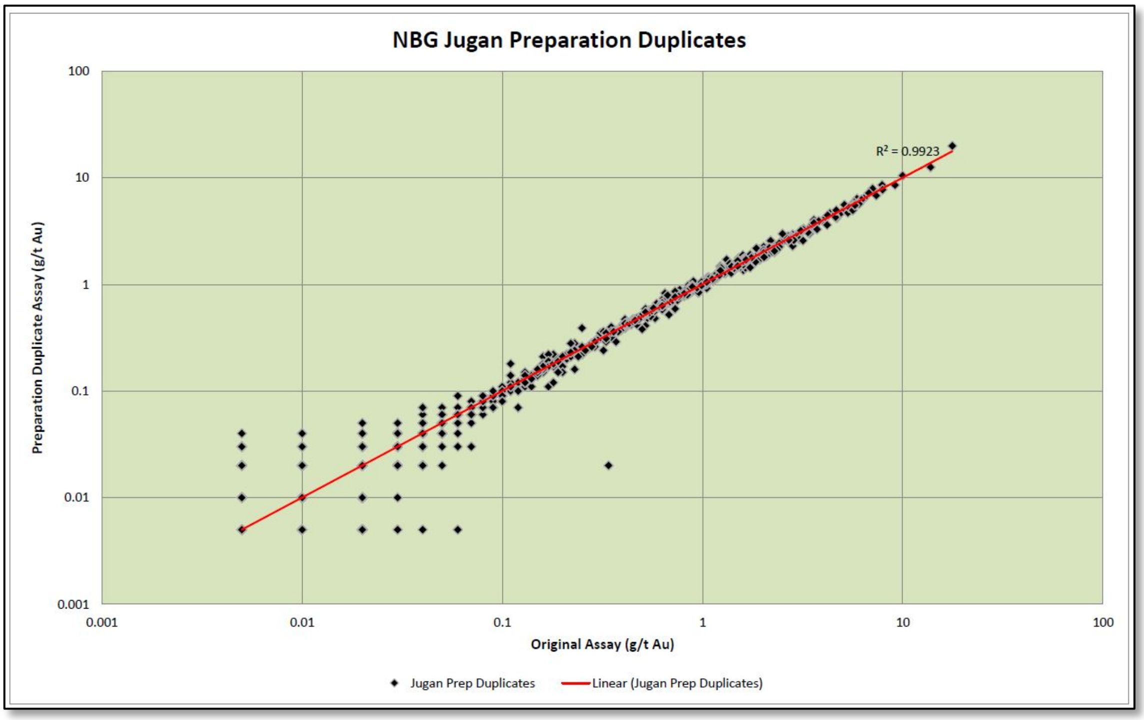


Figure 11-23: Log-log Plot, Preparation Duplicates, Jugan

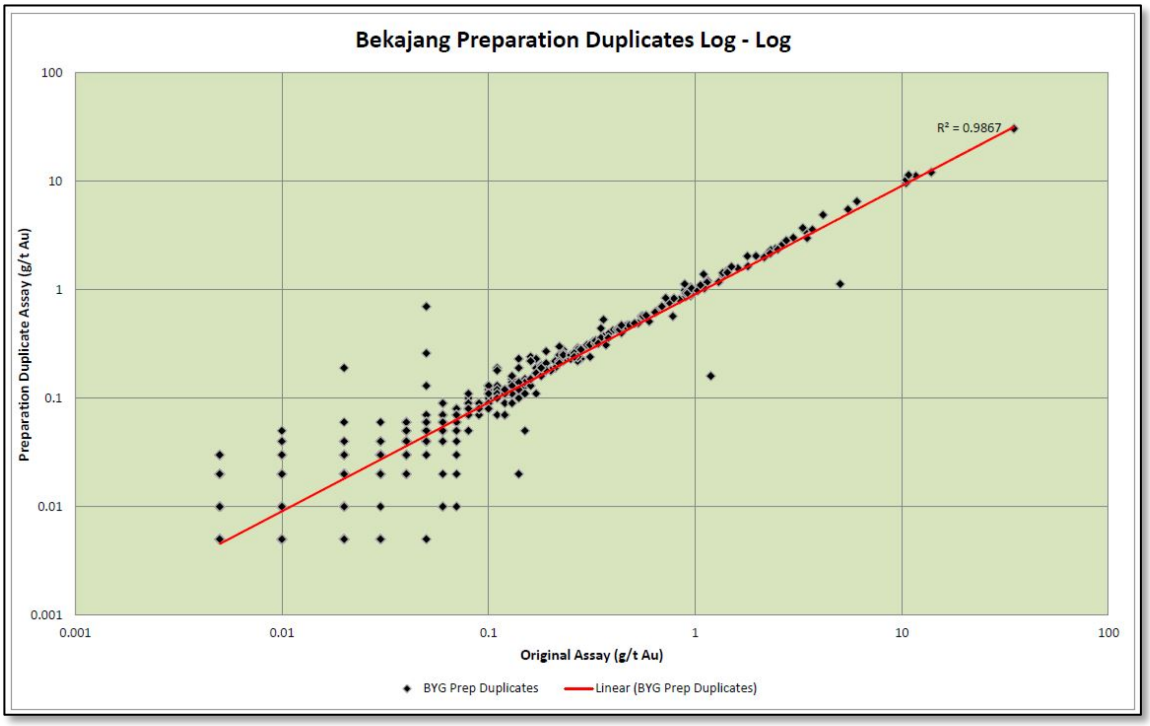


Figure 11-24: Log-log Plot, Preparation Duplicates, Bekajang

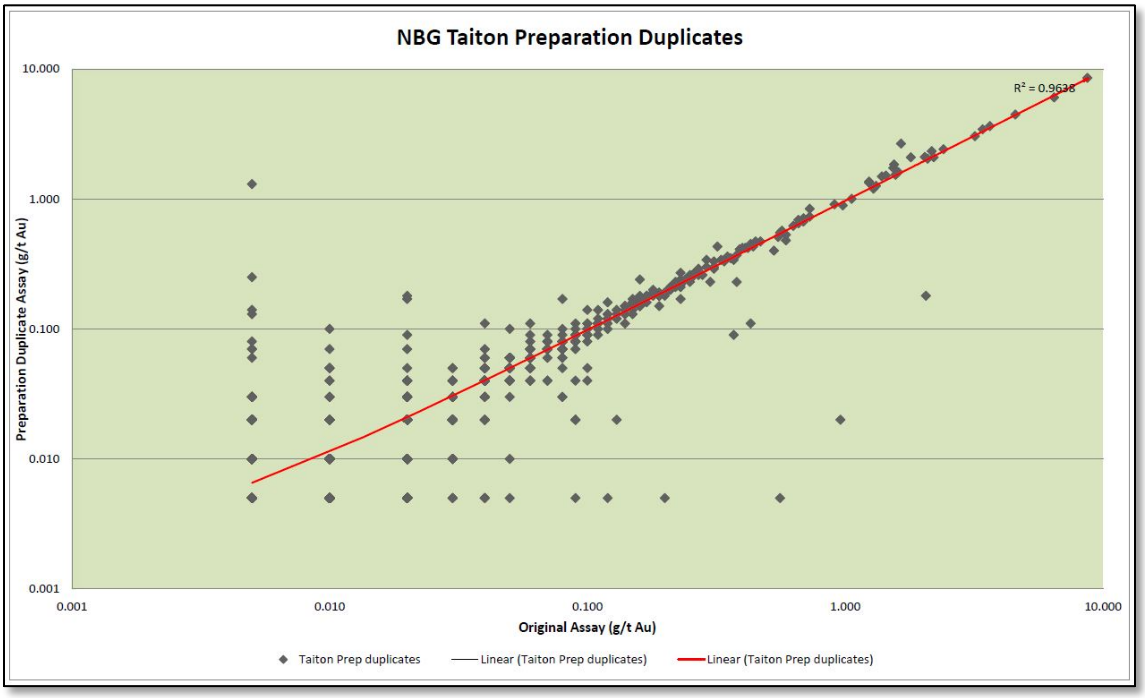


Figure 11-25: Log-log Plot, Preparation Duplicates, Taiton

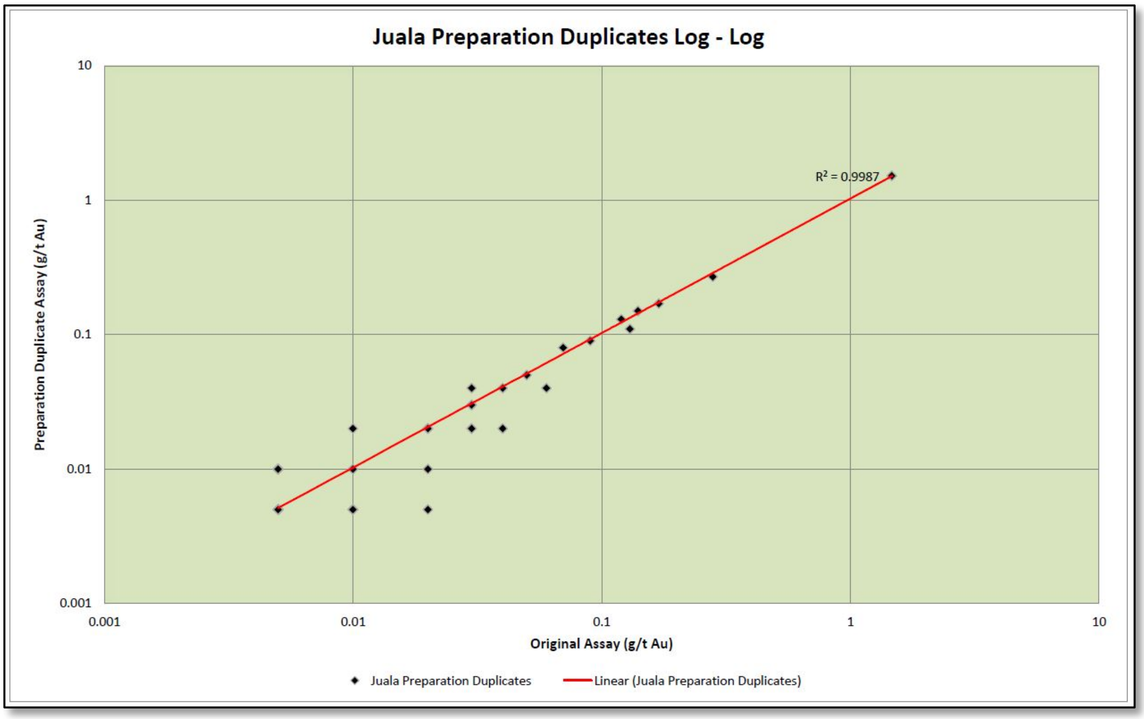


Figure 11-26: Log-log Plot, Preparation Duplicates, Juala

Comparison of the preparation duplicate plots shows that correlation coefficients for Taiton Jugan, Bekajang and Juala are all close to one, ranging from 0.9638 for Taiton to 0.9987 at Juala.

In the case of Taiton the R2 value is 0.9638. There are a number of samples in the lower grade ranges where there is clearly some disparity in the original and duplicate grades. Some of these are due to transposition errors. Overall the discrepancies lie mainly in the lower grade ranges where a small difference has a large effect especially where values below detection limit of 0.001 ppm are set at 0.005 ppm.

11.7.5. Laboratory Duplicates

As part of the QAQC procedure NBG also kept account of the duplicates assays conducted by SGS on NBG's samples. These are plotted in *Figure 11-27: Log-log Plot, SGS Duplicates* below.

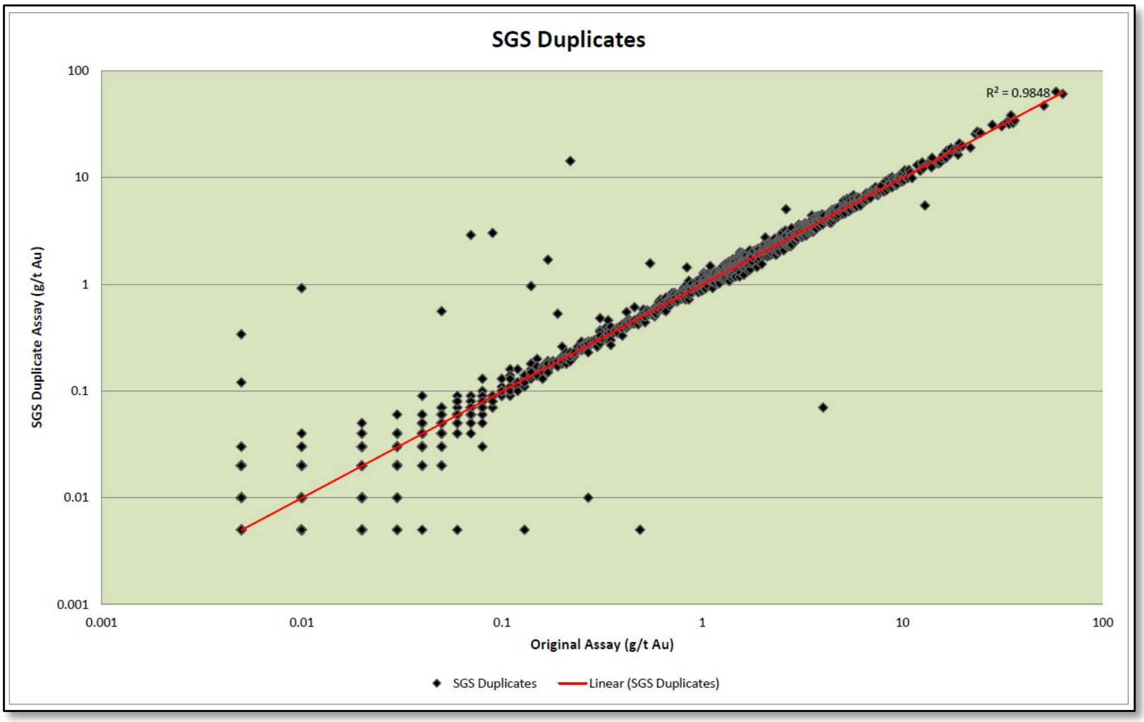


Figure 11-27: Log-log Plot, SGS Duplicates

The log-log plot of SGS duplicates compiled by NBG shows a correlation coefficient of 0.98. There are several samples (8 out of 2048) that show wide discrepancy between the original and the SGS duplicates. These amount to 0.39% and are statistically insignificant.

11.7.6. Blanks

NBG inserts a blank sample at a frequency of 1 in 30. The blank samples used comprise clean silica sand from a quarry outside the district and well away from any gold mineralisation. This blank has been used extensively for the project by NBG and SGS. This insertion of blank samples is primarily a check on the sample preparation procedure to pick up where sample contamination has occurred. This can happen if rushers or pulverisers have not been cleaned properly between samples. For instance a shot of silica sand is used to clean the LM2 ring mills between each sample.

The results of the blank sampling programme are shown in *Figure 11-28: Plot of Silica Blank Samples for the Bau Project* below.

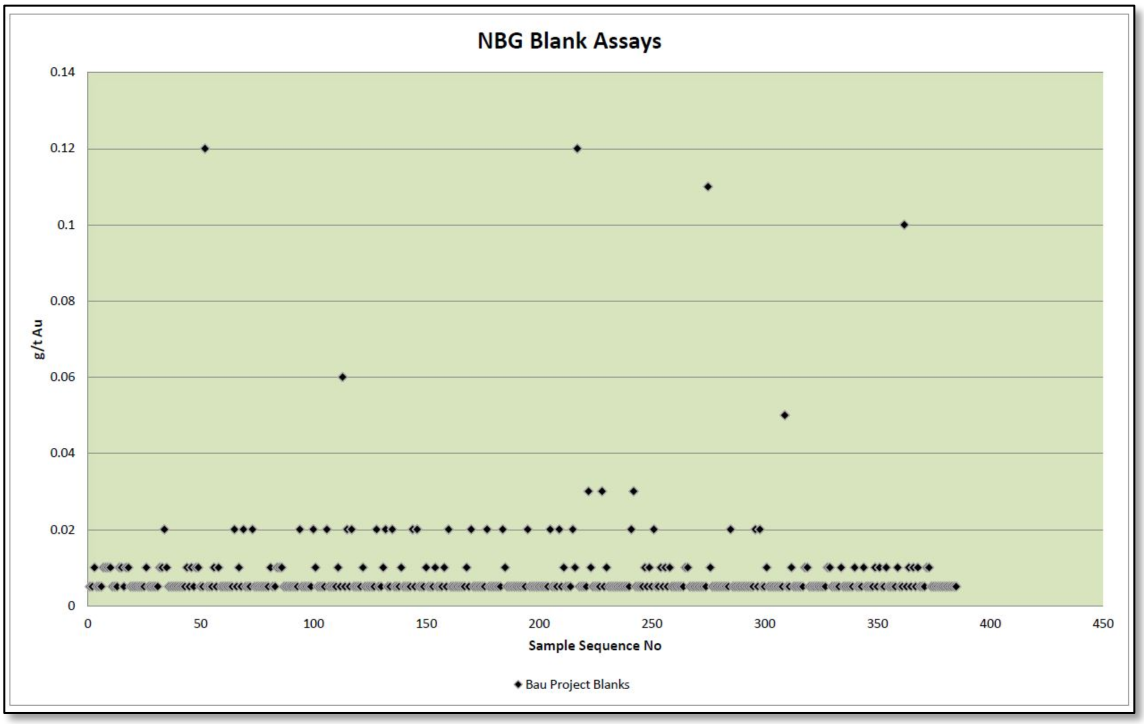


Figure 11-28: Plot of Silica Blank Samples for the Bau Project

In *Figure 11-28: Plot of Silica Blank Samples for the Bau Project* it can be seen that the vast majority of blank samples are below detection of 0.01 ppm Au. There some six (6) samples that fall outside the range of below detection to 0.04. These are not statistically significant and could be the result of minor contamination prior to the preparation procedure modifying the process to exclude roll mixing. The graph shows that overall there has been no systemic contamination from the sample preparation process.

11.7.7. Umpire Samples

NBG routinely sends pulps from approximately 10 % of all its samples to a separate independent laboratory for umpire analysis and the results compared. The Laboratory used is MAS in Thailand. The results are presented graphically in *Figure 11-29: Umpire Sample Log-Log Plot for the Bau Project Drilling* below.

NBG had MAS analyse 3171 samples representing 1:10 samples of the drill core from the drill programme. Overall the correlation between SGS and MAS is 0.9413. The greatest variance in comparative grades occurs in the low level gold grades. Where there is significant variation between samples at higher grades the relative number of samples does not show any significant bias.

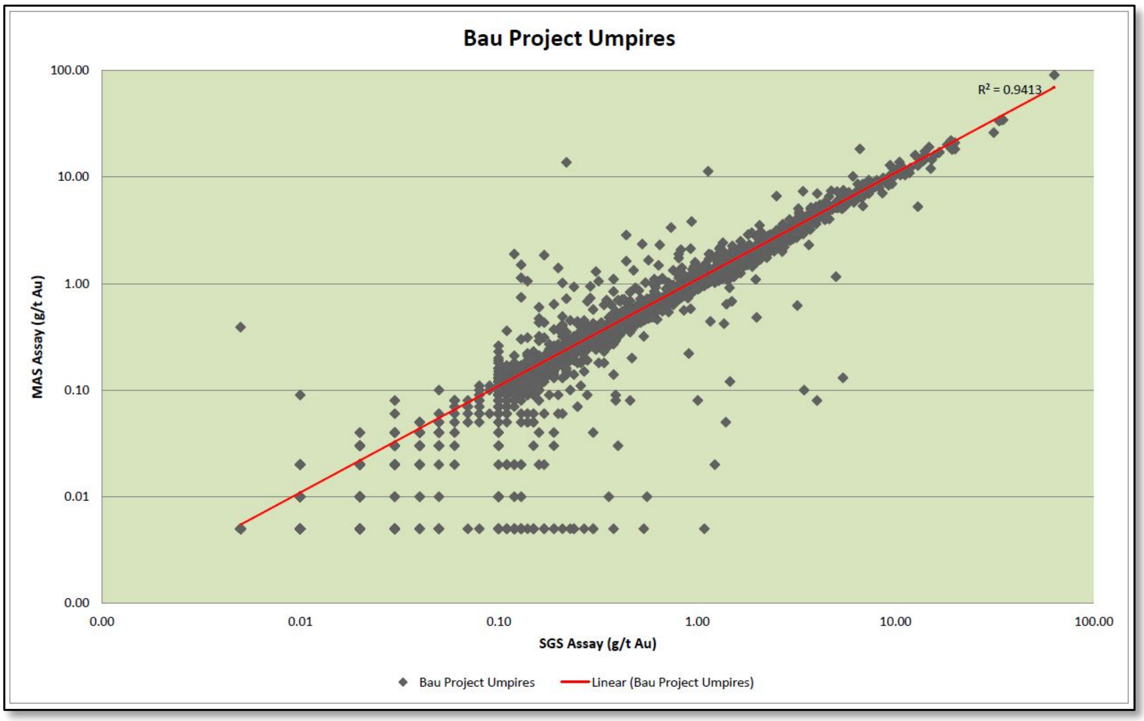


Figure 11-29: Umpire Sample Log-Log Plot for the Bau Project Drilling

11.7.8. Security

During the diamond drilling program since 2010, all drill core has been removed from drilling sites to secure sample preparation facilities at the field office in Bau as soon as practical under the supervision of the site geological staff.

The core logging and sample preparation areas are manned during working hours and have security patrols at night. The sample preparation and logging area are under the supervision of the senior site geologist, junior geologist and senior sample preparation staff. The Company employs on site security personnel and only authorised persons may enter the compound.

All samples are packaged in secure cloth bags and transported to SGS approximately 300 metres to SGS where they are received by SGS staff. The samples are recorded, batch numbers assigned by SGS and they pass into their system. Once samples are prepped the split for Fire Assay is retained at SGS for analysis while the split for ICP is sent via SGS's secure transport systems to SGS Perth or Port Klang via their freight system using DHL in Kuching.

Having the gold analyses carried out at SGS's laboratory on the Bau Mine Site eliminates a lot of security issues.

Only authorized NBG personnel are allowed access to the SGS sample preparation and laboratory areas and release of data only comes from the authorized laboratory manager to specific authorized senior personnel at NBG the Geology Manager, General Manager and Exploration Director.

Opinion on Adequacy of Sampling, Sample Preparation, Security and Analytical Procedures

The authors consider that the sampling, sample preparation, security and analytical procedures and results detailed in this report by and undertaken by NBG and SGS Laboratories have been carried out accordance with best industry practice in a systematic and secure manner. The authors consider that the data is valid for the purposes it is being used for in this report.

12. Data Verification

The data verification information below covers the work conducted for the August 2010 report plus the data verification work conducted since then for the February 2012 and November 2012 resource updates. All is included for completeness and disclosure purposes, as well as for easy reference within one document.

12.1. Data General

12.1.1. August 2010 Resource

The extensive site visits conducted by Terra Mining Consultants and Stevens & Associates during 2009 and 2010 have included visiting all major prospects that have been included in the resource estimates, a number of checks on data verification including visiting drill sites and key geological, reviewing existing reports on geology and mineralization and observing that the data fits the current mineralization and geological models for consistency with the resource modelling.

Exploration by BYG and its partner companies since the 1980's has produced a wealth of geological and geochemical data. Much of this is still intact and has been largely preserved by BYG, so while there were inconsistencies and errors found these have mostly been able to be verified, corrected or discarded as the case may be.

12.1.2. February & November 2010 Resource

Since 2010 all generated data from drilling and field work follows a set of standard procedures, logging checks and database verification. Many of these procedures and checks are covered in other sections within this report.

The database has been updated from information in hardcopy format has been regularly checked, cross-checked and verified before incorporation into the database. Many large format plans have been sourced from Besra's library, and from BYG storage, and converted to digital format for data capture.

12.2. Survey Control

12.2.1. August 2010 Resource

In collating the data for the resource modelling it became apparent that there were some issues with the survey control histroically. These largely stemmed from the use of various grids, mixing grids and datum's, and local grids for each project by past explorers. Issues encountered included drill holes collar coordinates in the database with elevation differences of tens of metres than on the ground at the same position, rotational errors with azimuth not consistently accounting for magnetic north/true north/grid north variations.

Menzies established their own datum based on UTM coordinates; however the parameters and conversions for this were found to be inconsistent and could not be duplicated.

Existing DEM and DTM models were mainly 10 metre or 20 metre contour intervals and accurate to +/- 10m in elevation. This has led to smoothing of the topography to the extent that drillholes could not be projected accurately to the surface in many instances.

In order to overcome these issues NBG decided to utilize existing aerial photography, establish survey control points and produce a DEM. The survey work was carried out by Resource Surveys Services, registered surveyors in Kuching. A number of survey control points were established at locations present at the time the air photos were taken and that could be verified today. In addition they surveyed a number of drill holes including all of NBG's drill holes. Data was captured in BRSO survey coordinates and converted to UTM coordinates. Elevations have been left as BRSO as there are no consistent control points for accurate conversion to UTM.

These control points were used by Precision Aerial Surveys of Auckland, New Zealand to produce a DEM to 1-2 metre accuracy. There are still some issues as the model is still a DEM but for the purposes of the resource modelling the elevation data now is far closer to reality than previously.

In addition, BYG had retained all the original hard copy survey records so it has been possible to reconstruct the BYG original survey control, done by traditional survey methods, establish local grid and BRSO and UTM coordinates for the same control points, drillholes, etc. and convert the old local grids to UTM. Where the orebody outcrops the ground surveyed topography has been used as collected by Resource Surveys or previous registered surveyors.

12.2.2. February & November 2010 Resource

During the course of the 2010, 2011 and 2012 drilling programmes and field work any historic drillholes have been resurveyed and their coordinates updated where applicable. Where original records or information has come to hand the original coordinates are compared to the current coordinates and verified. Some of these are in other recognised coordinate systems and have allowed the update of drillholes and other data, particularly those in local grid coordinates.

Updated topographic data was sourced from Malaysian government accredited aerial survey agents via our registered surveyor, Resource Surveys. This topographic information is based on radar aerial surveys and has an elevation accuracy of 1-5 metres depending upon vegetation cover. This topography covers all the areas of interest in Bau. Local survey updates are incorporated where applicable.

12.3. Drillhole & Sample Location

12.3.1. August 2010 Resource

Drillhole locations have been inspected by TMCSA. All NBG holes have been surveyed by registered surveyors. All NBG holes inspected had the collars set in concrete with the drillhole number, depth, declination, and start and completion date recorded. A selection of drill holes from past drilling campaigns have been checked using hand held GPS. Small discrepancies between the GPS readings and the surveyed positions in the database were consistent with accuracy limits of the handheld GPS.

Previous drillholes were captured by the mine surveyors during the BYG period and these drillholes have been converted from the local grid using the same survey control pegs whose coordinates have been verified by Resource Surveys the registered surveyors. These drillhole positions have also been cross-checked where available and are within reasonable tolerances.

With the recent survey work TMCSA have a greater level of confidence on drillhole locations for all phases of past work than previously available.

12.3.2. February & November 2010 Resource

All drillhole collars are surveyed by registered surveyors. Other control pegs and survey control lines are also surveyed by Besra's registered surveyors.

Other field work is surveyed by tape and compass from these known points (including drillholes) and verified by GPS as a cross-check. Where applicable these points are verified by a registered survey.

12.4. Geological Logging

12.4.1. August 2010 Resource

Representative drill core from all the prospects used in the resource modelling have been reviewed by TMCSA with drill core being compared with lithological descriptions in the drill logs. These were then checked against the lithological data entered into the database for the geological modelling.

Core logging has generally been descriptive and captured onto paper logs by all companies that have worked at Bau to date.

Menzies and RGC coded the paper logs and entered this coded data into geological databases.

Menzies captured the geological descriptions of their RC chip sampling on to paper logs. TMCSA reviewed these and have found them generally consistent and with geological descriptions generally correlating with geochemistry.

TMCSA are satisfied that the drill hole logging has been carried out in a professional manner, the data recorded and entered consistently into the database and is to accepted industry standards.

12.4.2. February & November 2010 Resource

Geological logging for drillholes, channels and trenches for 2010-2012 follows the Besra logging procedure and data validation procedures. The information is captured directly into electronic logging spreadsheets containing data validation routines and code tables. No paper based logging occurs and therefore transcription issues are removed.

12.5. Sample Data Verification

12.5.1. August 2010 Resource

NBG store all original signed assay sheets from its programs on the site office in Bau. These are in cupboards in an office complex that is locked outside work hours and with security guards on the premises. In addition, all historic paper records including dispatch sheets, original signed assay result sheets, and geological logs are stored in the same premises.

TMCSA have used these records extensively for checking and validating the databases. They have checked these against physical drill core from current and historic drill holes.

TMCSA are confident that the sample data has been verified to an acceptable level of confidence. Issues remain with some of the early fire assay data from the BYG laboratory where issues arose on converting from pennyweights to grams, and with the background/detection limits used. However, in most cases TMCSA have taken a conservative approach and generally where there are issues with fire assay data have used AAS data instead. In many cases the average grades of the AAS assays are less than those of the corresponding fire assay. Later assaying by the BYG laboratory has been independently checked by RGC and Menzies and issues identified, remedied or other independent and certified laboratories used.

NBG have used MAS in Thailand and ALS in Australia and TMCSA's investigations show this sample data to be valid.

12.5.2. February & November 2010 Resource

All the sample data for the 2010-2012 programmes have been assayed by SGS either in their Perth laboratory and/or the onsite laboratory (setup by SGS). SGS are ISO compliant and conduct a number of data verification and QA/QC procedures on the assay data.

Besra/NBG also conducts QA/QC and verification procedures on the data as well. More details on the QA/QC and other procedures can be found in other appropriate sections of this document. All sample data and returns are stored electronically and in hard copy for future reference and checking.

Sample data is also checked via umpire sampling at an alternate laboratory, namely MAS in Thailand. The umpire data and original assays have been cross-validated. This also is covered in other sections of this report.

12.6. Database Validation

12.6.1. August 2010 Resource

The following validation process was carried out on the primary data. Aspects of this are described in more detail in *Chapter 16* in relation to the resource modelling and using validation tools within the geological and mine modelling software.

- Take existing Access Database copy out relevant tables to Excel format on a project by project basis.
- Compile all recent data not in current database into project database, e.g. NBG data
- Check data for collar, surveys against original survey data sheets, check for duplication, omissions etc.
- Check assay data in database against original data from logs/assay sheets for Menzies/RGC/Gencor data.
- For BYG drill assay data, compile data in the existing database, enter primary data from original laboratory assay certificates if available and/or from hand entered data from drill logs, including fire assay, roasted fire assay, AAS, roasted AAS into separate columns. Compare with data in Access database, correct omissions, errors etc., derive an accepted value for the interval to use for the modelling.
- Geological logs: check codes on Access database, copy to excel on project by project basis. Modify codes where necessary; develop consistent coding system based on the existing Menzies coding system. Capture data from NBG paper logs into new database for each project modelled.

Overall some 1,614 drillholes within the resource areas modelled were validated in terms of collar, survey, geology, density, assay values and intervals. This included validation of 63,694 drill hole assay records and 1,610 channel/trench assay records.

Issues and errors found include missing assay data, missing drill collars, mis-plotted drillholes, different drill holes with same collar and survey data, etc. and these were systematically reviewed, rectified where possible or discarded if the data could not be verified of rectified.

As part of the validation process TMCSA collected representative samples from drill core of several projects and had them analysed independently at SGS Waihi, New Zealand in the case of core from Jugan, Pejiru and Sirenggok.

At Taiton, as this was a new project that had not been modelled previously samples were collected from several historic holes at Taiton A, Bungaat and Tabai. These were selected by the authors using drill logs and assay data and physical examination. The remaining core was

¼ cut using a diamond saw and prepared in the sample preparation facility on site and sent to MAS for analysis.

Table 12-1: SGS Waihi Check Verses Original Assays for Selected Drillholes show the comparative results of representative samples selected from Sirenggok, Jugan and Pejiru that were check assayed at Waihi, New Zealand.

Prospect	Drillhole No.	From (m)	To (m)	Sample No.	Original Au g/t	Check Au g/t
Sirenggok	SRDDH-01	122.00	123.00	231986	3.28	3.14
Sirenggok	SRDDH-01	158.00	159.00	231987	5.51	4.47
Jugan	JUDDH-03	28.00	29.00	231988	7.88	9.84
Jugan	JUDDH-04	85.00	86.00	231989	6.87	5.6
Pejiru	PJDDH-02	39.00	40.00	231990	5.2	4.98
Pejiru	PJDDH-03	36.00	37.00	231991	16.4	11.2

Table 12-1: SGS Waihi Check Verses Original Assays for Selected Drillholes

In the case of Waihi samples above the results are reasonably consistent and the variations are likely to be with the fact that ¼ core was chosen and reflects natural in homogeneity in the rock samples.

Table 12-2: Selected Assay Intervals from Taiton Database and *Table 12-3: Taiton Check Sampling Statistics* below show the check sampling results for Taiton.

BHID	FROM	TO	ORIGINAL SAMPLE NO	CHECK SAMPLE NO	ORIG. AU G/T	CHECK AU G/T	AREA
DDH104-36	17.90	18.75	BKTT480	232435	1.09	1.03	Overhead Tunnel
DDH104-36	18.75	19.65	BKTT481	232436	0.62	1.12	Overhead Tunnel
DDH104-36	22.45	22.75	BKTT483	232437	1.09	0.45	Overhead Tunnel
DDH104-36	22.75	23.00	BKTT484	232438	8.86	11.1	Overhead Tunnel
DDH104-36	23.00	23.80	BKTT485	232439	0.16	0.5	Overhead Tunnel
DDH104-36	23.80	24.00	BKTT486	232440	0.78	1.2	Overhead Tunnel
DDH104-162	34.55	35.55	2138	232441	0.75	0.13	Overhead Tunnel
DDH104-162	35.55	36.55	2139	232442	1.47	1.43	Overhead Tunnel
DDH104-162	36.55	38.55	2142	232443	9.43	6.75	Overhead Tunnel
DDH104-162	38.55	39.55	2143	232444	1.53	0.62	Overhead Tunnel
DDH104-162	39.55	40.55	2144	232446	1.08	0.15	Overhead Tunnel
DDH104-162	40.55	41.55	2145	232447	1.58	0.29	Overhead Tunnel
DDH104-162	41.55	45.55	2146	232448	1.80	0.59	Overhead Tunnel
DDH104-162	45.55	46.55	2147	232449	1.80	0.39	Overhead Tunnel
DDH104-18	0.00	1.54	BKTT284	232450	2.49	4.98	Bungaat
DDH104-18	1.54	3.08	BKTT285	232451	5.60	1.86	Bungaat
DDH104-18	3.08	4.62	BKTT286	232452	6.84	4.09	Bungaat
DDH104-18	4.62	5.20	BKTT287	232453	0.93	1.61	Bungaat
DDH104-18	5.20	6.60	BKTT288	232454	7.78	7.36	Bungaat
DDH104-18	6.60	7.40	BKTT289	232455	0.47	0.30	Bungaat

BHID	FROM	TO	ORIGINAL SAMPLE NO	CHECK SAMPLE NO	ORIG. AU G/T	CHECK AU G/T	AREA
DDH104-18	7.40	9.00	BKTT290	232457	0.78	0.86	Bungaat
DDH104-18	9.00	9.45	BKTT291	232458	2.18	2.12	Bungaat
DDH104-143	30.74	32.05	321	232459	20.25	9.08	Taiton A
DDH104-143	32.05	33.63	322	232460	17.62	21.60	Taiton A
DDH104-143	33.63	34.67	323	232461	4.92	23.20	Taiton A
DDH104-143	34.67	35.98	333	232462	10.93	11.50	Taiton A
DDH104-143	35.98	37.29	334	232463	9.81	5.76	Taiton A
DDH104-143	37.29	38.60	335	232464	3.32	1.35	Taiton A
DDH104-144	28.06	29.12	445	232465	11.61	13.20	Taiton A
DDH104-144	29.12	30.18	445	232466	17.80	20.00	Taiton A
DDH104-144	30.18	31.24	447	232468	29.01	20.20	Taiton A
DDH104-144	31.24	32.30	448	232469	2.07	2.16	Taiton A
DDH104-112	0.00	1.50	BKTT1571	232470	1.36	1.53	Tabai
DDH104-112	1.50	3.00	BKTT1572	232471	7.14	11.80	Tabai
DDH104-112	3.00	5.00	BKTT1573	232472	32.98	24.40	Tabai
DDH104-112	5.00	6.20	BKTT1574	232473	56.78	45.60	Tabai
DDH104-112	6.20	7.00	BKTT1575	232474	63.92	45.20	Tabai
DDH104-112	7.00	8.50	BKTT1576	232475	87.04	69.20	Tabai
DDH104-112	8.50	9.20	BKTT1577	232476	106.08	89.20	Tabai
DDH104-112	9.20	11.90	BKTT1578	232477	0.68	0.32	Tabai
DDH104-112	11.90	13.50	BKTT1579	232479	0.68	0.49	Tabai
DDH104-112	13.50	15.30	BKTT1580	232480	1.72	1.54	Tabai
DDH104-112	15.30	16.10		232481		0.19	Tabai
DDH104-123	1.30	2.30	BKTT1778	232482	4.46	3.08	Tabai
DDH104-123	2.30	4.00	BKTT1779	232483	12.56	5.56	Tabai
DDH104-123	4.00	5.50	BKTT1780	232484	2.10	0.58	Tabai
DDH104-123	5.50	7.60	BKTT1781	232485	3.21	0.42	Tabai
DDH104-123	7.60	9.10	BKTT1782	232486	3.07	1.14	Tabai
DDH104-123	9.10	10.60	BKTT1783	232487	2.80	10.10	Tabai
DDH104-123	10.60	12.10	BKTT1784	232488	5.14	7.08	Tabai
DDH104-123	12.10	13.60	BKTT1785	232490	12.00	18.70	Tabai
DDH104-123	13.60	15.10	BKTT1786	232491	6.85	7.27	Tabai

Table 12-2: Selected Assay Intervals from Taiton Database

FIELD	AU_CHK	AU_ORIG
No of Records	57	57
No of Samples	57	56
No of Missing Values	-	1
Minimum	0.13	0.16
Maximum	89.200	106.080
Range	89.070	105.924
Total of Values	549.790	622.623
Mean	9.645	11.118
Variance	278.939	423.931
Standard Deviation	16.701	20.590
Standard Error	2.212	2.751
Skewness	2.999	3.120
Kurtosis	9.680	9.648
Geometric Mean	2.680	3.756
Sum of Logs	56.189	74.108
Mean of Logs	0.986	1.323
Log Variance	2.956	2.098
Log Estimate of Mean	11.748	10.725

Table 12-3: Taiton Check Sampling Statistics

General observations with the Taiton data are that data range is higher in the original samples than in the check samples, but overall where there are high values in the original data there are high values in the check data. The samples were of ¼ cores from BQ sized core, whereas the original samples were ½ BQ. The aim of the check sampling was determine in the first instance that the gold content of the core was real. Similar orders of magnitude in comparative samples are generally observed.

From the database validation carried out, TMCSA are satisfied with the data integrity used in the resource modelling. Other database validation is covered in *Chapter 16* of this report.

12.6.2. February & November 2012 Resource

Database validation is conducted regularly and when the resource definition process begins using the various tools available within the standard mining software packages e.g. Datamine/CAE Mining.

13. Mineral Processing & Metallurgical Testing

13.1. Introduction

A selection of metallurgical testwork and studies has been undertaken for previous companies on the Bau Gold Project. This work focused on the Jugan and Pejiru deposits only and was compiled in six previous metallurgical reports issued between February, 1994 and August, 1998.

Starting in 2012 Olympus Pacific Minerals Ltd and later Besra Gold Inc. conducted metallurgical testing both in house and in outside laboratories, specifically on the Jugan ore, which will be processed first, to develop the best processing option for the recovery of gold from the ore. A summary of the historical testwork and work undertaken by or under supervision of Besra to date is provided in the sections below. The location of the various metallurgical drillholes is shown in *Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines)* below. The recent Jugan metallurgical drillcore is shown in blue on the diagram below. Historic drillholes are shown in green. The full ore outline is the dashed blue line with the surface expression and shallow orebody depth area shown by the red shading.

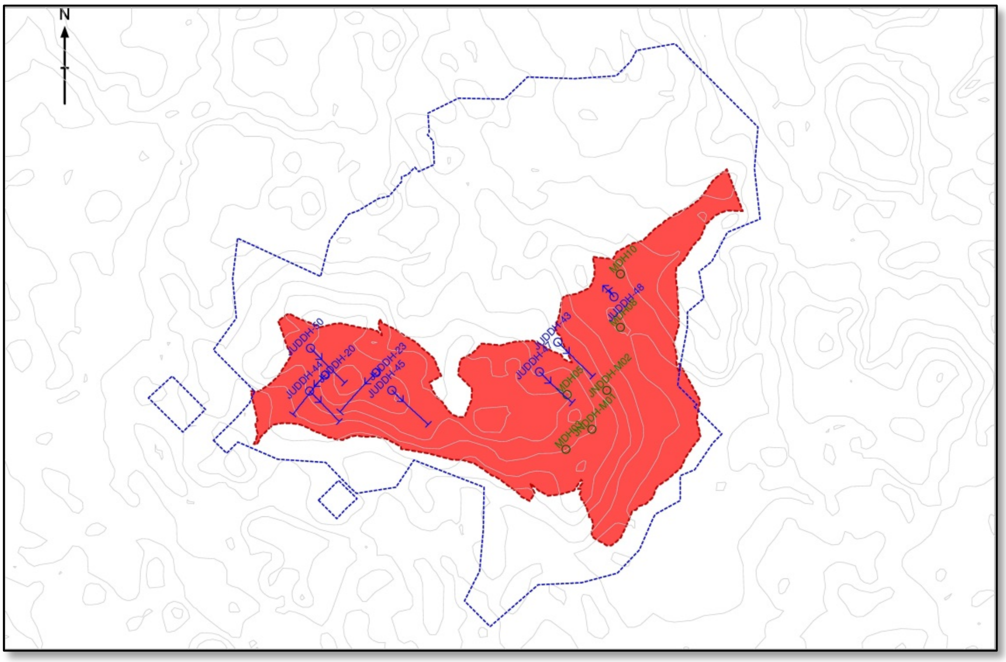


Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines)

Some of the historic testwork was conducted on Pejiru deposit. This deposit is not part of the current feasibility, but is included for completeness in terms of 43-101 requirements.

13.2. Historical Metallurgical Testwork

Orway Mineral Consultants (Orway) have summarised the historical metallurgical testwork in the report *"Bau Refractory Gold Ore Project Metallurgical Testwork", Orway Mineral Consultants, October 2008*. Portions of that summary have been extracted and are included in this section as the authors have determined that it is a professional and reasonable summary of the historical metallurgical testwork. Extracts from the Orway report are shown in italic font with no editing other than formatting for this report.

The reports detailing historical metallurgical testwork for the Jugan and Pejiru deposits are listed below:

- Gravity Concentration of Bau Ore Samples, Lakefield Oretest, Report No: 8793, 23 October 2001; **(Reference 5)**;
- Recovery of Gold from Bau Drill Core Samples, MIM-HRL Laboratory , Report No: 0616, 15 June 1997; **(Reference 6)**;
- Flotation of Jugan Hill Core Samples, GENCOR Process Research, Report No: 94/13, 16 February 1994; **(Reference 7)**;
- Bulk Sulphide Flotation Testwork Conducted Upon Samples of Ore from the Bau Gold Deposit for Menzies Gold N.L., AMMTEC Ltd., Report No: A6324, August 1998; **(Reference 8)**;
- Metallurgical Testwork Conducted Upon Pejiru Composite from Bau Gold Deposit for Project Advisory Services Pty. Ltd., AMMTEC Ltd., Report No: A5487, April 1997; **(Reference 9)**;
- Metallurgical Testwork Conducted Upon Jugan Composite from Bau Gold Deposit for Project Advisory Services Pty. Ltd., AMMTEC Ltd., Report No: A5517, April 1997; **(Reference 10)**.

All historical work was conducted on composite drill core samples. Locations and depth ranges of the samples were reported only for the MIM and the GENCOR testwork. About 110 kg each of Jugan and Pejiru ore samples were used for the gravity concentration work at Lakefield Orestest. Drill core samples from two east locations for Jugan (JNDDHM01 and JNDDHM02, see *Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines))* and three locations for Pejiru were supplied to the MIM-HRL laboratory. Four Jugan drillcore samples located in the east area of the Jugan property (MDHM-03, MDHM-05, MDHM-08 and MDHM-10, see *Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines))* were provided to Gencor Process Research. A total of fifty-five (55) samples were collected from quarter core (1 metre interval from 5 metres to about 60 metres) with a total weight of 57 kilograms. For the testwork conducted at AMMTEC in 1997, 400 kilograms each of Jugan and Pejiru composite samples were provided. For the bulk flotation work at AMMTEC in 1998, 200 kilograms each of Jugan and Pejiru were provided.

The Jugan composite samples were reasonably representative of the the Jugan deposit based on the assays given in *Table 13-1: Chemical Assays of the Jugan Ore Samples used in Metallurgical Testwork* below. The documented samples were taken from the East area of the Jugan deposit from vertical drillcores and no location variability testwork was conducted. The Pejiru composite samples were taken from selected high grade gold areas and were therefore less representative of the Pejiru gold deposit.

The historical metallurgical test work is summarised in the following sub-sections.

13.2.1. Summary of Historical Metallurgical Testwork

13.2.1.1. Chemical Composition of Jugan & Pejiru Ore Samples Tested

Table 13-1: Chemical Assays of the Jugan Ore Samples used in Metallurgical Testwork and *Table 13-2: Chemical Assays of the Pejiru Ore Samples used in Metallurgical Testwork* cover the available chemical assays of the Jugan and Pejiru ore samples used in metallurgical testwork. Chemical assays reported for both ore samples showed variations in each report. However, it is possible to see that the Pejiru ore has a higher grade as compared to Jugan. While Jugan has a higher arsenic content Pejiru had higher mercury levels. Pejiru seemed to contain more carbon (with a small organic carbon component) than Jugan.

The Jugan ore samples had higher arsenic, iron and sulphur levels than the Pejiru ore samples. This indicates that a higher mass will be associated with the flotation concentrate in terms of arsenopyrite and pyrite with the Jugan ore. The Jugan ore samples had higher aluminium levels and substantially lower calcium levels.

Element	Reference 5	Reference 6	Reference 7	Reference 8	Reference 10
Au	3.43 g/t	2.36 g/t	2.55 g/t	2.72/2.74 g/t	2.35/2.42 g/t
Ag	-	5 g/t	-	0.2 g/t	0.1/0.1 g/t
As	1.25 %	0.87 %	1.24 %	1.32 %	1.23/1.24 %
Al	-	-	-	8.63 %	9.19/9.2 %
Bi	-	-	-	<2 g/t	<5 g/t
C_{total}	-	-	-	-	1.68/1.67 %
$C_{organic}$	-	-	-	0.187 %	0.22/0.251 %
Ca	-	-	-	2.36 %	2.94/2.87 %
Cd	-	-	-	3 g/t	<2 g/t
Co	-	-	-	17 g/t	19 g/t
CO_3^{-2}	-	-	7.26 %	-	-
Cr	-	-	-	44 g/t	35/32 g/t
Cu	-	-	-	28 g/t	29/27 g/t
Fe	-	4.19 %	3.98 %	4.72 %	4.87/4.57 %
Hg	0.25 g/t	-	-	0.1 g/t	0.09/0.072 g/t
K	-	-	-	2.49 %	2.43/2.42 %
Mg	-	-	-	8107 g/t	9390/9100 g/t
Mn	-	-	-	802 g/t	1170/1089 g/t
Na	-	-	-	1598 g/t	1704/1650 g/t
Ni	-	-	-	19 g/t	19/16 g/t

Element	Reference 5	Reference 6	Reference 7	Reference 8	Reference 10
P	-	-	-	354 g/t	426/380 g/t
Pb	-	-	-	30 g/t	29/28 g/t
S_{total}	2.6 %	3.09 %	-	2.6 %	2.87/2.82 %
$S_{sulphide}$	-	-	2.93 %	-	2.82/2.76 %
Sr	-	-	-	189 g/t	239/235 g/t
Ti	-	-	-	3015 g/t	3100/3164 g/t
V	-	-	-	88 g/t	105/100 g/t
Zn	-	-		209 g/t	186/185 g/t
Zr	-			15 g/t	23/30 g/t

Table 13-1: Chemical Assays of the Jugan Ore Samples used in Metallurgical Testwork

Element	Reference 5	Reference 6	Reference 8	Reference 9
Au	3.28 g/t	5.22 g/t	6.12/6.08 g/t	5.42/5.54 g/t
Ag	-	5.1 g/t	1.8/1.7 g/t	1.6/1.6 g/t
As	0.6 %	<0.1 %	0.285/0.2825 %	0.2642/0.2611 %
Al	-	-	2786/2964 g/t	2462/2430 g/t
Bi	-	5.1 g/t	<2 g/t	<5 g/t
C_{total}	-	1.59 %	-	9.48/9.77 %
$C_{organic}$	-	-	0.068/0.065 %	0.053/0.059 %
Ca	-	-	31.6/31.2 %	15.9/16.0 %
Cd	-	-	<2 g/t	<2 g/t
Co	-	-	<5 g/t	<5 g/t
Cr	-	-	12/14 g/t	10/11 g/t
Cu	-	-	7/7 g/t	5/4 g/t
Fe	-	1.59 %	1.53/1.52 %	1.11/1.13 %
Hg	46.7 g/t	-	22.0/22.3 g/t	31/34 g/t
K	-	-	118/74 g/t	104/97 g/t
Mg	-	-	419/438 g/t	399/413 g/t
Mn	-	-	483/479 g/t	469/483 g/t
Na	-	-	28/33 g/t	42/43 g/t
Ni	-	-	4/5 g/t	8/6 g/t
P	-	-	170/187 g/t	193/202 g/t
Pb	-	-	30/29 g/t	33/31 g/t
S_{total}	2.9%	1.46 %	1.28/1.24 %	0.98/0.88 %
$S_{sulphide}$	-	-	-	0.86/0.76 %
Sr	-	-	54/59 g/t	53/52 g/t
Ti	-	-	121/129 g/t	1276/1344 g/t
V	-	-	7/7 g/t	8/9 g/t
Zn	-	-	26/25 g/t	21/23 g/t
Zr	-		<5 g/t	23/30 g/t

Table 13-2: Chemical Assays of the Pejiru Ore Samples used in Metallurgical Testwork

13.2.1.2. Mineralogical Composition of Jugan & Pejiru Samples Tested

References 9 and 10 indicated that the dominant mineral phase in Pejiru ore was pyrite whereas arsenopyrite was the dominating phase in Jugan ore sample.

The results of diagnostic leach studies given in these reports provided a good indication of the gold occurrence in Jugan and Pejiru ore samples used. These results are summarised below in *Table 13-3: Occurrence of Gold in Pejiru and Jugan Ore Samples as Established by Diagnostic Leaching*

Source	Pejiru	Jugan
Reference 9	*Free Gold: 16.48%* *Locked in FeAsS: 18.37%* *Locked in FeS2: 41.99%* *Encapsulated in SiO2; 3.16%*	-
Reference 10	-	*Free Gold: 0.66%* *Locked in FeAsS: 69.38%* *Locked in FeS2: 25.19 %* *Encapsulated in SiO2: 4.77%*

Table 13-3: Occurrence of Gold in Pejiru and Jugan Ore Samples as Established by Diagnostic Leaching

The Jugan ore sample had very low free gold but the Pejiru sample had a significant amount of free gold. It should be noted here that the free gold was based on the percentage of gold that was recovered by direct cyanidization of the feed ore and does not reflect free gold recoverable by gravity.

Both samples had pyrite and arsenopyrite as the main sulphide minerals hosting gold. Pyrite was the dominant gold hosting mineral in Pejiru ore sample whereas arsenopyrite was more abundant mineral in Jugan ore sample. A significant amount of gold was also associated with quartz in both samples.

13.2.1.3. Comminution Data on Jugan & Pejiru Ore Samples Tested

The established data on comminution characteristics of Pejiru and Jugan ores are summarised in *Table 13-4: Comminution Data on Pejiru and Jugan Ore Samples*

Source	Pejiru	Jugan
Reference 9	*Bond abrasion index (Ai) = 0.0616* *Bond rod mill work index (kWh/t) = 11.1* *Bond ball mill work index (kWh/t) = 9.7*	-

Source	Pejiru	Jugan
Reference 10	-	*Bond abrasion index (Ai) = 0.015* *Bond rod mill work index (kWh/t) = 13.5* *Bond ball mill work index (kWh/t) = 11.3*

Table 13-4: Comminution Data on Pejiru and Jugan Ore Samples

These results are characteristic of softer ores requiring low grinding energies. The abrasion index is very low.

13.2.1.4. Direct Cyanidation of Jugan & Pejiru Ore Samples

Only studies reported in References 6, 9 and 10 had data on direct cyanidation of Pejiru and Jugan ore samples. These results are summarised in *Table 13-5: Direct Cyanidation Results Reported for Pejiru and Jugan Ore Samples (P80 = 75µm)*

Source	Pejiru	Jugan
Reference 6	*Au recovery: 13.4%* *Ag recovery: 20%* *NaCN Consumption: 2.6 kg/t* *Lime Consumption: 1.8 kg/t*	*Au recovery: 4.6%* *Ag recovery: 20%* *NaCN consumption: 2.2 kg/t* *Lime consumption: 2.3 kg/t*
Reference 9	*Au recovery: 15.33%* *NaCN consumption: 1.56 kg/t* *Lime consumption: 1.43 kg/t*	-
Reference 10	-	*Au recovery: 0.62%* *NaCN consumption: 1.74 kg/t* *Lime consumption: 1.17 kg/t*

Table 13-5: Direct Cyanidation Results Reported for Pejiru and Jugan Ore Samples (P80 = 75µm)

Both ore samples responded poorly to direct cyanidation, with Jugan being less responsive in comparison with Pejiru. The low cyanidation gold extraction is typical of arsenopyrite-pyrite refractory ores.

13.2.1.5. Gravity Gold Recovery from Pejiru & Jugan Ore Samples

The testwork aimed at assessing the amenability of the ore samples to gravity concentration using a Falcon concentrator for varying grind sizes (P_{80} 106, 75 and 53µm) and to compare the Falcon and Knelson concentrators with the Kelsey Jig to see if a positive response to gravity concentration was obtained.

Gravity tests were performed on 100 kg samples from both the Pejiru and Jugan deposits. There was no information on sampling and mineralogy but the chemical analysis of the samples was provided. The sample head grade was 3.43 g/t Au for Jugan and 3.28 g/t Au for Pejiru.

Pejiru gravity concentration did not provide any significant upgrading and the gold recovery remained below 10% at all grind sizes. Further gravity testwork was not considered worthwhile.

In contrast, the Jugan ore sample demonstrated a positive response to gravity concentration; with gold recovery ranging from 30% up to 36% as the grind was reduced from 106µm to 53µm. In spite of these positive results, further gravity testwork was abandoned due to the more favourable results demonstrated by flotation – giving both higher gold recoveries and grades. These results are supported by mineralogical characterisation with the predominance of sulphide minerals and the absence of free gold (section 13.3.2.2 below).

13.2.1.6. Flotation of Jugan & Pejiru Ore Samples

Reported flotation test results both on Pejiru and Jugan ore samples are summarised below in Table 13-6: Flotation Test Results for Pejiru and Jugan Ore Samples.

Source	Pejiru	Jugan
Reference 6	Feed : 5.22 g Au/t 19.7 g Au/t in con; 0.606 g Au/t in tails; 91.2 % Au rec Slurry density: 25% Conditioning time: 20 mins Reagents: CuSO4, SIBX, MIMFloat Cumulative flot. Time: 35 mins Mass pull: 27.3 %	Feed: 2.36 g Au/t 8.87 g Au/t in con; 0.26 g Au/t in tails; 91.4 % Au rec Slurry density: 25% Conditioning time: 10 mins Reagents: CuSO4, SIBX, MIMFloat Cumulative flot. Time: 29 mins Mass pull: 29.8 %
Reference 7	-	Feed: 2.55 g Au/t Cleaner concentrate: 22.8 g Au/t; 24 % S; 10.5% wt. pull; 92.9% Au rec Tails: 0.204 g Au/t; 0.29%S CuSO4 100 g/t; SIBX 40 g/t; Senkol 294 40 g/t Flot. Time: 20 mins
Reference 8	Feed: 5.47 g Au/t Concentrate: 64.2 g Au/t; 16.9% S; 71.14% Au recovery Tail: 1.68 g Au/t; 0.05% S CuSO4 100 g/t; PAX per stage 20	Feed: 2.64 g Au/t Concentrate: 12.8 g Au/t; 13%S; 88.02% Au recovery Tail: 0.386 g Au/t; 0.22%S CuSO4 100 g/t; PAX per stage 20 g/t; Frother

Source	Pejiru	Jugan
	g/t; Frother 5 g/t *Flot. Time: 60-70 mins* *Mass Pull: 6 %*	*5 g/t* *Flot. Time: 60-70 mins* *Mass Pull: 18 %*
Reference 9 & 10	*Concentrate Tailing* *Au g/t* *Au g/t* *P80=106 µm 38.9 1.25* *P80=90 µm 39.5 1.16* *P80=75 µm 34.7 1.13* *P80=45 µm 29.8 0.93* *With reagent optimisation:* *P80=75 µm* *Concentrate gold recovery: 68.48 to 84.43 %* *Reagents used: CuSO4, AP238, PAX, SEX, SIBX, Frother*	*Concentrate Tailing* *Au g/t Au g/t* *P80=106 µm 5.87 0.512* *P80=90 µm 6.40 0.23* *P80=75 µm 6.27 0.22* *P80=45 µm 5.80 0.234* *With reagent optimisation:* *P80=75 µm* *Concentrate gold recovery: 87.69 to 96.12 %* *Reagents used: CuSO4, AP238, PAX, SEX, SIBX, Frother*

Table 13-6: Flotation Test Results for Pejiru and Jugan Ore Samples

Flotation test conditions and reagent schemes varied between the various laboratories. Recoveries are higher for the Jugan ore at about 92 %. Sulphur and gold extraction kinetics were slow due to the inhibiting effects of slimes. Incremental dosage of flotation reagents must be employed with the Jugan ore types. An investigation of desliming and its effects on gold and sulphur extraction kinetics to bring more light on this issue was recommended.

13.2.1.7. The BIOX technology for Gold Extraction from Jugan Ore

Minsaco Resources PTY Limited issued an internal report, "Bukit Young-Gencor, Jugan Hill Project, Final Feasibility report, May 1994", proposing the treatment of Jugan refractory ore in a flowsheet comprising comminution, flotation, bio-oxidation of the flotation concentrate (BIOX) and carbon-in-pulp cyanidation (CIL) for gold extraction.

BIOX is a bio hydrometallurgical process for the pre-cyanidation treatment of refractory gold ores. The process offers an alternative to conventional roasting or pressure oxidation techniques (POX). The BIOX process comprises contacting the refractory sulphide flotation concentrate with a strain of a BIOX mixed bacterial culture in a series of aerated reactors for a suitable treatment period while maintaining an optimum operating environment (39 ^{o}C to 41 ^{o}C, pH 1.4 to 1.6, dissolved O_2 3.5 to 4.5 ppm). A bacterial culture already adapted to a pyrite/arsenopyrite concentrate was used for the inoculum build-up test. The bacteria oxidise the sulphide minerals, thereby liberating the occluded gold for subsequent recovery by conventional cyanidation. Nutrients are provided to the reactors to maintain the bacterial population.

The report states that BIOX testwork carried out by Gencor in 1989 gave varying gold dissolutions between 73% and 92% in CIL on different sample runs. It was concluded that the Jugan Hill deposit is not homogeneous with respect to amenability to cyanidation after bacterial oxidation. However, the most representative sample tested gave dissolutions in excess of 90%.

The 1994 Gencor testwork targeted maximising gold recovery into a flotation concentrate suitable for Gencor's Biox process. Drill core samples were used in the tests and detailed information on the intervals sampled, the weight and grade of each interval were provided.

As the title implies, only samples from Jugan was tested. The head grade of the sample was 2.55 g/t Au with high arsenic content (1.24 %).The testwork program covered both grinding and flotation tests. The ore sample was found to be very friable and it was advised that care should be taken in plant design with milling residence times low enough to avoid over grinding.

Approximately, 95 % gold recovery to concentrate was reported for rougher flotation. The grades of the cleaner concentrate and tails were as below:

- *Concentrate - 22.8 g/t Au; 24 % S; 10.5 % weight pull; 92.9 % Au recovery.*
- *Tails - 0.204 g/t Au; 0.29 %S; 7.1 % Au to tails.*

The suitability of the concentrate to Biox was not commented on. There was also no indication of any Biox tests conducted.

13.2.1.8. The Albion Technology for Gold Extraction from Pejiru & Jugan Ores

The aim of the testwork was to determine the flotation characteristics of the ore samples both from Pejiru and Jugan and to investigate further processing of concentrates through the Albion process. Cyanide leaching of the oxidised residues was undertaken to determine the gold and silver recoveries from the samples.

The Albion is an alternate process to POX and BIOX for oxidation of flotation concentrate and consists of two process steps. The first step is ultrafine grinding of the refractory flotation concentrate which renders the particles more reactive to oxidation and greatly reduces particle passivation by oxidation products. The second step is an oxidative leach of the finely ground sulphides to breakdown the sulphide matrix and liberate the gold prior to cyanidation. The oxidation is carried under atmospheric conditions in a series of reactors with injection of oxygen. The oxidation produces heat and temperature is controlled by evaporative cooling below the boiling point.

The samples were provided as half 65 mm core sections. The Jugan sample head grade was 2.36 g Au/t and 5.22 g Au/t for Pejiru. Some limited mineralogical information on these samples was also provided.

The test program covered flotation, ultrafine grinding of the concentrates, hot oxidative leaching in acidic conditions, and iron precipitation in the form of goethite from the leach liquor. The concentrates of both samples responded well to ultrafine grinding and did not display a high viscosity at fine grind sizes.

The testwork program of a scoping level was designed to test the amenability of the ores to the Albion process under un-optimised and conservative conditions. Despite this, gold recoveries of around 85-88% were obtained from the oxidized flotation concentrate and overall recoveries of about 80 % with respect to the feed ores. This made it clear that an oxidative process is required to liberate gold associated with arsenopyrite and pyrite before conventional cyanidation routes. Further oxidative testwork is recommended to pursue this recovery route.

13.3. Metallurgical Testwork

13.3.1. Introduction

Refractory ores worldwide and at Bau are associated with the presence of sub-microscopic gold in arsenopyrite and pyrite. This means the gold is both present in ultra-fine colloidal form or in solid solution in the sulphide particles and not visible at high microscopic resolution. In this form the gold is not accessible to cyanide and an oxidation pre-treatment step is required to liberate the gold. Refractory gold ores often contain organic carbon which can cause preg-robbing during the gold extraction with cyanide after the pre-treatment step. The potential preg-robbing of organic carbon is minimised by a carbon pre-flotation step and cyanidation in the presence of carbon (CIL process).

In the past oxidation by roasting the ore was the process of choice but, due to environmental issues processes such as pressure oxidation (POX) and biological oxidation (BIOX) have, with a few exceptions, replaced roasting. Recently ultra-fine grinding followed by atmospheric oxidation (Albion Process) has been implemented at the Las Lagunas tailings re-treatment project in the Dominican Republic. Other technologies are available but have not reached full commercial status.

Based on review of the historical metallurgical work, Besra has undertaken an extensive metallurgical program for the beneficial extraction of gold from the Bau ores. The work was specifically undertaken on Jugan ore which is the most advanced project and will be implemented first on commercial scale. Although a fair bit of work was performed in the past on flotation to recover 90% plus of the gold in a reduced mass little work was done on the downstream gold extraction processing. There is indication that both the BIOX and Albion flotation concentrate oxidation processes can lead to 90% gold extraction from the concentrate by cyanide carbon-in-pulp leaching for an overall recovery of about 81% of the feed gold.

Besra decided, in a first phase of the metallurgical testwork, to confirm earlier results for the Albion process and compare these with POX, starting with the same flotation concentrate. The flotation concentrate was prepared by SGS Lakefield Oretest in Perth who also carried out the POX testwork on half of the flotation concentrate. The Albion testwork was carried out on the other half of the flotation concentrate by hrltesting in Brisbane under the supervision of Core Process Engineering.

The second phase of the metallurgical work focused on flotation optimisation on a master Jugan sample and three variability metallurgical core samples. This work was carried out at hrltesting Laboratories. Additional POX optimization work was performed on a master flotation concentrate bulk sample at SGS Lakefield Orestest. One additional Jugan metallurgical sample was floated separately at the Maelgwyn laboratories in South Africa to produce concentrate for BIOX amenability testing at SGS South Africa under the supervision of the Goldfields BIOX group (now Biomin).

The sections below summarize the results of the various test programs to date.

13.3.2. Chemical Composition and Mineralogy of the Jugan Feed Ore Samples

13.3.2.1. Chemical Assays

The chemical composition of the Jugan metallurgical drill core samples that were used in the various metallurgical testwork investigations are listed in *Table 13-7: Chemical Assays of Jugan Metallurgical Drill Core Samples* below. The assays of all elements are in within a narrow range with the exception of gold and silver. The variability samples were sent to hrltesting, with JUDDH-48 E as a master sample and JUDDH-43 E, JUDDH-44 W and JUDDH-50 W as variability samples with gold assays in the range 1.05 to 5.09 g/t. The location of the various metallurgical drillholes is shown in *Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines)*.

Laboratory	***SGS***	***hrlTesting***				***Maelgwyn***
Element	*JUDDH-20 & 23*	*JUDDH-48 East*	*JUDDH-43 East*	*JUDDH-44 West*	*JUDDH-50 West*	*JUDDH-45 & 47*
Au (g/t)	*2.83*	*1.91*	*1.18*	*5.09*	*2.39*	*1.47*
Ag (g/t)	*0.59*	*<0.2*	*0.25*	*0.58*	*0.2*	*0.26*
As (wt%)	*1.04*	*1.15*	*1.05*	*1.30*	*1.03*	*1.07*
S (wt%)	*3.00*	*2.92*	*2.82*	*2.93*	*3.11*	
Fe (wt%)	*4.18*	*4.82*	*4.58*	*4.34*	*4.36*	*4.78*
Al (wt%)	*0.32*	*0.32*	*0.44*	*0.41*	*0.34*	*0.35*
Cu (g/t)	*39*	*37*	*39*	*36*	*38*	*37*
Co (g/t)	*16*	*17*	*19*	*16*	*16*	*17*
Ni (g/t)	*36*	*37*	*41*	*35*	*31*	*37*
Zn (g/t)	*89*	*100*	*101*	*101*	*89*	*105*
Cd (g/t)	*<1*	*<1*	*<1*	*<1*	*<1*	*<1*
Mn (g/t)	*1,160*	*1,408*	*1,468*	*1,232*	*1,170*	*1,411*
Cr (g/t)	*10*	*7*	*9*	*9*	*6*	*10*
Sb (g/t)	*42*	*52*	*44*	*47*	*49*	*47*
Pb (g/t)		*12*	*17*	*17*	*15*	*13*

Laboratory	SGS	hrlTesting				Maelgwyn
Hg (g/t)	<1	<1	<1	<1	<1	<1
Ba (g/t)	25	29	39	28	25	28
Bi (g/t)	7	8	<5	<5	<5	<5
Ctotal (wt%)						
Corg (wt%)						
Ca (wt%)	2.47	2.62	2.77	2.49	2.51	2.93
Mg (wt%)	0.87	1.03	0.97	0.95	0.93	1.05
K (wt%)	0.162	0.15	0.19	0.19	0.16	0.17
Na (g/t)	398	960	1,100	290	693	640
Ti (g/t)	<10	<10	<10	<10	<10	<10
V (g/t)	9	11	13	11	11	12

Table 13-7: Chemical Assays of Jugan Metallurgical Drill Core Samples

Head grade assays provided by SGS Lakefield Oretest and hrltesting on the samples as received are given in *Table 13-8: Head Grade Assays* below.

	Na %	K %	Ca %	Mg %	Al %	As %	Mn %	Ti %	Fe %	Zn ppm	Cu ppm	Au ppm	S %
Jugan 20-23	0.13	2.59	2.48	1.13	8.00	1.13	-	-	4.41	-	-	-	3.02
Jugan 44 West	0.09	2.54	2.24	0.96	7.63	1.43	0.11	0.32	3.96	103	73	5.09	2.90
Jugan 43 East	0.19	2.38	2.52	0.93	7.70`	0.96	0.13	0.33	3.98	109	36	1.15	2.98
Jugan 50 West	0.14	2.51	2.51	0.95	8.02	1.18	0.11	0.34	4.18	111	40	2.67	2.95
Jugan 48 East	0.20	2.4	2.4	0.93	8.40	1.12	0.12	0.34	4.2	102	37	1.99	2.90

Table 13-8: Head Grade Assays

It can be noted that the Al, K and Ti feed assays are substantially higher than reported earlier by SGS at Bau. Perusal and comparison of the assay data for the Jugan ore zones suggests that there is very little difference with respect to mineral distributions in these ore zones apart from minor variations in arsenic and gold contents. This implies that the only visible difference may be that of minor variations in arsenopyrite content and further, that the concomitant increases in arsenic coinciding with that of increases in gold showing an evident correlation (see *Figure 13-2: Feed Gold Grade vs. Arsenic Content*). This implies either a physical style of gold entrapment or more probably a solid solution gold component in the arsenopyrite lattice. This would be in keeping with the inferred refractory 'Carlin style' gold emplacement that has been advocated for Bau Gold.

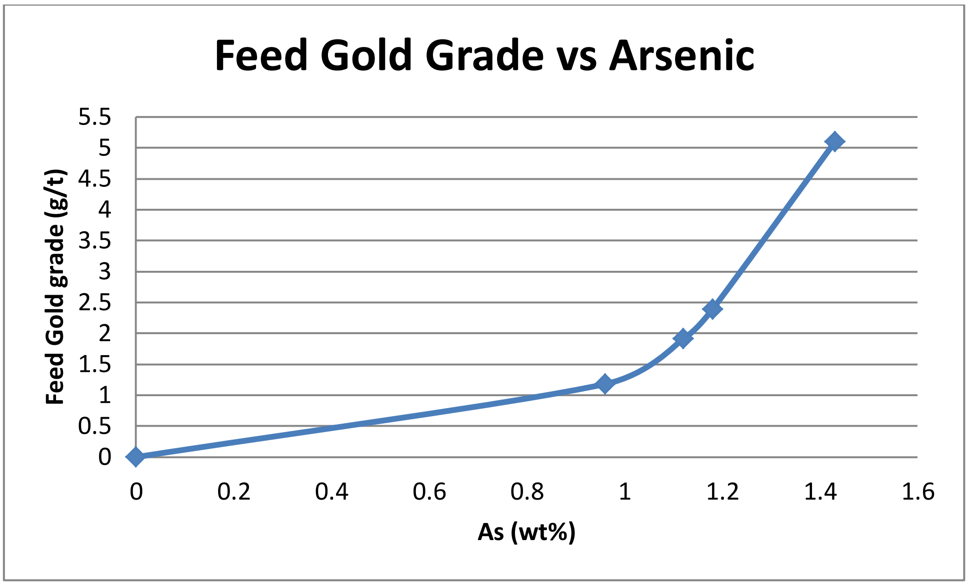


Figure 13-2: Feed Gold Grade vs. Arsenic Content

13.3.2.2. Minerology

An optical mineralogical examination and petrographic textural review of Jugan feed samples JUD 48E, 43E, 44W and 50W were conducted by hrltesting primarily to establish the degree of liberation of combined iron and iron arsenic sulphide components (Reference 1 & 2).

Apart from minor variations in mineral textural fabrics and thus style of preferred mineral associations presented by the particle populations the mineral assemblage is identical for all the Jugan ore zones.The bulk of the Jugan ore feeds comprises non sulphide gangue.

Mineralisation characteristics of the sedimentary rock hosted dissiminated Jugan gold deposit were reported by Kim Chee Goh (Reference) and concluded the following:

- Petrographic studies including pyrite etching yielded five pyrite types with different texture:
    - pyrite 1 – framboidal and aggregates,
    - pyrite 2a – porous and overgrown by other pyrite(s),
    - pyrite 2b – clean, euhedral shaped or thin bright lines,
    - pyrite 2c – 'busy' texture with internal fractures,
    - pyrite 3 – thin, subhedral to euhedral.
- Carbonate veinlets have chemical compositions of ankerite to ferroan dolomite. Calcite and siderite are also present. Clay minerals are dominated by illite with little evidence of kaolinite and dickite.
- Although clay minerals can be unequivocally identified under the microscope in this study, XRD analyses shows that illitic mica is the dominant mineral found up to 50 wt. % of the whole rock constituents. (*Table 13-9: Roughly Estimated Volumetric Modal Mineral Distribution in Bulk Jugan Feed Ores*). This mica is very fine-grained and poorly crystalline. Minor sphalerite was also found while a significant amount of rutile is found

in the sandstone samples by using the SEM. Gypsum also occurs in some of the shale samples but their origin is unknown.

Dominant x > 50 vol%	**Minor** 20 ≥ x ≥ 0.3 vol%	**Sparse** x<<1vol%
Silty Carbonaceous Shale Matrix nsg ≈ 50 Mosaic granular CO_3-rich qtz/carb matrix ≈ 32	Pyrite-FeS_2 ≈ 8 to 9 Arsenopyrite-FeAsS ≈5 to7	Marcasite- FeS_2 Rutile-TiO_2 Sphalerite-Zn(Fe)S Chalcopyrite-$CuFeS_2$ Tetrahedrite Series -$(Cu,Ag)_{10}(Zn,Fe)_2(Sb,As)_4S_{13}$ Gold ?-(Au)

Table 13-9: Roughly Estimated Volumetric Modal Mineral Distribution in Bulk Jugan Feed Ores

The population of sulphide bearing particles of various sieved particle fractions have been examined to determine a crude estimate of combined sulphide liberation (both single and locked sulphide entities). This is for a better appreciation to help improve concentration of sulphides for downstream gold extraction.

The sulphide liberation of the three Jugan ore sites for several chosen particle size ranges are tabled as follows:

Jugan 44 West

Size Fraction	*Sulphide Liberation*
-150 μm to +106 μm	*45%*
-106 μm to +75 μm	*52%*
-75 μm to +53 μm	*76%*

Jugan 43 East

Size Fraction	*Sulphide Liberation*
-150 μm to +106 μm	*22%*
-106 μm to +75 μm	*52%*
-75 μm to +53 μm	*38%*

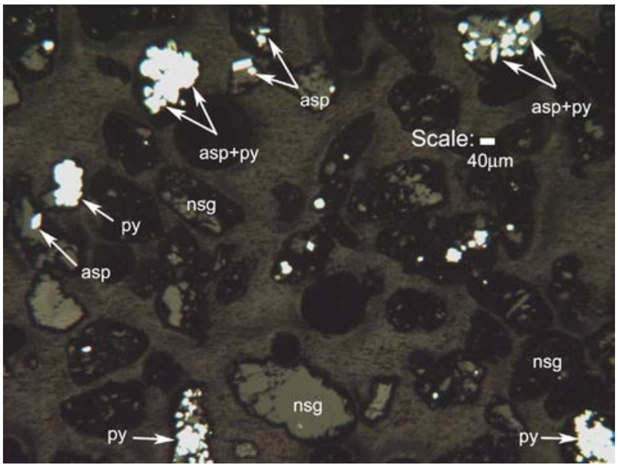
Jugan 50 West

Size Fraction	*Sulphide Liberation*
-150 μm to +106 μm	*17%*
-106 μm to +75 μm	*37%*
-75 μm to +53 μm	*63%*

Jugan 48 East

Size Fraction	*Sulphide Liberation*
+150 μm	*14%*
-150 μm to +125 μm	*23%*
-125 μm to +75 μm	*38%*

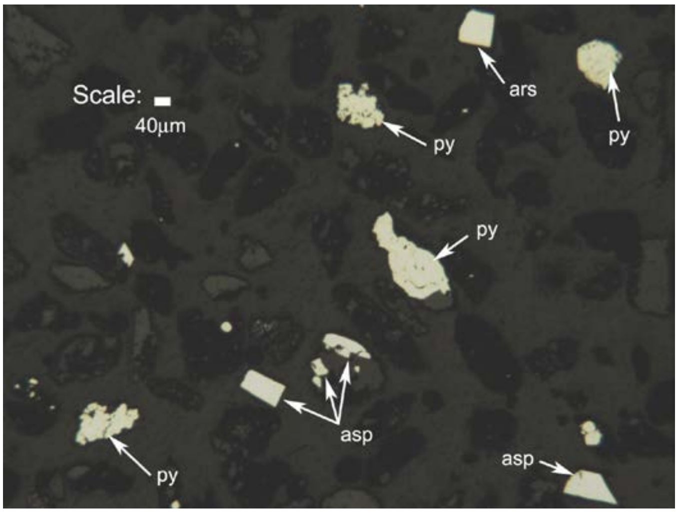
A visual comparison of the particle populations in the three size fractions for Jugan 48 East is illustrated in Photomicrographs 1, 11 and 16 below. (Reference 1)



py = pyrite   asp = arsenopyrite   asp+py =arsenopyrite + pyrite   nsg = non-sulphide gangue

Figure 13-3: Photomicrograph 1

This low magnification field of view of the +150 µm sieved fraction shows very little liberation of the sulphide fraction. The host rock matrix as depicted by the non-sulphide gangue (NSG) rich particles are visually subdivided on the basis of crude fabric differences to fine grained silty shale fabrics versus the coarser mosaic granular carbonate fabrics. There is a suggestion that the dark coloured fine grained silty shale fabrics have carbonaceous coatings similar to the Mt Isa silty shale members. At this low magnification exposure and under bright light conditions to highlight the NSG fabrics there is little contrast to distinguish pyrite from arsenopyrite excepting differences in their crystalline fabrics. Arsenopyrite characteristically shows rhombohedral crystal terminations or else lath shaped to prismatic outlines. Several such outlines are arrowed in the photomicrograph.

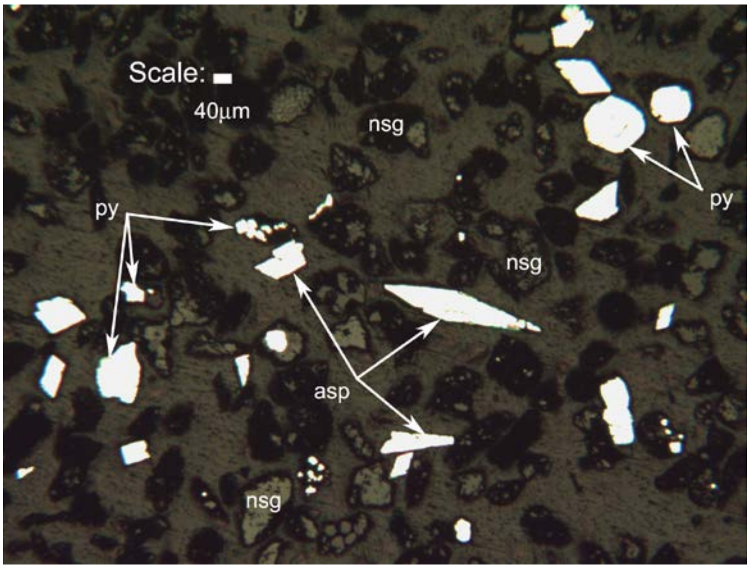


py = pyrite   asp = arsenopyrite   asp+py =arsenopyrite + pyrite   nsg = non-sulphide gangue

Figure 13-4: Photomicrograph 11

There would appear to be a slightly better liberation of the sulphide fraction here in this +125 µm sieved particle preparation by contrast to that demonstrated in the +150 µm sieved fraction illustrated in Photomicrograph 1.

Crude visual measurements suggest that here the liberation of sulphide particle population is 23 % as opposed to the 14 % for the +150 µm particle fraction. This does not take into account the actual yield of liberated pyrite since many of the sulphide bearing particles are low volume fraction sulphide bearing composites as opposed to the larger yield from liberated sulphide particles.



py = pyrite   asp = arsenopyrite   asp+py =arsenopyrite + pyrite   nsg = non-sulphide gangue

Figure 13-5: Photomicrograph 16

This photomicrograph is inserted to enable comparisons with the images of the +150 µm and +125 µm preparations (Photomicrographs 1 & 11) as a comparative indicator of the liberation characteristics of the sulphides in this +75 µm sieved fraction.

Once again crude measurements would suggest that liberation of the sulphide fraction either single mineral or liberated sulphide composites with respect to the rest of the sulphide bearing particle population is in the region of 38 %.

Although the mineralogy suggests that a grind size of 75 µm or finer will result in better liberation of sulphides, Besra has selected a target grind of 150 µm due to partial liberation with limited production of slimes. The company wants to minimize slimes production to optimize performance of the flotation process and the milling circuit has been designed with this in mind.

13.3.3. Comminution

Comminution testwork, carried out by SGS Lakefield Oretest is summarised in *Table 13-10: Comminution Test Results* (Reference 3), together with historical data and hrltesting data:

- The AI (abrasion index) value indicates very low abrasion characteristics; (see *Figure 13-6 - AI Ranked Against A.R. MacPherson Database of Abrasion Indices*)
- Both the BRWI (bond rod mill work index) and BBWI (bond ball mill work index) values indicate that the sample falls in the moderate hardness category; (see *Figure 13-7 - Jugan: BRWI vs. A.R. MacPherson Database* & *Figure 13-8 - Jugan: BBWI vs. A.R. MacPherson Database*)
- A single SPI test was conducted on a representative sub-sample of the master composite sample. The sample was prepared in accordance with the "Standard Crusher Test and SPI/Bond Sample Preparation Procedure". The SPI test measures the time required to grind 2 kg of sample from 80 % "minus" 12.7 mm to 80 % "minus" 1.70 mm in the SPI test mill. Quoted in minutes, SPI is a function of ore hardness as it applies to SAG milling. The sample was tested in accordance with the "Standard SPI Test Procedure". At 26.4 minutes, the sample tested falls in the soft category; (see *Figure 9 - Jugan SPI vs AR MacPherson Database*).

Test	Unit	Historical	SGS Lakefield	hrltesting
AI		0.015	0.012	
BBWI	kWh/t	11.3	13.2	12.6
BRWI	kWh/t	13.5	14.1	
SPI	minutes		26.4	

Table 13-10: Comminution Test Results

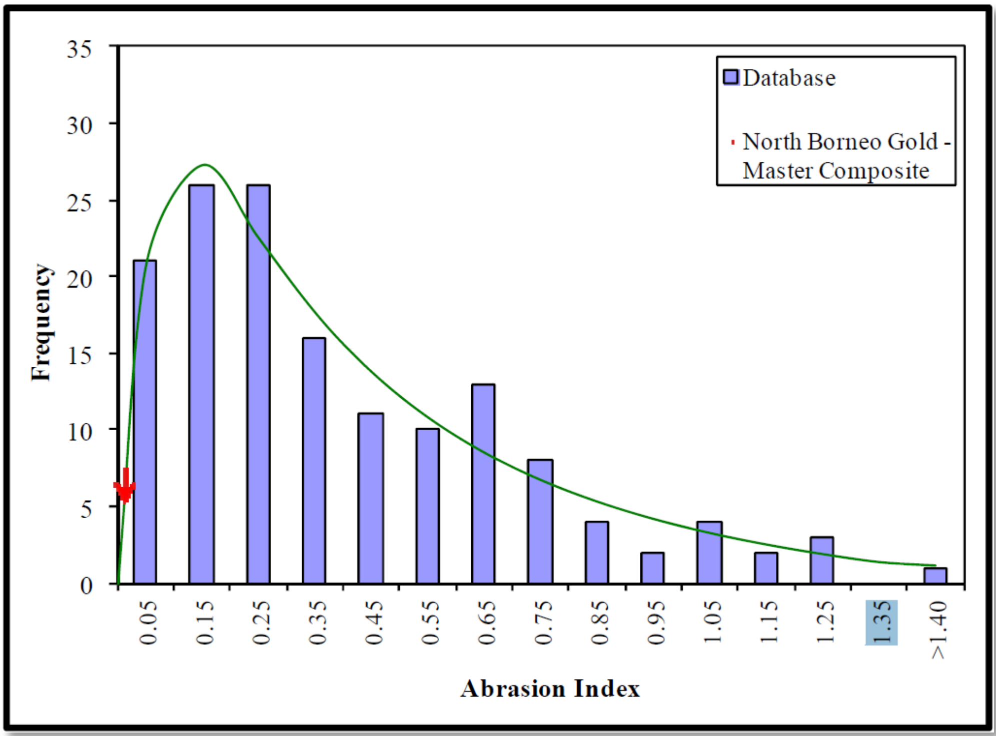


Figure 13-6 - AI Ranked Against A.R. MacPherson Database of Abrasion Indices

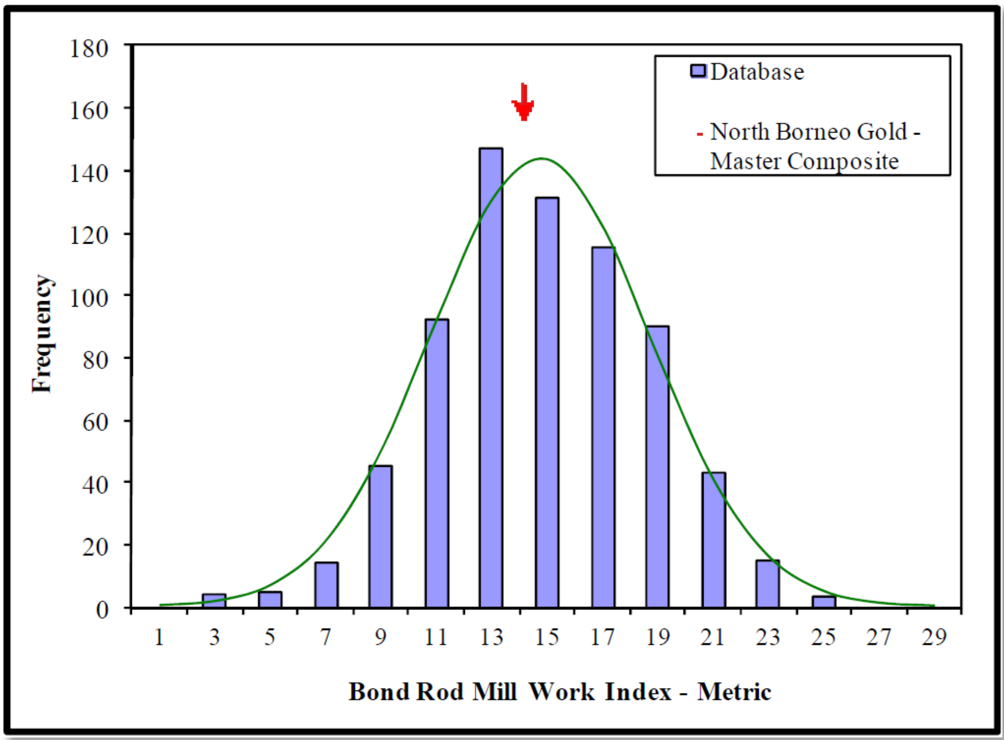


Figure 13-7 - Jugan: BRWI vs. A.R. MacPherson Database

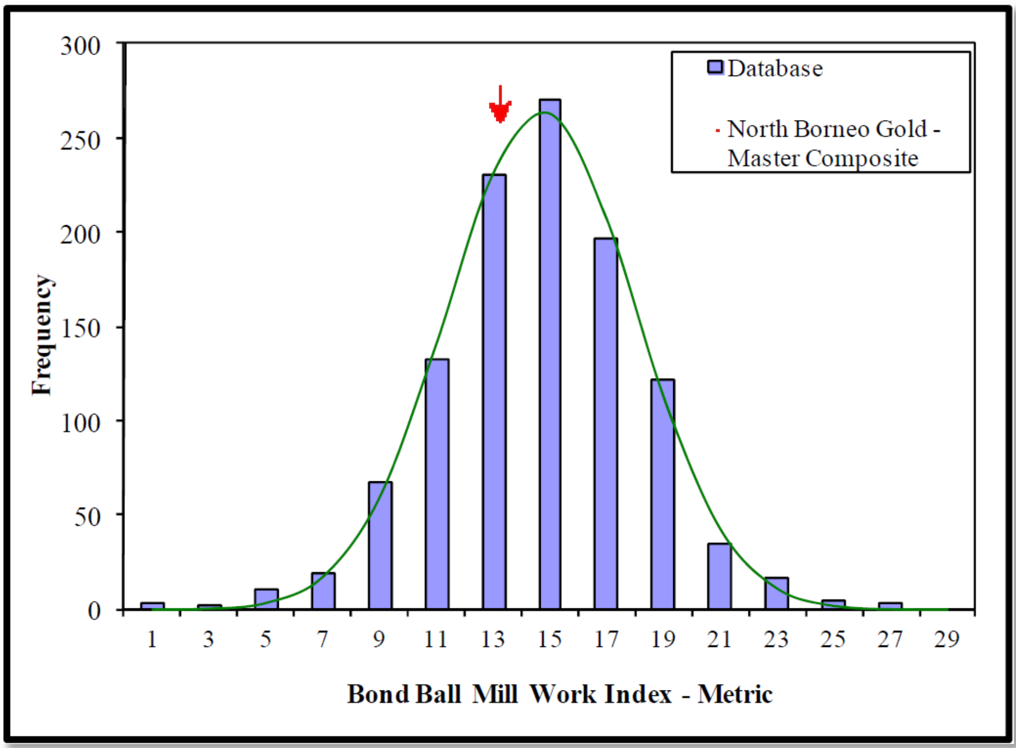


Figure 13-8 - Jugan: BBWI vs. A.R. MacPherson Database

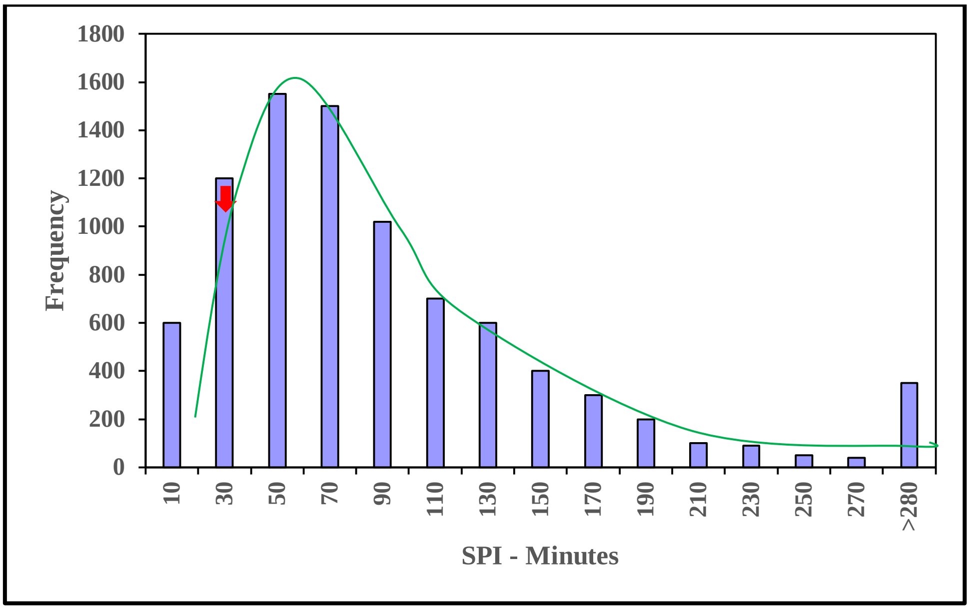


Figure 9 - Jugan SPI vs AR MacPherson Database

13.3.4. Flotation

13.3.4.1. Phase 1 – Rougher Flotation Results

For Phase 1, rougher flotation tests were carried out at SGS Lakefield Orestest and Maelgwyn Mineral Services.

SGS performed three rougher sighter batch flotation tests only on the composite Jugan oresample using an Essa laboratory flotation cell. (*Table 13-11: Jugan - Ore Head Assays* and *Table 13-12: Sighter Flotation Test Results*). The grind size (P_{80} = 75 µm) was nominated by the client for all tests. The following reagent scheme was used for each test:

- 100 g/t CuSO4 activator;
- 50 g/t AERO 407 (Test 1), AERO 6697 (Test 2) or AERO 3418A (Test 3) promoters added;
- 200 g/t PAX collector stage added;
- Polyfroth W22 added as required.

Sample ID	Au (g/t)	As (%)	Fe (%)	S_{tot} (%)	Ca (%)	K (%)	Mg (%)	Na (%)	Ni (ppm)	Co (ppm)	Sb (ppm)
Jugan 20-23	2.71	1.23	4.41	3	2.48	2.59	1.13	0.13	53	19	6 1

Table 13-11: Jugan - Ore Head Assays

Test No.	Combined Rougher Flotation Concentrate								
	Mass (%)	As		Au		Fe		S	
		Grade (%)	Dist'n (%)	Grade (g/t)	Dist'n (%)	Grade (%)	Dist'n (%)	Grade (%)	Dist'n (%)
Test 1	26.1	4.01	96.2	11.4	95.7	11.5	69.8	11.2	95.6
Test 2	27.1	4.15	96.0	11.2	95.3	11.2	69.0	10.6	95.4
Test 3	25.0	4.42	96.2	11.3	95.2	12.4	70.2	11.4	94.8

Table 13-12: Sighter Flotation Test Results

The rougher flotation showed a linear relationship between Au and As recovery consistent with the majority of the gold associated with arsenopyrite. Although in excess of 95 % of the gold can be recovered it is at the expense of a high mass pull. Slimes entrainment was observed from the onset of flotation and the flotation kinetics were very slow.

A bulk rougher flotation, under the conditions of Test 1, was performed to produce flotation concentrate for comparative pressure oxidation (POX) and ultra fine grinding and atmospheric oxidation (Albion process).

About 300 kg Jugan metallurgical drill core sample, grading 1.47 g/t Au, was dispatched to Maelgwyn Mineral Services in South Africa to compare standard cell rougher flotation and their G-Cell rougher flotation. The flow diagram of the G-cell set up is shown below.

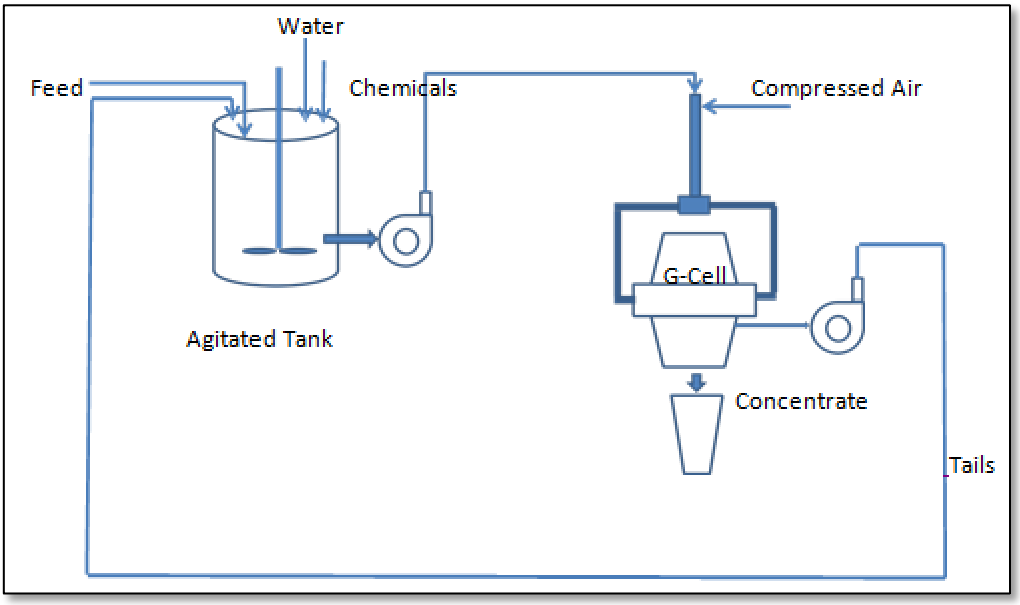


Figure 13-10 - Flow Diagram of the G-Cell

A 200 kilogram sample portion was milled to p_{80} 75 µm and placed into the feed tank and slurried with water to 20 % solids. The slurry was conditioned with the required flotation chemicals (shown below) and pumped through the G-cell to achieve flotation. The concentrate was removed continuously while the tails was continuously recycled. The recycling of the tails was continued to achieve two (2) theoretical passes of the material through the cell.

Results are shown in *Table 13-13: Conventional Pilot Flotation Results* and *Table 13-14: G-Cell Pilot Flotation Results* below. The bulk rougher concentrates were sent to SGS South Africa for BIOX amenability testing under the supervision of the Goldfields BIOX division. The flotation chemicals used are compatible with further BIOX oxidative treatment of the concentrate and were the same for the conventional flotation and G-Cell namely:

- 200g/t CuSO4
- 125g/t PAX
- 50g/t Senkol 294
- 300g/t PFD 100
- Senfroth 6005 as required

Product	Cum Time (min)	Mass (kg)	Mass (%)	Cum Mass %	Assays Au g/t	Assays S(t) %	Assays S^{2-} %	Cum Assays Au g/t	Cum Assays S(t) %	Cum Assays S^{2-} %	% Distribution Au	% Distribution S(t)	% Distribution S^{2-}	% Cum Distribution Au	% Cum Distribution S(t)	% Cum Distribution S^{2-}
Rougher concentrate 1	10	9.19	9.19	9.19	11.50	37.95	30.21	11.50	37.95	30.21	69.47	81.11	80.05	69.47	81.11	80.05
Rougher concentrate 2	20	2.98	2.98	12.17	6.20	9.32	8.23	10.20	30.94	24.83	12.14	6.46	7.07	81.62	87.57	87.12
Rougher concentrate 3	30	2.53	2.53	14.70	4.12	6.20	5.25	9.16	26.68	21.46	6.85	3.65	3.83	88.47	91.22	90.95
Rougher concentrate 4	40	1.88	1.88	16.58	3.12	6.78	5.15	8.47	24.43	19.61	3.86	2.96	2.79	92.32	94.18	93.75
Rougher Tails		83.42	83.42	100.00	0.14	0.30	0.26	1.52	4.30	3.47	7.68	5.82	6.25	100.00	100.00	100.00
Head (calc)		100.00	100.00		1.52	4.30	3.47				100.00	100.00	100.00			
Head (assayed)					1.47	3.78	2.39									

Table 13-13: Conventional Pilot Flotation Results

Product	Cell passes	Mass (kg)	Mass (%)	Assays Au g/t	Assays S(t) %	Assays S^{2-} %	% Distribution Au	% Distribution S(t)	% Distribution S^{2-}
Rougher concentrate	2	21.25	10.63	19.60	46.20	33.78	89.96	93.53	90.94
Rougher Tails		178.75	89.38	0.26	0.38	0.40	10.04	6.47	9.06
Head (calc)		200.00	100.00	2.31	5.25	3.95	100.00	100.00	100.00
Head (assayed)				1.47	3.78	2.39			

Table 13-14: G-Cell Pilot Flotation Results

Although the results of the G-Cell point to a vast improvement in flotation recovery versus mass pull, the calculated head gold grade for the G-Cell test was 57 % higher than the assayed head grade.

The flotation concentrates were combined and later reanalyzed at the SGS South Africa laboratories with a gold grade of 9.7 g/t. This suggests the G-Cell rougher concentrate head grade should have been 11.5 g/t with a recovery of only 84 % of the gold and a calculated head grade of 1.45 g/t. Therefore, we consider the above G-Cell test results inconclusive and it is likely that the pneumatic G-Cell offered no advantage over the conventional flotation cells under the conditions of the present testing.

13.3.4.2. Phase 2 – Flotation Optimization Test Results (Jugan 48E Master Composite)

Although gold recoveries in excess of 95 % can be realized in rougher flotation of Jugan ore it was imperative to maximize the gold grade of the flotation concentrate with minimum mass pull and at least 90 % gold recovery. The Phase 2 flotation optimization work was carried out in the hrltesting laboratories in Brisbane under the supervision of Core Process Engineering. Four (4) Jugan metallurgical drill core samples were dispatched to hrltesting, about 250 kg each. The samples were taken from different representative areas of the Jugan deposit to a depth of about 60 m and shown in *Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines)* above. JUDDH-48E was chosen as the master sample for the optimization work and JUDDH-43E, 44W and 50W were selected as the ore variability samples. Jugan 48E was selected over the other samples as it more closely resembles the anticipated plant feed grade over the Life-of-Mine (LOM).

The previous rougher flotation tests at SGS Oretest were conducted at a P_{80} of 75 µm, but it was felt that this grind was producing excessive amounts of fines which impeded on clean sulphide–gangue separation. In house tests (Reference) on portion of the Jugan metallurgical samples showed that optimum gold recovery was obtained at a P_{80} grind of 125μm, as shown in *Figure 13-10 - JUDDH-03 (4.15 g/t Au) - % Au Recovery vs. P80* and *Figure 13-11 - JUDDH-04 (1.07 g/t Au) - % Au Recovery vs. P80* below. The much lower gold recovery obtained for the JUDDH-03 at P_{80} of 150 μm is attributed to lack of PAX addition for the high grade ore as all tests were conducted with a constant PAX dose of 30 g/t for comparison.

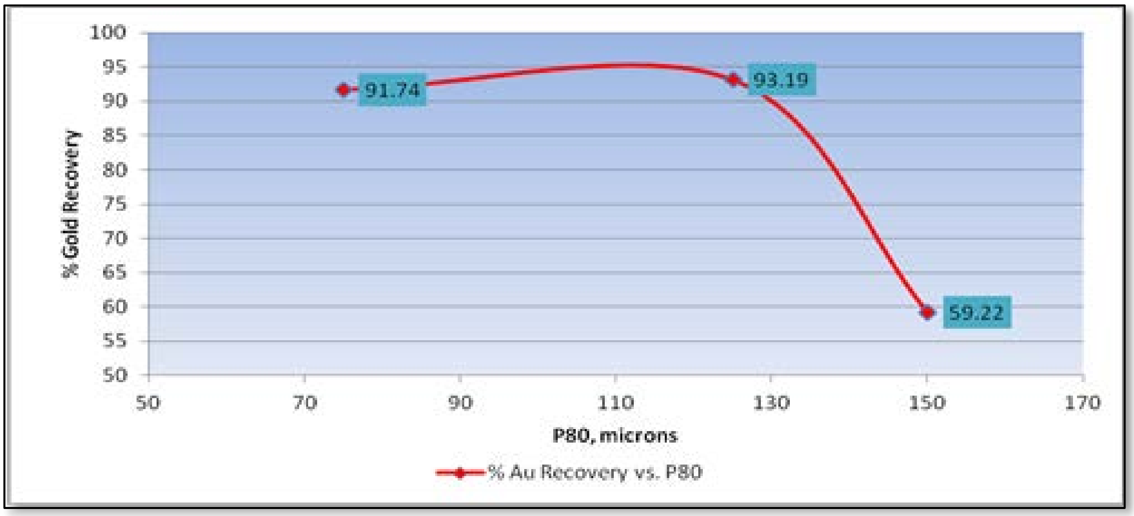


Figure 13-11 - JUDDH-03 (4.15 g/t Au) - % Au Recovery vs. P80

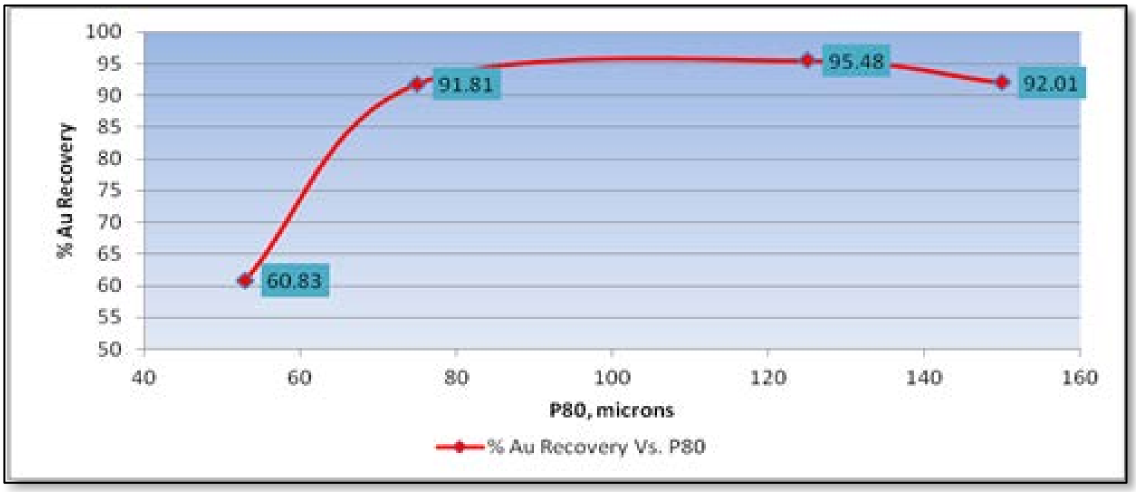


Figure 13-12 - JUDDH-04 (1.07 g/t Au) - % Au Recovery vs. P80

During the first stages of the testwork program for main Jugan 48E, problems of liquid-solid separation and production of colloidal slimes were observed. The initial suggested grind size of P_{80}=125 μm by Bau was used; however, a coarser grind of P_{80}=150 μm was tested due to the presence of significant slimes after milling.

The overall gold recoveries in the roughing and scavenging at the coarser grind size were comparable with relatively lesser slimes observed. Optical mineralogy for Jugan 48 East indicated that coarser than 125 μm grind size would be feasible to achieve good liberation of sought sulphide minerals, as shown in *Table 13-15: Comparison of Grind Size & Sulphide Liberation*. Hence, 150 μm grind size was tested showing similar flotation gold grade-recovery curve.

Jugan 48E Size Fraction	+150µm	(+125 to -150)µm	(+75 to -125)µm
Sulphide Liberation	14%	23%	38%

Table 13-15: Comparison of Grind Size & Sulphide Liberation

The flotation reagent schemes used were:

- CMC: 0 to 200 g/t
- $CuSO_4$: 200 g/t
- A 407: 50 to 75 g/t
- SIBX: 50 to 75 g/t
- MIBC-DF 250: as required

The standard rougher test gives a high gold and sulphur recovery. However, the mass recovery (17 %) is well above the minimum target 10 % target. In a first attempt to optimise the rougher performance, the use of dispersant addition during grinding (sodium carboxy-methyl-cellulose, CMC) and/or a pre-flotation step were examined.

The results are summarised in *Table 13-16: Rougher Tests Optimization (Jugan 48 E)*. With either option, the total mass recovery was reduced from 17 % to around 12 %. The pre-float stage recovers around 40 % of the organic carbon contained in the ore feed, removing around 3 % of the mass at a grade of 0.5 g/t Au (representing less than 1 % gold losses).

Test Number		Mass	Gold	Gold Grade	Sulphur	Sulphur
		%	Recovery %	g/t	Recovery %	%
Test 1	Standard	17.1	92.0	11.1	89.2	17.0
Test 2	With CMC to Mill	12.3	92.0	14.5	85.9	22.6
Test 3	With CMC to mill	14.3	92.0	11.8	88.0	18.1
Test 4	With Pre float	11.6	92.0	14.7	90.0	20.1
Test 5	Pre-float + CMC	15.7	92.0	11.0	89.9	16.0

Table 13-16: Rougher Tests Optimization (Jugan 48 E)

To move the initial POX test work programme forward, the production of a bulk concentrate using the Jugan 48E sample with previously established flotation conditions was done. These works were completed and the samples shipped to for testing in Perth. A summary of the flotation result for this bulk test is listed in *Table 13-17: Bulk (60 litre) Rougher Flotation Test (Jugan 48E)*.

Product	Weight	Cumulative	Gold Assay (ppm)		% Gold	Sulphur Assay (%)		% Sulphur
	(g)	Wt. (%)	Au	Cumulative	Recovery	%S	Cumulative	Recovery
Conc. 1	3,363	2.2	25.80	25.8	31.5	18.1	18.1	13.7
Conc. 2	6,540	6.6	15.40	18.9	68.0	31.1	26.7	59.6
Conc. 3	7,038	11.3	7.91	14.4	88.2	15.2	21.9	83.8
Conc. 4	5,130	14.7	2.15	11.5	92.2	3.92	17.7	88.3
Conc. 5	3,973	17.4	1.02	9.9	93.7	2.12	15.3	90.2

Product	Weight	Cumulative	Gold Assay (ppm)		% Gold	Sulphur Assay (%)		% Sulphur
	(g)	Wt. (%)	Au	Cumulative	Recovery	%S	Cumulative	Recovery
Tail	123,956		0.14			0.35		
TOTAL	150,000		1.84			2.95		

Table 13-17: Bulk (60 litre) Rougher Flotation Test (Jugan 48E)

A size by size analysis of the rougher concentrate was completed to help understand the mechanisms driving the high weight recovery to concentrate and also quantify the effect of de-sliming the rougher concentrate. A sub-sample of the rougher concentrate from the bulk test was screened at 38 microns, 20µm and 5µm. The mass and metal distribution is given in *Table 13-18: Size by Size Characterisation.*

Size	Gold		
Microns	Mass	Grade (g/t)	Distribution (%)
+ 38	40.9	16.0	68.4
+ 20	7.3	22.3	17
+ 5	4.9	18.6	9.4
- 5	47	1.1	5.2
Total	100	9.6	100

Table 13-18: Size by Size Characterisation

Almost half of the concentrate mass (47 % w/w) can be rejected at minus 5µm with around 5 % loss in total contained gold. As a result of the high mass rejection of largely barren clay materials to the minus 5µm slimes, the concentrate grade increases from 13.9 % sulphur to 24.0 % sulphur while the gold content jumps from 9.6 g/t to 17.1 g/t Au. The gold/sulphur ratio remains largely unchanged.

De-sliming provides a significant opportunity for improvement of process performance and will be targeted in future development testwork.

Given the large amount of ultra-fine slimes material reporting to the rougher concentrate in this instance, standard dilution cleaning is considered the most practical solution and was examined under a range of conditions (*Table 13-19: Rougher/Cleaner Test Results (Juan 48E)).*

Test Description	Overall Mass Recovery %	Overall Gold Recovery %	Overall Gold Grade g/t	Overall Sulphur Recovery %	Overall Sulphur wt %
Bench Scale Rougher/Cleaner	11.8	92.0	15.5	90.98	24.37
Bulk Rough/Cleaner	9.7	92.6	15.1	89.10	23.51
Bulk Rgh/Clr with Regrinding	7.8	89.3	17.6	88.18	28.85

Table 13-19: Rougher/Cleaner Test Results (Juan 48E)

Regrinding of the rougher concentrate, to around 80 % passing 30 µm prior to cleaning, resulted in the lowest mass recovered to concentrate. However, the total gold recovery was reduced by around 3 % to 89 %.

Cleaning the bulk rougher concentrate immediately without regrinding showed the most promise for lowering the overall mass recovery to concentrate to less than 10 % w/w while keeping the overall gold recovery about 92%. Given this, flotation circuit design is based on a gold upgrade factor of 9 in the rougher circuit and 2 in the cleaner stage. Verification of the design criteria will be further supported through ongoing testwork.

The main process criteria that drive the flotation flowsheet for the Bau Jugan 48E are the following:

- Primary Mill discharge grind size of P_{80}=150 µm for controlled slimes production for the rougher and scavenger circuits;
- Use of 200 to 250 grams CMC per tonne of ROM ore in the Mill feed during grinding should be adopted as a standard. This addition of CMC was observed to render the rougher floats with cleaner froth and less slimes and gangue entrainment during flotation. Hence, lower mass recoveries with better gold selectivity were achieved.
- Conditioning of the Rougher Feed with 200 to 250 grams of $CuSO_4$ per tonne feed for at least 15 minutes.
- It is important to note, that the use of 75 grams A-407 gold collector per tonne of feed should be added in the Conditioner ahead of the Roughers. This step will maximise selectivity of gold-bearing minerals and minimise slimes entrainment during the early stages of flotation, hence, mass recovery is lower with higher gold grade.
- The use of SIBX, instead of PAX, should be adopted in the Scavenger circuit, to achieve better selectivity of sulphides from the slimes-laden slurry, hence, reduce entrainment of non-sulphide gangue slimes. This should be added after the rougher cells, ahead of the scavengers, to give time for quick conditioning. SIBX is added as a ‘kicker collector’ only, to recover the remainder of liberated and partially liberated gold-bearing sulphides, at a dosage of 50 grams per tonne.
- Cleaning test by hrltesting indicated that re-grinding inhibits overall cleaning selectivity and recovery of gold-bearing minerals from the bulk concentrates. However, this still needs to be confirmed with additional testing.
- Frother in the form of DF-250 + MIBC mixture (75:25 ratio) is recommended and should be stage-added in moderation across the float banks. This frother addition strategy was observed to lessen entrainment of unwanted very fine slimes early on in the rougher/scavenger circuits.

A simplified configuration of the flowsheet used by hrltesting for the Bau Gold Concentrator is presented below in *Figure 13-12: Simplified Bau Flotation Circuit.*

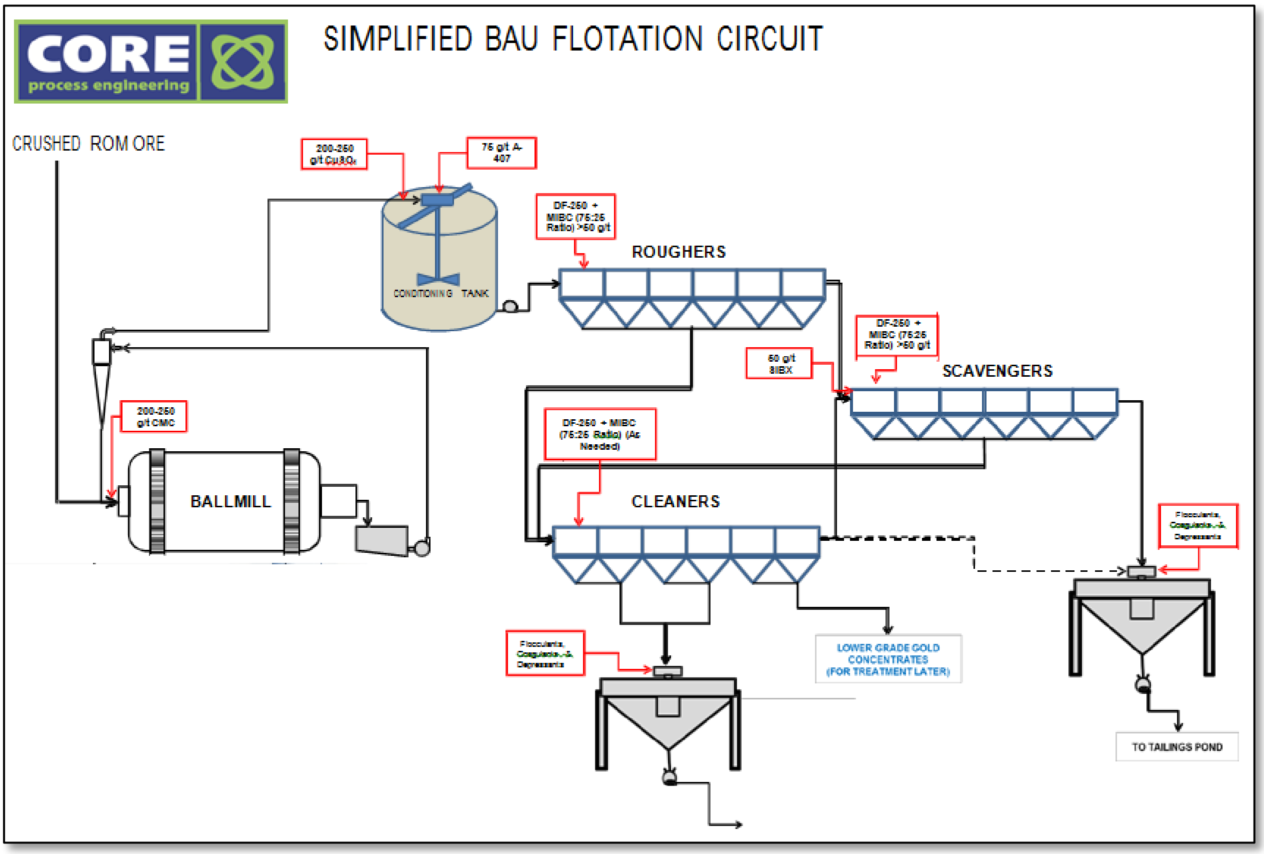


Figure 13-13: Simplified Bau Flotation Circuit

13.3.4.3. Phase 2 – Variability Testwork on 3 Orebody Samples (Jugan 44W, 50W & 43E)

For the treatment of the remaining three (3) orebody samples Jugan 44W, 50W and 43E (variability samples taken from different locations as shown in *Figure 13-1 - Jugan: Metallurgical Drillhole Locations (incl. Projected Ore Outlines)* above) testwork program the focus was on the production of the cleanest possible concentrates at acceptable gold grade and recovery - for consideration of selling the gold concentrates. Hence, roughing and cleaning tests for the variability samples were carried out using the best conditions achieved in the Jugan 48-East composite sample, as summarised in the previous section.

The scope of work included the following:

1. Grinding tests: Bond Work Indices;
2. Mineralogy (polished section, optical);
3. Flotation optimization tests with particular emphasis on the cleaner stages;
4. Full analysis of the optimum flotation concentrates (full ICP scan, Hg, Cd etc.);
5. Production of bulk concentrates, and then cleaner concentrates;
6. Pressure filtering tests on the cleaner concentrates produced.

Based on the standard rougher flotation conditions (with CMC addition to grinding) a series of flotation tests were carried out on the other samples from the Jugan deposit, being Jugan 43E, Jugan 44W and Jugan 50W.

The results from this initial round of characterisation testing where rougher gold recovery was maximised and reported at a 92 % recovery rate for comparison, are summarised in *Table 13-20: Variability Jugan Ore Samples – Standard Rougher Test Results* below.

Test Sample	Mass %	Gold Recovery %	Gold Grade g/t	Sulphur Recovery %	Sulphur wt %
Jugan 48 East	12.3	92.0	14.5	85.9	22.6
Jugan 43 East	20.0	92.0	5.2	91.4	13.2
Jugan 44 West	30.9	92.0	14.7	91.7	8.5
Jugan 50 West	28.0	92.0	9.7	90.2	11.7

Table 13-20: Variability Jugan Ore Samples – Standard Rougher Test Results

All of the other ore types show significantly higher mass recovery relative to the master composite from Jugan 48E at a the target gold recovery of 92 %. Whilst gold and sulphur recoveries are relatively consistent, the concentrate sulphur grade is highly variable ranging from 8.5 % to 22.6 % across the suite of ore samples tested due to variable slimes entrainment to the concentrate.

The chart below in *Figure 13-13: Rougher Flotation Results* summarises the flotation responses of the four (4) Jugan orebodies in the rougher and scavenger circuits using the best flotation conditions and reagent scheme developed for the Jugan 48E main sample.

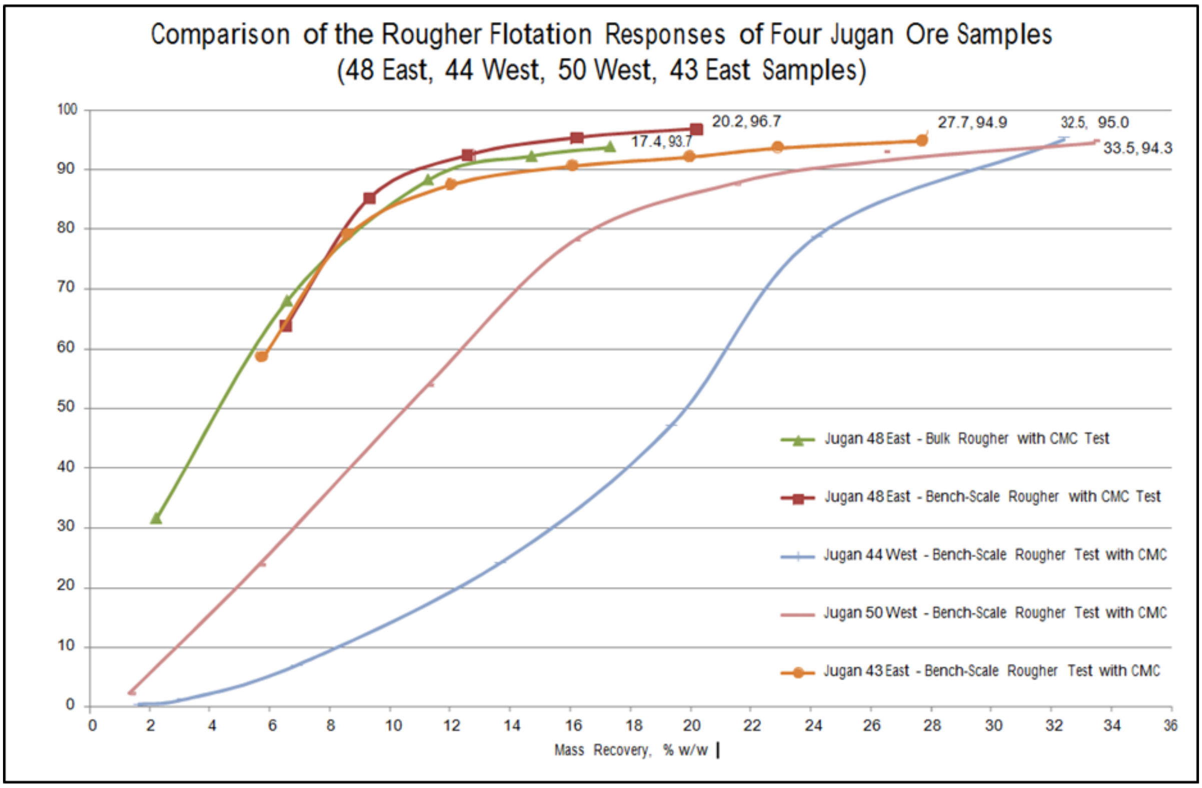


Figure 13-14: Rougher Flotation Results

Clearly, both the West orebodies from 44W and 50W Jugan areas showed higher amounts of gangue slimes entrainment during the bench-scale roughing and scavenging stages. Such resulted to mass pull of 32.5 % and 33.5 % w/w with overall gold recoveries of 94.3 % and 95 %, respectively.

Besra is currently investigating desliming of the flotation feed prior to rougher flotation not only in lower mass pull but more importantly to increase the gold upgrading factor with the aim of producing a high grade gold concentrate. Gravity separation of the bulk of the slimes is found to be the best option. This can be achieved by cycloning the flotation feed on plant scale and using Knelsons or tables on lab scale.

On the other hand, both the East orebodies from 48E and 43E Jugan areas displayed similar responses during bench-scale roughing and scavenging flotation. The lower grade 43E sample recovered higher mass at 27.7 % w/w with slightly lower overall gold recovery of 94.9 %, compared to the 48E sample with 20.2 % w/w mass pull at 96.7 % overall gold recovery.

Bulk roughing and scavenging response of the 48E sample clearly duplicated the response of the bench-scale test, with slightly slower kinetics which is expected using bigger size 60 litre pilot flotation cells.

Below is the chart (*Figure 13-14: Bench Scale Cleaner Flotation Results*) that summarises the bench scale flotation responses of the four (4) Jugan ore samples in the cleaner circuit using the best "dilution cleaning" conditions and reagent scheme employed and developed for the Jugan 48 East rougher/scavenger concentrates. The results are also summarised in *Table 13-21*

- *Variability Jugan Ore Samples – Bench Cleaner Test Results* below at the 90 % gold recovery rate for comparison.

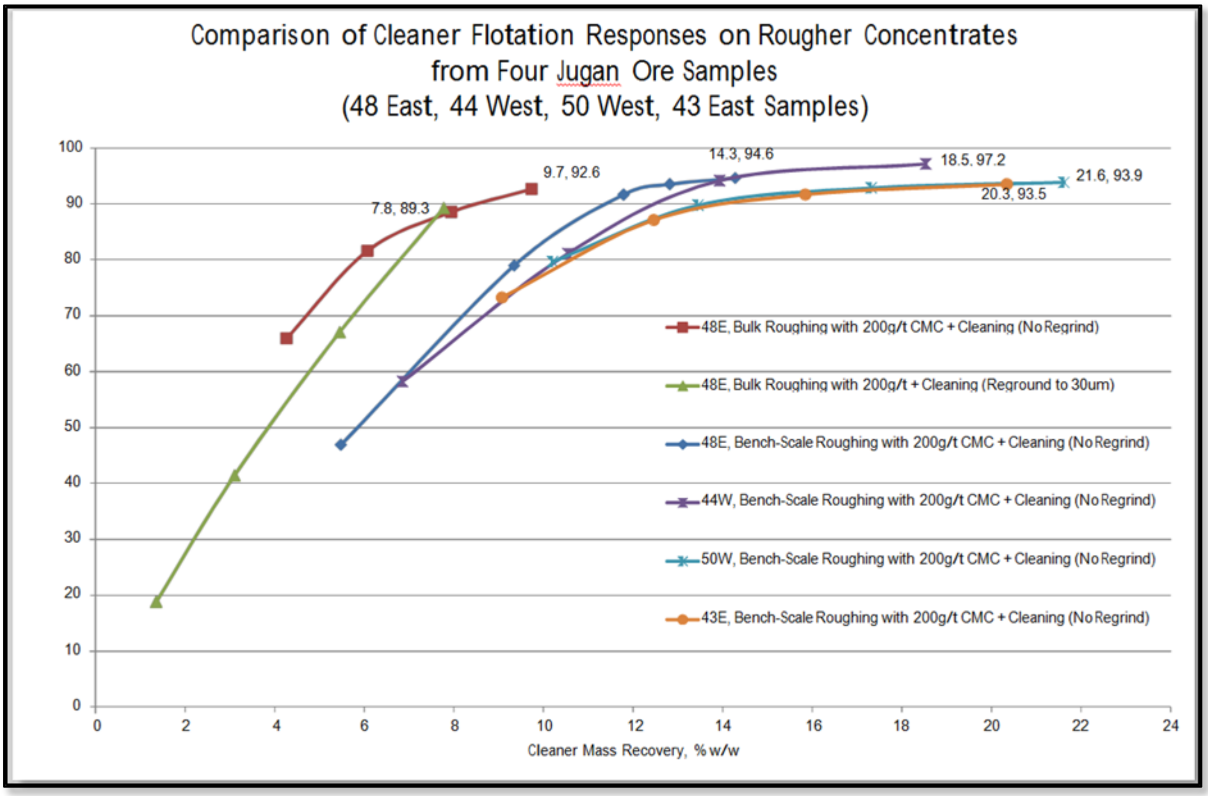


Figure 13-15: Bench Scale Cleaner Flotation Results

Test Sample	Mass %	Gold Recovery %	Gold Grade g/t	Sulphur Recovery %	Sulphur wt %
Jugan 48 East	11.5	90.0	15.4	89.6	24.0
Jugan 43 East	14.6	90.0	7.4	88.4	17.5
Jugan 44 West	12.8	90.0	35.9	88.9	20.9
Jugan 50 West	13.5	90.0	17.5	87.4	20.0

Table 13-21 - Variability Jugan Ore Samples – Bench Cleaner Test Results

It is interesting to note that, broadly, all of the Jugan orebody samples demonstrated similar trends of the cleaner mass recovery vs. gold recovery curves.

However, the Jugan 48E main sample showed the best in gangue rejection among the four (4) orebody samples with only 11.5 % w/w mass pull with 90% of the total gold recovered in the combined cleaner concentrates. Jugan 44 West sample also demonstrated good gangue rejection which is second to Jugan 48 East ore sample, with 12.8 % w/w mass and 90 % of the total gold recovered to the combined cleaner concentrates. The Jugan 50 West and Jugan 43 East had poorer gangue rejection at 13.5 % and 14.6 % mass pull at 90 % of total gold recovered, respectively.

The mineralogical composition of the Jugan 44 West sample at 36 g/t Au is shown in *Table 13-22 - Mineralogical Composition of the Jugan 44 West Cleaner Concentrate* below. The weight percentage of sulphides is about 67%.

Sample		Clnr Concentrate
Mineral Mass(%)	Gold	n.d
	Pyrite	16.24
	Pyrite_As	26.80
	Pyrrhotite	0.00
	Arsenopyrite	24.32
	Galena	0.00
	Bournonite	0.01
	Sphalerite	0.05
	Chalcopyrite	0.02
	Tetrahedrite	0.04
	Chalcostibite	0.00
	Quartz	5.67
	Plagioclase	0.02
	K-Feldspar	5.75
	Muscovite	16.61
	Biotite	0.29
	Clays	0.41
	Calcite	0.00
	Ankerite/Dolomite	3.02
	Siderite/Magnesite	0.07
	Accessories	0.06
	Ti & FeTi Oxides	0.48
	Fe Oxides/Hydroxides	0.01
	Other Minerals	0.13
	Total	100.00

n. d. - not detected

Table 13-22 - Mineralogical Composition of the Jugan 44 West Cleaner Concentrate

Also shown in *Figure 13-14: Bench Scale Cleaner Flotation Results* above, the bulk roughing and scavenging response of the 48E sample clearly demonstrated that regrinding the bulk concentrates before "dilution cleaning" affected selectivity and kinetics in the cleaner circuit.

However, it is clear that using bigger size 60 litre pilot flotation cells in the bulk dilution cleaning displayed better selectivity with good overall gold recovery. This indicates that selectivity will improve further in the concentrator plant scale due to improved energy and dilution factors.

Results of the bulk cleaner tests on the four (4) variability Jugan feeds are summarized in *Table 13-23 - Variability Jugan Ore Samples – Bulk Cleaner Test Results* below at the 90% gold recovery rate in the cleaner concentrates, for comparison.

Results of the bench scale and bulk rougher-cleaner flotation tests for the four (4) Jugan feeds are shown in *Figure 13-15: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 43E, 1.1* *g/t Au), Figure 13-16: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 48E, 2 g/t Au), Figure 13-17: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 50W, 2.67 g/t Au)* and *Figure 13-18: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 44W, 5.5 g/t Au)* below. Clearly better performance is observed with the bulk rougher-cleaner tests using a 60 litre flotation cell. About 90 % of the gold is recovered with about 10 % mass pull in all cases for the larger scale tests.

Test Sample	Mass %	Gold Recovery %	Gold Grade g/t	Sulphur Recovery %	Sulphur wt %
Jugan 48 East	9.6	90.0	18.0	86.0	25.3
Jugan 43 East	11.5	90.0	10.4	87.5	22.5
Jugan 44 West	10.2	90.0	50.4	88.2	28.1
Jugan 50 West	11.9	90.0	20.5	87.1	21.7

Table 13-23 - Variability Jugan Ore Samples – Bulk Cleaner Test Results

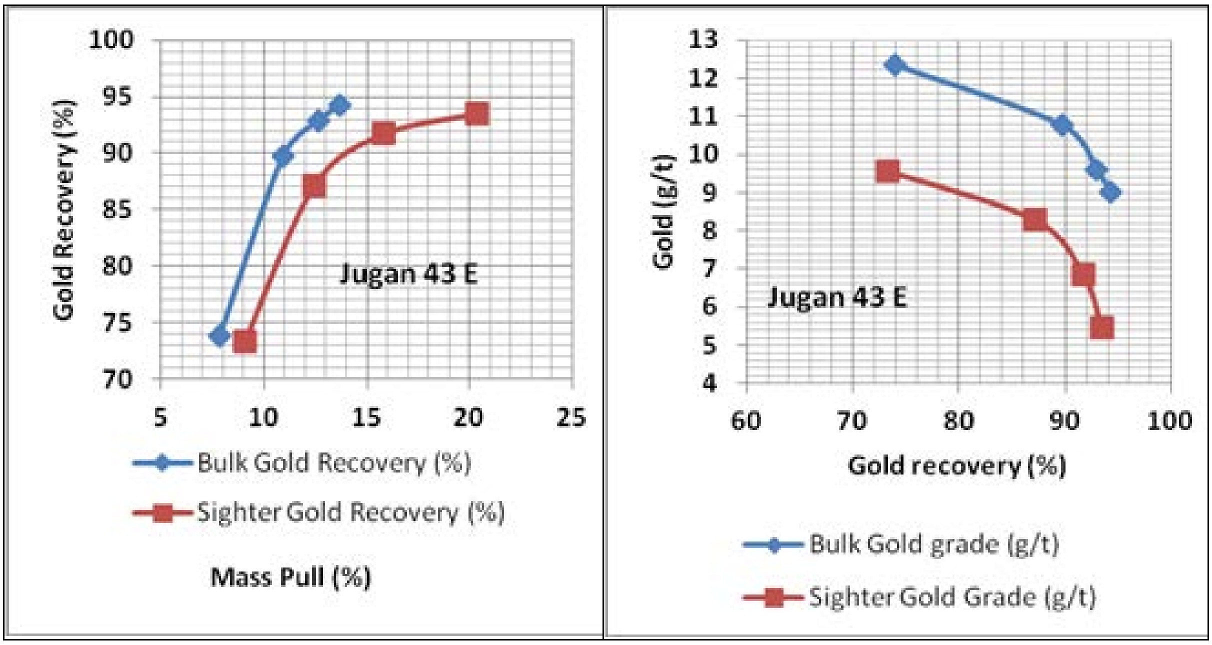


Figure 13-16: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 43E, 1.1 g/t Au)

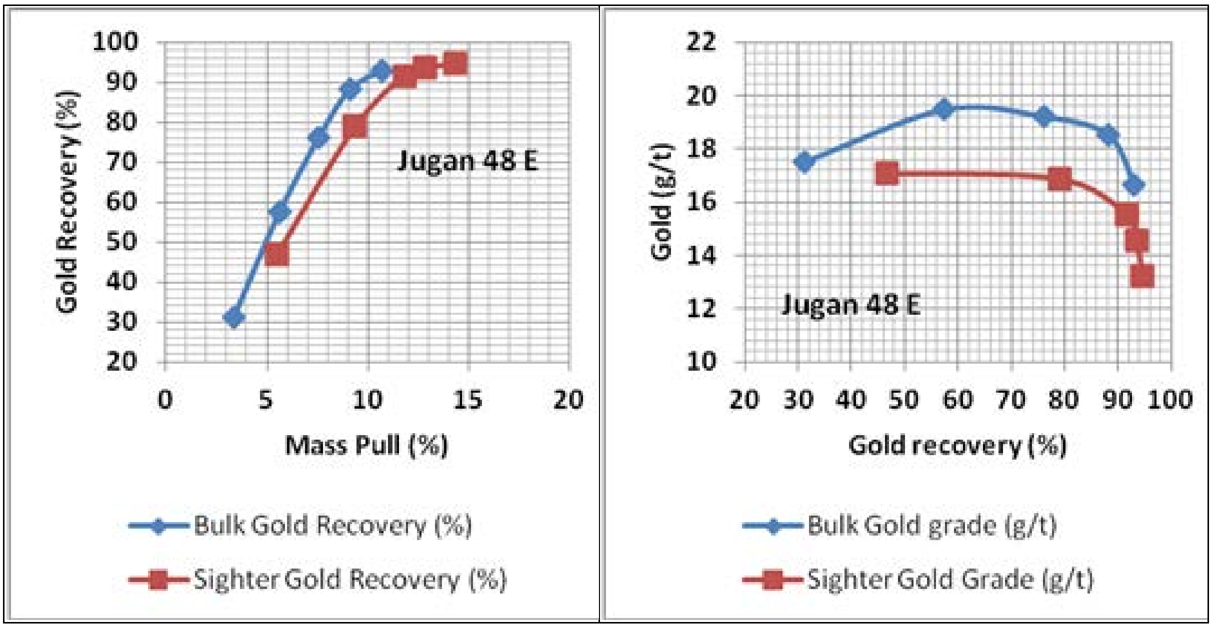


Figure 13-17: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 48E, 2 g/t Au)

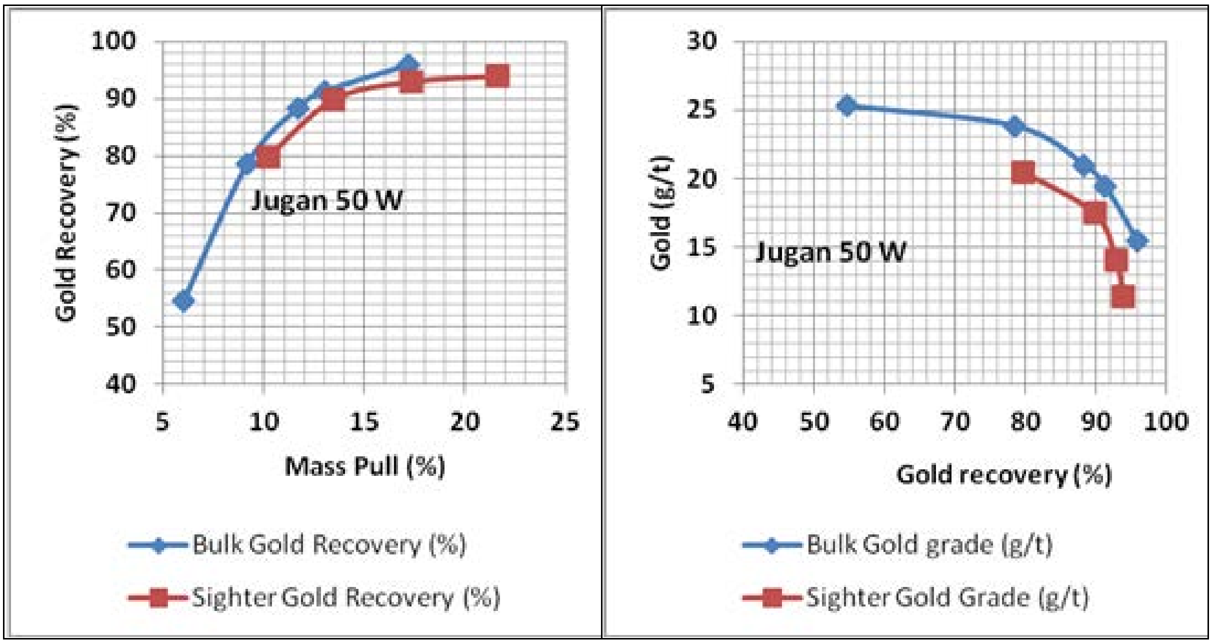


Figure 13-18: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 50W, 2.67 g/t Au)

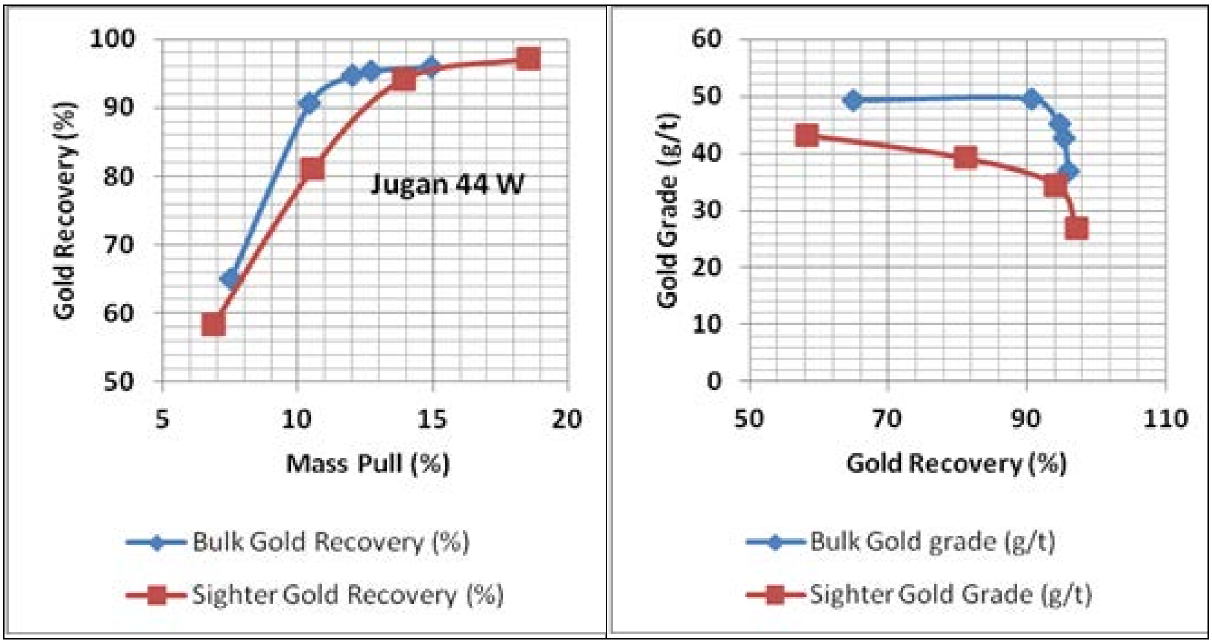


Figure 13-19: Bench Scale & Bulk Rougher-Cleaner Flotation Results (Jugan 44W, 5.5 g/t Au)

13.3.5. Pressure Oxidation (POX)

A representative sub-sample of the bulk flotation concentrate produced by SGS was subjected to batch POX testwork.

The POX test conditions are given in *Table 13-24: POX Test Conditions* and the POX results are summarised in *Table 13-25: Summary of POX Test 1 Results (180 °C)*, *Table 13-26: Summary of POX Test 2 Results (200 °C)* and *Table 13-27: Summary of POX Test 3 Results (220 °C)* for tests 1, 2 and 3 respectively.

Test Number	Pulp Density (%)	Temperature (°C)	Duration (min)	Total Pressure (kPa)	Oxygen O.P (kPa)
1	15.0	180	180	1,400	500
2	15.0	200	120	2,150	700
3	15.0	220	120	3,000	800

Table 13-24: POX Test Conditions

Sample (mins)	ORP[1] (mV)	Free Acid (g/l)	Elemental Dissolution (%)			Sulphide Oxidation (%)
			Fe	As	Sb	
15	433	13.6	7.24	7.24	0.47	51.3
30	456	30.1	18.7	11.6	1.37	65.0
60	519	49.0	45.8	11.9	4.11	96.8
90	549	52.4	49.9	11.7	3.70	96.9
180	588	52.4	50.6	13.1	2.88	96.4

Table 13-25: Summary of POX Test 1 Results (180 °C)

Sample (mins)	ORP[1] (mV)	Free Acid (g/l)	Elemental Dissolution (%)			Sulphide Oxidation (%)
			Fe	As	Sb	
15	475	35.4	24.7	0.00	0.00	74.6
30	538	49.5	48.1	16.4	8.75	80.9
60	593	53.4	34.9	13.7	3.36	82.3
90	595	51.5	26.3	12.5	1.82	86.4
120	592	50.5	23.7	11.5	1.53	90.0

Table 13-26: Summary of POX Test 2 Results (200 °C)

Sample (mins)	ORP[1] (mV)	Free Acid (g/l)	Elemental Dissolution (%)			Sulphide Oxidation (%)
			Fe	As	Sb	
15	575	52.4	26.9	0.00	0.00	70.8
30	584	51.5	21.1	12.1	2.99	71.3
60	593	50.0	16.6	11.5	1.70	73.4
90	596	49.5	12.8	11.2	1.18	77.5
120	607	47.6	12.8	11.8	1.09	78.7

Table 13-27: Summary of POX Test 3 Results (220 °C)

Note: (1) Oxidation-Reduction Potential (ORP) measured with a Pt vs Ag/AgCl electrode.

After removal of a sub-sample for chemical analysis, each POX residue was neutralised using lime. Summarised POX residue lime neutralisation results are given in *Figure 13-19: POX Residue Lime Neutralisation*.

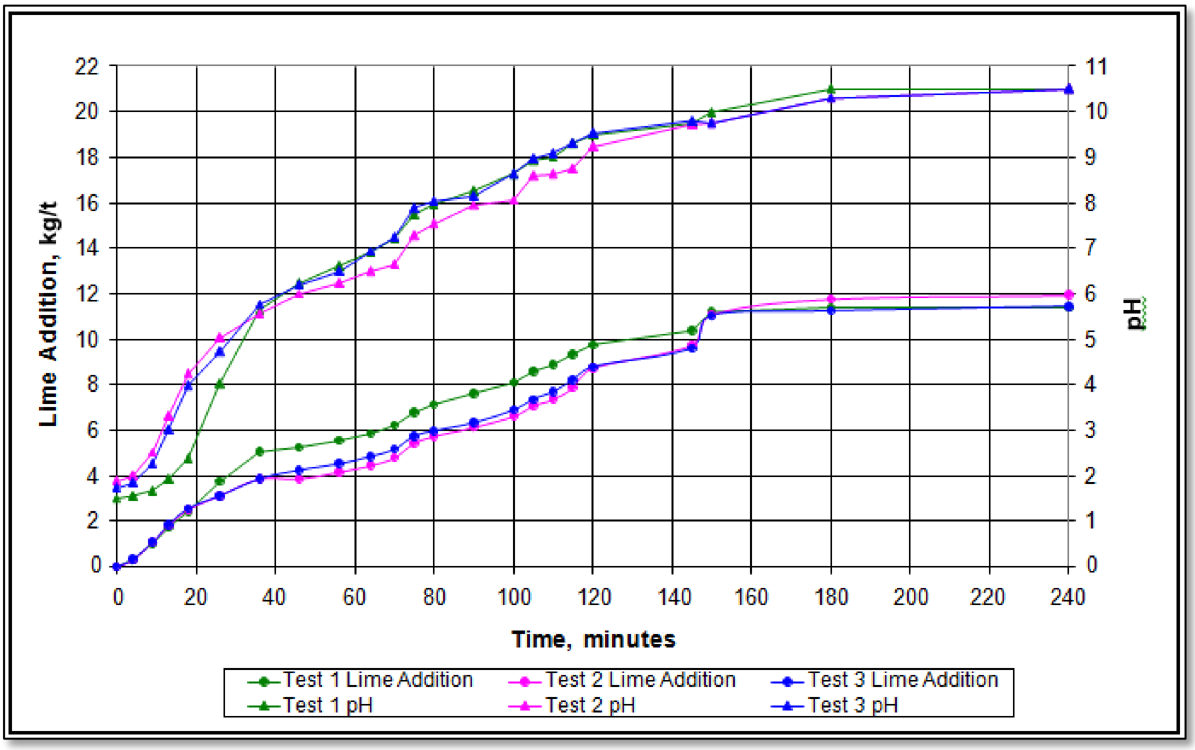


Figure 13-20: POX Residue Lime Neutralisation

13.3.5.1. Bottle Roll Cyanidation

After removal of a sub-sample for chemical analysis, each POX residue was neutralised with lime, and subjected to bottle roll cyanide leaching. The gold extraction realised with cyanide leaching of each POX residue are presented in *Table 13-28: Bottle Roll Leach Gold Extraction Results*.

Leach Time (hrs)	Au Extraction (%)		
	Test 1 (POX Temp. 180 °C)	Test 2 (POX Temp. 200 °C)	Test 3 (POX Temp. 220 °C)
2	75.3	86.8	77.3
4	90.4	89.7	84.3
8	93.4	92.3	83.4
12	95.7	94.3	83.2
24	97.2	94.1	83.3
48	97.9	93.1	83.1

Table 13-28: Bottle Roll Leach Gold Extraction Results

The gold dissolution profile for the samples over the 48 hour leaching period is illustrated in *Figure 13-20: Gold Dissolution vs. Leach Time* overleaf.

Figure 13-21: Gold Dissolution vs. Leach Time

The reagent consumption for each test is given in *Table 13-29: Bottle Roll Leach – Reagent Consumption.*

Test No.	POX Temp. (°C)	Reagent Consumption (kg/t)	
		NaCN	Lime
1	180	6.00	0.29
2	200	7.13	5.80
3	220	19.4	22.1

Table 13-29: Bottle Roll Leach – Reagent Consumption

POX and cyanidation testwork outcomes are:

- POX treatment of the flotation concentrates at 180 °C, resulted in 96.8 % sulphide oxidation within 60 minutes (Test 1). Increasing the temperature to 200 °C (Test 2) and 220 °C (Test 3), resulted in sulphide oxidations of 82.3 % and 73.4 % after 60 minutes respectively;
- The free acid concentrations obtained from the POX treatment ranged between 13.6 g/l and 52.4 g/l for Test 1, 35.4 g/l and 50.5 g/l for Test 2, and 52.4 g/l and 47.6 g/l for Test 3;
- Neutralisation of the POX residues indicated moderate lime requirement (11.4 kg/t to 11.9 kg/t) to achieve a final pH of ~10.5;
- Cyanide leach kinetics was rapid for all three tests, with gold recovery ranging between 84.3 % and 90.4 % after four (4) hours;

- Cyanidation results indicate that the gold recovery decreased with increased POX temperature;
- Cyanidation of the POX residue generated at 180 °C resulted in the highest gold recovery of 97.9 % and lowest reagent consumption compared to Test 2 and Test 3;
- Difficulty was experienced in maintaining a pH of greater than 10 in the cyanide leach conducted on the POX product generated at 220 °C. This residue also consumed significantly more lime (22.1 kg/t).
- In summary the POX process shows the potential for recovery of 98 % of the gold from the flotation concentrate with a NaCN consumption of 6 kg NaCN per tonne of concentrate.

13.3.6. Biological Oxidation (BIOX)

The following flotation concentrate sample was received by the Goldfields BioMet Department (now Biomin) to complete the BIOX® test program. The sample was prepared at the Maelgwyn laboratories from Jugan ore grading 1.46 g/t Au.

Sample	Approx. Wet Mass (kg)
Bau Lead Sample	1.5
Bau Property Concentrate	34.4

Table 13-30: BIOX Sample for BIOX Testing

On receipt, the BAU Property Conc was dried at 600 °C, crushed through an 850 µm screen to remove the lumps, blended and representative samples were split out for chemical analysis, particle size analysis and batch amenability testwork.

The results of the chemical analyses of the BAU Property Concentrate are given in the *Table 13-31: Property Concentrate Chemical Analysis Results* below.

Analysis	Unit	Concentration
Au	ppm	9.7
Hg	ppm	1.15
Sb	ppm	267
$Fe_{(T)}$	%	18.6
Si	%	16.5
As	%	5.81
$S_{(T)}$	%	19.0
S^{2-}	%	18.5
S^{o}	%	<0.5

Analysis	Unit	Concentration
$C_{(T)}$	%	1.83
$C_{(org)}$	%	0.64
CO_3	%	6.08
SG		4.0

Table 13-31: Property Concentrate Chemical Analysis Results

A BIOX® bacterial culture was adapted to the BAU Property Concentrate. The adaptation cultures were used to prepare inoculum for the various stages of the testwork. The BAU Property Concentrate is amenable to BIOX® treatment. This is proved by the following:

- The progress of bacterial culture adaptation and inoculum build up tests on the BAU Property;
- The ferrous ion of the batch amenability tests started to decrease from day 1. The ferric ion concentration increased progressively;
- The maximum extent of sulphide sulphur oxidation achieved in the BAU Property Concentrate was 98.7 %. The corresponding gold dissolution was 92.4 %. These were achieved after a bio-oxidation period of 19 days.

The progress of Bau Property concentrate batch tests is illustrated in *Figure 13-21 - Progress of the BAU Property Concentrate Batch Amenability Tests* below.

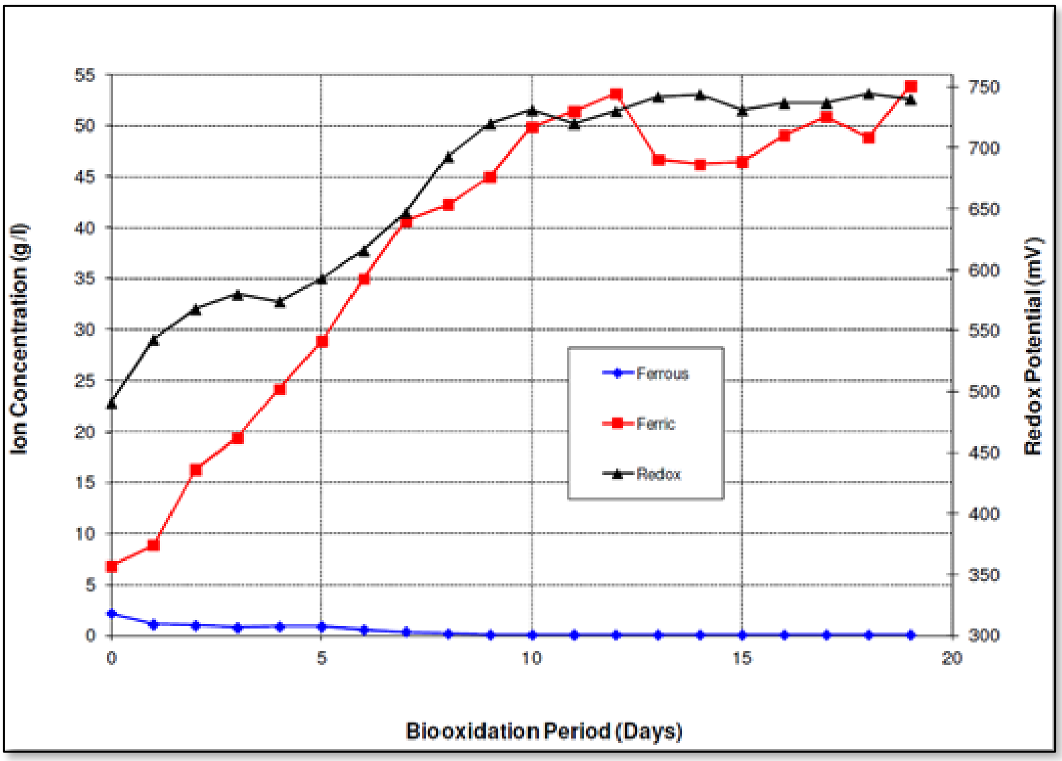


Figure 13-22 - Progress of the BAU Property Concentrate Batch Amenability Tests

The ferrous ion of the batch amenability tests started to decrease from day 1. The ferric ion concentration increased progressively. Decreases observed during biooxidation in the ferric ion concentrations of the tests could possibly been attributed due to the precipitation of iron salts (mainly as jarosite).This, however, does not necessarily indicate that dissolution of the iron sulphides was complete. Ferrous and ferric ion concentrations are used strictly for monitoring purposes during the batch tests. The results give an indication of bacterial activity due to their ability to oxidise ferrous ion to ferric ion. Due to the precipitation of jarosite and ferric arsenate during biooxidation, the iron content of the solution phase cannot be used directly to calculate the solubilisation of the sulphide minerals.

The BAU Property (Jugan) Concentrate batch tests showed an overall mass loss. The lime consumption during biooxidation varied between 39.9 kg/t and 81.9 kg/t feed respectively. The BAU Property (Jugan) Concentrate batch tests were net acid consuming during BIOX® for test until thirteen (13) days of treatment. The final two (2) tests at nineteen (19) days bio-oxidation treatment were acid generating.

The *Table 13-32 - Analyses of the Products of the BAU Property (Jugan) Concentrate BIOX® Tests* below summarizes the composition of the BIOX residue for each of the tests and *Table 13-33 - Analyses of the Solutions of the BAU Property (Jugan) Concentrate BIOX® Test* provides the corresponding solution assays. It can be seen that the sulphide sulphur decreases with the length of bio-oxidation, as would be expected. At the same time iron and arsenic are transferred to the solution phase.

The residue analysis in the table below has been corrected for mass change.

Test	Biooxidation Period (Days)	Residue Analysis (%)								
		S^T	S^{2-}	S^0	Fe^T	CO_3^{2-}	As	C^T	C^{org}	Si
BAT 5	3	15.76	13.39	< 0.5	12.29	0.84	3.43	0.81	0.57	15.7
BAT 3	6	11.33	4.74	< 0.5	5.76	1.25	1.12	0.99	0.81	18.0
BAT 4	8	10.38	2.99	< 0.5	3.65	0.84	2.01	0.90	0.73	15.8
BAT 6	10	8.71	1.66	< 0.5	1.76	1.27	0.38	0.92	0.73	15.6
BAT 1	13	10.74	0.91	< 0.5	1.56	1.17	0.41	0.98	0.65	15.2
BAT 7	19	11.11	0.22	< 0.5	1.27	0.42	0.29	0.86	0.63	15.2
BAT 2	19	11.90	1.18	< 0.5	2.35	0.45	0.89	0.86	0.65	15.3

Table 13-32 - Analyses of the Products of the BAU Property (Jugan) Concentrate BIOX® Tests

Test	Biooxidation Period (Days)	Solution Analysis (g/l)			
		Fe(II)	Fe(III)	As	S
BAT 5	5	0.7	22.2	9.11	23.0
BAT 3	3	0.4	34.9	15.28	24.5
BAT 4	4	0.4	37.3	13.91	24.3
BAT 6	6	0.1	42.7	14.01	28.4
BAT 1	1	0.1	49.6	18.28	36.5
BAT 7	7	0.1	43.9	16.97	32.5
BAT 2	2	0.1	44.3	17.02	34.0

Table 13-33 - Analyses of the Solutions of the BAU Property (Jugan) Concentrate BIOX® Test

The inverse trend in the liquor analyses confirms the oxidation of sulphide minerals and the dissolution of sulphur, iron and arsenic (shown in *Figure 13-22 - Sulphide Sulphur in Jugan Concentrate Solids and Total Sulphur in Solution Profiles* below).

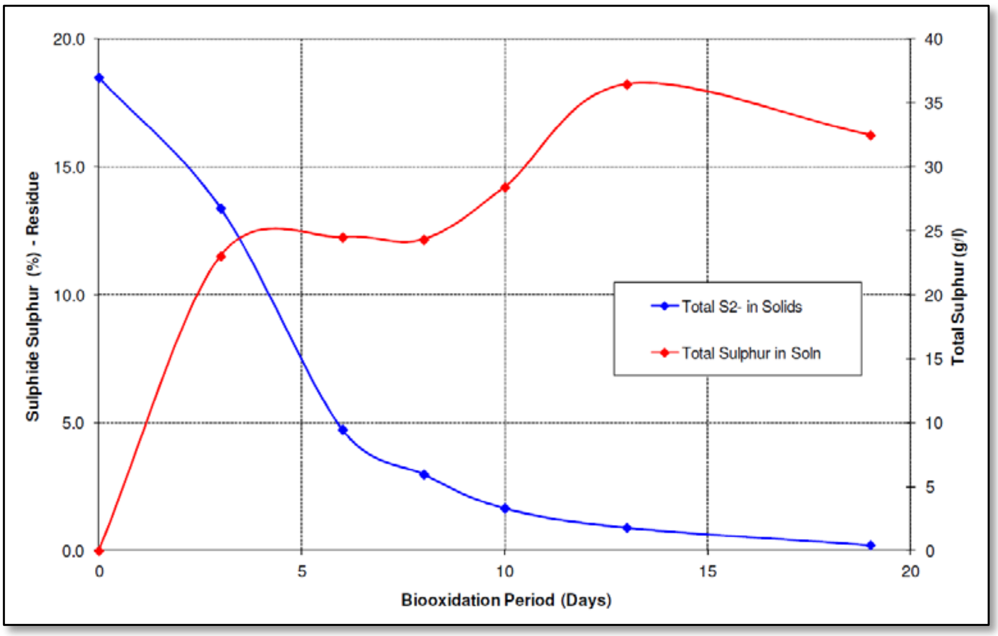


Figure 13-23 - Sulphide Sulphur in Jugan Concentrate Solids and Total Sulphur in Solution Profiles

The Fe:As (iron to Arsenic) molar ratios of the final BIOX® liquors are shown in *Table 13-34 - BAU Property (Jugan) Concentrate BIOX® Liquor Fe:As Ratios.*

Test	Biooxidation Treatment Period (Days)	Fe:As Molar Ratio
BAT 5	3	3.4
BAT 3	6	3.1
BAT 4	8	3.6
BAT 6	10	4.1
BAT 1	13	3.6
BAT 7	19	3.5
BAT 2	19	3.5

Table 13-34 - BAU Property (Jugan) Concentrate BIOX® Liquor Fe:As Ratios

The Fe:As molar ratios of the final BIOX® product solutions of the BAU Property Concentrate tests were all above 3:1. This implies that a stable precipitate would be formed during the neutralisation process.

The sulphide, arsenic, iron and the gold dissolution results are summarised in *Table 13-35 - Sulphide Removal, Arsenic Solubilisation and Corresponding Gold Dissolution during the Biooxidation of the BAU Property (Jugan) Concentrate* below. The sulphide oxidation results for the batch tests calculated from the un-oxidised solid head and biooxidised solid products corrected for mass change. Shown in *Figure 13-23 - Sulphide Sulphur Oxidation & Gold Dissolution vs Biooxidation Treatment Time* is the corresponding sulphide sulphur oxidation and gold dissolution curve.

Test	Biooxidation Treatment Period (Days)	Sulphide Removal (%)	Arsenic Dissolution (%)	Iron Dissolution (%)	Gold Dissolution (%)
BAT 5	3	21.2	35.8	28.3	59.5
BAT 3	6	72.1	79.0	66.4	85.9
BAT 4	8	82.4	62.4	78.7	87.3
BAT 6	10	90.2	93.0	89.7	91.3
BAT 1	13	94.7	92.3	90.9	91.2
BAT 7	19	98.7	94.7	92.6	92.4
BAT 2	19	93.1	83.4	86.3	87.7

Table 13-35 - Sulphide Removal, Arsenic Solubilisation and Corresponding Gold Dissolution during the Biooxidation of the BAU Property (Jugan) Concentrate

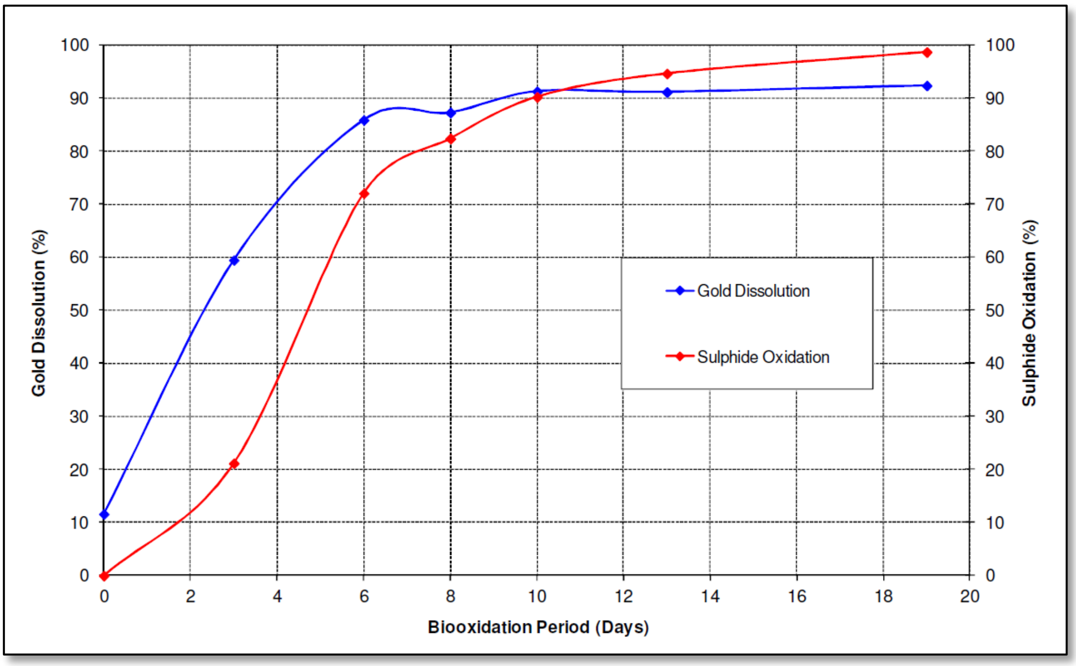


Figure 13-24 - Sulphide Sulphur Oxidation & Gold Dissolution vs Biooxidation Treatment Time

The laboratory batch treatment period to achieve the same extent of sulphide mineral oxidation is much longer than the corresponding residence time required in a full scale continuous plant. A laboratory treatment period of 20 to 30 days may translate to a plant residence time of only 4 to 5 days.

The results of the direct and BIOX® product cyanidation leach tests are given in *Table 13-36 - Gold Dissolutions vs. Treatment* Time below.

Test	Biooxidation Treatment Period (Days)	Head Gold Grade (g/t)	Residue Gold Grade (g/t)	Corrected Residue Gold Grade (g/t)	Gold Dissolution (%)	Corresponding Sulphide Sulphur Oxidation (%)
Direct	0	9.7	8.55	8.54	11.6	0.0
BAT 5	3	11.0	4.70	4.45	59.5	21.2
BAT 3	6	10.6	1.53	1.49	85.9	72.1
BAT 4	8	10.0	1.29	1.27	87.3	82.4
BAT 6	10	10.8	0.99	0.94	91.3	90.2
BAT 1	13	9.6	0.88	0.85	91.2	94.7
BAT 7	19	9.2	0.71	0.70	92.4	98.7
BAT 2	19	9.4	1.18	1.15	87.7	93.1

Table 13-36 - Gold Dissolutions vs. Treatment Time

Biooxidation of the BAU Property (Jugan) Concentrate improved the gold dissolution from 11.6 % on the unoxidised concentrate to 92.4 % for the batch amenability tests after nineteen (19) days.

The cyanide consumption during cyanidation of the BIOX® products tests varied between 10.7 and 15.2 kg NaCN/ton BIOX® concentrate feed. The lime consumptions for the same tests varied between 1.16 and 6.64 kg CaO/ton BIOX® concentrate feed, respectively.

The relationship between gold dissolution versus sulphide removal is shown in *Figure 13-24 - Gold Dissolution vs. Sulphide Removal* below.

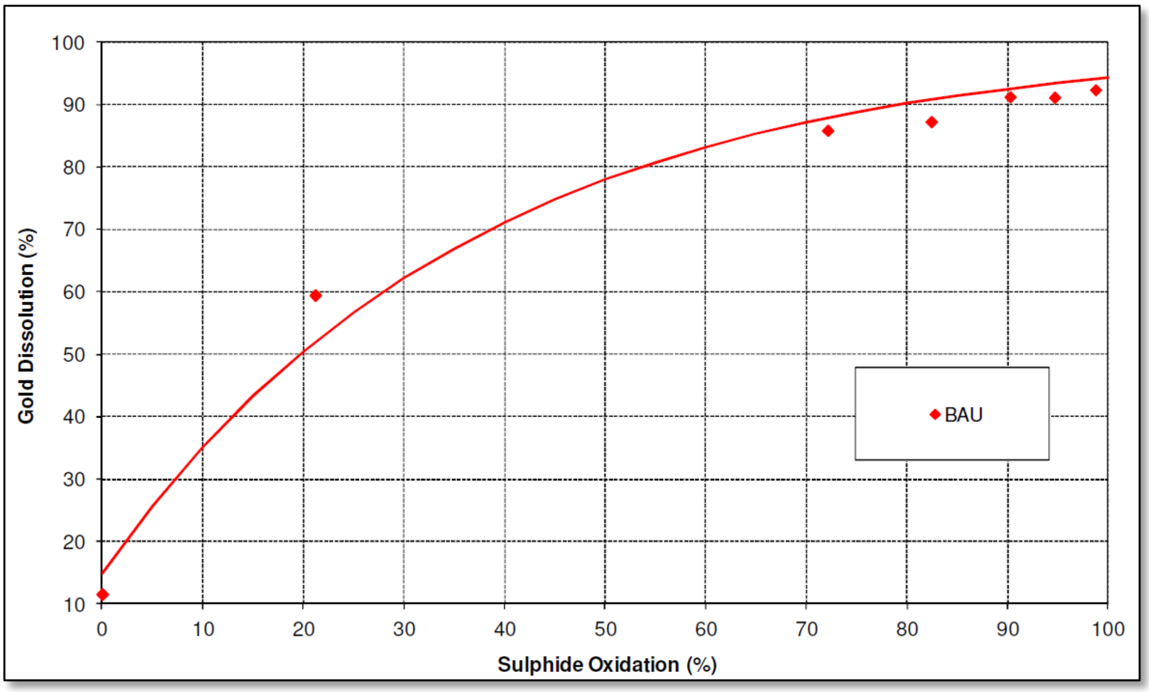


Figure 13-25 - Gold Dissolution vs. Sulphide Removal

The cyanide consumption during cyanidation of the BIOX® products of the tests varied between 10.7 and 15.2 kg NaCN/ton BIOX® concentrate feed. The lime consumptions for the same tests varied between 1.16 and 6.64 kg CaO/tonne BIOX® concentrate feed, respectively.

The results of the batch neutralisation tests conducted on the BAT 7 BIOX® liquor using AR-grade limestone and lime (Test 1), and AR-Grade lime only (Test 2) are summarised in *Table 13-37 - Results of the Batch Neutralisation of the Blended BIOX® Liquor.*

Feature	Units	Test 1	Test 2
Analysis of Neutralisation Feed:			
Fe(T)	g/l	15.3	14.7
As(T)	g/l	5.9	5.2
Fe:As	Molar	3.5	3.8

Feature	Units	Test 1	Test 2
Biooxidation liquor volume	ml	1000	1000
Water volume	ml	2000	2000
Total solution volume	ml	3000	3000
Analysis of Lime Feed:			
CaO Concentration	g/l	100	100
Volume added	ml	67	439
CaO Consumption	kg/t BIOX® feed	26.8	175.6
Total Ca^2+ consumption	kg/t BIOX® feed	128.0	125.5
Analysis of Neutralisation Precipitate:			
Fe	%	16.5	15.7
As	%	4.0	3.3
Analysis of Neutralisation Liquor:			
Fe	ppm	<0.02	0.47
As	ppm	<0.34	<0.34
Analysis of TCLP extract of Neutralisation precipitate:			
Fe	ppm	<0.05	<0.05
As	ppm	0.78	1.2

Table 13-37 - Results of the Batch Neutralisation of the Blended BIOX® Liquor

The results in *Table 13-37 - Results of the Batch Neutralisation of the Blended BIOX® Liquor* indicate that the arsenic in the BIOX® liquor can be successfully removed from solution by a two-stage treatment with AR-Grade Limestone to pH 5 for four (4) hours followed by the addition of AR-Grade lime to pH 7 or by a treatment with AR-Grade Lime only for five (5) hours. The neutralised effluents in all the tests contained <0.34 ppm As.

The toxicity characteristic leaching procedure (TCLP) testing of the neutralisation precipitates produced extracts containing <1 ppm As for Test 1 and 1.2 ppm for Test 2, well below the 5 ppm limit set by the EPA. The precipitates can be considered stable and thus acceptable for disposal on a tailings dam.

13.3.7. Albion

Orientation Albion Process$_{TM}$ testwork has been conducted by hrltesting (Brisbane, Australia) on a concentrate sample from the Bau Gold project provided by SGS Lakefield Oretest (Perth, Australia) during the months of March to May 2012.

The objectives of the orientation testwork program were to generate key process parameters and cost driver information needed to establish the technical feasibility and economic viability of the Albion Process$_{TM}$. These include:

- IsaMill grinding power requirements,
- Gold and silver (if appropriate) recovery versus sulphide sulphur oxidation,
- CIL gold/silver recovery from oxidized residues.

Key metallurgical results generated in this orientation Albion Process$_{TM}$ testwork program and their implications on the process are discussed in the following sections.

13.3.7.1. Head Assays

Key head assay data for the concentrate provided for testing is shown in the following *Table 13-38: Key Concentrate Head Assay Data*. Gold assay reported below was conducted in quadruplicate fire assay by GEKKO Systems.

Element/Component	Symbol	Unit	Assay
Gold	Au	(g/t)	21.2
Iron	Fe	(%w/w)	22.63
Arsenic	As	(%w/w)	8.97
Potassium	K	(%w/w)	1.21
Calcium	Ca	(%w/w)	0.64
Magnesium	Mg	(%w/w)	0.40
Aluminium	Al	(%w/w)	4.61
Carbonate	CO_3	(%w/w)	<3.0
Sulphur (Total)	S_T	(%w/w)	24.6
Sulphur (sulphide)	S^{2-}	(%w/w)	24.5

Table 13-38: Key Concentrate Head Assay Data

13.3.7.2. Mineralogy

Semi-quantitative XRD analysis was conducted at Queensland University of Technology X-ray Analysis Facility and revealed that pyrite and arsenopyrite were the major sulphide mineral phases. In addition, the XRD scan showed that the sample contained considerable non-sulphide gangue such as muscovite, ankerite and smectite all of which are considered acid consumers to varying degrees in the Albion Process. The following *Table 13-39: XRD Mineralogy Data* summarises the XRD results.

Component	Unit	Abundance
Pyrite	(%w/w)	38.3
Arsenopyrite	(%w/w)	16.7
Macasite	(%w/w)	1.0
Muscovite	(%w/w)	12.3
Ankerite	(%w/w)	3.1
Illite/Smectite	(%w/w)	3.4
Kaolinite	(%w/w)	0.9
Quartz	(%w/w)	16.8
Amorphous	(%w/w)	7.5

Table 13-39: XRD Mineralogy Data

13.3.7.3. Ultrafine Grinding

The first stage of the Albion Process$_{TM}$ is fine grinding of the concentrate. Most sulphide minerals cannot be oxidized at an acceptable rate under atmospheric pressure. The process of ultrafine grinding introduces a high degree of strain into the sulphide mineral lattice. As a result, the number of grain boundary fractures and lattice defects in the mineral increases by several orders of magnitude, relative to un-ground minerals. This introduction of strain lowers the activation energy for the oxidation of the sulphides, and enables leaching under atmospheric conditions. The rate of leaching is also enhanced, due to the increase in mineral surface area.

The indicative UFG curve for Bau concentrate material is shown in *Figure 13-25 - Indicative UFG Curve* below.

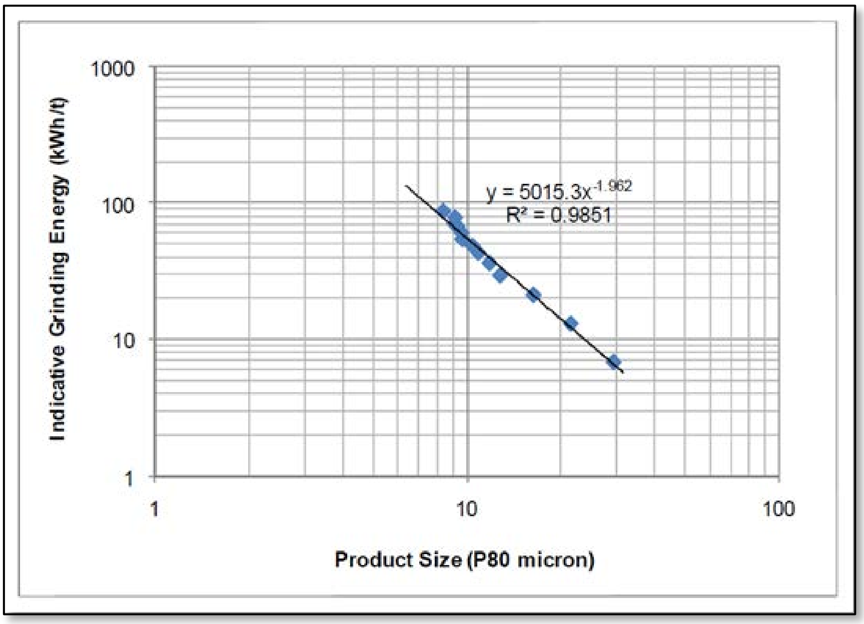


Figure 13-26 - Indicative UFG Curve

The indicative power draw measured for the two (2) coarser aliquot samples were:

- P_{80} ~12.5 µm required ~30 kWh/t
- P_{80} ~10.8 µm required ~43 kWh/t

Extrapolating the above preliminary indicative UFG data, the approximate specific power requirement to produce a product with a P_{80} of 10.5µm would be in the order of ±50 kWh/t for a commercial operation.

13.3.7.4. NAL Testwork Results

Key testwork results obtained in the orientation testwork program have been summarized in the following table.

Test No.	Albion Leach Conditions					Metallurgy Data		
	Grind Size (P_{80}, µm)	Av Temp (°C)	Time (h)	NaOH Add'n (kg/t)	Final pH	SOx[1] (%)	Limestone Add'n (t/t feed)	Residue Prod'n[2] (t/t feed)
AL1	8.5	94.9	48	33	5.5	65.5	469	1.90
AL2	8.5	94.9	54	66	5.5	78.5	410	1.50

[1] Based on S^{2-}unit available through out duration of the test to oxidize accounting for removal of kinetic samples
[2] Based on mass of test products

Table 13-40 - Orientation NAL Testwork Results Summary

Comparative sulphide sulphur oxidation (SOx) kinetic data for these two tests are shown in the following figure, and despite the dramatic increase in SOx after forty-eight (48) hours observed in AL2 the oxidation profiles are essentially the same.

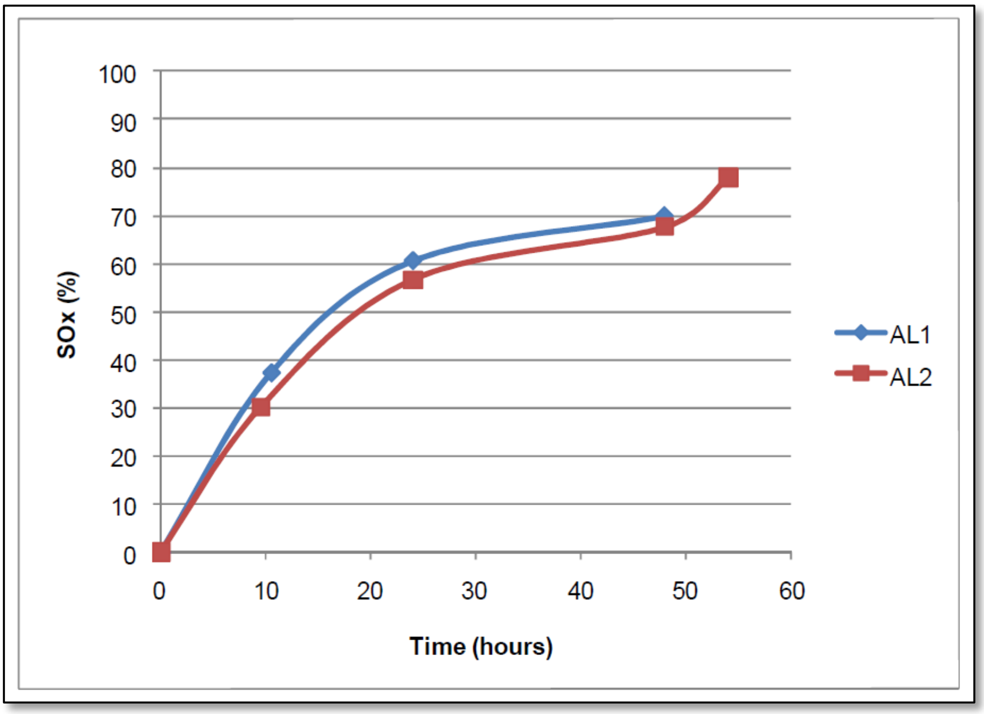


Figure 13-27 - NAL SO_x Kinetic Data

It is difficult to explain this sudden increase in SOx for AL2, but it could be related to sampling and/or assay issues. Ignoring the final oxidation point for AL2 it a ppears that there is little benefit in increasing the caustic addition, and in future testing it would be prudent to examine whether this reagent can be dropped altogether from the leach.

13.3.7.5. Gold Leaching

Two leaching methods have been employed to determine gold recovery on various oxidation products generated in the oxidative leach testwork, namely a diagnostic cyanide leach test (LeachWELL test) and Carbon-in-Leach (CIL) bottle roll test.

Kinetic samples from AL1 and AL2 have bee n subjected to a diagnostic gold leach test employing high cyanide conditions and the use of proprietary accelerants found in a LeachWELL tablet, which is commercially available. In Albion Process TM testwork this test is known as a LeachWELL test and conducted to provide a first approximation of the likely gold and silver recovery from oxidized residues. Typically these tests are conducted over six (6) hours.

LeachWELL residue gold assays and calculated recovery data for various kinetic samples are summarized in the following table.

Test	Sample Time	SOx %	Albion Sample Au Grade g/t	LeachWELL - Au Recovery: Test	Au Recovery %	LW Residue Au g/t	Test ID	CIL - Au Recovery: Re-Calc Au Grade g/t	CIL Au Ext. % (T/H)	NaCN Cons. kg/t	Lime Cons. kg/t
AL1	0	0.0	26.91		27.0	19.65					
	10.5	35.3	18.63		50.8	9.16					
	24	58.9	11.51		62.5	4.32					
	48	67.7	15.09	6h LW	80.4	2.96	CIL 1	9.6	89.7	9.5	9.6
				24h LW	85.6	2.18					
LimeBoil of AL1 Residue	6h Limeboil	75.4	13.12	6h LW	86.3	1.80	CIL 3	10.5	91.2	11.2	1.5
AL2	0	0.0	26.9		27.0	19.65					
	9.9	30.2	20.7		43.1	11.80					
	24.1	57.5	17.7		57.0	7.60					
	48.2	69.2	14.5		76.6	3.40					
	54.3	79.3	10.0	Original	61.4	3.85	CIL 2	6.7	84.8	17.0	16.9
				Repeat	73.7	2.62					

Table 13-41 - LeachWELL & CIL Gold Recovery

As part of the orientation testwork program three (3) CIL bottle tests have been completed. Two tests were conducted on final NAL residues from tests AL1 and AL2, identified as CIL1 and CIL2 respectively. The third test was conducted on AL1 residue subjected to lime boil pre-conditioning. CIL testwork conditions, feed properties and key metallurgical data for these tests have been summarised in the following table.

Test No.	CIL Feed: Oxidised Residue	CIL Feed: SOx (%)	Pre-treatment: Method	Pre-treatment: Temp (℃)	Pre-treatment: Time (h)	Pre-treatment: CaO (kg/t feed)	Pre-treatment: pH	CIL Leach Conditions: Time (h)	CIL Leach Conditions: Initial Solids (%w/w)	CIL Leach Conditions: pH	CIL Leach Conditions: Carbon Add'n (g/L)	CIL Leach Conditions: NaCN Level (mg/L)	Metallurgy Data: Gold Recovery Tails (g/t)	Metallurgy Data: Gold Recovery Yield (%)	Metallurgy Data: Reagent Consumption[α] NaCN (kg/t feed)	Metallurgy Data: Reagent Consumption[α] CaO (kg/t feed)
CIL1	AL1 residue	65.5	Agitation	20	20	CIL figure[γ]	10.4	24	39.7	10.7	24.0	1520	2.47	83.6		
								48		10.7		2300	1.55	89.7	9.5	9.6
CIL2	AL2 residue	78.5	Agitation	20	24	CIL figure[γ]	10.4	24	33.0	10.5	50.3	150	1.64	83.6		
								48		10.3		810	1.52	84.8	17.0	16.9
CIL3	AL1 residue	75.0	Lime boil[β]	80	8	25.6	9/10.6	48	34.4	11.0	26.6	2500	1.15	91.2		
								72		10.9		1660	1.15	91.2	11.2	1.5

[α] Consumption per tonne of CIL feed
[β] Limeboil residue conditioned for 1 hour at pH 10.5 using lime prior to CIL test
[γ] Pre-treatment lime conditioning figure included in CIL value

Table 13-42 - CIL Testwork Summary

Phase 1 orientation testwork has shown that the Albion Process$_{TM}$ is effective in oxidising Bau concentrate and releasing refractory gold for recovery using conventional CIL methods. Furthermore, despite the presence of considerable amounts of non-sulphide acid consuming and carbonate gangue minerals, moderate to high levels of sulphide oxidation could be readily achieved using the Albion Process, sufficient to liberate refractory gold.

A linear relationship between gold recovery and sulphide oxidation was observed in the testwork indicating an optimal SOx range between 70-75 % was required to obtain ≥90 % gold recovery in CIL testwork. Further treatment of the Albion Process residue using un-optimized lime boil conditions showed that a gold recovery of 91 % could be obtained in bench scale CIL testwork.

13.4. Metallurgical Summary & Conclusions

Both the historical and recent Besra metallurgical testwork on the Jugan ore deposit have demonstrated that about 95 % the gold is locked up in refractory arsenopyrite and pyrite with the remaining gold present in silicious gangue material. The recovery of gold from the ore requires a gold pre-concentration step in a treatment flowsheet comprising crushing, grinding, desliming and flotation to produce a high gold grade concentrate. For the base case and preferred option the flotation concentrate will be filtered to about 10 % moisture, packaged and sent to an outside smelting or gold refining operation. The sale of a flotation concentrate offers the lowest up front CAPEX and lowest OPEX as well as the the highest return on investement compared with treating concentrate on site.

Additional options which have been considered in the testwork include further treatment of the flotation concentrate in one of three oxidation processes described above (Albion, POX or BIOX). The oxidized concentrate is then treated by conventional carbon-in-pulp cyanide leaching (CIL), elution, gold electrowinning and gold dore melting. The CIL tailings are detoxified by the copper catalyzed SO_2/Air process and the eluted carbon regenerated for recycle to the CIL.

POX delivers the highest gold extraction (98%) at the lowest cyanide consumption rate (6kg/t). Gold extractions for both the BIOX and Albion are substantially lower at around 90 % with higher cyanide consumptions of about 15 kg/t. The unit cost of cyanide has a large impact on the operating cost. In addition to higher OPEX, the Albion process has the highest risk with only one commercial plant in operation at the Las Lagunas project in the Dominican Republic. The advantages of the POX are in part offset by a higher CAPEX than for BIOX and the Albion.

13.4.1. Flotation Concentrate (Base Case Option)

- The Jugan ore exhibits a very low abrasion index and moderate bond ball mill work index (11.3 to 13.2 kWh/t) and bond rod mill work index (13.5 to 14.1 kWh/t).
- The assay data for the Jugan ore zones indicate that there is very little difference with respect to mineral distributions in the ore zones apart from minor variations in arsenic and gold contents. The increases in arsenic coincide with increases in gold showing an evident correlation. Based on sulphide sulphur and arsenic assays the ore is estimated to contain between 2 and 2.5 wt % arsenopyrite and 4.5 to 5 wt % pyrite with a combined arsenopyrite-pyrite in the feed in the range 6.5 to 7.5 wt %. Therefore the maximum total sulphide upgrading factor is 15.4.
- The mineral assemblage is identical for all the Jugan ore zones tested across the deposit. The bulk of the Jugan ore feeds comprise non-sulphide gangue which is dominated by very fine grained Illite (mica) and silica. This results in production of excessive slimes after grinding.
- Gold deportment testing showed that very little gold is leached in whole ore cyanidation (0.6 to 2 %). About 70 % of the gold is associated with the arsenopyrite, 25 % with the pyrite and 5 % with silica.

- In excess of 95 % of the gold can be recovered in rougher – scavenging flotation. Due to varying slime entrainment the mass pull varied between 17 and 33 wt%. To mitigate the effect of feed slimes the flotation feed will be first deslimed by cyclone or a continuous Knelson. The flotation feed desliming work is still underway.
- Bulk rougher-scavenger followed by cleaner flotation without prior desliming has shown that 90 % of the gold can be recovered in a mass pull of 10 wt %. This corresponds to a gold upgrading ratio of 9:1 with respect to the feed grade. Mineralogical composition of a cleaner concentrate showed that the arsenopyrite and pyrite account for 67.4 wt% of the cleaner flotation concentrate. The remaining was comprised of 17 wt% mica (Muscovite), 6 wt% quartz, 6 wt% K-Feldspar, 3 wt% dolomite and minor rutile, sphalerite and siderite.
- The results indicate that inclusive of a desliming step, the flotation gold upgrade factor in the rougher circuit will be approximately 9 and in the cleaner stage greater than 2, giving an anticipated concentrate grade of +30g/tAu.

13.4.2. Pressure Oxidation

- The feed to the plant is the flotation concentrate produced in the base case.
- The best results for POX treatment of the flotation concentrate were obtained at 180 °C and 15 wt % solids, reaching 96.8 % sulphide oxidation after 60 minutes.
- There is a reduction of mass by up to 25 % after oxidation of the flotation concentrate.
- The molar ratio of iron over arsenic in solution was 5.2 in large excess over the molar ratio of 3 for precipitation of stable ferric arsenate.
- Direct cyanidation of the POX residue produced at 180 °C achieved 90 % gold dissolution after 2 hours and 97 % after 48 hours.

13.4.3. Biological Oxidation

- The feed to the BIOX plant will be the flotation concentrate produced in the base case.
- Successfull bacterial culture adaptation and inoculum build up tests on the BAU Property.
- The ferrous ion of the batch amenability tests started to decrease from day 1. The ferric ion concentration increased progressively;
- The maximum extent of sulphide sulphur oxidation achieved in the BAU Property Concentrate was 98.7%. The corresponding gold dissolution was 92.4%. These were achieved after a bio-oxidation period of 19 days.
- The batch treatment period to achieve the same extent of sulphide mineral oxidation is much longer than the corresponding residence time required in a continuous plant. A batch treatment period of 20 to 30 days may translate to a continuous plant residence time of only 4 to 5 days.
- There is a reduction in mass of about 10 % after the oxidation of the flotation concentrate.
- The results of neutralization indicate that the arsenic in the BIOX® liquor can be successfully removed from solution by a two-stage treatment with AR-Grade Limestone

to pH 5 for 4 hours followed by the addition of AR-Grade lime to pH 7. The neutralised effluents in all the tests contained <0.34 ppm As.

- The toxicity characteristic leaching procedure (TCLP) testing of the neutralisation precipitates produced extracts containing <1 ppm As, well below the 5 ppm limit set by the EPA. The precipitates can be considered stable and thus acceptable for disposal on a tailings dam.

13.4.4. Albion Method

- The feed to the Albion plant will be the flotation concentrate produced in the base case.
- The indicative power draw measured for the IsaMill ultrafine grinding to a P_{80} of 10.5 µm would be in the order of ±50 kWh/t for a commercial operation.
- A linear relationship between gold recovery and sulphide oxidation (SOx) was observed in the testwork indicating an optimal SO_x range between 70-75 % was required to obtain ⩾90 % gold recovery in CIL testwork. Further treatment of the Albion Process residue using lime boil conditions showed that a gold recovery of 91 % could be obtained in bench scale CIL testwork.
- An estimated retention time of 36 hours is required for the flotation concentrate oxidation at 10 wt % solids in the feed.
- The mass of the oxidation product is about 1.9 times that of the feed concentrate due to precipitation of arsenic as ferric arsenate, excess iron as goethite and sulphate as gypsum during oxidation.

14. Mineral Resources Estimates

14.1. Introduction

Besra Gold and North Borneo Gold personnel have carried out a resource update assessment for parts of the Bau Project based on the resource drilling and associated geological work conducted during the 2010 to 2012 period. The sectors (deposits) drilled and updated are Jugan, Taiton (Taiton A, Taiton B Extension & Tabai) and Bekajang (BYG-Krian).

Terra Mining Consultants/Stevens & Associates with the assistance of Olympus (now Besra Gold) and North Borneo Gold personnel carried out the original resource assessment at the Bau Project in 2010. Geological and resource modelling was undertaken at Jugan, Sirenggok, Taiton sector, Pejiru sector and Bekajang-Krian sector. The Taiton sector encompasses the Taiton A, Taiton B (excluding the underground deposit), Tabai and the Overhead Tunnel deposits. The Pejiru sector encompasses the Pejiru-Bogag, Pejiru Extension, Boring and Kapor deposits. The Bekajang-Krian sector encompasses the Bekajang North, Bekajang South, Johara, Karang Bila and BYG-Krian deposits. Jugan and Sirenggok are individual deposits in their own right.

The updated 2012 resource and associated 2011 resource updates as well as the 2010 original resource definition, is based on detailed resource drilling in 2010 to 2012. It also includes a review, validation and incorporation of all historic and recent drilling within the above areas; including geological re-interpretation. Estimation has been undertaken for gold only.

Note, the 2010 data is incorporated throughout the section for deposits defined in the 2010 resource definition but have not been updated in the interim except for the amendment to the lower cutoff limit. These have not materially changed since the 2010 work and the associated report. Reference could have been made to them but it was felt that for completeness sake they be incorporated here for ease of reference and referral.

A summary of resource totals by Resource Category is shown in *Table 14-1: Resource Update Summary by Category (November 2012)* and these updated resources by area/sector and deposit are also shown in *Table 14-2: Resource Update Summary by Sector/Area & Deposit (November 2012)* below.

Category	Tonnes (t)	Grade (g/t)
Measured	3,405,600	1.52
Indicated	17,879,700	1.67
Measured + Indicated	21,285,300	1.64
Inferred	51,329,000	1.32

Table 14-1: Resource Update Summary by Category (November 2012)

Sector	Deposit	Category	Cutoff (g/t)	Tonnes (t)	Grade (g/t)
Jugan	Jugan	Measured	0.5	3,405,600	1.52
		Indicated	0.5	14,505,700	1.51
		Inferred	0.5	1,774,000	1.57
Bekajang-Krian	BYG-Krian	Indicated	0.5	1,857,000	2.02
		Inferred	0.5	3,328,000	1.51
	Bekajang South	Inferred	0.5	2,294,000	1.60
	Bekajang North	Inferred	0.5	1,250,000	2.33
	Karang Bila	Inferred	0.5	628,000	2.50
	Tailings	Inferred	0.5	3,138,000	1.00
Taiton	Taiton A	Indicated	0.5	1,148,000	2.23
		Inferred	0.5	690,000	1.37
	Tabai (Open Pit)	Indicated	0.5	133,000	2.83
		Inferred	0.5	75,000	1.74
	Tabai (Underground)	Indicated	2.0	236,000	5.23
		Inferred	2.0	40,000	4.67
	Taiton B	Inferred	0.5	1,848,000	1.56
	Umbut	Inferred	0.5	690,000	2.26
	Overhead Tunnel	Inferred	2.0	76,000	3.36
Sirenggok	Sirenggok	Inferred	0.5	8,346,000	1.14
Pejiru	Pejiru-Bogag	Inferred	0.5	11,800,000	1.10
	Pejiru Extension	Inferred	0.5	7,053,000	1.14
	Kapor	Inferred	0.5	4,849,000	1.59
	Boring	Inferred	0.5	2,096,000	1.10
Say Seng	Bukit Sarin	Inferred	0.5	1,110,000	1.27
	Say Seng	Inferred	0.5	244,000	3.24

Table 14-2: Resource Update Summary by Sector/Area & Deposit (November 2012)

For the 2010 resource definition Terra Mining Consultants/Stevens & Associates have classified the defined mineralization according to the definitions of National Instrument 43-101, CIMM Definitions and the Australasian Institute of Mining & Metallurgy's JORC Code 2004. Similarly, the 2011 and 2012 resource updates have been classified in the same manner by Besra/NBG.

For the purposes of the report the relevant AusIMM resource definitions used for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code 2004) are listed below along with the comparative C.I.M.M. Standards for resources. *Table 14-3: AusIMM & CIMM Comparative Resource Definitions* below lists the comparative descriptions.

AusIMM JORC Code Definitions	C.I.M.M. Standards Definitions
A **'Mineral Resource'** is a concentration or occurrence of material of intrinsic economic interest in or on the Earth's crust in such form, quality and quantity that there are	A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and

AusIMM JORC Code Definitions	C.I.M.M. Standards Definitions
reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.	of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.	An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.
An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.	An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

AusIMM JORC Code Definitions	C.I.M.M. Standards Definitions
A **'Measured Mineral Resource'** is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and grade continuity.	A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

Table 14-3: AusIMM & CIMM Comparative Resource Definitions

Each of the areas/sectors and/or the deposits therein are discussed in more detail in the following sections.

14.2. Jugan Sector

Originally this resource section was completed in 2010 and is detailed in report "*Technical Report on Bau Project in Bau, Sarawak, East Malaysia*".

Since the above technical report a resource drilling campaign was undertaken in late 2010 and early 2011, and a further drilling programme in 2012. The change to the resource after both drilling campaigns was not significant in terms of the increase in resources and therefore a 43-101 report was not issued. However, the combined drilling along with the reserve definition and Feasibility Study work has necessitated a resource update including Jugan.

This section will outline the resource changes that occurred in November 2012. The February 2012 resource definition will be summarised in the previous resource sub-section below.

14.2.1. Introduction & General

The Jugan deposit is situated approximately 7 kilometres north of the town of Bau and is a single deposit outcropping as a small hillock.

The resource assessment conducted by Terra Mining Consultants/Stevens & Associates in August 2010 and by Besra (formally Olympus Pacific Minerals) in the February and November 2012 resource updates included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;
- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;
- Combining the above data into a clean and validated resource database with associated data being verified;
- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;
- Resource estimation work to determine the mineral resource using 3 different estimation techniques;

All data used for this resource update and previous updates was supplied by, or sourced from, Besra /North Borneo Gold or determined by Terra Mining Consultants/Stevens & Associates (in the case of previous estimates) from available information and data compiled from the drilling conducted since 2010. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

14.2.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of these are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;
- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

14.2.3. Ore Zone Definition

The ore zone at Jugan was defined in the following manner:

- Drillhole sections were created and interpreted faults, geological and mineralized zone grade boundaries (≥0.5 g/t Au lower cut-off) were drawn;
- The granodiorite dykes were also interpreted from drillholes and surface mapping;

- The grade boundaries were correlated from section to section and cross-checked in plan;
- In the absence of zone continuity, extrapolations were made in between the two drill sections, and up/down dip, using standard methodologies;
- The definition of the mineralized zones and the methodology used was validated visually on each section, and in 3D, and samples within the zone wireframe were analysed;
- The ore zone was terminated using the surveyed topography.

In the ore zone definition there are isolated cases of assay values below the lower cut-off value. These have only been included where they fall within samples above the cut-off, are of minor effect and cannot be excluded due to their isolated nature.

14.2.4. Statistical Analysis of Data

The 2010 Jugan database consisted of 173 drillhole collar entries, 173 collar survey entries, 7,064 assay records, 1,423 density records, and 12,425 lithology records; and 44 trench/costean collar records, 546 trench/costean survey entries, 72 trench/costean lithology entries and 545 trench/costean assay records.

In November 2012 (including the February 2012 update) the database consisted an additional of 79 drillhole collar entries, 756 collar survey entries, 13,310 assay records, 662 density records, and 1,781 lithology records; and 11 trench/costean collar records, 276 trench/costean survey entries, 91 trench/costean lithology entries and 756 trench/costean assay records.

Additionally, drillhole alteration, mineralisation, structure, veining, geotechnical, geomechanical and recovery data was also captured for the 2011/2012 drillhole programmes in line with NBG logging policies.

This results in a total database of 252 drillhole collar entries, 929 collar survey entries, 20,374 assay records, 2,085 density records, and 14,206 lithology records; and 55 trench/costean collar records, 822 trench/costean survey entries, 162 trench/costean lithology entries and 1,301 trench/costean assay records.

In the 2010 resource definition the database had a total of 17,769.05 metres of drilling was drilled in and around the Jugan deposit. The drillhole depths varied from 5 metres to 716 metres with an average depth of approximately 102 metres. The drillholes consisted of 82 RC holes and 91 diamond cored holes in BQ, NQ, HQ & PQ sizes. A total of 1,133.53 metres of trenching and costeaning was undertaken within the mineralised zone. Some trenching/costeaning occurred outside this mineralised zone and is not included. The trenches/costeans varied in length from 1.69 to 44 metres with an average length of 25.76 metres.

As a result of the 2011 and 2012 drilling campaigns the resource definition database had an additional total drilling meterage of 17,397.9 metres (including re-drills) in and around the Jugan deposit. The drillhole depths from these programmes varied from 42.3 metres to 478.3

metres with an average depth of approximately 220.23 metres. The drillholes consisted of 79 diamond cored holes in HQ & PQ sizes with a few (2 holes) being reduced to NQ when drilling problems were encountered. An additional amount of 746.1 metres of trenching and costeaning was undertaken within the mineralised zone. The trenches/costeans varied in length from 13 to 180 metres with an average length of 67.83 metres.

This gives a final Jugan database containing 35,166.95 metres of drilling and a total number of 252 drillholes (82 RC and 170 DDH). Overall the drillhole depths range from 5 metres to 716.01 metres, with an average depth of 139.55 metres. Also, there are now a total of 1,879.63 metres of trenching/costeaning with an average linear length of 34.17 metres.

In 2010 there was a total of 4,545 combined drillhole and trench/costean assay samples that fell within the mineralized zone at Jugan. In 2012 this is now 8,331 combined drillhole and trench/costean assay samples.

Statistics were calculated for gold, density and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-4: Jugan: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au	Density
Number of Records	8,471	8,471	8,471
Number of Samples	8,471	8,427	1,356
Missing Values	-	44	7,115
Minimum Value	0.00	-	1.53
Maximum Value	30.30	61.85	3.16
Range	30.30	61.85	1.63
Mean	1.08	1.46	2.64
Variance	0.43	4.05	0.03
Standard Deviation	0.66	2.01	0.18
Standard Error	0.01	0.02	0.00
Skewness	16.02	7.35	- 1.11
Kurtosis	572.90	137.39	3.10
Geometric Mean	0.93	0.68	2.63
Sum of Logs	- 659.31	- 3,203.55	1,311.47
Mean of Logs	- 0.08	- 0.38	0.97
Log Variance	0.44	2.32	0.01
Log Estimate of Mean	1.15	2.17	2.64

Table 14-4: Jugan: Ore Zone Drillhole Sample Statistics

Samples within the Jugan ore zone were composited to 1 metre lengths, resulting in 8,930 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-5: Jugan: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Jugan.

Drillhole Field	Length	Au	Density
Number of Records	8,930	8,930	8,930
Number of Samples	8,930	8,806	1,017
Missing Values	-	124	7,913
Minimum Value	0.50	-	1.71
Maximum Value	1.00	61.85	3.16
Range	0.50	61.85	1.45
Mean	1.00	1.48	2.64
Variance	0.00	3.85	0.03
Standard Deviation	0.03	1.96	0.17
Standard Error	0.00	0.02	0.01
Skewness	- 12.30	8.08	- 1.13
Kurtosis	159.55	164.63	2.69
Geometric Mean	1.00	0.75	2.63
Sum of Logs	- 34.19	- 2,495.40	983.62
Mean of Logs	- 0.00	- 0.28	0.97
Log Variance	0.00	2.00	0.00
Log Estimate of Mean	1.00	2.05	2.64

Table 14-5: Jugan: Ore Zone Composited Drillhole Sample Statistics

The Au data shown statistically above is also shown in graphical form below. *Figure 14-1: Jugan: Log Histogram of Au Ore Zone Composites* to *Figure 14-3: Jugan: Log Probability Plot of Au Ore Zone Composites* below displays the log histogram, cumulative log histogram and log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

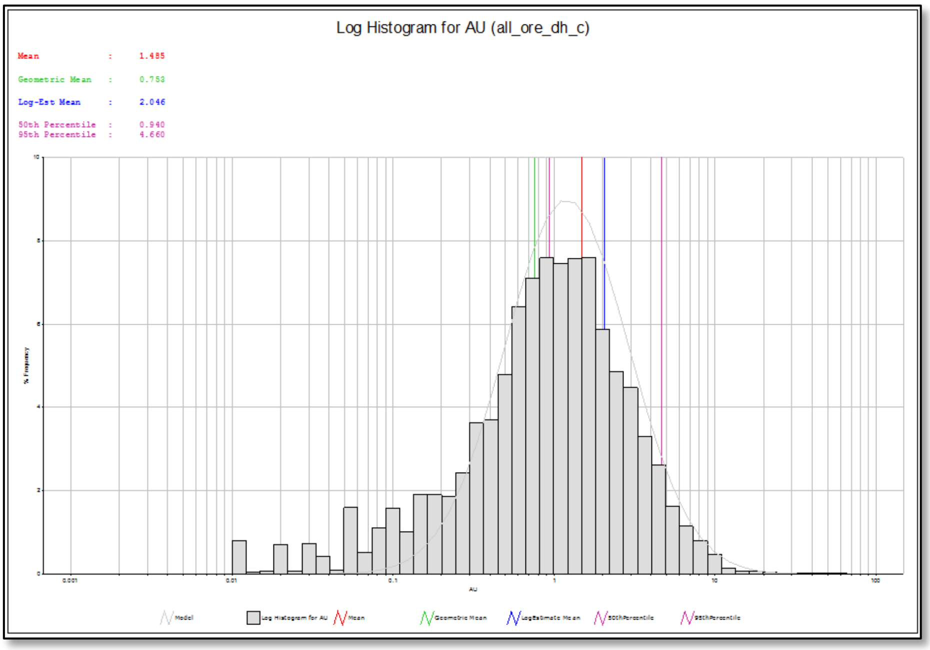


Figure 14-1: Jugan: Log Histogram of Au Ore Zone Composites

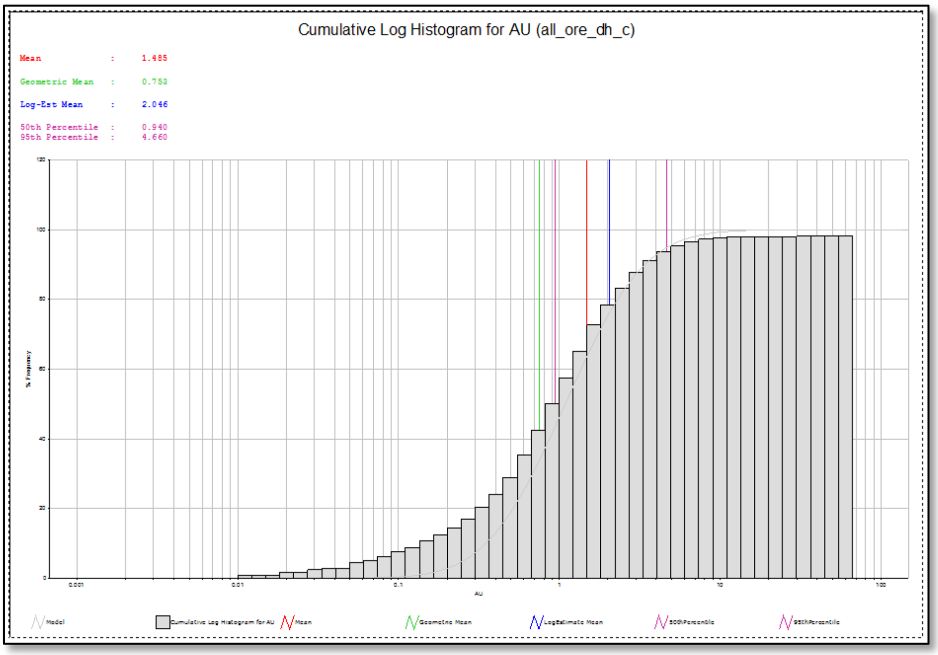


Figure 14-2: Jugan: Cumulative Log Histogram of Au Ore Zone Composites

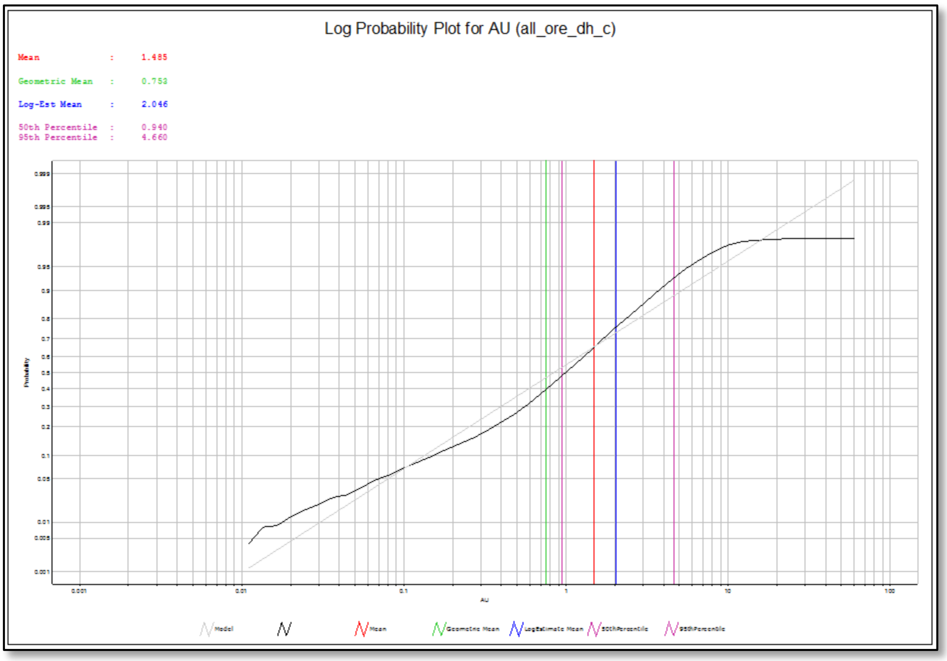


Figure 14-3: Jugan: Log Probability Plot of Au Ore Zone Composites

Also shown below is the normal histogram for all ore drillhole data.

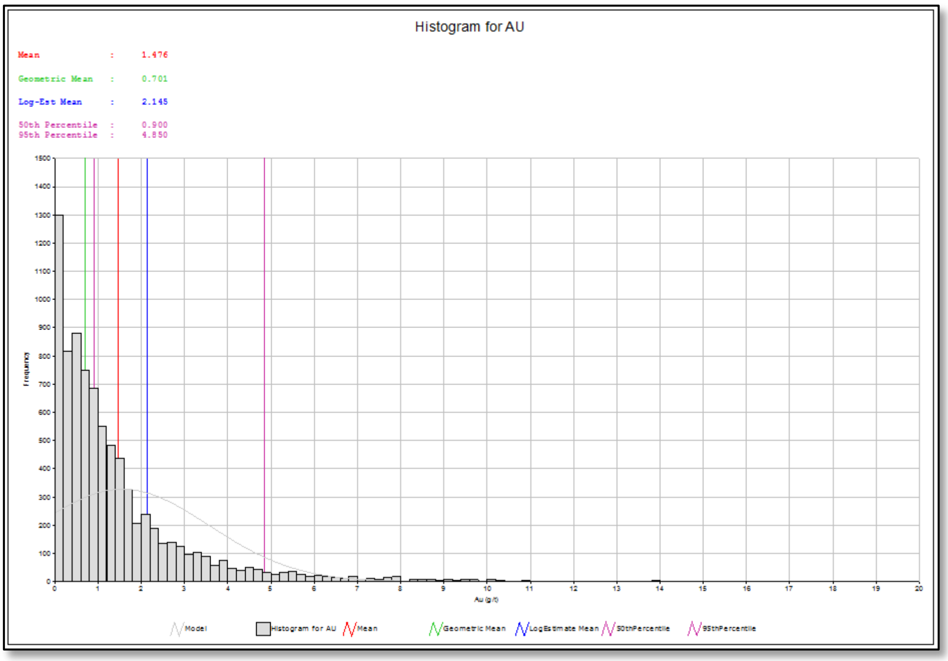


Figure 14-4: Jugan: Normal Histogram of Au Ore Zone Samples

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-6: Jugan: Quantile Analysis of Au Drillhole Composites* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Jugan Ore Zone.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	829	0.04	-	0.10	31.07	0.25
10	20	829	0.19	0.10	0.28	154.62	1.26
20	30	829	0.38	0.29	0.48	317.57	2.60
30	40	829	0.59	0.49	0.69	485.06	3.96
40	50	829	0.79	0.69	0.90	658.26	5.38
50	60	829	1.04	0.90	1.19	861.87	7.04
60	70	829	1.37	1.19	1.55	1,135.47	9.28
70	80	829	1.82	1.55	2.16	1,505.14	12.30
80	90	829	2.70	2.16	3.40	2,237.26	18.28
90	100	830	5.84	3.40	61.85	4,849.31	39.63
90	92.5	207	3.66	3.40	3.99	757.43	6.19
92.5	95	208	4.37	3.99	4.82	909.59	7.43
95	97.5	207	5.57	4.85	6.42	1,153.69	9.43
97.5	100	208	9.75	6.45	61.85	2,028.60	16.58
0	100	8291	1.48	-	61.85	12,235.62	100.00

Table 14-6: Jugan: Quantile Analysis of Au Drillhole Composites

Looking at the primary percentiles, it can be seen that approx. 39% of the metal percentage can be found in the top 10 % range (top 830 samples), and that there is a significant jump in the mean grade and metal content from the previous range. Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains nearly 17 % of the Au metal content.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top cut of 9.75 g/t Au (mean of the 97.5 percentile) should be applied to the samples above this value in order to remove any effect of the high grade samples in the estimation process.

Although not part of the resource the elements arsenic (As), iron (Fe) and sulphur (S) were analysed and modelled for geo-metallurgical reasons. The statistics and associated information for these elements are shown below for the sake of completeness and disclosure purposes.

Listed below in *Table 14-7: Jugan: Ore Zone Drillhole Sample Statistics for As, Fe & S* is the statistics for the three elements as found in the composited ore drillhole data.

Drillhole Field	As	Fe	S
Number of Records	8,929	8,929	8,929
Number of Samples	5,202	4,566	1,214
Missing Values	3,727	4,363	7,715
Minimum Value	16.00	4,123.00	1,300.00
Maximum Value	32,100.00	111,000.00	51,600.00
Range	32,084.00	106,877.00	50,300.00
Mean	9,737.98	43,500.95	23,160.99
Variance	20,052,587.78	78,664,410.86	47,249,213.98
Standard Deviation	4,478.01	8,869.30	6,873.81
Standard Error	62.09	131.26	197.28
Skewness	0.14	0.67	- 0.11
Kurtosis	0.78	4.75	1.26
Geometric Mean	7,800.47	42,541.83	21,791.78
Sum of Logs	46,620.01	48,665.54	12,127.00
Mean of Logs	8.96	10.66	9.99
Log Variance	0.86	0.05	0.16
Log Estimate of Mean	11,986.61	43,591.59	23,601.72

Table 14-7: Jugan: Ore Zone Drillhole Sample Statistics for As, Fe & S

The Arsenic, Iron and Sulphur data shown statistically above is also shown in graphical form below. *Figure 14-5: Jugan: Normal Histogram of As Ore Zone Samples* to *Figure 14-13: Jugan: Probability Plot of S Ore Zone Samples* below display the log histogram, cumulative log histogram and log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

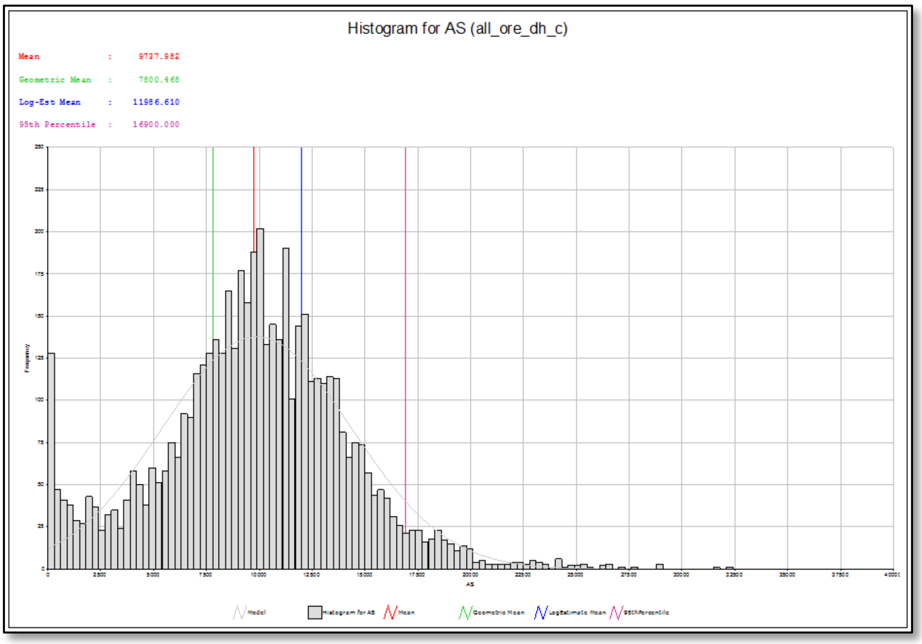


Figure 14-5: Jugan: Normal Histogram of As Ore Zone Samples

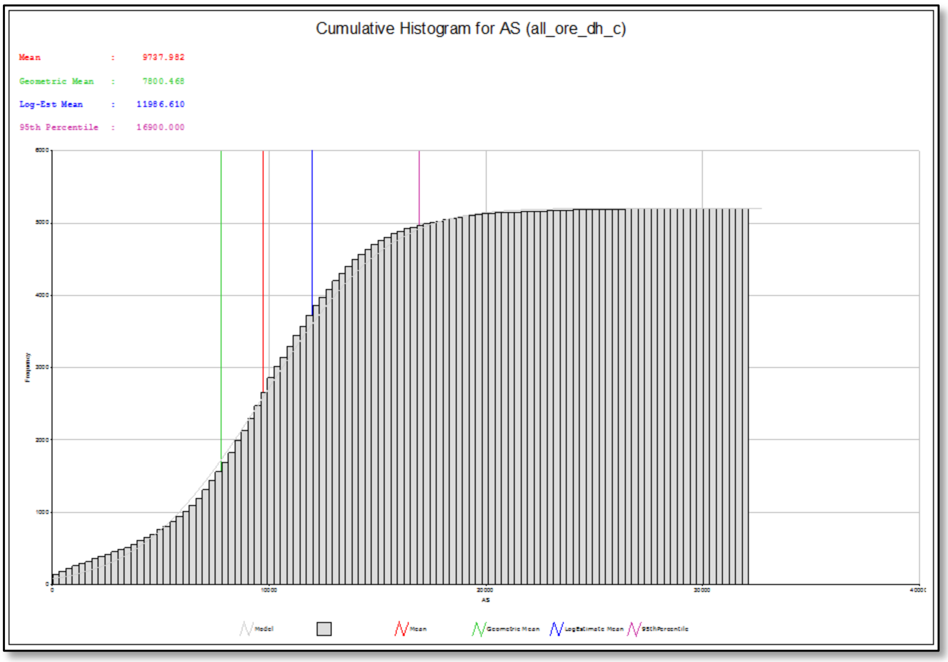


Figure 14-6: Jugan: Cumulative Histogram of As Ore Zone Samples

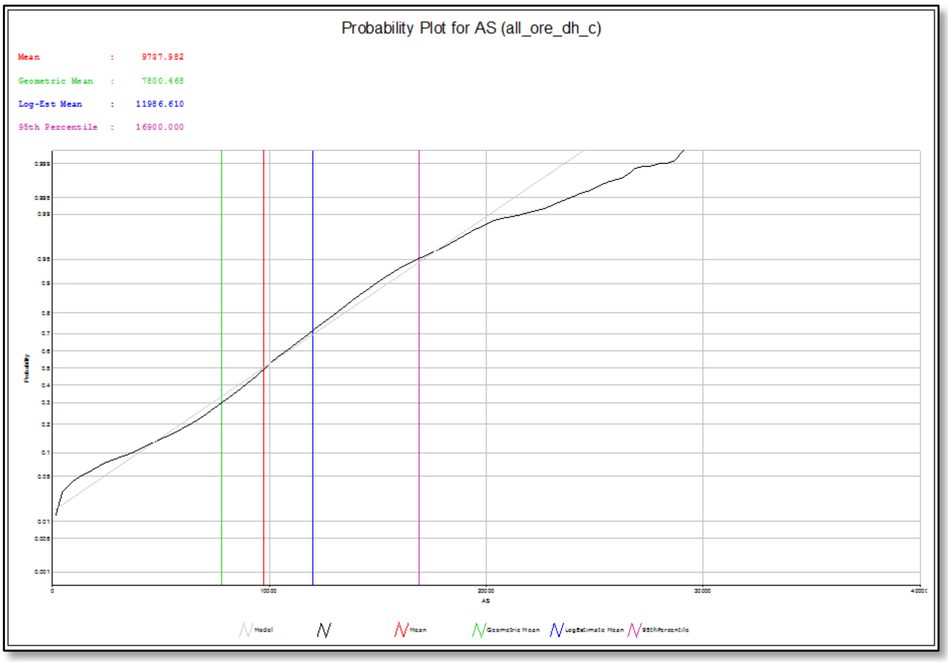


Figure 14-7: Jugan: Probability Plot of As Ore Zone Samples

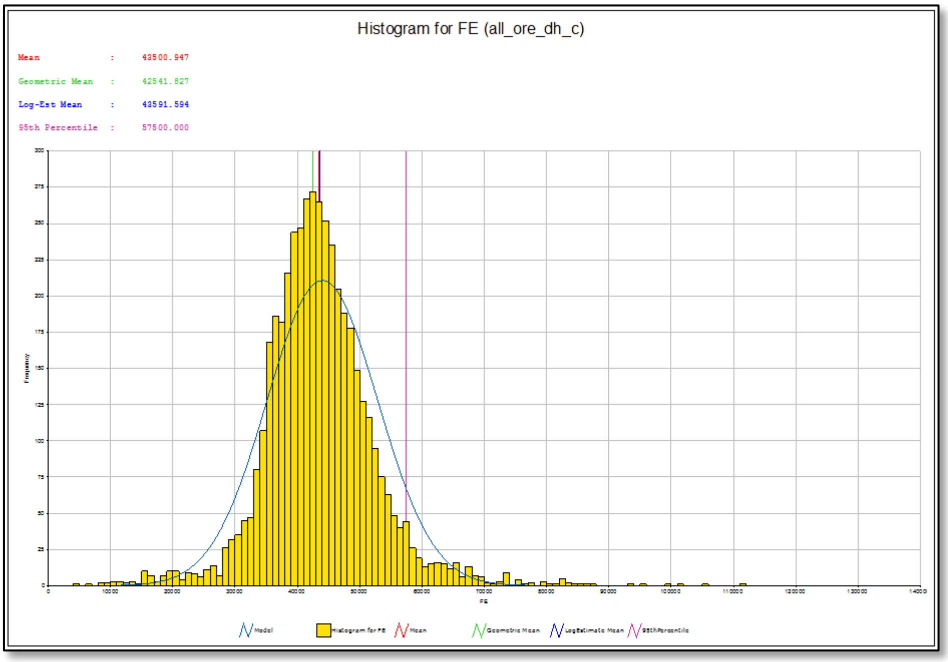


Figure 14-8: Jugan: Normal Histogram of Fe Ore Zone Samples

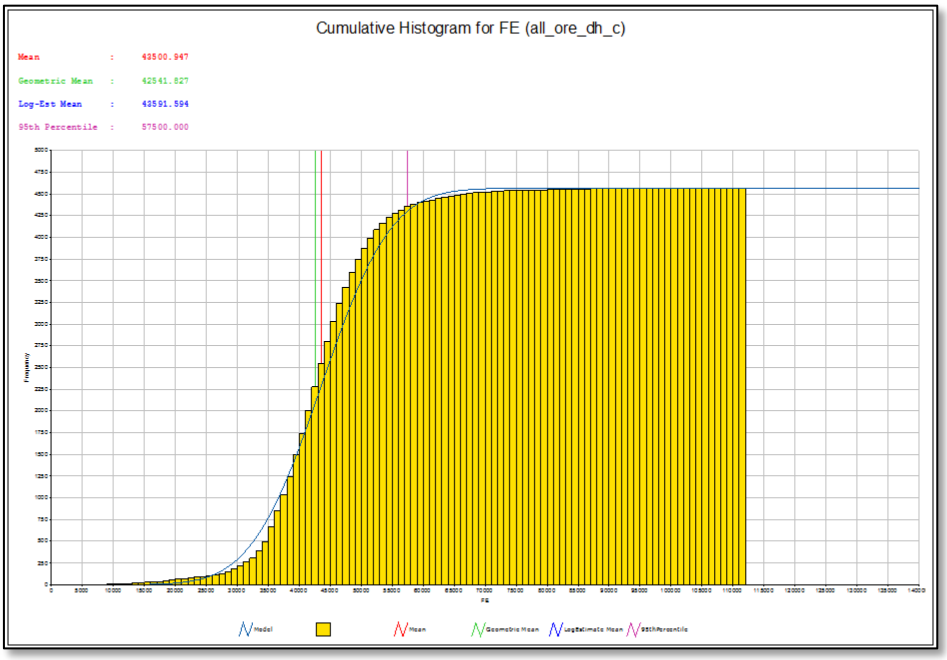


Figure 14-9: Jugan: Cumulative Histogram of Fe Ore Zone Samples

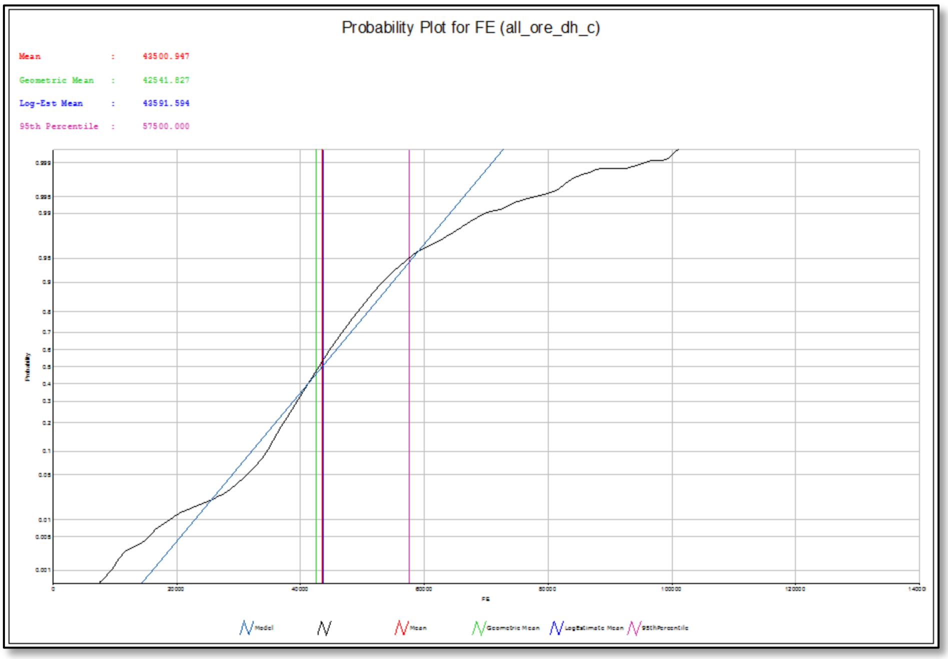


Figure 14-10: Jugan: Probability Plot of Fe Ore Zone Samples

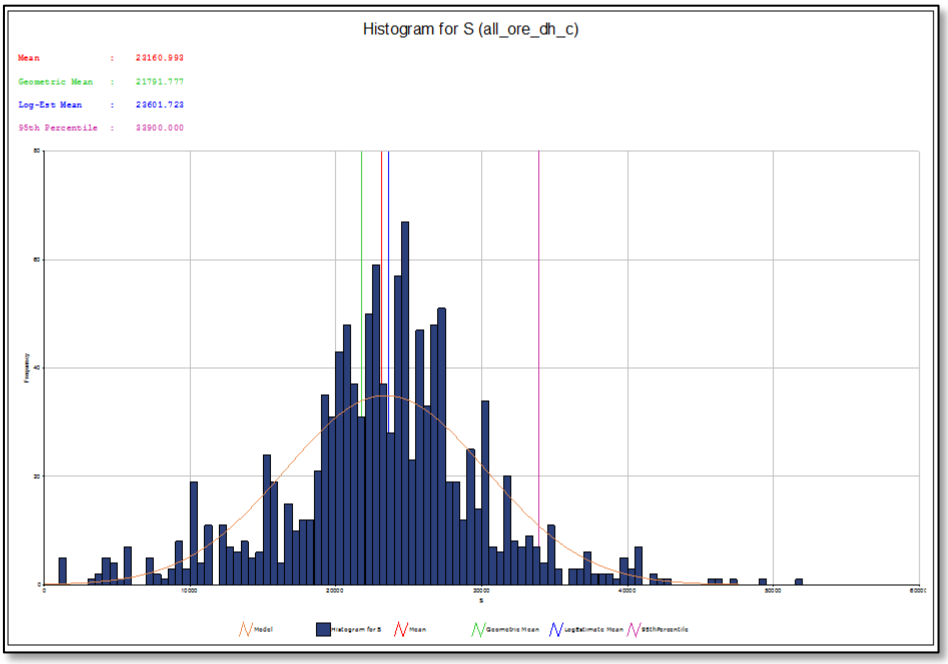


Figure 14-11: Jugan: Normal Histogram of S Ore Zone Samples

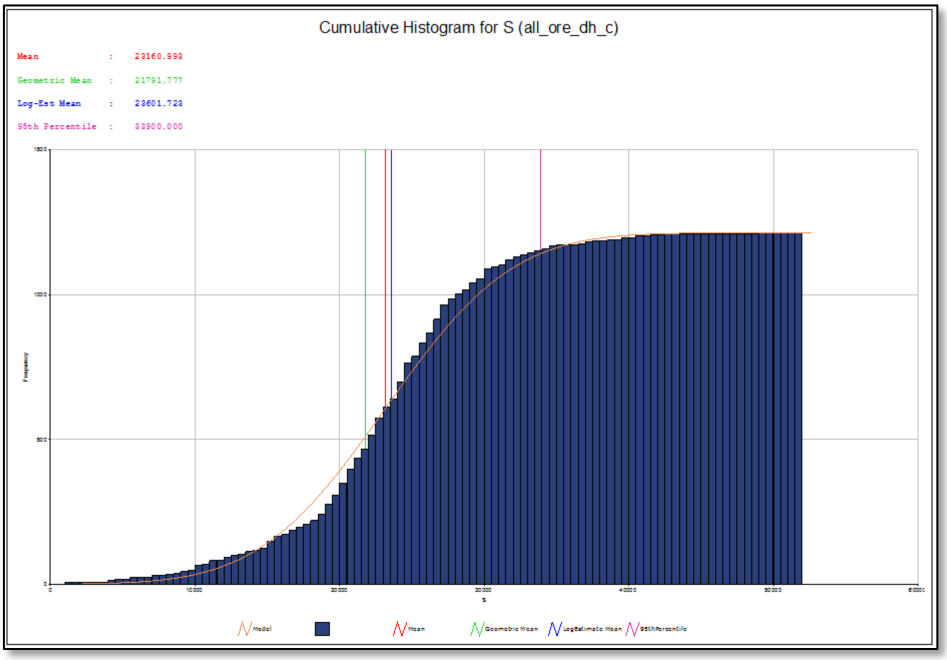


Figure 14-12: Jugan: Cumulative Histogram of S Ore Zone Samples

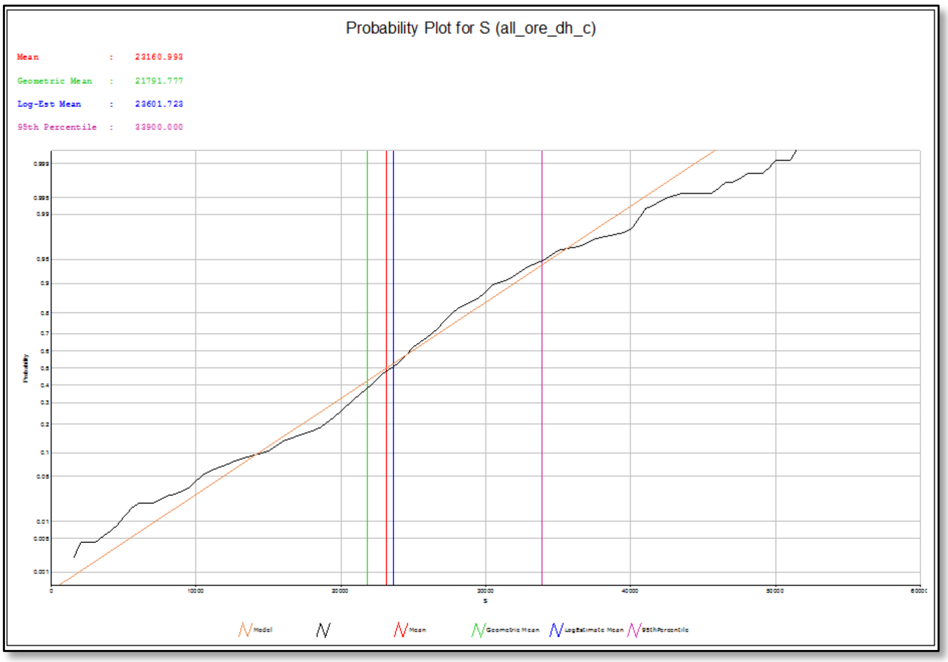


Figure 14-13: Jugan: Probability Plot of S Ore Zone Samples

14.2.5. Semi-Variogram Analysis

Semi-variogram analyses were undertaken to determine the semi-variogram parameters for use in the Ordinary Kriging. Downhole, horizontal and vertical increment semi-variograms were generated with the best semi-variograms selected that defines the strike, dip and dip direction. These semi-variograms were used to determine the nugget, sill values and ranges.

A log semi-variogram and two-range spherical model were used. A best fit model in the downhole semi-variogram was used to define the nugget. Subsequent model fitting was applied to the strike and dip/dip-direction to define the sill values by varying the ranges in these directions. The semi-variogram parameters are listed in Table 14-9: Jugan: Ordinary Kriging Estimation Parameters in *Section 16.2.7* below

The semi-variograms for Jugan are shown below in *Figure 14-14: Jugan: Downhole Semi-Variogram* to *Figure 14-16: Jugan: Dip/Dip Direction Semi-Variogram*

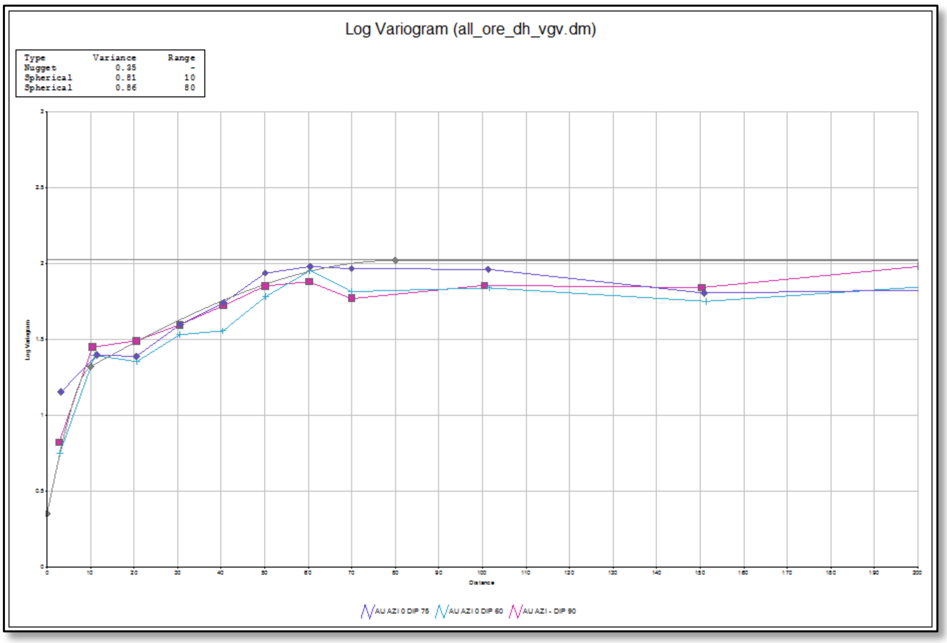


Figure 14-14: Jugan: Downhole Semi-Variogram

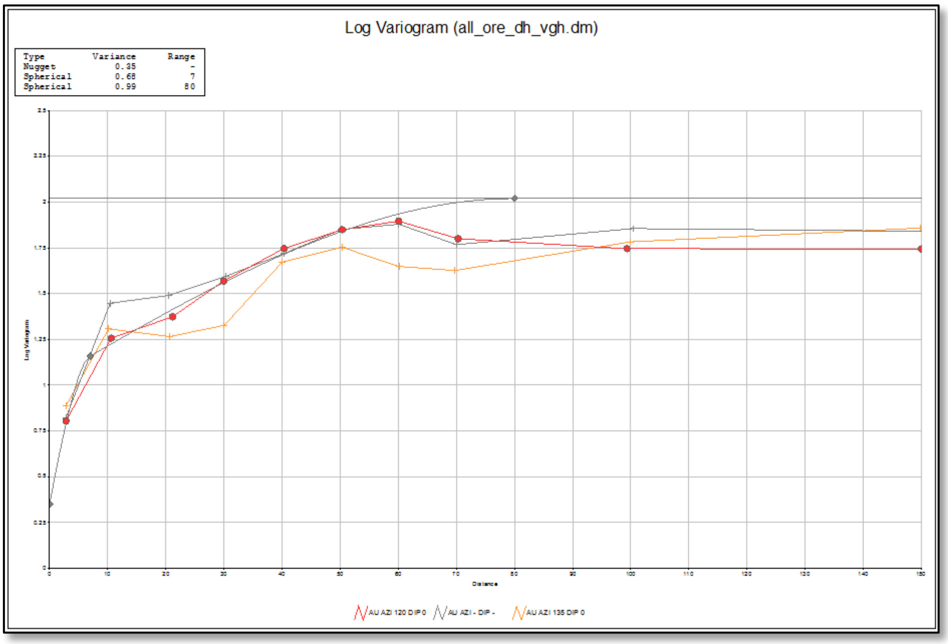


Figure 14-15: Jugan: Strike Semi-Variogram

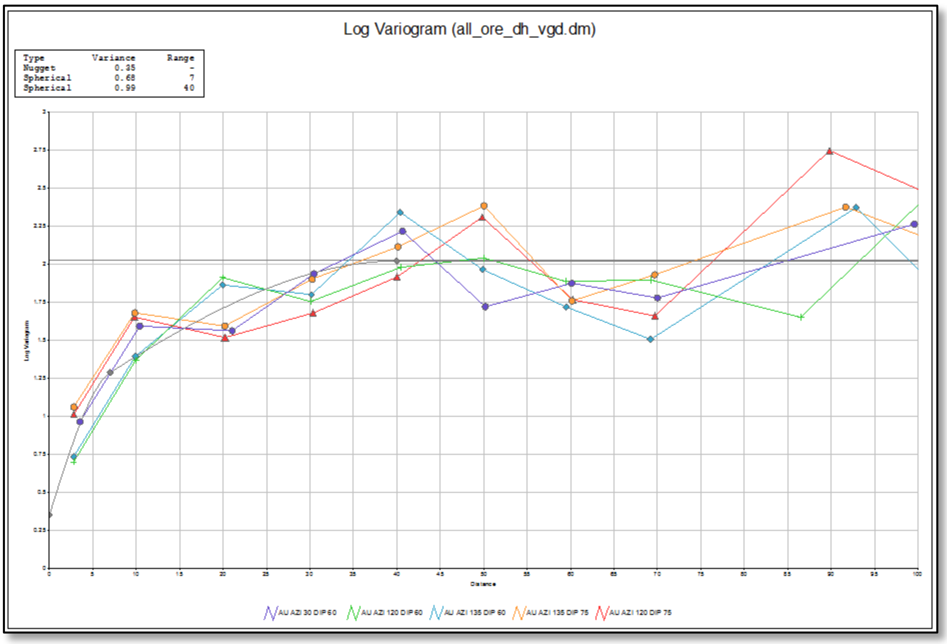


Figure 14-16: Jugan: Dip/Dip Direction Semi-Variogram

The modelled log semi-variogram values were back calculated to normal semi-variograms for use with Ordinary Kriging. The back transform is shown in *Figure 14-17: Jugan: Log to Normal Semi-Variogram Transform* below.

Converting log to normal variograms:

Enter the following data:

log variance :	2.02	**non-standardised**
log nugget :	0.35	0
log sill 1:	0.68	7
log sill 2:	0.99	80
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.34**
sill 1:	**0.27**
sill 2:	**0.39**
sill 3:	**0.00**

Figure 14-17: Jugan: Log to Normal Semi-Variogram Transform

Geostatistical analysis and modelling was done on the geo-metallurgical elements As, Fe and S. The results were mixed and inconclusive in some instances. The variograms that could be generated, particularly for As did show similar directions as for Au though the ranges appeared to be slightly shorter. *Figure 14-18: Jugan: Downhole Semi-Variogram for As* and *Figure 14-19: Jugan: Dip/Dip Direction Semi-Variogram for As* below show the variograms for As and are included only to demonstrate the similarities with the Au variograms.

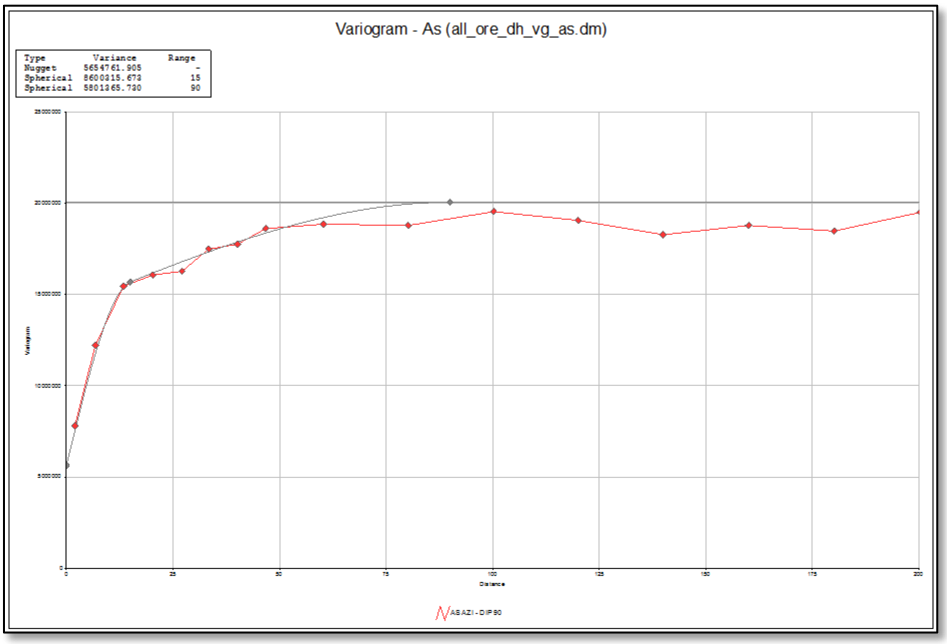


Figure 14-18: Jugan: Downhole Semi-Variogram for As

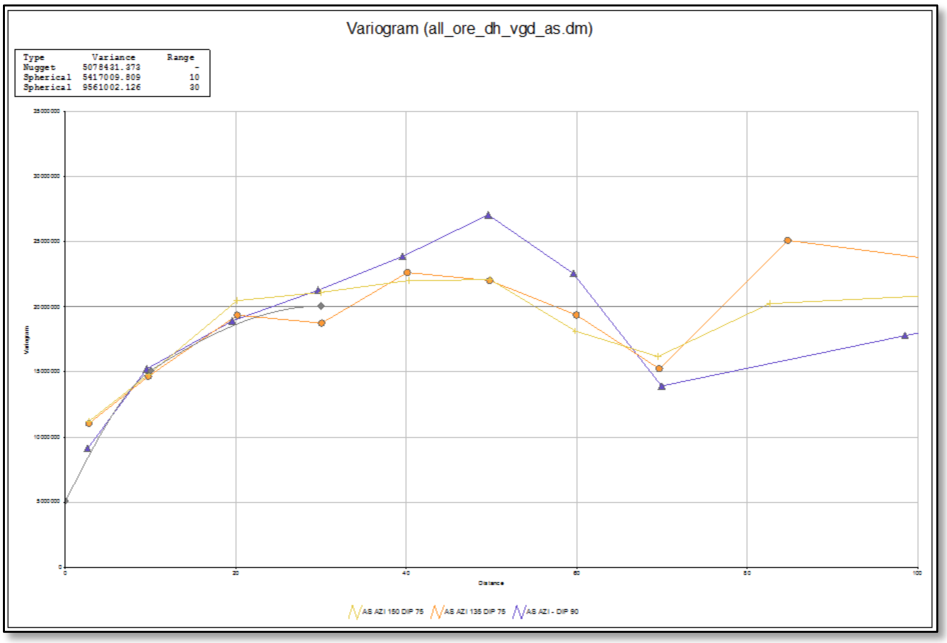


Figure 14-19: Jugan: Dip/Dip Direction Semi-Variogram for As

14.2.6. Previous Resource Estimates

The Jugan deposit has been the subject to a number of historic resource estimates (both internal and public) but the three public, historic resource estimates are the most significant. The following summary of the three public historic resource estimates completed prior to 2010, was extracted from Besra /North Borneo Gold sourced or supplied technical documents.

Some of these historic estimates were prepared pre-NI43-101 and Terra Mining Consultants/Stevens & Associates has neither audited them nor made any attempt to classify them according to NI43-101 standards.

Although some of the more recent resource estimates are purported to have been compiled in terms of the relevant AusIMM JORC Code at that point in time. They are presented because Besra (formally Olympus Pacific Minerals) and Terra Mining Consultants/Stevens & Associates consider them to be relevant and of historic significance.

Also included below are the two most recent estimates, the August 2010 resource definition and audit by Terra Mining Consultants/Stevens & Associates, and the February 2012 resource update by Besra.

- Snowden Mining Industry Consultants (Snowden) for BYG Services Pty Ltd in May 1997. Snowden defined an Indicated Resource (JORC 1996) of 7.724 million tonnes at 1.68 g/t Au. This was estimated using Indicator Kriging method, based on a cut-off of 1.0 g/t Au and the 97.5 percentile mean value for each ore zone was applied as a top cut with an average for all zones being 5.29 g/t (range of 4.51 to 6.82 g/t).
- Scott Andrew McManus (McManus) of Information Geoscience undertook a review and upgrade (JORC 2004) of the Snowden 1997 Resource Estimate in February 2007 for Zedex Ltd. McManus defined an Indicated Resource (JORC 2004) of 4.33 million tonnes at 2.04 g/t Au, using Indicator Kriging and at a cut-off grade of 1.5 g/t.
- John Ashby (Ashby) of Ashby & Associates for Zedex Ltd in October 2008. Ashby defined an Indicated Resource (JORC 2004) of 9.226 million tonnes at 1.66 g/t Au and an Inferred Resource (JORC 2004) of 2.514 million tonnes at 2.20 g/t Au, using a cutoff of 1.0 g/t Au.
- Terra Mining Consultants/Stevens & Associates (TMCSA) for Olympus Pacific Minerals in August 2010. TMCSA defined an Indicated Resource of 10.96 million tonnes at 1.63 g/t, using a 0.75 g/t cutoff.
- February 2012 by Besra (internal update) in February 2012. Besra defined a Measured Resource of 3.425 million tonnes at 1.44 g/t Au, an Indicated Resource of 10.259 million tonnes at 1.52 g/t Au, and an Inferred Resource of 0.507 million tonnes at 1.0 g/t Au, using a 0.5 g/t Au cutoff.

14.2.7. Modelling & Resource Estimation Parameters

The ore zone and intrusive dyke wireframes were generated in Datamine/CAE Mining by Besra/North Borneo Gold staff and validated. These were then filled with block model cells orientated orthogonally and given a separate zone code to differentiate the zones during the estimation process (i.e. no estimation in dyke). The block model parameters are listed in *Table 14-8: Jugan: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	5m x 5m x 2.5m
Zone Code	Ore Zone=1 & Dyke=2
Sub-Cell Size	0.625m x 0.625m x 0.5m

Table 14-8: Jugan: Block Model Parameters

For Jugan all assays within the ore zone volume were used in the estimate (zonal estimation). A top cut of 9.75 g/t Au was applied to all samples above this value. Density values found in the drillholes were used to model the density distribution within the model. The densities were determined using Inverse Distance Squared method with a search radius sufficient to fill the model. The resultant average density determined from this process is 2.64 t/m^3.

Search ellipse and Ordinary Kriging parameters were derived from the variogram analysis and are summarised in *Table 14-9: Jugan: Ordinary Kriging Estimation Parameters* below.

Estimation Parameter	Value
Search Orientation	75° dip at 300° azimuth and 30° plunge
Nugget	0.34
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.29
Sill (Range 2)	0.39
Range 1	7m x 10m x 7m
Range 2	40m x 80m x 40m
Search Volume	Range2 & 2x
Minimum Samples	2 (1)
Maximum Samples	32 (32)

Table 14-9: Jugan: Ordinary Kriging Estimation Parameters

Due to the inconclusive semi-variograms for the other elements these will be interpolated using the inverse distance method. However, the search ellipse and other inverse distance parameters will be as for Au. As some Fe and S data is not consistent throughout the drilling the search ellipse increments were increased in order to fill all cells as per the Au.

14.2.8. Resource & Comparative Estimates

The Ordinary Kriging resource for Jugan was determined at a variety of lower cutoffs. *Table 14-10: Jugan: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CATEGORY	CUTOFF	TONNES	AU	AS	FE	S
Measured	0.25	3,580,500	1.47	0.99	4.33	2.37
	0.5	3,405,600	1.52	1.00	4.32	2.37
	0.75	2,912,300	1.67	1.02	4.31	2.37
	1	2,398,600	1.85	1.05	4.29	2.35
	1.25	1,868,100	2.05	1.07	4.27	2.35
	1.5	1,364,300	2.30	1.09	4.26	2.35
	1.75	946,600	2.60	1.12	4.27	2.36
	2	656,200	2.93	1.16	4.29	2.36
Indicated	0.25	15,168,300	1.46	0.97	4.33	2.41
	0.5	14,505,800	1.51	0.98	4.33	2.41
	0.75	12,994,100	1.61	1.01	4.32	2.40
	1	10,738,400	1.76	1.02	4.31	2.39
	1.25	8,031,400	1.98	1.04	4.30	2.38
	1.5	5,758,900	2.22	1.07	4.30	2.39
	1.75	3,940,100	2.49	1.10	4.30	2.41
	2	2,662,500	2.79	1.12	4.31	2.42
Inferred	0.25	1,787,700	1.57	0.91	4.19	2.37
	0.5	1,774,000	1.57	0.91	4.19	2.37
	0.75	1,734,400	1.59	0.91	4.19	2.37
	1	1,510,000	1.70	0.91	4.13	2.35
	1.25	1,321,300	1.78	0.93	4.15	2.36
	1.5	756,500	2.09	1.05	4.27	2.50
	1.75	608,900	2.20	1.06	4.26	2.50
	2	209,800	2.85	1.08	4.14	2.29

Table 14-10: Jugan: Ordinary Kriging Resource at 0.25 g/t Increments

Previously a cutoff of 0.75 g/t Au was used in the August 2010 resource definition. However, preliminary pit design and costing work identified that the reserve would be potentially less than this value. Therefore in the February 2012 and November 2012 resource estimates a 0.5 g/t Au cutoff was used to define the final resource figure. This prevents the situation of having reserves not in resource.

Although the As, Fe and S are interpolated by Inverse Distance, they are included here with the Ordinary Kriging Au results for completeness sake and reference.

Figure 14-20: Jugan: NW-SE Section through Ordinary Kriging Resource Model below shows a slice through the Jugan gold resource model with the drillholes. Additionally, the ore zone, topography and dyke wireframe outlines are also shown. A plan view slice at -50mRL is also shown in *Figure 14-21: Jugan: Plan View (-50mRL) through Ordinary Kriging Resource Model* . Thirdly, a similar section to the first one is displayed in *Figure 14-22: Jugan: NW-SE Section through Arsenic Inverse Distance Model*, this time showing the arsenic (As) distribution.

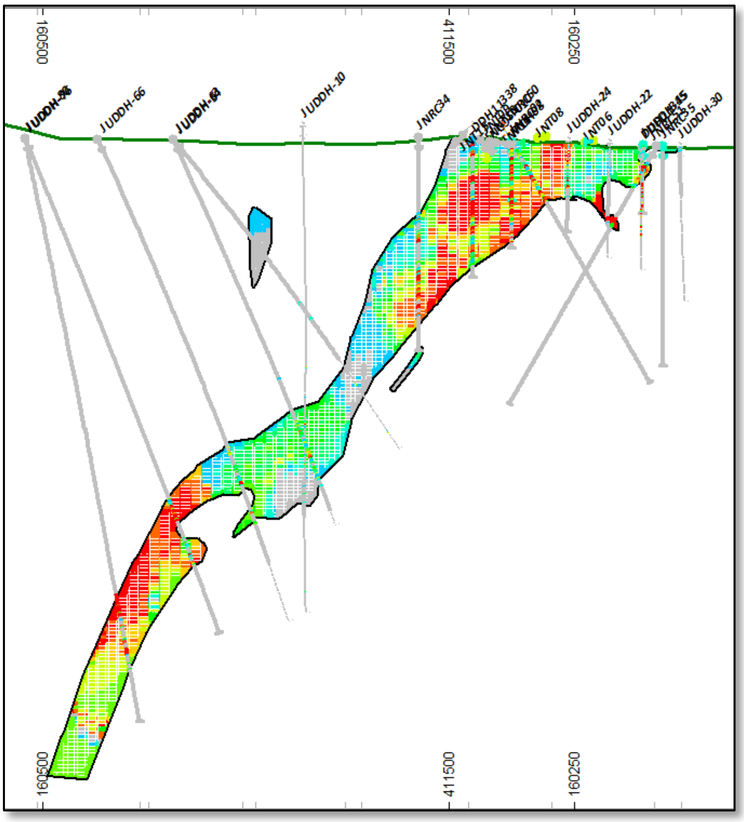


Figure 14-20: Jugan: NW-SE Section through Ordinary Kriging Resource Model

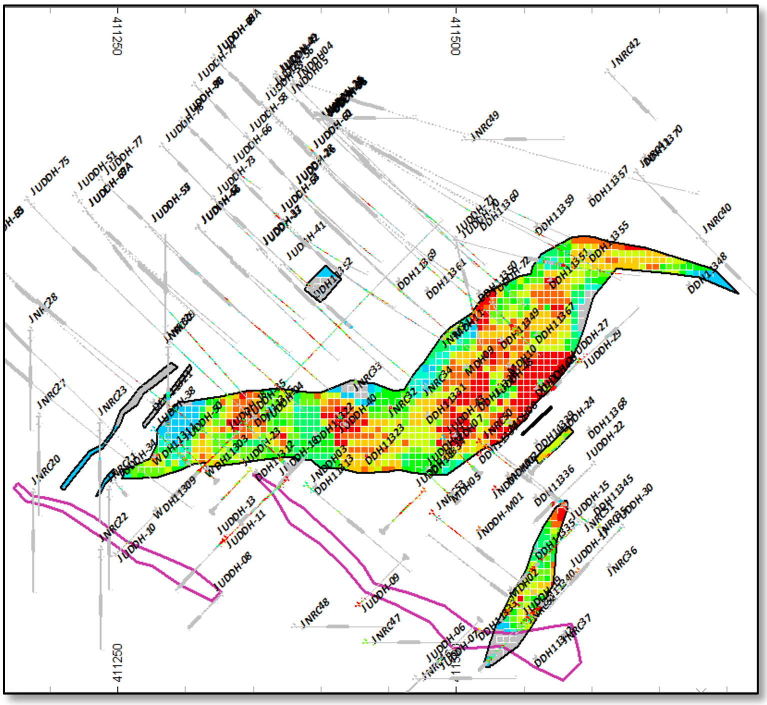


Figure 14-21: Jugan: Plan View (-50mRL) through Ordinary Kriging Resource Model

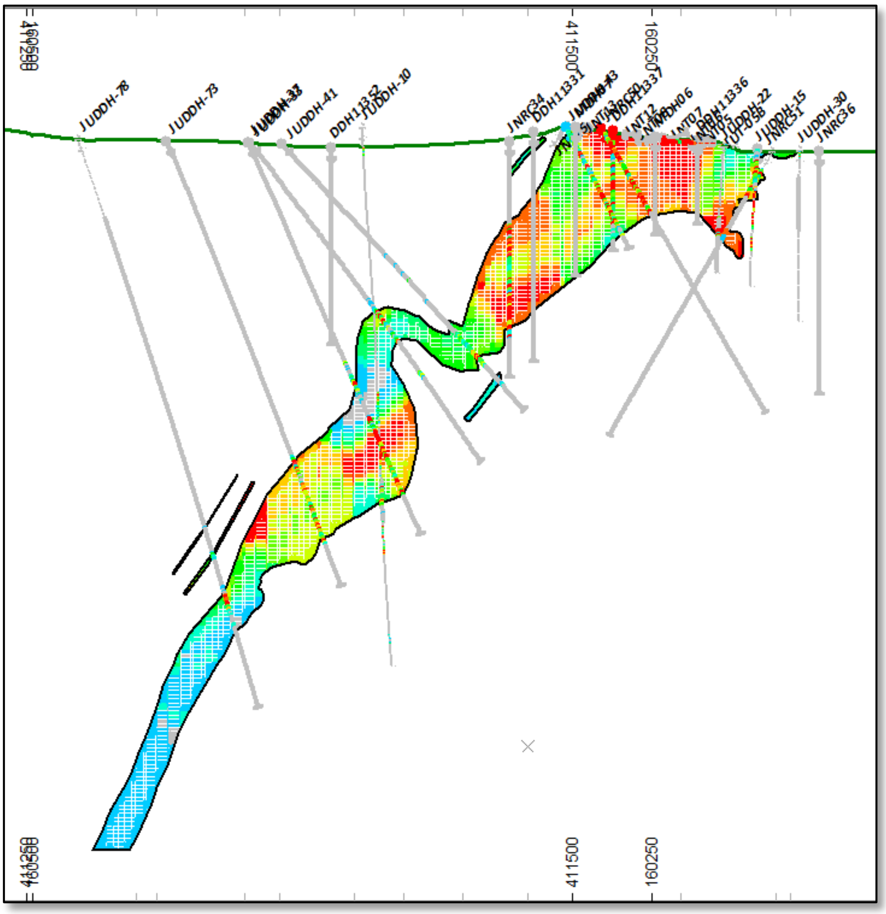


Figure 14-22: Jugan: NW-SE Section through Arsenic Inverse Distance Model

Resource model estimates are adjusted for topography or where excavations (underground and surface) exist. The resource model above topography or within known excavations is removed or subtracted from the final resource estimate.

Comparative estimations were conducted using Inverse Distance Squared and Nearest Neighbour (3D polygonal) methods. The estimation parameters used for these are listed in *Table 14-11: Jugan: Comparative Estimation Method Parameters* below.

Estimation Parameter	Value
Search Orientation	75° dip at 300° azimuth and 30° plunge
Search Ellipse Range	40m x 80m x 40m
Search Volume	Range & 2x
Minimum Samples	2 (1)
Maximum Samples	32 (32)

Table 14-11: Jugan: Comparative Estimation Method Parameters

Listed below, in *Table 14-12: Jugan: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-13: Jugan: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates.

CATEGORY	CUTOFF	TONNES	AU
Measured	0.25	3,580,500	1.46
	0.5	3,405,600	1.52
	0.75	2,912,300	1.67
	1	2,398,600	1.84
	1.25	1,868,100	2.05
	1.5	1,364,300	2.31
	1.75	946,600	2.63
	2	656,200	2.97
Indicated	0.25	15,168,300	1.48
	0.5	14,505,800	1.53
	0.75	12,994,100	1.64
	1	10,738,400	1.80
	1.25	8,031,400	2.02
	1.5	5,758,900	2.27
	1.75	3,940,100	2.56
	2	2,662,500	2.89
Inferred	0.25	1,787,700	1.62
	0.5	1,774,000	1.63
	0.75	1,734,400	1.65
	1	1,510,000	1.75
	1.25	1,321,300	1.82
	1.5	756,500	1.97
	1.75	608,900	2.07
	2	209,800	2.82

Table 14-12: Jugan: Inverse Distance Squared Resource at 0.25 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Measured	0.25	3,580,500	1.48
	0.5	3,405,600	1.54
	0.75	2,912,300	1.70
	1	2,398,600	1.89
	1.25	1,868,100	2.13
	1.5	1,364,300	2.42
	1.75	946,600	2.74
	2	656,200	3.05
Indicated	0.25	15,168,300	1.46
	0.5	14,505,800	1.51
	0.75	12,994,100	1.62
	1	10,738,400	1.79
	1.25	8,031,400	2.02
	1.5	5,758,900	2.29
	1.75	3,940,100	2.61
	2	2,662,500	2.96
Inferred	0.25	1,787,700	2.35
	0.5	1,774,000	2.37
	0.75	1,734,400	2.41
	1	1,510,000	2.59
	1.25	1,321,300	2.81
	1.5	756,500	4.02
	1.75	608,900	4.32
	2	209,800	2.93

Table 14-13: Jugan: Nearest Neighbour Resource at 0.25 g/t Increments

The comparative resource estimates for Jugan compares well with the Ordinary Kriging resource estimate and the minor differences probably reflect the interpolation techniques/application.

The blocks falling within the first search volume with sufficient number of samples has been classified as an Indicated Resource, with blocks in the second search volume classified as Inferred. Blocks outside these two search volumes remain unclassified (geological potential). The upper part of the Indicated Category zone was reviewed and due to the drilling density, number of confirmation holes (including metallurgical drillholes), extensive surface trenching the upper exposed part of the orebody (hill) down to a depth of -20 mRL (top 50-60 m) has been classified as Measured.

The other elements, namely arsenic (As), Iron (Fe) and Sulphur (S); though not part of the resource fall within the same category definitions and will be used for geo-metallurgical considerations during the planning and scheduling to assess the content of these elements produced and delivered to a future plant.

14.3. Bekajang / Krian Sector

This resource section was completed in 2010 and is detailed in the report "Technical Report on Bau Project in Bau, Sarawak, East Malaysia".

Since the above technical report a resource drilling campaign was undertaken in late 2010 and early 2011. The resource drilling campaign was undertaken for only part of the Sector, namely Bukit Young Pit (BYG Pit). The updated resource for this drilling was published in June 2011. The change to the resource was not significant in terms of the increase in resources and therefore a 43-101 report was not issued. The remaining deposits within the Sector remained unchanged at that time except for Krian and Johara which have been combined with the updated BYG Pit to form the new BYG-Krian deposit.

A revision to this resource was made in the February 2012 resource estimate release. No additional drilling or resource modelling was redone to the 2010 and 2011 resource estimates, only a change to the cut-off grade. The reason that the grade was lowered is that some preliminary feasibility work identified the possibility that the reserve cutoff grade could be lower than the resource cutoff grade creating a problem situation where there could be reserves not in resource. The previous cutoff grade was 0.75g/t and this was reduced to 0.5 g/t.

This section will outline the resource changes that occurred in June 2011 to the selected parts of the Sector. The subsequent amendments to the cut-off grade change (0.75 g/t Au to 0.5 g/t Au) for these deposits and the other deposits (where no update work has been undertaken since 2010) was applied in the February 2012 resource release. The data and analysis work for all deposits, whether updated in 2011 or not, is included below for completeness sake.

14.3.1. Introduction & General

The Taiton sector is situated approximately 0.5 kilometres from the town of Bau and is a set of five deposits based on discrete geographical areas as defined by the drilling to date. These deposits have been modelled separately and were Bekajang North, Bekajang South, Johara, Karang Bila and BYG Pit Extension-Krian in the 2010 resource audit. These have been re-organised into Bekajang North, Bekajang South, Karang Bila and BYG-Krian in the 2011 resource update. The tailings dam resource is situated in between the Bekajang North and Bekajang South deposits but has been dealt with separately in another section.

The resource assessment conducted by Terra Mining Consultants/Stevens & Associates in 2010 and by Besra staff in 2011 included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;
- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;
- Combining the above data into a clean and validated resource database with associated data being verified;

- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;
- Resource estimation work to determine the mineral resource using 3 different estimation techniques;

All data used for the 2010 resource definition and the 2011 resource update was supplied or sourced by/from NBG/Besra (formally Olympus Pacific Minerals) or determined by Terra Mining Consultants/Stevens & Associates (2010 resource audit) from available information. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

Historical documents and internal reports were reviewed as part of the resource update. Additionally, numerous notes, plans, sections, memoranda and other documents, both in digital and hardcopy format found in the office library and storage, were reviewed.

14.3.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of these are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;
- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

14.3.3. Ore Zone Definition

The ore zones at Bekajang North, Bekajang South, Johara, Karang Bila & BYG Pit Extension-Krian (2010) and the subsequent combined BYG-Krian ore zone (2011), was defined in the following manner:

- Drillhole sections were created and interpreted faults, geological and mineralized zone grade boundaries (≥0.5 g/t Au lower cut-off) were drawn;
- The grade boundaries were correlated from section to section and cross-checked in plan;
- In the absence of zone continuity, extrapolations were made in between the two drill sections, and up/down dip, using standard methodologies;

- The definition of the mineralized zones and the methodology used was validated visually on each section, and in 3D, and samples within the zone wireframe were analysed;
- The ore zone was terminated using the surveyed topography.
- In the ore zone definition there are isolated cases of assay values below the lower cut-off value. These have only been included where they fall within samples above the cut-off, are of minor effect and cannot be excluded due to their isolated nature.

14.3.4. Statistical Analysis of Data

In the 2010 resource definition, the full Bekajang-Krian database consisted of 690 drillhole collar entries, 791 survey entries, 18,365 assay records, 5,095 density records, and 34,031 lithology records. Drilling conducted at BYG pit in 2011 added a further 42 drillhole collar entries, 379 survey entries, 6,480 assay records, 277 density records and 884 lithology records. This gives a total of 734 drillhole collar entries 1,101 survey entries, 25,168 assay records, 5,372 density records and 34,997 lithology records.

The total data or records also include 2 exploration holes drilled in Bekajang North area during the 2011 drilling campaign. Also note the historic drillholes DDH102-034, DDH102-048, DDH102-058 & DDH102-059 were re-assayed and these records are included in the above totals for the 2011 drilling campaign. Additionally, drillhole alteration, mineralisation, structure, veining, geotechnical, geomechanical and recovery data was also captured for the 2011 drillhole programme in line with NBG logging policies.

A total of 59,027.44 metres of drilling was drilled in and around the Bekajang-Krian sector up to 2010 with an additional 7,159.8 metres drilled in 2011 – including re-drills. The drillhole depths up to 2010 varied from 4 metres to 535.95 metres with an average depth of approximately 85.56 metres. Drilling depths in the 2011 drilling programme varied from 39.2 metres to 389.4 metres with an average depth of 162.3 metres.

The drillholes consisted of 310 RC holes and 380 diamond cored holes in BQ, NQ, HQ & PQ sizes. All drillholes drilled in the 2011 drilling programme were diamond cored and orientated drillholes, and were PQ (collar) and HQ.

The Bekajang North deposit has 64 drillholes, Bekajang South deposit has 128 drillholes, Johara deposit has 15 drillholes, Karang Bila deposit has 16 drillholes and BYG Pit Extension-Krian deposit has 126 drillholes. The 2011 drilling programme added a further 42 drillholes to the combined BYG/Krian/Johara deposit (BYG-Krian) and 2 drillholes to the Bekajang North area. The remaining drillholes fall outside the defined deposits.

A total of 3131 drillhole assay samples fall within the mineralised zone at BYG-Krian. Statistics were calculated for gold, sample length and density fields in the drillhole database within the defined mineralised zones. *Table 14-14: BYG-Krian: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope including the 2011 drillhole data.

Drillhole Field	Length	Au	Density
Number of Records	3,131	3,131	3,131
Number of Samples	3,131	3,023	134
Missing Values	-	108	2,997
Minimum Value	-	-	1.70
Maximum Value	32.86	63.30	3.85
Range	32.86	63.30	2.15
Mean	1.08	2.08	2.54
Variance	1.26	18.76	0.08
Standard Deviation	1.12	4.33	0.28
Standard Error	0.02	0.08	0.02
Skewness	16.74	6.02	-0.70
Kurtosis	372.61	52.37	4.93
Geometric Mean	0.88	0.64	2.52
Sum of Logs	-413.13	-1,342.08	123.81
Mean of Logs	-0.13	-0.45	0.92
Log Variance	1.76	3.43	0.01
Log Estimate of Mean	2.12	3.55	2.54

Table 14-14: BYG-Krian: Ore Zone Drillhole Sample Statistics

The following statistics are from the 2010 resource definition. They are included here for the sake of completeness and easy reference. A total of 757 drillhole assay samples fall within the mineralized zone at Bekajang North. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-15: Bekajang North: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope. Note the 2 exploration drillholes drilled in the 2011 drilling programme are outside the current orebody and are not included here.

Drillhole Field	Length	Au
Number of Records	757	757
Number of Samples	757	757
Missing Values	-	-
Minimum Value	0.10	0.01
Maximum Value	3.10	132.03
Range	3.00	132.03
Mean	0.98	3.23
Variance	0.03	58.01
Standard Deviation	0.18	7.62
Standard Error	0.01	0.28
Skewness	1.45	8.76
Kurtosis	36.52	118.80
Geometric Mean	0.96	0.99
Sum of Logs	-31.35	-4.24
Mean of Logs	-0.04	-0.01
Log Variance	0.06	3.00
Log Estimate of Mean	0.99	4.45

Table 14-15: Bekajang North: Ore Zone Drillhole Sample Statistics

A total of 1,269 drillhole assay samples fall within the mineralized zone at Bekajang South. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-16: Bekajang South: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	1,269	1,269
Number of Samples	1,269	1,269
Missing Values	-	-
Minimum Value	0.01	0.01
Maximum Value	3.50	55.00
Range	3.49	55.00
Mean	1.06	1.73
Variance	0.13	11.87
Standard Deviation	0.37	3.45
Standard Error	0.01	0.10
Skewness	1.25	7.54
Kurtosis	6.76	79.06
Geometric Mean	0.98	0.83
Sum of Logs	- 25.20	- 243.23
Mean of Logs	- 0.02	- 0.19
Log Variance	0.23	1.79
Log Estimate of Mean	1.10	2.02

Table 14-16: Bekajang South: Ore Zone Drillhole Sample Statistics

A total of 149 drillhole assay samples fall within the mineralized zone at Karang Bila. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-17: Karang Bila: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	149	149
Number of Samples	149	149
Missing Values	-	-
Minimum Value	1.00	0.01
Maximum Value	1.00	14.60
Range	-	14.60
Mean	1.00	2.16
Variance	-	8.89
Standard Deviation	-	2.98
Standard Error	-	0.24
Skewness	-	2.55
Kurtosis	-	6.76
Geometric Mean	-	0.79
Sum of Logs	-	- 34.97
Mean of Logs	-	- 0.23
Log Variance	-	3.58
Log Estimate of Mean	-	4.74

Table 14-17: Karang Bila: Ore Zone Drillhole Sample Statistics

Samples within the BYG-Krian ore zone were composited to 1 metre lengths, resulting in 2,116 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-18: BYG-Krian: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for BYG-Krian.

Drillhole Field	Length	Au
Number of Records	2,116	2,116
Number of Samples	2,116	2,116
Missing Values	-	-
Minimum Value	0.50	0.01
Maximum Value	1.00	36.84
Range	0.50	36.84
Mean	0.97	2.38
Variance	0.01	15.21
Standard Deviation	0.10	3.90
Standard Error	0.00	0.08
Skewness	- 3.96	3.86
Kurtosis	14.53	19.23
Geometric Mean	0.97	1.05
Sum of Logs	- 69.76	100.02
Mean of Logs	- 0.03	0.05
Log Variance	0.02	1.91
Log Estimate of Mean	0.98	2.72

Table 14-18: BYG-Krian: Ore Zone Composited Drillhole Sample Statistics

The following drillhole composite statistics are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

Samples within the ore zone were composited to 1 metre lengths, resulting in 743 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-19: Bekajang North: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Bekajang North.

Drillhole Field	Length	Au
Number of Records	743	743
Number of Samples	743	743
Missing Values	-	-
Minimum Value	0.50	0.01
Maximum Value	1.00	83.63
Range	0.50	83.63
Mean	0.99	3.09
Variance	0.00	42.50
Standard Deviation	0.05	6.52
Standard Error	0.00	0.24
Skewness	- 7.51	6.08
Kurtosis	59.53	52.73
Geometric Mean	0.99	0.98
Sum of Logs	- 6.23	- 14.27
Mean of Logs	- 0.01	- 0.02
Log Variance	0.00	2.98
Log Estimate of Mean	0.99	4.35

Table 14-19: Bekajang North: Ore Zone Composited Drillhole Sample Statistics

Samples within the ore zone were composited to 1 metre lengths, resulting in 1,357 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-20: Bekajang South: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Bekajang South.

Drillhole Field	Length	Au
Number of Records	1,357	1,357
Number of Samples	1,357	1,357
Missing Values	-	-
Minimum Value	0.50	0.01
Maximum Value	1.00	55.00
Range	0.50	55.00
Mean	0.98	1.82
Variance	0.01	14.10
Standard Deviation	0.09	3.75
Standard Error	0.00	0.10
Skewness	- 4.85	8.04
Kurtosis	22.57	87.01
Geometric Mean	0.98	0.90
Sum of Logs	- 32.52	- 142.78
Mean of Logs	- 0.02	- 0.11
Log Variance	0.01	1.51
Log Estimate of Mean	0.98	1.92

Table 14-20: Bekajang South: Ore Zone Composited Drillhole Sample Statistics

Samples within the ore zone were composited to 1 metre lengths, resulting in 149 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-21: Karang Bila: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Karang Bila.

Drillhole Field	Length	Au
Number of Records	149	149
Number of Samples	149	149
Missing Values	-	-
Minimum Value	1.00	0.01
Maximum Value	1.00	14.60
Range	-	14.60
Mean	1.00	2.16
Variance	-	8.89
Standard Deviation	-	2.98
Standard Error	-	0.24
Skewness	-	2.55
Kurtosis	-	6.76
Geometric Mean	-	0.79
Sum of Logs	-	- 34.97
Mean of Logs	-	- 0.23
Log Variance	-	3.58
Log Estimate of Mean	-	4.74

Table 14-21: Karang Bila: Ore Zone Composited Drillhole Sample Statistics

The Au data for BYG-Krian shown statistically above is also shown in graphical form below. *Figure 14-23: BYG-Krian: Log Histogram of Au Ore Zone Composites* and *Figure 14-24: BYG-Krian: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

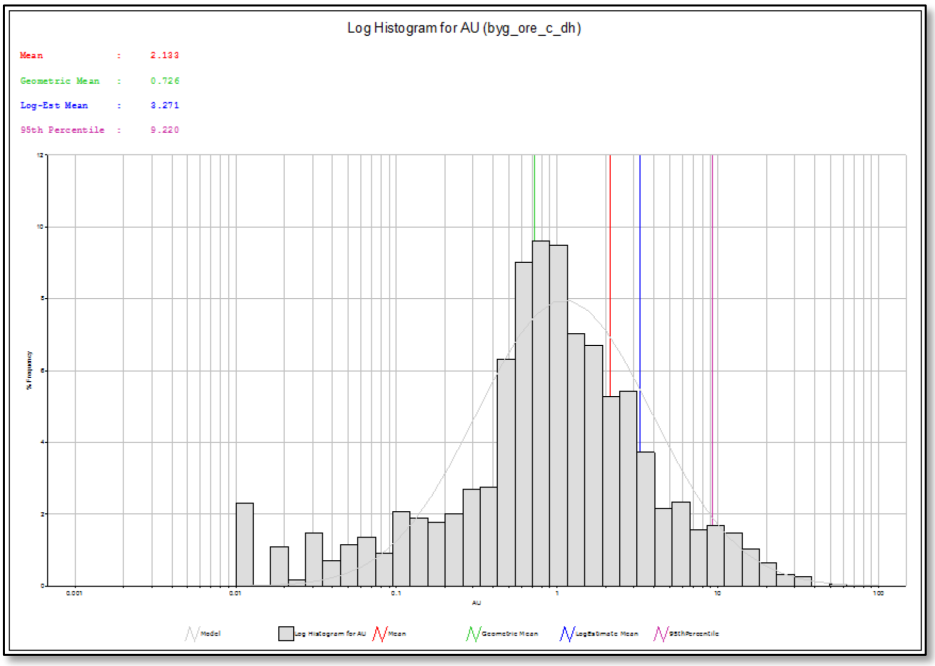


Figure 14-23: BYG-Krian: Log Histogram of Au Ore Zone Composites

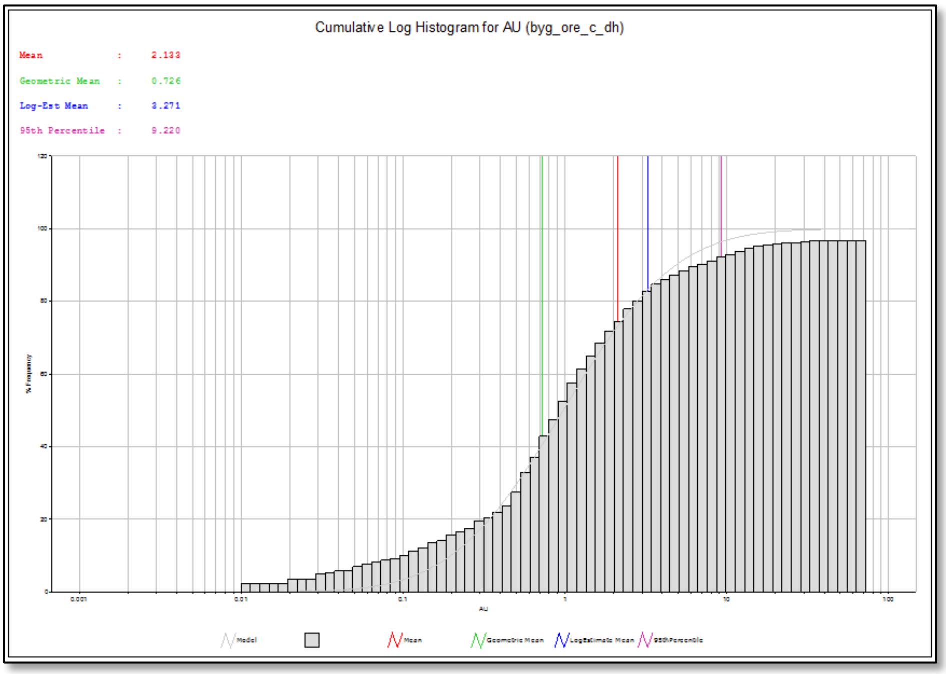


Figure 14-24: BYG-Krian: Cumulative Log Histogram of Au Ore Zone Composites

The following statistical plots and histograms are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

The Au data for Bekajang North shown statistically above is also shown in graphical form below. *Figure 14-25: Bekajang North: Log Histogram of Au Ore Zone Composites* and *Figure 14-26: Bekajang North: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

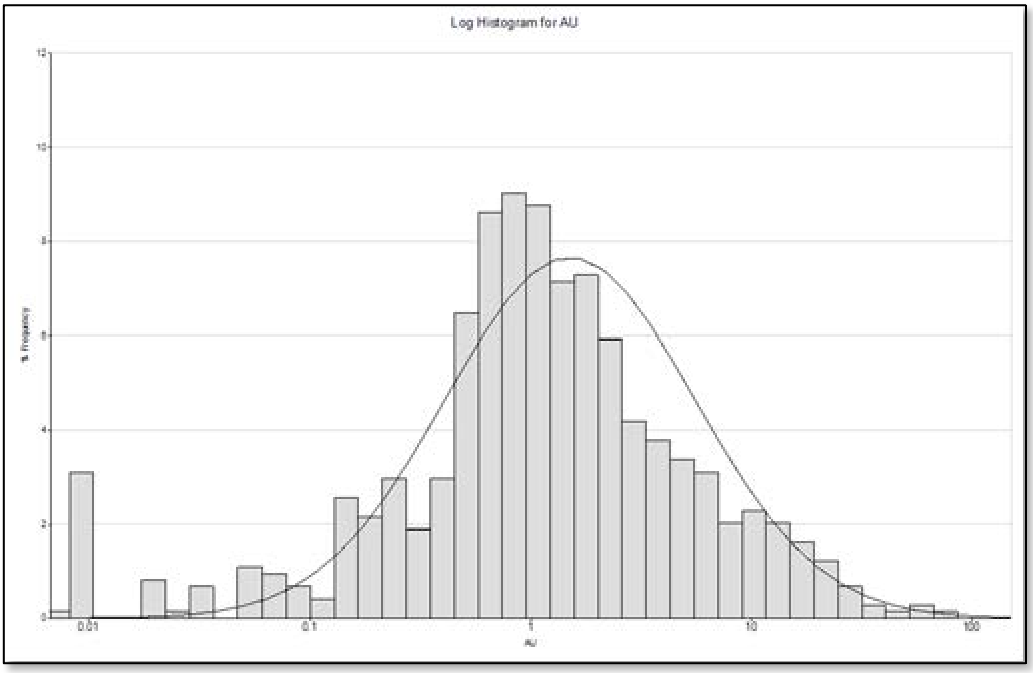


Figure 14-25: Bekajang North: Log Histogram of Au Ore Zone Composites

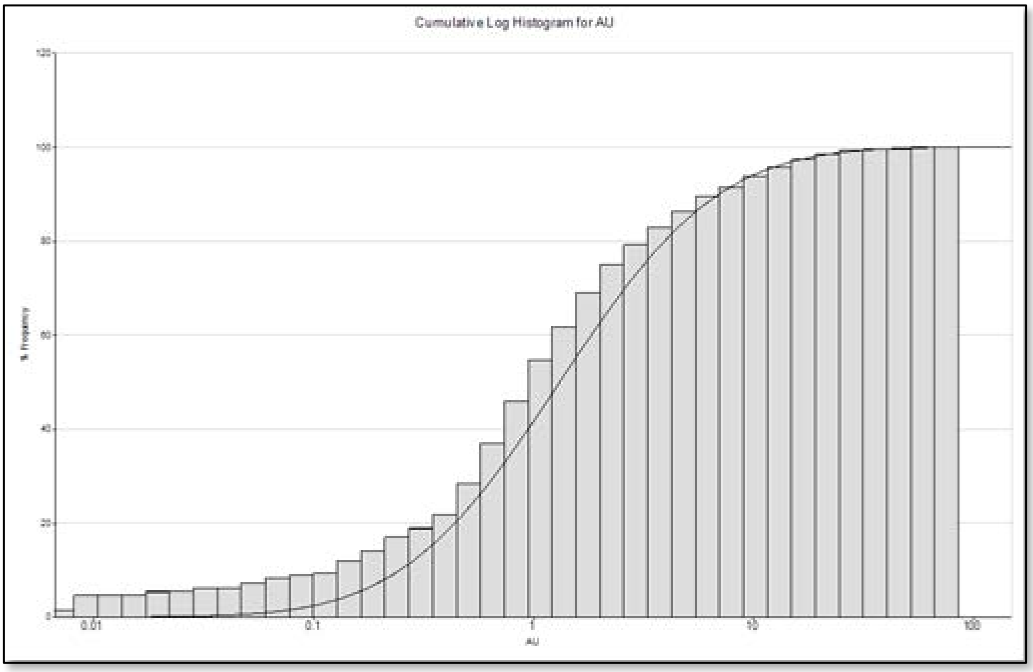


Figure 14-26: Bekajang North: Cumulative Log Histogram of Au Ore Zone Composites

The Au data for Bekajang South shown statistically above is also shown in graphical form below. *Figure 14-27: Bekajang South: Log Histogram of Au Ore Zone Composites* and *Figure 14-28: Bekajang South: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

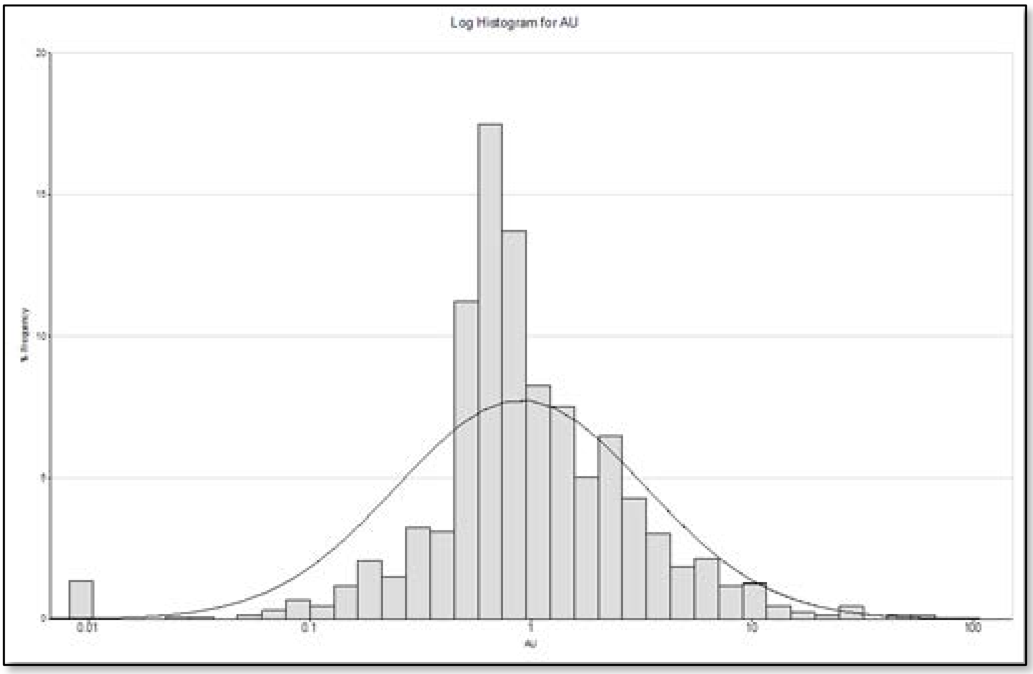


Figure 14-27: Bekajang South: Log Histogram of Au Ore Zone Composites

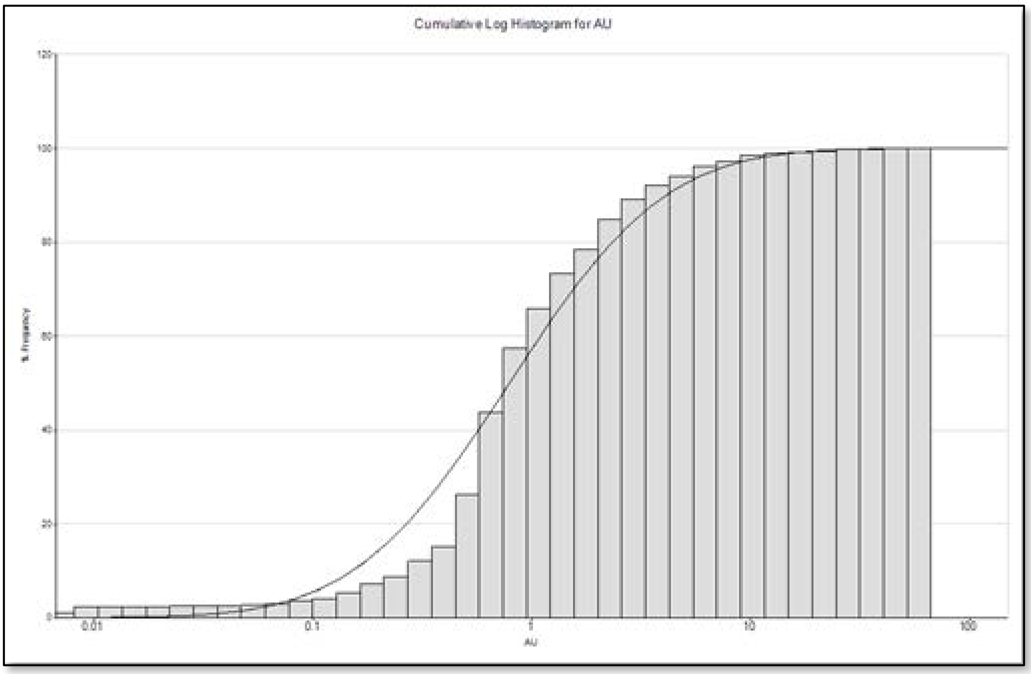


Figure 14-28: Bekajang South: Cumulative Log Histogram of Au Ore Zone Composites

The Au data for Karang Bila shown statistically above is also shown in graphical form below. *Figure 14-29: Karang Bila: Log Histogram of Au Ore Zone Composites* and *Figure 14-30: Karang Bila: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

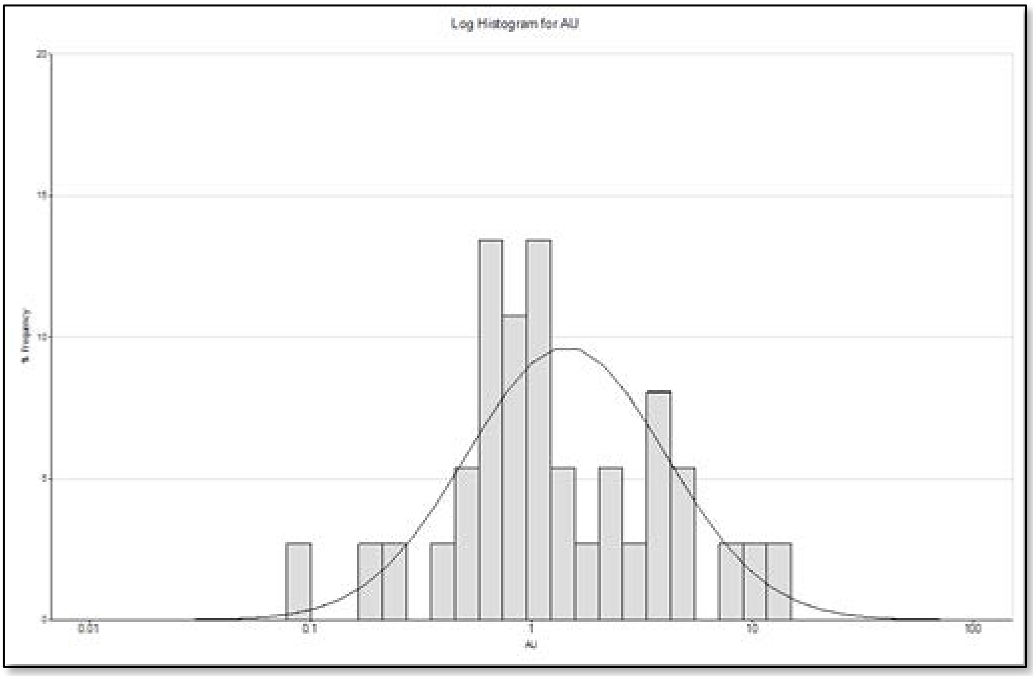


Figure 14-29: Karang Bila: Log Histogram of Au Ore Zone Composites

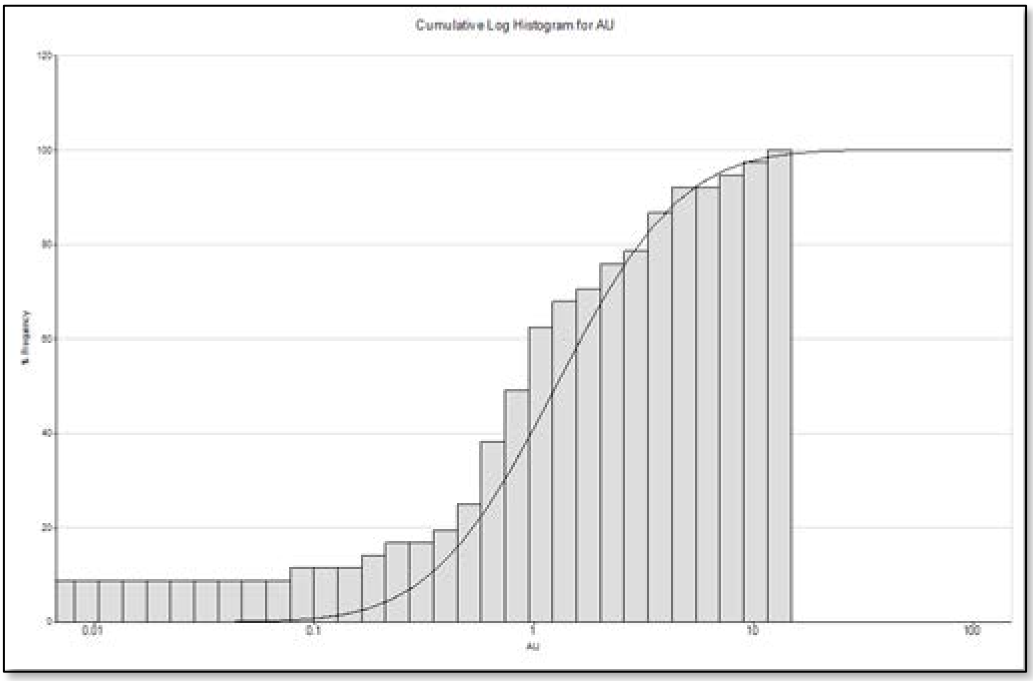


Figure 14-30: Karang Bila: Cumulative Log Histogram of Au Ore Zone Composites

For BYG-Krian resource update a quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-22: BYG-Krian: Quantile Analysis of Au Drillhole Composites* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	330	0.02	-	0.05	5.37	0.08
10	20	331	0.13	0.05	0.24	44.34	0.63
20	30	330	0.39	0.24	0.52	129.54	1.85
30	40	331	0.59	0.52	0.68	196.12	2.80
40	50	331	0.78	0.68	0.89	257.93	3.68
50	60	330	1.01	0.89	1.15	333.66	4.76
60	70	331	1.37	1.15	1.65	453.02	6.46
70	80	330	2.03	1.65	2.49	670.62	9.56
80	90	331	3.32	2.50	4.76	1,098.23	15.66
90	100	331	11.55	4.82	63.30	3,823.12	54.52
90	92.5	82	5.50	4.82	6.27	450.83	6.43
92.5	95	83	7.63	6.27	9.22	633.53	9.04
95	97.5	83	11.22	9.22	13.78	930.89	13.28
97.5	100	83	21.78	13.78	63.30	1,807.87	25.78
0	100	3306	2.12	-	63.30	7,011.96	100.00

Table 14-22: BYG-Krian: Quantile Analysis of Au Drillhole Composites

For BYG-Krian, looking at the primary percentiles, it can be seen that approximately 54.5 % of the metal percentage can be found in the top 10 % range, and that there is a significant jump in the mean grade and metal content from the previous range. Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains nearly 26 % of the Au metal content for BYG-Krian. Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top cut of 21.78 g/t Au (mean of the 97.5 percentile) should be applied to the BYG-Krian samples above this value in order to remove any effect of the high grade samples in the estimation process.

The following quantile analyses are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-23: Bekajang North: Quantile Analysis of Au Drillhole Composites* to *Table 14-25: Karang Bila: Quantile Analysis of Au Drillhole Composites* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Bekajang North, Bekajang South, and Karang Bila ore zones.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	74	0.04	0.01	0.13	2.94	0.13
10	20	74	0.24	0.14	0.40	17.99	0.78
20	30	74	0.52	0.40	0.60	38.64	1.68
30	40	75	0.72	0.60	0.81	53.78	2.34
40	50	74	0.93	0.82	1.06	69.02	3.00
50	60	74	1.27	1.07	1.48	93.71	4.08
60	70	75	1.73	1.48	2.13	129.52	5.64
70	80	74	2.68	2.15	3.52	198.39	8.63
80	90	74	5.19	3.54	7.82	384.17	16.71
90	100	75	17.47	8.02	83.63	1,310.23	57.01
90	92.5	18	8.77	8.02	9.60	157.89	6.87
92.5	95	19	11.21	9.61	12.90	213.08	9.27
95	97.5	19	16.31	13.10	19.70	309.83	13.48
97.5	100	19	33.13	19.80	83.63	629.43	27.39
0	100	743	3.09	0.01	83.63	2,298.40	100.00

Table 14-23: Bekajang North: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	135	0.14	0.01	0.30	19.53	0.79
10	20	136	0.43	0.30	0.52	59.10	2.39
20	30	136	0.56	0.52	0.61	76.47	3.09
30	40	135	0.66	0.61	0.70	88.76	3.59
40	50	136	0.75	0.70	0.82	102.59	4.15
50	60	136	0.91	0.82	1.01	123.52	5.00
60	70	135	1.20	1.01	1.40	161.64	6.54
70	80	136	1.72	1.40	2.13	234.37	9.48
80	90	136	2.72	2.13	3.59	369.59	14.95
90	100	136	9.10	3.60	55.00	1,237.21	50.03
90	92.5	34	3.99	3.60	4.51	135.52	5.48
92.5	95	34	5.42	4.51	6.53	184.30	7.45
95	97.5	34	7.68	6.61	9.37	261.06	10.56
97.5	100	34	19.30	9.49	55.00	656.33	26.54
0	100	1357	1.82	0.01	55.00	2,472.78	100.00

Table 14-24: Bekajang South: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	14	0.01	0.01	0.09	0.16	0.05
10	20	15	0.24	0.09	0.42	3.63	1.13
20	30	15	0.56	0.53	0.61	8.47	2.63
30	40	15	0.70	0.61	0.79	10.43	3.24
40	50	15	0.89	0.79	0.98	13.36	4.15
50	60	15	1.01	0.98	1.06	15.18	4.71
60	70	15	1.49	1.20	1.92	22.40	6.95
70	80	15	2.85	1.92	3.60	42.68	13.25
80	90	15	4.10	3.60	4.80	61.52	19.09
90	100	15	9.63	4.80	14.60	144.40	44.81
90	92.5	3	4.80	4.80	4.80	14.40	4.47
92.5	95	4	8.28	8.28	8.28	33.12	10.28
95	97.5	4	9.62	9.62	9.62	38.48	11.94
97.5	100	4	14.60	14.60	14.60	58.40	18.12
0	100	149	2.16	0.01	14.60	322.23	100.00

Table 14-25: Karang Bila: Quantile Analysis of Au Drillhole Composites

For Bekajang North, looking at the primary percentiles, it can be seen that approximately 57 % of the metal percentage can be found in the top 10 % range, and that there is a significant jump in the mean grade and metal content from the previous range. For Bekajang South this is approximately 50 % and Karang Bila approximately 45 %.

Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains nearly 27 % of the Au metal content for Bekajang North, 27 % for Bekajang South and 18 % for Karang Bila.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top cut of 33.13 g/t Au (mean of the 97.5 percentile) should be applied to the Bekajang North samples above this value in order to remove any effect of the high grade samples in the estimation process. Similarly, a top cut of 19.30 g/t Au for Bekajang South needs be applied. A value of 10.00 g/t Au for Karang Bila was applied as the maximum grade and the mean of the 97.5 percentile are the same, so the value used lies between the 95 and 97.5 percentile.

14.3.5. Semi-Variogram Analysis

Semi-variogram analyses were undertaken to determine the semi-variogram parameters for use in the Ordinary Kriging. Downhole, horizontal and vertical increment semi-variograms were generated with the best semi-variograms selected that defines the strike, dip and dip direction. These semi-variograms were used to determine the nugget, sill values and ranges.

A log semi-variogram and two-range spherical model were used. A best fit model in the downhole semi-variogram was used to define the nugget. Subsequent model fitting was applied to the strike and dip/dip-direction to define the sill values by varying the ranges in these directions. The semi-variogram parameters are listed in *Table 14-26: BYG-Krian: Block Model Parameters* to *Table 14-30: Bekajang South: Ordinary Kriging Estimation Parameters* in *Section 16.1.3.7* below.

The semi-variograms for BYG-Krian are shown below in *Figure 14-31: BYG-Krian: Downhole Semi-Variogram to Figure 14-33: BYG-Krian: Directional Semi-Variogram.*

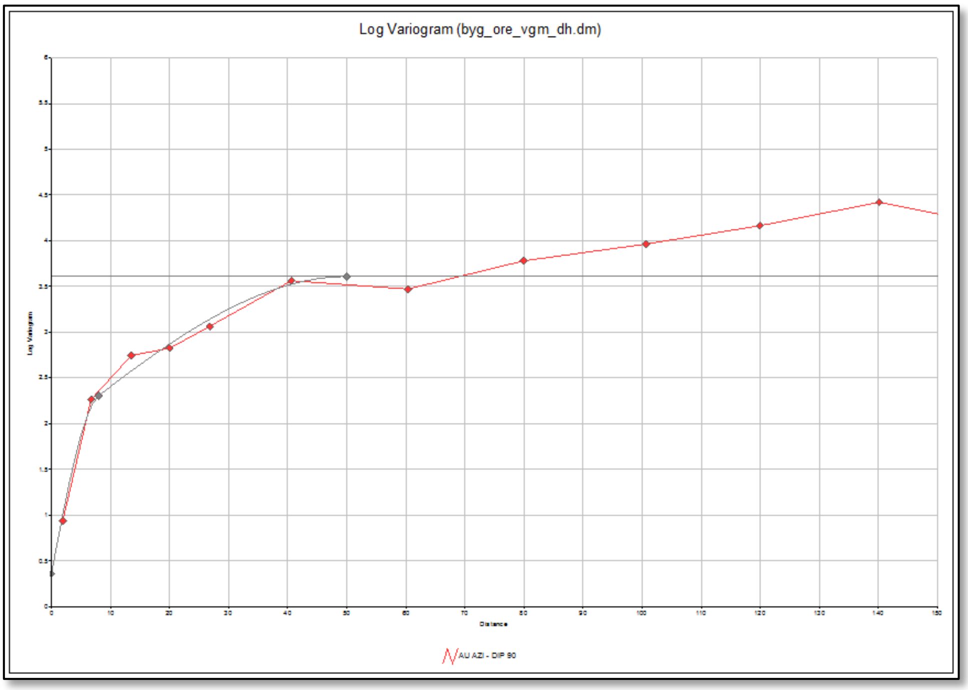


Figure 14-31: BYG-Krian: Downhole Semi-Variogram

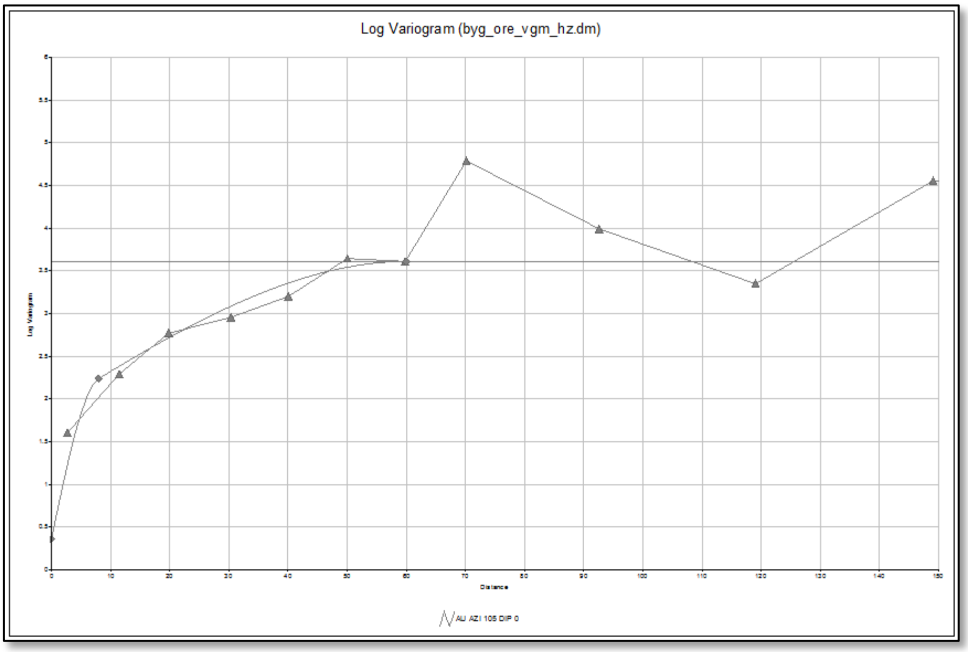


Figure 14-32: BYG-Krian: Horizontal Semi-Variogram

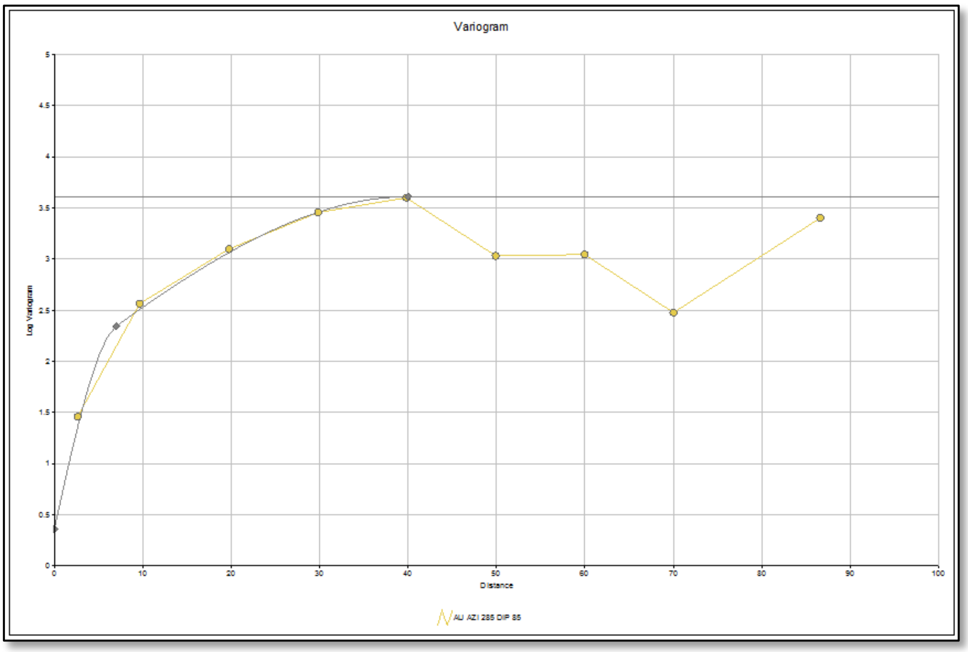


Figure 14-33: BYG-Krian: Directional Semi-Variogram

The following semi-variograms are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

The semi-variograms for Bekajang North are shown below in *Figure 14-34: Bekajang North: Downhole Semi-Variogram* and *Figure 14-35: Bekajang North: Horizontal Semi-Variogram* .

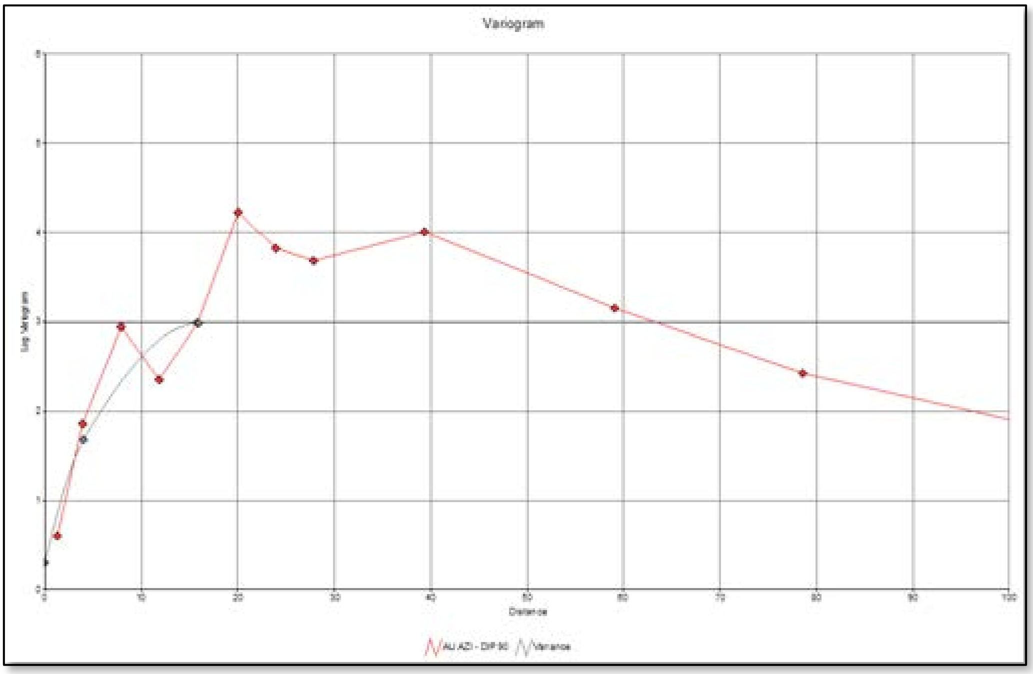


Figure 14-34: Bekajang North: Downhole Semi-Variogram

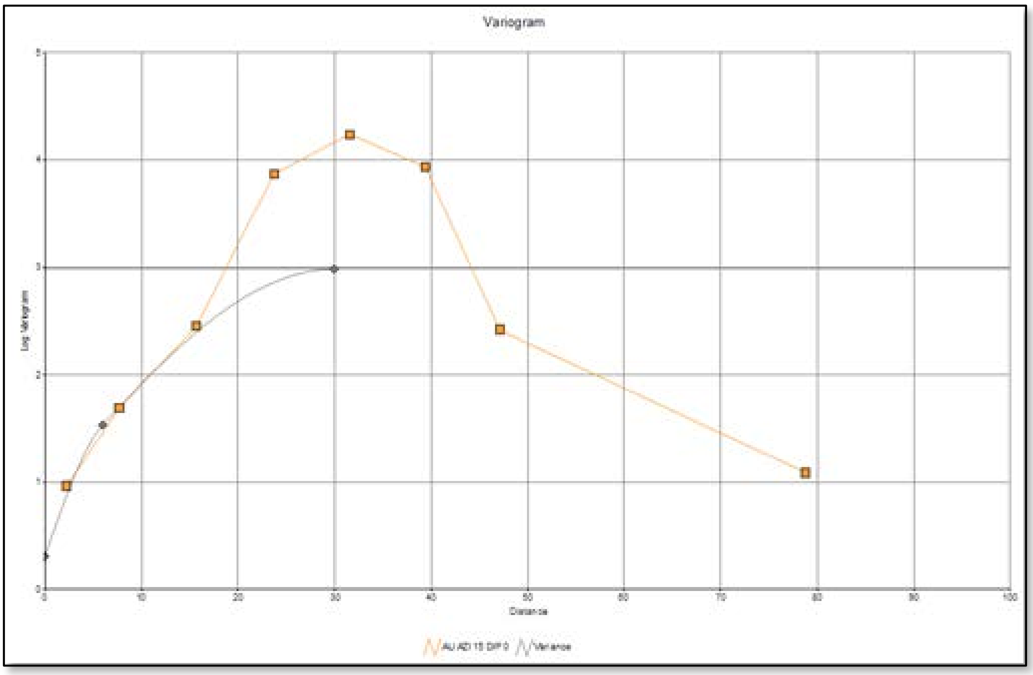


Figure 14-35: Bekajang North: Horizontal Semi-Variogram

The semi-variograms for Bekajang South are shown below in *Figure 14-36: Bekajang South: Downhole Semi-Variogram* to *Figure 14-38: Bekajang South: Alternate Directional Semi-Variogram.*

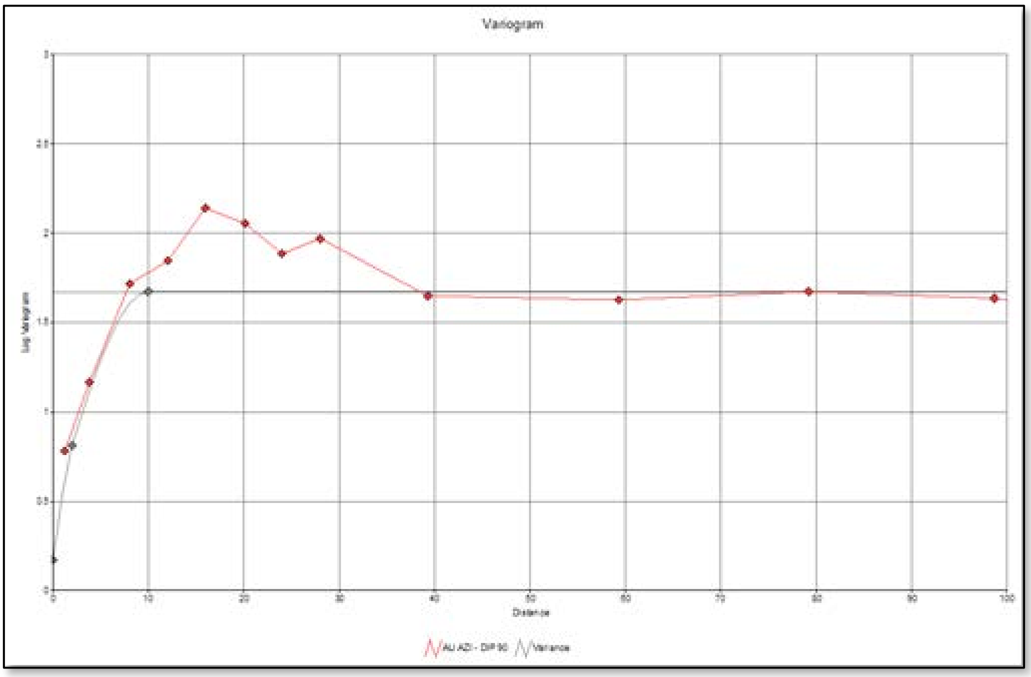


Figure 14-36: Bekajang South: Downhole Semi-Variogram

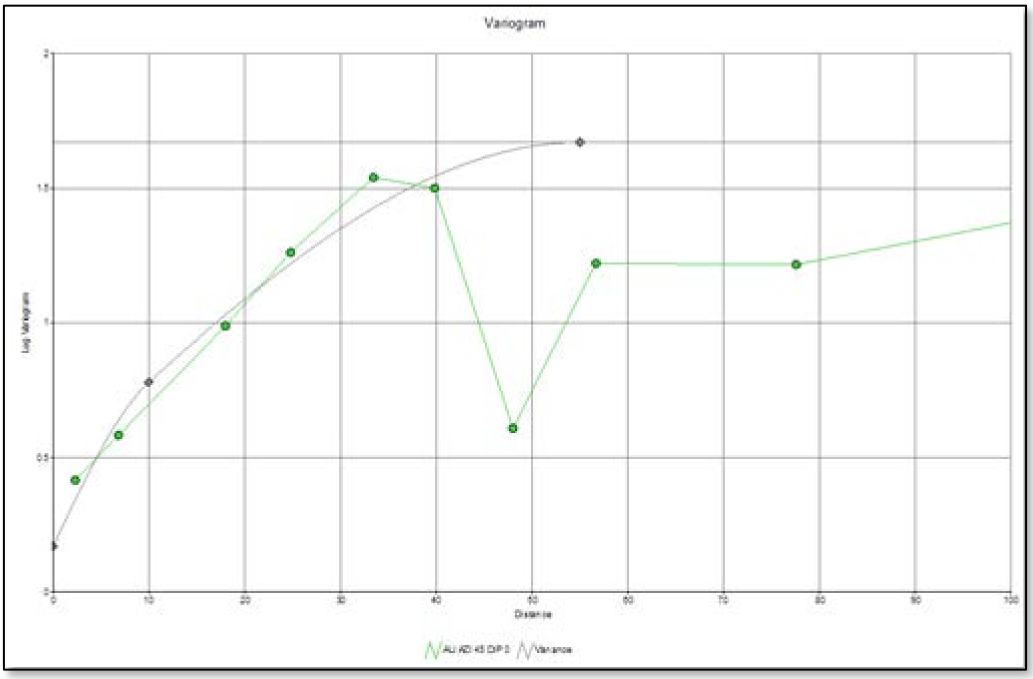


Figure 14-37: Bekajang South: Horizontal Semi-Variogram

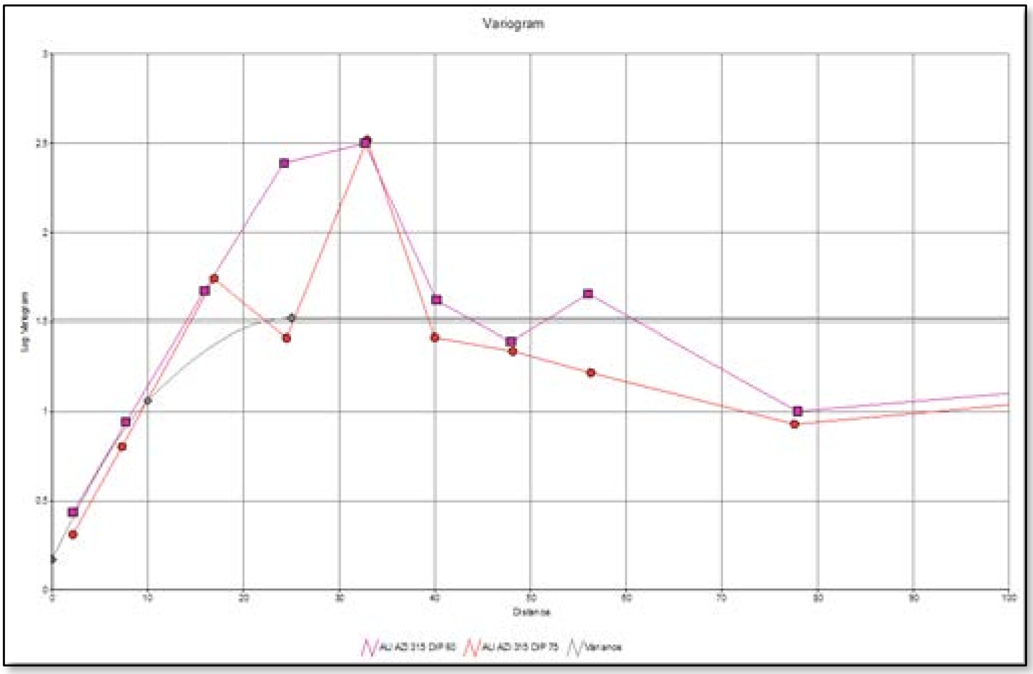


Figure 14-38: Bekajang South: Alternate Directional Semi-Variogram

No adequate semi-variograms were definable for the Karang Bila deposit and these have not been included above. Due to this the Karang Bila deposit was estimated using Inverse Distance Squared method and no Ordinary Kriging was undertaken.

The BYG-Krian modelled log semi-variogram values were back calculated to normal semi-variograms for use with Ordinary Kriging, see *Figure 14-39: BYG-Krian: Log to Normal Semi-Variogram Transform* below.

Converting log to normal variograms:		
Enter the following data:		
log variance :	3.61	**non-standardised**
log nugget :	0.36	0
log sill 1:	1.54	8
log sill 2:	1.71	60
log sill 3:	0.00	
The transformed nugget and sills are:		
nugget:	**0.31**	
sill 1:	**0.33**	
sill 2:	**0.36**	
sill 3:	**0.00**	

Figure 14-39: BYG-Krian: Log to Normal Semi-Variogram Transform

The following back calculations from log to normal semi-variograms are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

The modelled log semi-variogram values were back calculated to normal semi-variograms for use with Ordinary Kriging. The back transform for Bekajang North and Bekajang South are shown in *Figure 14-40: Bekajang North: Log to Normal Semi-Variogram Transform* and *Figure 14-41: Bekajang South: Log to Normal Semi-Variogram Transform* below.

Converting log to normal variograms:		
Enter the following data:		
log variance :	2.98	**non-standardised**
log nugget :	0.30	0
log sill 1:	0.62	5
log sill 2:	2.06	30
log sill 3:	0.00	
The transformed nugget and sills are:		
nugget:	**0.27**	
sill 1:	**0.17**	
sill 2:	**0.56**	
sill 3:	**0.00**	

Figure 14-40: Bekajang North: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:

Enter the following data:

log variance :	1.67	non-standardised
log nugget :	0.17	0
log sill 1:	0.28	10
log sill 2:	1.22	55
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	0.19
sill 1:	0.15
sill 2:	0.66
sill 3:	0.00

Figure 14-41: Bekajang South: Log to Normal Semi-Variogram Transform

No back calculation was done for Karang Bila as no semi-variogram defined and inverse distance used.

14.3.6. Previous Resource Estimates

The Bekajang-Krian sector and deposits has been the subject of only one previous resource estimate. This resource estimate was conducted by Terra Mining Consultants/Stevens and Associates in their 2010 report. The Bekajang North, Bekajang South and Karang Bila resource estimations conducted in 2010 are included in this report in the interests of completeness.

The BYG-Krian 2011 resource update are included as part of this report update. Note this resource update is a combination of the previous BYG Pit-Krian Extension and Johara resource areas as they are reasonably contiguous. The historic estimates for these areas can be found in the 2010 technical report titled "*Technical Report on Bau Project in Bau, Sarawak, East Malaysia*" compiled by Terra Mining Consultants/Stevens & Associates, and are summarised below:

- BYG Pit- Krian Extension 2010 resource as at August 2010 was 3.1 million tonnes at 2.22 g/t Au at a 0.75 g/t Au cutoff, and the Johara resource was 448,000 tonnes at 2.19 g/t Au at a 0.75 g/t Au cutoff.

14.3.7. Modelling & Resource Estimation Parameters

The ore zone wireframes were generated in Gemcom/Datamine/CAE Mining by Besra/North Borneo Gold geological staff and imported into Datamine/CAE Mining and validated. These were then filled with block model cells orientated orthogonally.

The block model parameters for BYG-Krian are listed in *Table 14-26: BYG-Krian: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	4m x 4m x 2m
Zone Code	Ore Zone=1 Ore Zone=2 Ore Zone=3 Ore Zone=4 Ore Zone=5 Ore Zone=6 Ore Zone=7 Ore Zone=8 Ore Zone=9 Ore Zone=10 Ore Zone=11 Ore Zone=12
Sub-Cell Size	0.5m x 0.5m x 0.25m

Table 14-26: BYG-Krian: Block Model Parameters

The following block model parameters are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

The block model parameters for Bekajang North, Bekajang South and Karang Bila are listed in *Table 14-27: Bekajang-Krian: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	5m x 5m x 5m
Zone Code	Ore Zone=1
Sub-Cell Size	0.5m x 0.5m x 0.5m

Table 14-27: Bekajang-Krian: Block Model Parameters

For BYG-Krian all assays within the ore zone volume were used in the estimate (zonal estimation). A top cut of 21.78 g/t Au was applied to all samples above this value for BYG-Krian.

For Bekajang North, Bekajang South and Karang Bila all assays within the ore zone volume were used in the estimate (zonal estimation). A top cut of 33.13 g/t Au was applied to all samples above this value for Bekajang North. Similarly, for Bekajang South a top-cut of 19.30 g/t Au was applied. A value of 10.00 g/t Au for Karang Bila was applied as the maximum grade and the mean of the 97.5 percentile are the same, so the value used lies between the 95 and 97.5 percentile.

Regular and systematic density sampling was conducted during the 2011 drilling, and these density values were interpolated into the block models using the inverse distance technique. This was applied to the BYG-Krian deposit that was updated during 2011.

For the 2010 resource areas there was limited density values were found in the a few drillholes from the Taiton and Bekajang-Krian areas. The average density was determined from these density samples by formation and applied to the Taiton data. The average was 2.594 t/m^3 for Bau Limestone, 2.406 t/m^3 for Intrusive, 2.589 t/m^3 for Krian Sandstone, 2.365 t/m^3 for Pedawan Shale, 1.98 t/m^3 for Quaternary deposits and 2.751 t/m^3 for Serian Volcanics; with a default of 2.5 being applied as required.

Search ellipse and Ordinary Kriging parameters for BYG-Krian were derived from the variogram analysis and are summarised in *Table 14-28: BYG-Krian: Ordinary Kriging Estimation Parameters* below.

Estimation Parameter	Value
Search Orientation	85° dip at 285° azimuth
Nugget	0.31
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.33
Sill (Range 2)	0.36
Range 1	7m x 7m x 8m
Range 2	60m x 40m x 50m
Search Volume	Range 2 & 2x
Minimum Samples	2 (1)
Maximum Samples	32 (32)

Table 14-28: BYG-Krian: Ordinary Kriging Estimation Parameters

The following Ordinary Kriging parameters and search ellipse details are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

Search ellipse and Ordinary Kriging parameters for Bekajang North and Bekajang South were derived from the variogram analysis and are summarised in *Table 14-29: Bekajang North: Ordinary Kriging Estimation Parameters* and *Table 14-30: Bekajang South: Ordinary Kriging Estimation Parameters* below.

Estimation Parameter	Value
Search Orientation	0° dip at 15° azimuth
Nugget	0.27
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.17
Sill (Range 2)	0.56
Range 1	8m x 5m x 5m

Estimation Parameter	Value
Range 2	30m x 30m x 16m
Search Volume	Range 2
Minimum Samples	2
Maximum Samples	32

Table 14-29: Bekajang North: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 45° azimuth
Nugget	0.19
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.15
Sill (Range 2)	0.66
Range 1	10m x 10m x 2m
Range 2	55m x 25m x 10m
Search Volume	Range 2
Minimum Samples	2
Maximum Samples	32

Table 14-30: Bekajang South: Ordinary Kriging Estimation Parameters

Karang Bila resource was estimated by the Inverse Distance Squared and the parameters for this estimation are included *Table 14-38: Karang Bila: Comparative Estimation Method Parameters* in the next Section.

14.3.8. Resource & Comparative Estimates

The resource for BYG-Krian was determined at a variety of lower cutoffs within each resource category (Inferred & Indicated). *Table 14-31: BYG-Krian: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CATEGORY	CUTOFF	TONNES	AU
Inferred	0.5	2,972,000	1.39
	0.75	2,647,000	1.48
	1	1,591,000	1.88
	1.25	1,101,000	2.22
	1.5	817,000	2.52
	1.75	606,000	2.83
	2	500,000	3.03
Indicated	0.5	1,534,000	2.45
	0.75	1,278,000	2.81
	1	1,014,000	3.32
	1.25	839,000	3.78
	1.5	728,000	4.15
	1.75	643,000	4.48
	2	587,000	4.73

Table 14-31: BYG-Krian: Ordinary Kriging Resource at 0.25 g/t Increments

The following Ordinary Kriging resource estimates are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

The resource for Bekajang North was determined at a variety of lower cutoffs. *Table 14-32: Bekajang North: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	1,250,000	2.33
0.75	1,178,000	2.44
1	1,024,000	2.67
1.25	868,000	2.94
1.5	699,000	3.32
1.75	548,000	3.80
2	459,000	4.17

Table 14-32: Bekajang North: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Bekajang South was determined at a variety of lower cutoffs. *Table 14-33: Bekajang South: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	2,294,000	1.60
0.75	1,704,000	1.93
1	1,353,000	2.21
1.25	1,053,000	2.52
1.5	758,000	2.97
1.75	570,000	3.41
2	451,000	3.82

Table 14-33: Bekajang South: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Karang Bila was determined at a variety of lower cutoffs. *Table 14-34: Karang Bila: Inverse Distance Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	774,000	2.56
0.75	637,000	2.98
1	526,000	3.42
1.25	439,000	3.88
1.5	407,000	4.08
1.75	385,000	4.22
2	359,000	4.39

Table 14-34: Karang Bila: Inverse Distance Resource at 0.25 g/t Increments

For the 2010 resources the original cutoff grade of 0.75 g/t Au has been lowered to 0.5 g/t Au in line with potential reserve cutoffs being lower and a review of the statistics. For the 2011 resource update for BYG-Krian a 0.5 g/t Au cutoff was applied.

Figure 14-42: BYG-Krian: W-E Section through Ordinary Kriging Resource Model below shows a slice through the BYG-Krian gold resource model with the drillholes.

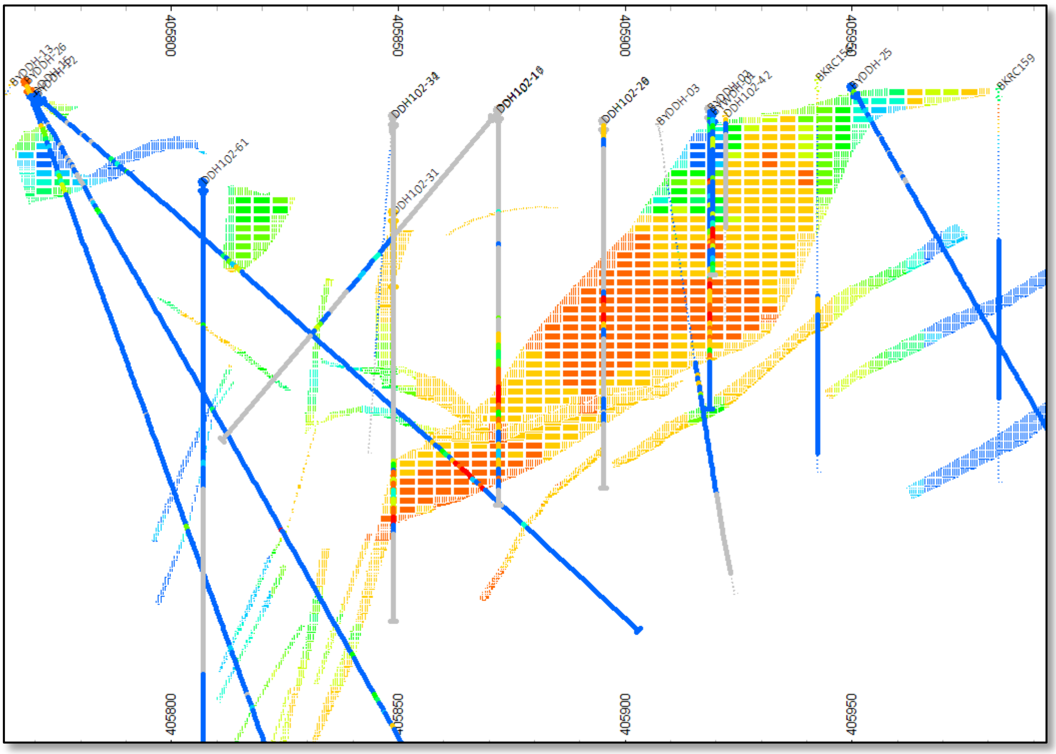


Figure 14-42: BYG-Krian: W-E Section through Ordinary Kriging Resource Model

The following model sections are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

Figure 14-43: Bekajang North: N-S Section through Ordinary Kriging Resource Model below shows a slice through the Bekajang North gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

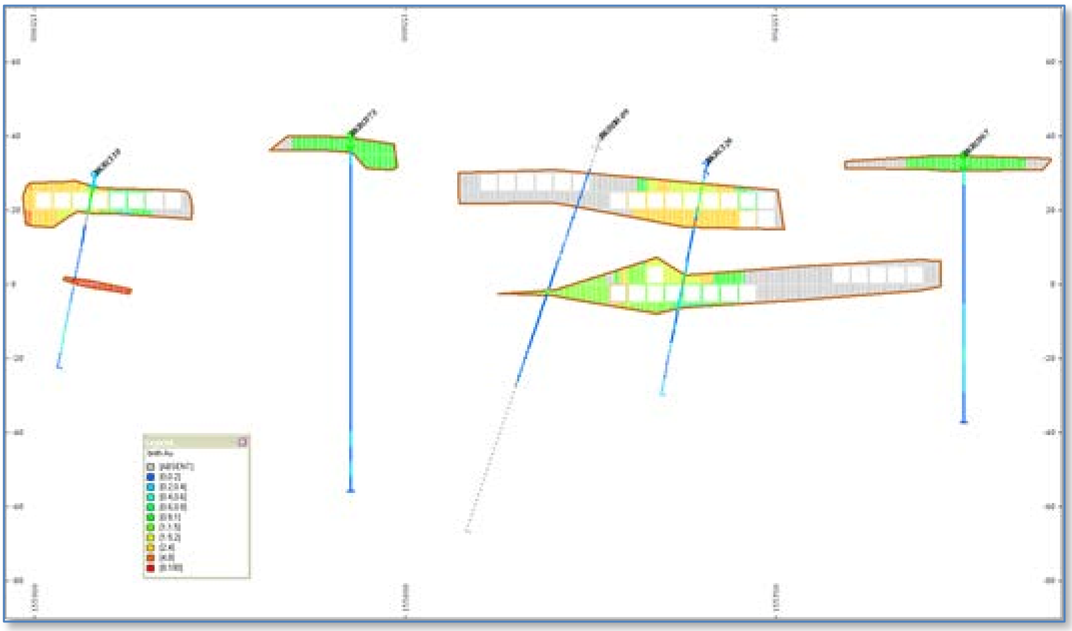


Figure 14-43: Bekajang North: N-S Section through Ordinary Kriging Resource Model

Figure 14-44: Bekajang South: SW-NE Section through Ordinary Kriging Resource Model below shows a slice through the Bekajang South gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

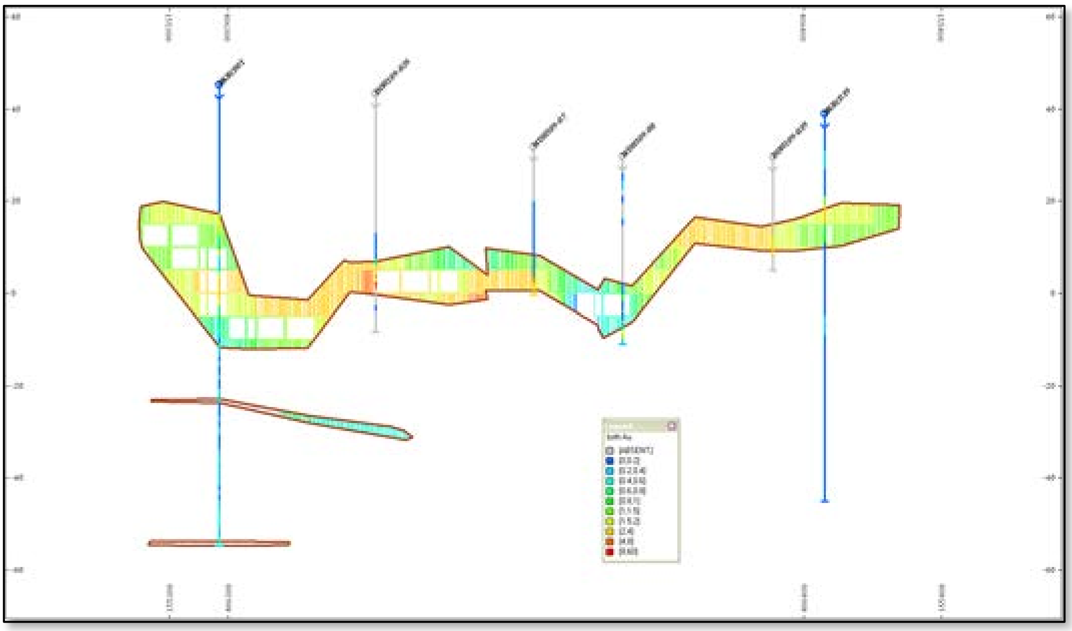


Figure 14-44: Bekajang South: SW-NE Section through Ordinary Kriging Resource Model

Figure 14-45: Karang Bila: SW-NE Section through Inverse Distance Resource Model below shows a slice through the Karang Bila gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

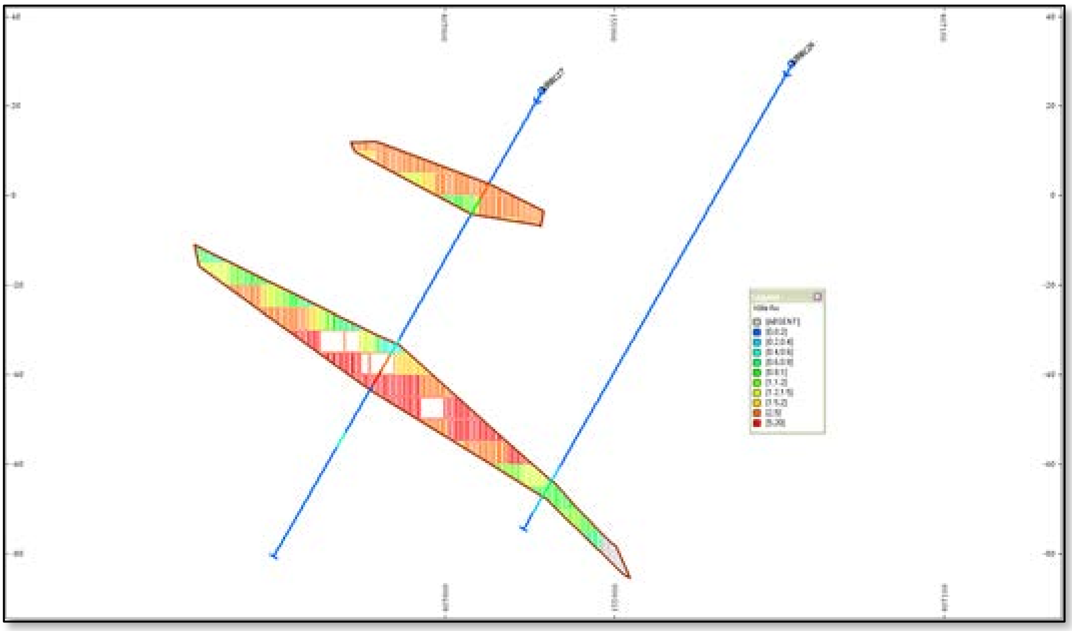


Figure 14-45: Karang Bila: SW-NE Section through Inverse Distance Resource Model

Resource model estimates are adjusted for topography or where excavations (underground and surface) exist. The resource model above topography or within known excavations is removed or subtracted from the final resource estimate. The old Tai Parit and Bukit Young pit topography was used to remove mined ore as modelled from historic drillholes.

Comparative estimations were conducted using the Inverse Distance Squared method. The estimation parameters used for this method are listed below in *Table 14-35: BYG-Krian: Comparative Estimation Method Parameters* for BYG-Krian.

Estimation Parameter	Value
Search Orientation	85° dip at 285° azimuth
Search Ellipse Range	60m x 40m x 50m
Search Volume	Range & 2x
Minimum Samples	2 (1)
Maximum Samples	32 (32)

Table 14-35: BYG-Krian: Comparative Estimation Method Parameters

The following comparative resource estimation parameters are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

Comparative estimations were conducted using Inverse Distance Squared and/or Nearest Neighbour (3D polygonal) methods. The estimation parameters used for these are listed below in *Table 14-36: Bekajang North: Comparative Estimation Method Parameters* for Bekajang North, *Table 14-37: Bekajang South: Comparative Estimation Method Parameters* for Bekajang South and *Table 14-38: Karang Bila: Comparative Estimation Method Parameters* for Karang Bila.

Estimation Parameter	Value
Search Orientation	0° dip at 15° azimuth
Search Ellipse Range	30m x 30m x 16m
Search Volume	Range
Minimum Samples	2
Maximum Samples	32

Table 14-36: Bekajang North: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 45° azimuth
Search Ellipse Range	55m x 25m x 10m
Search Volume	Range
Minimum Samples	2
Maximum Samples	32

Table 14-37: Bekajang South: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	25° dip at 65° azimuth
Search Ellipse Range	40m x 40m x 10m
Search Volume	Range
Minimum Samples	2
Maximum Samples	32

Table 14-38: Karang Bila: Comparative Estimation Method Parameters

Listed below, in *Table 14-39: BYG-Krian: Inverse Distance Squared Resource at 0.25 g/t Increments* is the Inverse Distance comparative estimates for BYG-Krian resource update.

CATEGORY	CUTOFF	TONNES	AU
Inferred	0.5	2,972,000	1.45
	0.75	2,647,000	1.55
	1	1,591,000	1.97
	1.25	1,101,000	2.36
	1.5	817,000	2.70
	1.75	606,000	3.08
	2	500,000	3.31
Indicated	0.5	1,534,000	2.66
	0.75	1,278,000	3.06
	1	1,014,000	3.61
	1.25	839,000	4.12
	1.5	728,000	4.53
	1.75	643,000	4.90
	2	587,000	5.16

Table 14-39: BYG-Krian: Inverse Distance Squared Resource at 0.25 g/t Increments

The following comparative resource estimations are from the 2010 resource definition. They are included here for the sake of completeness and easy reference.

Listed below, in *Table 14-40: Bekajang North: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-41: Bekajang North: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Bekajang North.

CUTOFF	TONNES	AU
0.5	1,249,000	2.41
0.75	1,172,000	2.53
1	1,007,000	2.80
1.25	846,000	3.12
1.5	650,000	3.64
1.75	532,000	4.09
2	428,000	4.63

Table 14-40: Bekajang North: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	1,064,000	2.76
0.75	839,000	3.33
1	662,000	3.98
1.25	582,000	4.38
1.5	498,000	4.88
1.75	403,000	5.65
2	367,000	6.02

Table 14-41: Bekajang North: Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-42: Bekajang South: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-43: Bekajang South: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Bekajang South.

CUTOFF	TONNES	AU
0.5	2,260,000	1.62
0.75	1,658,000	1.98
1	1,336,000	2.25
1.25	1,000,000	2.62
1.5	719,000	3.11
1.75	552,000	3.57
2	438,000	4.01

Table 14-42: Bekajang South: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	2,918,000	1.71
0.75	1,814,000	2.37
1	1,148,000	3.26
1.25	915,000	3.80
1.5	765,000	4.28
1.75	684,000	4.59
2	577,000	5.11

Table 14-43: Bekajang South: Nearest Neighbour Resource at 0.25 g/t Increments

Table 14-44: Karang Bila: Nearest Neighbour Resource at 0.25 g/t Increments lists the Nearest Neighbour comparative estimate for the Karang Bila deposit.

CUTOFF	TONNES	AU
0.5	755,000	2.74
0.75	513,000	3.75
1	434,000	4.27
1.25	388,000	4.65
1.5	363,000	4.86
1.75	339,000	5.10
2.5	328,000	5.21

Table 14-44: Karang Bila: Nearest Neighbour Resource at 0.25 g/t Increments

The BYG-Krian comparative resource estimate compares well with the Ordinary Kriging resource estimates and the minor differences probably reflect the interpolation techniques/application.

The resource has been, divided into and, classified as Indicated and Inferred. An incremental search volume approach was used to determine the resource category in conjunction with the number of samples. Search volume 1 to be set to Indicated category and volume 2 to be set to Inferred category. Upon review this was modified as follows:

- Area of Indicated category material reduced to area of 2011 drilling with areas outside reset to Inferred category. Rationale is that areas outside 2011 drilling had not been verified by recent drilling and associated information not in historic data (e.g. Density) not present;

- Area of Indicated category material below the -55 metre elevation is to be reset to Inferred category. Rationale is that below this level the combination of historic drilling and recent drilling density is not sufficient.

The comparative resource estimates for Bekajang North and Bekajang South compare well with the Ordinary Kriging resource estimates and the minor differences probably reflect the interpolation techniques/application. In the case of Karang Bila the comparison with the Inverse Distance resource estimate also compares well considering the estimation technique differences.

The resource has been classified as Inferred. Some areas of the deposit(s) could potentially have been classified as Indicated based purely on the drilling density. However, one or more of the following issues gave rise to an Inferred classification:

- Large number of RC drillholes with few diamond core holes;
- Smaller drillhole sizes in some instances (e.g. BQ);
- Lack of extensive and systematic density determinations throughout the deposit;
- Gaps in the drillhole spacing or coverage and/or larger distances between drillholes;
- Difficulty in domaining of the data to remove possible mixed populations in some instances.

14.4. Sirenggok Sector

This resource section was completed in 2010 and is detailed in report *"Technical Report on Bau Project in Bau, Sarawak, East Malaysia"*. It has been included here for the sake of completeness.

A revision to this resource was made in the February 2012 resource estimate release. No changes or modelling was redone only a change to the cut-off grade and therefore the following information is still valid. The reason that the grade was lowered is that some preliminary feasibility work identified the possibility that the reserve cutoff grade could be lower than the resource cutoff grade creating a problem situation where there could be reserves not in resource. The previous cutoff grade was 0.75 g/t Au and this was reduced to 0.5 g/t Au.

14.4.1. Introduction & General

The Sirenggok deposit is situated approximately 1.5 kilometres from the town of Bau and is a single deposit.

The resource assessment conducted by Terra Mining Consultants/Stevens & Associates included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;
- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;

- Combining the above data into a clean and validated resource database with associated data being verified;
- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;
- Resource estimation work to determine the mineral resource using 3 different estimation techniques;

All data used for this resource update was supplied or sourced by Olympus/North Borneo Gold or determined by Terra Mining Consultants/Stevens & Associates from available information. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

Historical documents and reports were reviewed as part of the resource update and these are listed below and in *Section 27 - References*. Additionally, numerous notes, plans, sections, memoranda and other documents, both in digital and hardcopy format found in the office library and storage, were reviewed.

- Review of Ashby & Associates, June 2008 preliminary draft report (incomplete) titled "Investigation of the Sirenggok Database".

14.4.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of these are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;
- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

14.4.3. Ore Zone Definition

The ore zone at Sirenggok was defined in the following manner:

- Drillhole sections were created and interpreted faults, geological and mineralized zone grade boundaries (≥0.5 g/t Au lower cut-off) were drawn;
- The grade boundaries were correlated from section to section and cross-checked in plan;

- In the absence of zone continuity, extrapolations were made in between the two drill sections, and up/down dip, using standard methodologies;
- The definition of the mineralized zones and the methodology used was validated visually on each section, and in 3D, and samples within the zone wireframe were analysed;
- The ore zone was terminated using the surveyed topography.

In the ore zone definition there are isolated cases of assay values below the lower cut-off value. These have only been included where they fall within samples above the cut-off, are of minor effect and cannot be excluded due to their isolated nature.

14.4.4. Statistical Analysis of Data

The full Sirenggok database consisted of 72 drillhole collar entries, 119 collar survey entries, 6,351 assay records, 20 density records, and 3,061 lithology records; and 39 trench/costean collar records, 1,616 trench/costean survey entries and 1,619 trench/costean assay records.

A total of 11,163.10 metres of drilling was drilled in and around the Sirenggok deposit. The drillhole depths varied from 6 metres to 489.55 metres with an average depth of approximately 155.04 metres. The drillholes consisted of 13 RC holes, 3 diamond cored holes pre-collared by RC drilling and 56 fully diamond cored holes in BQ, NQ, HQ & PQ sizes.

A total of 1,174 metres of trenching and costeaning was undertaken within the mineralised zone. Some trenching/costeaning occurred outside this mineralised zone and is not included. The trenches/costeans varied in length from 2 to 43 metres with an average length of 65.68 metres.

A total of 2,881 combined drillhole and trench/costean assay samples fall within the mineralized zone at Sirenggok. Statistics were calculated in Datamine for gold, density and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-45: Sirenggok: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	2,881	2,881
Number of Samples	2,881	2,880
Missing Values	-	1
Minimum Value	0.02	0.01
Maximum Value	4.50	33.40
Range	4.48	33.40
Mean	1.28	0.94
Variance	0.20	2.72
Standard Deviation	0.44	1.65
Standard Error	0.01	0.03
Skewness	0.71	7.62
Kurtosis	2.13	109.13

Drillhole Field	Length	Au
Geometric Mean	1.19	0.38
Sum of Logs	508.10	-2,819.96
Mean of Logs	0.18	-0.98
Log Variance	0.17	2.33
Log Estimate of Mean	1.30	1.20

Table 14-45: Sirenggok: Ore Zone Drillhole Sample Statistics

Samples within the orezone were composited to 1 metre lengths, resulting in 3,705 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-46: Sirenggok: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Sirenggok.

Drillhole Field	Length	Au
Number of Records	3,705	3,705
Number of Samples	3,705	3,703
Missing Values	-	2
Minimum Value	0.50	0.01
Maximum Value	1.00	32.20
Range	0.50	32.20
Mean	0.99	0.90
Variance	0.00	2.11
Standard Deviation	0.06	1.45
Standard Error	0.00	0.02
Skewness	-8.39	6.95
Kurtosis	69.57	92.99
Geometric Mean	0.99	0.40
Sum of Logs	-34.06	-3,424.04
Mean of Logs	-0.01	-0.92
Log Variance	0.01	2.11
Log Estimate of Mean	0.99	1.14

Table 14-46: Sirenggok: Ore Zone Composited Drillhole Sample Statistics

The Au data shown statistically above is also shown in graphical form below. *Figure 14-46: Sirenggok: Log Histogram of Au Ore Zone Composites* and *Figure 14-47: Sirenggok: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine.

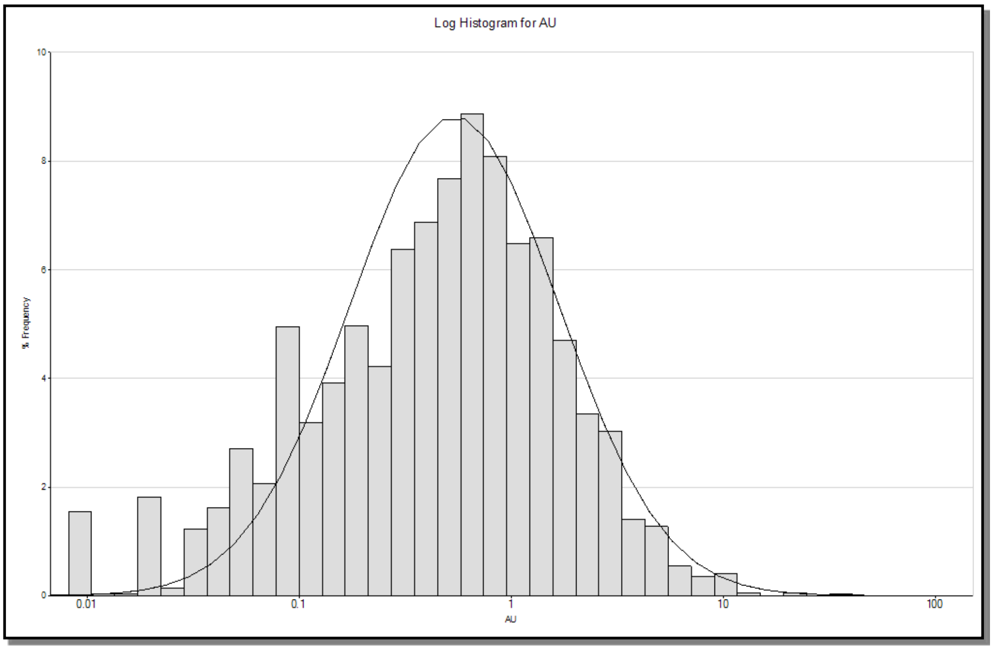


Figure 14-46: Sirenggok: Log Histogram of Au Ore Zone Composites

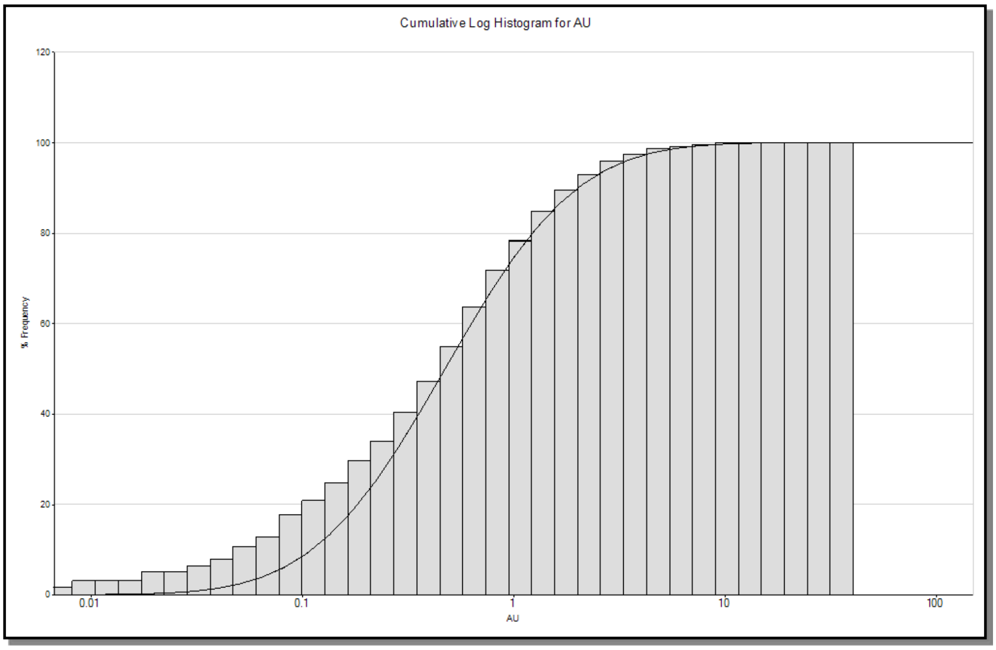


Figure 14-47: Sirenggok: Cumulative Log Histogram of Au Ore Zone Composites

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-47: Sirenggok: Quantile Analysis of Au Drillhole Composites* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Sirenggok Ore Zone.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	370	0.03	0.01	0.06	10.14	0.30
10	20	370	0.09	0.06	0.12	32.86	0.98
20	30	370	0.17	0.12	0.22	62.35	1.86
30	40	371	0.28	0.22	0.35	104.95	3.14
40	50	370	0.43	0.35	0.50	157.28	4.70
50	60	370	0.58	0.50	0.66	213.99	6.40
60	70	371	0.78	0.67	0.91	288.18	8.62
70	80	370	1.09	0.91	1.30	401.64	12.01
80	90	370	1.62	1.30	2.08	598.40	17.89
90	100	371	3.97	2.08	32.20	1,474.47	44.09
90	92.5	92	2.29	2.08	2.54	210.33	6.29
92.5	95	93	2.72	2.54	2.93	253.35	7.58
95	97.5	93	3.56	2.94	4.44	331.04	9.90
97.5	100	93	7.31	4.47	32.20	679.75	20.33
0	100	3703	0.90	0.01	32.20	3,344.27	100.00

Table 14-47: Sirenggok: Quantile Analysis of Au Drillhole Composites

Looking at the primary percentiles, it can be seen that approximately 44 % of the metal percentage can be found in the top 10 % range (top 371 samples), and that there is a significant jump in the mean grade and metal content from the previous range. Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains approximately 20 % of the Au metal content.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top-cut of 7.31 g/t (mean of the 97.5 percentile) should be applied to the samples above this value in order to remove any effect of the high grade samples in the estimation process.

14.4.5. Semi-Variogram Analysis

Semi-variogram analyses were undertaken to determine the semi-variogram parameters for use in the Ordinary Kriging. Downhole, horizontal and vertical increment semi-variograms were generated with the best semi-variograms selected that defines the strike, dip and dip direction. These semi-variograms were used to determine the nugget, sill values and ranges.

A log semi-variogram and two-range spherical model were used. A best fit model in the downhole semi-variogram was used to define the nugget. Subsequent model fitting was applied to the strike and dip/dip-direction to define the sill values by varying the ranges in these directions. The semi-variogram parameters are listed in Table 79 - Sirenggok: Ordinary Kriging Estimation Parameters in Section 16.4.7 below

The semi-variograms for Sirenggok are shown below in *Figure 14-48: Sirenggok: Strike/Downhole Semi-Variogram* and *Figure 14-49: Sirenggok: Dip Direction/Downhole Semi-Variogram*.

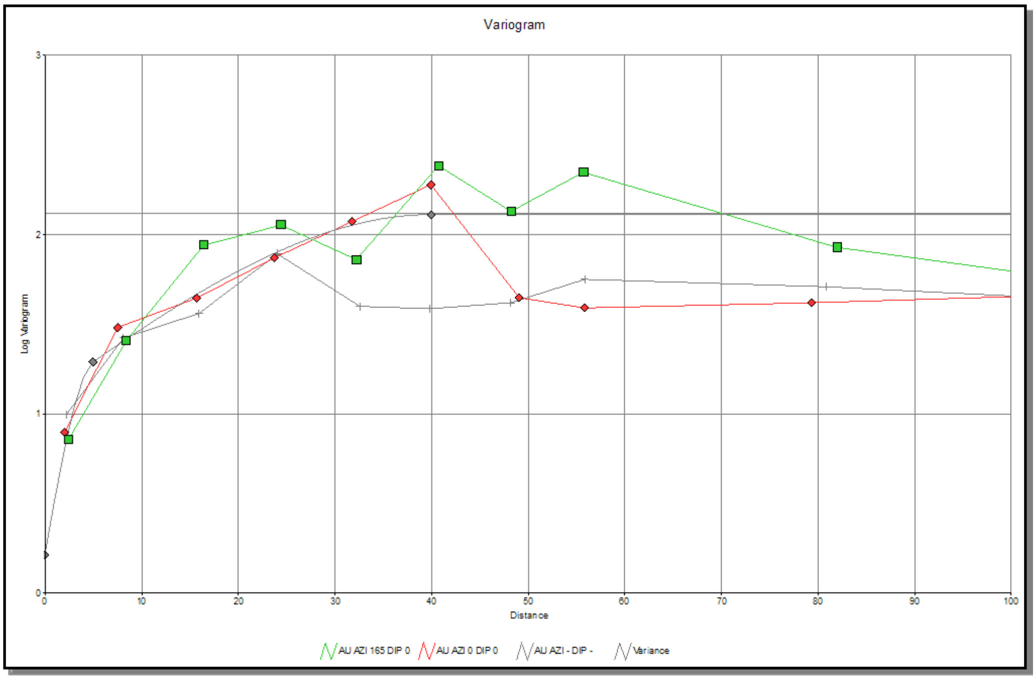


Figure 14-48: Sirenggok: Strike/Downhole Semi-Variogram

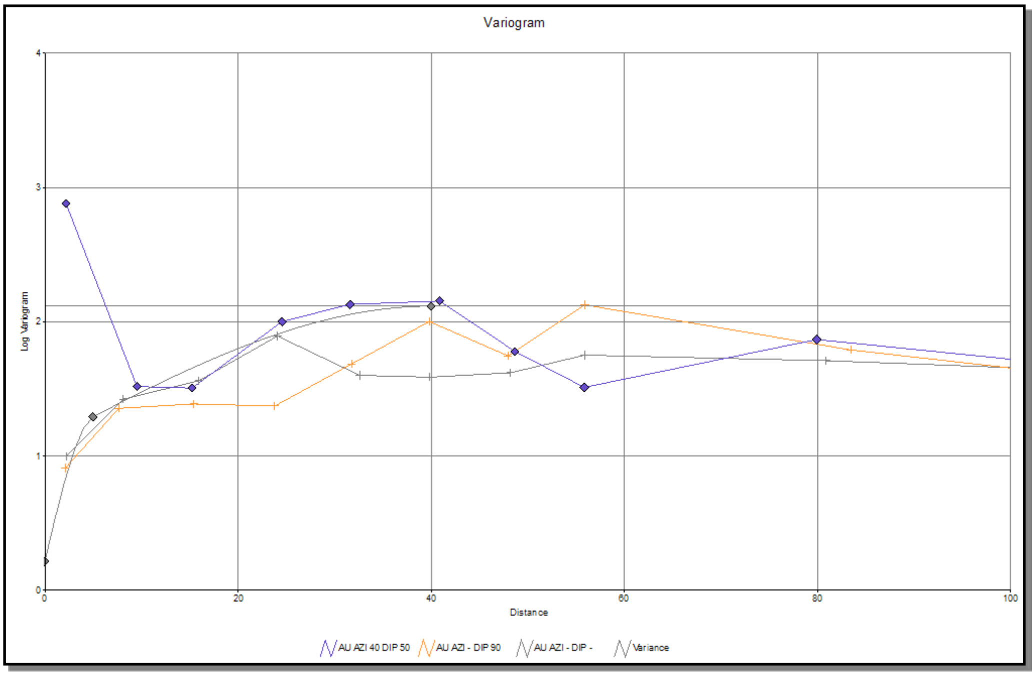


Figure 14-49: Sirenggok: Dip Direction/Downhole Semi-Variogram

The modelled log semi-variogram values were back calculated to normal semi-variograms for use with Ordinary Kriging. The back transform is shown in *Figure 14-50: Sirenggok: Log to Normal Semi-Variogram Transform* below.

Converting log to normal variograms:

Enter the following data:

log variance :	2.11	**non-standardised**
log nugget :	0.21	0
log sill 1:	0.89	5
log sill 2:	1.01	40
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.22**
sill 1:	**0.37**
sill 2:	**0.42**
sill 3:	**0.00**

Figure 14-50: Sirenggok: Log to Normal Semi-Variogram Transform

14.4.6. Previous Resource Estimates

The Sirenggok deposit has been the subject to a number of historic resource estimates (both internal and public) but the single public resource estimates is the most significant. The following summary of the single public, historic resource estimate completed prior to 2010, was extracted from Olympus/North Borneo Gold sourced or supplied technical documents. Some of these historic estimates were prepared pre-NI43-101 and Terra Mining Consultants/Stevens & Associates has neither audited them nor made any attempt to classify them according to NI43-101 standards. Although some of the more recent resource estimates are purported to have been compiled in terms of the relevant AusIMM JORC Code at that point in time. They are presented because Olympus and Terra Mining Consultants/Stevens & Associates consider them to be relevant and of historic significance.

- John Ashby (Ashby) of Ashby & Associates for Zedex Ltd in October 2008. Ashby defined an Inferred Resource (JORC 2004) of 8.702 million tonnes at 1.109 g/t Au, using a cutoff of 0.75 g/t Au.

14.4.7. Modelling & Resource Estimation Parameters

The ore zone wireframes were generated in Gemcom by Olympus/North Borneo Gold staff and imported into Datamine and validated. These were then filled with block model cells orientated orthogonally. The block model parameters are listed in *Table 14-48: Sirenggok: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	10m x 10m x 5m
Zone Code	Ore Zone=1
Sub-Cell Size	2.5m x 2.5m x 0.5m

Table 14-48: Sirenggok: Block Model Parameters

For Sirenggok all assays within the ore zone volume were used in the estimate (zonal estimation). A top-cut of 7.31 g/t Au was applied to all samples above this value. Limited density values were found in the a few drillholes. The average density determined from these density samples was 2.65 t/m^3.

Search ellipse and Ordinary Kriging parameters were derived from the variogram analysis and are summarised in *Table 14-49: Sirenggok: Ordinary Kriging Estimation Parameters* below.

Estimation Parameter	Value
Search Orientation	50° dip at 40° azimuth
Nugget	0.22
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.37
Sill (Range 2)	0.42
Range 1	5m x 5m x 5m
Range 2	40m x 40m x 40m
Minimum Samples	2
Maximum Samples	32

Table 14-49: Sirenggok: Ordinary Kriging Estimation Parameters

14.4.8. Resource & Comparative Estimates

The resource for Sirenggok was determined at a variety of lower cutoffs. *Table 14-50: Sirenggok: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	8,346,000	1.14
0.75	5,953,000	1.35
1	3,920,000	1.60
1.25	2,243,000	1.97
1.5	1,183,000	2.51
1.75	586,000	3.43
2	271,000	5.24

Table 14-50: Sirenggok: Ordinary Kriging Resource at 0.25 g/t Increments

A lower cutoff grade of 0.75 g/t Au was selected as this is a typical cutoff value used in other Malaysian operations and in known deposits mining similarly refractory ore.

Figure 14-51: Sirenggok: SW-NE Section through Ordinary Kriging Resource Model below shows a slice through the Sirenggok gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

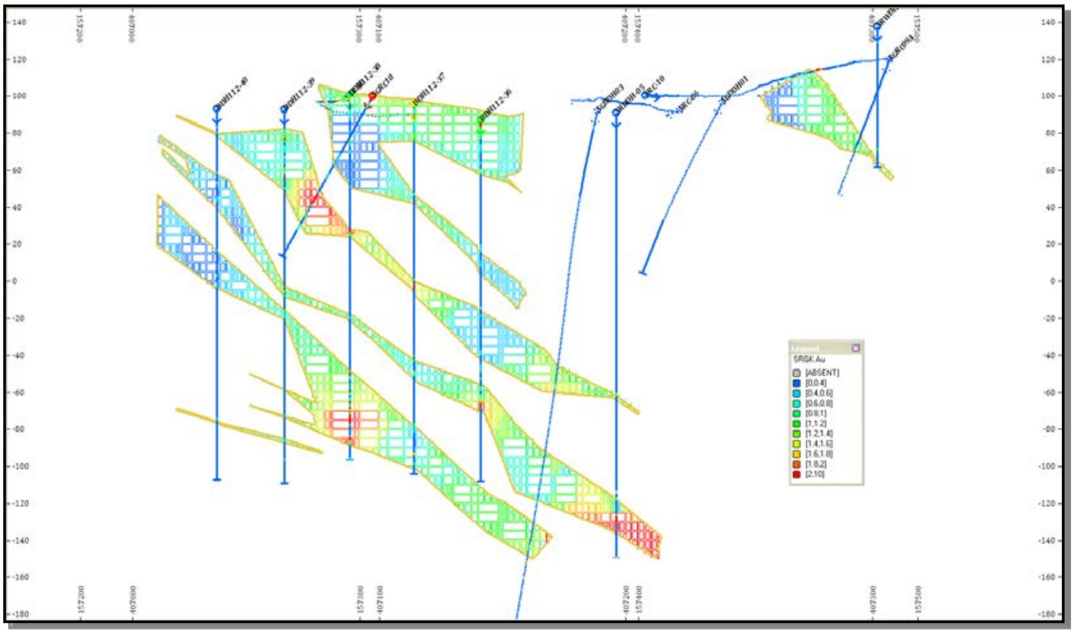


Figure 14-51: Sirenggok: SW-NE Section through Ordinary Kriging Resource Model

Resource model estimates are adjusted for topography or where excavations (underground and surface) exist. The resource model above topography or within known excavations is removed or subtracted from the final resource estimate.

Comparative estimations were conducted using Inverse Distance Squared and Nearest Neighbour (3D polygonal) methods. The estimation parameters used for these are listed below in *Table 14-51: Sirenggok: Comparative Estimation Method Parameters* for Sirenggok.

Estimation Parameter	Value
Search Orientation	50° dip at 40° azimuth
Search Ellipse Range	40m x 40m x 40m
Minimum Samples	2
Maximum Samples	32

Table 14-51: Sirenggok: Comparative Estimation Method Parameters

Listed below, in *Table 14-52: Sirenggok: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-53: Sirenggok: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Sirenggok.

CUTOFF	TONNES	AU
0.5	7,881,000	1.14
0.75	5,207,000	1.41
1	3,458,000	1.69
1.25	2,158,000	2.03
1.5	1,265,000	2.50
1.75	678,000	3.28
2	388,000	4.33

Table 14-52: Sirenggok: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	6,299,000	1.65
0.75	4,374,000	2.10
1	3,250,000	2.53
1.25	2,579,000	2.90
1.5	1,893,000	3.46
1.75	1,349,000	4.20
2	1,079,000	4.79

Table 14-53: Sirenggok: Nearest Neighbour Resource at 0.25 g/t Increments

The comparative resource estimates for Sirenggok compares well with the Ordinary Kriging resource estimate and the minor differences probably reflect the interpolation techniques/application.

The resource has been classified as Inferred. Some areas of the deposit(s) could potentially have been classified as Indicated based purely on the drilling density. However, one or more of the following issues gave rise to an Inferred classification:

- Large number of RC drillholes with few diamond core holes;
- Smaller drillhole sizes in some instances (e.g. BQ);
- Lack of extensive and systematic density determinations throughout the deposit;
- Gaps in the drillhole spacing or coverage and/or larger distances between drillholes;
- Difficulty in domaining of the data to remove possible mixed populations in some instances.

14.5. Pejiru Sector

This resource section was completed in 2010 and is detailed in report *"Technical Report on Bau Project in Bau, Sarawak, East Malaysia"*. It has been included here for the sake of completeness.

A revision to this resource was made in the February 2012 resource estimate release. No changes or modelling was redone only a change to the cut-off grade and therefore the following information is still valid. The reason that the grade was lowered is that some preliminary feasibility work identified the possibility that the reserve cutoff grade could be lower than the resource cutoff grade creating a problem situation where there could be reserves not in resource. The previous cutoff grade was 0.75 g/t and this was reduced to 0.5 g/t.

14.5.1. Introduction & General

The Pejiru sector is situated approximately 5-8 kilometres south of the town of Bau and is a set of four deposits based on discrete geographical areas as defined by the drilling to date. These deposits have been modelled separately and are Pejiru-Bogag, Boring, Pejiru Extension and Kapor.

The resource assessment conducted by Terra Mining Consultants/Stevens & Associates included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;
- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;
- Combining the above data into a clean and validated resource database with associated data being verified;
- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;
- Resource estimation work to determine the mineral resource using 3 different estimation techniques.

All data used for this resource update was supplied or sourced by Olympus/North Borneo Gold or determined by Terra Mining Consultants/Stevens & Associates from available information. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

Historical documents and reports were reviewed as part of the resource update and these are listed below and in *Section 20 – References*. Additionally, numerous notes, plans, sections, memoranda and other documents, both in digital and hardcopy format found in the office library and storage, were reviewed.

- Review of Sue Border, GEOS Mining Mineral Consultants, June 2007 report titled "Pejiru Preliminary Resources Report".
- Review of Ashby & Associates, June 2008 preliminary draft report (incomplete) titled "Investigation of the Pejiru Database (including Boring & Bogag)".

14.5.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of these are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;

- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

14.5.3. Ore Zone Definition

The ore zone at Pejiru-Bogag, Boring, Pejiru Extension and Kapor were defined in the following manner:

- Drillhole sections were created and interpreted faults, geological and mineralized zone grade boundaries (⩾0.5 g/t Au lower cut-off) were drawn;
- The grade boundaries were correlated from section to section and cross-checked in plan;
- In the absence of zone continuity, extrapolations were made in between the two drill sections, and up/down dip, using standard methodologies;
- The definition of the mineralized zones and the methodology used was validated visually on each section, and in 3D, and samples within the zone wireframe were analysed;
- The ore zone was terminated using the surveyed topography.

In the ore zone definition there are isolated cases of assay values below the lower cut-off value. These have only been included where they fall within samples above the cut-off, are of minor effect and cannot be excluded due to their isolated nature.

14.5.4. Statistical Analysis of Data

The full Pejiru database consisted of 704 drillhole collar entries, 704 collar survey entries, 25,276 assay records, 265 density records, and 50,542 lithology records.

A total of 51,956.31 metres of drilling was drilled in and around the Pejiru sector. The drillhole depths varied from 4 metres to 500 metres with an average depth of approximately 73.8 metres. The drillholes consisted of 682 RC holes and 22 diamond cored holes in BQ, NQ, HQ & PQ sizes.

The Pejiru-Bogag deposit has 237 drillholes, Boring deposit has 54 drillholes, Pejiru Extension deposit has 102 drillholes and Kapor deposit has 51 drillholes. The remaining drillholes fall outside the defined deposits.

A total of 8,255 drillhole assay samples fall within the mineralized zone at Pejiru-Bogag. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-54: Pejiru-Bogag: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	8,255	8,255
Number of Samples	8,255	8,126
Missing Values	-	129
Minimum Value	0.03	0.01
Maximum Value	8.50	90.90
Range	8.47	90.90
Mean	0.97	0.88
Variance	0.03	8.27
Standard Deviation	0.17	2.88
Standard Error	0.00	0.03
Skewness	12.29	16.30
Kurtosis	605.03	360.89
Geometric Mean	0.95	0.28
Sum of Logs	-407.03	-10,228.79
Mean of Logs	-0.05	-1.26
Log Variance	0.07	2.39
Log Estimate of Mean	0.99	0.94

Table 14-54: Pejiru-Bogag: Ore Zone Drillhole Sample Statistics

A total of 972 drillhole assay samples fall within the mineralized zone at Boring. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-55: Boring: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	972	972
Number of Samples	972	913
Missing Values	-	59
Minimum Value	0.50	0.01
Maximum Value	1.50	10.70
Range	1.00	10.70
Mean	1.00	0.74
Variance	0.00	1.60
Standard Deviation	0.02	1.26
Standard Error	0.00	0.04
Skewness	-	3.84
Kurtosis	483.00	19.35
Geometric Mean	1.00	0.29
Sum of Logs	-0.29	-1,119.64
Mean of Logs	-0.00	-1.23
Log Variance	0.00	2.02
Log Estimate of Mean	1.00	0.80

Table 14-55: Boring: Ore Zone Drillhole Sample Statistics

A total of 2,239 drillhole assay samples fall within the mineralized zone at Pejiru Extension. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-56: Pejiru Extension: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	2,329	2,329
Number of Samples	2,329	2,271
Missing Values	-	58
Minimum Value	0.50	0.01
Maximum Value	1.00	404.00
Range	0.50	404.00
Mean	1.00	0.86
Variance	0.00	73.16
Standard Deviation	0.01	8.55
Standard Error	0.00	0.18
Skewness	-48.23	46.12
Kurtosis	2,324.00	2,170.19
Geometric Mean	1.00	0.23
Sum of Logs	-0.69	-3,314.02
Mean of Logs	-0.00	-1.46
Log Variance	0.00	2.56
Log Estimate of Mean	1.00	0.84

Table 14-56: Pejiru Extension: Ore Zone Drillhole Sample Statistics

A total of 1,723 drillhole assay samples fall within the mineralized zone at Kapor. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-57: Kapor: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	1,723	1,723
Number of Samples	1,723	1,687
Missing Values	-	36
Minimum Value	0.40	0.01
Maximum Value	1.50	69.60
Range	1.10	69.59
Mean	1.00	1.32
Variance	0.00	14.35
Standard Deviation	0.02	3.79
Standard Error	0.00	0.09
Skewness	-7.65	8.41
Kurtosis	858.50	100.13
Geometric Mean	1.00	0.39
Sum of Logs	-0.42	- 1,575.83
Mean of Logs	-0.00	-0.93

Drillhole Field	Length	Au
Log Variance	0.00	2.35
Log Estimate of Mean	1.00	1.27

Table 14-57: Kapor: Ore Zone Drillhole Sample Statistics

Samples within the ore zone were composited to 1 metre lengths, resulting in 8,037 composites for Pejiru-Bogag, 973 composites for Boring, 2,329 composites for Pejiru Extension and 1,723 composites for Kapor. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length.

Table 14-58: Pejiru-Bogag: Ore Zone Composited Drillhole Sample Statistics lists the statistics for the composited drillholes for Pejiru-Bogag.

Drillhole Field	Length	Au
Number of Records	8,037	8,037
Number of Samples	8,037	7,910
Missing Values	-	127
Minimum Value	0.50	0.01
Maximum Value	1.00	90.90
Range	0.50	90.90
Mean	1.00	0.88
Variance	0.00	7.76
Standard Deviation	0.01	2.79
Standard Error	0.00	0.03
Skewness	-36.03	16.31
Kurtosis	1,411.43	371.26
Geometric Mean	1.00	0.29
Sum of Logs	-2.93	-9,842.95
Mean of Logs	-0.00	-1.24
Log Variance	0.00	2.37
Log Estimate of Mean	1.00	0.94

Table 14-58: Pejiru-Bogag: Ore Zone Composited Drillhole Sample Statistics

Table 14-59: Boring: Ore Zone Composited Drillhole Sample Statistics lists the statistics for the composited drillholes for Boring.

Drillhole Field	Length	Au
Number of Records	973	973
Number of Samples	973	914
Missing Values	-	59
Minimum Value	0.50	0.01
Maximum Value	1.00	10.70
Range	0.50	10.70
Mean	1.00	0.74

Drillhole Field	Length	Au
Variance	0.00	1.59
Standard Deviation	0.02	1.26
Standard Error	0.00	0.04
Skewness	-21.99	3.84
Kurtosis	481.50	19.36
Geometric Mean	1.00	0.29
Sum of Logs	-1.39	-1,119.55
Mean of Logs	-0.00	-1.22
Log Variance	0.00	2.02
Log Estimate of Mean	1.00	0.80

Table 14-59: Boring: Ore Zone Composited Drillhole Sample Statistics

Table 14-60: Pejiru Extension: Ore Zone Composited Drillhole Sample Statistics lists the statistics for the composited drillholes for Pejiru Extension.

Drillhole Field	Length	Au
Number of Records	2,329	2,329
Number of Samples	2,329	2,271
Missing Values	-	58
Minimum Value	0.50	0.01
Maximum Value	1.00	404.00
Range	0.50	404.00
Mean	1.00	0.86
Variance	0.00	73.16
Standard Deviation	0.01	8.55
Standard Error	0.00	0.18
Skewness	-48.23	46.12
Kurtosis	2,324.00	2,170.19
Geometric Mean	1.00	0.23
Sum of Logs	-0.69	-3,314.02
Mean of Logs	-0.00	-1.46
Log Variance	0.00	2.56
Log Estimate of Mean	1.00	0.84

Table 14-60: Pejiru Extension: Ore Zone Composited Drillhole Sample Statistics

Table 14-61: Kapor: Ore Zone Composited Drillhole Sample Statistics lists the statistics for the composited drillholes for Kapor.

Drillhole Field	Length	Au
Number of Records	1,723	1,723
Number of Samples	1,723	1,688
Missing Values	-	35
Minimum Value	1.00	0.01
Maximum Value	1.00	69.60
Range	-	69.59

Drillhole Field	Length	Au
Mean	1.00	1.32
Variance	-	14.34
Standard Deviation	-	3.79
Standard Error	-	0.09
Skewness	-	8.42
Kurtosis	-	100.19
Geometric Mean	-	0.39
Sum of Logs	-	-1,576.01
Mean of Logs	-	-0.93
Log Variance	-	2.34
Log Estimate of Mean	-	1.27

Table 14-61: Kapor: Ore Zone Composited Drillhole Sample Statistics

The Pejiru-Bogag Au data shown statistically above is also shown in graphical form below. *Figure 14-52: Pejiru-Bogag: Log Histogram of Au Ore Zone Composites* and *Figure 14-53: Pejiru-Bogag: Cumulative Log Histogram of Au Ore Zone Composites* below display the Pejiru-Bogag log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine Studio.

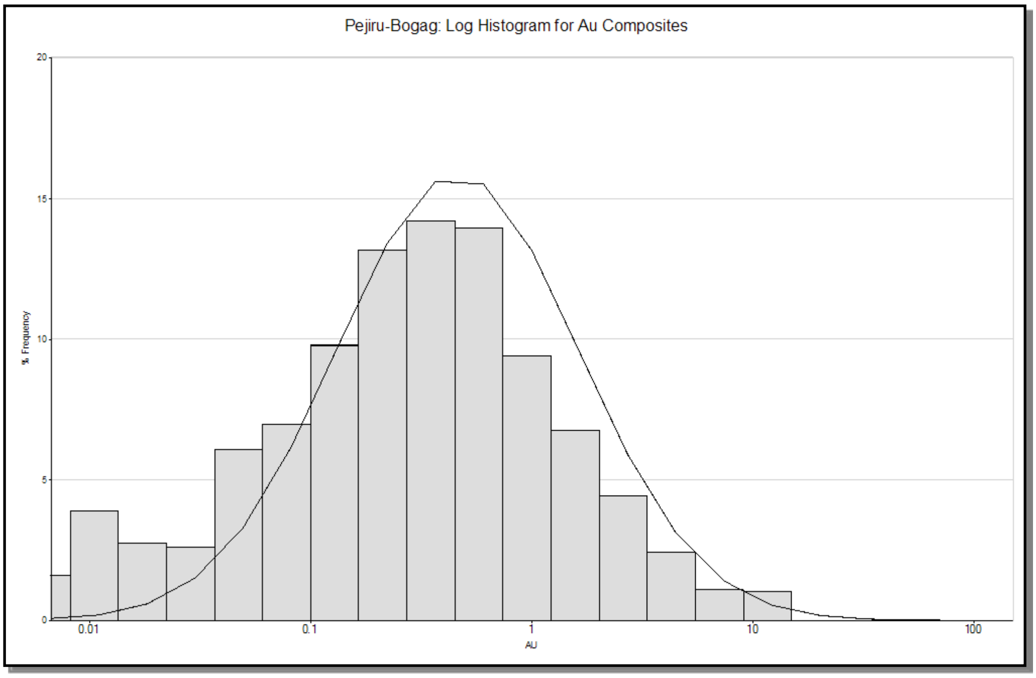


Figure 14-52: Pejiru-Bogag: Log Histogram of Au Ore Zone Composites

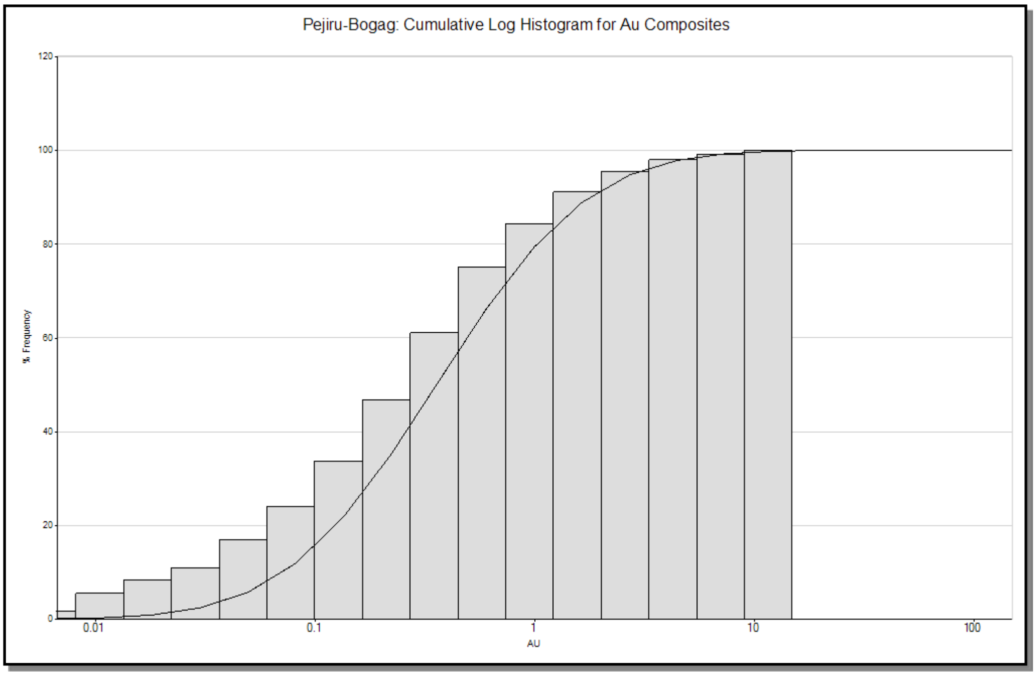


Figure 14-53: Pejiru-Bogag: Cumulative Log Histogram of Au Ore Zone Composites

The Boring Au data shown statistically above is also shown in graphical form below. *Figure 14-54: Boring: Log Histogram of Au Ore Zone Composites* and *Figure 14-55: Boring: Cumulative Log Histogram of Au Ore Zone Composites* below display the Boring log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine Studio.

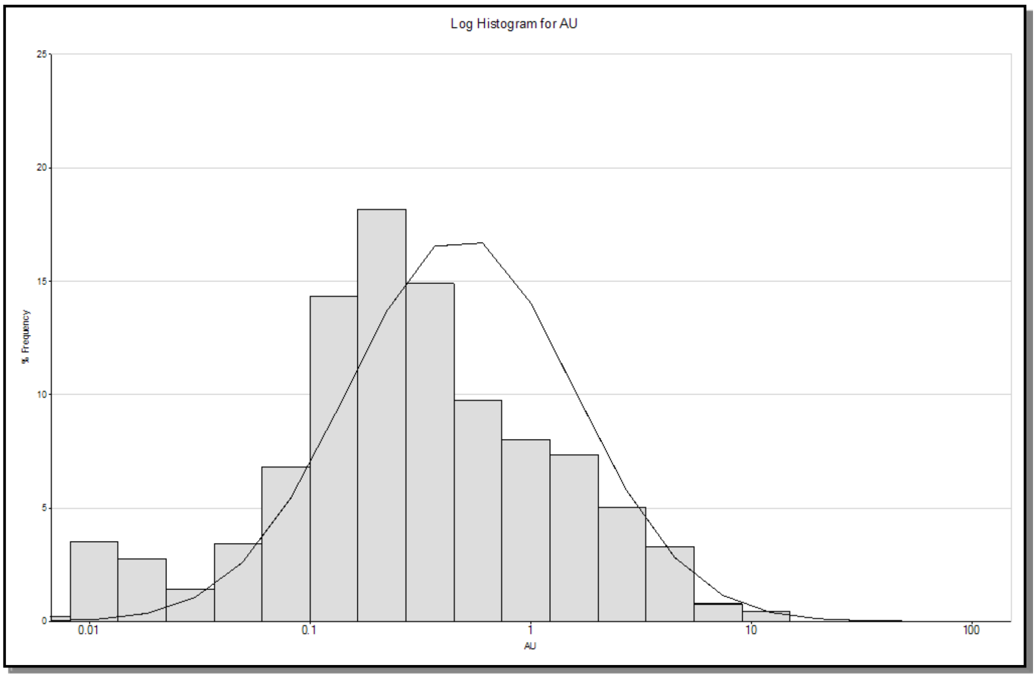


Figure 14-54: Boring: Log Histogram of Au Ore Zone Composites

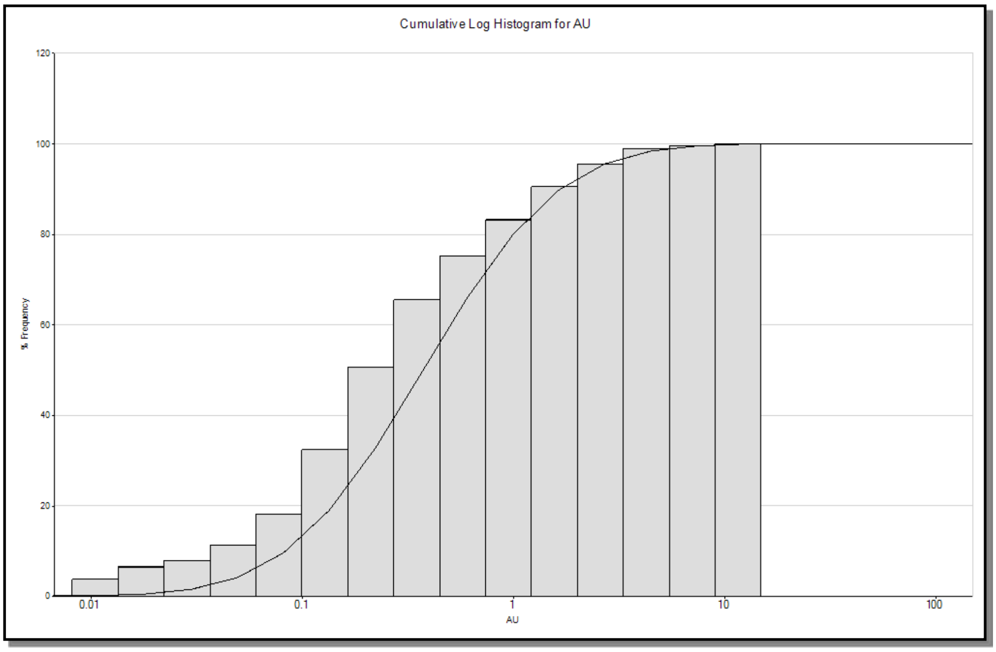


Figure 14-55: Boring: Cumulative Log Histogram of Au Ore Zone Composites

The Pejiru Extension Au data shown statistically above is also shown in graphical form below. *Figure 14-56: Pejiru Extension: Log Histogram of Au Ore Zone Composites* and *Figure 14-57: Pejiru Extension: Cumulative Log Histogram of Au Ore Zone Composites* below display the Pejiru Extension log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine Studio.

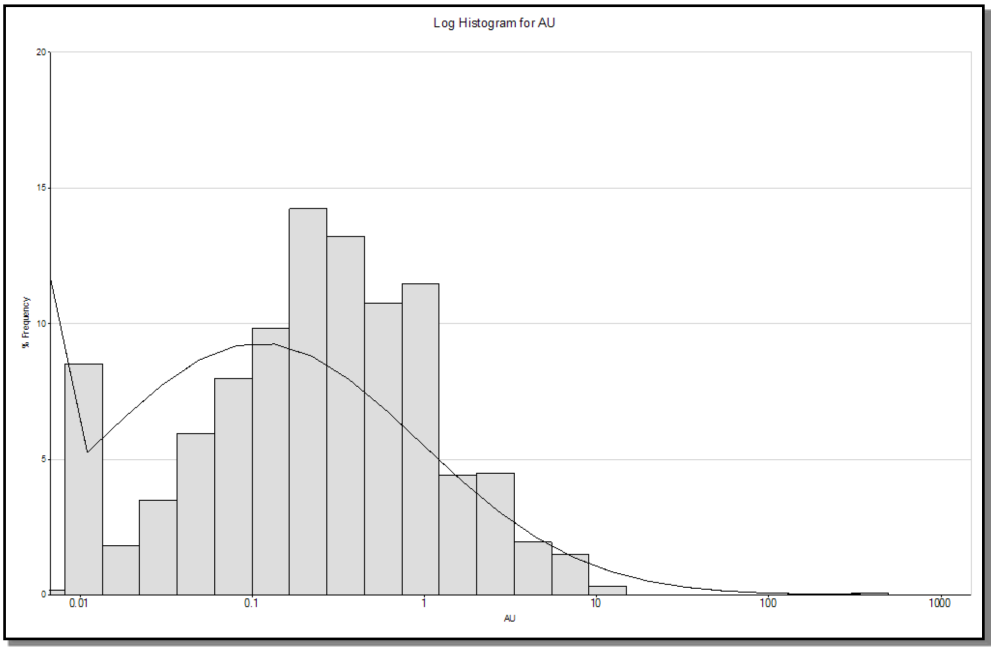


Figure 14-56: Pejiru Extension: Log Histogram of Au Ore Zone Composites

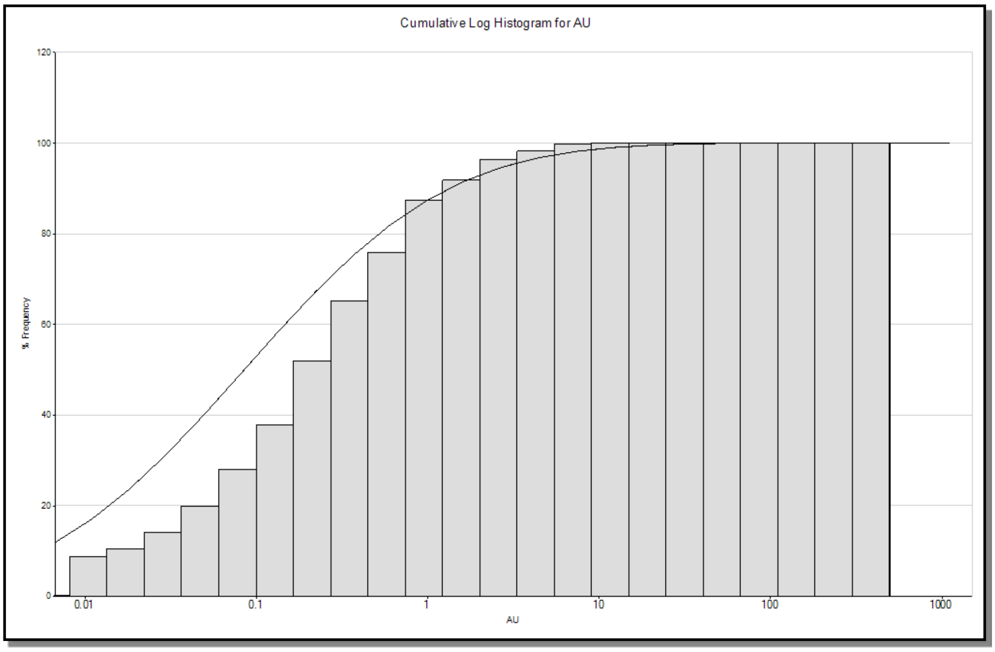


Figure 14-57: Pejiru Extension: Cumulative Log Histogram of Au Ore Zone Composites

The Kapor Au data shown statistically above is also shown in graphical form below. *Figure 14-58: Kapor: Log Histogram of Au Ore Zone Composites* and *Figure 14-59: Kapor: Cumulative Log Histogram of Au Ore Zone Composites* below display the Kapor log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine.

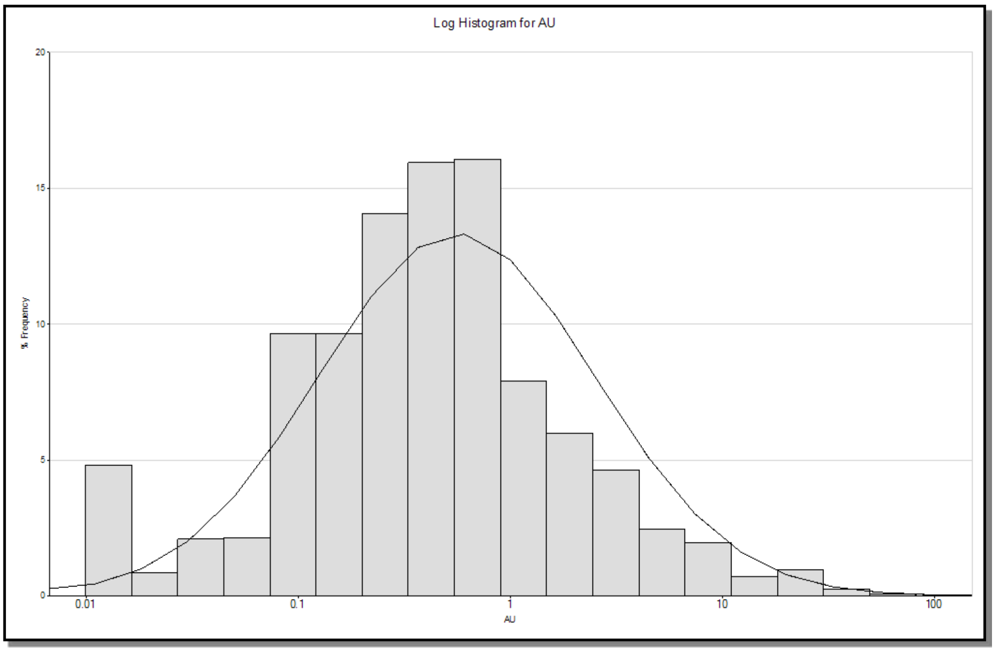


Figure 14-58: Kapor: Log Histogram of Au Ore Zone Composites

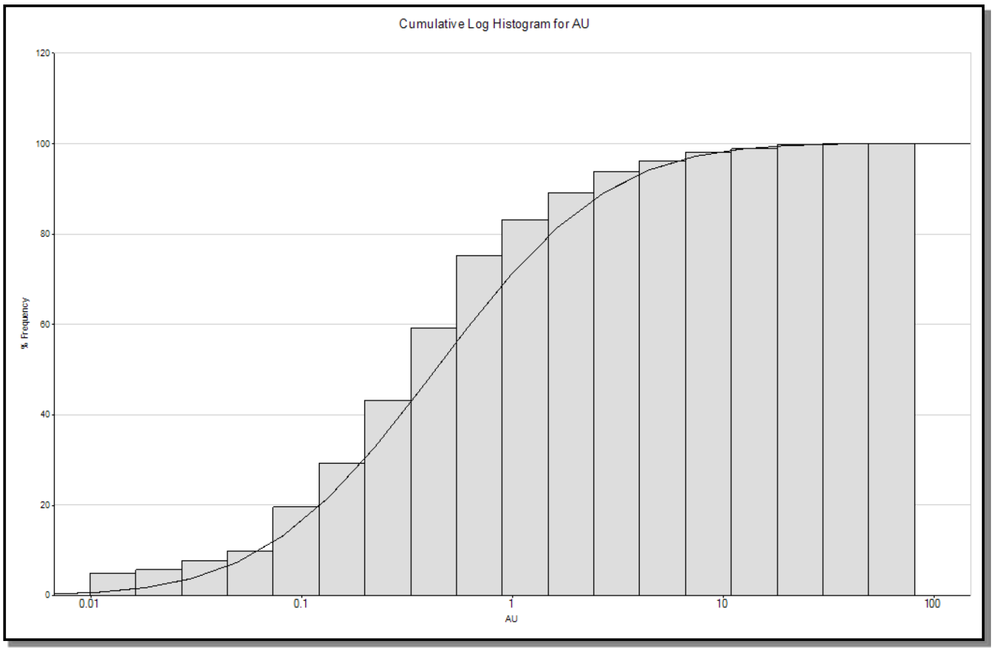


Figure 14-59: Kapor: Cumulative Log Histogram of Au Ore Zone Composites

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile.

Table 14-62: Pejiru-Bogag: Quantile Analysis of Au Drillhole Composites to *Table 14-65: Kapor: Quantile Analysis of Au Drillhole Composites* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Pejiru-Bogag, Boring, Pejiru Extension and Kapor Ore Zones.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	792	0.02	0.01	0.04	13.56	0.20
10	20	792	0.06	0.04	0.09	50.18	0.72
20	30	792	0.12	0.09	0.15	96.66	1.39
30	40	792	0.19	0.15	0.23	149.32	2.15
40	50	792	0.27	0.23	0.33	217.56	3.14
50	60	792	0.38	0.33	0.46	302.85	4.37
60	70	792	0.54	0.46	0.63	428.25	6.18
70	80	792	0.77	0.63	0.95	610.27	8.80
80	90	792	1.30	0.95	1.84	1,030.83	14.87
90	100	793	5.08	1.84	77.24	4,032.35	58.17
90	92.5	198	2.05	1.84	2.29	404.92	5.84
92.5	95	198	2.68	2.29	3.11	530.87	7.66
95	97.5	198	3.81	3.11	4.75	755.18	10.89
97.5	100	199	11.77	4.75	77.24	2,341.37	33.78
0	100	7921	0.88	0.01	77.24	6,931.82	100.00

Table 14-62: Pejiru-Bogag: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	91	0.02	0.01	0.05	2.03	0.30
10	20	91	0.09	0.05	0.11	8.36	1.23
20	30	92	0.13	0.12	0.15	12.01	1.77
30	40	91	0.18	0.15	0.20	16.12	2.37
40	50	92	0.24	0.20	0.27	21.67	3.19
50	60	91	0.31	0.27	0.36	28.47	4.19
60	70	91	0.46	0.36	0.60	41.60	6.12
70	80	92	0.76	0.60	1.00	70.31	10.35
80	90	91	1.43	1.00	1.95	130.38	19.20
90	100	92	3.79	1.98	10.70	348.24	51.27
90	92.5	23	2.16	1.98	2.38	49.72	7.32
92.5	95	23	2.80	2.46	3.25	64.41	9.48
95	97.5	23	3.71	3.31	4.21	85.34	12.57
97.5	100	23	6.47	4.36	10.70	148.77	21.90
0	100	914	0.74	0.01	10.70	679.19	100.00

Table 14-63: Boring: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	227	0.01	0.01	0.02	2.55	0.13
10	20	227	0.04	0.02	0.07	9.45	0.48
20	30	227	0.09	0.07	0.12	21.13	1.08
30	40	227	0.15	0.12	0.18	33.32	1.71
40	50	227	0.21	0.18	0.26	48.40	2.48
50	60	227	0.31	0.26	0.37	70.57	3.61
60	70	227	0.46	0.37	0.56	103.45	5.30
70	80	227	0.73	0.56	0.93	164.67	8.43
80	90	227	1.14	0.93	1.63	258.10	13.22
90	100	228	5.44	1.64	404.00	1,240.74	63.55
90	92.5	57	1.92	1.64	2.21	109.57	5.61
92.5	95	57	2.55	2.22	2.90	145.52	7.45
95	97.5	57	3.64	2.90	4.74	207.74	10.64
97.5	100	57	13.65	4.75	404.00	777.91	39.84
0	100	2271	0.86	0.01	404.00	1,952.38	100.00

Table 14-64: Pejiru Extension: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	168	0.03	0.01	0.08	4.66	0.21
10	20	169	0.11	0.08	0.13	17.75	0.80
20	30	168	0.16	0.13	0.21	27.61	1.24
30	40	169	0.25	0.21	0.29	41.73	1.87
40	50	169	0.36	0.29	0.42	60.15	2.70
50	60	168	0.49	0.42	0.55	82.00	3.68
60	70	169	0.63	0.55	0.75	106.50	4.77

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
70	80	168	0.91	0.75	1.18	153.61	6.89
80	90	169	1.77	1.18	2.62	299.86	13.44
90	100	169	8.50	2.64	69.60	1,437.01	64.41
90	92.5	42	2.93	2.64	3.40	122.88	5.51
92.5	95	42	3.99	3.41	4.68	167.64	7.51
95	97.5	42	6.70	4.70	8.94	281.46	12.62
97.5	100	43	20.12	9.06	69.60	865.03	38.78
0	100	1686	1.32	0.01	69.60	2,230.88	100.00

Table 14-65: Kapor: Quantile Analysis of Au Drillhole Composites

For Pejiru-Bogag, looking at the primary percentiles, it can be seen that approximately 58 % of the metal percentage can be found in the top 10 % range, and that there is a significant jump in the mean grade and metal content from the previous range. For Boring this is approximately 51 %, Pejiru Extension approximately 64 % and Kapor 64 %.

Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains nearly 34 % of the Au metal content for Pejiru-Bogag, 22 % for Boring, 40 % for Pejiru Extension and 39 % for Kapor.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top-cut of 11.77 g/t Au (mean of the 97.5 percentile) should be applied to the Pejiru-Bogag samples above this value in order to remove any effect of the high grade samples in the estimation process. Similarly, a top-cut of 6.47 g/t Au for Boring, 13.65 g/t Au for Pejiru Extension and 20.12 g/t Au for Kapor.

14.5.5. Semi-Variogram Analysis

Semi-variogram analyses were undertaken to determine the semi-variogram parameters for use in the Ordinary Kriging. Downhole, horizontal and vertical increment semi-variograms were generated with the best semi-variograms selected that defines the strike, dip and dip direction. These semi-variograms were used to determine the nugget, sill values and ranges.

A log semi-variogram and two-range spherical model were used. A best fit model in the downhole semi-variogram was used to define the nugget. Subsequent model fitting was applied to the strike and dip/dip-direction to define the sill values by varying the ranges in these directions. The semi-variogram parameters are listed in *Table 14-67: Pejiru-Bogag: Ordinary Kriging Estimation Parameters* to *Table 14-70: Kapor: Ordinary Kriging Estimation Parameters* in Section 16.1.5.7 below

The semi-variograms for Pejiru-Bogag are shown below in *Figure 14-60: Pejiru-Bogag: Downhole Semi-Variogram* and *Figure 14-61: Pejiru-Bogag: Strike/Dip Direction Semi-Variogram*.

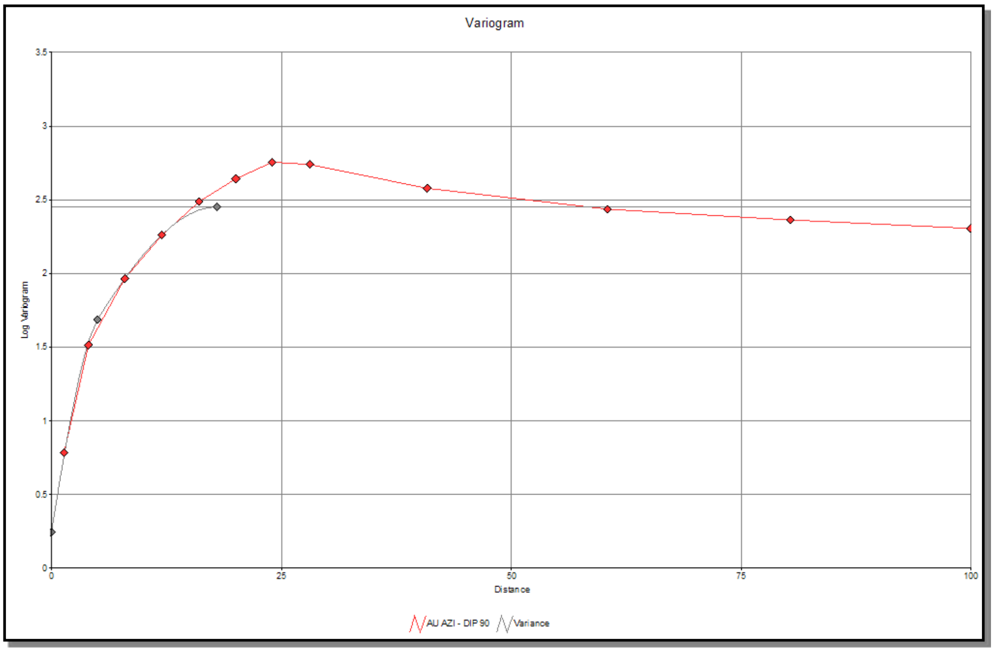


Figure 14-60: Pejiru-Bogag: Downhole Semi-Variogram

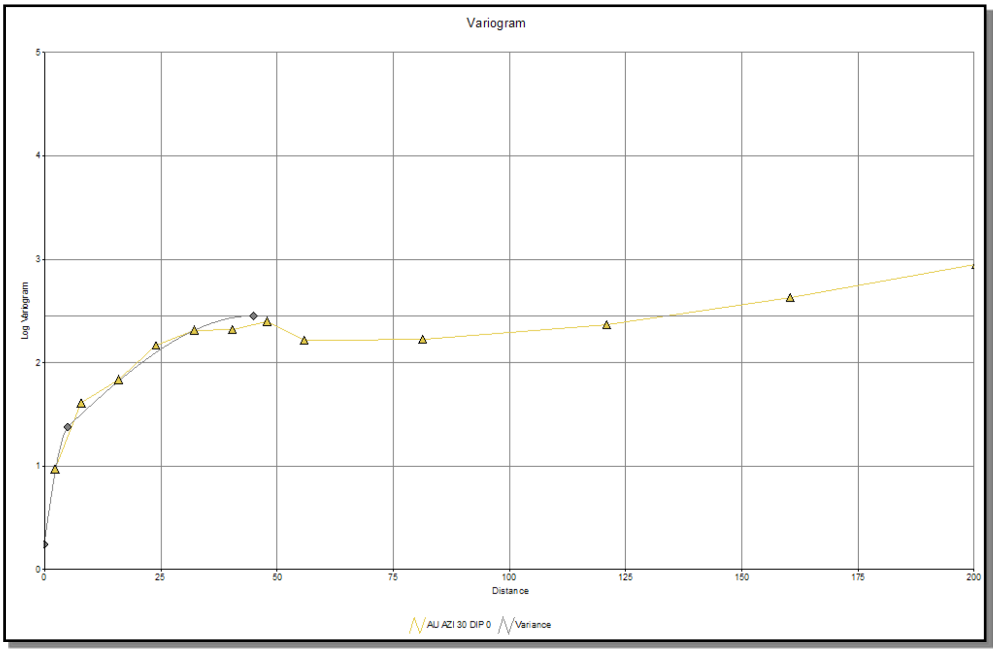


Figure 14-61: Pejiru-Bogag: Strike/Dip Direction Semi-Variogram

The semi-variograms for Boring are shown below in *Figure 14-62: Boring: Downhole Semi-Variogram* to *Figure 14-63: Boring: Strike/Dip Direction Semi-Variogram.*

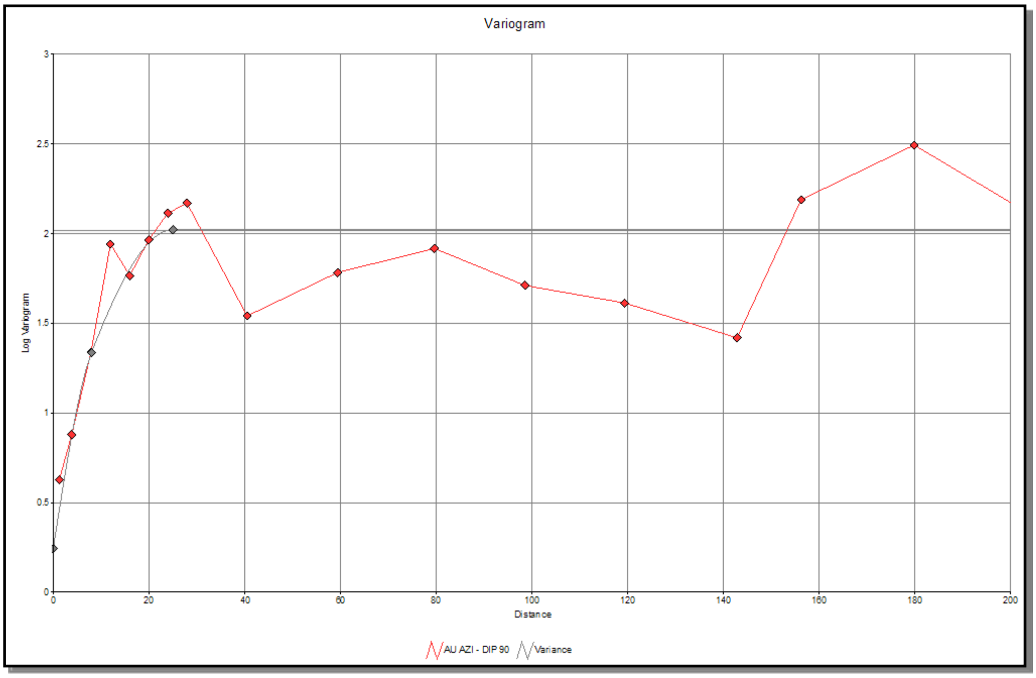


Figure 14-62: Boring: Downhole Semi-Variogram

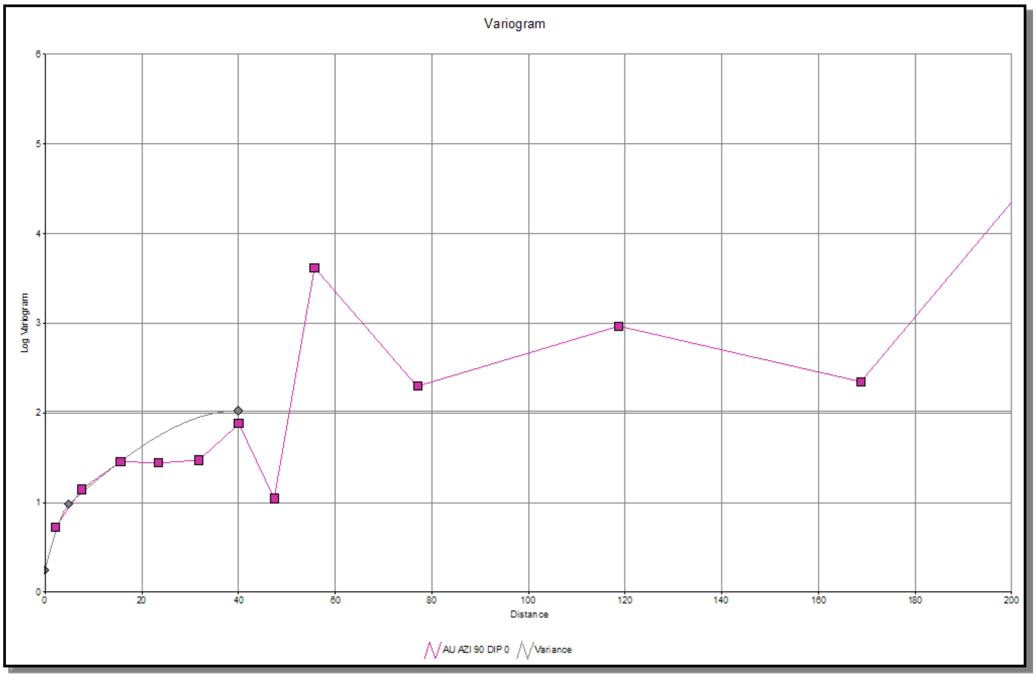


Figure 14-63: Boring: Strike/Dip Direction Semi-Variogram

The semi-variograms for Pejiru Extension are shown below in *Figure 14-64: Pejiru Extension: Downhole Semi-Variogram* and *Figure 14-65: Pejiru Extension: Strike/Dip Direction Semi-Variogram*

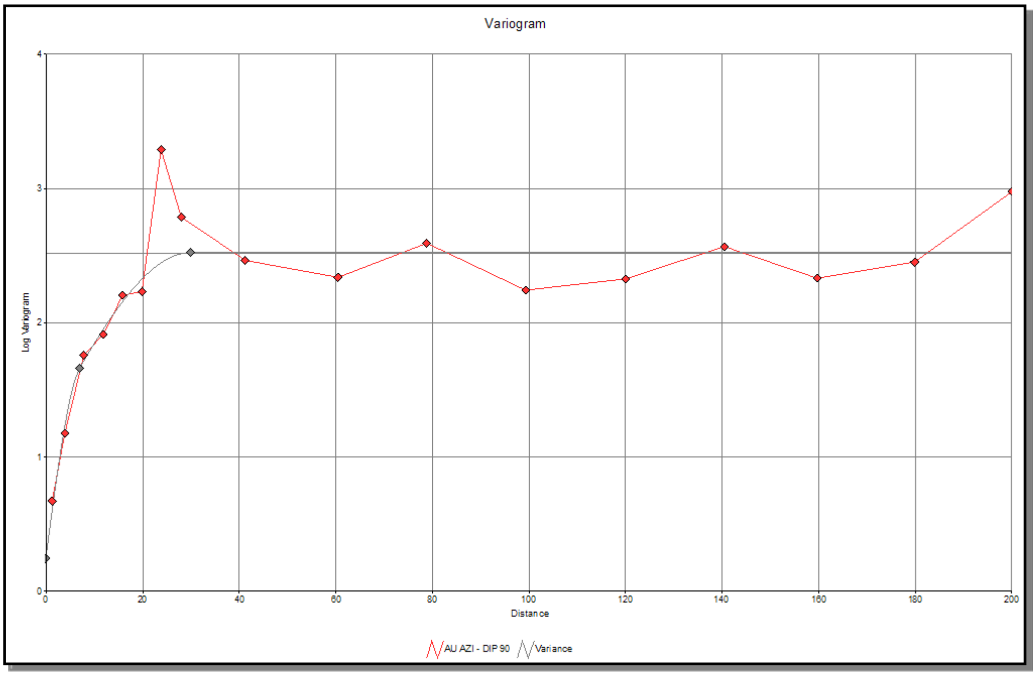


Figure 14-64: Pejiru Extension: Downhole Semi-Variogram

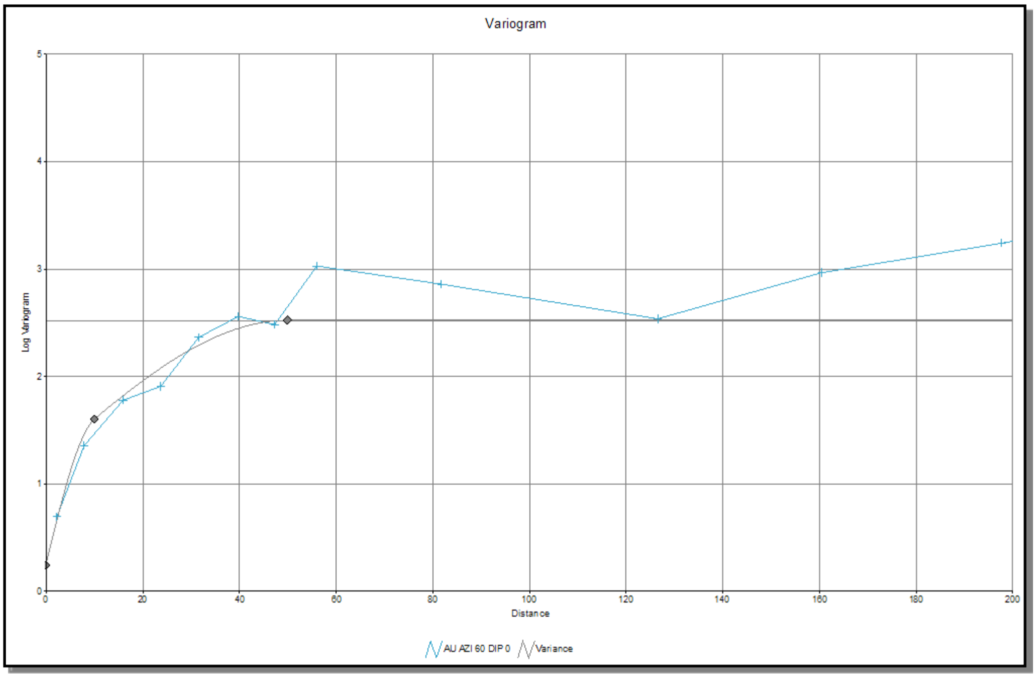


Figure 14-65: Pejiru Extension: Strike/Dip Direction Semi-Variogram

The semi-variograms for Kapor are shown below in *Figure 14-66: Kapor: Downhole Semi-Variogram* and *Figure 14-67: Kapor: Strike/Dip Direction Semi-Variogram.*

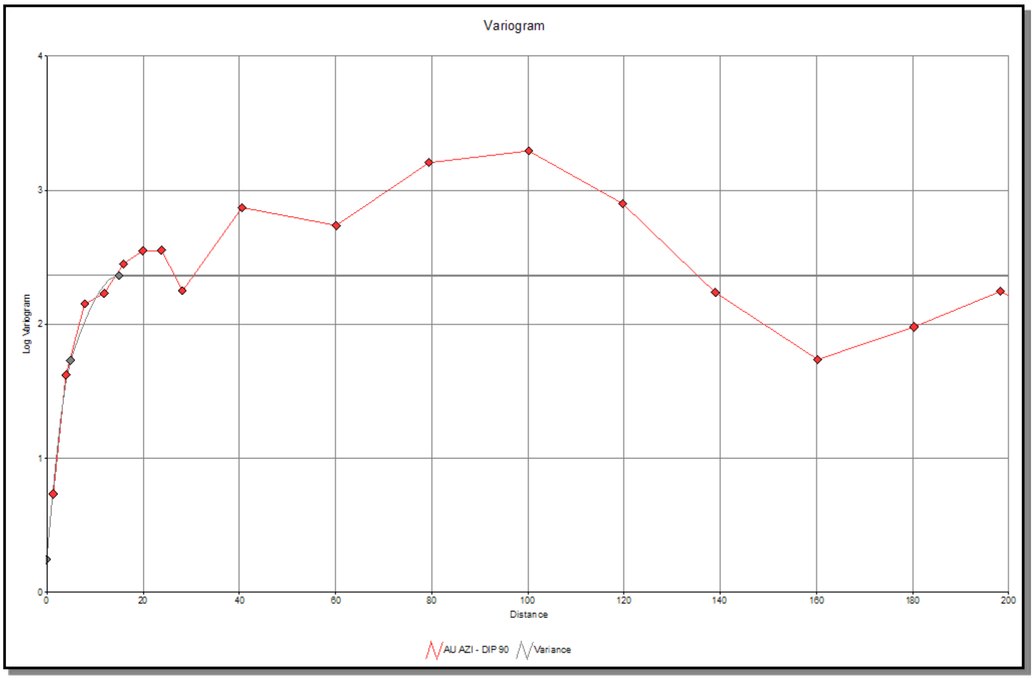


Figure 14-66: Kapor: Downhole Semi-Variogram

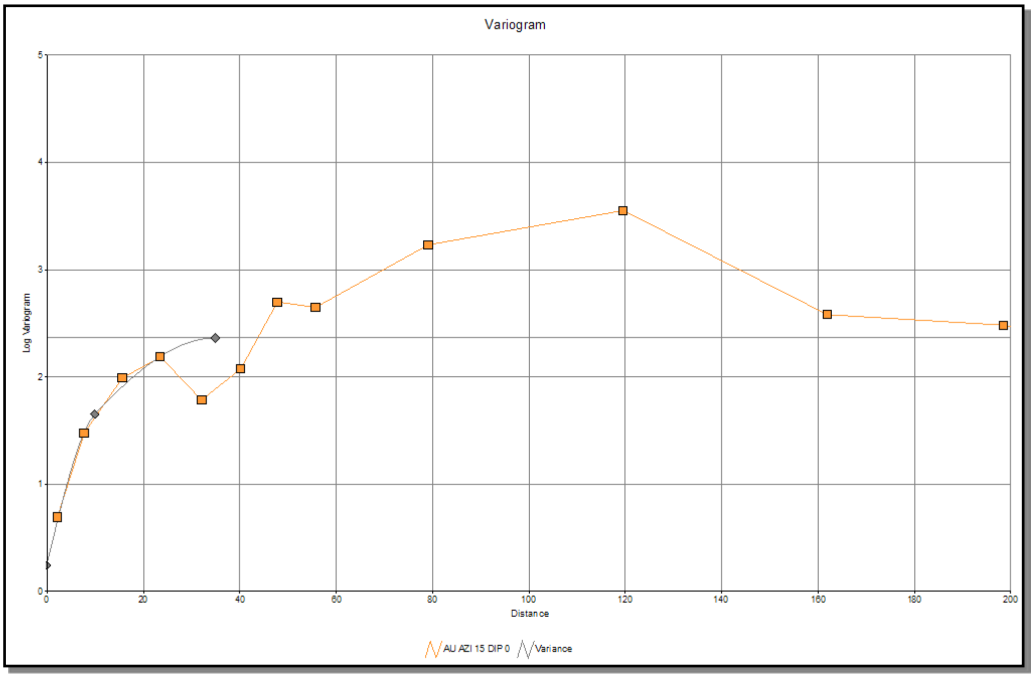


Figure 14-67: Kapor: Strike/Dip Direction Semi-Variogram

The modelled log semi-variogram values were back calculated to normal semi-variograms for use with Ordinary Kriging. The back transform for Pejiru-Bogag is shown in *Figure 154: Pejiru-Bogag: Log to Normal Semi-Variogram Transform* below, with Boring shown in *Figure 155: Boring: Log to Normal Semi-Variogram Transform*, Pejiru Extension in *Figure 14-70: Pejiru Extension: Log to Normal Semi-Variogram Transform* and Kapor in *Figure 14-71: Kapor: Log to Normal Semi-Variogram Transform.*

Converting log to normal variograms:

Enter the following data:

log variance :	2.45	**non-standardised**
log nugget :	0.24	0
log sill 1:	0.92	10
log sill 2:	1.29	35
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.23**
sill 1:	**0.32**
sill 2:	**0.45**
sill 3:	**0.00**

Figure 14-68: Pejiru-Bogag: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:

Enter the following data:

log variance :	2.02	**non-standardised**
log nugget :	0.24	0
log sill 1:	0.50	5
log sill 2:	1.28	40
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.25**
sill 1:	**0.21**
sill 2:	**0.54**
sill 3:	**0.00**

Figure 14-69: Boring: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:

Enter the following data:

log variance :	2.52	**non-standardised**
log nugget :	0.24	0
log sill 1:	0.97	10
log sill 2:	1.31	35
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.23**
sill 1:	**0.33**
sill 2:	**0.44**
sill 3:	**0.00**

Figure 14-70: Pejiru Extension: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:

Enter the following data:

log variance :	2.36	non-standardised
log nugget :	0.24	0
log sill 1:	0.90	10
log sill 2:	1.22	35
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	0.24
sill 1:	0.32
sill 2:	0.44
sill 3:	0.00

Figure 14-71: Kapor: Log to Normal Semi-Variogram Transform

14.5.6. Previous Resource Estimates

The Pejiru sector and deposits has been the subject to a number of historic resource estimates (both internal and public) but the two public resource estimates are the most significant. The following summary of the two public, historic resource estimates completed prior to 2010, was extracted from Olympus/North Borneo Gold sourced or supplied technical documents.

Some of these historic estimates were prepared pre-NI43-101 and Terra Mining Consultants/Stevens & Associates has neither audited them nor made any attempt to classify them according to NI43-101 standards. Although some of the more recent resource estimates are purported to have been compiled in terms of the relevant AusIMM JORC Code at that point in time. They are presented because Olympus and Terra Mining Consultants/Stevens & Associates consider them to be relevant and of historic significance.

- Sue Border of GEOS Mining (GEOS) for Zedex Ltd in June 2007. Border defined an Inferred Resource (JORC 2004) of 3.34 million tonnes at 1.55 g/t Au within a limited area around Pejiru only. This was estimated using Inverse Distance Squared method, based on a cut-off of 0.5 g/t Au.
- John Ashby (Ashby) of Ashby & Associates for Zedex Ltd in October 2008. Ashby defined an Inferred Resource (JORC 2004) of 5.582 million tonnes at 2.14 g/t Au at Pejiru (included Bogag and Boring) and an Inferred Resource (JORC 2004) of 1.052 million tonnes at 3.34 g/t Au at the Kapor deposit, using a cutoff of 1.0 g/t Au for both estimates.

14.5.7. Modelling & Resource Estimation Parameters

The ore zone wireframes were generated in Gemcom by Olympus/North Borneo Gold staff and imported into Datamine and validated. These were then filled with block model cells orientated orthogonally. The block model parameters for Pejiru-Bogag, Boring, Pejiru Extension and Kapor are listed in *Table 14-66: All Pejiru Deposits: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	10m x 10m x 5m
Zone Code	Ore Zone=1
Sub-Cell Size	2.5m x 2.5m x 0.5m

Table 14-66: All Pejiru Deposits: Block Model Parameters

For Pejiru-Bogag, Boring, Pejiru Extension and Kapor, all assays within the ore zone volume were used in the estimate (zonal estimation). A top-cut of 11.77 g/t Au was applied to all samples above this value for Pejiru-Bogag deposit, 6.47 g/t Au for Boring, 13.65 g/t Au for Pejiru Extension and 20.12 g/t Au for Kapor.

Limited density values were found in the a few drillholes. The average density determined from these density samples was 2.61 t/m^3.

Search ellipse and Ordinary Kriging parameters were derived from the variogram analysis and are summarised below in *Table 14-67: Pejiru-Bogag: Ordinary Kriging Estimation Parameters* for Pejiru-Bogag, *Table 14-68: Boring: Ordinary Kriging Estimation Parameters* for Boring, *Table 14-69: Pejiru Extension: Ordinary Kriging Estimation Parameters* for Pejiru Extension and *Table 14-70: Kapor: Ordinary Kriging Estimation Parameters* for Kapor.

Estimation Parameter	Value
Search Orientation	0° dip at 30° azimuth
Nugget	0.23
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.32
Sill (Range 2)	0.45
Range 1	5m x 5m x 5m
Range 2	45m x 45m x 18m
Minimum Samples	2
Maximum Samples	32

Table 14-67: Pejiru-Bogag: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 90° azimuth
Nugget	0.25
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.21
Sill (Range 2)	0.54
Range 1	5m x 5m x 8m
Range 2	40m x 40m x 25m
Minimum Samples	2
Maximum Samples	32

Table 14-68: Boring: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 60° azimuth
Nugget	0.23
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.33
Sill (Range 2)	0.44
Range 1	10m x 10m x 7m
Range 2	50m x 50m x 30m
Minimum Samples	2
Maximum Samples	32

Table 14-69: Pejiru Extension: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 15° azimuth
Nugget	0.24
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.32
Sill (Range 2)	0.44
Range 1	10m x 10m x 5m
Range 2	35m x 35m x 15m
Minimum Samples	2
Maximum Samples	32

Table 14-70: Kapor: Ordinary Kriging Estimation Parameters

14.5.8. Resource & Comparative Estimates

The resource for Pejiru-Bogag was determined at a variety of lower cutoffs. *Table 14-71: Pejiru-Bogag: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	11,800,000	1.10
0.75	7,328,000	1.40
1	4,714,000	1.70
1.25	3,189,000	1.98
1.5	2,131,000	2.28
1.75	1,412,000	2.62
2	993,000	2.94

Table 14-71: Pejiru-Bogag: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Boring was determined at a variety of lower cutoffs. *Table 14-72: Boring: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	2,096,000	1.10
0.75	1,476,000	1.30
1	935,000	1.54
1.25	588,000	1.79
1.5	373,000	2.04
1.75	234,000	2.29
2	132,000	2.62

Table 14-72: Boring: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Pejiru Extension was determined at a variety of lower cutoffs. *Table 14-73: Pejiru Extension: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	7,053,000	1.14
0.75	5,028,000	1.35
1	3,486,000	1.55
1.25	2,068,000	1.88
1.5	1,480,000	2.08
1.75	1,046,000	2.27
2	776,000	2.41

Table 14-73: Pejiru Extension: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Kapor was determined at a variety of lower cutoffs. *Table 14-74: Kapor: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	4,849,000	1.59
0.75	3,175,000	2.11
1	2,316,000	2.57
1.25	1,808,000	2.98
1.5	1,491,000	3.32
1.75	1,202,000	3.73
2	1,016,000	4.07

Table 14-74: Kapor: Ordinary Kriging Resource at 0.25 g/t Increments

The original cutoff grade of 0.75 g/t Au has been lowered to 0.5 g/t Au in line with potential reserve cutoffs being lower and a review of the statistics.

Figure 14-72: Pejiru-Bogag: NS Section through Ordinary Kriging Resource Model below shows a slice through the Pejiru-Bogag gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

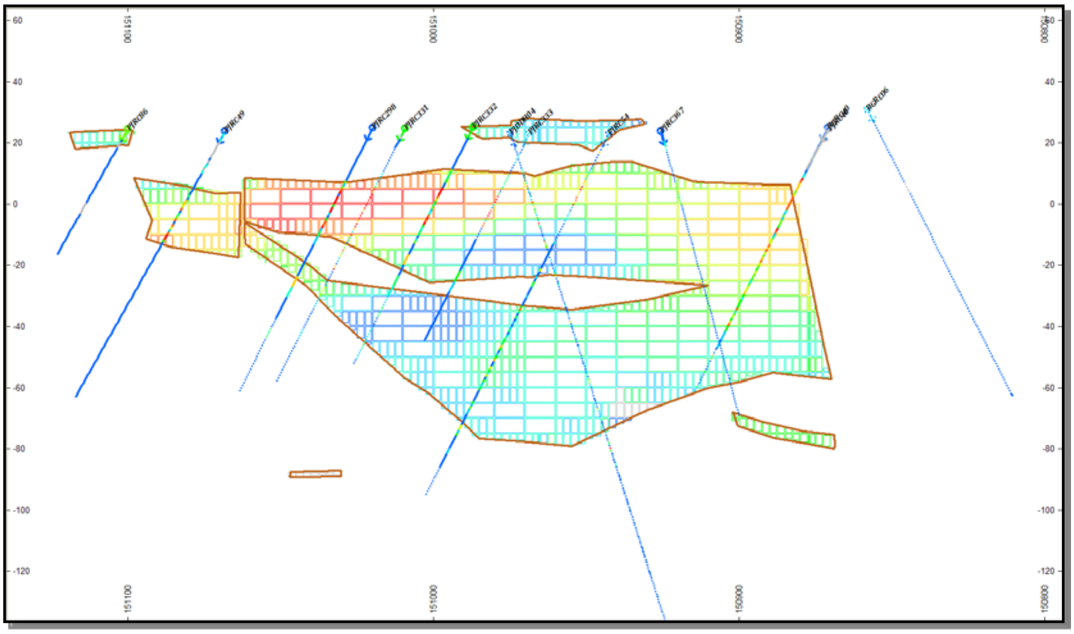


Figure 14-72: Pejiru-Bogag: NS Section through Ordinary Kriging Resource Model

Figure 14-73: Boring: WE Section through Ordinary Kriging Resource Model below shows a slice through the Boring gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

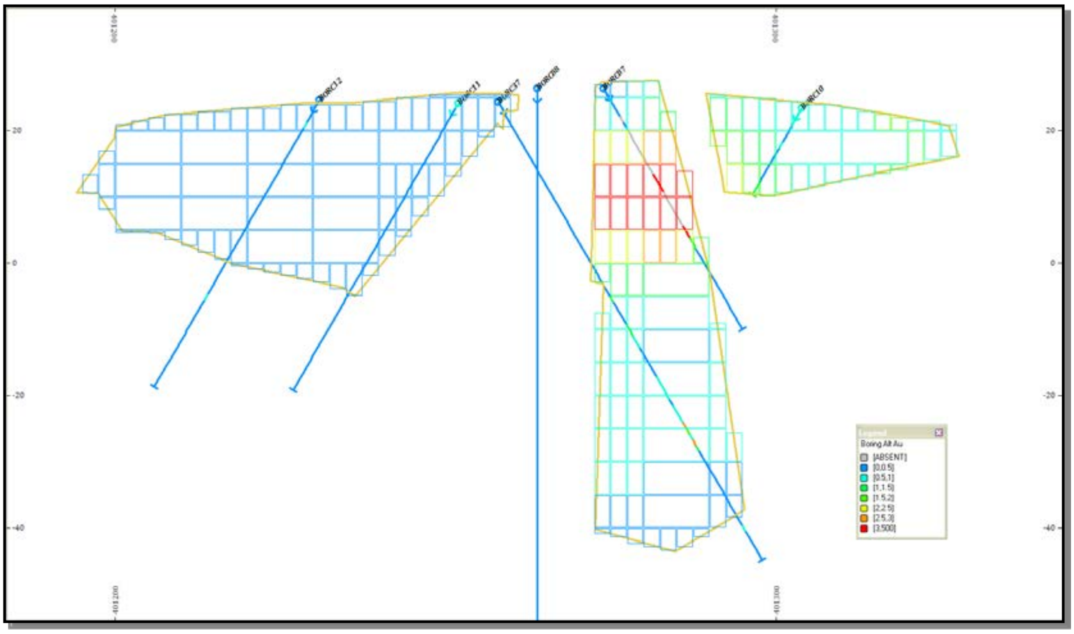


Figure 14-73: Boring: WE Section through Ordinary Kriging Resource Model

Figure 14-74: Pejiru Extension: WE Section through Ordinary Kriging Resource Model below shows a slice through the Pejiru Extension gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

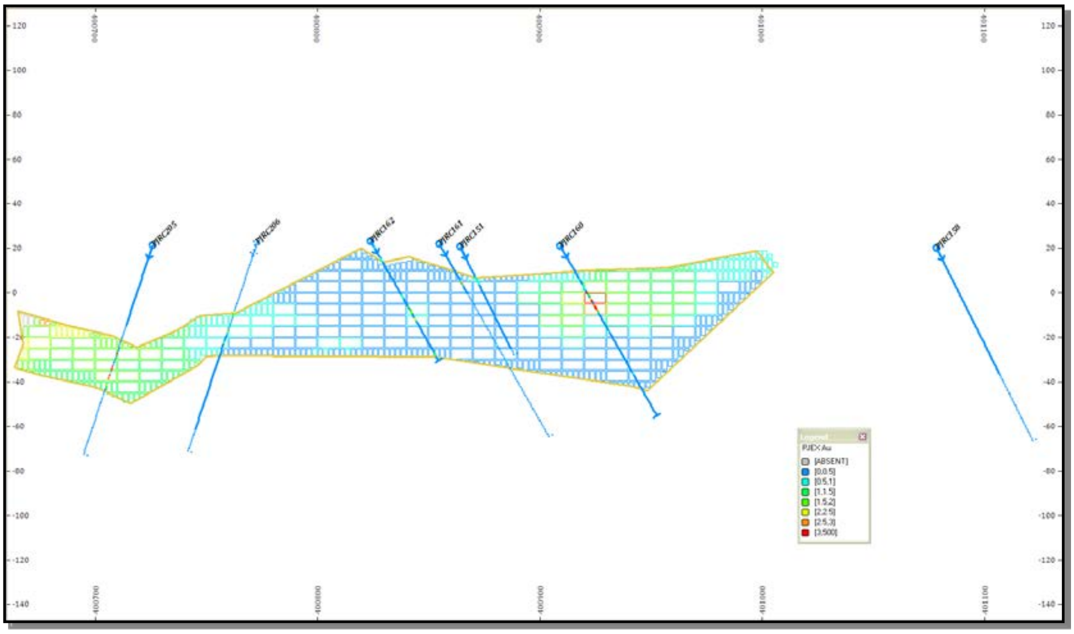


Figure 14-74: Pejiru Extension: WE Section through Ordinary Kriging Resource Model

Figure 14-75: Kapor: NS Section through Ordinary Kriging Resource Model below shows a slice through the Kapor gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

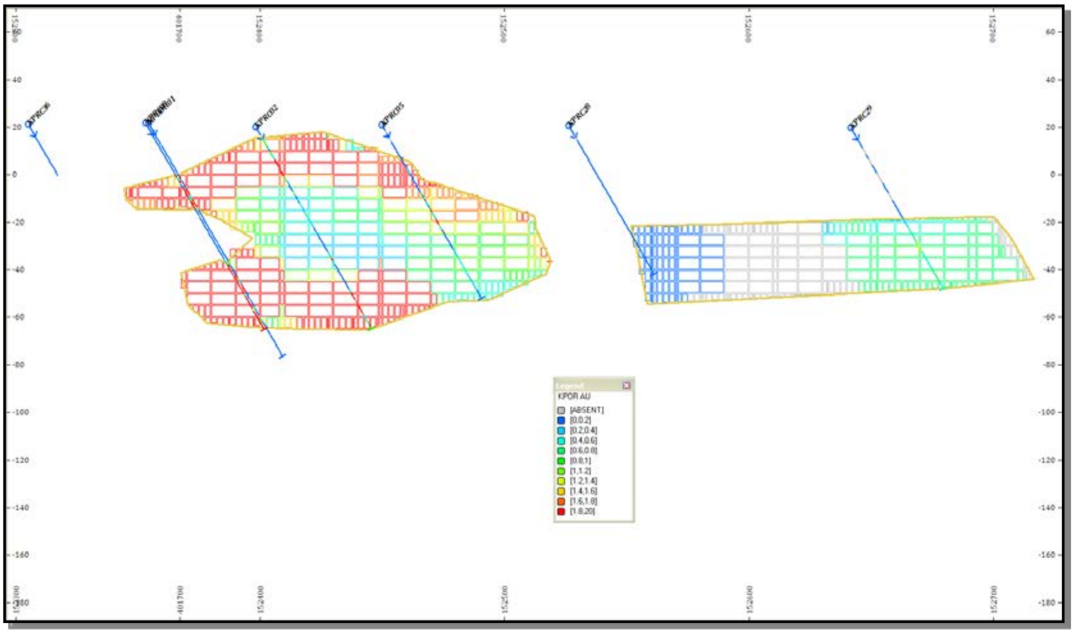


Figure 14-75: Kapor: NS Section through Ordinary Kriging Resource Model

Resource model estimates are adjusted for topography or where excavations (underground and surface) exist. The resource model above topography or within known excavations is removed or subtracted from the final resource estimate.

Comparative estimations were conducted using Inverse Distance Squared and Nearest Neighbour (3D polygonal) methods. The estimation parameters used for these are listed below in *Table 14-75: Pejiru-Bogag: Comparative Estimation Method Parameters* for Pejiru-Bogag, *Table 14-76: Boring: Comparative Estimation Method Parameters* for Boring, *Table 14-77: Pejiru Extension: Comparative Estimation Method Parameters* for Pejiru Extension and *Table 14-78: Kapor: Comparative Estimation Method Parameters* for Kapor.

Estimation Parameter	Value
Search Orientation	0° dip at 30° azimuth
Search Ellipse Range	45m x 45m x 18m
Minimum Samples	2
Maximum Samples	32

Table 14-75: Pejiru-Bogag: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 90° azimuth
Search Ellipse Range	40m x 40m x 25m
Minimum Samples	2
Maximum Samples	32

Table 14-76: Boring: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 60° azimuth
Search Ellipse Range	50m x 50m x 30m
Minimum Samples	2
Maximum Samples	32

Table 14-77: Pejiru Extension: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 15° azimuth
Search Ellipse Range	35m x 35m x 15m
Minimum Samples	2
Maximum Samples	32

Table 14-78: Kapor: Comparative Estimation Method Parameters

Listed below, in *Table 14-79: Pejiru-Bogag: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-80: Pejiru-Bogag: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Pejiru-Bogag.

CUTOFF	TONNES	AU
0.5	11,580,000	1.15
0.75	7,490,000	1.44
1	4,858,000	1.75
1.25	3,310,000	2.05
1.5	2,277,000	2.35

CUTOFF	TONNES	AU
1.75	1,500,000	2.73
2	1,078,000	3.07

Table 14-79: Pejiru-Bogag: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	8,261,000	1.62
0.75	5,170,000	2.23
1	3,819,000	2.72
1.25	3,030,000	3.13
1.5	2,479,000	3.53
1.75	2,159,000	3.81
2	1,821,000	4.18

Table 14-80: Pejiru-Bogag: Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-81: Boring: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-82: Boring: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Boring.

CUTOFF	TONNES	AU
0.5	1,897,000	1.16
0.75	1,417,000	1.35
1	998,000	1.54
1.25	600,000	1.83
1.5	378,000	2.11
1.75	250,000	2.36
2	135,000	2.79

Table 14-81: Boring: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	1,194,000	1.71
0.75	841,000	2.16
1	652,000	2.54
1.25	513,000	2.93
1.5	424,000	3.26
1.75	387,000	3.42
2	308,000	3.82

Table 14-82: Boring: Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-83: Pejiru Extension: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-84: Pejiru Extension: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Pejiru Extension.

CUTOFF	TONNES	AU
0.5	7,120,000	1.20
0.75	5,155,000	1.43
1	3,715,000	1.63
1.25	2,320,000	1.97
1.5	1,687,000	2.20
1.75	1,151,000	2.47
2	886,000	2.65

Table 14-83: Pejiru Extension: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	5,356,000	1.70
0.75	3,923,000	2.10
1	3,112,000	2.42
1.25	1,901,000	3.31
1.5	1,643,000	3.61
1.75	1,403,000	3.95
2	1,168,000	4.37

Table 14-84: Pejiru Extension: Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-85: Kapor: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-86: Kapor: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Kapor.

CUTOFF	TONNES	AU
0.5	4,808,000	1.67
0.75	3,255,000	2.18
1	2,281,000	2.74
1.25	1,829,000	3.14
1.5	1,542,000	3.46
1.75	1,252,000	3.89
2	1,071,000	4.24

Table 14-85: Kapor: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	3,343,000	2.23
0.75	2,220,000	3.06
1	1,624,000	3.87
1.25	1,297,000	4.57
1.5	1,112,000	5.10
1.75	977,000	5.58
2	874,000	6.02

Table 14-86: Kapor: Nearest Neighbour Resource at 0.25 g/t Increments

The comparative resource estimates for Pejiru-Bogag, Boring, Pejiru Extension and Kapor compare well with the Ordinary Kriging resource estimates and the minor differences probably reflect the interpolation techniques/application.

The resource has been classified as Inferred. Some areas of the deposit(s) could potentially have been classified as Indicated based purely on the drilling density. However, one or more of the following issues gave rise to an Inferred classification:

- Large number of RC drillholes with few diamond core holes;
- Smaller drillhole sizes in some instances (e.g. BQ);
- Lack of extensive and systematic density determinations throughout the deposit;
- Gaps in the drillhole spacing or coverage and/or larger distances between drillholes;
- Difficulty in domaining of the data to remove possible mixed populations in some instances.

14.6. Taiton Sector

This resource section was completed in 2010 and is detailed in report *"Technical Report on Bau Project in Bau, Sarawak, East Malaysia"*.

Since the above technical report a resource drilling campaign was undertaken in late 2010 and early 2011. The resource drilling campaign was undertaken for only part of the Sector, namely Taiton A (but excluding the Bungaat part), Tabai and an extension to Taiton B. The updated resource for this drilling was published in June 2011. The change to the resource was not significant in terms of the increase in resources and therefore a 43-101 report was not issued. The remaining deposits within the Sector remained unchanged at that time.

A revision to this resource was made in the February 2012 resource estimate release. No additional drilling or resource modelling was done subsequent to the 2010 and 2011 resource estimates; however, there was a change to the cut-off grade. The reason that the grade was lowered is that some preliminary feasibility work identified the possibility that the reserve cutoff grade could be lower than the resource cutoff grade creating a problem situation where there could be reserves not in resource. The previous cutoff grade was 0.75 g/t and this was reduced to 0.5 g/t.

This section will outline the resource changes that occurred in June 2011 to the selected parts of the Sector. The subsequent amendments to the cut-off grade change (0.75 g/t Au to 0.5 g/t Au) for these deposits and the other deposits (where no update work has been undertaken since 2010) was applied in the February 2012 resource release. The data and analysis work for all deposits, whether updated in 2011 or not, is included below for completeness sake.

14.6.1. Introduction & General

The Taiton Sector is situated approximately 2 kilometres south of the town of Bau and is a set of five deposits based on discrete geographical areas as defined by the drilling to date. These deposits have been modelled separately and are Tabai, Overhead Tunnel (combined with Tabai in resource table), Taiton A (including Bungaat), Taiton B/C (excluding underground deposit at Gunung Palaat) and Umbut.

The 2010 resource assessment conducted by Terra Mining Consultants/Stevens & Associates included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;
- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;
- Combining the above data into a clean and validated resource database with associated data being verified;
- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;

Resource estimation work to determine the mineral resource using 3 different estimation techniques;

All data used for this 2010 resource update was supplied or sourced by Besra Gold Inc. (formally Olympus/North Borneo Gold) or determined by Terra Mining Consultants/Stevens & Associates from available information. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

Historical documents and internal reports were reviewed as part of the resource update. Additionally, numerous notes, plans, sections, memoranda and other documents, both in digital and hardcopy format found in the office library and storage, were reviewed.

The 2011 resource update (for certain deposits) followed the industry standard logging, QA/QC and other in-house processes/procedures as defined in previous sections of this report. Any additional historic data (both digital and hardcopy) found in the interim was also included after suitable review and validation as outlined below.

14.6.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of the validations applied are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;

- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

Data from the 2011 drilling campaign was also incorporated with the historic data and followed the standard QA/QC procedures, and the above data review and validation.

14.6.3. Ore Zone Definition

The ore zones at Taiton A, Taiton B/C, Overhead Tunnel, Tabai and Umbut were defined in the 2010 resource work, in the following manner:

- Drillhole sections were created and interpreted faults, geological and mineralized zone grade boundaries (≥0.5 g/t Au lower cut-off) were drawn;
- The grade boundaries were correlated from section to section and cross-checked in plan;
- In the absence of zone continuity, extrapolations were made in between the two drill sections, and up/down dip, using standard methodologies;
- The definition of the mineralized zones and the methodology used was validated visually on each section, and in 3D, and samples within the zone wireframe were analysed;
- The ore zone was terminated using the surveyed topography.

The above methodology was repeated for the 2011 resource update using the historic holes and the holes from the 2010/2011 resource drilling. This applies to the Taiton A, Tabai and Taiton B extension areas.

In the ore zone definition there are isolated cases of assay values below the lower cut-off value. These have only been included where they fall within samples above the cut-off, are of minor effect and cannot be excluded due to their isolated nature.

14.6.4. Statistical Analysis of Data

For the 2010 resource definition, the full Taiton database consisted of 300 drillhole collar entries, 300 drillhole survey entries, 6,078 assay records and 12,029 lithology records.

The historic drilling totalled 19,125.58 metres of drilling in and around the Taiton Sector. The drillhole depths varied from 5 metres to 202.55 metres with an average depth of approximately 64.18 metres. The drillholes consisted of 120 RC holes and 180 diamond cored holes in BQ, NQ, HQ & PQ sizes.

An additional 78 diamond drillholes were completed in the 2011 resource update at the Taiton A, Tabai and Taiton B Extension areas. These additional drillholes comprised 514 drillhole survey entries, 11,592 assay records, 1,948 lithology records as well as density, recovery, mineralisation, alteration, geotechnical and structural data.

The additional drilling, comprising 13,663.85 metres, increased the total drilling to 32,789.43 metres. The new drillhole depths ranged from 49.7 to 509.1 metres with an average depth of 178.87 metres. Overall this approximately doubled the average depth and the maximum depth. All drillholes were collared in PQ to an average depth of 48 metres with the remainder drilled in HQ, except for 4 holes that had to be reduced to NQ where drilling, drillhole depth and drillhole conditions required the hole diameter reduction.

In the 2010 resource definition, the Taiton A deposit (incl. Bungaat) has 54 drillholes, Taiton B/C deposit has 48 drillholes, Tabai deposit has 68 drillholes, Overhead Tunnel deposit has 19 drillholes and Umbut deposit has 78 drillholes. Not all holes were used in the resource definition or estimation.

As a result of the 2010-2011 resource drilling programme, Taiton A (excl. Bungaat zone) has 91 drillholes, Taiton B/C deposit has 60 drillholes, Tabai deposit has 105 drillholes, Overhead Tunnel deposit has 19 drillholes and Umbut deposit has 78 drillholes.

A total of 681 drillhole assay samples fall within the mineralized zone at Taiton A for the 2011 update (663 samples in 2010). Statistics were calculated for gold, density and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-87:Taiton A: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope, inclusive of Bungaat area (which was not altered or updated).

Drillhole Field	Length	Au	Density
Number of Records	681	681	681
Number of Samples	681	661	15
Missing Values	-	20	666
Minimum Value	-	0.01	2.39
Maximum Value	12.80	39.81	2.83
Range	12.80	39.81	0.44
Mean	0.80	3.42	2.66
Variance	0.43	41.67	0.01
Standard Deviation	0.66	6.46	0.09
Standard Error	0.03	0.25	0.02
Skewness	9.19	3.19	- 1.47
Kurtosis	162.85	10.74	4.29
Geometric Mean	0.49	0.87	2.66
Sum of Logs	- 491.58	- 95.58	14.67
Mean of Logs	- 0.72	- 0.14	0.98
Log Variance	7.83	3.82	0.00
Log Estimate of Mean	24.31	5.85	2.66

Table 14-87:Taiton A: Ore Zone Drillhole Sample Statistics

A total of 317 drillhole assay samples fall within the mineralized zone at Taiton B in the 2010 resource definition. For the 2010/11 update an additional 78 drillhole assays resulted from the drilling of the Taiton B extension. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones for the 2010 resource definition and the 2011 update. *Table 14-88: Taiton B: Ore Zone Drillhole Sample* Statistics lists the statistics for the drillhole samples within the mineralised envelope for 2010 resource definition. *Table 14-89: Taiton B Extension: Ore Zone Drillhole Sample Statistics* lists the statistics for the Taiton B Extension used in the 2011 update.

Drillhole Field	Length	Au
Number of Records	317	317
Number of Samples	317	311
Missing Values	-	6
Minimum Value	0.10	0.01
Maximum Value	3.18	14.46
Range	3.08	14.46
Mean	1.04	1.51
Variance	0.10	3.75
Standard Deviation	0.32	1.94
Standard Error	0.02	0.11
Skewness	1.74	4.09
Kurtosis	11.03	21.22
Geometric Mean	0.99	0.66
Sum of Logs	- 4.17	- 127.11
Mean of Logs	- 0.01	- 0.41
Log Variance	0.13	3.34
Log Estimate of Mean	1.05	3.53

Table 14-88: Taiton B: Ore Zone Drillhole Sample Statistics

Drillhole Field	Length	Au
Number of Records	78	78
Number of Samples	78	76
Missing Values	-	2
Minimum Value	0.20	0.01
Maximum Value	1.61	48.80
Range	1.41	48.80
Mean	1.01	2.00
Variance	0.04	33.55
Standard Deviation	0.20	5.79
Standard Error	0.02	0.66
Skewness	- 0.32	7.17
Kurtosis	5.57	53.69
Geometric Mean	0.98	0.68
Sum of Logs	- 1.39	- 28.94
Mean of Logs	- 0.02	- 0.38
Log Variance	0.07	2.83
Log Estimate of Mean	1.02	2.81

Table 14-89: Taiton B Extension: Ore Zone Drillhole Sample Statistics

A total of 1,229 drillhole assay samples fall within the mineralized zone at Tabai for the 2010/11 update (676 samples in 2010). Statistics were calculated for gold, density and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-90: Tabai:*

Ore Zone Drillhole Sample Statistics lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au	Density
Number of Records	1,229	1,229	1,229
Number of Samples	1,229	1,181	18
Missing Values	-	48	1,211
Minimum Value	-	0.01	2.55
Maximum Value	18.50	106.08	2.70
Range	18.50	106.08	0.14
Mean	1.04	2.48	2.65
Variance	1.42	60.15	0.00
Standard Deviation	1.19	7.76	0.04
Standard Error	0.03	0.23	0.01
Skewness	9.00	7.90	- 0.94
Kurtosis	106.79	76.24	- 0.08
Geometric Mean	0.71	0.58	2.65
Sum of Logs	- 414.72	- 634.26	17.54
Mean of Logs	- 0.34	- 0.54	0.97
Log Variance	3.44	3.61	0.00
Log Estimate of Mean	3.99	3.55	2.65

Table 14-90: Tabai: Ore Zone Drillhole Sample Statistics

A total of 496 drillhole assay samples fall within the mineralized zone at Overhead Tunnel for the 2010 resource definition. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-91: Overhead Tunnel: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	496	496
Number of Samples	496	485
Missing Values	-	11
Minimum Value	0.01	0.01
Maximum Value	8.35	31.41
Range	8.34	31.41
Mean	0.82	1.87
Variance	0.37	6.77
Standard Deviation	0.61	2.60
Standard Error	0.03	0.12
Skewness	4.30	5.21
Kurtosis	47.38	41.42
Geometric Mean	0.61	1.11
Sum of Logs	- 244.22	48.52
Mean of Logs	- 0.49	0.10
Log Variance	0.79	1.13
Log Estimate of Mean	0.91	1.95

Table 14-91: Overhead Tunnel: Ore Zone Drillhole Sample Statistics

A total of 338 drillhole assay samples fall within the mineralized zone at Umbut for the 2010 resource definition. Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-92: Umbut: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope.

Drillhole Field	Length	Au
Number of Records	338	338
Number of Samples	338	332
Missing Values	-	6
Minimum Value	0.05	0.01
Maximum Value	6.80	42.00
Range	6.75	42.00
Mean	1.20	2.32
Variance	0.32	17.92
Standard Deviation	0.57	4.23
Standard Error	0.03	0.23
Skewness	3.48	4.73
Kurtosis	27.99	29.56
Geometric Mean	1.09	1.12
Sum of Logs	28.26	36.24
Mean of Logs	0.08	0.11
Log Variance	0.23	1.35
Log Estimate of Mean	1.22	2.19

Table 14-92: Umbut: Ore Zone Drillhole Sample Statistics

Samples within the Taiton A orezone (excl. Bungaat zone) were composited to 1 metre lengths, resulting in 547 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-93: Taiton A: Main Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Taiton A main zone.

Drillhole Field	Length	Au
Number of Records	547	547
Number of Samples	547	523
Missing Values	-	24
Minimum Value	0.50	0.01
Maximum Value	1.00	39.81
Range	0.50	39.81
Mean	0.98	2.54
Variance	0.01	19.99
Standard Deviation	0.08	4.47
Standard Error	0.00	0.20
Skewness	- 3.97	3.83
Kurtosis	15.76	18.52
Geometric Mean	0.98	0.78
Sum of Logs	- 13.55	- 130.71
Mean of Logs	- 0.02	- 0.25
Log Variance	0.01	3.63
Log Estimate of Mean	0.98	4.79

Table 14-93: Taiton A: Main Ore Zone Composited Drillhole Sample Statistics

Samples within the Taiton B orezone were composited to 1 metre lengths, resulting in 332 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-94: Taiton B: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Taiton B. Taiton B Extension drillholes were predominantly in 1 metre lengths compositing was not required, and when statistics determined they did not materially alter the non-composited values.

Drillhole Field	Length	Au
Number of Records	332	332
Number of Samples	332	328
Missing Values	-	4
Minimum Value	0.50	0.01
Maximum Value	1.00	14.46
Range	0.50	14.46
Mean	0.99	1.47
Variance	0.00	3.23
Standard Deviation	0.07	1.80
Standard Error	0.00	0.10
Skewness	- 6.29	4.21
Kurtosis	39.48	23.97
Geometric Mean	0.99	0.67
Sum of Logs	- 4.70	- 131.80
Mean of Logs	- 0.01	- 0.40
Log Variance	0.01	3.24
Log Estimate of Mean	0.99	3.38

Table 14-94: Taiton B: Ore Zone Composited Drillhole Sample Statistics

Samples within the Tabai orezone were composited to 1 metre lengths, resulting in 1,307 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-95: Tabai: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Tabai.

Drillhole Field	Length	Au
Number of Records	1,307	1,307
Number of Samples	1,307	1,157
Missing Values	-	150
Minimum Value	0.50	0.01
Maximum Value	1.00	98.60
Range	0.50	98.60
Mean	0.97	2.45
Variance	0.01	53.69
Standard Deviation	0.11	7.33
Standard Error	0.00	0.22
Skewness	- 3.39	8.18
Kurtosis	10.22	82.50
Geometric Mean	0.96	0.65
Sum of Logs	- 56.34	- 502.99
Mean of Logs	- 0.04	- 0.43
Log Variance	0.02	3.44
Log Estimate of Mean	0.97	3.61

Table 14-95: Tabai: Ore Zone Composited Drillhole Sample Statistics

Samples within the Overhead Tunnel orezone were composited to 1 metre lengths, resulting in 405 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-96: Overhead Tunnel: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Overhead Tunnel.

Drillhole Field	Length	Au
Number of Records	405	405
Number of Samples	405	394
Missing Values	-	11
Minimum Value	0.50	0.01
Maximum Value	1.00	31.41
Range	0.50	31.41
Mean	0.99	1.94
Variance	0.01	7.96
Standard Deviation	0.07	2.82
Standard Error	0.00	0.14
Skewness	- 5.48	6.60
Kurtosis	30.02	60.22
Geometric Mean	0.98	1.18
Sum of Logs	- 7.19	65.63
Mean of Logs	- 0.02	0.17
Log Variance	0.01	1.04
Log Estimate of Mean	0.99	1.99

Table 14-96: Overhead Tunnel: Ore Zone Composited Drillhole Sample Statistics

Samples within the orezone were composited to 1 metre lengths, resulting in 412 composites. Composites were set at 1 metre as this was the pre*dominant sample length and close to the average sample length*. Table 14-97: Umbut: Ore Zone Composited Drillhole Sample Statistics lists the statistics for the composited drillholes for Umbut.

Drillhole Field	Length	Au
Number of Records	412	412
Number of Samples	412	402
Missing Values	-	10
Minimum Value	0.50	0.01
Maximum Value	1.00	38.27
Range	0.50	38.26
Mean	0.97	2.45
Variance	0.01	16.40
Standard Deviation	0.10	4.05
Standard Error	0.01	0.20
Skewness	- 3.93	4.01
Kurtosis	14.22	21.05
Geometric Mean	0.97	1.24
Sum of Logs	- 14.21	85.28
Mean of Logs	- 0.03	0.21
Log Variance	0.02	1.24
Log Estimate of Mean	0.98	2.29

Table 14-97: Umbut: Ore Zone Composited Drillhole Sample Statistics

The Taiton A Au data shown statistically above is also shown in graphical form below. *Figure 14-76: Taiton A: Log Histogram of Au Ore Zone Composites* and *Figure 14-77: Taiton A: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

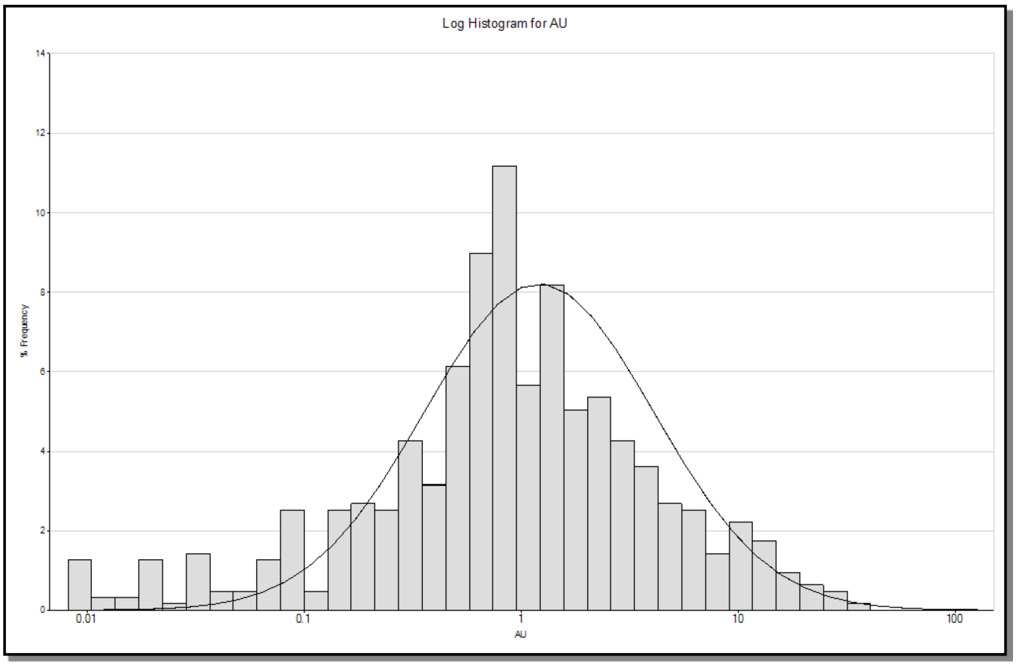


Figure 14-76: Taiton A: Log Histogram of Au Ore Zone Composites

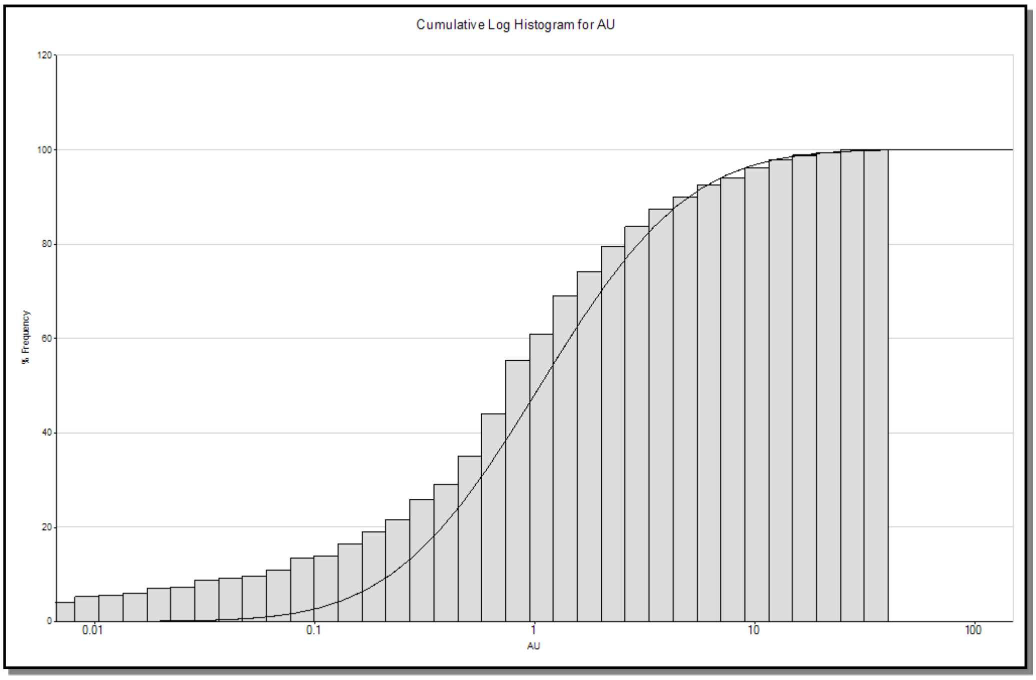


Figure 14-77: Taiton A: Cumulative Log Histogram of Au Ore Zone Composites

The Taiton B Au data shown statistically above is also shown in graphical form below. *Figure 14-78: Taiton B: Log Histogram of Au Ore Zone Composites* and *Figure 14-79: Taiton B: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

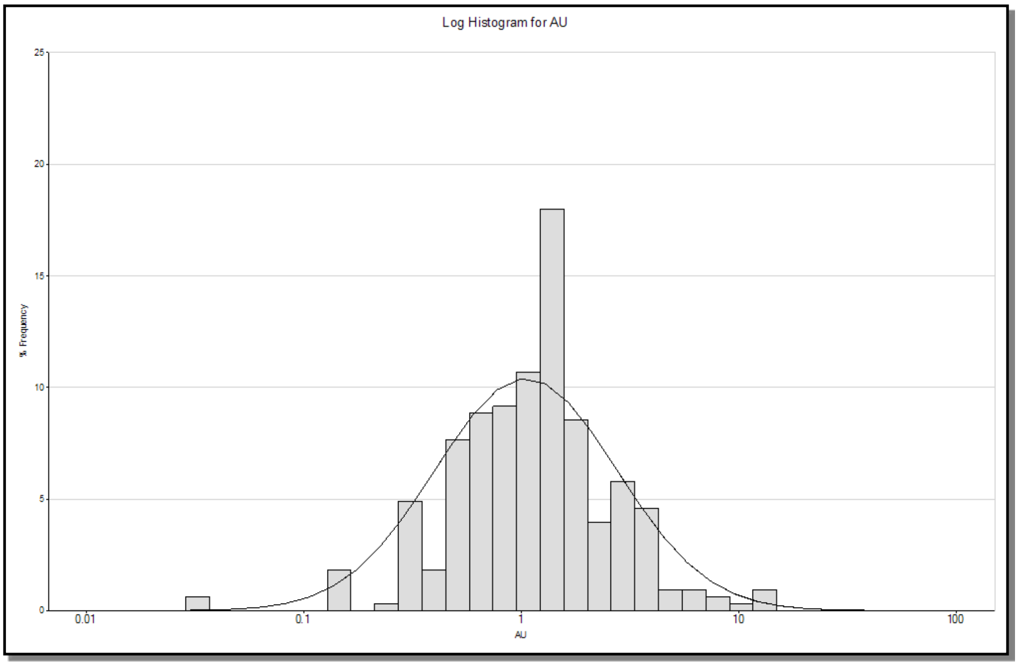


Figure 14-78: Taiton B: Log Histogram of Au Ore Zone Composites

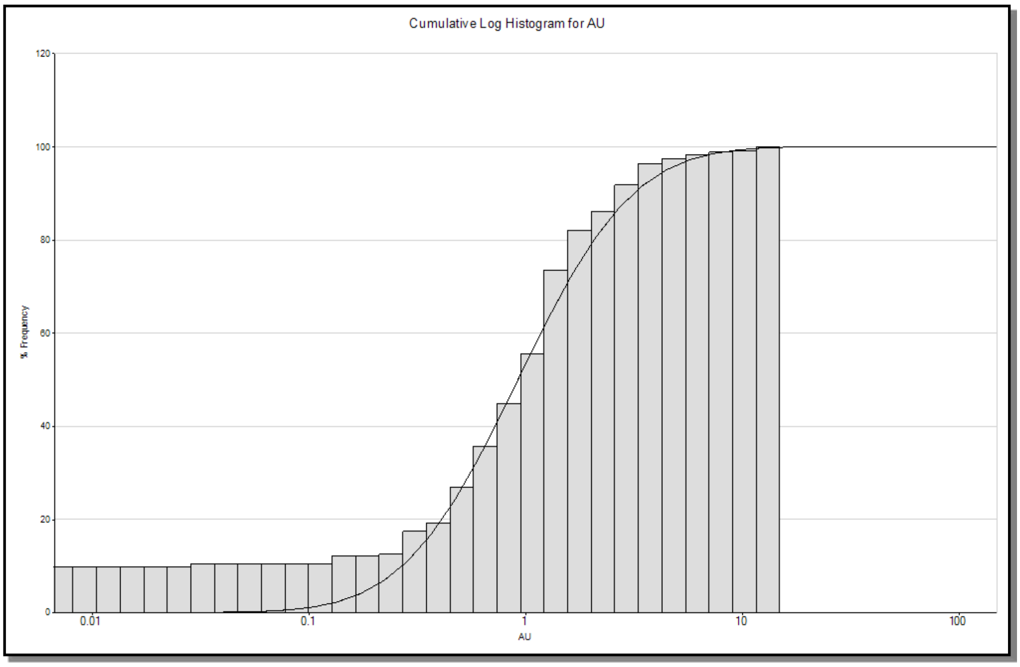


Figure 14-79: Taiton B: Cumulative Log Histogram of Au Ore Zone Composites

The Tabai Au data shown statistically above is also shown in graphical form below. *Figure 14-80: Tabai: Log Histogram of Au Ore Zone Composites* and *Figure 14-81: Tabai: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

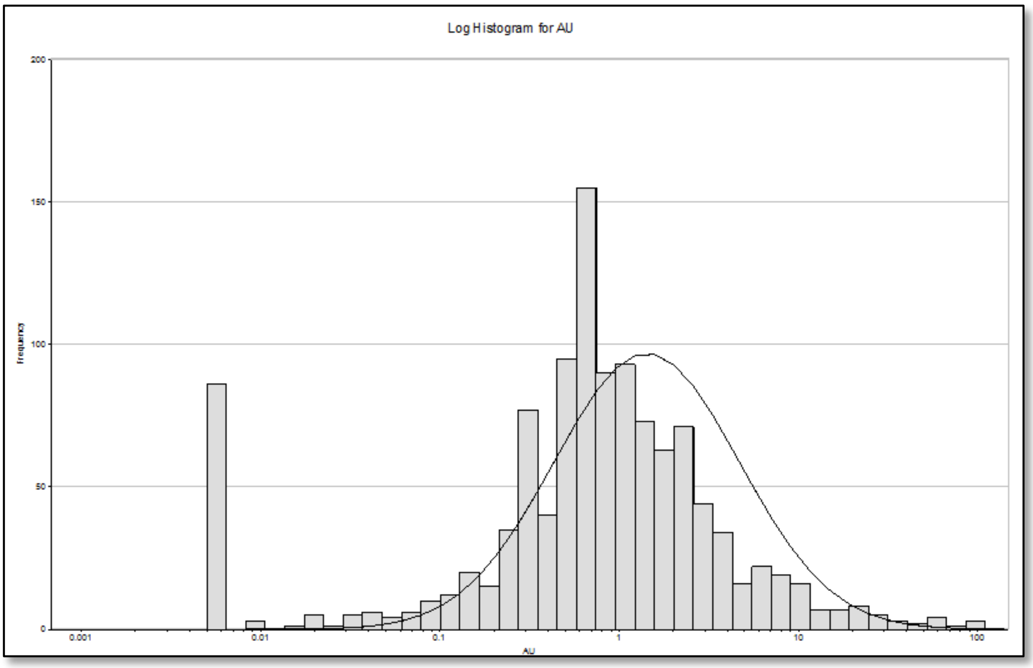


Figure 14-80: Tabai: Log Histogram of Au Ore Zone Composites

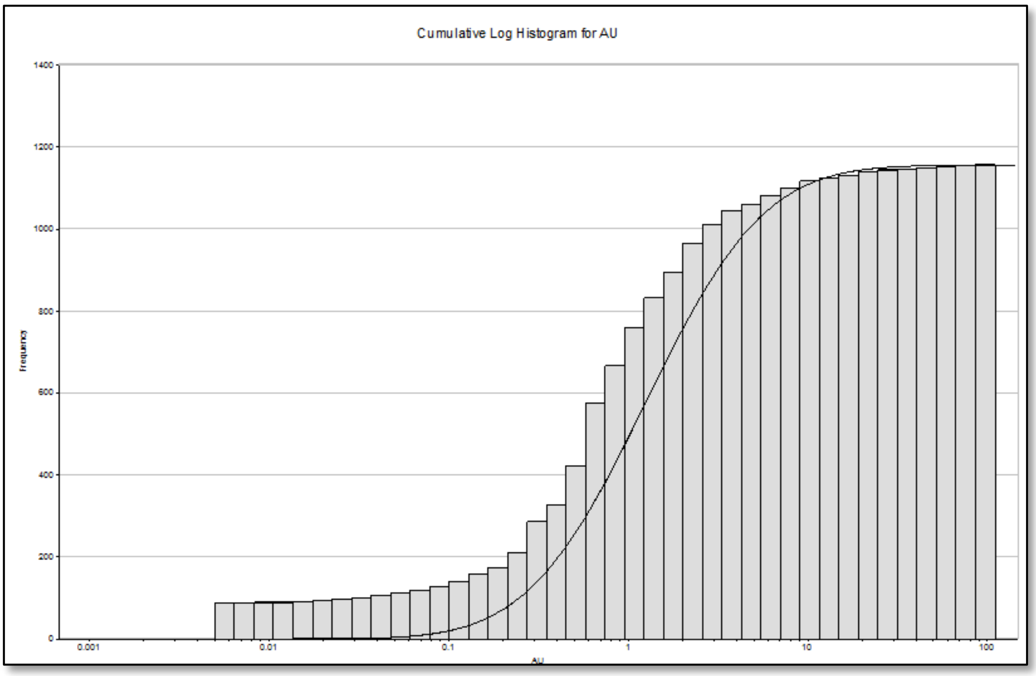


Figure 14-81: Tabai: Cumulative Log Histogram of Au Ore Zone Composites

The Overhead Tunnel Au data shown statistically above is also shown in graphical form below. *Figure 14-82: Overhead Tunnel: Log Histogram of Au Ore Zone Composites* and *Figure 14-83: Overhead Tunnel: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

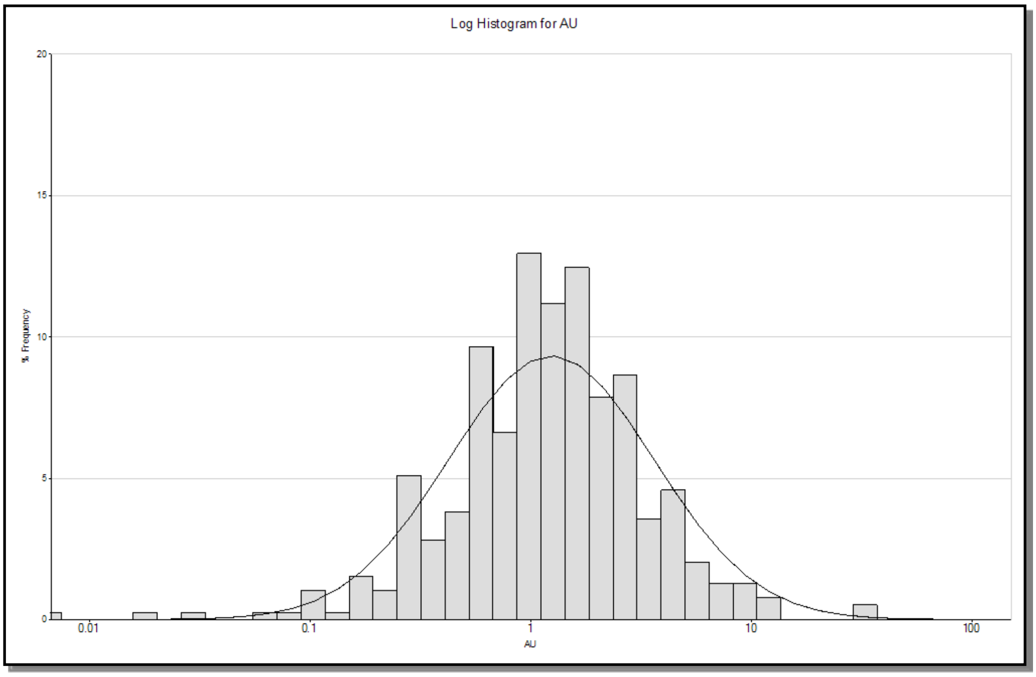


Figure 14-82: Overhead Tunnel: Log Histogram of Au Ore Zone Composites

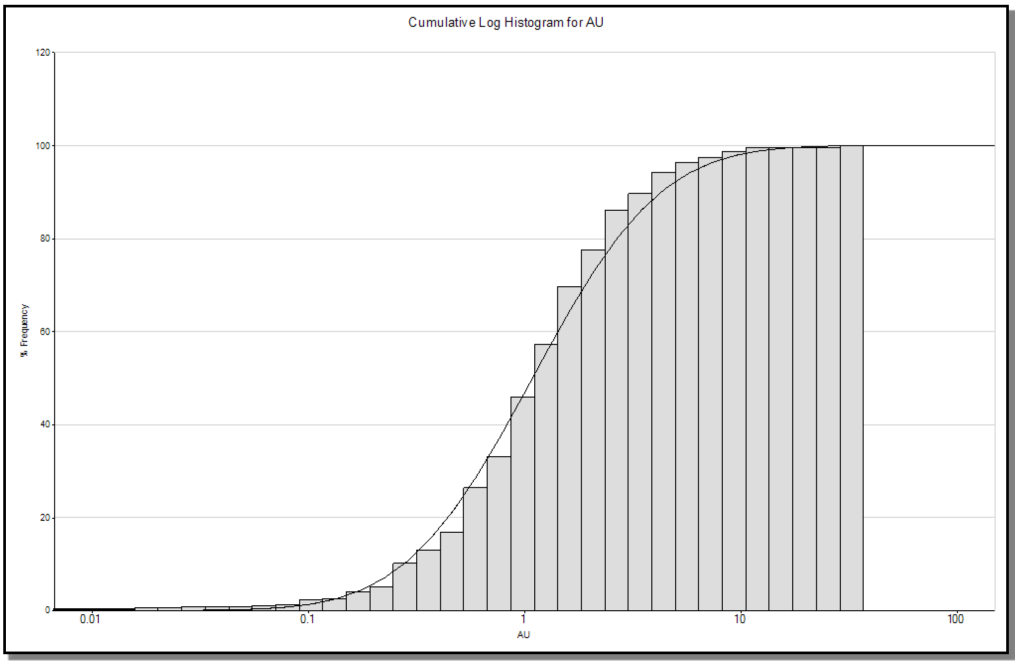


Figure 14-83: Overhead Tunnel: Cumulative Log Histogram of Au Ore Zone Composites

The Umbut Au data shown statistically above is also shown in graphical form below. *Figure 14-84: Umbut: Log Histogram of Au Ore Zone Composites* and *Figure 14-85: Umbut: Cumulative Log Histogram of Au Ore Zone Composites* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

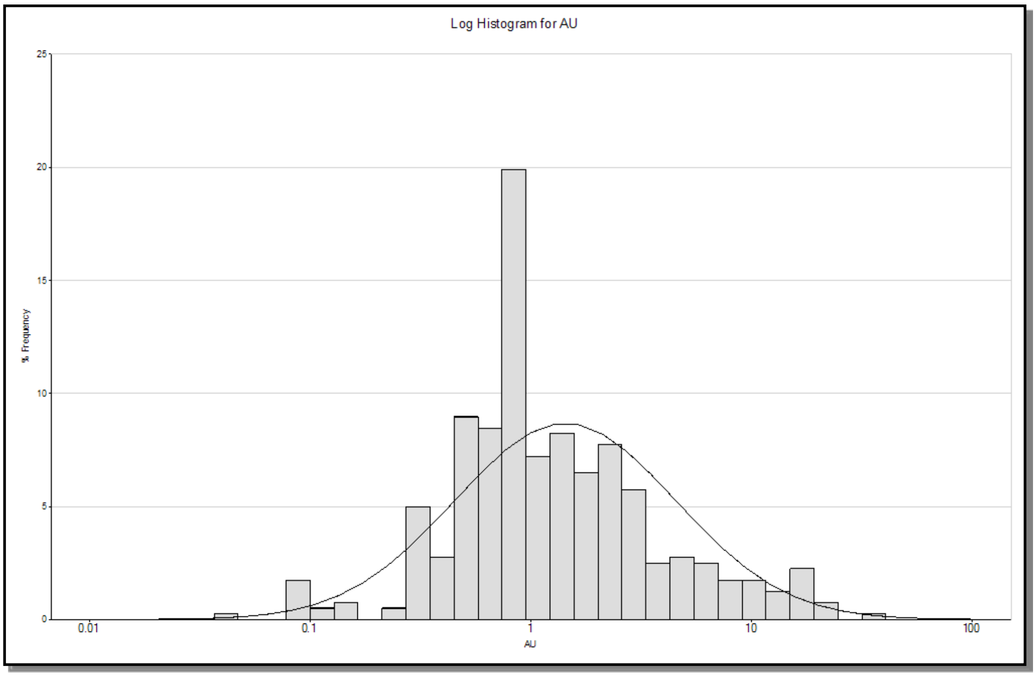


Figure 14-84: Umbut: Log Histogram of Au Ore Zone Composites

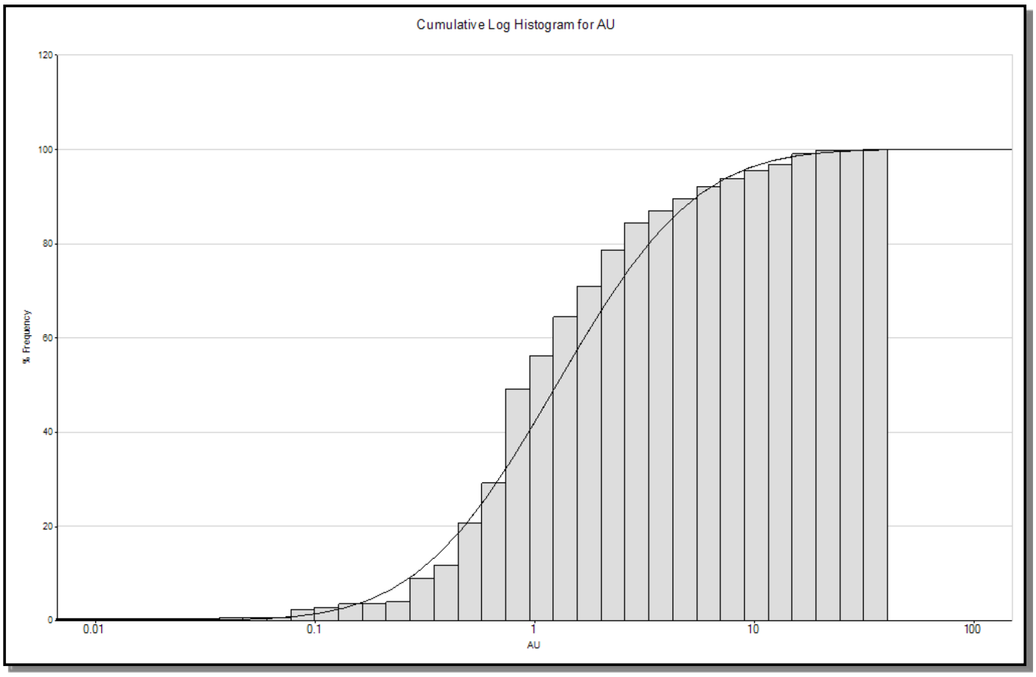


Figure 14-85: Umbut: Cumulative Log Histogram of Au Ore Zone Composites

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-98: Taiton A: Quantile Analysis of Au Drillhole Composites* and *Table 14-103: Umbut: Quantile Analysis of Au Drillhole Composites* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Taiton A, Taiton B, Taiton B Extension, Overhead Tunnel, Tabai and Umbut Ore Zones. The Taiton B, Overhead Tunnel and Umbut are based on the 2010 data with the 2011 data shown in the Taiton A (excl. Bungaat zone), Tabai and Taiton B Extension.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	52	0.01	0.01	0.04	0.65	0.05
10	20	52	0.15	0.04	0.27	7.73	0.58
20	30	52	0.47	0.28	0.59	24.31	1.83
30	40	53	0.67	0.59	0.78	35.65	2.68
40	50	52	0.86	0.78	0.96	44.88	3.38
50	60	52	1.17	0.96	1.40	60.98	4.59
60	70	53	1.66	1.40	1.96	87.83	6.61
70	80	52	2.37	1.97	3.06	123.18	9.28
80	90	52	4.36	3.10	6.29	226.90	17.09
90	100	53	13.51	6.32	39.81	715.79	53.90
90	92.5	13	7.31	6.32	8.43	95.01	7.15
92.5	95	13	9.84	8.75	10.89	127.93	9.63
95	97.5	13	12.86	10.89	15.94	167.17	12.59
97.5	100	14	23.26	16.47	39.81	325.69	24.53
0	100	523	2.54	0.01	39.81	1,327.90	100.00

Table 14-98: Taiton A: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	32	0.01	0.01	0.01	0.16	0.03
10	20	33	0.30	0.03	0.48	9.85	2.05
20	30	33	0.56	0.50	0.65	18.37	3.82
30	40	33	0.73	0.66	0.85	24.09	5.01
40	50	33	0.97	0.86	1.09	31.90	6.63
50	60	32	1.20	1.10	1.25	38.43	7.99
60	70	33	1.38	1.25	1.49	45.48	9.45
70	80	33	1.66	1.50	2.00	54.87	11.40
80	90	33	2.51	2.00	3.18	82.87	17.22
90	100	33	5.31	3.18	14.46	175.20	36.41
90	92.5	8	3.30	3.18	3.36	26.36	5.48
92.5	95	8	3.40	3.36	3.45	27.22	5.66
95	97.5	8	4.21	3.45	5.44	33.67	7.00
97.5	100	9	9.77	6.07	14.46	87.95	18.28
0	100	328	1.47	0.01	14.46	481.22	100.00

Table 14-99: Taiton B: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	7	0.02	0.01	0.12	0.16	0.11
10	20	8	0.26	0.12	0.42	2.06	1.35
20	30	7	0.51	0.45	0.56	3.59	2.36
30	40	8	0.60	0.57	0.64	4.81	3.16
40	50	8	0.68	0.65	0.76	5.47	3.59
50	60	7	0.97	0.78	1.14	6.80	4.46
60	70	8	1.32	1.14	1.46	10.52	6.90
70	80	7	1.67	1.50	1.83	11.69	7.67
80	90	8	2.49	1.90	3.09	19.95	13.09
90	100	8	10.92	3.09	48.80	87.33	57.31
90	92.5	2	3.11	3.09	3.13	6.22	4.08
92.5	95	2	3.41	3.33	3.48	6.81	4.47
95	97.5	2	4.15	3.80	4.50	8.30	5.45
97.5	100	2	33.00	17.20	48.80	66.00	43.31
0	100	76	2.00	0.01	48.80	152.38	100.00

Table 14-100: Taiton B Extension: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	118	0.01	0.01	0.05	1.26	0.04
10	20	118	0.14	0.05	0.24	16.63	0.57
20	30	118	0.32	0.24	0.38	37.70	1.29
30	40	118	0.50	0.38	0.58	58.88	2.01
40	50	118	0.65	0.59	0.69	76.86	2.63
50	60	118	0.82	0.69	1.00	96.90	3.31
60	70	118	1.14	1.01	1.36	134.56	4.60
70	80	118	1.68	1.36	2.06	198.10	6.78
80	90	118	2.79	2.06	3.95	329.37	11.27
90	100	119	16.58	3.97	106.08	1,973.20	67.50
90	92.5	29	4.80	3.97	5.87	139.31	4.77
92.5	95	30	7.18	6.12	8.32	215.28	7.36
95	97.5	30	12.11	8.73	15.04	363.32	12.43
97.5	100	30	41.84	15.04	106.08	1,255.29	42.94
0	100	1181	2.48	0.01	106.08	2,923.46	100.00

Table 14-101: Tabai: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	39	0.21	0.01	0.31	8.17	1.07
10	20	39	0.47	0.31	0.60	18.24	2.39
20	30	40	0.66	0.61	0.75	26.54	3.48
30	40	39	0.91	0.78	1.02	35.36	4.64
40	50	40	1.12	1.02	1.25	44.97	5.90
50	60	39	1.37	1.25	1.49	53.54	7.02
60	70	39	1.67	1.50	1.89	65.19	8.55
70	80	40	2.23	1.90	2.49	89.22	11.70
80	90	39	3.02	2.55	3.93	117.66	15.43
90	100	40	7.59	3.94	31.41	303.62	39.82
90	92.5	10	4.14	3.94	4.34	41.43	5.43
92.5	95	10	4.84	4.47	5.49	48.40	6.35
95	97.5	10	6.52	5.55	7.85	65.16	8.55
97.5	100	10	14.86	8.59	31.41	148.62	19.49
0	100	394	1.94	0.01	31.41	762.52	100.00

Table 14-102: Overhead Tunnel: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	40	0.24	0.01	0.39	9.58	0.97
10	20	40	0.48	0.39	0.57	19.30	1.96
20	30	40	0.67	0.57	0.78	26.74	2.72
30	40	40	0.83	0.78	0.88	33.10	3.36
40	50	41	0.92	0.88	0.98	37.90	3.85
50	60	40	1.15	0.98	1.32	46.18	4.69
60	70	40	1.62	1.34	1.96	64.78	6.58
70	80	40	2.29	1.96	2.71	91.56	9.31
80	90	40	3.69	2.71	5.60	147.60	15.00
90	100	41	12.37	5.63	38.27	507.08	51.54
90	92.5	10	6.25	5.63	7.12	62.47	6.35
92.5	95	10	8.70	7.70	9.99	86.97	8.84
95	97.5	10	12.98	9.99	16.27	129.77	13.19
97.5	100	11	20.72	16.62	38.27	227.88	23.16
0	100	402	2.45	0.01	38.27	983.82	100.00

Table 14-103: Umbut: Quantile Analysis of Au Drillhole Composites

For Taiton A, looking at the primary percentiles, it can be seen that approximately 54 % of the metal percentage can be found in the top 10 % range, and that there is a significant jump in the mean grade and metal content from the previous range. For Taiton B this is approximately 36 %, Taiton B Extension this is approximately 57 %, Tabai approximately 67.5 %, Overhead Tunnel approximately 40 % and Umbut 52 %.

Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains nearly 24.5 % of the Au metal content for Taiton A, 18 % for Taiton B, 43 % for Taiton B Extension, 43 % for Tabai, 19% for Overhead Tunnel and 23 % for Umbut.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top-cut of 23.26 g/t Au (mean of the 97.5 percentile) should be applied to the Taiton A samples above this value in order to remove any effect of the high grade samples in the estimation process. Similarly, a top-cut of 9.77 g/t Au for Taiton B, 33 g/t Au for Taiton B Extension, 41.84 g/t Au for Tabai, 14.86 g/t Au for Overhead Tunnel and 20.72 g/t Au for Umbut.

14.6.5. Semi-Variogram Analysis

Semi-variogram analyses were undertaken to determine the semi-variogram parameters for use in the Ordinary Kriging. Downhole, horizontal and vertical increment semi-variograms were generated with the best semi-variograms selected that defines the strike, dip and dip direction. These semi-variograms were used to determine the nugget, sill values and ranges.

A log semi-variogram and two-range spherical model were used. A best fit model in the downhole semi-variogram was used to define the nugget. Subsequent model fitting was applied to the strike and dip/dip-direction to define the sill values by varying the ranges in these directions. The semi-variogram parameters are listed in *Table 14-106: Taiton A: Ordinary Kriging Estimation Parameters* to *Table 14-111: Umbut: Ordinary Kriging Estimation Parameters* in *Section 16.1.6.7* below.

The semi-variograms for Taiton A are shown below in *Figure 14-86: Taiton A: Downhole Semi-Variogram and Figure 14-87: Taiton A: Horizontal Semi-Variogram.*

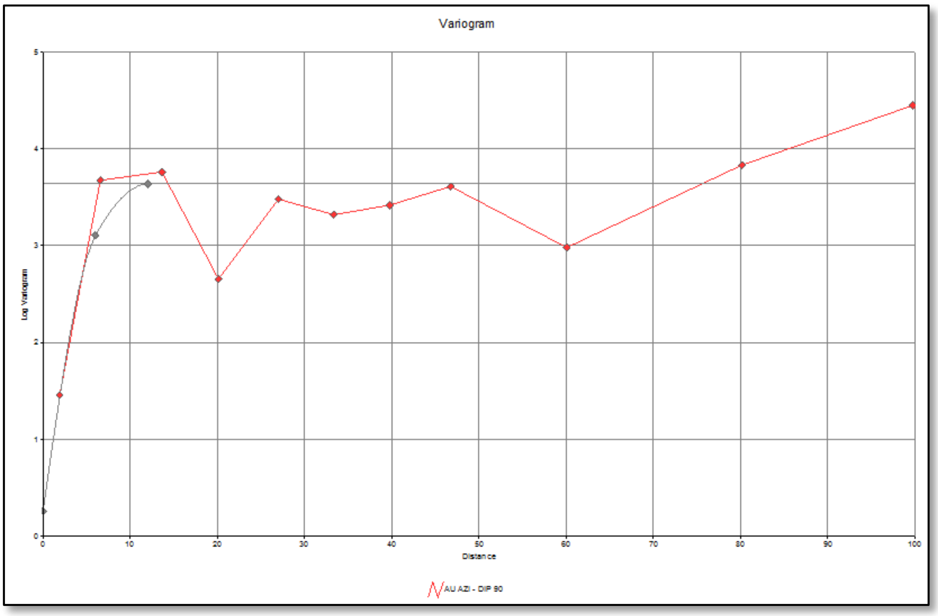


Figure 14-86: Taiton A: Downhole Semi-Variogram

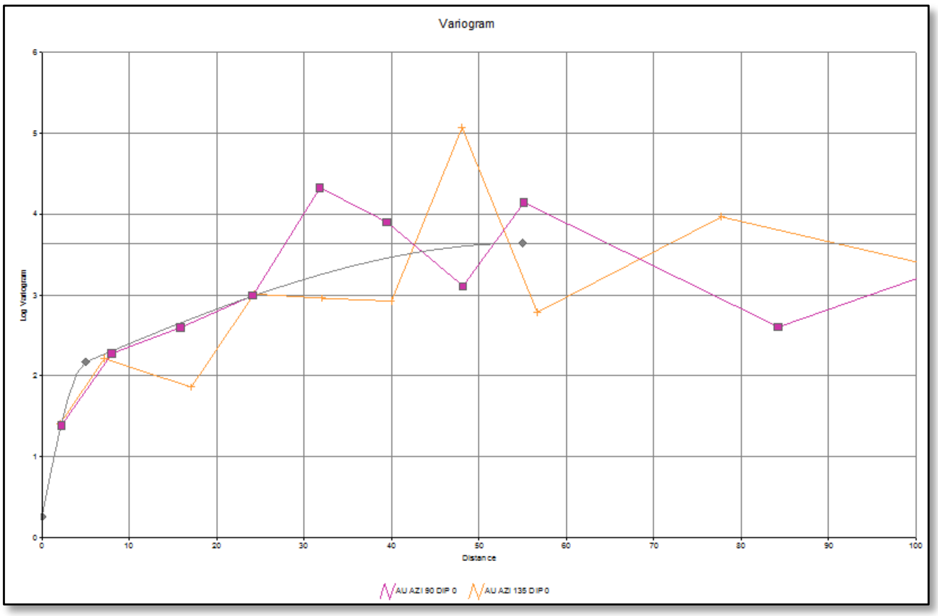


Figure 14-87: Taiton A: Horizontal Semi-Variogram

The semi-variograms for Taiton B are shown below in *Figure 14-88: Taiton B: Downhole Semi-Variogram* and *Figure 14-89: Taiton B: Uni-Directional Semi-Variogram.*

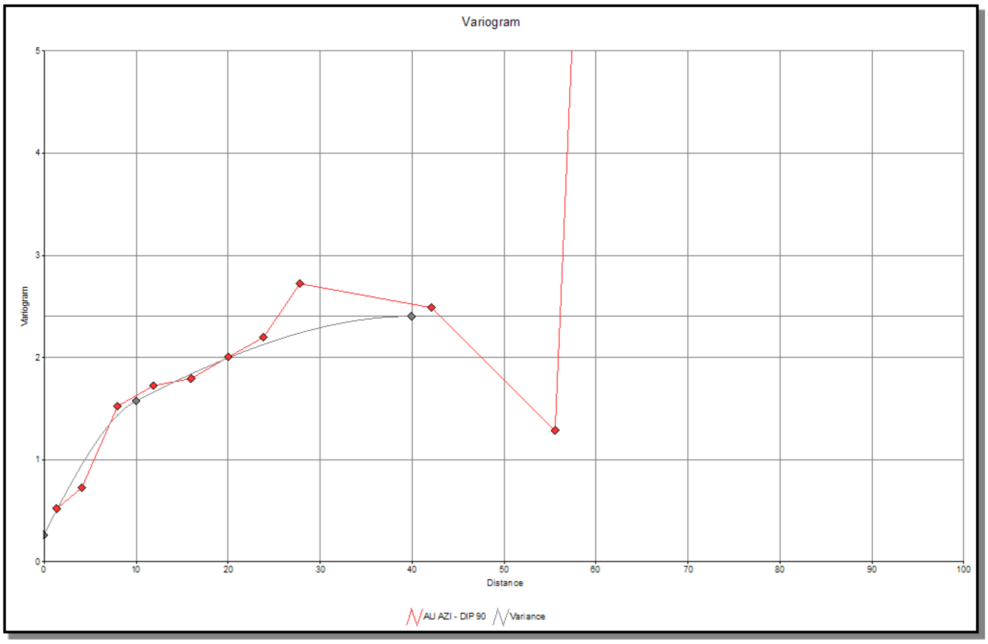


Figure 14-88: Taiton B: Downhole Semi-Variogram

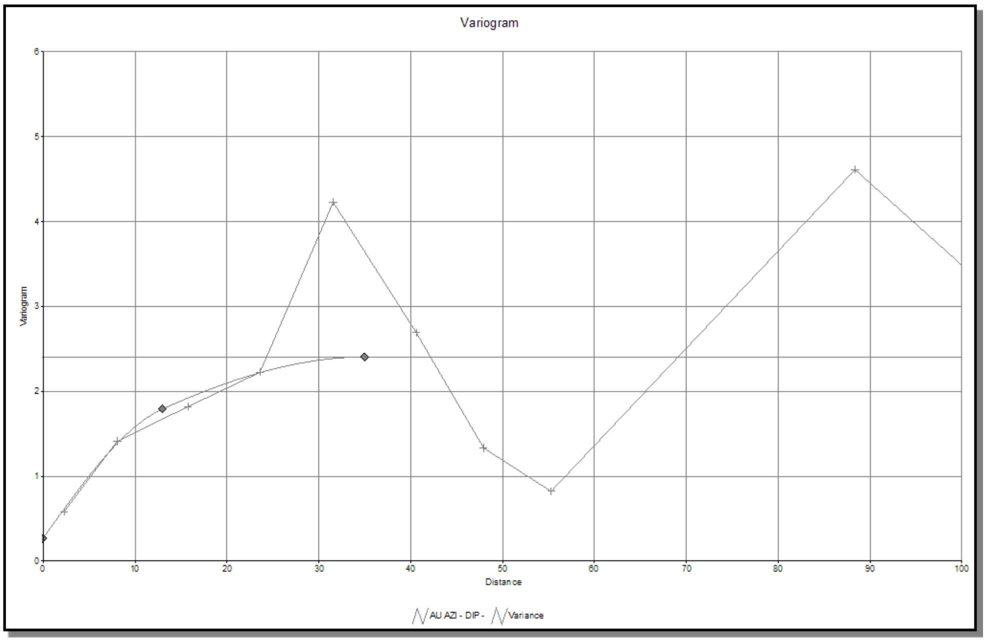


Figure 14-89: Taiton B: Uni-Directional Semi-Variogram

The semi-variograms for Tabai are shown below in *Figure 14-90: Tabai: Downhole Semi-Variogram* and *Figure 14-92: Tabai: Inclined Semi-Variograms.*

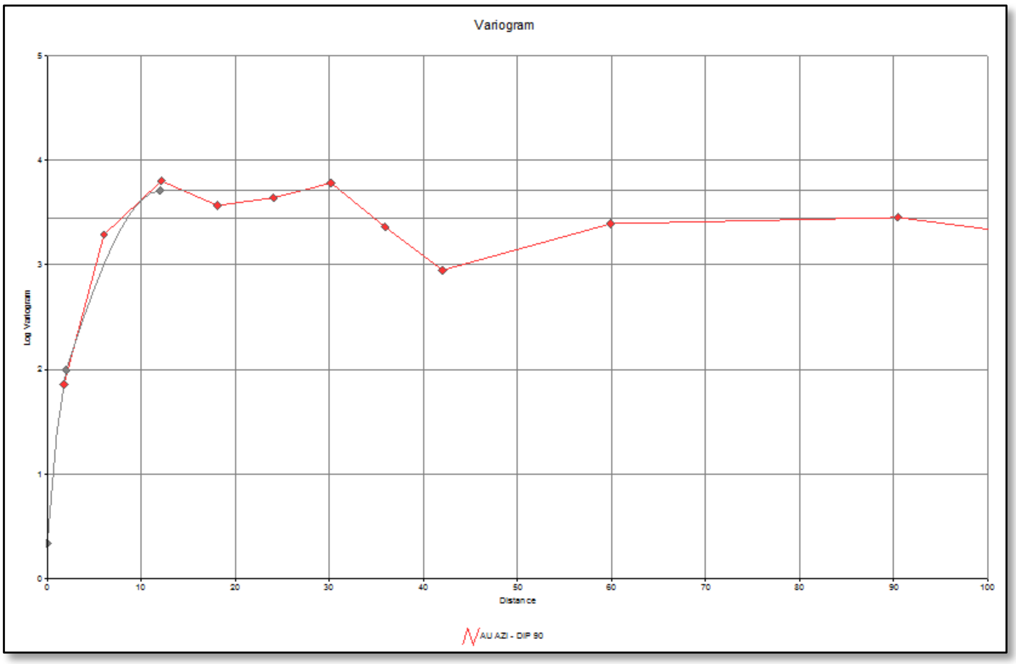


Figure 14-90: Tabai: Downhole Semi-Variogram

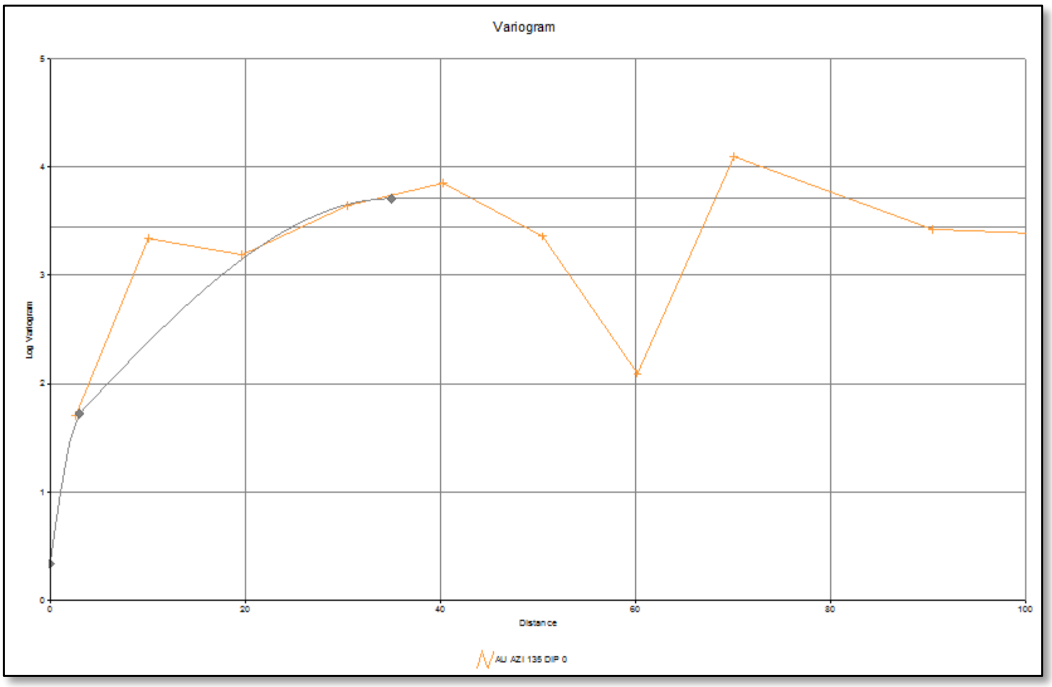


Figure 14-91: Tabai: Horizontal Semi-Variogram

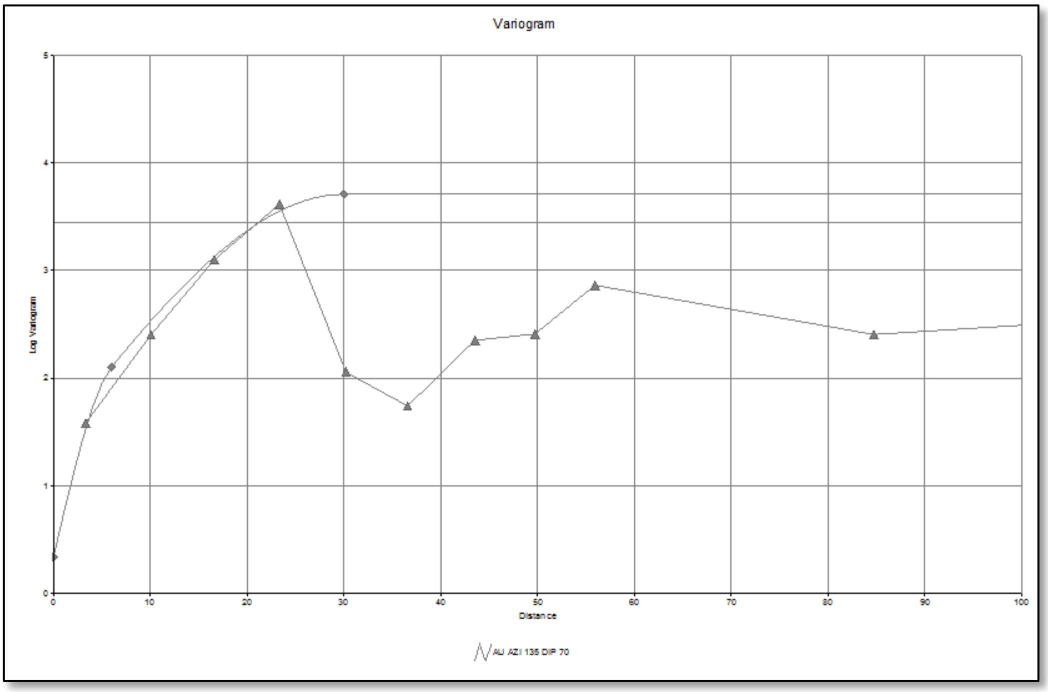


Figure 14-92: Tabai: Inclined Semi-Variograms

The semi-variograms for Overhead Tunnel are shown below in *Figure 14-93: Overhead Tunnel: Downhole Semi-Variogram* and *Figure 14-94: Overhead Tunnel: Horizontal Semi-Variogram.*

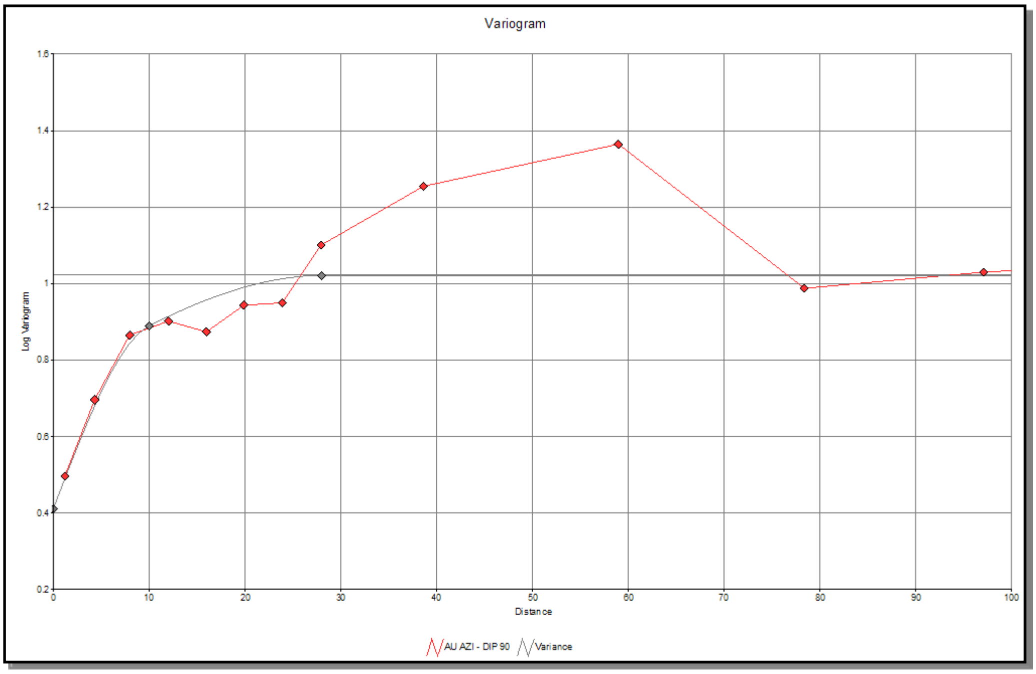


Figure 14-93: Overhead Tunnel: Downhole Semi-Variogram

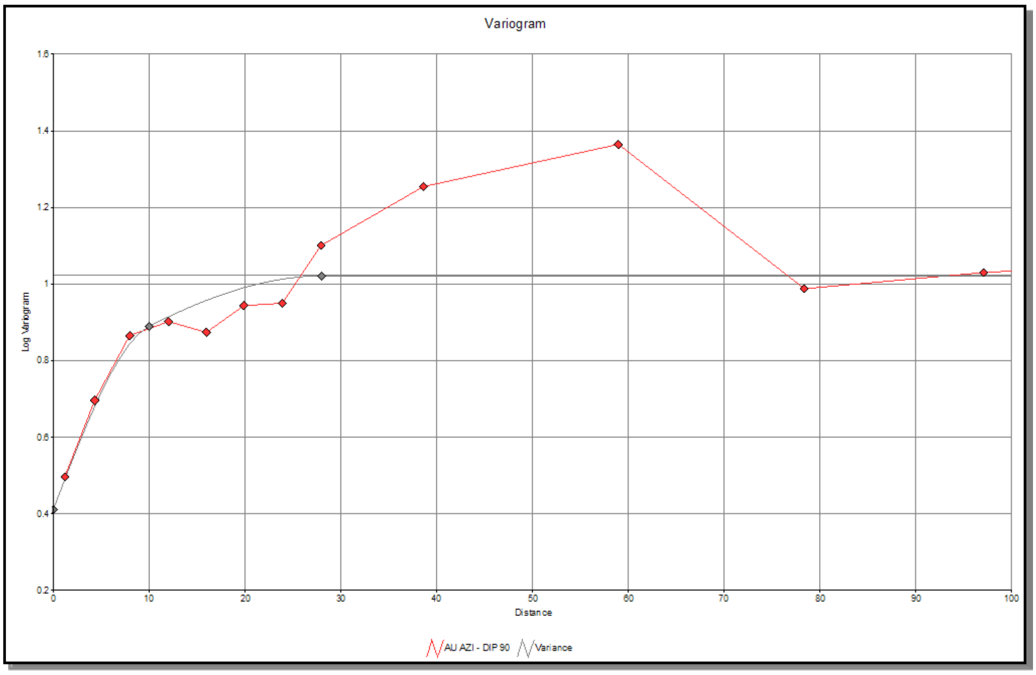


Figure 14-94: Overhead Tunnel: Horizontal Semi-Variogram

The semi-variograms for Umbut are shown below in *Figure 14-95: Umbut: Downhole Semi-Variogram* and *Figure 14-97: Umbut: Alternate Inclined Semi-Variogram*.

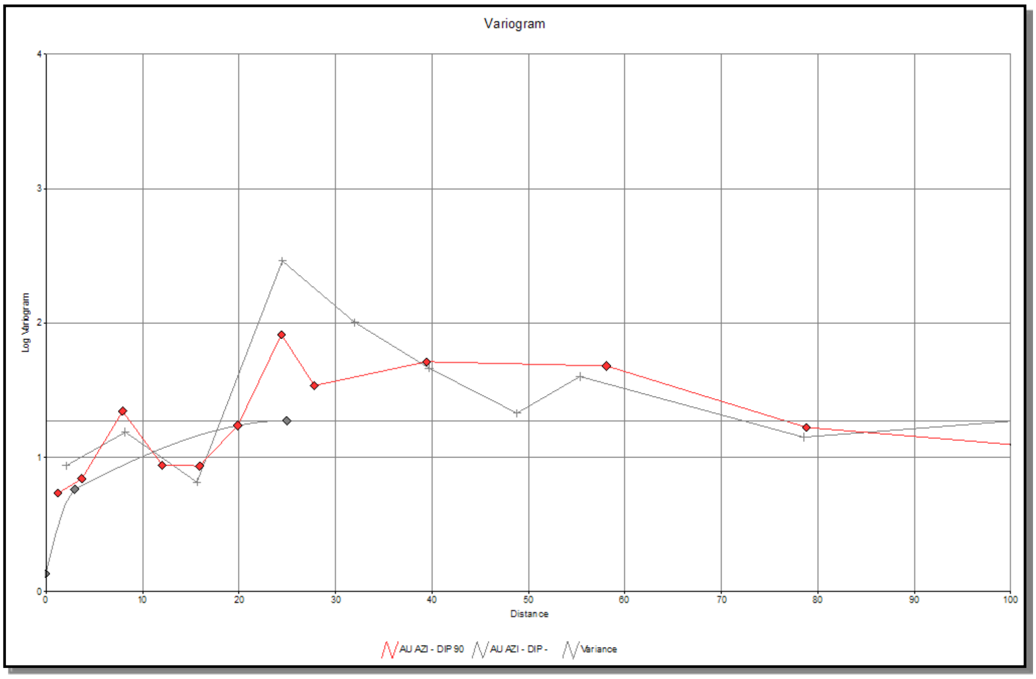


Figure 14-95: Umbut: Downhole Semi-Variogram

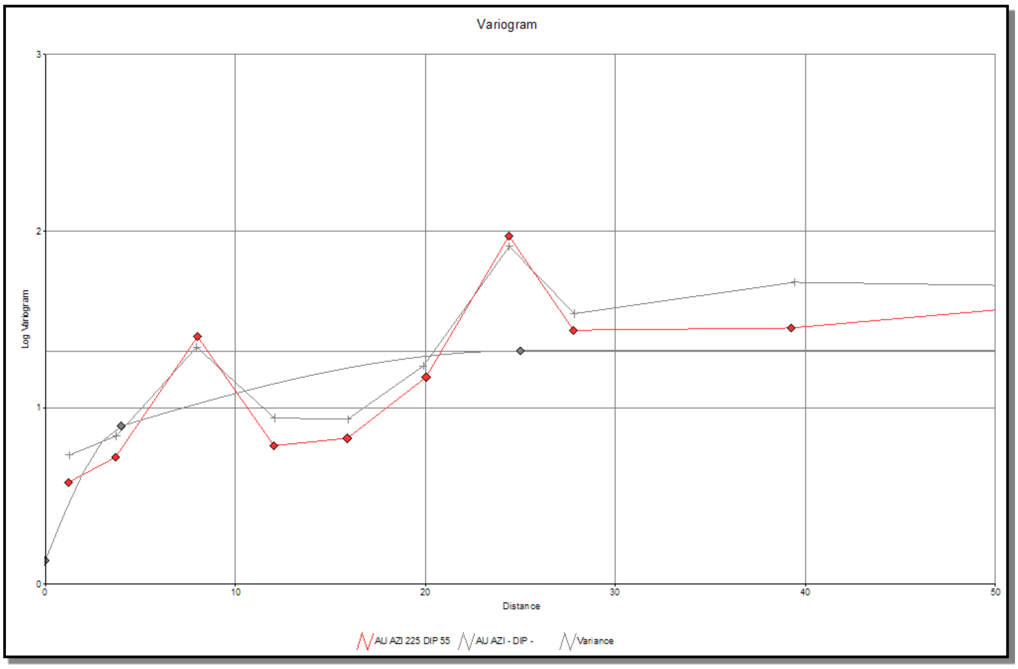


Figure 14-96: Umbut: Inclined Semi-Variogram

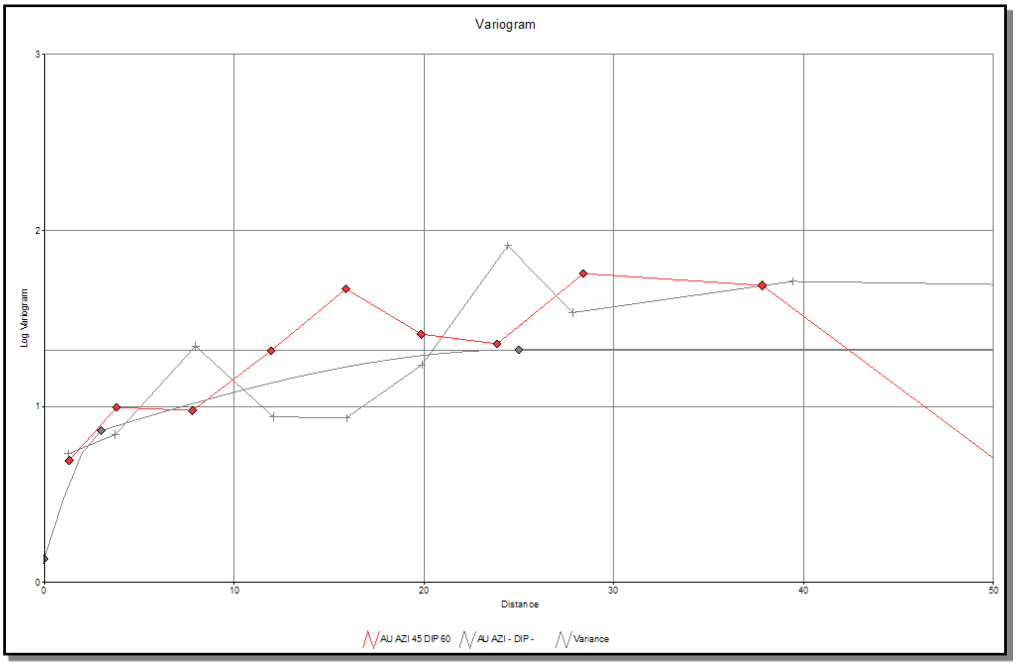


Figure 14-97: Umbut: Alternate Inclined Semi-Variogram

The modelled log semi-variogram values were back calculated to normal semi-variograms for use with Ordinary Kriging. The back transform for Taiton A, Taiton B, Tabai, Overhead Tunnel and Umbut are shown in *Figure 14-98: Taiton A: Log to Normal Semi-Variogram Transform* and *Figure 14-102: Umbut: Log to Normal Semi-Variogram Transform.*

Converting log to normal variograms:

Enter the following data:

log variance :	3.64	**non-standardised**
log nugget :	0.26	0
log sill 1:	1.68	5
log sill 2:	1.70	55
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.24**
sill 1:	**0.38**
sill 2:	**0.38**
sill 3:	**0.00**

Figure 14-98: Taiton A: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:		
Enter the following data:		
log variance :	2.40	non-standardised
log nugget :	0.26	0
log sill 1:	0.83	5
log sill 2:	1.31	55
log sill 3:	0.00	
The transformed nugget and sills are:		
nugget:	0.25	
sill 1:	0.29	
sill 2:	0.46	
sill 3:	0.00	

Figure 14-99: Taiton B: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:		
Enter the following data:		
log variance :	3.71	non-standardised
log nugget :	0.34	0
log sill 1:	1.09	12
log sill 2:	2.28	35
log sill 3:	0.00	
The transformed nugget and sills are:		
nugget:	0.30	
sill 1:	0.23	
sill 2:	0.48	
sill 3:	0.00	

Figure 14-100: Tabai: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:		
Enter the following data:		
log variance :	1.02	non-standardised
log nugget :	0.41	0
log sill 1:	0.34	5
log sill 2:	0.27	55
log sill 3:	0.00	
The transformed nugget and sills are:		
nugget:	0.53	
sill 1:	0.26	
sill 2:	0.21	
sill 3:	0.00	

Figure 14-101: Overhead Tunnel: Log to Normal Semi-Variogram Transform

Converting log to normal variograms:

Enter the following data:

log variance :	1.32	**non-standardised**
log nugget :	0.13	0
log sill 1:	0.63	3
log sill 2:	0.56	25
log sill 3:	0.00	

The transformed nugget and sills are:

nugget:	**0.17**
sill 1:	**0.44**
sill 2:	**0.39**
sill 3:	**0.00**

Figure 14-102: Umbut: Log to Normal Semi-Variogram Transform

14.6.6. Previous Resource Estimates

The Taiton sector and deposits has been the subject of only one previous resource estimate, which was conducted by Terra Mining Consultants/Stevens and Associates in their 2010 report. The Taiton B, Overhead Tunnel and Umbut resource estimations conducted in 2010 are included in this report in the interests of completeness. The Taiton A (excl. Bungaat zone), Taiton B and Tabai 2011 resource updates are included as part of this report update. The historic estimates for these areas can be found in the 2010 technical report titled "*Technical Report on Bau Project in Bau, Sarawak, East Malaysia*" compiled by Terra Mining Consultants/Stevens & Associates, and are summarised below:

- Taiton A 2010 resource as at August 2010 was 1.23 million tonnes at 2.20 g/t Au at a 0.75 g/t Au cutoff, and the Tabai resource was 267,000 tonnes at 4.65 g/t Au at a 2 g/t Au cutoff. The original Tabai resource (excluding the 2011 extension) was 1.6 million tonnes at 1.58 g/t Au, also at 0.75 g/t Au cutoff.

14.6.7. Modelling & Resource Estimation Parameters

The ore zone wireframes were generated in Gemcom/Datamine/CAE Mining by Besra/North Borneo Gold geological staff and imported into Datamine/CAE Mining and validated. These were then filled with block model cells orientated orthogonally. The block model parameters for Taiton A, Taiton B, Overhead Tunnel and Umbut are listed in *Table 14-104: Taiton A, Taiton B, Overhead Tunnel & Umbut: Block Model Parameters* below. Those for Tabai are listed in *Table 14-105: Tabai: Block Model Parameters* and are different due to the narrow vertical nature of this deposit.

Block Model Parameter	Block Model Value
Parent Block Cell Size	10m x 10m x 5m
Zone Code	Ore Zone=1
Sub-Cell Size	2.5m x 2.5m x 0.5m

Table 14-104: Taiton A, Taiton B, Overhead Tunnel & Umbut: Block Model Parameters

Block Model Parameter	Block Model Value
Parent Block Cell Size	5m x 5m x 5m
Zone Code	Ore Zone=1
Sub-Cell Size	0.5m x 0.5m x 0.5m

Table 14-105: Tabai: Block Model Parameters

For Taiton A, Taiton B, Tabai, Overhead Tunnel and Umbut all assays within the ore zone volume were used in the estimate (zonal estimation). A top-cut of 23.26 g/t Au was applied to all samples above this value for Taiton A. Similarly, for Taiton B a top-cut of 9.77 g/t Au was applied, 33 g/t Au for Taiton B Extension, 41.84 g/t Au for Tabai, 14.86 g/t Au for Overhead Tunnel and 20.72 g/t Au for Umbut.

Limited density values were found in the a few drillholes from the Taiton and Bekajang-Krian areas. For the 2010 resource definition the average density was determined from these density samples by formation and applied to the Taiton data. The average was 2.594 t/m^3 for Bau Limestone, 2.406 t/m^3 for Intrusive, 2.589 t/m^3 for Krian Sandstone, 2.365 t/m^3 for Pedawan Shale, 1.98 t/m^3 for Quaternary deposits and 2.751 t/m^3 for Serian Volcanics; with a default of 2.5 being applied as required.

Regular and systematic density sampling was conducted during the 2011 drilling, and these density values were interpolated into the block models using the inverse distance technique. This was applied to the Taiton A, Taiton B Extension and Tabai deposits that were updated during 2011.

Search ellipse and Ordinary Kriging parameters were derived from the variogram analysis and are summarised in *Table 14-106: Taiton A: Ordinary Kriging Estimation Parameters* to *Table 14-111: Umbut: Ordinary Kriging Estimation Parameters* below.

Estimation Parameter	Value
Search Orientation	90° dip at 90° azimuth
Nugget	0.24
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.36
Sill (Range 2)	0.40
Range 1	5m x 5m x 2m
Range 2	55m x 55m x 16m
Minimum Samples	2 (1)
Maximum Samples	32 (32)
Search Volumes/Factor	2/2x

Table 14-106: Taiton A: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	60° dip at 90° azimuth
Nugget	0.25

Estimation Parameter	Value
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.29
Sill (Range 2)	0.46
Range 1	10m x 13m x 13m
Range 2	40m x 35m x 35m
Minimum Samples	2
Maximum Samples	32

Table 14-107: Taiton B: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	140° dip at -80° azimuth
Nugget	0.27
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.41
Sill (Range 2)	0.33
Range 1	5m x 5m x 5m
Range 2	60m x 60m x 10m
Minimum Samples	2 (1)
Maximum Samples	32 (32)
Search Volumes/Factor	2/2x

Table 14-108: Taiton B Extension: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	135° dip at -75° azimuth
Nugget	0.27
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.24
Sill (Range 2)	0.49
Range 1	3m x 6m x 2m
Range 2	35m x 30m x 12m
Minimum Samples	2 (1)
Maximum Samples	32 (32)
Search Volumes/Factor	2/2x

Table 14-109: Tabai: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 150° azimuth
Nugget	0.53
Variogram Type	Spherical (2 range)

Estimation Parameter	Value
Sill (Range 1)	0.26
Sill (Range 2)	0.21
Range 1	10m x 10m x 10m
Range 2	32m x 32m x 28m
Minimum Samples	2
Maximum Samples	32

Table 14-110: Overhead Tunnel: Ordinary Kriging Estimation Parameters

Estimation Parameter	Value
Search Orientation	55° dip at 225° azimuth
Nugget	0.14
Variogram Type	Spherical (2 range)
Sill (Range 1)	0.44
Sill (Range 2)	0.39
Range 1	3m x 3m x 3m
Range 2	25m x 25m x 25m
Minimum Samples	2
Maximum Samples	32

Table 14-111: Umbut: Ordinary Kriging Estimation Parameters

14.6.8. Resource & Comparative Estimates

The resource for Taiton A was determined at a variety of lower cutoffs by resource category. *Table 14-112: Taiton A: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	1,263,000	2.44
	0.75	1,133,000	2.64
	1	1,001,000	2.87
	1.25	855,000	3.17
	1.5	702,000	3.56
	1.75	596,000	3.90
	2	516,000	4.22
Inferred	0.5	240,000	1.46
	0.75	217,000	1.61
	1	190,000	1.76
	1.25	142,000	2.18
	1.5	88,000	2.91
	1.75	60,000	3.03
	2	32,000	4.27

Table 14-112: Taiton A: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Taiton B was determined at a variety of lower cutoffs by resource category. *Table 14-113: Taiton B: Ordinary Kriging Resource at 0.1 g/t Increments* below displays the results at each 0.1 g/t Au cutoff grade increment.

CATEGORY	CUTOFF	TONNES	AU
Inferred	0.4	1,862,000	1.55
	0.5	1,848,000	1.56
	0.6	1,786,000	1.60
	0.7	1,700,000	1.64

Table 14-113: Taiton B: Ordinary Kriging Resource at 0.1 g/t Increments

The Tabai resource has been split into two areas – one area is potentially mineable by open pit methods and the other by underground methods. The resource for Tabai was determined at a variety of lower cutoffs for each method and resource category. *Table 14-114: Tabai (Open Pit): Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment (open pit area) and *Table 14-115: Tabai (Underground): Ordinary Kriging Resource at 0.25 g/t Increments* displays the results at each 0.25 g/t Au cutoff grade (underground portion).

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	119,000	2.87
	0.75	83,000	2.90
	1	64,000	3.40
	1.25	48,000	3.60
	1.5	43,000	3.64
	1.75	37,000	3.97
	2	32,000	4.16
Inferred	0.5	78,000	1.69
	0.75	76,000	1.70
	1	75,000	1.72
	1.25	37,000	1.97
	1.5	31,000	2.06
	1.75	17,000	2.29
	2	15,000	2.46

Table 14-114: Tabai (Open Pit): Ordinary Kriging Resource at 0.25 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	769,000	2.02
	0.75	479,000	2.49
	1	390,000	2.65
	1.25	302,000	2.99
	1.5	231,000	3.48
	1.75	178,000	3.94
	2	163,000	4.00
Inferred	0.5	318,000	1.35
	0.75	157,000	1.57
	1	141,000	1.51
	1.25	85,000	1.83
	1.5	57,000	2.21
	1.75	40,000	2.75
	2	40,000	2.75

Table 14-115: Tabai (Underground): Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Overhead Tunnel was determined at a variety of lower cutoffs (Inferred Category only). *Table 14-116: Overhead Tunnel: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	372,000	1.67
0.75	349,000	1.74
1	299,000	1.88
1.25	229,000	2.11
1.5	135,000	2.62
1.75	97,000	3.02
2	76,000	3.34

Table 14-116: Overhead Tunnel: Ordinary Kriging Resource at 0.25 g/t Increments

The resource for Umbut was determined at a variety of lower cutoffs (Inferred Category only)... *Table 14-117: Umbut: Ordinary Kriging Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	473,000	2.23
0.75	401,000	2.52
1	296,000	3.10
1.25	263,000	3.34
1.5	221,000	3.72
1.75	171,000	4.33
2	152,000	4.63

Table 14-117: Umbut: Ordinary Kriging Resource at 0.25 g/t Increments

The original cutoff grade of 0.75 g/t Au has been lowered to 0.5 g/t Au in line with potential reserve cutoffs being lower and a review of the statistics. Deposits with resource estimates from the 2010 work only changed due to the lower cutoff, 2011 resource updates were evaluated at 0.5 g/t Au cutoff in line with their modification and update work. The two likely underground deposits (Tabai and Overhead Tunnel) have the higher cutoff grade of 2 g/t Au.

Figure 14-103: Taiton A: NW-SE Section through Ordinary Kriging Resource Model below shows a slice through the Taiton A gold resource model with the drillholes. Additionally, the ore zone and pit excavation wireframe outlines are also shown.

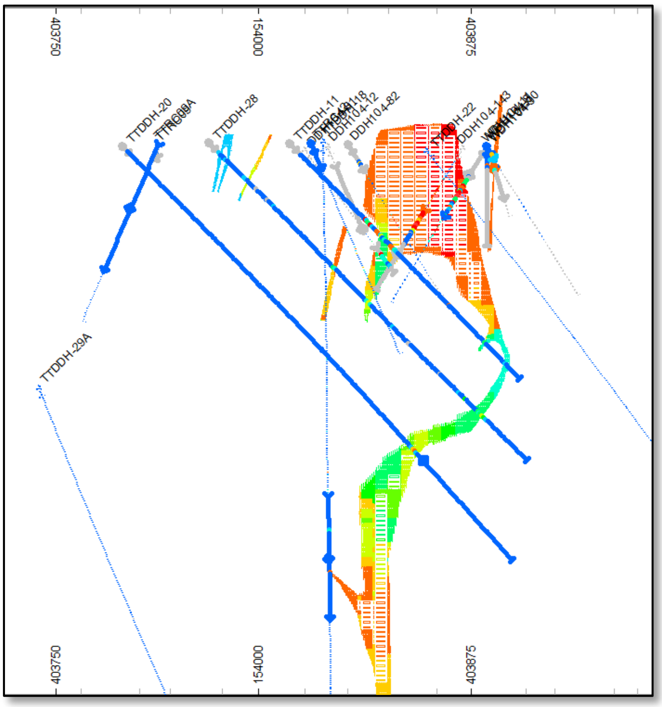


Figure 14-103: Taiton A: NW-SE Section through Ordinary Kriging Resource Model

Figure 14-104: Taiton B: NW-SE Section through Ordinary Kriging Resource Model below shows a slice through the Taiton B gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

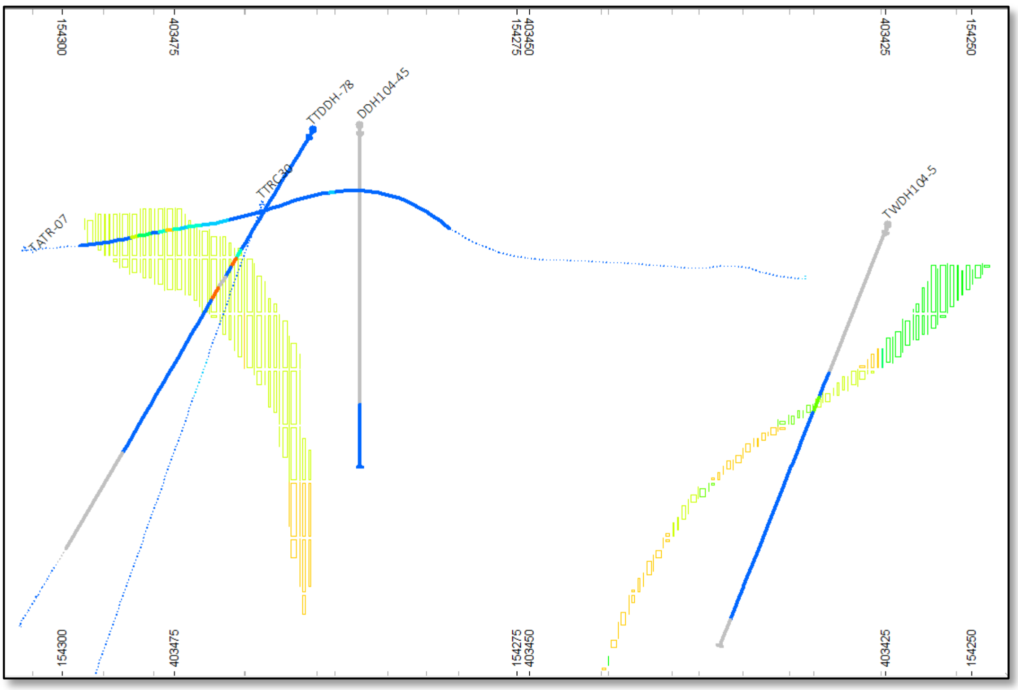


Figure 14-104: Taiton B: NW-SE Section through Ordinary Kriging Resource Model

Figure 14-105: Tabai: W-E Section through Ordinary Kriging Resource Model below shows a slice through the Tabai gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

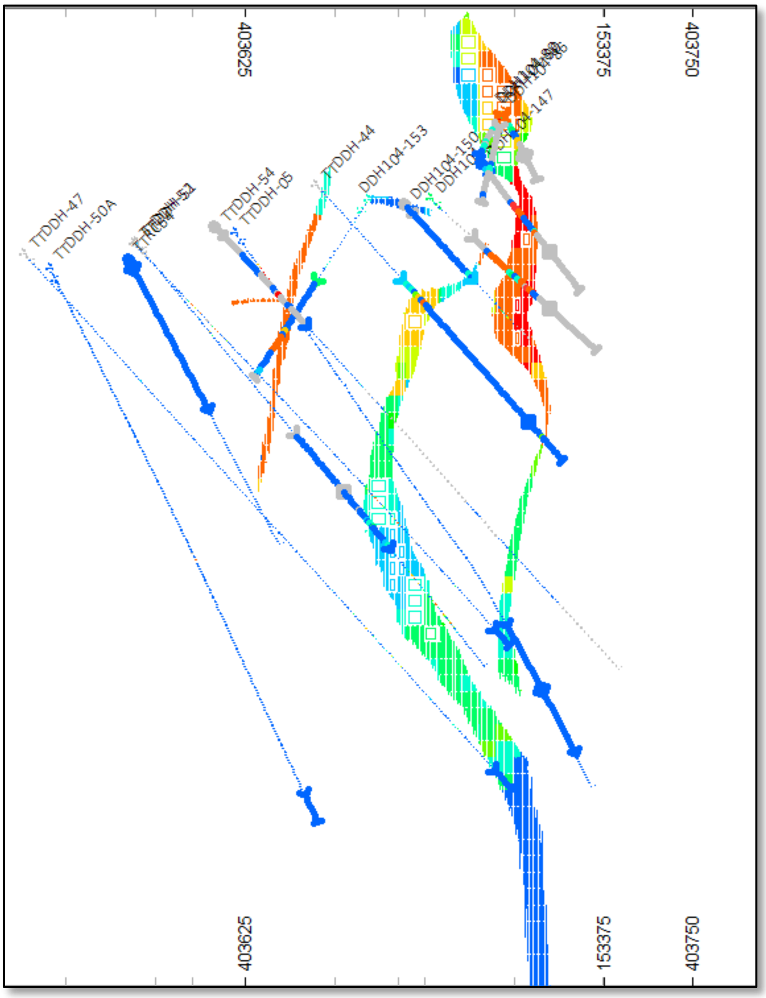


Figure 14-105: Tabai: W-E Section through Ordinary Kriging Resource Model

Figure 14-106: Overhead Tunnel: SW-NE Section through Ordinary Kriging Resource Model below shows a slice through the Overhead Tunnel gold resource model with the drillholes. Additionally, the ore zone and tunnel excavation wireframe outlines are also shown.

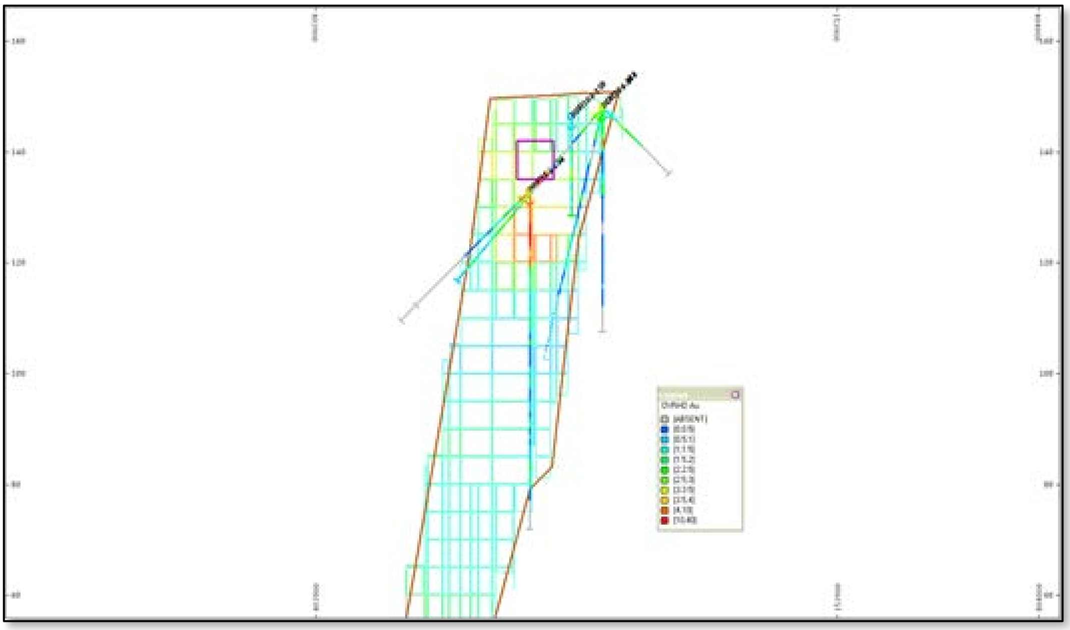


Figure 14-106: Overhead Tunnel: SW-NE Section through Ordinary Kriging Resource Model

Figure 14-107: Umbut: SW-NE Section through Ordinary Kriging Resource Model below shows a slice through the Umbut gold resource model with the drillholes. Additionally, the ore zone wireframe outlines are also shown.

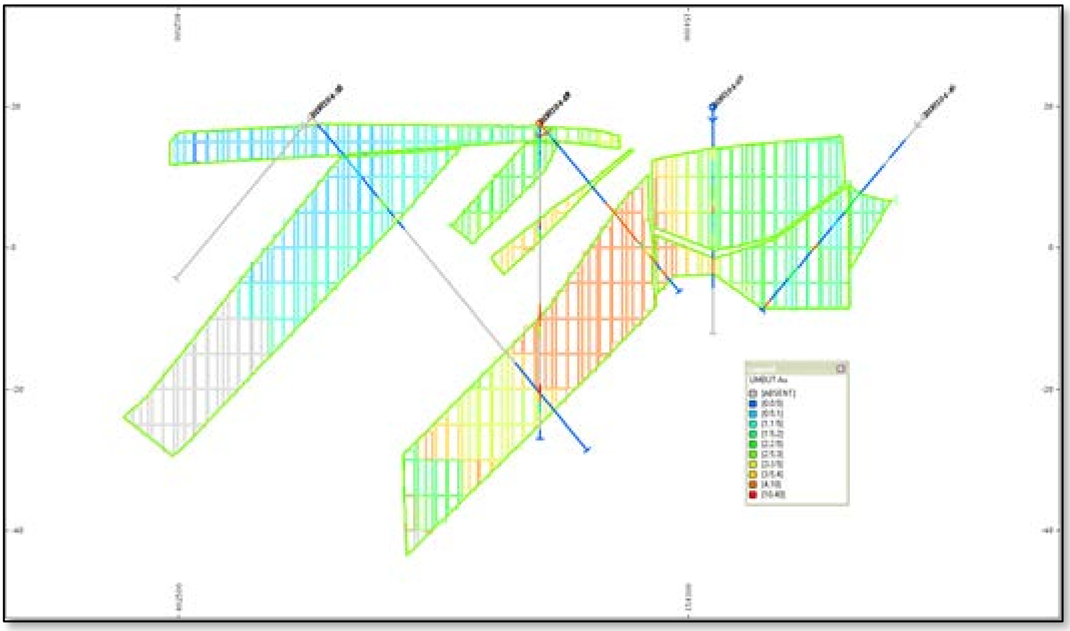


Figure 14-107: Umbut: SW-NE Section through Ordinary Kriging Resource Model

Resource model estimates are adjusted for topography or where excavations (underground and surface) exist. The resource model above topography or within known excavations is removed or subtracted from the final resource estimate.

Comparative estimations were conducted using Inverse Distance Squared and Nearest Neighbour (3D polygonal) methods. The estimation parameters used for these are listed below in *Table 14-118*: Taiton A: Comparative Estimation Method Parameters for Taiton A, *Table 14-119*: Taiton B: Comparative Estimation Method Parameters for Taiton B, *Table 14-120*: Taiton B Extension: Comparative Estimation Method Parameters for Taiton B Extension, *Table 14-121*: Tabai: Comparative Estimation Method Parameters for Tabai, *Table 14-122*: Overhead Tunnel: Comparative Estimation Method Parameters for Overhead Tunnel and *Table 14-123*: Umbut: Comparative Estimation Method Parameters for Umbut.

Estimation Parameter	Value
Search Orientation	90° dip at 90° azimuth
Search Ellipse Range	55m x 55m x 16m
Minimum Samples	2 (1)
Maximum Samples	32 (32)

Table 14-118: Taiton A: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	60° dip at 90° azimuth
Search Ellipse Range	40m x 35m x 35m
Minimum Samples	2
Maximum Samples	32

Table 14-119: Taiton B: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	140° dip at -80° azimuth
Search Ellipse Range	60m x 60m x 10m
Minimum Samples	2
Maximum Samples	32

Table 14-120: Taiton B Extension: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	135° dip at -75° azimuth
Search Ellipse Range	35m x 30m x 12m
Minimum Samples	2 (1)
Maximum Samples	32 (32)

Table 14-121: Tabai: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	0° dip at 150° azimuth

Estimation Parameter	Value
Search Ellipse Range	32m x 32m x 28m
Minimum Samples	2
Maximum Samples	32

Table 14-122: Overhead Tunnel: Comparative Estimation Method Parameters

Estimation Parameter	Value
Search Orientation	55° dip at 225° azimuth
Search Ellipse Range	25m x 25m x 25m
Minimum Samples	2
Maximum Samples	32

Table 14-123: Umbut: Comparative Estimation Method Parameters

Listed below, in *Table 14-124: Taiton A: Inverse Distance Squared Resource at 0.25 g/t Increments* and Table 14-125: Taiton A: Nearest Neighbour Resource at 0.25 g/t Increments, are the Inverse Distance and Nearest Neighbour comparative estimates for Taiton A.

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	1,263,000	2.44
	0.75	1,133,000	2.64
	1	1,001,000	2.88
	1.25	855,000	3.17
	1.5	702,000	3.54
	1.75	596,000	3.88
	2	516,000	4.18
Inferred	0.5	240,000	1.39
	0.75	217,000	1.47
	1	190,000	1.56
	1.25	142,000	1.68
	1.5	88,000	1.82
	1.75	60,000	1.95
	2	32,000	2.23

Table 14-124: Taiton A: Inverse Distance Squared Resource at 0.25 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	1,263,000	2.49
	0.75	1,133,000	2.71
	1	1,001,000	2.93
	1.25	855,000	3.25
	1.5	702,000	3.71
	1.75	596,000	4.06
	2	516,000	4.37
Inferred	0.5	240,000	1.58
	0.75	217,000	1.66
	1	190,000	1.76
	1.25	142,000	1.95
	1.5	88,000	2.60
	1.75	60,000	2.95
	2	32,000	2.45

Table 14-125: Taiton A: Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-126: Taiton B: Inverse Distance Squared Resource at 0.1 g/t Increments* and *Table 14-127: Taiton B: Nearest Neighbour Resource at 0.1 g/t Increments* , are the Inverse Distance and Nearest Neighbour comparative estimates for Taiton B.

CATEGORY	CUTOFF	TONNES	AU
Inferred	0.4	1,844,000	1.53
	0.5	1,829,000	1.54
	0.6	1,761,000	1.58
	0.7	1,678,000	1.63

Table 14-126: Taiton B: Inverse Distance Squared Resource at 0.1 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Inferred	0.4	1,749,000	1.73
	0.5	1,724,000	1.75
	0.6	1,514,000	1.91
	0.7	1,297,000	2.13

Table 14-127: Taiton B: Nearest Neighbour Resource at 0.1 g/t Increments

Listed below, in *Table 14-128: Tabai (Open Pit): Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-129: Tabai (Open Pit): Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Tabai (Open Pit). *Table 14-130: Tabai (Underground): Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-131: Tabai (Underground): Nearest Neighbour Resource at 0.25 g/t Increments* is the same but for Tabai (Underground) areas.

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	119,000	2.96
	0.75	83,000	3.07
	1	64,000	3.58
	1.25	48,000	3.79
	1.5	43,000	3.95
	1.75	37,000	4.33
	2	32,000	4.61
Inferred	0.5	78,000	1.84
	0.75	76,000	1.86
	1	75,000	1.87
	1.25	37,000	2.16
	1.5	31,000	2.31
	1.75	17,000	2.50
	2	15,000	2.69

Table 14-128: Tabai (Open Pit): Inverse Distance Squared Resource at 0.25 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	119,000	3.18
	0.75	83,000	4.28
	1	64,000	5.31
	1.25	48,000	6.71
	1.5	43,000	7.35
	1.75	37,000	8.25
	2	32,000	9.26
Inferred	0.5	78,000	1.52
	0.75	76,000	1.56
	1	75,000	1.57
	1.25	37,000	2.59
	1.5	31,000	2.92
	1.75	17,000	4.41
	2	15,000	4.86

Table 14-129: Tabai (Open Pit): Nearest Neighbour Resource at 0.25 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	769,000	2.23
	0.75	479,000	2.80
	1	390,000	2.95
	1.25	302,000	3.35
	1.5	231,000	3.94
	1.75	178,000	4.46
	2	163,000	4.53
Inferred	0.5	318,000	1.49
	0.75	157,000	1.75
	1	141,000	1.65
	1.25	85,000	2.05
	1.5	57,000	2.56
	1.75	40,000	3.24
	2	40,000	3.24

Table 14-130: Tabai (Underground): Inverse Distance Squared Resource at 0.25 g/t Increments

CATEGORY	CUTOFF	TONNES	AU
Indicated	0.5	769,000	2.21
	0.75	479,000	3.17
	1	390,000	3.70
	1.25	302,000	4.46
	1.5	231,000	5.41
	1.75	178,000	6.53
	2	163,000	6.96
Inferred	0.5	318,000	1.23
	0.75	157,000	1.89
	1	141,000	2.01
	1.25	85,000	2.64
	1.5	57,000	3.24
	1.75	40,000	3.91
	2	40,000	3.91

Table 14-131: Tabai (Underground): Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-132: Overhead Tunnel: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-133: Overhead Tunnel: Nearest Neighbour Resource at 0.25 g/t*

Increments, are the Inverse Distance and Nearest Neighbour comparative estimates for Overhead Tunnel.

CUTOFF	TONNES	AU
0.5	372,000	1.58
0.75	358,000	1.61
1	277,000	1.83
1.25	193,000	2.14
1.5	125,000	2.56
1.75	88,000	2.97
2	71,000	3.23

Table 14-132: Overhead Tunnel: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	346,000	2.01
0.75	252,000	2.54
1	235,000	2.65
1.25	164,000	3.33
1.5	135,000	3.74
1.75	127,000	3.86
2	119,000	4.01

Table 14-133: Overhead Tunnel: Nearest Neighbour Resource at 0.25 g/t Increments

Listed below, in *Table 14-134: Umbut: Inverse Distance Squared Resource at 0.25 g/t Increments* and *Table 14-135: Umbut: Nearest Neighbour Resource at 0.25 g/t Increments*, are the Inverse Distance and Nearest Neighbour comparative estimates for Umbut.

CUTOFF	TONNES	AU
0.5	447,000	2.47
0.75	393,000	2.73
1	290,000	3.38
1.25	257,000	3.67
1.5	220,000	4.05
1.75	183,000	4.55
2	148,000	5.18

Table 14-134: Umbut: Inverse Distance Squared Resource at 0.25 g/t Increments

CUTOFF	TONNES	AU
0.5	415,000	2.42
0.75	347,000	2.76
1	235,000	3.66
1.25	200,000	4.11
1.5	173,000	4.54
1.75	163,000	4.71
2	132,000	5.37

Table 14-135: Umbut: Nearest Neighbour Resource at 0.25 g/t Increments

The comparative resource estimates for Taiton A, Taiton B, Tabai, Overhead Tunnel and Umbut compare well with the Ordinary Kriging resource estimates and the minor differences probably reflect the interpolation techniques/application.

Other techniques tend to overestimate grades in comparison to the Ordinary Kriging, except for low grade Nearest Neighbour which tends to underestimate in some cases. Some instances of Nearest Neighbour have quite high grade values for higher cutoffs and this is likely due to very high samples heavily weighting the higher grade ranges or cutoffs. This is as expected from these techniques and hence why Ordinary Kriging is used in this instance. These techniques are just used as a comparative estimate to check for any issues or errors with the Ordinary Kriging method and parameters.

The deposits from the 2010 resource definition work (Umbut & Overhead Tunnel) have been classified as Inferred. Some areas of the deposit(s) could potentially have been classified as Indicated based purely on the drilling density. However, one or more of the following issues gave rise to an Inferred classification:

- Large number of RC drillholes with few diamond core holes;
- Smaller drillhole sizes in some instances (e.g. BQ);
- Lack of extensive and systematic density determinations throughout the deposit;
- Gaps in the drillhole spacing or coverage and/or larger distances between drillholes;
- Difficulty in domaining of the data to remove possible mixed populations in some instances.

Subsequent to the 2010-11 drilling and resource update a sufficient amount of drilling and associated QA/QC was conducted, along with sufficient additional information (e.g. density), to classify parts of the sector as Indicated. The criteria for classification to Indicated was where sufficient recent drilling was done to confirm the historic drilling, where the drilling density was less than or equal to 25 metres spacing and there were sufficient samples (>20) used in the estimation, and the blocks were within the first search radius in the estimation. Using this criteria part of the recently drilled Taiton A block and Tabai deposits warranted an upgrade in the resource classification to Indicated.

14.7. Say Seng Sector

The Say Seng sector deposits were added to the resource base after investigations and analysis of historic information during 2011. It was determined that there was sufficient information in order to define small resources using the historic drilling and data.

14.7.1. Introduction & General

The Say Seng sector is situated approximately 3-5 kilometres north-east of the town of Bau and is a set of two deposits based on discrete geographical areas as defined by the drilling to date. These deposits have been modelled separately and are Bukit Sarin and Say Seng.

The resource assessment conducted, included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;

- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;
- Combining the above data into a clean and validated resource database with associated data being verified;
- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;
- Resource estimation work to determine the mineral resource using 3 different estimation techniques.

All data used for this resource update was supplied or sourced by Besra (previously Olympus)/North Borneo Gold or determined from available information, both digital and hardcopy. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

14.7.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of these are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;
- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

14.7.3. Ore Zone Definition

The ore zones at Bukit Sarin and Say Seng were defined in the following manner:

- Drillhole sections were created and interpreted faults, geological and mineralized zone grade boundaries (≥0.5 g/t Au lower cut-off) were drawn;
- The grade boundaries were correlated from section to section and cross-checked in plan;
- In the absence of zone continuity, extrapolations were made in between the two drill sections, and up/down dip, using standard methodologies;
- The definition of the mineralized zones and the methodology used was validated visually on each section, and in 3D, and samples within the zone wireframe were analysed;

- The ore zone was terminated using the surveyed topography.

In the ore zone definition there are isolated cases of assay values below the lower cut-off value. These have only been included where they fall within samples above the cut-off, are of minor effect and cannot be excluded due to their isolated nature. A typical section/plan through Bukit Sarin Deposit is shown below in *Figure 14-108: Bukit Sarin - Drillhole Section and Associated Plan Used for Geological/Ore Zone Definition* with the drillholes coloured by lithology and the Au assays as histograms on the side of the drillhole trace.

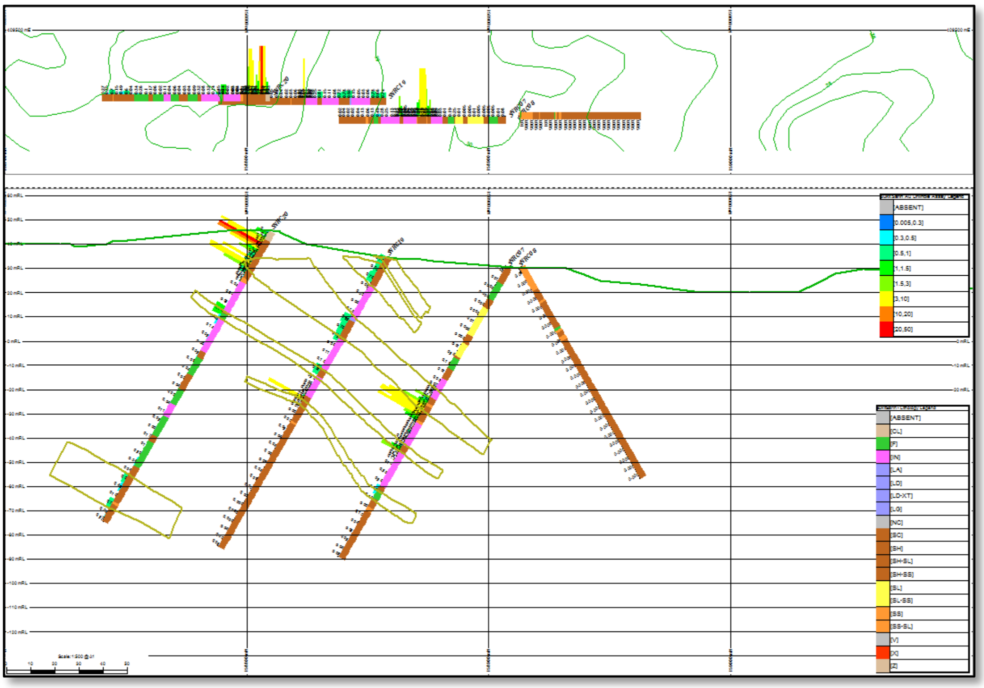


Figure 14-108: Bukit Sarin - Drillhole Section and Associated Plan Used for Geological/Ore Zone Definition

14.7.4. Statistical Analysis of Data

The Bukit Sarin database consisted of 26 drillhole collar entries, 26 collar survey entries, 1,018 assay records and 3,280 lithology records.

A total of 3,281 metres of drilling was drilled at Bukit Sarin. The drillhole depths varied from 12 metres to 150 metres with an average depth of approximately 126.2 metres. The drillholes consisted of 26 RC holes (including 1 re-drill).

The Say Seng database consisted of 13 drillhole collar entries, 13 collar survey entries, 772 assay records, and 363 lithology records.

A total of 1,989.2 metres of drilling was drilled at Say Seng. The drillhole depths varied from 87.6 metres to 205.65 metres with an average depth of approximately 153.02 metres. The drillholes consisted of 13 diamond cored holes in HQ & NQ sizes.

Statistics were calculated for gold and sample length fields in the drillhole database within the defined mineralized zones at both deposits. *Table 14-136: Bukit Sarin: Ore Zone Drillhole Sample Statistics* lists the statistics for the drillhole samples within the mineralised envelope for Bukit Sarin, and *Table 14-137: Say Seng: Ore Zone Drillhole Sample Statistics* lists the same information for Say Seng.

Drillhole Field	Length	Au
Number of Records	314	314
Number of Samples	314	314
Missing Values	-	-
Minimum Value	1.00	0.02
Maximum Value	1.00	8.87
Range	-	8.85
Mean	1.00	1.18
Variance	-	1.72
Standard Deviation	-	1.31
Standard Error	-	0.07
Skewness	-	2.97
Kurtosis	-	10.67
Geometric Mean	-	0.77
Sum of Logs	-	- 83.78
Mean of Logs	-	- 0.27
Log Variance	-	0.94
Log Estimate of Mean	-	1.22

Table 14-136: Bukit Sarin: Ore Zone Drillhole Sample Statistics

Drillhole Field	Length	Au
Number of Records	117	117
Number of Samples	117	107
Missing Values	-	10
Minimum Value	0.10	0.01
Maximum Value	8.55	59.30
Range	8.45	59.30
Mean	1.03	3.98
Variance	1.08	75.36
Standard Deviation	1.04	8.68
Standard Error	0.10	0.84
Skewness	4.74	4.05
Kurtosis	26.88	19.66
Geometric Mean	0.80	0.34
Sum of Logs	- 25.51	- 116.24
Mean of Logs	- 0.22	- 1.09
Log Variance	0.47	7.70
Log Estimate of Mean	1.02	15.84

Table 14-137: Say Seng: Ore Zone Drillhole Sample Statistics

All sample lengths at Bukit Sarin were 1 metre so no compositing was required.

Samples within the Say Seng ore zone were composited to 1 metre lengths, resulting in 148 composites. Composites were set at 1 metre as this was the predominant sample length and close to the average sample length. *Table 14-138: Say Seng: Ore Zone Composited Drillhole Sample Statistics* lists the statistics for the composited drillholes for Say Seng.

Drillhole Field	Length	Au
Number of Records	148	148
Number of Samples	148	119
Missing Values	-	29
Minimum Value	0.50	0.01
Maximum Value	1.00	59.30
Range	0.50	59.30
Mean	0.96	3.39
Variance	0.01	66.43
Standard Deviation	0.11	8.15
Standard Error	0.01	0.75
Skewness	- 3.10	4.93
Kurtosis	8.46	28.10
Geometric Mean	0.96	0.44
Sum of Logs	- 6.75	- 98.72
Mean of Logs	- 0.05	- 0.83
Log Variance	0.02	5.96
Log Estimate of Mean	0.97	8.59

Table 14-138: Say Seng: Ore Zone Composited Drillhole Sample Statistics

The Bukit Sarin Au data shown statistically above is also shown in graphical form. *Figure 14-109: Bukit Sarin: Log Histogram of Au Ore Zone Samples* and *Figure 14-110: Bukit Sarin: Cumulative Log Histogram of Au Ore Zone Samples* below display the log histogram and cumulative log probability plots, for Au ore samples, which were plotted in Datamine/CAE Mining.

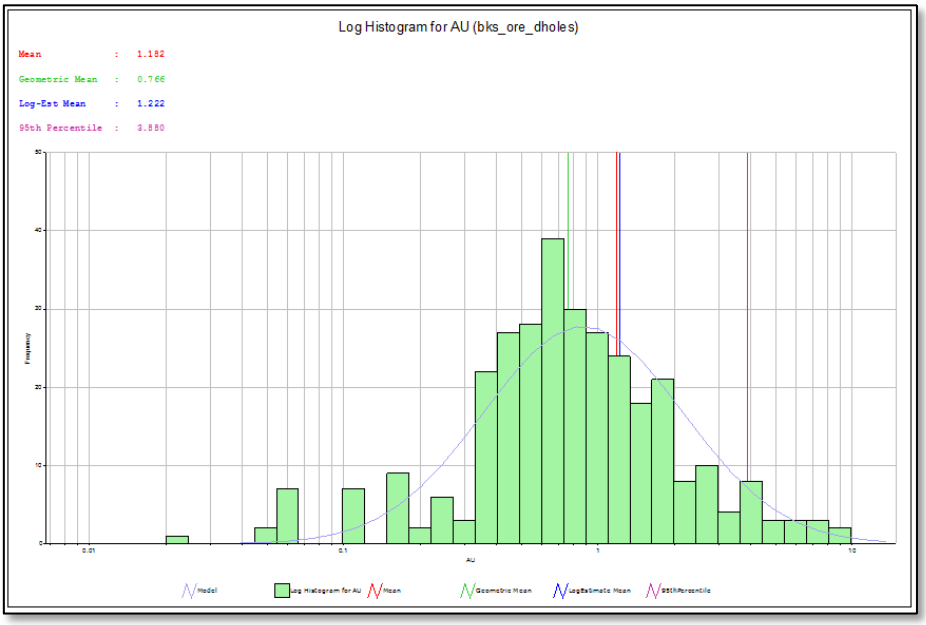


Figure 14-109: Bukit Sarin: Log Histogram of Au Ore Zone Samples

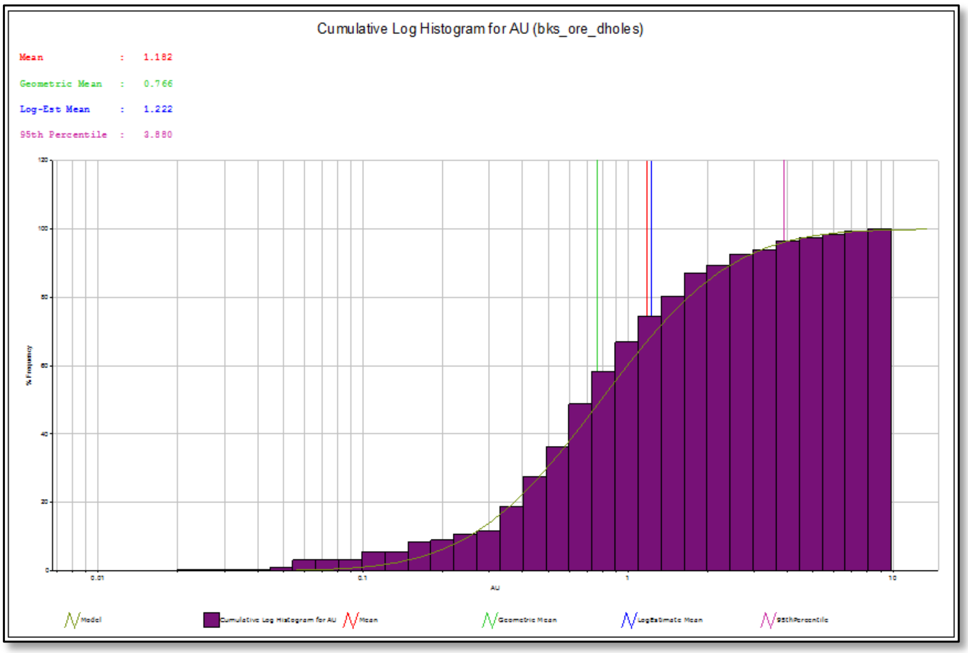


Figure 14-110: Bukit Sarin: Cumulative Log Histogram of Au Ore Zone Samples

The Say Seng Au data shown statistically above is also shown in graphical form below. *Figure 14-111: Say Seng: Log Histogram of Au Ore Zone Samples* and *Figure 14-112: Say Seng: Cumulative Log Histogram of Au Ore Zone Samples* below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine/CAE Mining.

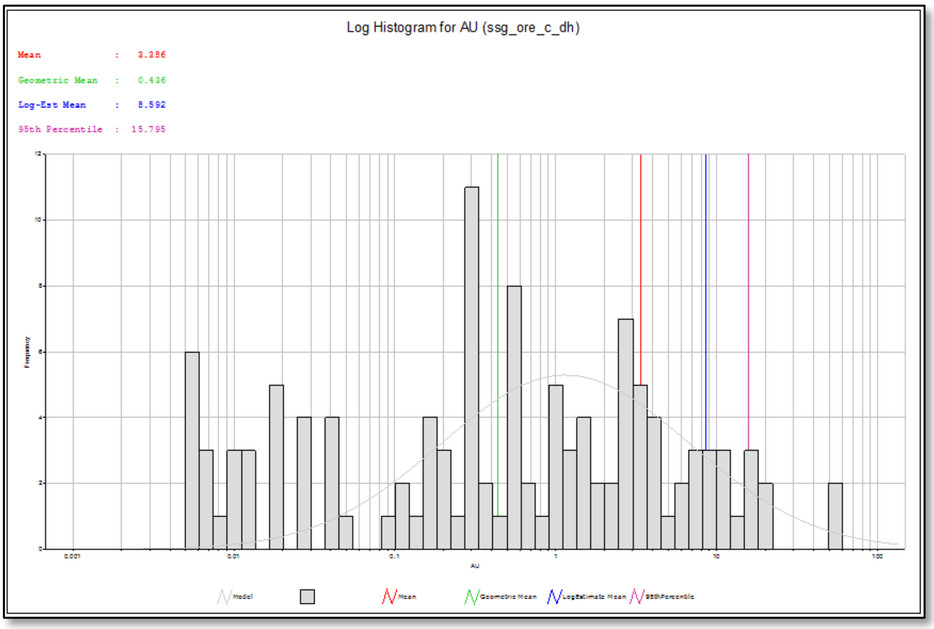


Figure 14-111: Say Seng: Log Histogram of Au Ore Zone Samples

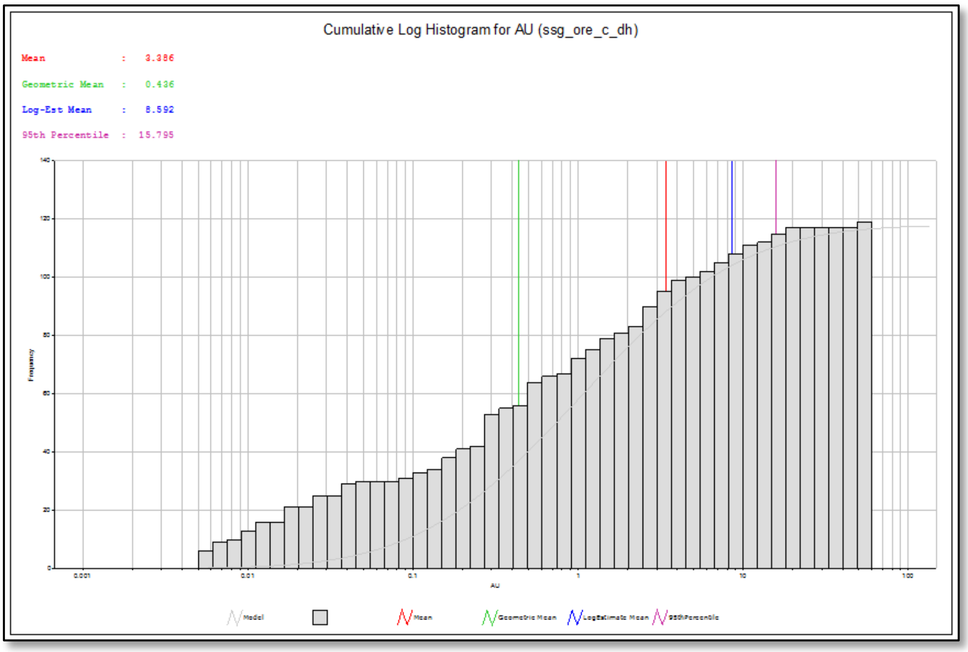


Figure 14-112: Say Seng: Cumulative Log Histogram of Au Ore Zone Samples

The Say Seng data shows mixed populations and a large inclusion of low grade material within the ore envelope. The deposit requires more work to clearly identify the discrete ore zones within the current ore envelope, and allow better definition or domaining. This would require additional drilling. The resource is small and limited in extent and the data is sufficient to model an Inferred resource.

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-139: Bukit Sarin: Quantile Analysis of Au Drillhole Composites* and *Table 14-140: Say Seng: Quantile Analysis of Au Drillhole*

Composites below displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Bukit Sarin and Say Seng Ore Zones.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	31	0.12	0.02	0.25	3.83	1.03
10	20	31	0.36	0.26	0.41	11.13	3.00
20	30	32	0.48	0.41	0.53	15.51	4.18
30	40	31	0.59	0.54	0.65	18.30	4.93
40	50	32	0.69	0.66	0.74	22.08	5.95
50	60	31	0.83	0.78	0.92	25.69	6.92
60	70	31	1.03	0.93	1.17	32.06	8.64
70	80	32	1.35	1.17	1.58	43.22	11.64
80	90	31	1.91	1.59	2.44	59.19	15.94
90	100	32	4.38	2.50	8.87	140.26	37.78
90	92.5	8	2.67	2.50	2.82	21.36	5.75
92.5	95	8	3.43	2.83	3.88	27.43	7.39
95	97.5	8	4.43	4.05	5.18	35.41	9.54
97.5	100	8	7.01	5.79	8.87	56.06	15.10
0	100	314	1.18	0.02	8.87	371.27	100.00

Table 14-139: Bukit Sarin: Quantile Analysis of Au Drillhole Composites

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	11	0.01	0.01	0.01	0.07	0.02
10	20	12	0.02	0.01	0.03	0.20	0.05
20	30	12	0.07	0.03	0.16	0.89	0.22
30	40	12	0.24	0.16	0.31	2.91	0.72
40	50	12	0.39	0.31	0.53	4.71	1.17
50	60	12	0.74	0.54	1.09	8.85	2.20
60	70	12	1.59	1.09	2.29	19.09	4.74
70	80	12	2.96	2.62	3.36	35.51	8.81
80	90	12	5.98	3.74	8.60	71.72	17.80
90	100	12	21.58	8.91	59.30	258.95	64.27
90	92.5	3	10.29	8.91	11.16	30.87	7.66
92.5	95	3	13.40	11.36	15.80	40.20	9.98
95	97.5	3	18.05	17.74	18.21	54.14	13.44
97.5	100	3	44.58	21.02	59.30	133.74	33.20
0	100	119	3.39	0.01	59.30	402.90	100.00

Table 14-140: Say Seng: Quantile Analysis of Au Drillhole Composites

For Bukit Sarin, looking at the primary percentiles, it can be seen that approximately 38 % of the metal percentage can be found in the top 10 % range, and that there is a significant jump in the mean grade and metal content from the previous range. For Say Seng this is approximately 64 %.

Closer inspection of the secondary percentiles indicates that the Au metal content changes abruptly at the 97.5 percentile, and contains nearly 15 % of the Au metal content for Bukit Sarin and 33 % for Say Seng.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top-cut of 7 g/t Au (mean of the 97.5 percentile) should be applied to the Bukit Sarin samples above this value in order to remove any effect of the high grade samples in the estimation process. Similarly, a top-cut of 44.6 g/t Au for Say Seng should be applied.

14.7.5. Semi-Variogram Analysis

Semi-variogram analyses were undertaken to determine the semi-variogram parameters for use in the Ordinary Kriging. Downhole, horizontal and vertical increment semi-variograms were generated with the best semi-variograms selected that defines the strike, dip and dip direction.

Resulting from these analyses it was determined that Ordinary Kriging could not be applied. This was due to not being able to define a semi-variogram for the Say Seng deposit, and inconclusive semi-variograms from the Bukit Sarin deposit. Therefore, both deposits were determined by the Inverse Distance method with the Nearest Neighbour (3D polygonal) method as a check.

14.7.6. Previous Resource Estimates

No previous resource estimate has been conducted on these deposits.

14.7.7. Modelling & Resource Estimation Parameters

The ore zone wireframes were generated in Datamine/CAE Mining by Besra/North Borneo Gold geological staff and validated. These were then filled with block model cells orientated orthogonally. The block model parameters for Bukit Sarin and Say Seng are listed in *Table 14-141: Bukit Sarin: Block Model Parameters* and *Table 14-142: Say Seng: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	5m x 5m x 2.5m
Zone Code	Ore Zone=1
Sub-Cell Size	0.625m x 0.625m x 0.25m

Table 14-141: Bukit Sarin: Block Model Parameters

Block Model Parameter	Block Model Value
Parent Block Cell Size	4m x 4m x 2m
Zone Code	Ore Zone=1
Sub-Cell Size	0.5m x 0.5m x 0.25m

Table 14-142: Say Seng: Block Model Parameters

For Bukit Sarin and Say Seng all assays within the ore zone volume were used in the estimate (zonal estimation). A top-cut of 7 g/t Au was applied to all samples above this value for Bukit Sarin. Similarly, for Say Seng a top-cut of 44.6 g/t Au was applied.

Limited or no density values were found in the a few drillholes from the Bukit Sarin and Say Seng. For the Bukit Sarin resource definition the average density was determined from the average value of the shales/mudstones/sandstones from the nearby Jugan deposit and is 2.625 t/m^3. For Say Seng the average density for limestone (host rock) was used, namely 2.6 t/m^3.

Search ellipse and Inverse Distance estimation parameters for Bukit Sarin and Say Seng are summarised in *Table 14-143: Bukit Sarin: Inverse Distance Estimation Parameters* and *Table 14-144: Say Seng: Inverse Distance Estimation Parameters* below.

Estimation Parameter	Value
Search Orientation	-45° dip at 810° azimuth
Range	40m x 40m x 10m
Search Volume	Range
Minimum Samples	2
Maximum Samples	32
Inverse Power	2
Search Volumes/Factor	1/1x

Table 14-143: Bukit Sarin: Inverse Distance Estimation Parameters

Estimation Parameter	Value
Search Orientation	60° dip at 60° azimuth
Range	40m x 40m x 10m
Search Volume	Range
Minimum Samples	2
Maximum Samples	32
Inverse Power	2
Search Volumes/Factor	1/1x

Table 14-144: Say Seng: Inverse Distance Estimation Parameters

14.7.8. Resource & Comparative Estimates

The resource for Bukit Sarin was determined at a variety of lower cutoffs by resource category. *Table 14-145: Bukit Sarin: Inverse Distance Resource at 0.1 g/t Increments* below displays the results at each 0.1 g/t Au cutoff grade increment.

Cutoff	Tonnage	Grade
	(t)	(g/t)
0.4	1,238,000	1.19
0.5	1,110,000	1.28
0.6	1,009,000	1.35
0.7	932,000	1.41
0.8	854,000	1.46
0.9	762,000	1.54
1	692,000	1.60

Table 14-145: Bukit Sarin: Inverse Distance Resource at 0.1 g/t Increments

The resource for Say Seng was determined at a variety of lower cutoffs by resource category. *Table 14-146: Say Seng: Inverse Distance Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

Cutoff	Tonnage	Grade
	(t)	(g/t)
0.5	244,000	3.24
0.75	239,000	3.29
1	235,000	3.33
1.25	230,000	3.38
1.5	213,000	3.54
1.75	196,000	3.71
2	159,000	4.14

Table 14-146: Say Seng: Inverse Distance Resource at 0.25 g/t Increments

The final cutoff for the resource definition was 0.5 g/t Au cutoff in line with other deposits.

Figure 14-113: Bukit Sarin: SW-NE Section through Inverse Distance Resource Model below shows a slice through the Bukit Sarin gold resource model with the drillholes.

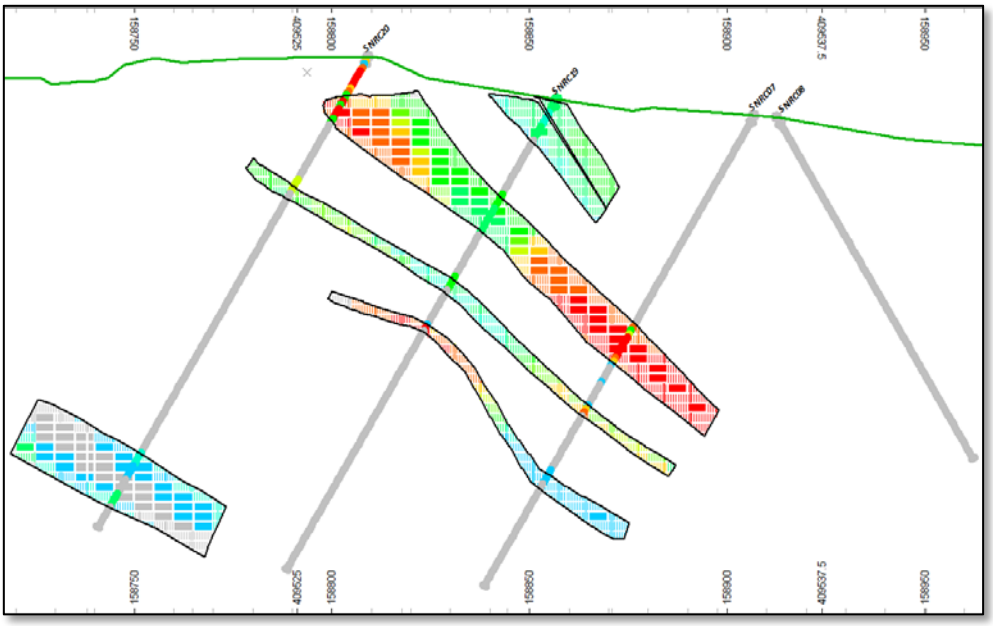


Figure 14-113: Bukit Sarin: SW-NE Section through Inverse Distance Resource Model

Figure 14-114: Say Seng: W-E Section through Inverse Distance Resource Model below shows a slice through the Say Seng gold resource model with the drillholes.

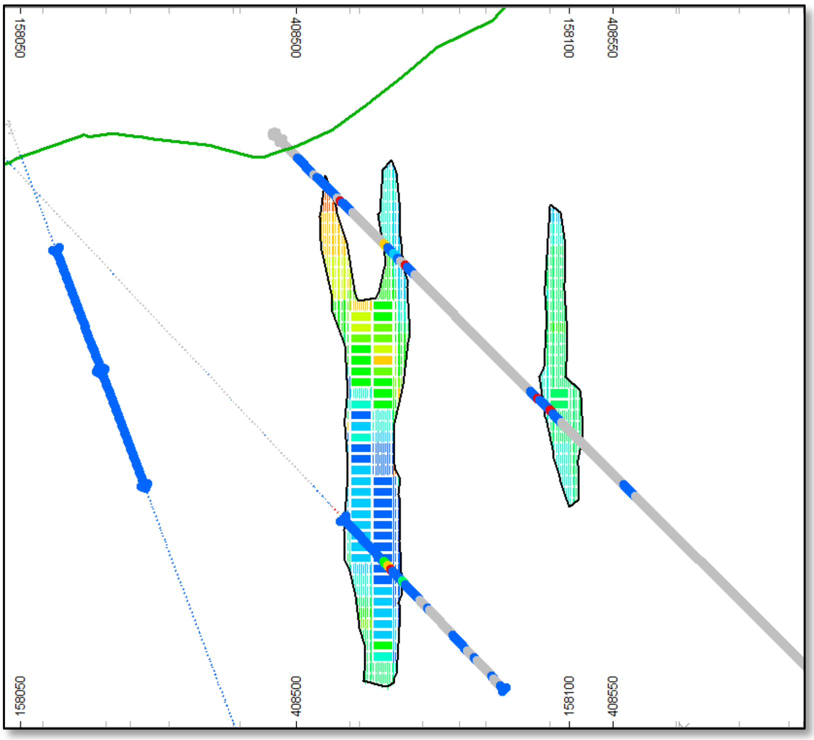


Figure 14-114: Say Seng: W-E Section through Inverse Distance Resource Model

Resource model estimates are adjusted for topography or where excavations (underground and surface) exist. The resource model above topography or within known excavations is removed or subtracted from the final resource estimate.

Comparative estimations were conducted using Nearest Neighbour (3D polygonal) methods. The estimation parameters used for these are as listed in the Inverse Distance tables above.

Listed below, in *Table 14-147: Bukit Sarin: Nearest Neighbour Resource at 0.1 g/t Increments*, is the Nearest Neighbour comparative estimate for Bukit Sarin.

Cutoff	Tonnage (t)	Grade (g/t)
0.4	1,238,000	1.17
0.5	1,110,000	1.26
0.6	1,009,000	1.32
0.7	932,000	1.38
0.8	854,000	1.43
0.9	762,000	1.51
1	692,000	1.58

Table 14-147: Bukit Sarin: Nearest Neighbour Resource at 0.1 g/t Increments

Listed below, in *Table 14-148: Say Seng: Nearest Neighbour Resource at 0.25 g/t Increments*, is the Nearest Neighbour comparative estimate for Say Seng.

Cutoff	Tonnage (t)	Grade (g/t)
0.5	244,000	4.20
0.75	239,000	4.27
1	235,000	4.32
1.25	230,000	4.41
1.5	213,000	4.71
1.75	196,000	5.05
2	159,000	5.95

Table 14-148: Say Seng: Nearest Neighbour Resource at 0.25 g/t Increments

The comparative resource estimates for Bukit Sarin and Say Seng compare well with the Inverse Distance resource estimates and the minor differences probably reflect the interpolation techniques/application.

The deposits have been classified as Inferred. Some areas of the deposit(s) could potentially have been classified as Indicated based purely on the drilling density. However, one or more of the following issues gave rise to an Inferred classification:

- Large number of RC drillholes with few diamond core holes;
- Smaller drillhole sizes in some instances (e.g. BQ);
- Lack of extensive and systematic density determinations throughout the deposit;
- Gaps in the drillhole spacing or coverage and/or larger distances between drillholes;
- Difficulty in domaining of the data to remove possible mixed populations in some instances.

14.8. Tailings

This resource section was completed in 2010 and is detailed in report *"Technical Report on Bau Project in Bau, Sarawak, East Malaysia"*. It has been included here for the sake of completeness.

14.8.1. Introduction & General

The historic tailings dam resource is situated in the Bekajang area between the Bekajang North and South deposits and is approximately 1 kilometre from the town of Bau. This resource assessment is of the residual processed tails from the Bukit-Young Gold Mines operations and plant during the 1980-90's.

No changes were made to this resource from the previous 2010 resource estimate and it is included for completeness.

The resource assessment conducted by Terra Mining Consultants/Stevens & Associates in 2010 included:

- Review of previous resource estimate work and geological interpretations;
- Review and validation of the current resource database and associated data;
- Review, capture and validation of information and data not captured in the above database (hardcopy format) including other digital data;
- Combining the above data into a clean and validated resource database with associated data being verified;
- Analysis and assessment of the resource data;
- Geological modelling and interpretation of the resource;
- Resource estimation work to determine the mineral resource using 3 different estimation techniques.

All data used for this resource update was supplied or sourced by Olympus/North Borneo Gold or determined by Terra Mining Consultants/Stevens & Associates from available information. An extensive data validation, cross checking and rectification process was undertaken prior to all resource modelling to verify all data and sources as best as possible, particularly with respect to the historic data.

Historical documents and internal reports were reviewed as part of the resource update. Additionally, numerous notes, plans, sections, memoranda and other documents, both in digital and hardcopy format found in the office library and storage, were reviewed.

14.8.2. Data Review & Validation

All data in digital format or captured from hardcopy format has gone through an extensive set of data validation steps and processes. Where any errors existed these have been checked and rectified where applicable, with those that could not be verified being removed from the database. Some of these are listed below:

- Cross-checking data against original forms, documents, logs or field notes;
- Check surveying of drillhole and topographic data in the field and comparing with the database value;
- Systematic checking of all assay, geology, density, survey and collar information;
- Use of the mining software validation tools to detect errors, e.g. sample from/to overlaps;
- Visual verification where applicable;
- Statistical and other checks.

14.8.3. Ore Zone Definition

The tailings impoundment was defined in the following manner:

- Digitise the hydrographic survey of the original Bekajang Lake and incorporate into the 1978 topography as determined from the aerial photogrammetry work;
- Capture the final tailings topography surface and limits, projecting these boundaries down at the angle of the bund construction;

- This process defines the tailings impoundment volume which was used to define the tailings “resource” volume.

14.8.4. Statistical Analysis of Data

The full database consisted of 237 auger drillhole collar entries and 937 assay records. All augers were assumed to be vertical.

A total of 916.8 metres of auger drilling was drilled in the accessible part of the tailings impoundment. The auger drill depths varied from 0.3 metres to 7.4 metres with an average depth of approximately 3.87 metres.

All auger drillholes fell within the tailings impoundment zone. Statistics were calculated in Datamine for gold, density and sample length fields in the drillhole database within the defined mineralized zones. *Table 14-149: Tailings: Impoundment Drillhole Sample Statistics* lists the statistics for the drillhole samples within the tailings impoundment envelope.

Drillhole Field	Length	Au
Number of Records	937	937
Number of Missing Samples	937	937
Missing Values	-	-
Minimum Value	0.10	0.55
Maximum Value	1.00	8.25
Range	0.90	7.70
Mean	0.98	1.39
Variance	0.01	0.24
Standard Deviation	0.10	0.49
Standard Error	0.00	0.02
Skewness	-5.96	4.60
Kurtosis	35.94	47.78
Geometric Mean	0.97	1.33
Sum of Logs	-27.75	268.70
Mean of Logs	-0.03	0.29
Log Variance	0.03	0.08
Log Estimate of Mean	0.99	1.39

Table 14-149: Tailings: Impoundment Drillhole Sample Statistics

Samples within the tailings impoundment were not composited as the sample intervals were 1 metre and any sub-metre intervals were at the end of the holes which would have not changed in the composite process.

The Au data shown statistically above is also shown in graphical form below. *Figure 14-115: Tailings: Log Histogram of Au Impoundment Samples* and *Figure 14-116: Tailings: Cumulative Log*

Histogram of Au Impoundment Samples below display the log histogram and cumulative log probability plots, for composited Au samples, which were plotted in Datamine.

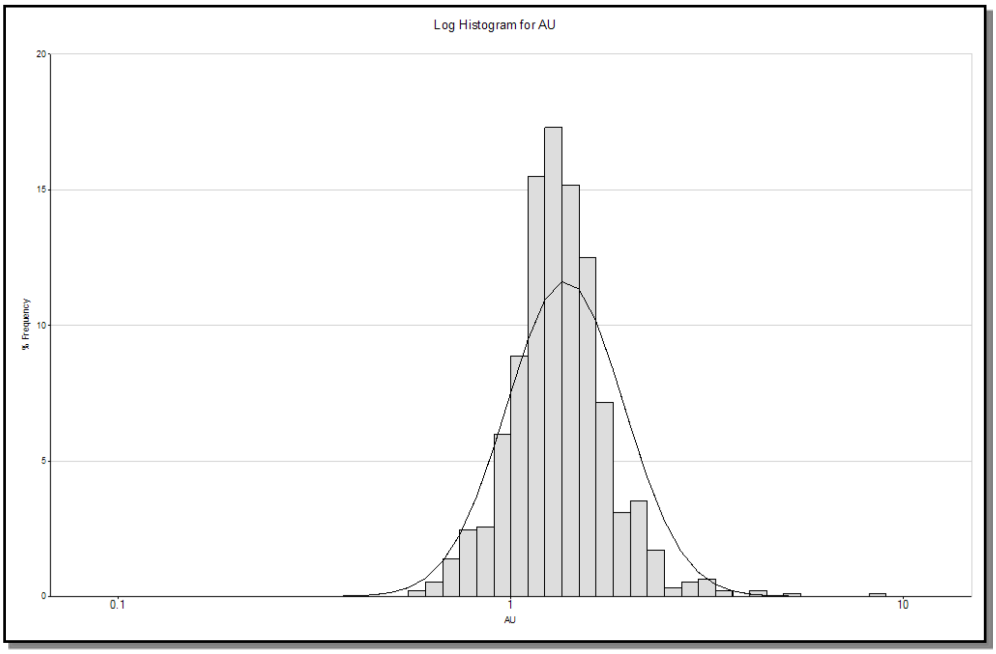


Figure 14-115: Tailings: Log Histogram of Au Impoundment Samples

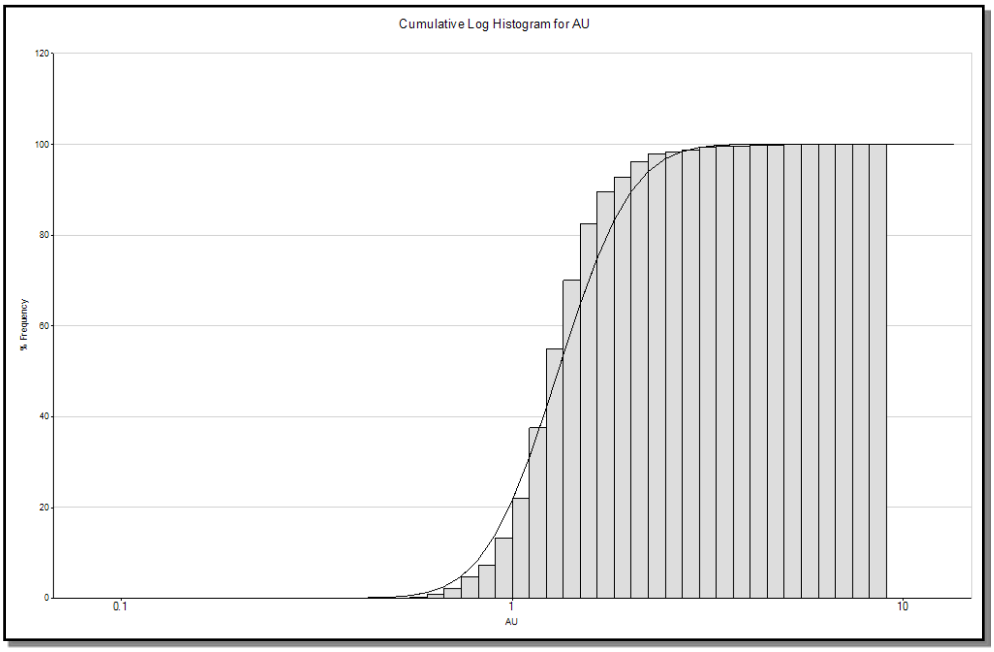


Figure 14-116: Tailings: Cumulative Log Histogram of Au Impoundment Samples

A quantile analysis was run for Au at ten primary percentiles (10 % ranges) with four secondary percentiles (2.5 % ranges) for the last primary percentile. *Table 14-150: Tailings: Quantile Analysis of Au Auger Samples* displays the primary and secondary percentiles; the mean, minimum and maximum grades; and the metal content and percentage per range for the Tailings Impoundment.

Percent From	Percent To	Number Samples	Mean	Minimum	Maximum	Metal Content	Metal Percent
0	10	93	0.82	0.55	0.95	76.60	5.87
10	20	94	1.03	0.95	1.09	96.74	7.42
20	30	94	1.13	1.09	1.17	106.35	8.15
30	40	93	1.20	1.17	1.24	112.03	8.59
40	50	94	1.28	1.24	1.32	120.33	9.22
50	60	94	1.36	1.32	1.40	127.81	9.80
60	70	93	1.44	1.40	1.48	133.95	10.27
70	80	94	1.55	1.50	1.62	145.49	11.15
80	90	94	1.72	1.62	1.85	161.46	12.38
90	100	94	2.38	1.85	8.25	223.73	17.15
90	92.5	23	1.90	1.85	2.00	43.76	3.35
92.5	95	24	2.06	2.00	2.10	49.34	3.78
95	97.5	23	2.23	2.10	2.35	51.37	3.94
97.5	100	24	3.30	2.37	8.25	79.26	6.08
0	100	937	1.39	0.55	8.25	1,304.49	100.00

Table 14-150: Tailings: Quantile Analysis of Au Auger Samples

Looking at the primary percentiles, it can be seen that approx. 17 % of the metal percentage can be found in the top 10 % range (top 94 samples), and that there is a jump in the mean grade and metal content from the previous range. Closer inspection of the secondary percentiles indicates that the Au metal content changes at the 97.5 percentile, and contains nearly 6 % of the Au metal content.

Reviewing the log histograms, cumulative log histograms and the quantile analysis suggests that a top-cut of 3.30 g/t Au (mean of the 97.5 percentile) should be applied to the samples above this value in order to remove any effect of the high grade samples in the estimation process.

14.8.5. Semi-Variogram Analysis

The Tailings resource was estimated using the Inverse Distance Squared method and no semi-variogram analysis was conducted.

14.8.6. Previous Resource Estimates

The Tailing resource has been the subject to a number of historic resource estimates (both internal and public) but the single public resource estimates is the most significant. The following summary of the single public, historic resource estimate completed prior to 2010, was extracted from Olympus/North Borneo Gold sourced or supplied technical documents. Some of these historic estimates were prepared pre-NI43-101 and Terra Mining Consultants/Stevens & Associates has neither audited them nor made any attempt to classify them according to NI43-101 standards.

Although some of the more recent resource estimates are purported to have been compiled in terms of the relevant AusIMM JORC Code at that point in time. They are presented because Olympus and Terra Mining Consultants/Stevens & Associates consider them to be relevant and of historic significance.

- John Ashby (Ashby) of Ashby & Associates for Zedex Ltd in October 2008. Ashby defined an Inferred Resource (JORC 2004) of 1.291 million tonnes at 1.332 g/t Au based on the modelling and an Inferred Resource (JORC 2004) of 1.878 million tonnes at 1.332 g/t Au for the remaining historic tailings outside the modelled area, using a cutoff of 0.87 g/t Au and 0.62 g/t Au respectively.

14.8.7. Modelling & Resource Estimation Parameters

The Tailings impoundment resource wireframes were generated in Datamine and split into a north and south impoundment wireframe. These were then filled with block model cells orientated orthogonally. The block model parameters are listed in *Table 14-151: Tailings: Block Model Parameters* below.

Block Model Parameter	Block Model Value
Parent Block Cell Size	10m x 10m x 1m
Zone Code	Zone=1 & 2
Sub-Cell Size	2.5m x 2.5m x 0.25m

Table 14-151: Tailings: Block Model Parameters

For the Tailings all assays within the impoundment volume were used in the estimate. A top cut of 3.30 g/t Au was applied to all samples above this value. Limited density values were found determined from a few samples.

The average density was determined from these limited density samples and applied to the block model. The average was 1.80 t/m3 for the tailings impoundment material.

Search ellipse and Inverse Distance Squared estimation parameters were derived and are summarised in *Table 14-152: Tailings: Inverse Distance Estimation Method Parameters* below.

Estimation Parameter	Value
Search Orientation – North Impoundment	0° dip at 120° azimuth
Search Orientation – South Impoundment	0° dip at 300° azimuth
Search Ellipse Range	95m x 45m x 2m
Minimum Samples	5
Maximum Samples	20

Table 14-152: Tailings: Inverse Distance Estimation Method Parameters

14.8.8. Resource & Comparative Estimates

The resource for the Tailings impoundment was determined at a variety of lower cutoffs. *Table 14-153: Tailings: Inverse Distance Resource at 0.25 g/t Increments* below displays the results at each 0.25 g/t Au cutoff grade increment.

CUTOFF	TONNES	AU
0.5	1,400,000	1.34
0.75	1,379,000	1.35
1	1,289,000	1.38
1.25	849,000	1.50
1.5	342,000	1.72
1.75	119,000	1.91
2	25,000	2.16

Table 14-153: Tailings: Inverse Distance Resource at 0.25 g/t Increments

A lower cutoff grade of 0.5 g/t Au was selected for the tailings impoundment as this would be a reasonable cutoff value used in defining tailings resources.

A comparative estimate was undertaken using the Nearest Neighbour (3D polygonal) method. The 0.25 g/t cutoff grade increments for this estimation are shown in *Table 14-154: Tailings: Comparative Nearest Neighbour Resource at 0.25 g/t Increments* below.

CUTOFF	TONNES	AU
0.5	1,400,000	1.36
0.75	1,354,000	1.38
1	1,188,000	1.45
1.25	720,000	1.66
1.5	347,000	2.00
1.75	202,000	2.28
2	145,000	2.47

Table 14-154: Tailings: Comparative Nearest Neighbour Resource at 0.25 g/t Increments

The comparative resource estimates for the Tailings compares well with the Inverse Distance resource estimate and the minor differences probably reflect the interpolation techniques/application.

Due to the extent of the auger drilling and sampling only a portion of the modelled tailings impoundment has been estimated, and this is represented by the above resource. This resource represents approximately 60,400 ozs Au, and the remainder of the tailings resource has been calculated from the official annual tailings records and the above resource. *Figure 14-117: Tailings: Au Model Slice & Auger Positions* shows a slice through the Tailings impoundment model coloured by Au grade ranges.

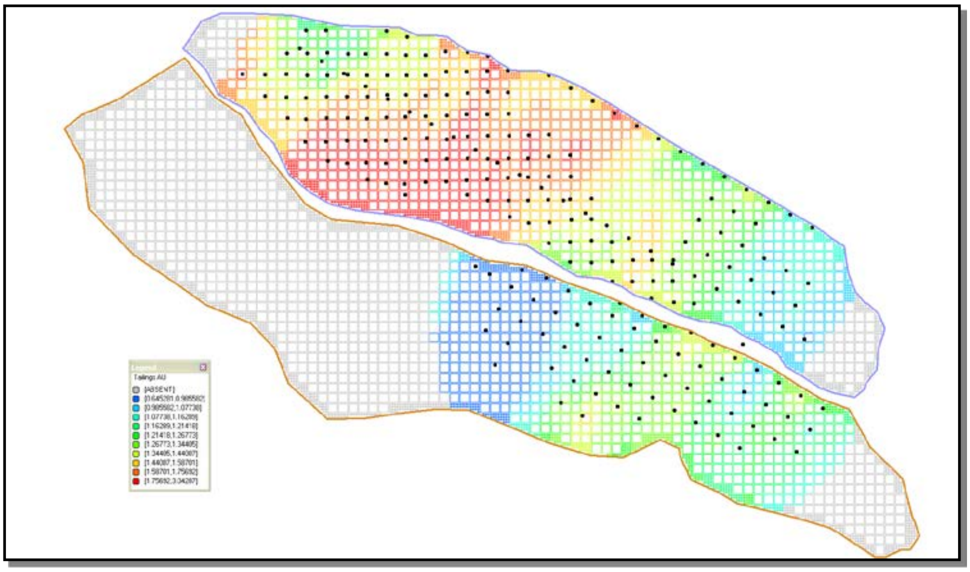


Figure 14-117: Tailings: Au Model Slice & Auger Positions

The remaining resource is 1,738,000 tonnes at 0.71 g/t Au. The total resource, modelled plus calculated remaining is 3,138,000 tonnes at 1.0 g/t Au.

The resultant resource is therefore equal to the total recorded gold placed in the tailings. *Table 14-155: Bukit Young Historic Ore Treatment & Tailings* below lists the annualized recorded tailings placement and value.

Year	Tonnes Treated (t)	Gold Content (g)	Gold Recovered (g)	Gold Recovery (%)	Gold in Tailings (g)	Gold in Tailings (ozs)	Notes
1983	*530*	-	*1,608*				Not in TSF
1984	*15,640*	-	*5,936*				Not in TSF
1985	159,832	268,635	129,336	48%	139,299	4,479	
1986	274,440	533,790	215,965	40%	317,825	10,218	Start CIL
1987	484,168	664,600	300,695	45%	363,905	11,700	
1988	514,473	732,350	428,266	58%	304,084	9,777	Start Milling
1989	360,597	477,580	239,090	50%	238,490	7,668	
1990	216,070	249,980	138,879	56%	111,101	3,572	
1991	193,970	466,830	288,705	62%	178,125	5,727	
1992	177,529	793,344	520,311	66%	273,033	8,778	
1993	280,404	1,930,705	1,546,395	80%	384,310	12,356	
1994	204,054	1,224,042	905,811	74%	318,231	10,231	
1995	161,913	712,873	460,068	65%	252,805	8,128	
1996	126,706	512,308	272,005	53%	240,303	7,726	Closure
Total	***3,154,156***	***8,567,037***	***5,445,526***	***64%***	***3,121,511***	***100,359***	
Note: Total excludes ore treated in 1983/1984 not in TSF							

Table 14-155: Bukit Young Historic Ore Treatment & Tailings

The tonnage calculated from the modelled tailings impoundment, and using an average density of 1.8 t/m^3, is 3,138,000 tonnes. This equates to 99.5 % of the tonnage in the above table and is well within an acceptable margin of error.

The resource has been classified as Inferred. Some areas of the deposit(s) could potentially have been classified as Indicated based purely on the drilling density. However, one or more of the following issues gave rise to an Inferred classification:

- Large number of RC/auger drillholes with few or no diamond core holes;
- Lack of extensive and systematic density determinations throughout the deposit;
- Gaps in the drillhole spacing or coverage and/or larger distances between drillholes;
- Difficulty in domaining of the data to remove possible mixed populations in some instances.

15. Mineral Reserve Estimates

15.1. Introduction

Besra Gold and North Borneo Gold personnel have carried out a reserve definition and assessment for parts of the Bau Project based on the associated mineral resources. The sectors (deposits) having reserve definitions and assessments conducted are Jugan and Bekajang (BYG-Krian). These deposits, or parts thereof, (along with parts of the Taiton Sector) have resources at the suitable resource confidence level for reserves to be defined, i.e. Measured and Indicated. At this stage no reserve definition work has been conducted in the Taiton Sector.

A summary of reserve totals, for the contract mining base case, by Reserve Category is shown in *Table 15-1: Reserve Summary by Category (November 2013)* and these reserves by area/sector and deposit are also shown in *Table 15-2: Reserve Summary by Sector/Area & Deposit (November 2013)* below.

Reserve Category	Tonnes (t)	Grade (g/t)
Proven	3,418,650	1.47
Probable	7,243,920	1.81
Proven + Probable	10,662,570	1.70

Table 15-1: Reserve Summary by Category (November 2013)

Sector	Reserve Category	Tonnes (t)	Grade (g/t)
Jugan	Proven	3,418,650	1.47
	Probable	6,368,190	1.61
	Proven + Probable	9,786,840	1.56
Bukit Young	Proven	0	0
	Probable	875,730	3.31
	Proven + Probable	875,730	3.31

Table 15-2: Reserve Summary by Sector/Area & Deposit (November 2013)

For the reserve definition work found in this report, Besra/NBG have classified the ore/mineral reserves according to the definitions in the National Instrument 43-101, CIMM Definitions and the Australasian Institute of Mining & Metallurgy's JORC Code 2012. A checklist from the Code covering Section 4 – Ore Reserves is included in *Appendix A15-2*. Mineral/Ore Reserves above are contained within Mineral Resources.

For the purposes of the report the relevant AusIMM reserve definitions used for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code) are listed below along with the comparative C.I.M.M. Standards for reserves. *Table 15-3: AusIMM & CIMM Comparative Reserve Definitions* below, lists comparative descriptions.

AusIMM JORC Code Definitions	C.I.M.M. Standards Definitions
An '**Ore Reserve**' is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified. Ore Reserves are sub-divided in order of increasing confidence into Probable Ore Reserves and Proved Ore Reserves.	A '**Mineral Reserve**' is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.
A **'Probable Ore Reserve'** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors These assessments demonstrate at the time of reporting that extraction could reasonably be justified.	A **'Probable Mineral Reserve'** is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.
A **'Proved Ore Reserve'** is the economically mineable part of a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed	A **'Proven Mineral Reserve'** is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic

AusIMM JORC Code Definitions	C.I.M.M. Standards Definitions
mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified.	extraction is justified.

Table 15-3: AusIMM & CIMM Comparative Reserve Definitions

For open pit inventory, the resource block model estimation methodology incorporates adequate dilution and provides a reasonable estimate of mined tonnage and grades.

Also, due to the nature of the orebody there are small waste zones, which are unable to be model discretely, and are incorporated within the overall ore zone. These can be found in the grade model with no or minor Au grade. This internal dilution is included within the overall reserves and would form the highest percentage of dilution. Peripheral waste would be avoided by detailed mapping, grade control and careful mining practices, and would be minor compared to the internal dilution and has been included at 5 % in the reserves for that reason.

Each of the areas/sectors and/or the deposits that have reserves therein are discussed in more detail in the following sections.

15.2. Jugan Sector

The Jugan reserve estimate represents that part of the Measured and Indicated Resource which can be economically mined and for which the necessary design work and mine planning have been carried out. Proved and Probable reserve blocks are based on Measured and Indicated resource blocks respectively. To be included, the blocks have to show reasonable continuity of mineralization.

Inferred blocks are considered to be inadequately defined and therefore are not included in reported reserves, although if they fall within the pit outlines they do represent potential additions to ore mined if confirmed by grade control drilling. For the purposes of these reserves they are treated as waste and not included in the reserve figures.

The Reserves are included within the overall Resource figures. Additionally, mineralised blocks below cutoff are reported as waste with no grade, although they contain low Au values.

A number of scenarios were investigated with the main options or variables as process type (POX, BIOX, Albion or Concentrate), mining type (owner-operator vs. contractor), production levels (4,000 tpd to 12,000 tpd) and plant position (central, regional, local and local front-end/central back-end). These scenarios presented a number of mining and processing cost options for optimisation work using CAE Mining's NPV Scheduler.

The potential reserves from each optimisation scenario is summarised in *Table 15-4: Jugan: Pit Optimisation – Scenario Potential Reserves (@ 8,000tpd)*. Optimisation reserves for all tonnage options are included in *Appendix A15-1*.

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	3,444,580	1.47
	Contract Mining-POX Processing	2,892,650	1.64
	Contract Mining-BIOX Processing	2,664,500	1.72
	Contract Mining-Albion Processing	2,145,010	1.91
	Owner Mining-Concentrate Production	3,452,670	1.47
	Owner Mining-POX Processing	2,912,750	1.64
	Owner Mining-BIOX Processing	2,674,220	1.72
	Owner Mining-Albion Processing	2,153,290	1.90
Probable	Contract Mining-Concentrate Production	6,475,920	1.61
	Contract Mining-POX Processing	5,539,620	1.74
	Contract Mining-BIOX Processing	5,248,330	1.78
	Contract Mining-Albion Processing	4,285,540	1.91
	Owner Mining-Concentrate Production	6,705,100	1.59
	Owner Mining-POX Processing	5,788,260	1.73
	Owner Mining-BIOX Processing	5,476,530	1.77
	Owner Mining-Albion Processing	4,532,840	1.90

Table 15-4: Jugan: Pit Optimisation – Scenario Potential Reserves (@ 8,000tpd)

The economic pit limit evaluations, open pit development sequence plans, and reserve estimates are based on a gold price of $1,500/oz. This is the gold price used in the optimisation to define the ultimate pit, with the optimal pit used being within this limit. Differing gold prices have been used in the cost models and sensitivity optimisations were done at $1,200 and $2,000 per ounce gold prices.

Results from the $1,200/oz and $1,500/oz optimisations showed only a small difference in tonnes (<1%) and grade (<1%). A variety of gold prices have been reviewed and a sensitivity analysis conducted. These results are presented in *Section 28* and risk aspects are covered in *Section 29*.

Process recoveries used are an effective recovery of 77 % for the base case concentrate option. The concentrate recovery option is based on a 90 % flotation recovery, recovery for contract processing facility and their percentage of metal content (80%). For the optimisations using the other metallurgical processes, the following recoveries were used - 85 % (POX), 80 % (BIOX & ALBION).

Mining recovery is assumed at 95 % with a 5 % dilution factor applied.

Base mining cost used for 8,000tpd and concentrate base case was $1.735/t and this relates to overburden removal, with mine cost adjustment factors (MCAF) of 1.518 and 1.338 for ore mining and waste mining respectively. Processing cost for the base case concentrate option was $7.57/t with the G&A's and other selling costs as $0.16/g. Mining, processing and other costs for other processes are detailed in *Chapter 21* of this report.

Detailed optimisation parameters for all of the options, plus details about the optimisation process are outlined in the following *Chapter 17*.

Using the base case optimisation scenario (flotation concentrate, 8,000 tpd and owner-operator) a detailed pit design was undertaken and the reserves from that work are shown in *Table 15-5: Jugan: Detailed Design Pit Reserves (Owner-Operator)* below and grouped with BYG-Krian in *Table 15-2: Reserve Summary by Sector/Area & Deposit (July 2013)* in *Section 16.2.1*. Note that Mineral/Ore Reserves below are contained within Mineral Resources.

Reserve Category	Tonnes (t)	Grade (g/t Au)
Proven	3,412,860	1.47
Probable	6,525,210	1.61
Proven + Probable	9,938,070	1.56

Table 15-5: Jugan: Detailed Design Pit Reserves (Owner-Operator)

Detailed design parameters, results plus details about the design process for the above Jugan reserves are outlined in the following *Chapter 15*.

Comparing the designed total pit reserves (9,938,070 tonnes ore at 1.56 g/t Au) and that generated via the optimisation software (10,157,780 tonnes ore at 1.55 g/t Au), for the same scenario, the reserves (Proven + Probable) are comparable and show 2.2 % difference in tonnage and 0.5 % difference in grade. This difference is negligible in relation to the orebody modelling and design resolution. Therefore, the optimised schedules can be accepted as reasonable level for reserve generation for the open pits.

The comparison for the other base case (flotation concentrate, 8,000 tpd and contract-mining) is 9,786,840 tonnes at 1.56 g/t Au for design and 9,920,500 tonnes at 1.56 g/t Au for optimisation, which is a difference of 1.4 % in tonnage and 0.4 % difference in grade. The contract mining basecase reserves are detailed in *Table 15-6 - Jugan: Detailed Design Pit Reserves (Contract-Mining)* below.

Reserve Category	Tonnes (t)	Grade (g/t Au)
Proven	3,418,650	1.47
Probable	6,368,190	1.61
Proven + Probable	9,786,840	1.56

Table 15-6 - Jugan: Detailed Design Pit Reserves (Contract-Mining)

Based on the optimisation runs and the applied parameters a cut-off grade of 0.39 to 0.44 g/t Au is realised for the Jugan reserves, with a strip ratio of 1.604/1.472 for owner-operator and contract-mining options, respectively.

For open pit inventory, the resource block model estimation methodology incorporates dilution and provides a reasonable estimate of mined tonnage and grades. However, an additional 5 % dilution is added with a 95 % mining recovery have been included as an additional factor in the pit optimisation process and in the reserves.

15.3. Bekajang/Krian Sector

The BYG-Krian reserve estimate is treated in the same fashion as the Jugan reserves above, and those notes apply here. Note that Mineral/Ore Reserves below are contained within Mineral Resources.

The potential reserves from pit optimisation work, base case pit optimisation parameters for the BYG-Krian pit and detailed design reserves are summarised in *Table 15-7: BYG-Krian: Pit Optimisation – Scenario Potential Reserves (@ 8,000 tpd)* and *Table 15-8: BYG-Krian: Detailed Design Pit Reserves (Owner-Operator)* and *Table 15-9 - BYG-Krian: Detailed Design Pit Reserves (Contract-Mining)*. Optimisation reserves for all tonnage options are included in *Appendix A16-1*.

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	0	0
	Contract Mining-POX Processing	0	0
	Contract Mining-BIOX Processing	0	0
	Contract Mining-Albion Processing	0	0
	Owner Mining-Concentrate Production	0	0
	Owner Mining-POX Processing	0	0
	Owner Mining-BIOX Processing	0	0
	Owner Mining-Albion Processing	0	0
Probable	Contract Mining-Concentrate Production	1,007,380	3.13
	Contract Mining-POX Processing	901,240	3.42
	Contract Mining-BIOX Processing	818,860	3.62
	Contract Mining-Albion Processing	722,380	3.95
	Owner Mining-Concentrate Production	1,051,310	3.08
	Owner Mining-POX Processing	931,540	3.38
	Owner Mining-BIOX Processing	869,050	3.52
	Owner Mining-Albion Processing	770,760	3.85

Table 15-7: BYG-Krian: Pit Optimisation – Scenario Potential Reserves (@ 8,000 tpd)

Pit optimisation parameters for BYG-Krian are as per Jugan. Pit slope and some other pit optimisation aspects are also different but these are detailed in the *Chapter 16*.

Reserve Category	**Tonnes (t)**	**Grade (g/t)**
Proven	0	0
Probable	939,030	3.10
Proven + Probable	939,030	3.10

Table 15-8: BYG-Krian: Detailed Design Pit Reserves (Owner-Operator)

Reserve Category	**Tonnes (t)**	**Grade (g/t)**
Proven	0	0
Probable	875,730	3.31
Proven + Probable	875,730	3.31

Table 15-9 - BYG-Krian: Detailed Design Pit Reserves (Contract-Mining)

Detailed design parameters, results plus details about the design process for the above BYG-Krian reserves are outlined in the following *Chapter 16*.

As per Jugan the above detailed design reserves are comparable with the optimised reserves.

Based on the optimisation runs and the applied parameters a cut-off grade of 0.58 to 0.65 g/t Au is realised for the BYG-Krian reserves, with a strip ratio of 4.41/3.94 for owner-operator and contract-mining options, respectively.

For open pit inventory, the resource block model estimation methodology incorporates dilution and provides a reasonable estimate of mined tonnage and grades. However, an additional 5 % dilution is added with a 95 % mining recovery have been included as an additional factor in the pit optimisation process.

Although the BYG-Krian pit is small when considering the Indicated only, it has additional potential in terms of the inferred both under the pit and indicated zone but also in shallow extensions around. As the resource is Inferred, in this case it cannot be considered in the reserves, the potential for pit expansion is significant in terms of the current reserves.

This can easily be upgraded with some additional resource drilling and conversion to indicated. Listed below in *Table 15-10 – Comparison of Potential between Indicted Only & Indicated-Inferred Resources at BYG-Krian* is a comparison of the resources if Inferred was available as Indicated and *Figure 15-1 - 3D Comparison of Indicated Only (top) & Indicated-Inferred (bottom) Pits* below shows this impact visually.

	Using Indicated Only		Using Indicated & Inferred	
	Tonnes	**Au (g/t)**	**Tonnes**	**Au (g/t)**
Ultimate Pit (Shell 65)	1,026,890	3.13	2,808,890	2.09
Optimal Pit (Shell 50)	1,007,380	3.11	2,696,450	2.11
Designed Pit	875,730	3.31	2,093,510	2.35

Table 15-10 – Comparison of Potential between Indicted Only & Indicated-Inferred Resources at BYG-Krian

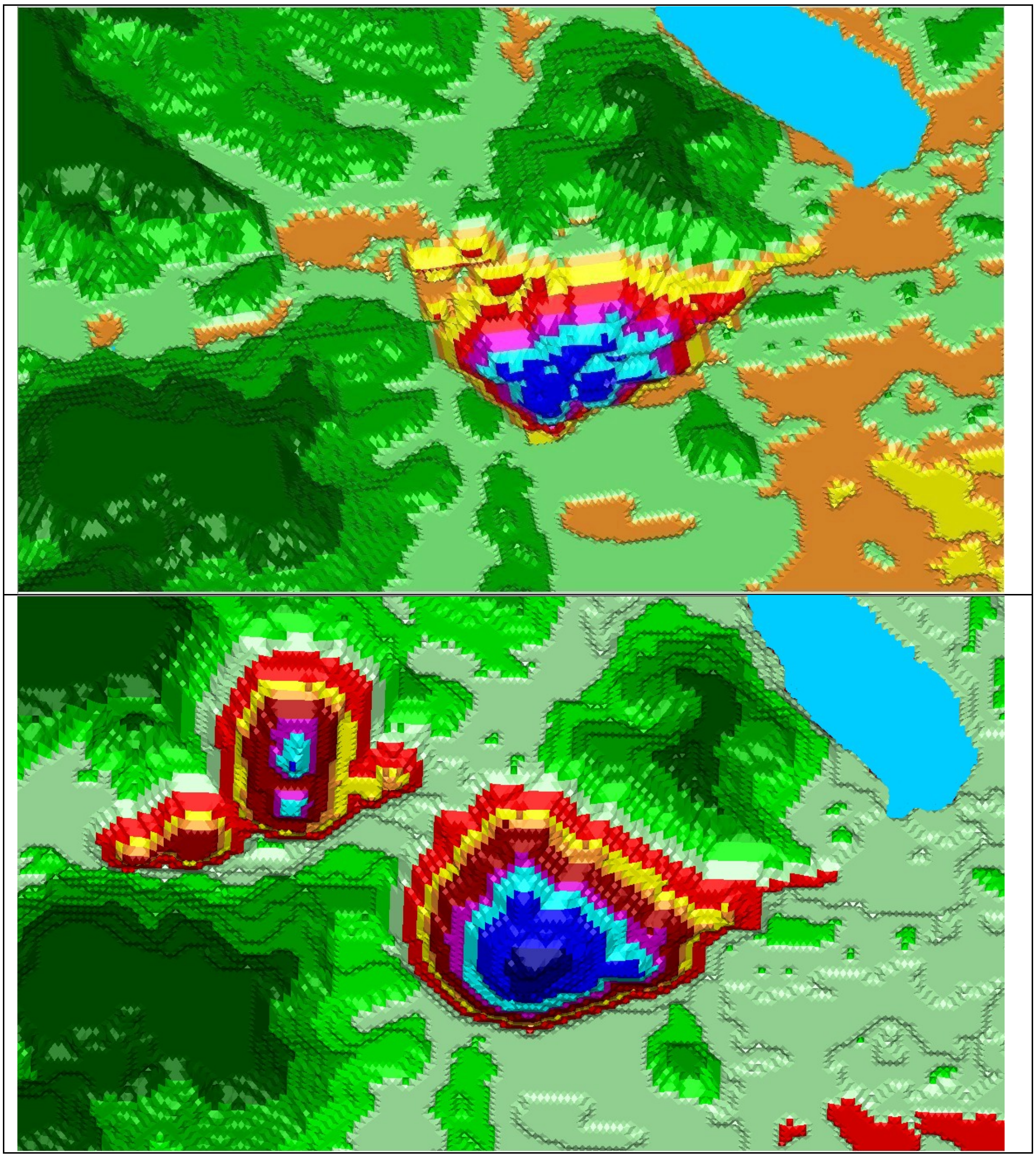


Figure 15-1 - 3D Comparison of Indicated Only (top) & Indicated-Inferred (bottom) Pits

It should be noted the above is only included for comparative purposes and should not be considered as reserves.

16. Mine Planning & Scheduling

16.1. Introduction & Mining Method

Due to the orebody outcropping as a hill and having significant resources at a shallow depth the initial method of extraction of ore is by open pit methods. The orebody does extend down to a depth of approximately 400m and is anticipated to carry on to further depths. Future mining will probably be by underground methods. However, for the purposes of this Feasibility Study the shallower open pit extraction portion is applicable and has been used.

This section will outline the economic model generation and pit optimisation work and parameters, along with the detailed pit design and associated mining feature designs, namely the waste engineered landform (disposal) area and tailings storage facility.

16.2. Pit Optimisation

16.2.1. Mining Model & Ore Characterisation

The resource models for Jugan and BYG-Krian were processed and characterised to derive a "mining" block model for use in the optimisation process and for future design work. For Jugan the surrounding waste blocks were included along with the waste intrusives and combined with the ore model to generate a total model. Likewise the BYG-Krian ore model was added to the waste model. Note all models were trimmed to topography (if not previously done) and modified for any previous mining.

Waste and ore zones were coded for identification based on the respective resource/reserve category and major orezone type.

For Jugan, when loading into the optimiser, the waste (Orezone=3), intrusive (Orezone=2), Inferred (Orezone=1 and Resource Category=3) and geological potential (Orezone=1 and Resource Category=NULL) were defined as waste. Ore was defined using Measured (Orezone=1 and Resource Category=1) and Indicated (Orezone=1 and Resource Category=2) resources.

For BYG-Krian the ore was defined as a combination of Indicated (Resource Category=2) and ore resource zones (1 to 14). Waste was based on the waste blocks (Zone=20) and the Inferred (Resource Category=1) and geological potential (Resource Category=-).

Also when loading into the optimiser the default density was set to 2.6 for Jugan and BYG-Krian. This will be applied where no density value exists. Also other attributes are loaded for reporting purposes only. These are Arsenic (As), Iron (Fe) and Sulphur (S). They are reported to determine the ore feed characteristics, for these elements, to the plant over time.

16.2.2. Economic Model Parameters

As part of the Feasibility Study a number of scenarios are being investigated to determine the best approach and result for the Bau Project, and the initial mining at Jugan with follow-up mining at BYG-Krian. These scenarios are based on a combination of contractor mining vs. owner-operator mining options and process options, namely Pressure Oxidation (POX), Biological Oxidation (BIOX), Albion Process and Flotation Concentrate option (with supply to external toll treatment facility). Each of the process and mining combinations was run at different production levels ranging from 4,000 tpd to 12,000 tpd.

Also, plant position options were also considered in combination with the mining and process option combinations, and this is relevant in terms of transport costs mainly. Other options (e.g. heap leach of low grade ore) are also provided for as scenario options but will be investigated later and are not part of this current study.

16.2.2.1. Economic Model Parameters – Jugan

The economic model parameters used to derive the economic model for the base case (8,000tpd) for Jugan are summarised in *Table 16-1: Jugan - Economic Model Parameters for Pit* Optimisation below.

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Base Mining Cost (Contractor)	$/tonne	1.735	1.735	1.735	1.735
MCAF – Ore		1.518	1.518	1.518	1.518
MCAF – Waste/Intrusive		1.338	1.338	1.338	1.338
Base Mining Cost (Owner)	$/tonne	2.1675	2.1675	2.1675	2.1675
MCAF – Ore		1.6183	1.6183	1.6183	1.6183
MCAF – Waste/Intrusive		1.4755	1.4755	1.4755	1.4755
Incremental Cost per Bench	$/tonne	0.05	0.05	0.05	0.05
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost	$/g	2.91	-	-	-

Table 16-1: Jugan - Economic Model Parameters for Pit Optimisation

16.2.2.2. Economic Model Parameters – BYG-Krian

The economic model parameters used to derive the economic model for the base case (8,000tpd) for BYG-Krian are summarised in *Table 16-2: BYG-Krian – Economic Model Parameters for Pit Optimisation* below.

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Reference Mining Cost	$/tonne	1.735	1.735	1.735	1.735
MCAF – Ore		1.518	1.518	1.518	1.518
MCAF – Waste/Intrusive		1.338	1.338	1.338	1.338
Base Mining Cost (Owner)	$/tonne	2.1675	2.1675	2.1675	2.1675
MCAF – Ore		1.6183	1.6183	1.6183	1.6183
MCAF – Waste/Intrusive		1.4755	1.4755	1.4755	1.4755
Incremental Cost per Bench	$/tonne	0.05	0.05	0.05	0.05
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost	$/g	1.90	-	-	-

Table 16-2: BYG-Krian – Economic Model Parameters for Pit Optimisation

From the above parameters an economic model is built for each deposit, attributing value to each model cell based on what type of cell it is and the relative contribution (positive or negative) that block provides.

The economic parameters for the other production tonnage options, for both BYG-Krian and Jugan, are listed in *Appendix A17-1*.

16.2.3. Ultimate Pit & Parameters

16.2.3.1. Ultimate Pit & Parameters – Jugan

Using the developed economic models for each mining/process option and applying the ultimate pit parameters, a set of ultimate pits is defined. The ultimate pit parameters used to determine the ultimate pit for Jugan is listed in *Table 16-3 - Jugan - Ultimate Pit Parameters for Pit Optimisation* below.

Parameters	Units	Flotation	POX	BIOX	ALBION
Discount Rate	%	8	8	8	8
Ore Extraction Rate (Minimum)	tpd	4,000	4,000	4,000	4,000

Parameters	Units	Flotation	POX	BIOX	ALBION
Ore Extraction Rate (Maximum)	tpd	12,000	12,000	12,000	12,000
Ore Extraction Rate (Increment)	tpd	2,000	2,000	2,000	2,000
Pit Overall Slope Angle	°				
Azimuth = 0	°	43	43	43	43
Azimuth = 45	°	45	45	45	45
Azimuth = 120	°	40	40	40	40
Azimuth = 180	°	45	45	45	45
Azimuth =270	°	45	45	45	45

Table 16-3 - Jugan - Ultimate Pit Parameters for Pit Optimisation

Some of the resulting ultimate pits for Jugan are shown graphically in 3D in *Figure 16-1 - Jugan: Original Pit Optimisation Topography* to *Figure 16-5: Jugan - Contract Mining-ALBION Ultimate Pit* below, for each mining/process scenario option along with the original topography.

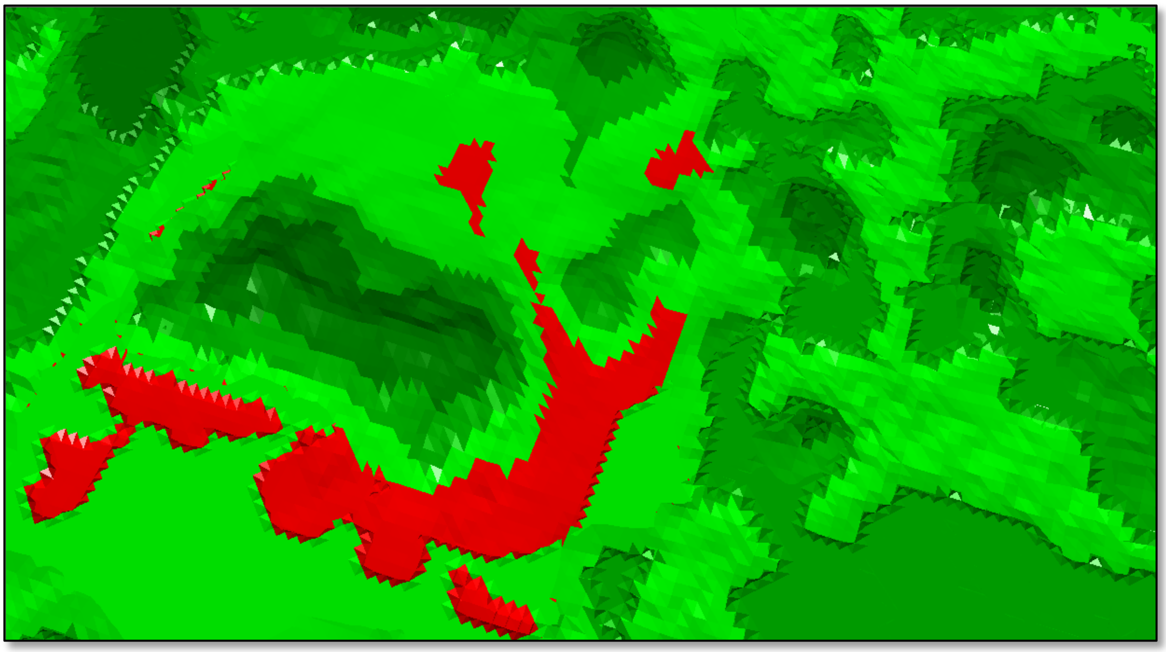


Figure 16-1 - Jugan: Original Pit Optimisation Topography

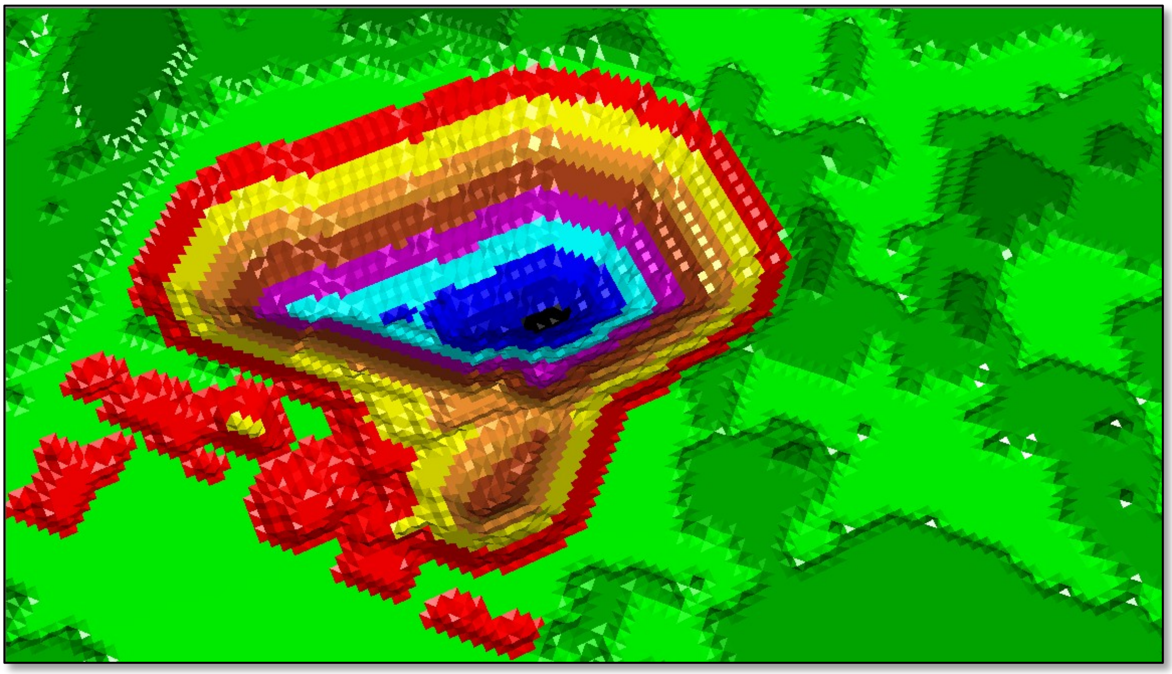


Figure 16-2: Jugan - Contract Mining-Flotation Ultimate Pit

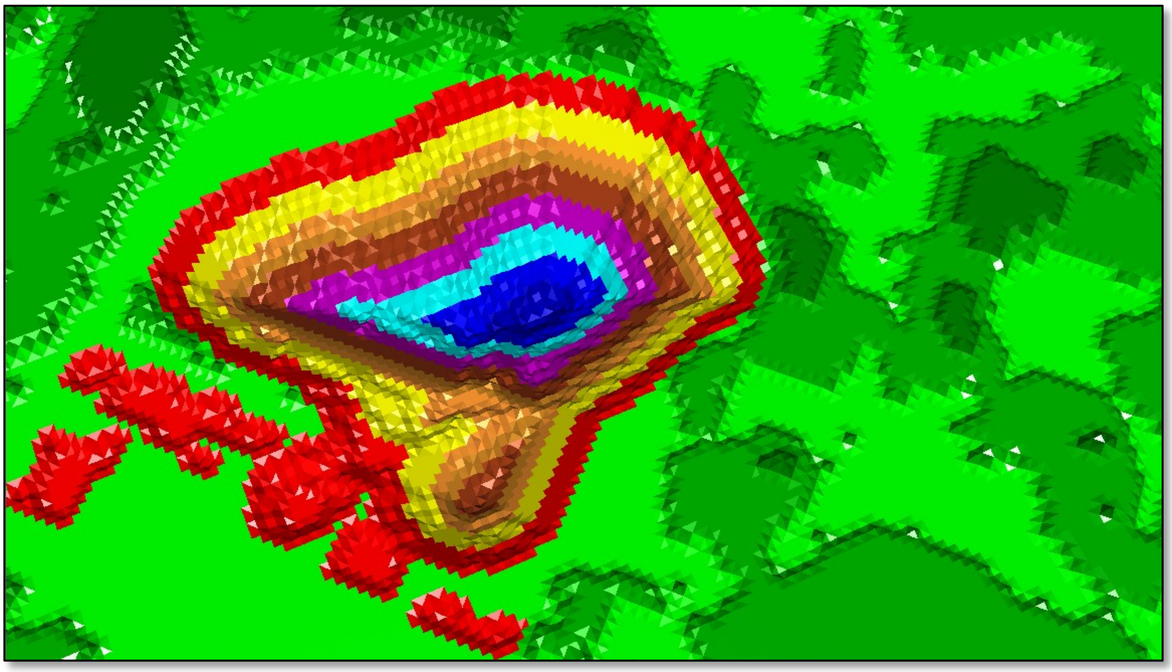


Figure 16-3: Jugan - Contract Mining-POX Ultimate Pit

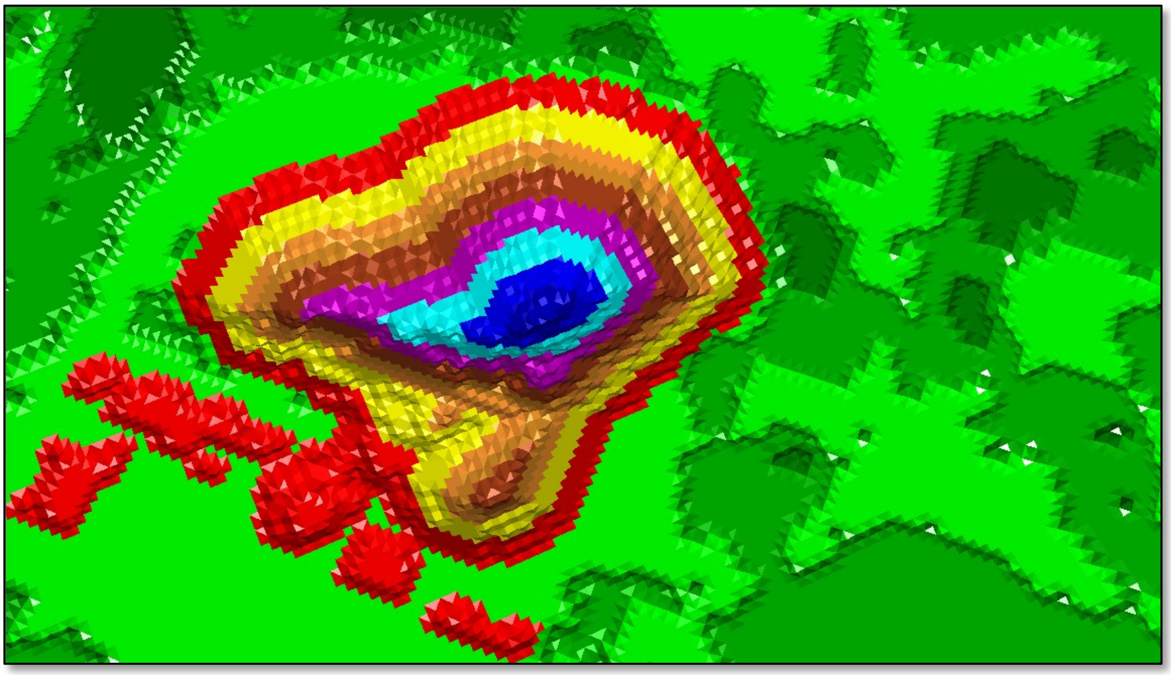


Figure 16-4: Jugan - Contract Mining-BIOX Ultimate Pit

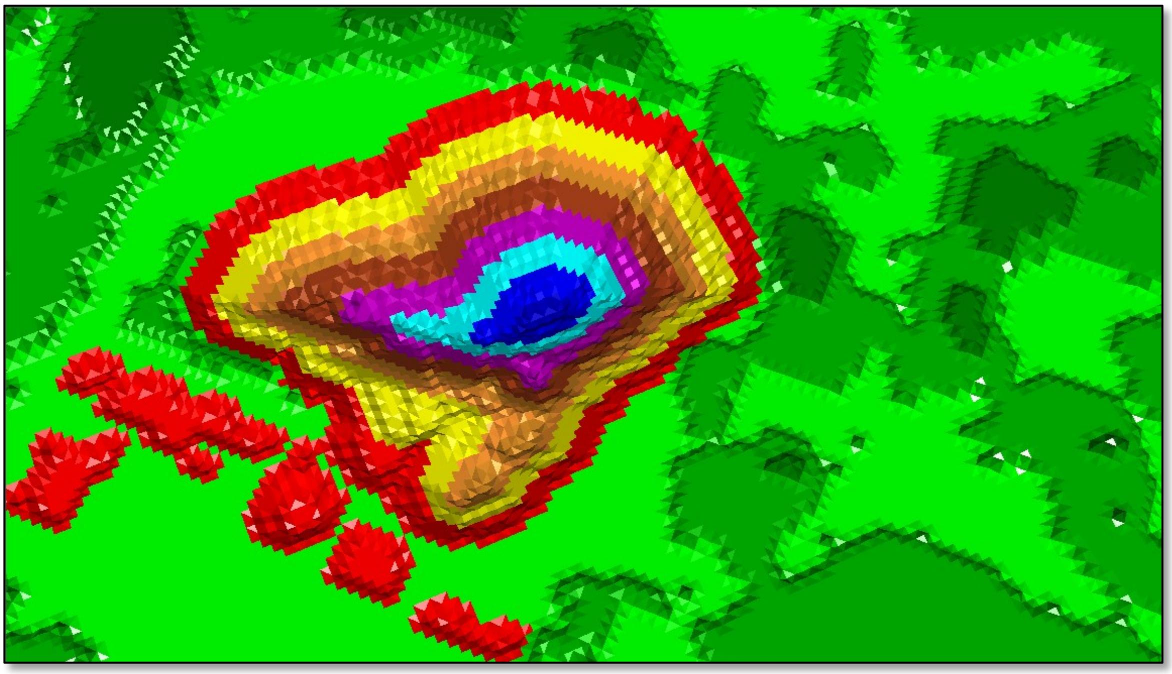


Figure 16-5: Jugan - Contract Mining-ALBION Ultimate Pit

The ore and key results of the ultimate pit process are listed in *Table 16-4 - Jugan Owner Operator - Ultimate Pit Results* and *Table 16-5 - Jugan Contract Mining - Ultimate Pit Results* below. Detailed summary tabulations are listed in *Appendix A17-2*.

Ultimate Pit Description	**Tonnage (t)**	**Grade (g/t)**	**Pit Shell Number**	**Strip Ratio**
Owner-Operator_4000 TPD_Flotation	10,218,950	1.546	Pit 67	1.646
Owner-Operator_6000 TPD_Flotation	10,282,950	1.543	Pit 68	1.683

Ultimate Pit Description	Tonnage (t)	Grade (g/t)	Pit Shell Number	Strip Ratio
Owner-Operator_8000 TPD_Flotation (base case)	12,916,270	1.439	Pit 68	3.324
Owner-Operator_10000 TPD_Flotation	13,109,200	1.433	Pit 67	3.430
Owner-Operator_12000 TPD_Flotation	13,113,110	1.433	Pit 67	3.434
Owner-Operator_4000 TPD_POX	8,672,520	1.700	Pit 70	1.735
Owner-Operator_6000 TPD_POX	8,784,610	1.693	Pit 65	1.781
Owner-Operator_8000 TPD_POX	8,851,760	1.688	Pit 64	1.800
Owner-Operator_10000 TPD_POX	8,875,790	1.687	Pit 65	1.810
Owner-Operator_12000 TPD_POX	8,888,790	1.686	Pit 64	1.827
Owner-Operator_4000 TPD_BIOX	7,996,940	1.759	Pit 58	1.700
Owner-Operator_6000 TPD_BIOX	8,099,590	1.754	Pit 61	1.752
Owner-Operator_8000 TPD_BIOX	8,164,720	1.754	Pit 62	1.842
Owner-Operator_10000 TPD_BIOX	8,191,230	1.753	Pit 69	1.853
Owner-Operator_12000 TPD_BIOX	8,204,850	1.752	Pit 64	1.868
Owner-Operator_4000 TPD_ALBION	6,556,500	1.904	Pit 65	1.881
Owner-Operator_6000 TPD_ALBION	6,680,180	1.904	Pit 62	2.030
Owner-Operator_8000 TPD_ALBION	6,733,720	1.9015	Pit 59	2.065
Owner-Operator_10000 TPD_ALBION	6,744,460	1.901	Pit 63	2.066
Owner-Operator_12000 TPD_ALBION	6,796,850	1.900	Pit 64	2.146

Table 16-4 - Jugan Owner Operator - Ultimate Pit Results

Ultimate Pit Description	Tonnage (t)	Grade (g/t)	Pit Shell Number	Strip Ratio
Contract-Mining_4000 TPD_Flotation	10,005,520	1.555	Pit 68	1.505
Contract-Mining_6000 TPD_Flotation	10,031,950	1.553	Pit 62	1.516
Contract-Mining_8000 TPD_Flotation (base case)	10,114,130	1.552	Pit 65	1.587
Contract-Mining_10000 TPD_Flotation	10,201,610	1.548	Pit 67	1.641
Contract-Mining_12000 TPD_Flotation	10,234,560	1.546	Pit 66	1.661
Contract-Mining_4000 TPD_POX	8,395,700	1.711	Pit 66	1.568
Contract-Mining_6000 TPD_POX	8,473,030	1.707	Pit 67	1.602
Contract-Mining_8000 TPD_POX	8,547,660	1.704	Pit 68	1.641
Contract-Mining_10000 TPD_POX	8,626,890	1.703	Pit 69	1.726
Contract-Mining_12000 TPD_POX	8,672,520	1.700	Pit 70	1.735
Contract-Mining_4000 TPD_BIOX	7,790,120	1.761	Pit 59	1.523
Contract-Mining_6000 TPD_BIOX	7,858,320	1.759	Pit 65	1.563
Contract-Mining_8000 TPD_BIOX	7,923,630	1.763	Pit 62	1.664
Contract-Mining_10000 TPD_BIOX	7,988,860	1.759	Pit 59	1.690
Contract-Mining_12000 TPD_BIOX	7,996,940	1.759	Pit 63	1.700
Contract-Mining_4000 TPD_Albion	6,320,730	1.908	Pit 69	1.715

Ultimate Pit Description	Tonnage (t)	Grade (g/t)	Pit Shell Number	Strip Ratio
Contract-Mining_6000 TPD_Albion	6,440,380	1.910	Pit 67	1.830
Contract-Mining_8000 TPD_Albion	6,462,550	1.908	Pit 68	1.833
Contract-Mining_10000 TPD_Albion	6,545,900	1.904	Pit 70	1.874
Contract-Mining_12000 TPD_Albion	6,555,290	1.904	Pit 66	1.881

Table 16-5 - Jugan Contract Mining - Ultimate Pit Results

16.2.3.2. Ultimate Pit & Parameters – BYG-Krian

The ultimate pit parameters used to determine the ultimate pit for BYG-Krian are listed in *Table 16-6: BYG-Krian - Ultimate Pit Parameters for Pit Optimisation* below.

Parameters	Units	Flotation	POX	BIOX	ALBION
Discount Rate	%	8	8	8	8
Ore Extraction Rate (Minimum)	tpd	4,000	4,000	4,000	4,000
Ore Extraction Rate (Maximum)	tpd	12,000	12,000	12,000	12,000
Ore Extraction Rate (Increment)	tpd	2,000	2,000	2,000	2,000
Pit Overall Slope Angle	°	47	47	47	47

Table 16-6: BYG-Krian - Ultimate Pit Parameters for Pit Optimisation

Similarly, for BYG-Krian ultimate pit, a 3D example of the current topography (*Figure 16-6 - BYG-Krian: Original Pit Optimisation Topography*) and BYG ultimate pit (*Figure 16-7: BYG-Krian – Contract Mining-Flotation Ultimate Pit*) is shown below and the ore and key results are shown in *Table 16-7 - BYG-Krian Owner-Operator – Ultimate Pit Results* and *Table 16-8 - BYG-Krian Contract Mining – Ultimate Pit Results* below the images. A detailed summary tabulation is listed in *Appendix A17-2*.

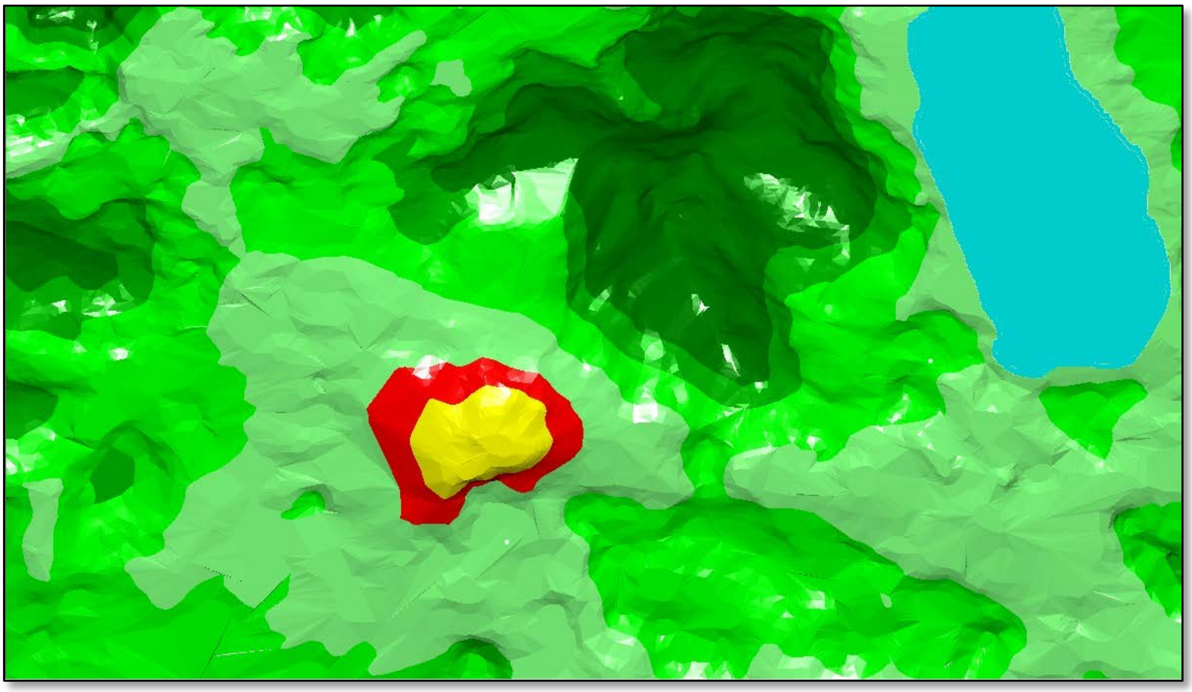


Figure 16-6 - BYG-Krian: Original Pit Optimisation Topography

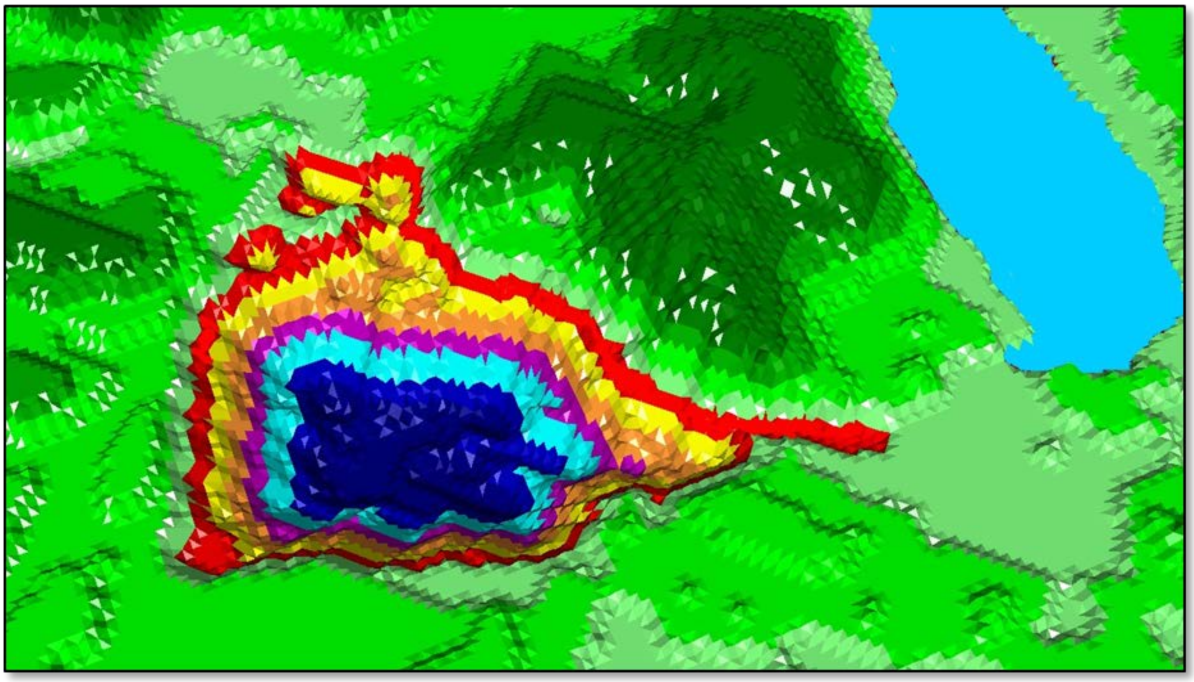


Figure 16-7: BYG-Krian – Contract Mining-Flotation Ultimate Pit

Ultimate Pit Description	Tonnage (t)	Grade (g/t)	Pit Shell Number	Strip Ratio
Owner-Operator_4000 TPD_Flotation	1,046,790	3.086	Pit 67	4.403
Owner-Operator_6000 TPD_Flotation	1,050,270	3.080	Pit 68	4.420
Owner-Operator_8000 TPD_Flotation (base case)	1,060,190	3.065	Pit 65	4.535
Owner-Operator_10000 TPD_Flotation	1,217,360	2.754	Pit 67	4.640
Owner-Operator_12000 TPD_Flotation	1,219,780	2.752	Pit 66	4.672
Owner-Operator_4000 TPD_POX	922,580	3.393	Pit 58	4.588
Owner-Operator_6000 TPD_POX	929,790	3.382	Pit 53	4.673
Owner-Operator_8000 TPD_POX	934,800	3.373	Pit 59	4.732
Owner-Operator_10000 TPD_POX	944,330	3.354	Pit 61	4.799
Owner-Operator_12000 TPD_POX	945,090	3.353	Pit 62	4.823
Owner-Operator_4000 TPD_BIOX	866,330	3.531	Pit 56	4.516
Owner-Operator_6000 TPD_BIOX	874,770	3.516	Pit 58	4.603
Owner-Operator_8000 TPD_BIOX	884,140	3.511	Pit 54	4.866
Owner-Operator_10000 TPD_BIOX	888,310	3.501	Pit 60	4.880
Owner-Operator_12000 TPD_BIOX	889,040	3.499	Pit 56	4.884
Owner-Operator_4000 TPD_ALBION	742,790	3.901	Pit 55	4.633
Owner-Operator_6000 TPD_ALBION	744,710	3.898	Pit 51	4.673
Owner-Operator_8000 TPD_ALBION	774,910	3.848	Pit 56	5.317
Owner-Operator_10000 TPD_ALBION	776,890	3.844	Pit 64	5.350
Owner-Operator_12000 TPD_ALBION	779,400	3.841	Pit 61	5.418

Table 16-7 - BYG-Krian Owner-Operator – Ultimate Pit Results

Ultimate Pit Description	Tonnage (t)	Grade (g/t)	Pit Shell Number	Strip Ratio
Contract_Mining_4000 TPD_Flotation	1,004,060	3.141	Pit 63	3.931
Contract_Mining_6000 TPD_Flotation	1,012,330	3.127	Pit 64	3.981
Contract_Mining_8000 TPD_Flotation (base case)	1,026,890	3.109	Pit 63	4.140
Contract_Mining_10000 TPD_Flotation	1,046,250	3.087	Pit 64	4.399
Contract_Mining_12000 TPD_Flotation	1,046,920	3.086	Pit 63	4.405
Contract_Mining_4000 TPD_POX	869,190	3.482	Pit 49	4.025
Contract_Mining_6000 TPD_POX	881,140	3.456	Pit 53	4.085
Contract_Mining_8000 TPD_POX	908,500	3.412	Pit 55	4.393
Contract_Mining_10000 TPD_POX	921,390	3.395	Pit 54	4.567
Contract_Mining_12000 TPD_POX	924,570	3.393	Pit 57	4.644
Contract_Mining_4000 TPD_BIOX	820,620	3.619	Pit 56	4.084
Contract_Mining_6000 TPD_BIOX	827,040	3.607	Pit 54	4.143
Contract_Mining_8000 TPD_BIOX	836,700	3.585	Pit 50	4.196

Ultimate Pit Description	Tonnage (t)	Grade (g/t)	Pit Shell Number	Strip Ratio
Contract_Mining_10000 TPD_BIOX	858,780	3.542	Pit 52	4.403
Contract_Mining_12000 TPD_BIOX	865,700	3.533	Pit 54	4.519
Contract_Mining _4000 TPD_ALBION	717,410	3.973	Pit 56	4.519
Contract_Mining _6000 TPD_ALBION	723,500	3.956	Pit 56	4.551
Contract_Mining_8000 TPD_ALBION	731,180	3.934	Pit 51	4.584
Contract_Mining_10000 TPD_ALBION	741,150	3.907	Pit 55	4.638
Contract_Mining_12000 TPD_ALBION	742,790	3.901	Pit 56	4.633

Table 16-8 - BYG-Krian Contract Mining – Ultimate Pit Results

16.2.4. Optimal Pit Selection & Pit Design

Reviewing the results of the ultimate pit runs for each scenario option and considering the NPV values, ore tonnage (incremental and cumulative), waste tonnage (incremental and cumulative), strip ratios (incremental and cumulative) and practical considerations; and the optimal pit shell is selected, which is equal to or lesser than the ultimate pit.

Once the optimal pit shell has been selected for each scenario option the shell is used in conjunction with the geotechnical data and design parameters to design a detailed optimal pit. This pit shell is then used in the subsequent pushback and optimisation scheduler to define the major extraction phases, pit schedules and reserves for the base case scenario and other scenario options.

16.2.4.1. Optimal Pit & Design – Jugan

The basis for selecting the optimal shell was to choose the shell which maximizes the NPV whilst minimising the waste mining and strip ratio. At some point the minimal increment in the NPV is offset by the rise in, or additional waste produced or where there is a significant step in the waste production. The same applies to the strip ratio.

In the base case (owner-operator) the pit shell selected is Pit_Shell_54. The waste rises by 526,460 tonnes after this point which correlates to a 3.2 % increase, whilst the increase in the ore tonnes and NPV is 0.5 % and 0.08 % respectively. At Pit_shell_66 the waste tonnage jumps to 21,133,710 tonnes. Strip ratio also rises individually and cumulatively by 162% and 107%.

The pit shell selection (owner-operator) is also demonstrated in *Table 16-9 – Jugan: Incremental Pit Shell Data* (Owner-Operator) and *Table 16-10 – Jugan: Cumulative Pit Shell Data* (Owner-Operator), and also shown in the graph in *Figure 16-8 - Jugan: Graph of Cumulative Ore & Waste Tonnes and NPV (Owner-Operator)* below.

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 1	161,730,427	5,411,960	4,241,697	1,170,263	0.276

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 2	21,456,232	949,327	560,447	388,880	0.694
Pit 3	13,641,977	651,173	366,855	284,318	0.775
Pit 4	18,441,573	956,363	477,962	478,401	1.001
Pit 5	18,334,516	1,137,190	541,320	595,870	1.101
Pit 6	6,631,664	470,218	193,744	276,474	1.427
Pit 7	13,386,290	1,030,177	410,322	619,855	1.511
Pit 8	5,808,166	492,150	186,568	305,582	1.638
Pit 9	10,866,610	985,932	347,578	638,354	1.837
Pit 10	6,999,891	644,388	198,758	445,630	2.242
Pit 11	7,198,046	747,576	235,650	511,927	2.172
Pit 12	1,975,507	251,205	80,535	170,670	2.119
Pit 13	6,466,233	765,437	202,010	563,426	2.789
Pit 14	322,381	44,410	15,810	28,600	1.809
Pit 15	7,104,086	1,067,046	289,185	777,861	2.690
Pit 16	601,316	90,565	19,896	70,670	3.552
Pit 17	379,972	89,770	19,616	70,154	3.576
Pit 18	4,560,600	713,216	159,257	553,959	3.478
Pit 19	545,023	95,824	25,395	70,429	2.773
Pit 20	5,807,777	1,170,166	243,740	926,427	3.801
Pit 21	718,033	168,011	40,394	127,617	3.159
Pit 22	2,056,989	440,503	99,450	341,053	3.429
Pit 23	504,469	115,724	20,113	95,610	4.754
Pit 24	694,426	152,992	27,382	125,610	4.587
Pit 25	2,306,495	571,179	64,057	507,122	7.917
Pit 26	1,793,777	500,212	109,931	390,281	3.550
Pit 27	950,048	296,773	47,822	248,951	5.206
Pit 28	418,450	162,155	39,541	122,614	3.101
Pit 29	3,640,345	1,081,976	181,774	900,202	4.952
Pit 30	540,442	194,134	31,797	162,337	5.105
Pit 31	300,078	106,958	17,226	89,732	5.209
Pit 32	521,153	225,362	54,288	171,074	3.151
Pit 33	2,387,923	790,335	97,349	692,985	7.119
Pit 34	314,688	170,596	32,680	137,915	4.220
Pit 35	317,291	127,506	19,387	108,119	5.577
Pit 36	52,259	31,421	7,572	23,849	3.150
Pit 37	581,978	347,456	68,996	278,460	4.036
Pit 38	772,715	417,504	63,253	354,251	5.601
Pit 39	416,590	284,397	56,881	227,516	4.000
Pit 40	1,392,268	756,142	54,045	702,097	12.991
Pit 41	107,249	94,662	7,847	86,815	11.063
Pit 42	418,094	327,338	54,808	272,530	4.972
Pit 43	117,775	114,733	20,440	94,293	4.613
Pit 44	2,027	3,968	588	3,379	5.745

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 45	29,567	23,321	1,906	21,416	11.237
Pit 46	123,938	141,107	22,008	119,099	5.412
Pit 47	20,119	20,835	2,559	18,276	7.143
Pit 48	26,653	48,708	6,437	42,271	6.567
Pit 49	582,192	673,223	37,469	635,754	16.967
Pit 50	1,175	3,896	784	3,112	3.967
Pit 51	138,430	232,114	42,989	189,125	4.399
Pit 52	1,934	5,192	372	4,820	12.952
Pit 53	14,032	38,924	6,529	32,395	4.962
Pit 54	4,850	12,703	2,757	9,947	3.608
Pit 55	263,317	581,960	55,505	526,455	9.485
Pit 56	5,435	15,609	1,076	14,533	13.505
Pit 57	18,412	52,428	9,768	42,660	4.368
Pit 58	70,151	196,637	15,609	181,028	11.598
Pit 59	17,116	75,206	10,617	64,590	6.084
Pit 60	32,846	130,710	24,004	106,706	4.445
Pit 61	27,105	103,523	9,968	93,554	9.385
Pit 62	35,785	251,950	30,683	221,267	7.211
Pit 63	98,263	631,529	23,491	608,037	25.884
Pit 64	26,563	243,677	20,467	223,211	10.906
Pit 65	366.4708	6,304	873	5,431	6.224
Pit 66	393,832	23,389,930	2,256,217	21,133,714	9.367
Pit 67	4,230	83,435	4,528	78,907	17.427
Pit 68	50,037	3,634,950	295,687	3,339,263	11.293
TOTAL	**335,570,199**	**55,844,000**	**12,916,266**	**42,927,735**	**3.324**

Table 16-9 – Jugan: Incremental Pit Shell Data (Owner-Operator)

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 1	161,730,427	5,411,960	4,241,697	1,170,263	0.276
Pit 2	183,186,659	6,361,287	4,802,144	1,559,143	0.325
Pit 3	196,828,637	7,012,460	5,168,999	1,843,461	0.357
Pit 4	215,270,210	7,968,822	5,646,960	2,321,862	0.411
Pit 5	233,604,726	9,106,012	6,188,281	2,917,732	0.472
Pit 6	240,236,390	9,576,230	6,382,025	3,194,206	0.501
Pit 7	253,622,680	10,606,407	6,792,347	3,814,061	0.562
Pit 8	259,430,845	11,098,557	6,978,915	4,119,643	0.590
Pit 9	270,297,456	12,084,489	7,326,493	4,757,996	0.649
Pit 10	277,297,347	12,728,876	7,525,250	5,203,627	0.692
Pit 11	284,495,392	13,476,453	7,760,900	5,715,553	0.737
Pit 12	286,470,899	13,727,658	7,841,435	5,886,224	0.751

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 13	292,937,132	14,493,095	8,043,445	6,449,650	0.802
Pit 14	293,259,513	14,537,505	8,059,255	6,478,250	0.804
Pit 15	300,363,599	15,604,551	8,348,440	7,256,111	0.869
Pit 16	300,964,915	15,695,116	8,368,336	7,326,780	0.876
Pit 17	301,344,887	15,784,886	8,387,952	7,396,935	0.882
Pit 18	305,905,487	16,498,102	8,547,209	7,950,894	0.930
Pit 19	306,450,509	16,593,926	8,572,604	8,021,323	0.936
Pit 20	312,258,286	17,764,092	8,816,343	8,947,750	1.015
Pit 21	312,976,319	17,932,103	8,856,737	9,075,367	1.025
Pit 22	315,033,308	18,372,606	8,956,187	9,416,420	1.051
Pit 23	315,537,778	18,488,330	8,976,300	9,512,030	1.060
Pit 24	316,232,204	18,641,322	9,003,682	9,637,641	1.070
Pit 25	318,538,698	19,212,501	9,067,739	10,144,763	1.119
Pit 26	320,332,475	19,712,714	9,177,670	10,535,045	1.148
Pit 27	321,282,523	20,009,486	9,225,491	10,783,996	1.169
Pit 28	321,700,973	20,171,641	9,265,032	10,906,609	1.177
Pit 29	325,341,318	21,253,617	9,446,806	11,806,812	1.250
Pit 30	325,881,760	21,447,751	9,478,603	11,969,148	1.263
Pit 31	326,181,839	21,554,709	9,495,829	12,058,881	1.270
Pit 32	326,702,991	21,780,071	9,550,117	12,229,954	1.281
Pit 33	329,090,915	22,570,405	9,647,467	12,922,939	1.340
Pit 34	329,405,603	22,741,001	9,680,147	13,060,855	1.349
Pit 35	329,722,894	22,868,507	9,699,534	13,168,974	1.358
Pit 36	329,775,153	22,899,928	9,707,105	13,192,823	1.359
Pit 37	330,357,131	23,247,384	9,776,101	13,471,283	1.378
Pit 38	331,129,847	23,664,888	9,839,355	13,825,534	1.405
Pit 39	331,546,437	23,949,286	9,896,236	14,053,050	1.420
Pit 40	332,938,705	24,705,428	9,950,281	14,755,147	1.483
Pit 41	333,045,954	24,800,090	9,958,128	14,841,962	1.490
Pit 42	333,464,048	25,127,427	10,012,936	15,114,492	1.510
Pit 43	333,581,823	25,242,161	10,033,377	15,208,785	1.516
Pit 44	333,583,850	25,246,129	10,033,965	15,212,164	1.516
Pit 45	333,613,417	25,269,450	10,035,871	15,233,580	1.518
Pit 46	333,737,355	25,410,557	10,057,878	15,352,679	1.526
Pit 47	333,757,473	25,431,392	10,060,437	15,370,956	1.528
Pit 48	333,784,126	25,480,100	10,066,874	15,413,227	1.531
Pit 49	334,366,318	26,153,323	10,104,343	16,048,980	1.588
Pit 50	334,367,494	26,157,219	10,105,127	16,052,092	1.589
Pit 51	334,505,924	26,389,333	10,148,116	16,241,217	1.600
Pit 52	334,507,858	26,394,524	10,148,488	16,246,036	1.601
Pit 53	334,521,890	26,433,448	10,155,017	16,278,431	1.603
Pit 54	334,526,740	26,446,151	10,157,774	16,288,378	1.604
Pit 55	334,790,057	27,028,111	10,213,279	16,814,833	1.646

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 56	334,795,492	27,043,720	10,214,355	16,829,366	1.648
Pit 57	334,813,904	27,096,148	10,224,122	16,872,026	1.650
Pit 58	334,884,056	27,292,785	10,239,731	17,053,054	1.665
Pit 59	334,901,172	27,367,991	10,250,348	17,117,644	1.670
Pit 60	334,934,018	27,498,701	10,274,352	17,224,350	1.676
Pit 61	334,961,123	27,602,224	10,284,321	17,317,905	1.684
Pit 62	334,996,908	27,854,174	10,315,004	17,539,171	1.700
Pit 63	335,095,171	28,485,703	10,338,495	18,147,209	1.755
Pit 64	335,121,733	28,729,381	10,358,962	18,370,419	1.773
Pit 65	335,122,100	28,735,684	10,359,835	18,375,850	1.774
Pit 66	335,515,932	52,125,615	12,616,051	39,509,564	3.132
Pit 67	335,520,162	52,209,050	12,620,579	39,588,471	3.137
Pit 68	335,570,199	55,844,000	12,916,266	42,927,735	3.324

Table 16-10 – Jugan: Cumulative Pit Shell Data (Owner-Operator)

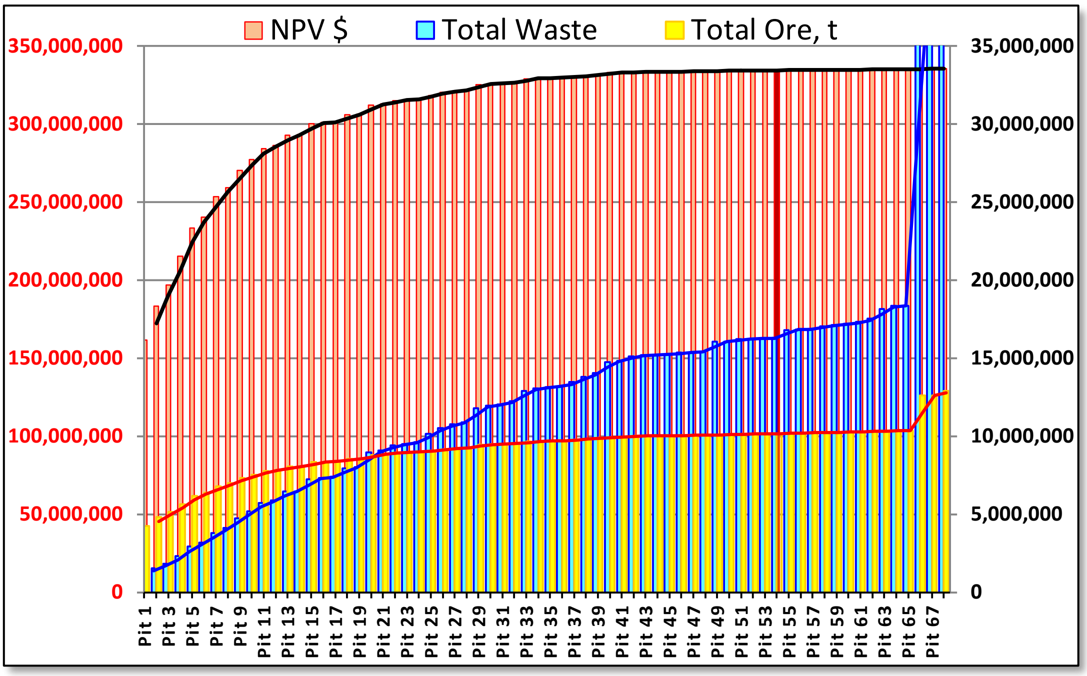


Figure 16-8 - Jugan: Graph of Cumulative Ore & Waste Tonnes and NPV (Owner-Operator)

To show the impact graphically and in terms of the 3D pit the optimal pit shell (Pit_Shell_54) and the ultimate pit shell (Pit_shell_68) are shown side by side. *Figure 16-9 - Jugan: 3D View Comparison Between Optimal & Ultimate Pits (Owner-Operator)* shows the two pits below and it can be seen that there are very marginal differences between each.

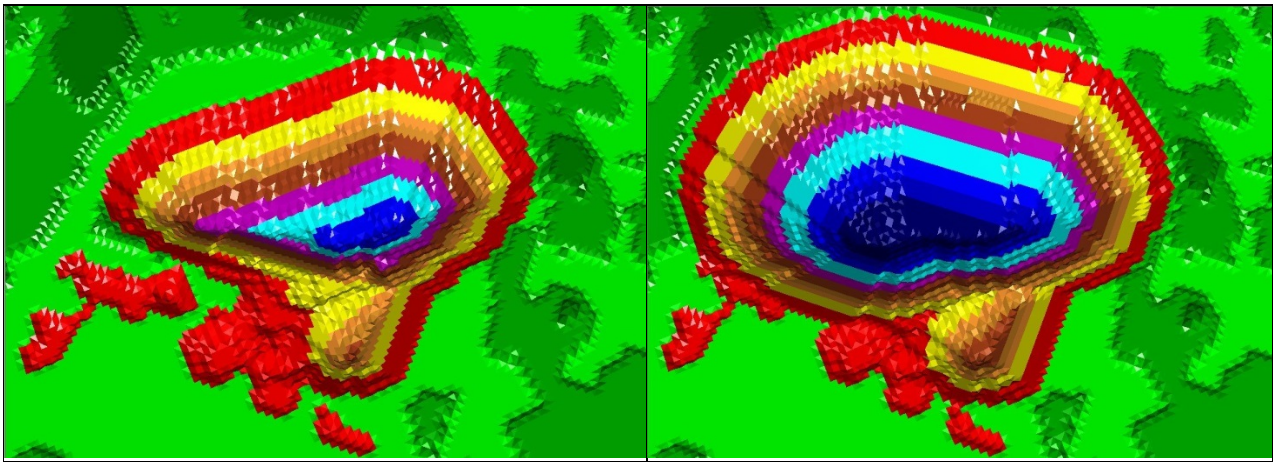


Figure 16-9 - Jugan: 3D View Comparison Between Optimal & Ultimate Pits (Owner-Operator)

The selected optimal pits (owner-operator) for all options with corresponding tonnage, grade and strip ratio are shown in *Table 16-11 - Jugan Owner-Operator: Selected Pit Shell Tabulations – All Options* below.

Selected Optimal Pit Description	Selected Pit Shell	Tonnage (t)	Grade (g/t)	Strip Ratio
Owner-Operator_4000 TPD_Flotation	Pit 52	9,951,930	1.557	1.482
Owner-Operator_6000 TPD_Flotation	Pit 55	10,055,790	1.552	1.525
Owner-Operator_8000 TPD_Flotation (base case)	Pit 54	10,157,770	1.549	1.604
Owner-Operator_10000 TPD_Flotation	Pit 54	10,216,610	1.547	1.647
Owner-Operator_12000 TPD_Flotation	Pit 61	12,618,900	1.446	3.132
Owner-Operator_4000 TPD_POX	Pit 56	8,476,360	1.707	1.605
Owner-Operator_6000 TPD_POX	Pit 53	8,517,820	1.704	1.614
Owner-Operator_8000 TPD_POX	Pit 56	8,701,010	1.698	1.739
Owner-Operator_10000 TPD_POX	Pit 54	8,725,420	1.696	1.748
Owner-Operator_12000 TPD_POX	Pit 54	8,732,590	1.695	1.751
Owner-Operator_4000 TPD_BIOX	Pit 54	7,970,700	1.759	1.674
Owner-Operator_6000 TPD_BIOX	Pit 55	7,993,200	1.759	1.692
Owner-Operator_000 TPD_BIOX	Pit 59	8,150,750	1.755	1.836
Owner-Operator_10000 TPD_BIOX	Pit 62	8,156,920	1.755	1.839
Owner-Operator_12000 TPD_BIOX	Pit 57	8,162,360	1.755	1.842
Owner-Operator_4000 TPD_ALBION	Pit 59	6,506,470	1.905	1.842
Owner-Operator_6000 TPD_ALBION	Pit 57	6,550,170	1.904	1.873
Owner-Operator_8000 TPD_ALBION	Pit 56	6,686,130	1.903	2.030
Owner-Operator_10000 TPD_ALBION	Pit 58	6,696,250	1.902	2.030
Owner-Operator_12000 TPD_ALBION	Pit 57	6,697,480	1.902	2.030

Table 16-11 - Jugan Owner-Operator: Selected Pit Shell Tabulations – All Options

In the base case (contract-mining) the pit shell selected is Pit_Shell_52. At Pit_Shell_52, the cumulative NPV has almost reached the peak with no significant incremental increase or incremental NPV higher than this pit shell. For Pit_Shell_52 the higher incremental waste is off-set by the higher incremental ore and large NPV amount. If there had been a significant increase in waste without the corresponding increase in ore and NPV, then the previous pit shell would have been selected (Pit_shell_51). Pit_Shell_52 is the last pit which has the higher Au ounces compared to the remaining pits.

The pit shell selection (contract-mining) is also demonstrated in *Table 16-12 - Jugan: Incremental Pit Shell Data (Contract-Mining)* and *Table 16-13 - Jugan: Cumulative Pit Shell Data (Contract-Mining),* and also shown in the graph in *Figure 16-10 - Jugan: Graph of Cumulative Ore & Waste Tonnes and NPV (Contract-Mining)* below.

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 1	145,368,655	4,752,939	3,847,216	905,723	0.235
Pit 2	22,000,885	1,074,764	663,713	411,052	0.619
Pit 3	14,114,326	635,834	377,105	258,729	0.686
Pit 4	10,077,840	507,749	276,370	231,379	0.837
Pit 5	17,543,267	943,672	470,490	473,182	1.006
Pit 6	9,471,968	583,834	281,637	302,197	1.073
Pit 7	8,350,897	555,628	266,239	289,389	1.087
Pit 8	6,180,498	449,515	185,833	263,682	1.419
Pit 9	10,146,127	766,844	309,726	457,118	1.476
Pit 10	5,860,488	528,608	215,038	313,570	1.458
Pit 11	6,779,279	594,050	201,194	392,855	1.953
Pit 12	4,225,984	475,313	169,025	306,289	1.812
Pit 13	6,102,192	589,908	187,056	402,852	2.154
Pit 14	8,002,186	830,666	258,055	572,611	2.219
Pit 15	570,208	74,505	24,453	50,053	2.047
Pit 16	2,048,920	312,225	102,247	209,978	2.054
Pit 17	752,796	97,222	25,585	71,637	2.800
Pit 18	5,216,453	656,253	174,671	481,581	2.757
Pit 19	294,641	44,410	15,810	28,600	1.809
Pit 20	6,761,210	1,090,594	295,840	794,755	2.686
Pit 21	201,637	43,323	7,774	35,548	4.573
Pit 22	227,435	40,719	11,472	29,247	2.550
Pit 23	4,175,987	721,008	163,035	557,974	3.422
Pit 24	63,155	15,674	3,206	12,468	3.889
Pit 25	259,726	49,337	13,714	35,623	2.598
Pit 26	245,880	56,920	13,590	43,330	3.188
Pit 27	4,498,585	939,540	184,530	755,009	4.092
Pit 28	860,149	267,925	72,954	194,971	2.673
Pit 29	353,485	131,160	28,232	102,929	3.646
Pit 30	1,820,160	488,656	104,607	384,049	3.671

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 31	344,078	79,741	15,500	64,242	4.145
Pit 32	2,425,575	578,688	78,598	500,090	6.363
Pit 33	12,648	15,252	1,290	13,962	10.823
Pit 34	958,995	298,359	62,608	235,751	3.766
Pit 35	789,704	267,053	54,097	212,957	3.937
Pit 36	635,968	329,864	42,570	287,294	6.749
Pit 37	133,601	80,571	15,369	65,202	4.242
Pit 38	2,924,409	1,096,452	196,743	899,708	4.573
Pit 39	429,028	145,946	23,244	122,701	5.279
Pit 40	330,716	123,742	19,816	103,927	5.245
Pit 41	153,333	90,741	12,698	78,043	6.146
Pit 42	2,172,166	915,235	128,111	787,123	6.144
Pit 43	217,929	117,664	27,464	90,199	3.284
Pit 44	399,970	220,284	32,581	187,703	5.761
Pit 45	49,504	31,434	5,337	26,097	4.890
Pit 46	23,141	33,597	6,296	27,301	4.336
Pit 47	67,995	81,307	20,812	60,495	2.907
Pit 48	352,603	282,135	58,893	223,242	3.791
Pit 49	519,030	407,027	60,534	346,493	5.724
Pit 50	78,233	91,582	18,808	72,775	3.869
Pit 51	135,983	130,737	25,811	104,926	4.065
Pit 52	1,007,381	789,996	62,902	727,094	11.559
Pit 53	13,758	12,978	2,578	10,400	4.034
Pit 54	49,705	75,809	16,096	59,712	3.710
Pit 55	42,613	52,236	7,770	44,466	5.723
Pit 56	10,482	25,821	3,257	22,564	6.927
Pit 57	109,024	136,118	16,791	119,327	7.107
Pit 58	134,297	233,676	39,842	193,834	4.865
Pit 59	16,393	23,321	1,906	21,416	11.237
Pit 60	24,612	46,882	9,308	37,575	4.037
Pit 61	27,176	67,851	11,132	56,719	5.095
Pit 62	26,020	59,810	5,662	54,147	9.562
Pit 63	227,065	788,284	54,901	733,384	13.358
Pit 64	3,667	28,648	5,198	23,450	4.511
Pit 65	6,304	83,814	19,192	64,622	3.367

Table 16-12 - Jugan: Incremental Pit Shell Data (Contract-Mining)

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 1	145,368,655	4,752,939	3,847,216	905,723	0.235
Pit 2	167,369,540	5,827,703	4,510,929	1,316,775	0.292
Pit 3	181,483,866	6,463,537	4,888,033	1,575,504	0.322
Pit 4	191,561,706	6,971,286	5,164,404	1,806,883	0.350

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 5	209,104,973	7,914,958	5,634,893	2,280,065	0.405
Pit 6	218,576,941	8,498,792	5,916,531	2,582,261	0.436
Pit 7	226,927,838	9,054,419	6,182,770	2,871,650	0.465
Pit 8	233,108,336	9,503,934	6,368,603	3,135,332	0.492
Pit 9	243,254,463	10,270,778	6,678,329	3,592,450	0.538
Pit 10	249,114,951	10,799,386	6,893,367	3,906,020	0.567
Pit 11	255,894,230	11,393,436	7,094,561	4,298,876	0.606
Pit 12	260,120,214	11,868,749	7,263,585	4,605,164	0.634
Pit 13	266,222,406	12,458,657	7,450,641	5,008,016	0.672
Pit 14	274,224,591	13,289,323	7,708,696	5,580,627	0.724
Pit 15	274,794,800	13,363,828	7,733,149	5,630,680	0.728
Pit 16	276,843,720	13,676,053	7,835,396	5,840,658	0.745
Pit 17	277,596,515	13,773,275	7,860,981	5,912,294	0.752
Pit 18	282,812,969	14,429,528	8,035,653	6,393,875	0.796
Pit 19	283,107,609	14,473,938	8,051,463	6,422,475	0.798
Pit 20	289,868,819	15,564,532	8,347,303	7,217,230	0.865
Pit 21	290,070,456	15,607,855	8,355,077	7,252,778	0.868
Pit 22	290,297,891	15,648,574	8,366,549	7,282,025	0.870
Pit 23	294,473,878	16,369,582	8,529,584	7,839,999	0.919
Pit 24	294,537,033	16,385,256	8,532,789	7,852,467	0.920
Pit 25	294,796,758	16,434,593	8,546,503	7,888,090	0.923
Pit 26	295,042,639	16,491,513	8,560,093	7,931,420	0.927
Pit 27	299,541,224	17,431,053	8,744,624	8,686,430	0.993
Pit 28	300,401,373	17,698,978	8,817,578	8,881,401	1.007
Pit 29	300,754,857	17,830,138	8,845,809	8,984,330	1.016
Pit 30	302,575,017	18,318,794	8,950,416	9,368,379	1.047
Pit 31	302,919,095	18,398,535	8,965,916	9,432,620	1.052
Pit 32	305,344,670	18,977,223	9,044,514	9,932,710	1.098
Pit 33	305,357,319	18,992,476	9,045,804	9,946,672	1.100
Pit 34	306,316,314	19,290,835	9,108,412	10,182,424	1.118
Pit 35	307,106,018	19,557,888	9,162,508	10,395,380	1.135
Pit 36	307,741,986	19,887,752	9,205,078	10,682,674	1.161
Pit 37	307,875,587	19,968,322	9,220,447	10,747,876	1.166
Pit 38	310,799,996	21,064,774	9,417,191	11,647,584	1.237
Pit 39	311,229,024	21,210,720	9,440,435	11,770,286	1.247
Pit 40	311,559,739	21,334,462	9,460,250	11,874,212	1.255
Pit 41	311,713,072	21,425,203	9,472,948	11,952,255	1.262
Pit 42	313,885,237	22,340,438	9,601,059	12,739,379	1.327
Pit 43	314,103,166	22,458,101	9,628,524	12,829,578	1.333
Pit 44	314,503,137	22,678,385	9,661,105	13,017,281	1.347
Pit 45	314,552,640	22,709,819	9,666,442	13,043,378	1.349
Pit 46	314,575,781	22,743,416	9,672,738	13,070,678	1.351
Pit 47	314,643,776	22,824,724	9,693,551	13,131,173	1.355

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 48	314,996,379	23,106,859	9,752,444	13,354,416	1.369
Pit 49	315,515,409	23,513,886	9,812,977	13,700,909	1.396
Pit 50	315,593,642	23,605,468	9,831,785	13,773,684	1.401
Pit 51	315,729,625	23,736,205	9,857,595	13,878,610	1.408
Pit 52	316,737,006	24,526,201	9,920,497	14,605,704	1.472
Pit 53	316,750,764	24,539,179	9,923,076	14,616,104	1.473
Pit 54	316,800,469	24,614,988	9,939,172	14,675,816	1.477
Pit 55	316,843,082	24,667,223	9,946,942	14,720,282	1.480
Pit 56	316,853,563	24,693,044	9,950,199	14,742,846	1.482
Pit 57	316,962,587	24,829,163	9,966,991	14,862,173	1.491
Pit 58	317,096,884	25,062,838	10,006,833	15,056,006	1.505
Pit 59	317,113,276	25,086,160	10,008,738	15,077,422	1.506
Pit 60	317,137,889	25,133,042	10,018,046	15,114,997	1.509
Pit 61	317,165,065	25,200,893	10,029,179	15,171,715	1.513
Pit 62	317,191,085	25,260,703	10,034,841	15,225,862	1.517
Pit 63	317,418,150	26,048,987	10,089,742	15,959,246	1.582
Pit 64	317,421,818	26,077,635	10,094,940	15,982,696	1.583
Pit 65	317,428,122	26,161,449	10,114,132	16,047,318	1.587

Table 16-13 - Jugan: Cumulative Pit Shell Data (Contract-Mining)

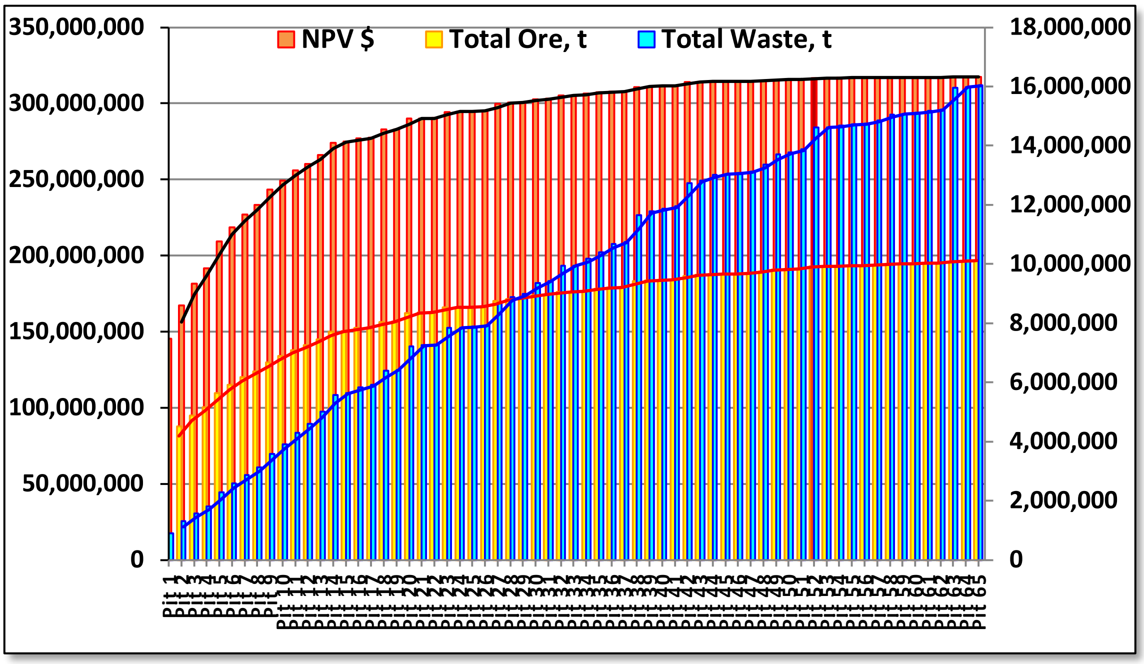


Figure 16-10 - Jugan: Graph of Cumulative Ore & Waste Tonnes and NPV (Contract-Mining)

To show the impact graphically and in terms of the 3D pit the optimal pit shell (Pit_Shell_65) and the ultimate pit shell (Pit_shell_52) are shown side by side. *Figure 16-11 - Jugan: 3D View Comparison between Optimal & Ultimate Pits (Contract-Mining)* shows the two pits below and it can be seen that there are very marginal differences between each.

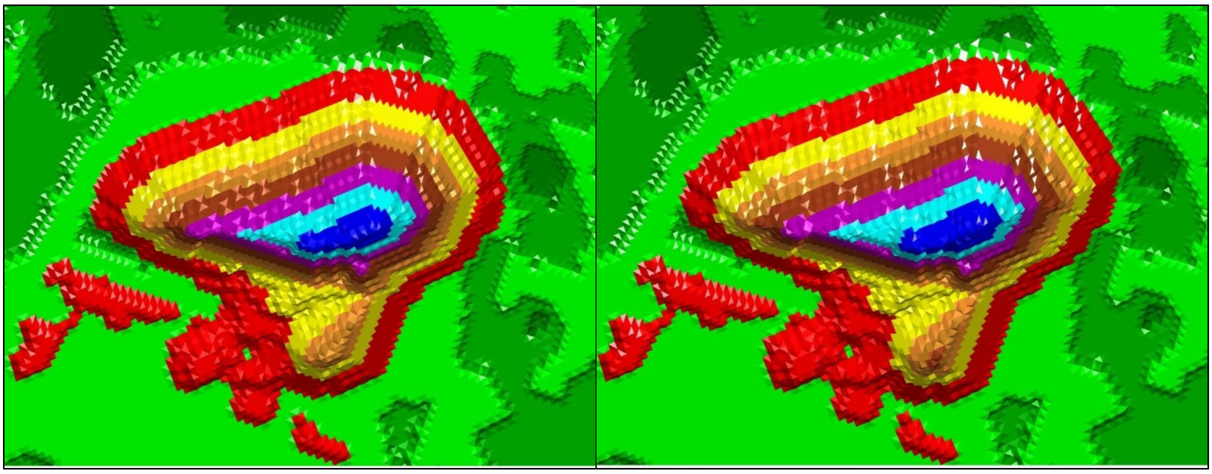


Figure 16-11 - Jugan: 3D View Comparison between Optimal & Ultimate Pits (Contract-Mining)

The selected optimal pits (contract-mining) for all options with corresponding tonnage, grade and strip ratio are shown in *Table 16-14 - Jugan Contract-Mining: Selected Pit Shell Tabulations – All Options* below.

Selected Optimal Pit Description	Selected Pit Shell	Tonnage (t)	Grade (g/t)	Strip Ratio
Contract-Mining_4000 TPD_Flotation	Pit 61	9,913,621	1.560	1.470
Contract-Mining_6000 TPD_Flotation	Pit 53	9,917,605	1.560	1.470
Contract-Mining_8000 TPD_Flotation (base case)	Pit 52	9,920,498	1.559	1.472
Contract-Mining_10000 TPD_Flotation	Pit 61	10,092,966	1.554	1.582
Contract-Mining_12000 TPD_Flotation	Pit 59	10,094,006	1.554	1.582
Contract-Mining_4000 TPD_POX	Pit 55	8,189,576	1.715	1.433
Contract-Mining_6000 TPD_POX	Pit 63	8,396,940	1.711	1.568
Contract-Mining_8000 TPD_POX	Pit 59	8,432,274	1.709	1.588
Contract-Mining_10000 TPD_POX	Pit 60	8,472,537	1.707	1.604
Contract-Mining_12000 TPD_POX	Pit 58	8,475,565	1.707	1.605
Contract-Mining_4000 TPD_BIOX	Pit 53	7,727,498	1.763	1.491
Contract-Mining_6000 TPD_BIOX	Pit 54	7,740,633	1.762	1.493
Contract-Mining_000 TPD_BIOX	Pit 59	7,912,835	1.763	1.656
Contract-Mining_10000 TPD_BIOX	Pit 57	7,943,170	1.761	1.663
Contract-Mining_12000 TPD_BIOX	Pit 59	7,944,160	1.761	1.664
Contract-Mining_4000 TPD_ALBION	Pit 65	6,178,660	1.909	1.610
Contract-Mining_6000 TPD_ALBION	Pit 62	6,331,948	1.907	1.715
Contract-Mining_8000 TPD_ALBION	Pit 64	6,430,553	1.910	1.827
Contract-Mining_10000 TPD_ALBION	Pit 64	6,505,264	1.905	1.842
Contract-Mining_12000 TPD_ALBION	Pit 60	6,505,264	1.905	1.842

Table 16-14 - Jugan Contract-Mining: Selected Pit Shell Tabulations – All Options

A full summary of optimal pit shell results for Jugan are listed in *Appendix A17-3*.

16.2.4.2. Optimal Pit & Design – BYG-Krian

In the case of BYG-Krian the optimal pits for the basecase owner-operator option is Pit_Shell_56, and for the basecase contract-mining option is Pit_Shell_50.

For the contract-mining option this pit shell is selected as it is the last pit that has the highest incremental values compared to the other remaining pits; and is almost the ultimate pit with the highwall at the West side already reached, and no more resource area for the pit to expand other than small incremental trimming of the West highwall.

For the owner-operator option this pit shell is the last pit with the highest NPV values compared to the remaining pits; and like the contract-mining option, is almost the ultimate pit with the highwall at the West side already reached, and no more resource area for the pit to expand other than small incremental trimming of the West highwall.

Table 16-15 - BYG-Krian: Incremental Pit Shell Data (Owner-Operator) to *Table 16-19 - BYG-Krian: Cumulative Pit Shell Data (Contract-Mining)* show the BYG-Krian pit shells with their associated information – for both owner-operator and contract-mining, and incremental and cumulative respectively. And the ultimate and optimal pits are shown in *Figure 16-13 - BYG-Krian: 3D View Comparison between Optimal & Ultimate Pits (Owner-Operator)* with the graph of the waste and ore tonnes plus the NPV is shown in *Figure 16-12 - BYG-Krian: Graph of Cumulative Ore & Waste Tonnes and NPV (Owner-Operator)* and *Figure 16-14 - BYG-Krian: Graph of Cumulative Ore & Waste Tonnes and NPV (Contract-Mining)*.

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 1	47,094,725	1,358,635	545,012	813,624	1.493
Pit 2	7,145,273	172,462	39,884	132,578	3.324
Pit 3	9,618,253	337,671	77,000	260,671	3.385
Pit 4	1,254,337	91,622	16,377	75,244	4.594
Pit 5	2,333,470	143,948	31,768	112,180	3.531
Pit 6	678,285	37,288	5,074	32,214	6.349
Pit 7	1,720,149	130,250	23,482	106,768	4.547
Pit 8	272,817	41,572	8,523	33,049	3.878
Pit 9	623,122	69,537	9,980	59,556	5.967
Pit 10	2,333,905	203,535	24,819	178,716	7.201
Pit 11	3,303,201	353,821	42,714	311,107	7.284
Pit 12	493,963	72,354	12,318	60,036	4.874
Pit 13	1,142,866	110,866	12,044	98,821	8.205
Pit 14	159,018	26,473	3,914	22,559	5.764
Pit 15	682,410	92,191	11,325	80,865	7.140
Pit 16	461,290	54,664	5,335	49,329	9.247
Pit 17	223,960	27,038	2,537	24,502	9.659
Pit 18	768,383	123,584	10,571	113,013	10.691
Pit 19	483,994	91,603	7,393	84,209	11.390
Pit 20	385,285	92,014	10,390	81,623	7.856

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 21	36,167	7,700	1,638	6,062	3.701
Pit 22	655,337	126,831	8,611	118,220	13.729
Pit 23	1,068,156	223,624	26,648	196,977	7.392
Pit 24	357,402	102,452	11,800	90,652	7.682
Pit 25	79,378	17,910	1,543	16,367	10.606
Pit 26	112,633	26,858	2,086	24,772	11.876
Pit 27	293,569	58,615	2,721	55,894	20.540
Pit 28	186,635	57,477	3,196	54,281	16.985
Pit 29	268,354	73,797	6,465	67,332	10.415
Pit 30	6,828	2,652	414	2,237	5.401
Pit 31	232,469	72,140	6,862	65,278	9.513
Pit 32	20,139	8,209	1,037	7,172	6.917
Pit 33	709,019	241,833	15,465	226,368	14.638
Pit 34	7,242	3,017	384	2,633	6.852
Pit 35	38,403	23,950	2,056	21,894	10.649
Pit 36	117,958	55,830	4,153	51,677	12.443
Pit 37	6,551	3,322	531	2,790	5.251
Pit 38	10,740	7,200	938	6,262	6.675
Pit 39	83,093	40,747	2,573	38,174	14.838
Pit 40	13,315	11,339	1,502	9,837	6.550
Pit 41	401,734	235,965	13,374	222,591	16.643
Pit 42	15,821	17,262	1,738	15,524	8.935
Pit 43	13,965	23,707	2,461	21,246	8.633
Pit 44	69,955	67,965	5,679	62,285	10.967
Pit 45	18,405	16,685	1,716	14,969	8.722
Pit 46	40,526	62,108	5,064	57,045	11.265
Pit 47	103,267	92,214	3,699	88,515	23.929
Pit 48	10,761	9,941	1,104	8,837	8.002
Pit 49	1,769	6,414	1,352	5,062	3.745
Pit 50	5,203	6,662	220	6,442	29.270
Pit 51	5,102	7,099	112	6,986	62.143
Pit 52	33,030	43,182	4,488	38,694	8.622
Pit 53	136,819	198,812	4,959	193,853	39.088
Pit 54	30,038	45,697	1,313	44,383	33.791
Pit 55	9,535	13,885	1,743	12,142	6.965
Pit 56	27,494	44,985	1,204	43,781	36.371
Pit 57	5,567	13,317	318	12,999	40.907
Pit 58	2,820	13,011	1,783	11,228	6.298
Pit 59	4,924	33,364	2,020	31,344	15.516
Pit 60	1,163	3,996	127	3,868	30.447
Pit 61	396	5,280	220	5,060	22.977
Pit 62	1,799	7,978	209	7,770	37.250
Pit 63	8,581	62,077	3,496	58,581	16.755

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 64	1,936	14,622	256	14,367	56.157
Pit 65	133	24,959	453	24,506	54.070

Table 16-15 - BYG-Krian: Incremental Pit Shell Data (Owner-Operator)

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 1	47,094,725	1,358,635	545,012	813,624	1.493
Pit 2	54,239,998	1,531,097	584,896	946,201	1.618
Pit 3	63,858,252	1,868,768	661,895	1,206,873	1.823
Pit 4	65,112,589	1,960,390	678,273	1,282,117	1.890
Pit 5	67,446,058	2,104,338	710,040	1,394,297	1.964
Pit 6	68,124,343	2,141,625	715,114	1,426,511	1.995
Pit 7	69,844,492	2,271,876	738,596	1,533,280	2.076
Pit 8	70,117,309	2,313,448	747,119	1,566,329	2.097
Pit 9	70,740,431	2,382,984	757,099	1,625,885	2.148
Pit 10	73,074,336	2,586,519	781,918	1,804,601	2.308
Pit 11	76,377,537	2,940,340	824,632	2,115,708	2.566
Pit 12	76,871,500	3,012,694	836,950	2,175,744	2.600
Pit 13	78,014,366	3,123,560	848,995	2,274,565	2.679
Pit 14	78,173,384	3,150,033	852,909	2,297,124	2.693
Pit 15	78,855,794	3,242,224	864,234	2,377,989	2.752
Pit 16	79,317,085	3,296,887	869,569	2,427,318	2.791
Pit 17	79,541,045	3,323,926	872,106	2,451,820	2.811
Pit 18	80,309,428	3,447,510	882,677	2,564,834	2.906
Pit 19	80,793,422	3,539,113	890,070	2,649,043	2.976
Pit 20	81,178,707	3,631,127	900,460	2,730,666	3.033
Pit 21	81,214,874	3,638,827	902,098	2,736,728	3.034
Pit 22	81,870,211	3,765,657	910,710	2,854,948	3.135
Pit 23	82,938,366	3,989,282	937,357	3,051,925	3.256
Pit 24	83,295,768	4,091,734	949,157	3,142,577	3.311
Pit 25	83,375,146	4,109,644	950,700	3,158,944	3.323
Pit 26	83,487,779	4,136,502	952,786	3,183,716	3.342
Pit 27	83,781,348	4,195,117	955,507	3,239,610	3.391
Pit 28	83,967,984	4,252,594	958,703	3,293,891	3.436
Pit 29	84,236,337	4,326,391	965,168	3,361,223	3.483
Pit 30	84,243,166	4,329,043	965,583	3,363,460	3.483
Pit 31	84,475,635	4,401,183	972,445	3,428,738	3.526
Pit 32	84,495,773	4,409,392	973,482	3,435,910	3.530
Pit 33	85,204,793	4,651,225	988,947	3,662,278	3.703
Pit 34	85,212,035	4,654,242	989,331	3,664,911	3.704
Pit 35	85,250,438	4,678,193	991,387	3,686,806	3.719
Pit 36	85,368,396	4,734,023	995,540	3,738,483	3.755
Pit 37	85,374,947	4,737,345	996,071	3,741,273	3.756

Phase	NPV	Rock	Total Ore	Total Waste	Strip
	($)	(t)	(t)	(t)	Ratio
Pit 38	85,385,687	4,744,544	997,010	3,747,535	3.759
Pit 39	85,468,780	4,785,291	999,582	3,785,709	3.787
Pit 40	85,482,095	4,796,630	1,001,084	3,795,546	3.791
Pit 41	85,883,829	5,032,595	1,014,459	4,018,137	3.961
Pit 42	85,899,650	5,049,857	1,016,196	4,033,661	3.969
Pit 43	85,913,615	5,073,563	1,018,657	4,054,907	3.981
Pit 44	85,983,570	5,141,528	1,024,336	4,117,192	4.019
Pit 45	86,001,975	5,158,213	1,026,053	4,132,161	4.027
Pit 46	86,042,501	5,220,322	1,031,116	4,189,205	4.063
Pit 47	86,145,767	5,312,535	1,034,815	4,277,720	4.134
Pit 48	86,156,528	5,322,477	1,035,920	4,286,557	4.138
Pit 49	86,158,298	5,328,891	1,037,272	4,291,619	4.137
Pit 50	86,163,501	5,335,553	1,037,492	4,298,061	4.143
Pit 51	86,168,603	5,342,651	1,037,604	4,305,047	4.149
Pit 52	86,201,633	5,385,833	1,042,092	4,343,741	4.168
Pit 53	86,338,452	5,584,645	1,047,051	4,537,594	4.334
Pit 54	86,368,490	5,630,342	1,048,365	4,581,977	4.371
Pit 55	86,378,025	5,644,227	1,050,108	4,594,119	4.375
Pit 56	86,405,519	5,689,211	1,051,312	4,637,900	4.412
Pit 57	86,411,086	5,702,528	1,051,629	4,650,899	4.423
Pit 58	86,413,906	5,715,539	1,053,412	4,662,127	4.426
Pit 59	86,418,830	5,748,903	1,055,432	4,693,471	4.447
Pit 60	86,419,993	5,752,899	1,055,559	4,697,339	4.450
Pit 61	86,420,389	5,758,179	1,055,780	4,702,400	4.454
Pit 62	86,422,188	5,766,158	1,055,988	4,710,169	4.460
Pit 63	86,430,769	5,828,235	1,059,484	4,768,750	4.501
Pit 64	86,432,705	5,842,857	1,059,740	4,783,117	4.514
Pit 65	86,432,838	5,867,816	1,060,194	4,807,622	4.535

Table 16-16 - BYG-Krian: Cumulative Pit Shell Data (Owner-Operator)

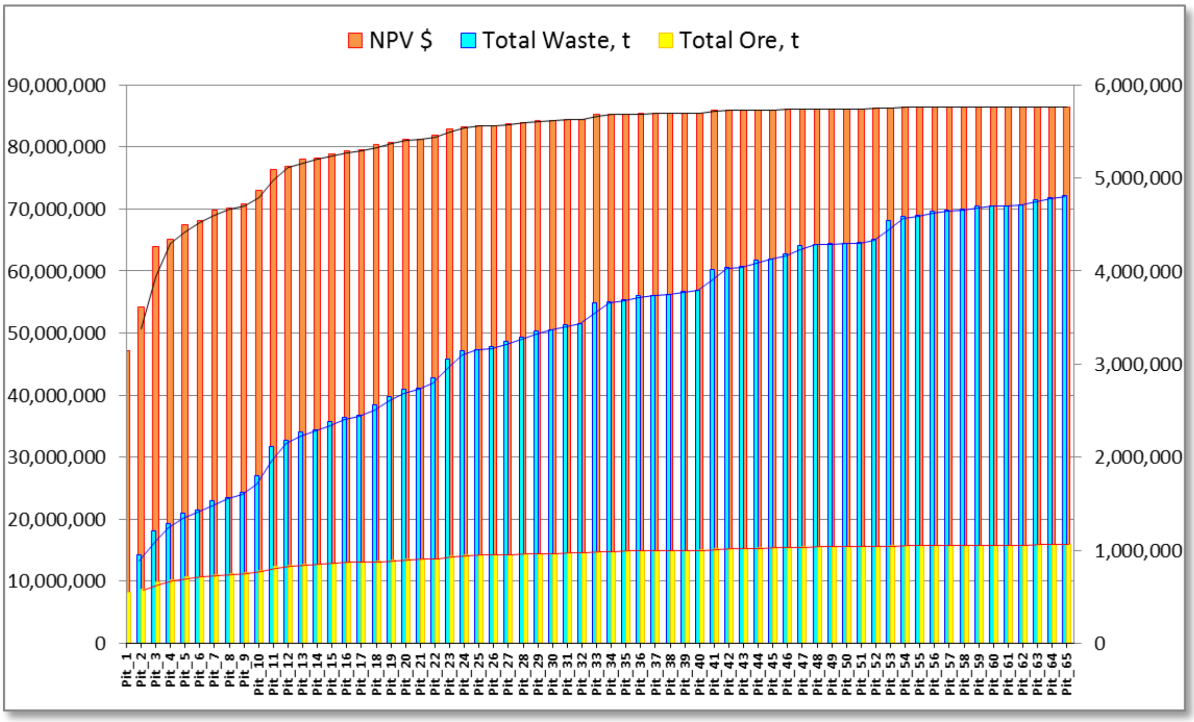


Figure 16-12 - BYG-Krian: Graph of Cumulative Ore & Waste Tonnes and NPV (Owner-Operator)

To show the impact graphically and in terms of the 3D pit the optimal pit shell (Pit_Shell_65) and the ultimate pit shell (Pit_shell_56) are shown side by side. *Figure 16-13 - BYG-Krian: 3D View Comparison between Optimal & Ultimate Pits (Owner-Operator)* shows the two pits below and it can be seen that there are very marginal differences between each.

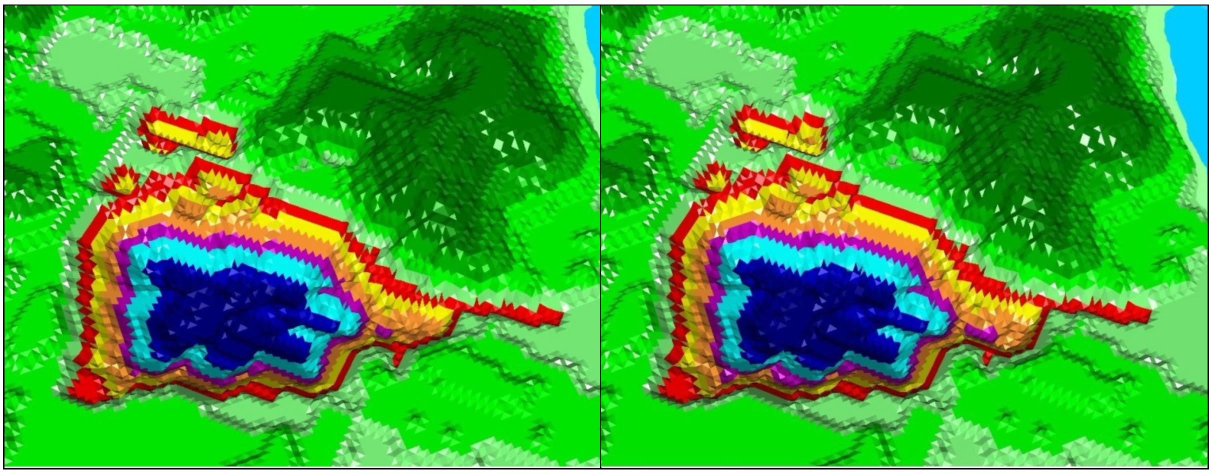


Figure 16-13 - BYG-Krian: 3D View Comparison between Optimal & Ultimate Pits (Owner-Operator)

The selected optimal pits (owner-operator) for all options with corresponding tonnage, grade and strip ratio are shown in *Table 16-17 - BYG-Krian Owner-Operator: Selected Pit Shell Tabulations – All Options* below.

Selected Optimal Pit Description	Selected Pit Shell	Tonnage (t)	Grade (g/t)	Strip Ratio
Owner-Operator_4000 TPD_Flotation	Pit 61	1,036,976	3.088	4.177

Selected Optimal Pit Description	Selected Pit Shell	Tonnage (t)	Grade (g/t)	Strip Ratio
Owner-Operator_6000 TPD_Flotation	Pit 62	1,044,129	3.086	4.329
Owner-Operator_8000 TPD_Flotation (base case)	Pit_56	1,051,312	3.077	4.412
Owner-Operator_10000 TPD_Flotation	Pit 58	1,060,487	3.062	4.501
Owner-Operator_12000 TPD_Flotation	Pit 56	1,060,720	3.062	4.504
Owner-Operator_4000 TPD_POX	Pit 57	922,020	3.393	4.573
Owner-Operator_6000 TPD_POX	Pit 49	924,908	3.388	4.590
Owner-Operator_8000 TPD_POX	Pit 52	931,535	3.378	4.680
Owner-Operator_10000 TPD_POX	Pit 51	936,021	3.367	4.679
Owner-Operator_12000 TPD_POX	Pit 50	936,487	3.366	4.683
Owner-Operator_4000 TPD_BIOX	Pit 54	861,631	3.535	4.411
Owner-Operator_6000 TPD_BIOX	Pit 55	870,967	3.520	4.528
Owner-Operator_000 TPD_BIOX	Pit 46	869,054	3.516	4.423
Owner-Operator_10000 TPD_BIOX	Pit 55	886,374	3.504	4.851
Owner-Operator_12000 TPD_BIOX	Pit 50	886,642	3.503	4.852
Owner-Operator_4000 TPD_ALBION	Pit 54	742,295	3.902	4.619
Owner-Operator_6000 TPD_ALBION	Pit 48	743,193	3.898	4.617
Owner-Operator_8000 TPD_ALBION	Pit 53	770,764	3.854	5.223
Owner-Operator_10000 TPD_ALBION	Pit 58	771,907	3.851	5.230
Owner-Operator_12000 TPD_ALBION	Pit 55	772,022	3.851	5.232

Table 16-17 - BYG-Krian Owner-Operator: Selected Pit Shell Tabulations – All Options

And for Contract-Mining at BYG-Krian the pit shells are:

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 1	45,395,766	1,280,673	533,940	746,733	1.399
Pit 2	1,840,943	30,488	8,636	21,851	2.530
Pit 3	8,352,881	222,242	46,441	175,801	3.786
Pit 4	6,860,254	282,928	67,466	215,462	3.194
Pit 5	1,137,855	96,597	15,276	81,321	5.323
Pit 6	496,599	64,974	16,809	48,165	2.865
Pit 7	2,262,174	115,840	19,878	95,962	4.828
Pit 8	157,997	13,479	2,372	11,107	4.683
Pit 9	1,592,736	128,077	22,728	105,349	4.635
Pit 10	30,115	2,154	850	1,304	1.533
Pit 11	220,219	35,410	7,120	28,290	3.973
Pit 12	550,324	55,825	7,744	48,082	6.209
Pit 13	2,180,043	211,886	26,200	185,686	7.087
Pit 14	3,092,705	365,176	44,964	320,212	7.122
Pit 15	401,777	60,030	8,927	51,103	5.724

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 16	1,018,672	102,916	12,895	90,022	6.981
Pit 17	211,760	38,002	5,723	32,279	5.640
Pit 18	508,477	73,844	8,758	65,086	7.432
Pit 19	116,851	24,432	3,077	21,356	6.941
Pit 20	380,382	39,986	3,474	36,512	10.510
Pit 21	381,371	57,619	5,720	51,899	9.073
Pit 22	809,523	144,181	11,010	133,171	12.095
Pit 23	126,121	41,689	4,697	36,992	7.876
Pit 24	318,708	86,761	9,934	76,827	7.734
Pit 25	447,405	89,337	5,365	83,971	15.651
Pit 26	135,603	38,396	3,691	34,704	9.402
Pit 27	929,017	233,008	27,236	205,772	7.555
Pit 28	12,359	4,212	954	3,258	3.415
Pit 29	152,214	49,406	4,064	45,342	11.158
Pit 30	183,056	65,705	8,324	57,381	6.893
Pit 31	347,902	87,265	5,050	82,215	16.279
Pit 32	3,660	2,101	337	1,764	5.236
Pit 33	261,369	91,622	5,913	85,710	14.496
Pit 34	81,610	29,310	2,224	27,086	12.180
Pit 35	21,520	6,665	441	6,224	14.120
Pit 36	160,195	65,677	5,905	59,771	10.122
Pit 37	21,097	14,310	2,010	12,300	6.120
Pit 38	14,001	5,328	224	5,105	22.804
Pit 39	518,386	233,811	15,781	218,030	13.816
Pit 40	5,486	2,870	414	2,456	5.937
Pit 41	11,486	9,324	341	8,983	26.324
Pit 42	4,355	4,297	572	3,725	6.509
Pit 43	3,048	7,710	1,366	6,345	4.646
Pit 44	74,746	53,165	3,930	49,235	12.528
Pit 45	19,233	19,893	1,494	18,400	12.319
Pit 46	4,112	3,322	531	2,790	5.251
Pit 47	53,687	40,747	2,573	38,174	14.838
Pit 48	74,978	55,304	1,820	53,484	29.380
Pit 49	7,436	11,197	1,192	10,005	8.393
Pit 50	154,120	171,805	10,987	160,818	14.636
Pit 51	6,359	13,919	1,915	12,003	6.266
Pit 52	16,707	27,028	1,691	25,337	14.984
Pit 53	1,144	2,663	258	2,405	9.332
Pit 54	6,394	11,670	712	10,958	15.387
Pit 55	2,556	4,795	422	4,372	10.353
Pit 56	5,393	11,455	1,207	10,248	8.494
Pit 57	5,589	12,864	440	12,423	28.219
Pit 58	30,741	81,994	3,459	78,536	22.706

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 59	3,122	15,951	732	15,219	20.779
Pit 60	4,425	15,172	1,614	13,558	8.400
Pit 61	4,174	47,962	1,397	46,566	33.343
Pit 62	1,771	54,403	5,547	48,857	8.808
Pit 63	26	7,099	112	6,986	62.143

Table 16-18 - BYG-Krian: Incremental Pit Shell Data (Contract-Mining)

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 1	45,395,766	1,280,673	533,940	746,733	0.235
Pit 2	47,236,709	1,311,161	542,577	768,584	0.292
Pit 3	55,589,591	1,533,403	589,017	944,385	0.322
Pit 4	62,449,845	1,816,331	656,483	1,159,848	0.350
Pit 5	63,587,701	1,912,928	671,760	1,241,168	0.405
Pit 6	64,084,299	1,977,902	688,569	1,289,333	0.436
Pit 7	66,346,473	2,093,742	708,447	1,385,295	0.465
Pit 8	66,504,471	2,107,221	710,819	1,396,402	0.492
Pit 9	68,097,207	2,235,298	733,547	1,501,751	0.538
Pit 10	68,127,322	2,237,452	734,398	1,503,055	0.567
Pit 11	68,347,541	2,272,863	741,518	1,531,345	0.606
Pit 12	68,897,865	2,328,688	749,261	1,579,427	0.634
Pit 13	71,077,908	2,540,574	775,461	1,765,113	0.672
Pit 14	74,170,613	2,905,750	820,425	2,085,325	0.724
Pit 15	74,572,390	2,965,780	829,352	2,136,428	0.728
Pit 16	75,591,062	3,068,696	842,247	2,226,449	0.745
Pit 17	75,802,822	3,106,699	847,970	2,258,729	0.752
Pit 18	76,311,299	3,180,542	856,728	2,323,814	0.796
Pit 19	76,428,150	3,204,974	859,804	2,345,170	0.798
Pit 20	76,808,532	3,244,961	863,279	2,381,682	0.865
Pit 21	77,189,903	3,302,580	868,999	2,433,581	0.868
Pit 22	77,999,426	3,446,761	880,009	2,566,752	0.870
Pit 23	78,125,548	3,488,450	884,706	2,603,744	0.919
Pit 24	78,444,255	3,575,211	894,640	2,680,571	0.920
Pit 25	78,891,660	3,664,548	900,005	2,764,543	0.923
Pit 26	79,027,263	3,702,944	903,697	2,799,247	0.927
Pit 27	79,956,280	3,935,951	930,933	3,005,019	0.993
Pit 28	79,968,639	3,940,164	931,887	3,008,277	1.007
Pit 29	80,120,853	3,989,570	935,951	3,053,619	1.016
Pit 30	80,303,909	4,055,275	944,275	3,111,000	1.047
Pit 31	80,651,811	4,142,540	949,325	3,193,215	1.052
Pit 32	80,655,472	4,144,641	949,662	3,194,979	1.098
Pit 33	80,916,840	4,236,263	955,575	3,280,688	1.100
Pit 34	80,998,451	4,265,573	957,799	3,307,774	1.118

Phase	NPV ($)	Rock (t)	Total Ore (t)	Total Waste (t)	Strip Ratio
Pit 35	81,019,970	4,272,238	958,240	3,313,999	1.135
Pit 36	81,180,165	4,337,915	964,145	3,373,770	1.161
Pit 37	81,201,262	4,352,225	966,155	3,386,070	1.166
Pit 38	81,215,263	4,357,554	966,379	3,391,175	1.237
Pit 39	81,733,649	4,591,364	982,159	3,609,205	1.247
Pit 40	81,739,135	4,594,234	982,573	3,611,661	1.255
Pit 41	81,750,621	4,603,558	982,914	3,620,644	1.262
Pit 42	81,754,976	4,607,855	983,487	3,624,369	1.327
Pit 43	81,758,024	4,615,566	984,852	3,630,713	1.333
Pit 44	81,832,770	4,668,730	988,782	3,679,948	1.347
Pit 45	81,852,003	4,688,624	990,276	3,698,348	1.349
Pit 46	81,856,115	4,691,945	990,807	3,701,138	1.351
Pit 47	81,909,802	4,732,692	993,380	3,739,312	1.355
Pit 48	81,984,780	4,787,997	995,200	3,792,796	1.369
Pit 49	81,992,215	4,799,194	996,392	3,802,801	1.396
Pit 50	82,146,335	4,970,999	1,007,380	3,963,619	1.401
Pit 51	82,152,695	4,984,918	1,009,295	3,975,622	1.408
Pit 52	82,169,402	5,011,945	1,010,986	4,000,959	1.472
Pit 53	82,170,546	5,014,608	1,011,244	4,003,364	1.473
Pit 54	82,176,940	5,026,278	1,011,956	4,014,321	1.477
Pit 55	82,179,496	5,031,072	1,012,378	4,018,694	1.480
Pit 56	82,184,889	5,042,527	1,013,585	4,028,942	1.482
Pit 57	82,190,478	5,055,391	1,014,025	4,041,365	1.491
Pit 58	82,221,219	5,137,385	1,017,484	4,119,901	1.505
Pit 59	82,224,341	5,153,336	1,018,217	4,135,120	1.506
Pit 60	82,228,765	5,168,508	1,019,831	4,148,678	1.509
Pit 61	82,232,939	5,216,470	1,021,227	4,195,243	1.513
Pit 62	82,234,710	5,270,874	1,026,774	4,244,100	1.517
Pit 63	82,234,736	5,277,972	1,026,886	4,251,086	1.582

Table 16-19 - BYG-Krian: Cumulative Pit Shell Data (Contract-Mining)

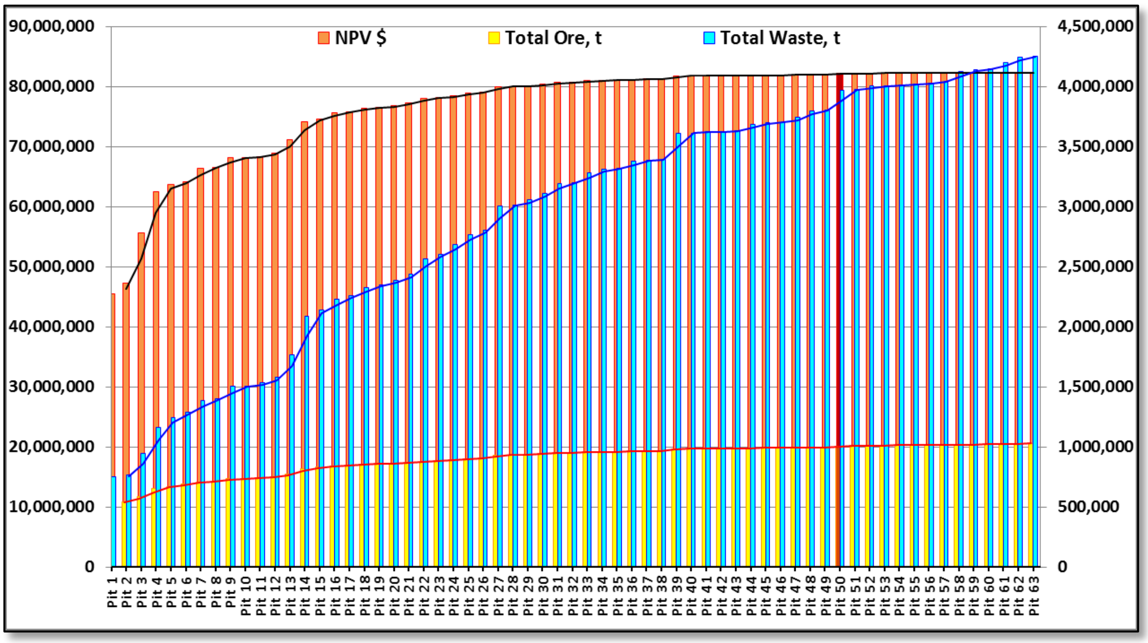


Figure 16-14 - BYG-Krian: Graph of Cumulative Ore & Waste Tonnes and NPV (Contract-Mining)

To show the impact graphically and in terms of the 3D pit the optimal pit shell (Pit_Shell_50) and the ultimate pit shell (Pit_shell_63) are shown side by side. *Figure 16-15 - BYG-Krian: 3D View Comparison between Optimal & Ultimate Pits (Contract-Mining)* shows the two pits below and it can be seen that there are very marginal differences between each.

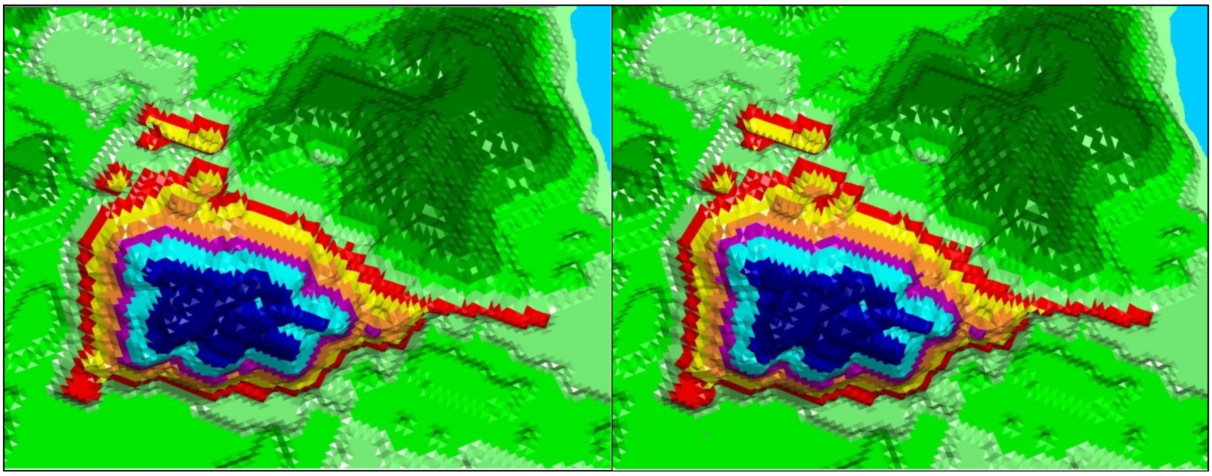


Figure 16-15 - BYG-Krian: 3D View Comparison between Optimal & Ultimate Pits (Contract-Mining)

The selected optimal pits (contract-mining) for all options with corresponding tonnage, grade and strip ratio are shown in *Table 16-20 - BYG-Krian Contract-Mining: Selected Pit Shell Tabulations – All Options* below.

Selected Optimal Pit Description	Selected Pit Shell	Tonnage (t)	Grade (g/t)	Strip Ratio
Contract-Mining_4000 TPD_Flotation	Pit 49	972,764	3.177	3.632
Contract-Mining_6000 TPD_Flotation	Pit 56	1,005,425	3.137	3.927

Selected Optimal Pit Description	Selected Pit Shell	Tonnage (t)	Grade (g/t)	Strip Ratio
Contract-Mining_8000 TPD_Flotation (base case)	Pit 50	1,007,380	3.133	3.935
Contract-Mining_10000 TPD_Flotation	Pit 46	1,008,087	3.132	3.936
Contract-Mining_12000 TPD_Flotation	Pit 53	1,021,034	3.116	4.075
Contract-Mining_4000 TPD_POX	Pit 49	869,185	3.482	4.025
Contract-Mining_6000 TPD_POX	Pit 51	874,158	3.469	4.030
Contract-Mining_8000 TPD_POX	Pit 48	901,235	3.419	4.253
Contract-Mining_10000 TPD_POX	Pit 47	906,610	3.405	4.242
Contract-Mining_12000 TPD_POX	Pit 47	907,239	3.404	4.248
Contract-Mining_4000 TPD_BIOX	Pit 49	814,266	3.628	4.009
Contract-Mining_6000 TPD_BIOX	Pit 48	816,923	3.620	4.007
Contract-Mining_000 TPD_BIOX	Pit 42	818,861	3.616	4.021
Contract-Mining_10000 TPD_BIOX	Pit 41	828,120	3.592	4.035
Contract-Mining_12000 TPD_BIOX	Pit 51	861,004	3.537	4.413
Contract-Mining_4000 TPD_ALBION	Pit 52	711,614	3.980	4.414
Contract-Mining_6000 TPD_ALBION	Pit 51	716,966	3.967	4.460
Contract-Mining_8000 TPD_ALBION	Pit 45	722,377	3.951	4.471
Contract-Mining_10000 TPD_ALBION	Pit 49	733,809	3.923	4.561
Contract-Mining_12000 TPD_ALBION	Pit 49	734,133	3.922	4.556

Table 16-20 - BYG-Krian Contract-Mining: Selected Pit Shell Tabulations – All Options

16.2.5. Pushbacks & Parameters

Upon completion of the ultimate pit process, optimal pit selection and pit design the next step is to define the pit pushbacks in order to define the major expansion phases and generic extraction sequence. These phases or pushbacks are developed in order to maximise productivity and delay waste mining.

16.2.5.1. Pushbacks & Parameters – Jugan

For the Jugan pushbacks the "optimal pit" shell was used as the limit and the number of pushbacks was defined to meet the pit expansion phases, which in the basecase was six (6). Other parameters were applied such as minimum size/number of blocks (10,000 m^2) within a pushback so as to ensure no islands are left the sequence, and minimum width for pushback, which was 20 metres.

Listed below in *Table 16-21 - Jugan: Pushback Parameters (Owner-Operator)* and *Table 16-22 - Jugan: Pushback Parameters (Contract-Mining)* are the pushback parameters for the different options (owner-operator & contract-mining) at Jugan.

Metallurgical Process	Tonnes Per Day	Selected Pit (Optimal)	Ore Size Control	No. of Pushback
By Flotation Process	t/d	Pit Shell No.	Ore (tonnes)	
Flotation1	4,000	52	960,000	6
Flotation2	6,000	55	1,080,000	6
Flotation3	8,000	54	1,200,000	6
Flotation4	10,000	54	1,800,000	5
Flotation5	12,000	61	1,500,000	6
General criteria for size control;				
a) Ore tonnes = or > 3 months ore production for all process				
b) Total material (Ore + Waste) should not exceed mining fleet capacity				
By POX Process	t/d	Pit Shell No.	Ore (tonnes)	
POX1	4,000	56	1,080,000	6
POX2	6,000	53	1,080,000	6
POX3	8,000	56	1,280,000	6
POX4	10,000	54	1,200,000	6
POX5	12,000	54	1,440,000	5
By BIOX Process	t/d	Pit Shell No.	Ore (tonnes)	
BIOX1	4,000	54	960,000	6
BIOX2	6,000	55	1,080,000	6
BIOX3	8,000	59	1,320,000	6
BIOX4	10,000	62	1,200,000	6
BIOX5	12,000	57	1,800,000	4
By ALBION Process	t/d	Pit Shell No.	Ore (tonnes)	
ALBN1	4,000	59	840,000	6
ALBN2	6,000	57	1,080,000	5
ALBN3	8,000	56	900,000	6
ALBN4	10,000	58	1,200,000	4
ALBN5	12,000	57	1,800,000	4

Table 16-21 - Jugan: Pushback Parameters (Owner-Operator)

Metallurgical Process	Tonnes Per Day	Selected Pit (Optimal)	Ore Size Control	No. of Pushback
By Flotation Process	t/d	Pit Shell No.	Ore (tonnes)	
Flotation1	4,000	61	960,000	7
Flotation2	6,000	53	1,080,000	6
Flotation3	8,000	52	1,200,000	6
Flotation4	10,000	61	1,800,000	5

Metallurgical Process	Tonnes Per Day	Selected Pit (Optimal)	Ore Size Control	No. of Pushback
Flotation5	12,000	59	1,500,000	6
General criteria for size control;				
a) Ore tonnes = or > 3 months ore production for all process				
b) Total material (Ore + Waste) should not exceed mining fleet capacity				
By POX Process	t/d	Pit Shell No.	Ore (tonnes)	
POX1	4,000	55	1,080,000	6
POX2	6,000	63	1,080,000	6
POX3	8,000	59	1,280,000	6
POX4	10,000	60	1,200,000	5
POX5	12,000	58	1,440,000	5
By BIOX Process	t/d	Pit Shell No.	Ore (tonnes)	
BIOX1	4,000	53	960,000	6
BIOX2	6,000	54	1,080,000	6
BIOX3	8,000	59	1,320,000	5
BIOX4	10,000	57	1,200,000	5
BIOX5	12,000	59	1,800,000	4
By ALBION Process	t/d	Pit Shell No.	Ore (tonnes)	
ALBN1	4,000	65	840,000	5
ALBN2	6,000	62	1,080,000	4
ALBN3	8,000	64	900,000	4
ALBN4	10,000	64	1,200,000	4
ALBN5	12,000	60	1,800,000	4

Table 16-22 - Jugan: Pushback Parameters (Contract-Mining)

For the basecase the pushbacks are also checked against the pit shell defined by the detailed pit design, which in turn is the design based on the optimal pit shell, but incorporates the detailed design elements.

The pushbacks or expansion phases and sequence for the basecase are shown graphically in *Figure 16-16 – Jugan Basecase: Original* Topography to *Figure 16-22 - Jugan Basecase: Pit Pushback 6* below. Note the first figure is the original topography.

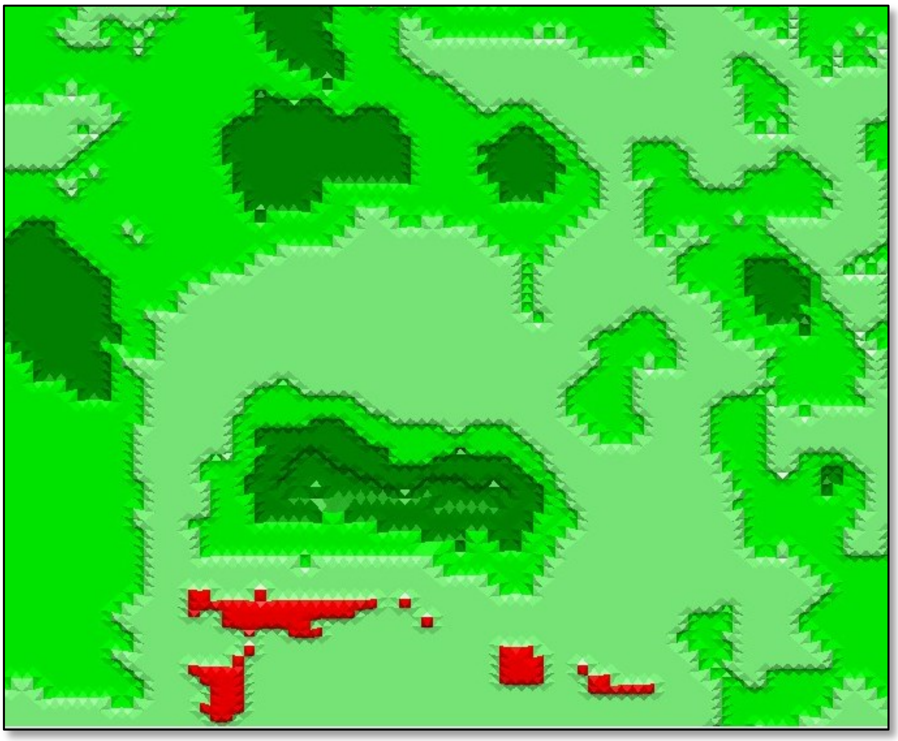


Figure 16-16 – Jugan Basecase: Original Topography

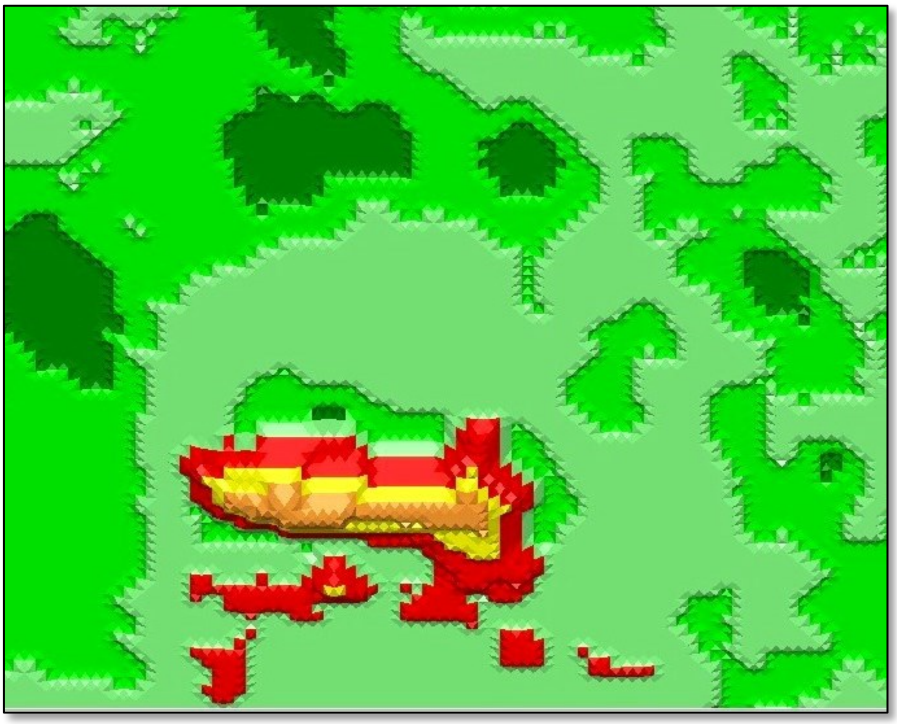


Figure 16-17 – Jugan Basecase: Pit Pushback 1

Figure 16-18 - Jugan Basecase: Pit Pushback 2

Figure 16-19 - Jugan Basecase: Pit Pushback 3

Figure 16-20 - Jugan Basecase: Pit Pushback 4

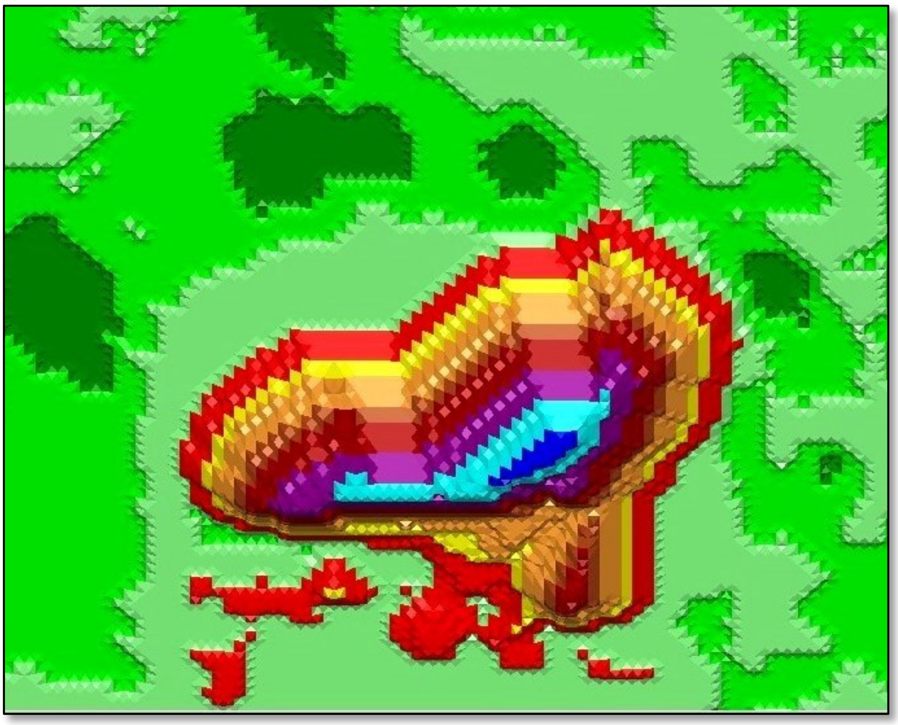


Figure 16-21 - Jugan Basecase: Pit Pushback 5

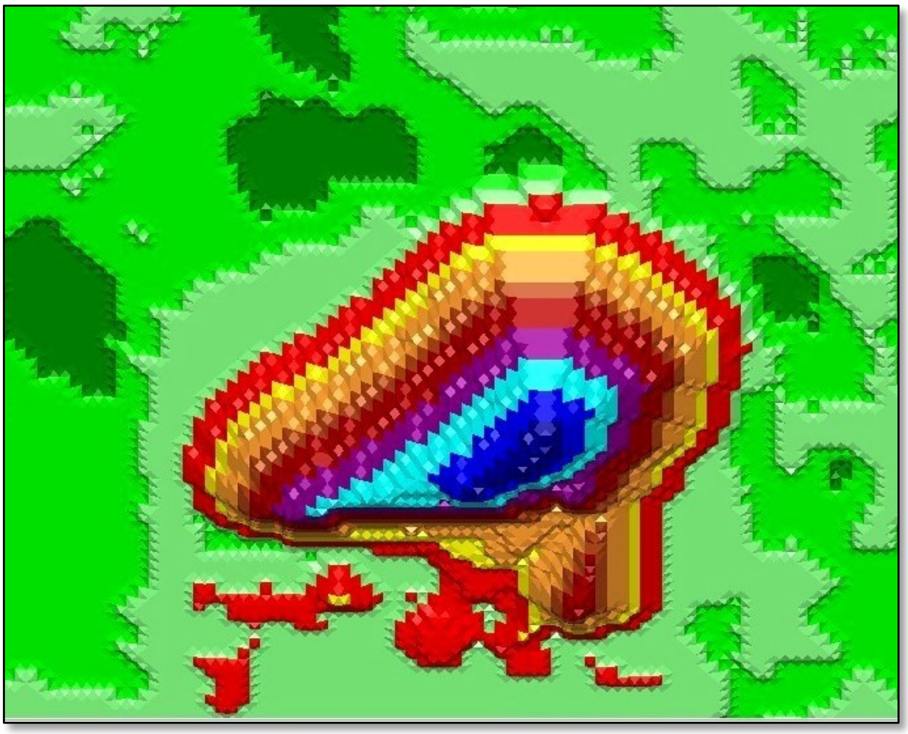


Figure 16-22 - Jugan Basecase: Pit Pushback 6

16.2.5.2. Pushbacks & Parameters – BYG-Krian

Similarly, for BYG-Krian the pit shells used were based on the "optimal pit" shell, minimum size is 10,000 m^2 and minimum pushback distance is 20 metres. The parameters used for BYG-Krian are shown in *Table 16-23 - BYG-Krian: Pushback Parameters (Owner-Operator)* and *Table 16-24 - BYG-Krian: Pushback Parameters (Contract-Mining)* below.

Metallurgical Process	Tonnes Per Day	Selected Pit (Optimal)	Ore Size Control	No. of Pushback
By Flotation Process	t/d	Pit Shell No.	Ore (tonnes)	
Flotation1	4,000	61	960,000	2
Flotation2	6,000	62	1,080,000	2
Flotation3	8,000	56	1,200,000	1
Flotation4	10,000	58	1,800,000	1
Flotation5	12,000	56	1,500,000	1
General criteria for size control;				
a) Ore tonnes = or > 3 months ore production for all process				
b) Total material (Ore + Waste) should not exceed mining fleet capacity				
By POX Process	t/d	Pit Shell No.	Ore (tonnes)	
POX1	4,000	57	1,080,000	2
POX2	6,000	49	1,080,000	2
POX3	8,000	52	1,280,000	1

Metallurgical Process	Tonnes Per Day	Selected Pit (Optimal)	Ore Size Control	No. of Pushback
POX4	10,000	51	1,200,000	1
POX5	12,000	50	1,440,000	1
By BIOX Process	t/d	Pit Shell No.	Ore (tonnes)	
BIOX1	4,000	54	960,000	2
BIOX2	6,000	55	1,080,000	2
BIOX3	8,000	46	1,320,000	1
BIOX4	10,000	55	1,200,000	1
BIOX5	12,000	50	1,800,000	1
By ALBION Process	t/d	Pit Shell No.	Ore (tonnes)	
ALBN1	4,000	54	840,000	1
ALBN2	6,000	48	1,080,000	1
ALBN3	8,000	53	900,000	1
ALBN4	10,000	58	1,200,000	1
ALBN5	12,000	55	1,800,000	1

Table 16-23 - BYG-Krian: Pushback Parameters (Owner-Operator)

Metallurgical Process	Tonnes Per Day	Selected Pit (Optimal)	Ore Size Control	No. of Pushback
By Flotation Process	t/d	Pit Shell No.	Ore (tonnes)	
Flotation1	4,000	49	960,000	1
Flotation2	6,000	56	1,080,000	1
Flotation3	8,000	50	1,200,000	1
Flotation4	10,000	46	1,800,000	1
Flotation5	12,000	53	1,500,000	1
General criteria for size control;				
a) Ore tonnes = or > 3 months ore production for all process				
b) Total material (Ore + Waste) should not exceed mining fleet capacity				
By POX Process	t/d	Pit Shell No.	Ore (tonnes)	
POX1	4,000	49	1,080,000	1
POX2	6,000	51	1,080,000	1
POX3	8,000	48	1,280,000	1
POX4	10,000	47	1,200,000	1
POX5	12,000	47	1,440,000	1
By BIOX Process	t/d	Pit Shell No.	Ore (tonnes)	
BIOX1	4,000	49	960,000	1
BIOX2	6,000	48	1,080,000	1

Metallurgical Process	**Tonnes Per Day**	**Selected Pit (Optimal)**	**Ore Size Control**	**No. of Pushback**
BIOX3	8,000	42	1,320,000	1
BIOX4	10,000	41	1,200,000	1
BIOX5	12,000	51	1,800,000	1
By ALBION Process	t/d	Pit Shell No.	Ore (tonnes)	
ALBN1	4,000	52	840,000	1
ALBN2	6,000	51	1,080,000	1
ALBN3	8,000	45	900,000	1
ALBN4	10,000	49	1,200,000	1
ALBN5	12,000	49	1,800,000	1

Table 16-24 - BYG-Krian: Pushback Parameters (Contract-Mining)

Shown graphically below are the pushbacks in *Figure 16-23 - BYG-Krian Basecase: Original Topography* to *Figure 16-24 - BYG-Krian Basecase: Pit Pushback.*

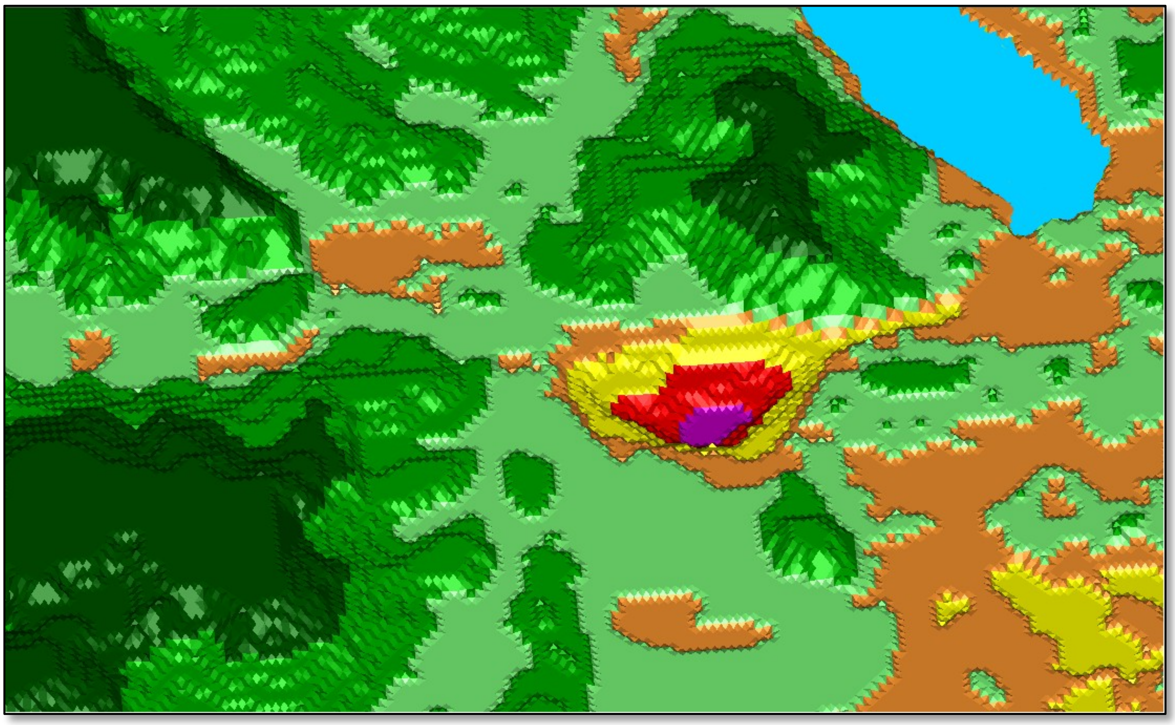


Figure 16-23 - BYG-Krian Basecase: Original Topography

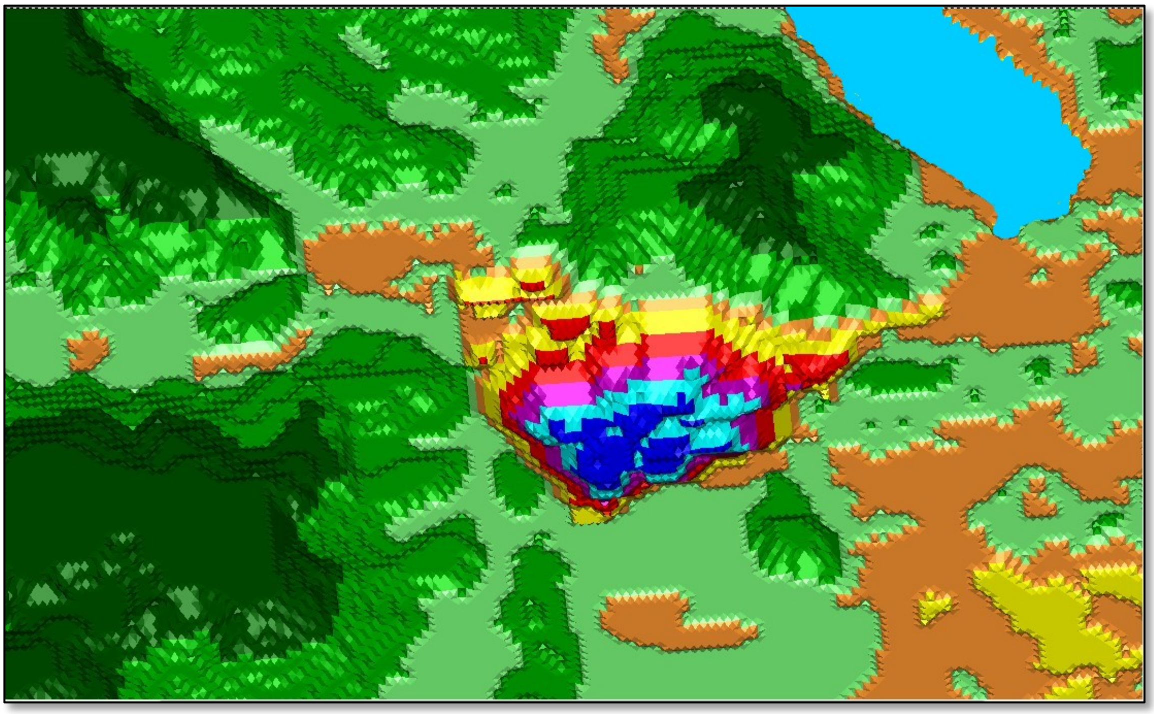


Figure 16-24 - BYG-Krian Basecase: Pit Pushback

16.2.6. Optimisation Schedule, Parameters & Results

For each Jugan pit design and set of pushbacks, encompassing each combination of mining type and process option, a set of optimisation schedules have been developed at 4,000 tpd to 12,000 tpd in 2,000 tpd increments. The schedules are defined to maximise NPV. This culminated in running a total of eighty (80) options for Jugan and BYG.

Listed below is the development schedule for the base case, i.e. 8,000 tpd flotation concentrate option with contractor mining. All other schedules are listed in detail in *Appendix A17-4 Pit Optimisation – Schedule Results*. Each pit extraction schedule is also included in the Cost Model scenario spreadsheet, which is used to determine the project costings and analysis, for each scenario.

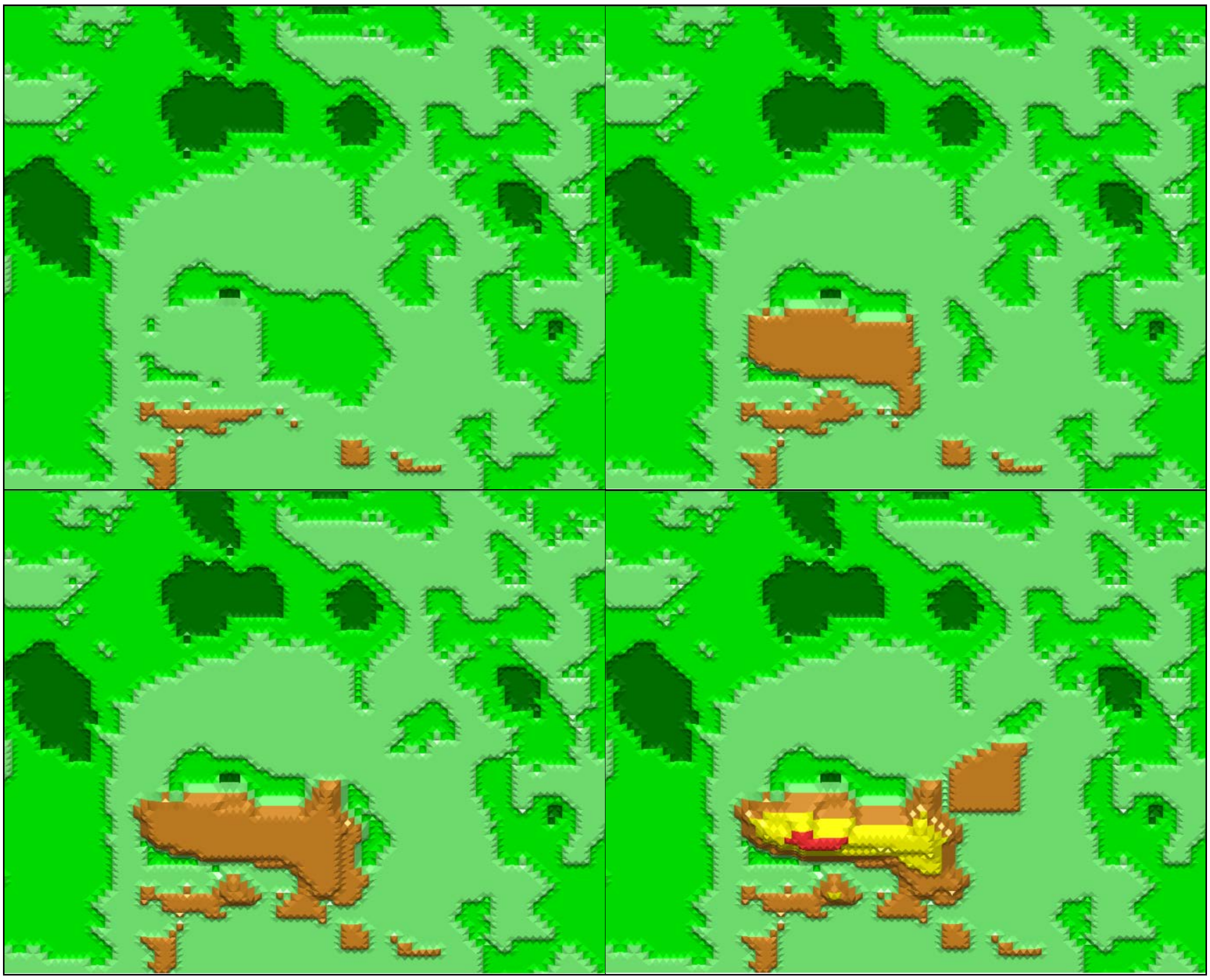
Table 16-25 - Jugan & BYG-Krian: Combined Base Case Pit Extraction Schedule (Owner-Operator & Contract-Mining Respectively) lists the combination of the Jugan and BYG-Krian pit development and extraction schedule for the base case.

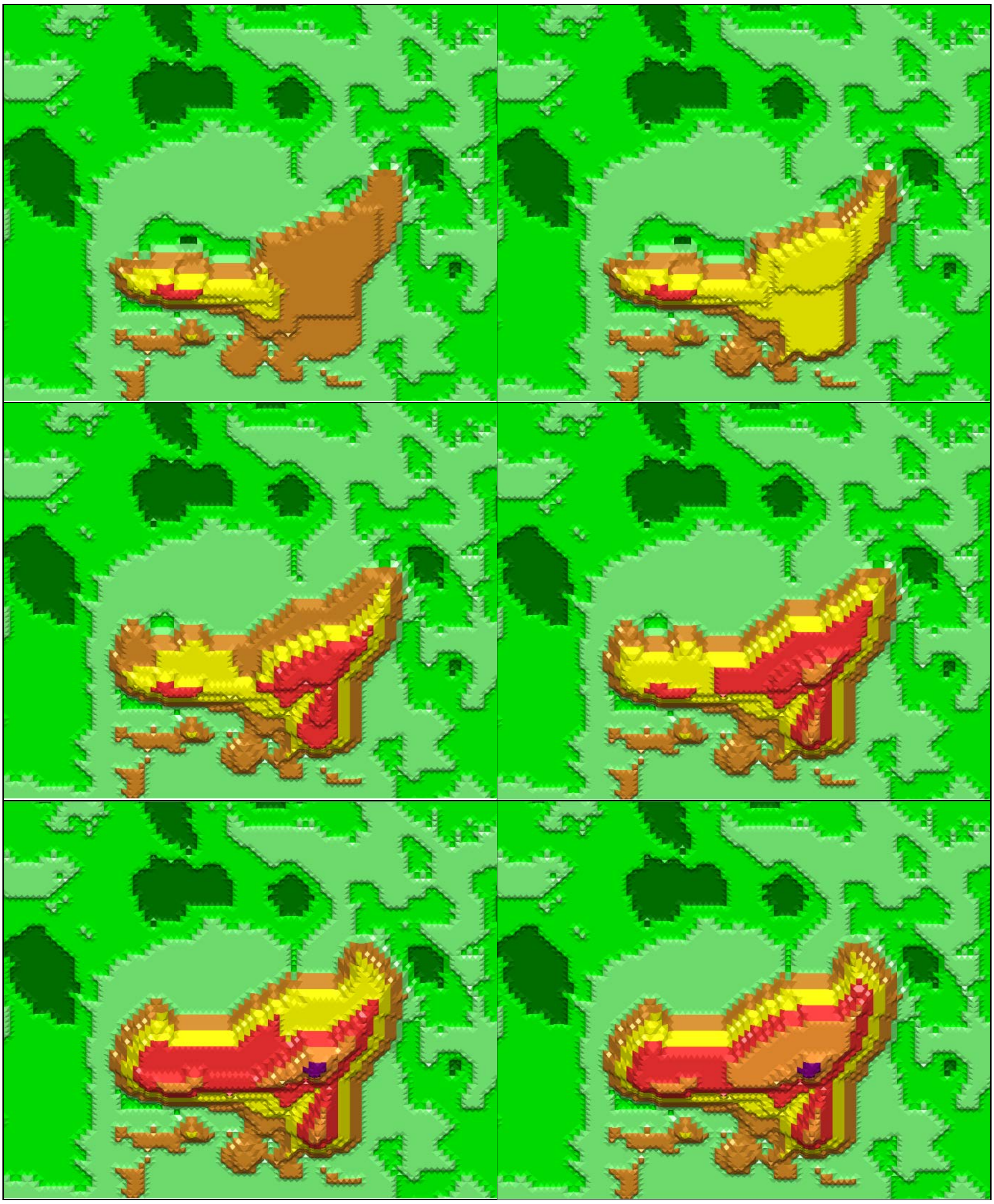
Production Option	Schedule Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4			
			Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
8_FLOT_C2	Mined Ore Tonnes	10,927,863	240,914	489,128	730,042	730,042	730,042	730,024	730,024	730,024	730,024	730,138	730,138	730,138	730,138	729,360	721,682	716,007	
8_FLOT_C2	Mined Au Grade	1.70	1.53	1.53	1.53	1.53	1.53	1.58	1.58	1.58	1.58	1.56	1.56	1.56	1.56	1.59	2.11	3.24	
8_FLOT_C2	Mined Au Ounces	598,806	11,866	24,092	35,958	35,958	35,958	36,990	36,990	36,990	36,990	36,597	36,597	36,597	36,597	37,191	48,940	74,496	
8_FLOT_C2	Mined Fe Percent	4.04	-	4.27	4.12	4.33	4.44	4.45	4.50	4.56	4.51	4.52	4.43	4.49	4.59	4.35	4.39	-	
8_FLOT_C2	Mined As Percent	0.92	-	1.06	1.04	1.00	0.92	0.97	1.06	1.08	1.02	1.06	1.04	0.99	0.99	0.95	0.96	-	
8_FLOT_C2	Mined S Percent	2.18	-	2.43	2.35	2.35	2.29	2.31	2.31	2.41	2.43	2.41	2.46	2.44	2.47	2.49	2.38	-	
8_FLOT_C2	Processed Ore Tonnes	10,927,863	0	730,042	730,042	730,042	730,042	730,024	730,024	730,024	730,024	730,138	730,138	730,138	730,138	729,360	721,682	716,007	
8_FLOT_C2	Recovered Au Grade	1.31	-	1.18	1.18	1.18	1.18	1.21	1.21	1.21	1.21	1.20	1.20	1.20	1.20	1.22	1.62	2.49	
8_FLOT_C2	Recovered Au Ounces	461,081	0	27,688	27,688	27,688	27,688	28,482	28,482	28,482	28,482	28,179	28,179	28,179	28,179	28,637	37,684	57,362	
8_FLOT_C2	Waste Tonnes	18,569,290	118,147	239,875	358,022	358,022	358,022	922,594	922,594	922,594	922,594	1,650,146	1,650,146	1,650,146	1,650,146	1,812,992	2,320,907	2,712,344	
8_FLOT_C2	Strip Ratio	1.70	0.49	0.49	0.49	0.49	0.49	1.26	1.26	1.26	1.26	2.26	2.26	2.26	2.26	2.49	3.22	3.79	

Production Option	Schedule Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4			
			Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
8_FLOT_B2	Mined Ore Tonnes	11,209,070	241,057	489,418	730,475	730,475	730,475	729,860	729,860	729,860	729,860	730,310	730,310	730,310	730,310	726,665	668,517	728,268	323,043
8_FLOT_B2	Mined Au Grade	1.69	1.53	1.53	1.53	1.53	1.53	1.56	1.56	1.56	1.56	1.55	1.55	1.55	1.55	1.55	1.55	2.74	3.83
8_FLOT_B2	Mined Au Ounces	609,895	11,873	24,106	35,979	35,979	35,979	36,677	36,677	36,677	36,677	36,394	36,394	36,394	36,394	36,306	33,401	64,179	39,810
8_FLOT_B2	Mined Fe Percent	3.92	-	4.24	4.11	4.27	4.42	4.60	4.67	4.38	4.55	4.45	4.49	4.51	4.48	4.46	4.38	-	-
8_FLOT_B2	Mined As Percent	0.89	-	1.04	1.07	1.02	0.95	0.96	1.07	1.06	1.06	1.06	0.97	1.01	0.97	0.93	0.93	-	-
8_FLOT_B2	Mined S Percent	2.12	-	2.41	2.35	2.27	2.35	2.37	2.27	2.39	2.39	2.34	2.52	2.42	2.50	2.49	2.50	-	-
8_FLOT_B2	Processed Ore Tonnes	11,209,070	-	730,475	730,475	730,475	730,475	729,860	729,860	729,860	729,860	730,310	730,310	730,310	730,310	726,665	668,517	728,268	323,043
8_FLOT_B2	Recovered Au Grade	1.30	-	1.18	1.18	1.18	1.18	1.20	1.20	1.20	1.20	1.19	1.19	1.19	1.19	1.20	1.20	2.11	2.95
8_FLOT_B2	Recovered Au Ounces	469,619	-	27,704	27,704	27,704	27,704	28,241	28,241	28,241	28,241	28,023	28,023	28,023	28,023	27,955	25,718	49,418	30,654
8_FLOT_B2	Waste Tonnes	20,926,239	126,519	256,871	383,390	383,390	383,390	983,217	983,217	983,217	983,217	1,713,535	1,713,535	1,713,535	1,713,535	2,066,570	1,901,205	3,541,970	1,095,930
8_FLOT_B2	Strip Ratio	1.87	0.52	0.52	0.52	0.52	0.52	1.35	1.35	1.35	1.35	2.35	2.35	2.35	2.35	2.84	2.84	4.86	3.39

Table 16-25 - Jugan & BYG-Krian: Combined Base Case Pit Extraction Schedule (Owner-Operator & Contract-Mining Respectively)

A 3D visual example of the extraction schedule for the Jugan base case is demonstrated in the following sequence of fourteen (14) 3D images.





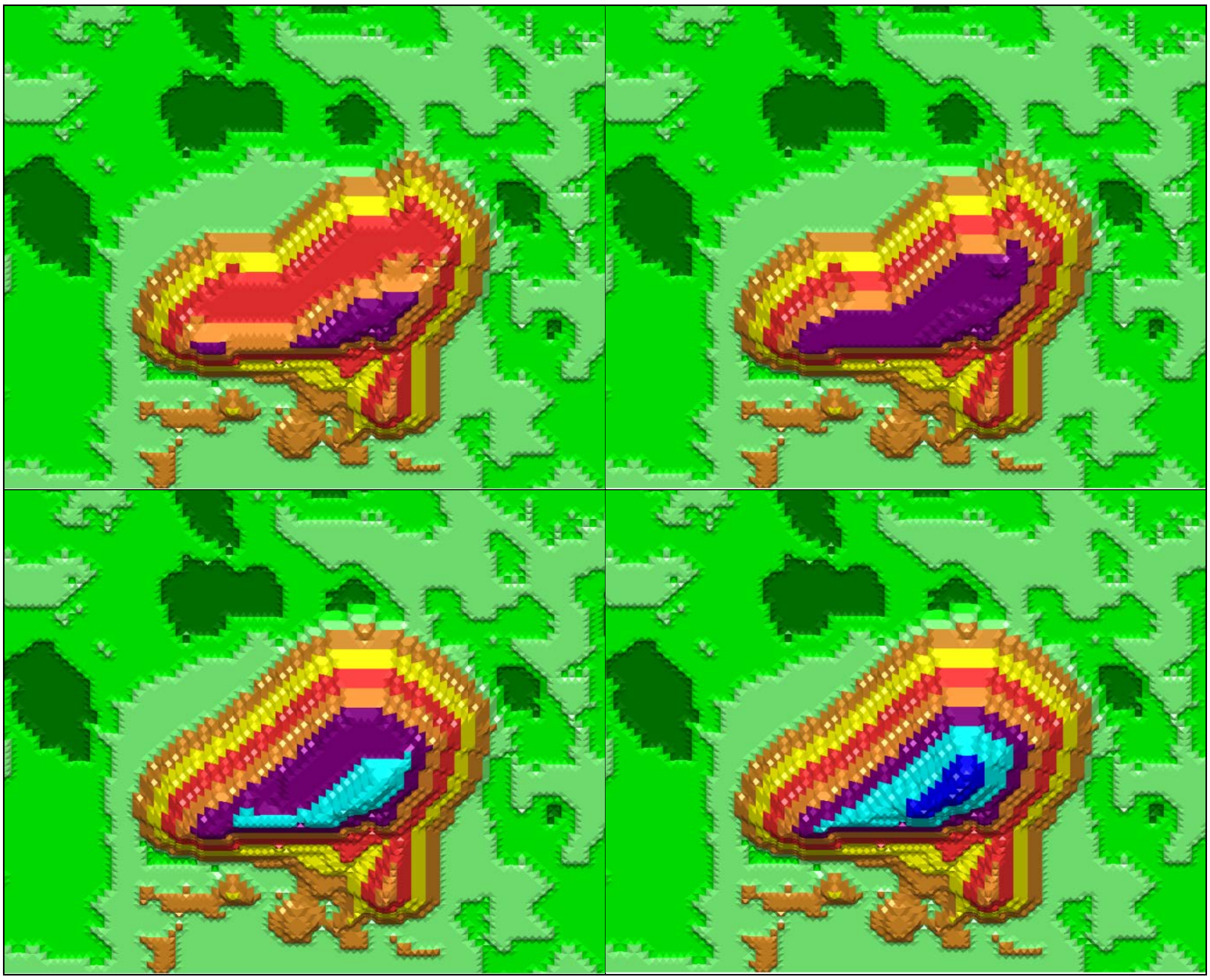


Figure 16-25 - Jugan: 3D Visual Extraction Schedule Steps - 8,000 tpd Base Case Option

16.3. Pit Design

16.3.1. Introduction & Design Methodology

The Jugan and BYG-Krian open pit designs were developed using the CAE Studio 5D Planning software. The detailed design of benches (toe & crest) and ramps was undertaken using the selected pit shell for each scenario option as an outline guide; use of the geotechnical model defining the face angle value per RMR rock zone; the final pit design parameters (based on geotechnical input); and practical design judgments of the mine planner.

16.3.2. Geotechnical Data Used

Geotechnical and geomechanical logging work, modelling and parameters used in the detailed design are described in more detail in *Section 24 – Other Information*. Key aspects and a summary are included below.

16.3.2.1. Geotechnical Data – Jugan

The geotechnical data for the Jugan deposit has indicated that there would be some regions within the ore deposit that have poor rock mass ratings based on the current modelling. The poor RMR Rating (RMR = 30 to 40) areas are located at the north and south-east region of the Jugan deposit as shown in *Figure 16-26: Jugan - Plan View RMR Model at 0mRL with Pit Layout* and *Figure 16-27: Jugan - Plan View RMR Model at -85mRL with Pit Layout* below.

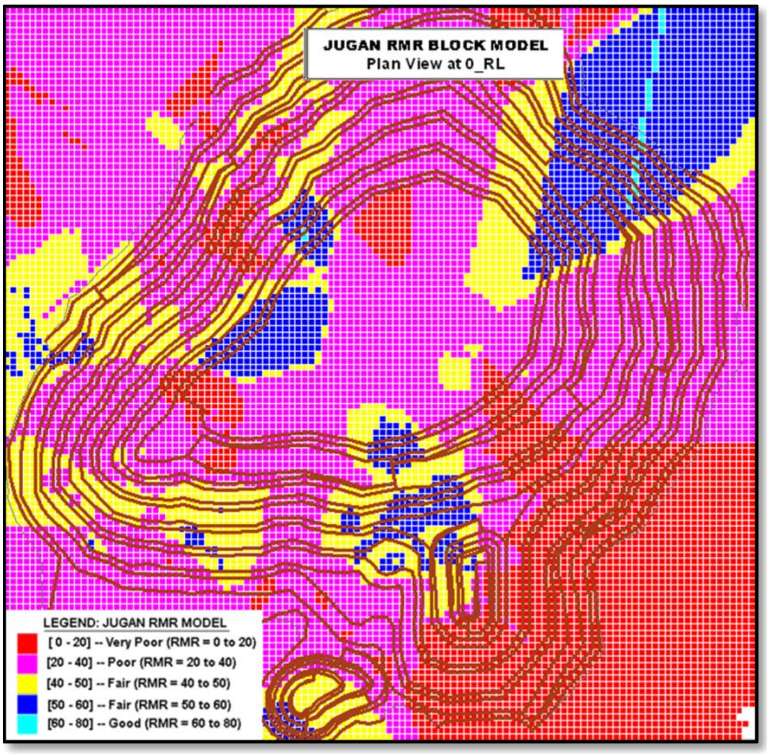


Figure 16-26: Jugan - Plan View RMR Model at 0mRL with Pit Layout

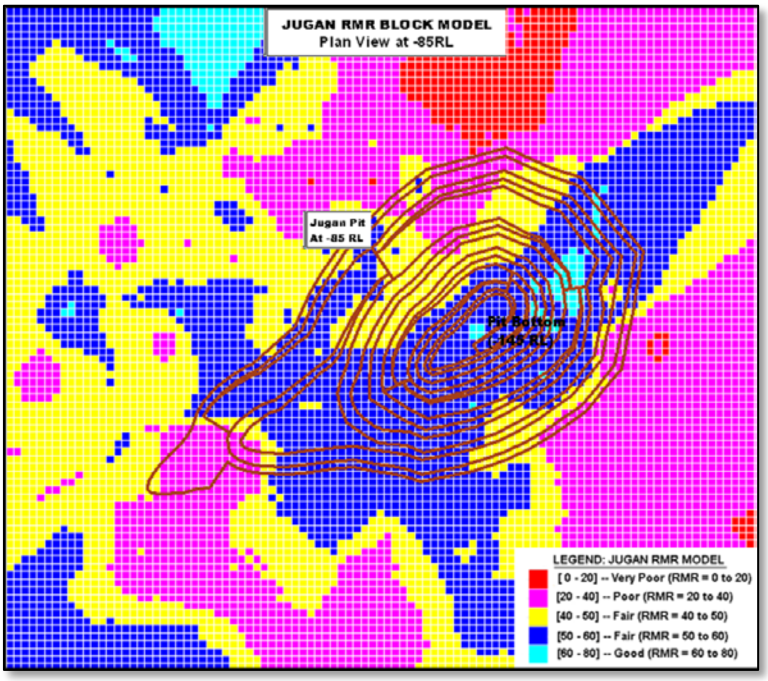


Figure 16-27: Jugan - Plan View RMR Model at -85mRL with Pit Layout

The geotechnical ratings and the associated geotechnical values are listed in *Table 16-26: Jugan - RMR Rating & Associated Values* below, with the ratings and pit design guidelines in *Table 16-27: Jugan - RMR Rating & Pit Design Guidelines.*

RMR Descriptio n	RMR Range	% Population	Est. Friction Angle (φ)	Friction Angle Wt. Ave (φ)	Est. Cohesion (kPa)	Cohesion Wt. Ave, (kPa)
Very Poor	0-20	6.15	<15	15.00	<100	100.00
Poor	20-30	30.67	15-20	16.36	100-150	113.62
Poor	30-40	24.49	20-25	23.76	150-200	187.64
Fair	40-50	29.35	25-30	26.03	200-250	210.30
Fair	50-60	8.75	30-35	32.80	250-300	278.06
Good	60-80	0.58	35-45	40.00	300-400	350.00

Table 16-26: Jugan - RMR Rating & Associated Values

RMR Description	RMR Range	Bench Height (m)	Berm Width (m)	Face Slope Angle	Overall Pit Slope Angle
Very Poor	0-20	15.00	5.00	40.00	33.30
Poor	20-30	15.00	5.00	45.00	36.90
Poor	30-40	15.00	5.00	50.00	40.50
Fair	40-50	15.00	5.00	55.00	44.00
Fair	50-60	15.00	5.00	60.00	47.70
Good	60-80	15.00	5.00	65.00	51.30

Table 16-27: Jugan - RMR Rating & Pit Design Guidelines

16.3.2.2. Geotechnical Data – BYG-Krian

The *Table 16-28 - BYG-Krian: RMR Rating & Associated Values* below and the following plan views (*Figure 16-28 – BYG-Krian: RMR Model – Plan View at 0 mRL* to *Figure 16-30 - BYG-Krian: RMR Model – Plan View at -60 mRL*) illustrate the geotechnical data of the Bukit Young deposit as mostly fair to good rock mass especially at depth. Poor RMR starts at 0 mRL which is more obvious at the south region of the deposit.

For the purpose of pit designs, the SLOPE angle and BERMWDTH fields were added into the RMR Block Model. Then in the pit design, using CAE Studio5D, these two fields were used as incumbent parameters by activating the `use model' function. The resulting overall pit slope of BYG Design Pit is much steeper than that of the Jugan Pit using the same design methodology. The design parameters are listed in *Table 16-29 - BYG-Krian: RMR Rating & Pit Design Guidelines* below.

RMR Descriptio n	RMR Range	% Population	Est. Friction Angle (φ)	Friction Angle Wt. Ave (φ)	Est. Cohesion (kPa)	Cohesion Wt. Ave, (kPa)
Very Poor	0-20	13.02	<15	15.00	<100	100.00
Poor	20-30	10.42	15-20	16.36	100-150	113.62
Poor	30-40	14.71	20-25	23.76	150-200	187.64
Fair	40-50	20.74	25-30	26.03	200-250	210.30
Fair	50-60	21.71	30-35	32.80	250-300	278.06
Good	60-80	19.37	35-45	40.00	300-400	350.00
Very Good	80-100	0.04	>45	>40	>400	>350

Table 16-28 - BYG-Krian: RMR Rating & Associated Values

RMR Description	RMR Range	Bench Height (m)	Berm Width (m)	Face Slope Angle	Overall Pit Slope Angle
Very Poor	0-20	15.00	5.00	40.00	33.30
Poor	20-30	15.00	5.00	45.00	36.90
Poor	30-40	15.00	5.00	50.00	40.50
Fair	40-50	15.00	5.00	55.00	44.00
Fair	50-60	15.00	5.00	60.00	47.70
Good	60-80	15.00	5.00	65.00	51.30
Very Good	80-100	15.00	5.00	70.00	55.00

Table 16-29 - BYG-Krian: RMR Rating & Pit Design Guidelines

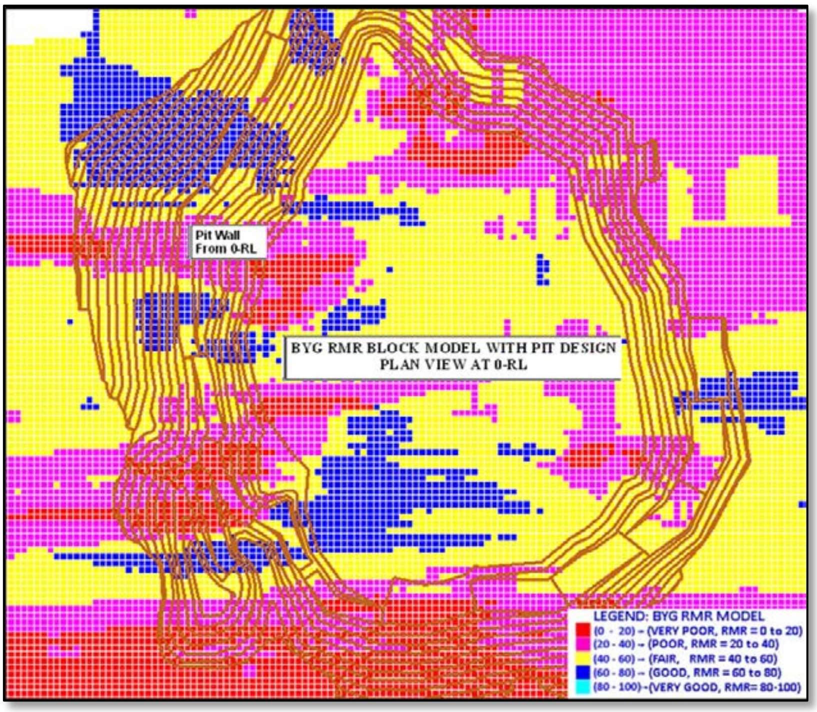


Figure 16-28 – BYG-Krian: RMR Model – Plan View at 0 mRL

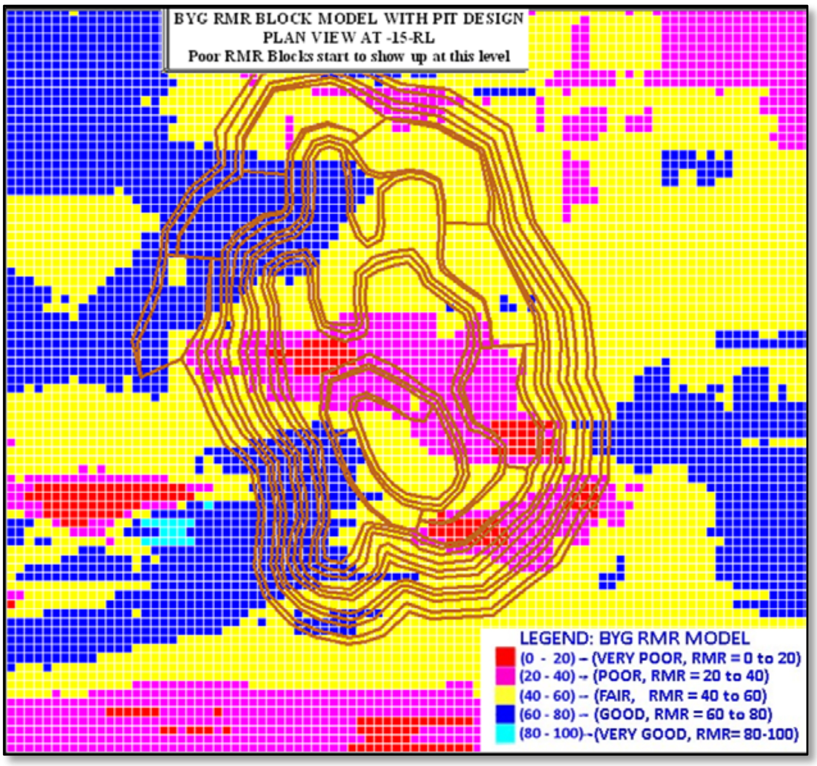


Figure 16-29 - BYG-Krian: RMR Model – Plan View at -15 mRL

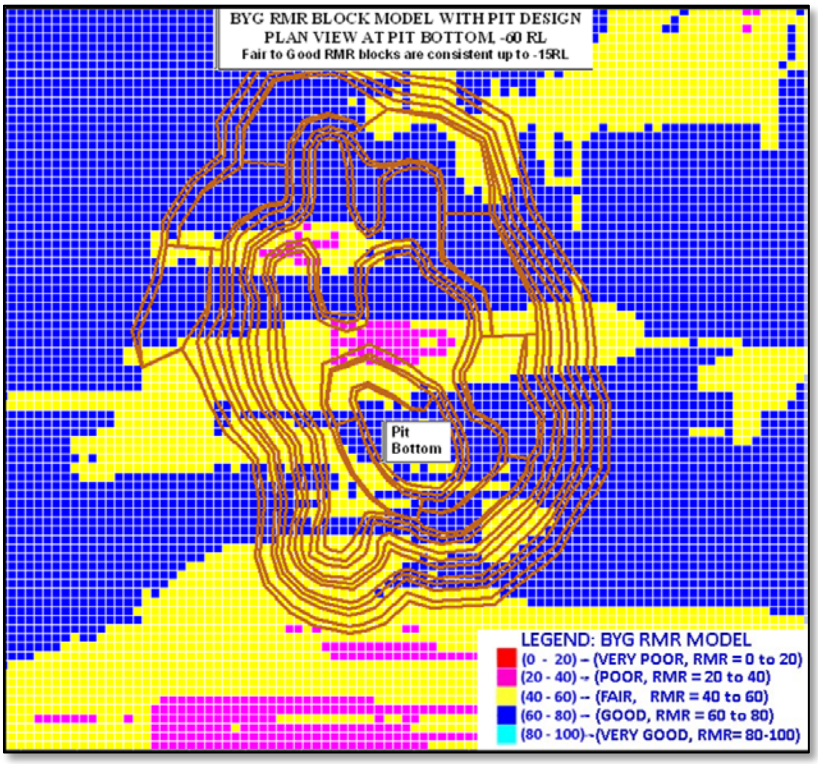


Figure 16-30 - BYG-Krian: RMR Model – Plan View at -60 mRL

16.3.3. Detailed Pit Design Parameters

As part of the pit optimisation process the practical pit designs need to be incorporated along with the specific geotechnical parameters. Outlined in the next sub-sections are the parameters and geotechnical guidelines used to design a practicable pit using the optimal pit as a guideline and comparison.

16.3.3.1. Pit Design Parameters – Jugan

The Jugan pit was designed using the RMR model with the SLOPE and BERMWDTH fields set to the appropriate values based on the above geotechnical and design parameters. This allows the software to adjust the face slope angle for each RMR zone. This is of particular significance where this zone changes part way up a bench lift, and allows for the detailed setting per zone. Other techniques, like overall slope angle settings and manual settings, are more generic in nature with the use of rosettes somewhere in between. The rosettes option was used as a cross-check against the model option for Jugan.

The values and parameters for the detailed pit design are listed in *Table 16-30: Jugan Detailed Design Parameters Using Model* below. These parameters are final pit parameters and not working pit values. Note, where "Use Model" is indicated in the table, please refer to the geotechnical parameters in the previous section.

Parameters	Value
Slope face angle (batter angle)	"Use Model"
Bench Height	15m
Ramp Width	20m
Ramp Width (-140 to -115)	10m
Berm Width	"Use Model"
Ramp Gradient	10%

Table 16-30: Jugan Detailed Design Parameters Using Model

The alternate design parameters using the rosettes function is detailed in *Table 16-31: Jugan Detailed Design Parameters Using Rosettes* below.

Parameters	Value	
Pit Slope Angle	Use Rosette	
Bench Height	15m	
Ramp Width	20m	
Ramp Width (-140 to -115)	10m	
Berm Width	5m	
Ramp Gradient	10%	
ROSETTE Settings:	**Azi**	**Slope**
Location (411575mE; 160340mN)	*Point 1*	
	0	48
	120	40
	225	48
	270	48
Location (411520mE; 160470mN)	*Point 2*	
	0	40
	90	48
	180	48
	270	44

Table 16-31: Jugan Detailed Design Parameters Using Rosettes

The resulting overall pit slopes are shown in *Table 16-32: Jugan Comparison of Overall Pit Slope Angles* below for both the model option and the alternate rosettes option.

Parameters	Without Ramp	With Ramp
Design Using Model:		
Overall PIT Slope (N & SE) measured bottom to top	36° to 39°	32° to 35°
Overall PIT Slope (W) measured bottom to top	42° to 45°	29° to 36°
Design Using Rosettes:		

Parameters	Without Ramp	With Ramp
Overall PIT Slope (N & SE) measured bottom to top	39° to 41°	34° to 36°
Overall PIT Slope (W) measured bottom to top	41° to 42.5°	30° to 36°

Table 16-32: Jugan Comparison of Overall Pit Slope Angles

The final design arising from the RMR Model and the design parameters has been accepted as the definitive pit for the base scenario option and each alternate scenario option (POX, BIOX and ALBION + Owner/Contractor). The base case pit design is shown below in *Section17.3.3.3*.

16.3.3.2. Pit Design Parameters – BYG-Krian

For the BYG-Krian pit design the model option was used. The design parameters for this pit are displayed in *Table 16-33: Jugan Detailed Design Parameters Using Model*.

Parameters	Value
Slope face angle (batter angle)	“Use Model”
Bench Height	10m
Ramp Width	20m
Ramp Width (pit bottom)	10m
Berm Width	“Use Model”
Ramp Gradient	10%

Table 16-33: Jugan Detailed Design Parameters Using Model

The overall slope angles for the designed pit at BYG-Krian are 47° without a ramp and 36° to 42° with a ramp. This pit has steeper face angles than the Jugan pit due to the host rock being primarily limestone and not shale. The final pit design for the base case is also shown below. A 10 m bench height was used at BYG-Krian instead of a 15 m height as this bench height suits the orebody better, has a lower strip ratio (less waste), slightly more reserves and better NPV.

16.3.3.3. Pit Design Output & Results – Jugan

The resultant pit design for Jugan is shown in *Figure 16-31 – Jugan: Plan View Pit Design Layout* with a 3D perspective view shown in *Figure 16-32 - Jugan: 3D View Pit Design Layout* below.

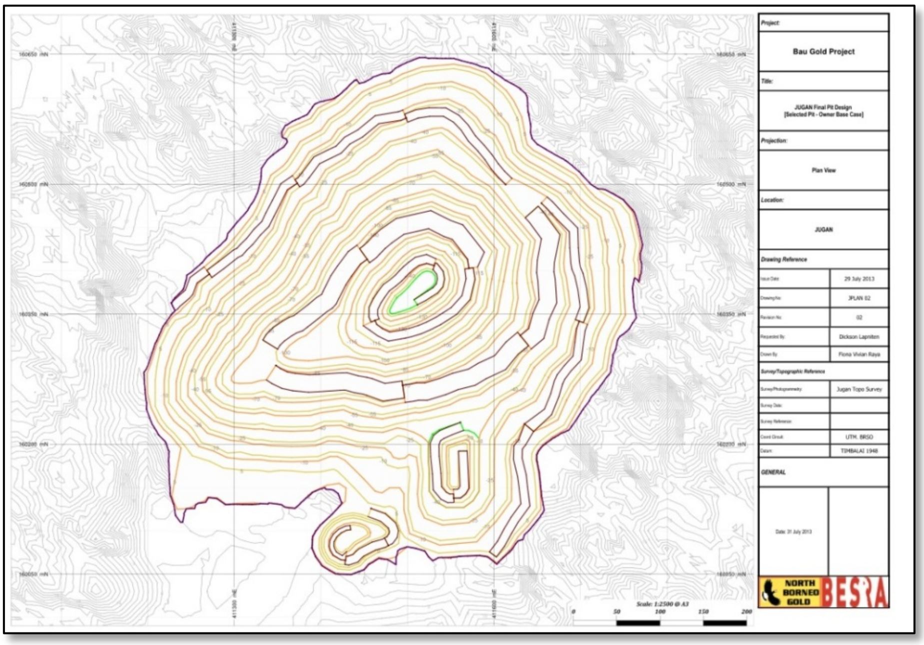


Figure 16-31 – Jugan: Plan View Pit Design Layout

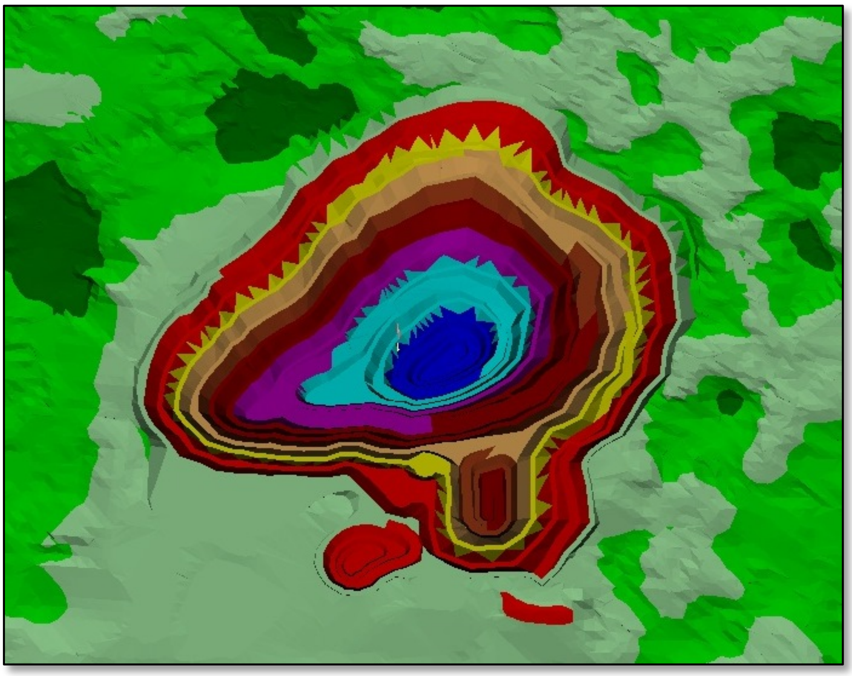


Figure 16-32 - Jugan: 3D View Pit Design Layout

16.3.3.4. Pit Design Output & Results – BYG Krian

The resultant pit design for BYG-Krian is shown in *Figure 16-33 - BYG-Krian: Plan View Pit Design Layout* with a 3D perspective view shown in *Figure 16-34 - BYG-Krian: 3D View Pit Design Layout* below.

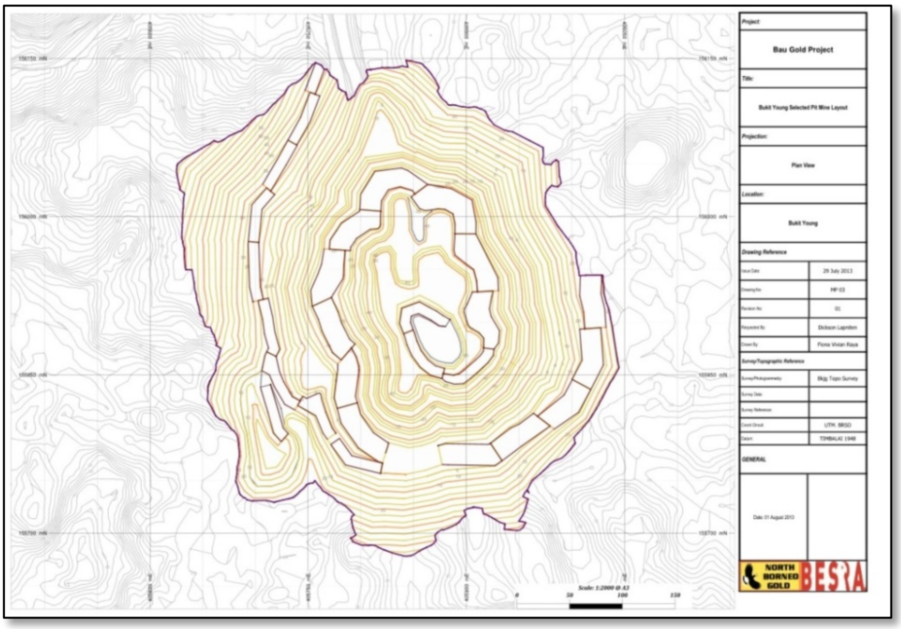


Figure 16-33 - BYG-Krian: Plan View Pit Design Layout

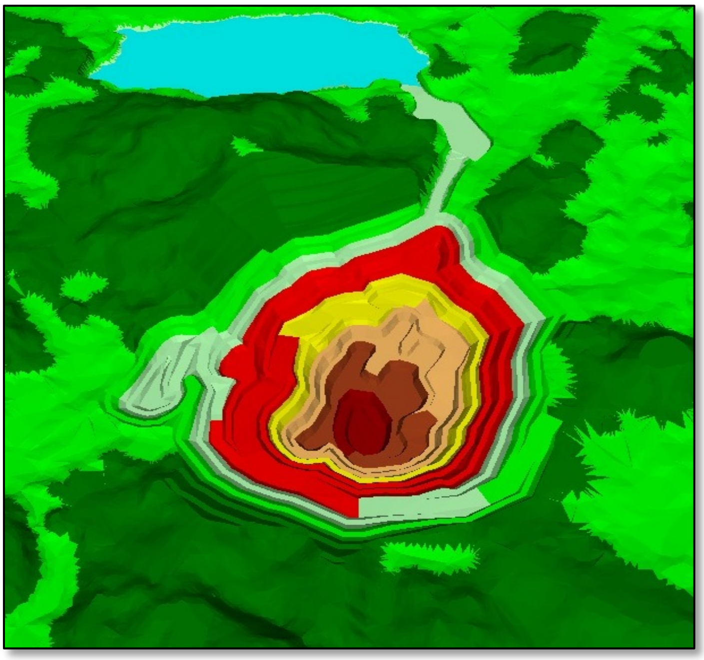


Figure 16-34 - BYG-Krian: 3D View Pit Design Layout

16.4. Pre-Production Work

Before production ore and waste mining commences at Jugan Hill, pre-production work includes removing vegetation from Jugan Hill; clearing the mining area surrounding the hill and in particular the old tailings area; pumping water from the existing ponds and diverting the streams in the immediate pit area; preparing ROM/stockpile sites and establishing working access to the hill. Prior to the commencement of mining the construction contractor will also build the permanent haul roads outside of the final pit limits, to the ROM/stockpile area and the TSF site as well as ancillary roads where applicable. An initial tailings impoundment area will also need to be constructed.

16.4.1. Vegetation Clearing, Stripping and Grubbing

Initially the vegetation on the actual Jugan will need to be cleared along with any minor surficial soils or non-ore material. This material will be minor in volume and extent. The construction contractor will clear any designated areas of the site surrounding Jugan Hill by felling all trees, shrubs and vegetation to within 500 mm of the ground. The area will then be stripped and grubbed by complete removal of all vegetation and organic matter and grubbing to remove all roots and stumps as well as the topsoil. All vegetable matter, roots and stumps, and topsoil are to be recovered from disturbed areas and stockpiled in areas adjacent to or within the mine boundaries for subsequent use as part of the closure requirements, or as otherwise required for the Reclamation Plan.

16.4.2. Removal of Miscellaneous Surface Materials

Because of the history of mining in the area, there are pockets of old tailings to the NE and SW of Jugan Hill. This area is low lying and will need to be drained and infilled. Some of this material may contain gold and needs to be tested. If suitable quantities of gold present the material will need to be dried and stored for processing. If no contained gold this material will need to be stored in a suitable impoundment due to its fine sized nature.

16.4.3. Surface De-Watering and Waterway Realignment

The area around Jugan is a shale peneplain with elevations between 10-20 metres above mean sea level. There are areas of low lying ground which is swampy in nature; current and old fish ponds or mine pits; and some streams and small waterways. The water features and swampy areas will need to be pumped and drained, and any waterways that are affected by the mining will need to be re-aligned.

16.4.4. Haul Road Construction

Pre-production construction of permanent haul roads outside of the pit limits will require the placement of 47,000 m^3 of sub-grade fill, 134,500 m^3 of base course and 67,300 m^3 of wearing layer for a total of 248,800 m^3 (317,000 m^3 for the alternate option layout). An additional 356,000 m^3 is required for stockpile areas, ROM pad and dump bases and drainage. The local shale would not be suitable for the road construction, and limestone aggregates will be sourced from the local quarries operated by our JV partner. Detailed breakdowns of the road volumes are listed in the Appendices.

It is imperative that all haul roads be constructed and maintained to high standards. The combination of heavy trucks, shale and heavy rain do not mix and the only way to maintain consistent production is to make sure that equipment can operate as much of the time as possible, which means minimum down time due to weather. Suitable road drainage infrastructure (drains, culverts, etc.) will also need to be constructed for long term operation. A total of 8.2 kilometres of drains and culverts will be needed along the roads with additional drainage around building structures, ROM pads, plant area, etc.

16.4.5. Ore Stockpile/ROM Pad Preparation

A stockpile area has been defined and has an area of 67,282 m^2. This area has been designed to cater for a 3 month (or 1 quarter) production capacity of 0.73 Mt with suitable buffers, with a stockpile maximum height of 5 metres, assuming the base case 8,000 tpd. However, it is likely that the actual ROM ore requirement will be 1-2 months capacity. The three month capacity is to allow for storage options for such situations as pre-mining prior to plant commencement; continued mining during a major plant shotdown; factors that may affect the pit production (e.g. seasonal rains); storage of marginal grade material separate from the main ROM ore for selected blending; or, storage of other materials as required.

The ore stockpile/ROM pad is situated adjacent to the plant to limit any re-handling haulage. The stockpiled ore can either be rehandled into trucks for delivery to the plant feed bin, or delivered directly by front-end loader, or loaded into a mobile conveyor. The rehandling option will be dependent upon the position of the material within the stockpile area and is relative position to the plant feed position.

The stockpile area will need to be cleared, levelled and prepared. The floor of the stockpile area will need to be laid with suitable material and/or concrete. The waste rock is predominantly shale with mudstones, siltstones and sandstones. This material breaks up when exposed, contains a lot of fines and gets slippery when wet. Some of the waste shale also contains sulphides. Therefore, the waste rock would not be suitable as a base for the stockpile/ROM pad area. However, suitable limestone is available from 4 nearby quarries owned and perated by our joint venture partner. Care needs to be taken to prevent ore loss or dilution from this stockpile/ROM pad floor.

To limit ore/gold loss, diution from floor, operation under wet conditions, water manmagement and for better manangement of ROM ore and vehicles operating, it is suggested that a concrete pad be constructed to cater for the basic or ongoing ROM ore with the remaining area covered with a limestone floor where low grade stockpiles and other material can be stored that are handled less often or require longer term storage, plus are not critical to the ongoing operation. Should the concrete pad be too expensive then a suitable limestone floor constructed and careful operational procedures implemented for ore re-handling.

Suitable construction of the stockpile area needs allow for drainage and water run-off and to prevent ponding within the stockpile area. Ring drainage and suitable ponding and silt retention infrastructure constructed. Water treatment will also need to be catered for in the water handling arrangements.

An alternate or additional option would be to provide a covered area for the ROM pad but this is envisaged as being too expensive and is not considered at this stage.

For the BYG-Krian operations, there exists areas within the old mine site, including the old ROM pad for storage of ore. This can then be fed into a plant at BYG-Krian or transported thereafter to the Jugan plant site. This is a number of years down the line and can be adjusted to suit the situation at that point in time, as things do change.

16.4.6. BYG-Krian Site Preparation Work

Currently the BYG-Krian pit is connected to the Tasik Biru Lake (formally the Tai Parit mine) via a channel that was previously the haul road from Tai Parit operations to the ROM pad behind the BYG-Krian pit. Both pits are now filled with water and there exists a water connection between via the slot or old haul road. The BYG-Krian historic pit will need to be de-watered prior to the commencement of mining operations. This slot between the pits will need to be dammed to prevent water entering from Tasik Biru once pumping out of the BYG-Krian pit commences.

Water from the catchment up-dip of the BYG-Krian pit currently enters the old workings from the opposite end of the pit area from the exit to Tasik Biru. Before operations commence and during the proposed mining operations the catchment water will need to be diverted via a water channel to Tasik Biru.

When the water diversion channel and slot plug are complete the current BYG-Krian pit can be de-watered via suitably sized pumps, treated and discharged suitably into Tasik Biru Lake or other suitable places. The plug/dam and water diversion are diagrammatically shown in *Figure 16-35 - BYG-Krian Indicative Diagram of Site Preparation Work* below.

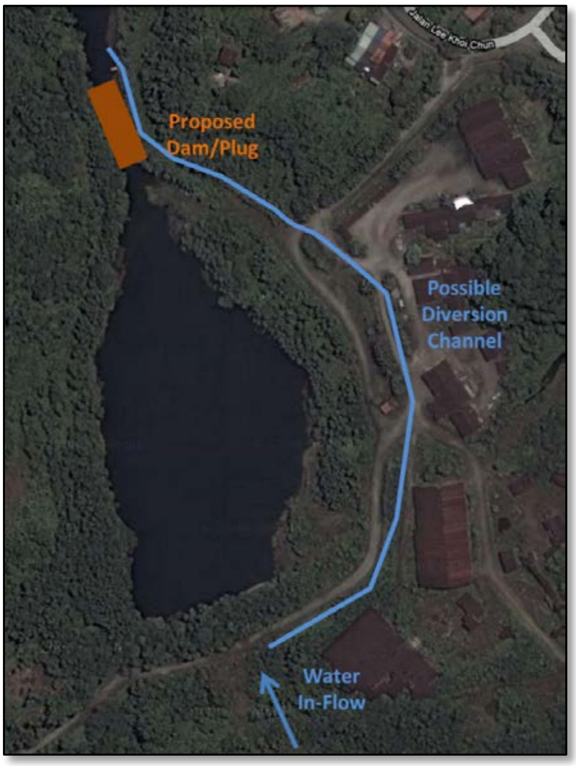


Figure 16-35 - BYG-Krian Indicative Diagram of Site Preparation Work

16.5. Overburden Removal & Storage

At Jugan the orebody outcrops and in fact Jugan Hill is the orebody. Where ore is not exposed there is a surficial layer of soil or organics and vegetation. This material and vegetation can be removed very easily and quickly allowing for the almost immediate extraction of ore. Jugan Hill has the benefit of no pre-strip and the associated capital costs of this and very good strip ratios initially. Overburden removal is likely to be conducted by local earth moving or civil contractor's who have good experience with this work and the site materials in question. The top twenty (20) metres or so of the hill can then be lowered but cutting the hill down progressively.

Once the hill removal reaches the same elevation as the surrounding land the overburden from the surrounding areas will need to be removed at a suitable point around this time. All soils will be removed first and stored in the soil storage landform for use in ongoing and final rehabilitation work. Any clays encountered can ether be stockpiled for use or used directly in the TSF lining, waste landform lining or other use within the operation. Vegetation removed for mining or for construction will be mulched/composted and stored for ongoing or final rehabilitation work or as feed to the proposed nursery.

When the mining at Jugan is coming to a close and operations are due to begin at BYG-Krian, this site will need to be prepared. As the porposed site is already a water-filled historic pit (see previous section) there will be no immediate need for overburden removal. This is compounded by the fact that the surrounding area is covered by historic mullock which is averaging a few grams/tonne Au. These top few metres can be stripped and processed. Obviously, any vegetation or infrastructure will need to also be removed at this time. Again due to the situation this site also does not have a large or significant pre-strip requirement, which also positively impacts on the project, other than the de-watering required and associated minor civil works (dam/plus and water channel).

16.6. Waste Mining & Storage

At Jugan, waste mining will not commence immediately as the hill is mined downwards to the surrounding land level, other than some surficial material or low grade ore below cutoff which will go to waste (see Ore Mining & Grade Control section). Once the open pit has progressed downwards sufficiently the first overburden and waste cutback will be undertaken.

As discussed above the overburden materials will be stored as required. The waste rock is planned to be used for the containment bund for the TSF. The TSF bund is planned to be built in stages and the waste material mined is scheduled for this construction. The TSF stages are covered in more detail in *Section 24 – Other Information* & *Section 18 - Infrastructure*.

Any waste in excess of the TSF bund requirement will be placed on the waste disposal landform. The TSF bund, less clay lining, is 3.73 M m^3. The total waste produced is 10.25 Mm^3 and therefore the remaining 7.92 Mm^3 will need to be catered for in the designated waste landform. Suitable factors for bulking have been applied. A placed, engineered and compacted waste material bulking factor of 1.1 has been applied.

Figure 16-36 – Jugan: Pit, TSF, Waste Landform & Mine Infrastructure below shows a plan view of the proposed TSF and waste landform position and size at Jugan, along with the pit and infrastructure. *Figure 16-37 - Jugan: 3D View of Pit, TSF, Waste Landform & Mine Infrastructure* shows a 3D view of the same landforms.

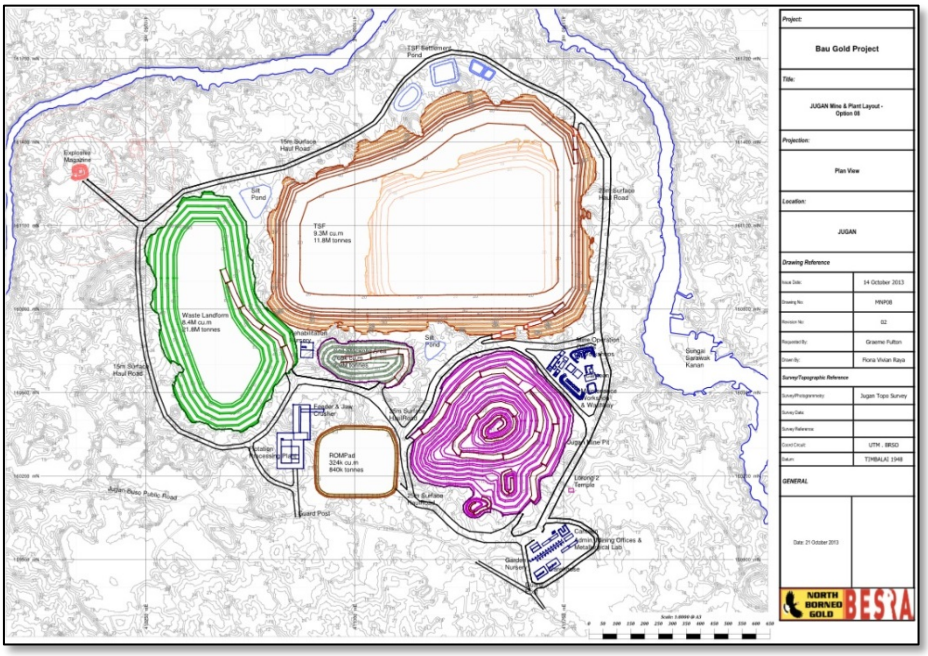


Figure 16-36 – Jugan: Pit, TSF, Waste Landform & Mine Infrastructure

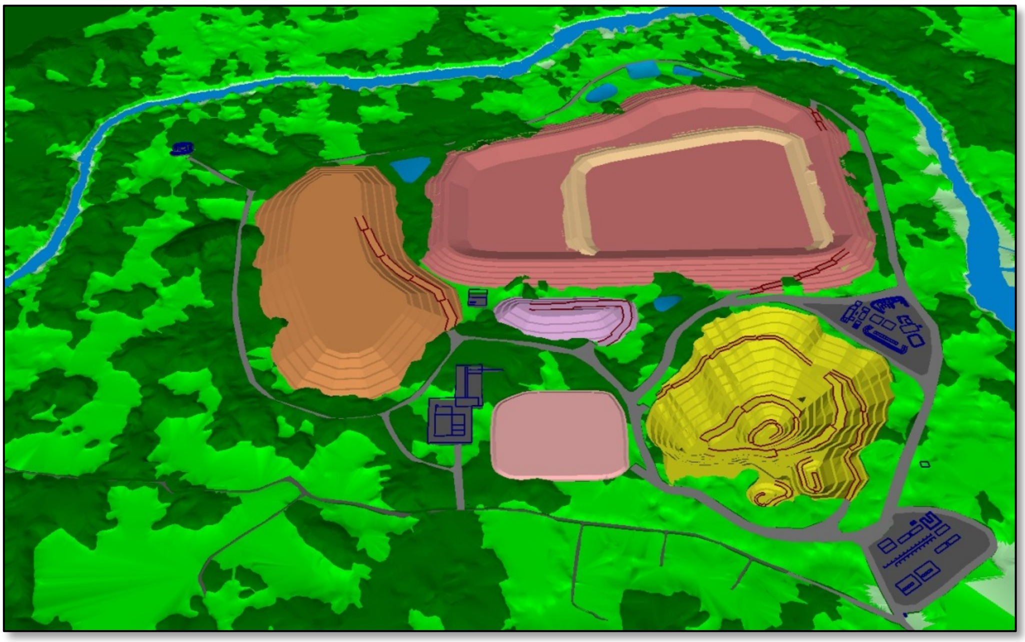


Figure 16-37 - Jugan: 3D View of Pit, TSF, Waste Landform & Mine Infrastructure

There is the possibility of potential extensions to the SW of the Jugan orebody and pit and therefore an alternate mining infrastructure layout has been designed. The alternate design is shown in *Figure 16-38 - Jugan: Alternate Mine Infrastructure Layout* and 3D view in *Figure 16-39 - Jugan: 3D View of Alternate Mine Infrastructure* below.

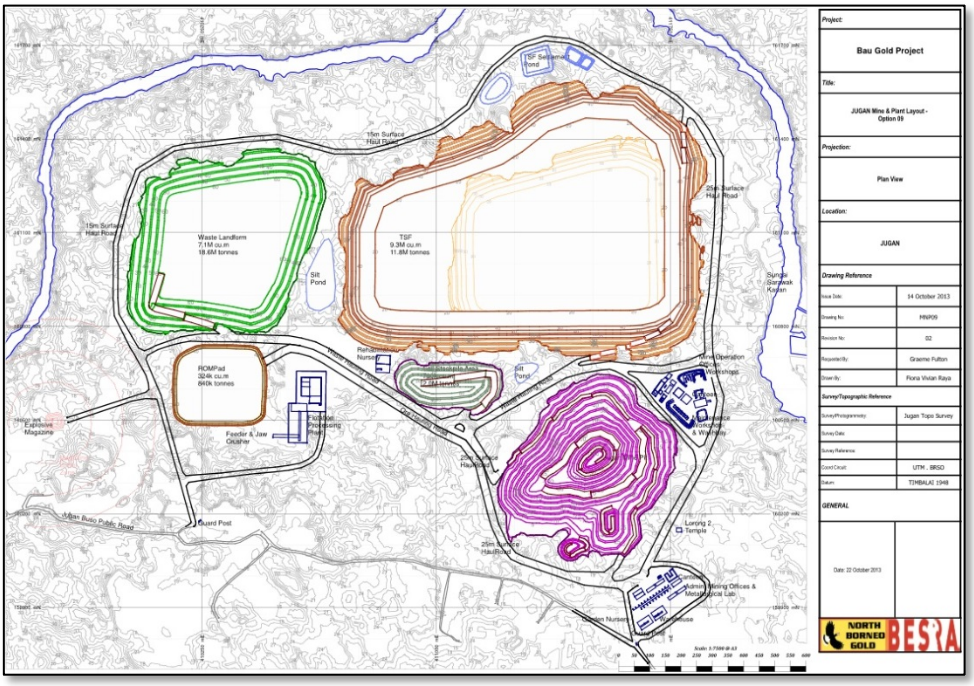


Figure 16-38 - Jugan: Alternate Mine Infrastructure Layout

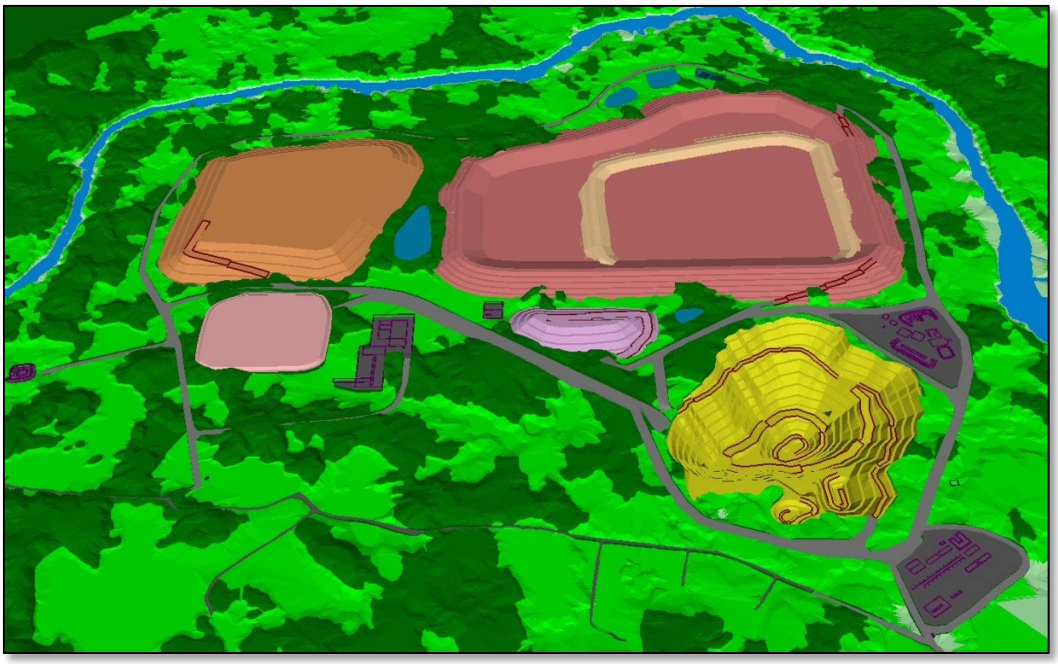


Figure 16-39 - Jugan: 3D View of Alternate Mine Infrastructure

Listed below in *Table 16-34 - Waste Material Balance Summary (Owner-Operator)* is a waste material balance summary showing the construction cut-and-fill, waste production and waste requirements for the TSF. Note: the “resultant expansion factor” is a combination of swell factor and compaction factors.

Period		Volumes of Total Available Material		Resultant Expansion Factor (110%)		TSF Embankment Fill Volume		Total Available Material	Balance - TSF Embankment Construction	Allocation of Waste Material to ELF	
Year	Qtr	m^3	Cumm.	m^3	Cumm.	m^3	Cumm.	m^3	m^3	m^3	Cumm.
	-1	110,667	110,667	121,733	121,733	110,667	110,667	121,733	11,067		
Year-1	1	601,231	711,898	661,354	783,088	601,231	711,898	754,677	153,446		
	2	601,231	1,313,129	661,354	1,444,442	601,231	1,313,129	935,047	333,816		
	3	435,231	1,748,360	478,754	1,923,196	300,616	1,613,745	873,436	572,821	258,302	258,302
	4	379,898	2,128,257	417,887	2,341,083	200,410	1,814,155	852,651	652,241	197,436	455,738
Year-2	5	572,704	2,700,961	629,974	2,971,057	572,704	2,386,859	1,376,404	803,700		455,738
	6	657,692	3,358,654	723,462	3,694,519	657,692	3,044,551	1,653,365	995,673		455,738
	7	576,923	3,935,577	634,615	4,329,134	288,462	3,333,013	1,638,365	1,349,904	423,077	878,815
	8	557,692	4,493,269	613,462	4,942,596	173,077	3,506,090	1,655,673	1,482,596	423,077	1,301,892
Year-3	9	519,231	5,012,500	571,154	5,513,750	220,145	3,726,235	1,409,551	1,189,406	856,745	2,158,638
	10	923,077	5,935,577	1,015,385	6,529,134		3,726,235			1,015,385	3,174,022
	11	1,153,846	7,089,423	1,269,231	7,798,365					1,269,231	4,443,253
	12	1,153,846	8,243,269	1,269,231	9,067,596					1,269,231	5,712,484
Year-4	13	1,153,846	9,397,115	1,269,231	10,336,827					1,269,231	6,981,715
	14	850,082	10,247,197	935,091	11,271,917					935,091	7,916,805
	15										
	16										
Year-5	17										
	18										
	19										
	20										
TOTAL		**10,247,197**		**11,271,917**		**3,726,235**				**7,916,805**	
									With 10% Extra Allowance Volume	**8,708,486**	

Table 16-34 - Waste Material Balance Summary (Owner-Operator)

For the contract-mining option the waste material balance is shown in *Table 16-35 - Waste Material Balance Summary (Contract-Mining)* below.

Period		Volumes of Total Available Material		Resultant Expansion Factor (110%)		TSF Embankment Fill Volume		Total Available Material	Balance - TSF Embankment Construction	Allocation of Waste Material to ELF	
Year	Qtr	m^3	Cumm.	m^3	Cumm.	m^3	Cumm.	m^3	m^3	m^3	Cumm.
	-1	110,667	110,667	121,733	121,733	110,667	110,667	121,733	11,067		
Year-1	1	601,231	711,898	661,354	783,088	601,231	711,898	754,677	153,446		
	2	601,231	1,313,129	661,354	1,444,442	601,231	1,313,129	935,047	333,816		
	3	435,231	1,748,360	478,754	1,923,196	300,616	1,613,745	873,436	572,821	258,302	258,302
	4	379,898	2,128,257	417,887	2,341,083	200,410	1,814,155	852,651	652,241	197,436	455,738
Year-2	5	572,704	2,700,961	629,974	2,971,057	572,704	2,386,859	1,376,404	803,700		455,738
	6	657,692	3,358,654	723,462	3,694,519	657,692	3,044,551	1,653,365	995,673		455,738
	7	576,923	3,935,577	634,615	4,329,134	288,462	3,333,013	1,638,365	1,349,904	423,077	878,815
	8	557,692	4,493,269	613,462	4,942,596	173,077	3,506,090	1,655,673	1,482,596	423,077	1,301,892
Year-3	9	519,231	5,012,500	571,154	5,513,750	220,145	3,726,235	1,409,551	1,189,406	856,745	2,158,638
	10	653,846	5,666,346	719,231	6,232,980					719,231	2,877,868
	11	923,077	6,589,423	1,015,385	7,248,365					1,015,385	3,893,253
	12	923,077	7,512,500	1,015,385	8,263,750					1,015,385	4,908,638
Year-4	13	923,077	8,435,577	1,015,385	9,279,134					1,015,385	5,924,022
	14	845,724	9,281,301	930,297	10,209,431					930,297	6,854,319
	15										
	16										
Year-5	17										
	18										
	19										
	20										
TOTAL		**9,281,301**		**10,209,431**		**3,726,235**				**6,854,319**	
									With 10% Extra Allowance Volume	**7,539,751**	

Table 16-35 - Waste Material Balance Summary (Contract-Mining)

The waste material in the pit will be defined by geological mapping and standard grade control processes/procedures and this material will be marked as such if defined as waste or below cut-off.

Initially the waste (and ore) will be free dug by excavators as the material is soft and friable and has been exposed to air. This is particularly the case on Jugan Hill itself. At some depth the ore will need to be ripped by a bulldozer and dug out by excavators. At some depth the waste (and ore) will need to be blasted before being dug out. All costs applied to the waste (and ore) have assumed that it will be blasted and dug. This allows for the worst case scenario and there is potential for cost savings due to the non-blasting extraction method in the early stages.

Waste material mined at the BYG-Krian deposit will need to be stored in a suitable area. The area surrounding the pit is an old mine site area and under a current Mining Lease. The waste disposal landform is 3.01 Mm^3 or 7.84 Mt (2.53 Mm^3 or 5.86 Mt for contractor option). Suitable factors for bulking have been applied. A placed, engineered and compacted waste material bulking factor of 1.1 has been applied.

Figure 16-40 - BYG-Krian: Pit & Waste Landform Design below shows a plan view of the proposed waste landform position and size at BYG-Krian along with the pit. *Figure 16-41 - BYG-Krian: 3D View of Pit & Waste Landform Design* shows a 3D view of the same landforms.

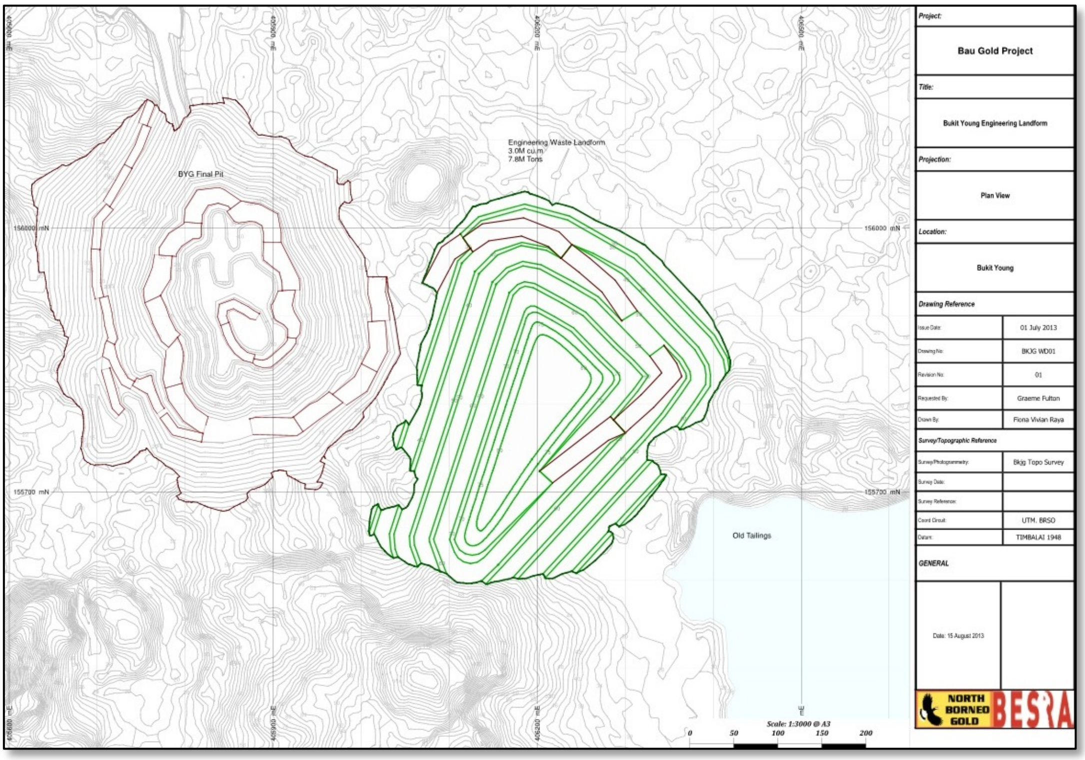


Figure 16-40 - BYG-Krian: Pit & Waste Landform Design

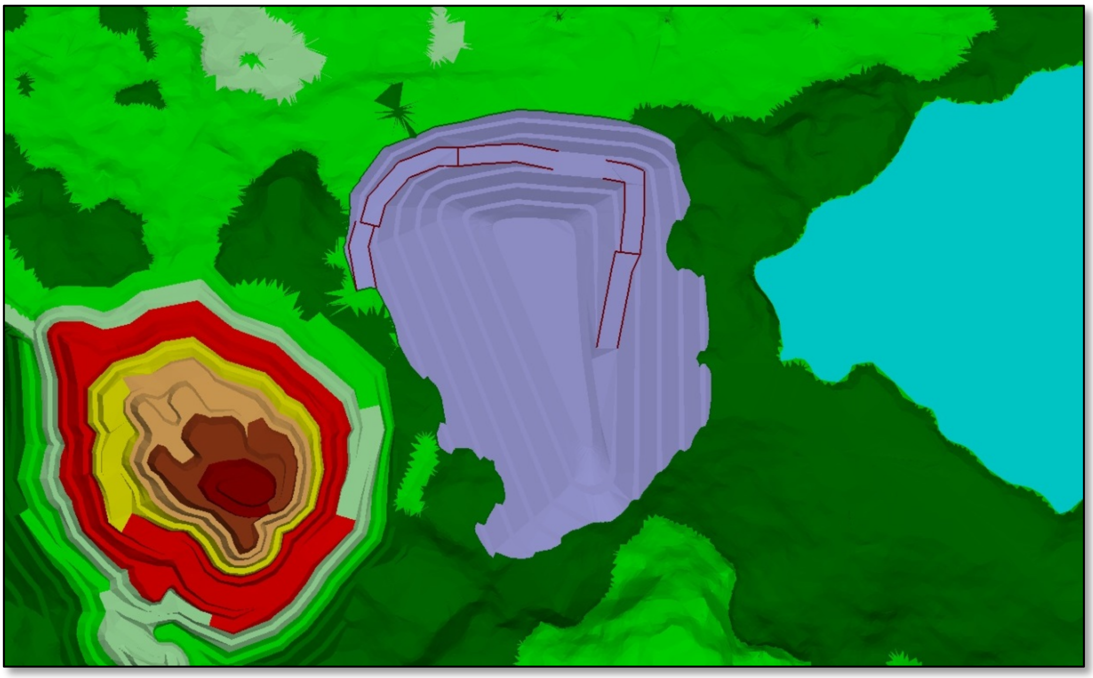


Figure 16-41 - BYG-Krian: 3D View of Pit & Waste Landform Design

All waste material produced will go to the waste landform at BYG-Krian.

16.7. Ore Mining & Grade Control

As for the waste, the ore material will be free dug by excavators as the material is soft and friable and has been exposed to air. This is particularly the case on Jugan Hill itself. At some depth the ore will need to be ripped by a bulldozer and dug out by excavators. At some depth the ore (and waste) will need to be blasted before being dug out. All costs applied to the ore (and waste) have assumed that it will be blasted and dug. This allows for the worst case scenario and there is potential for cost savings due to the non-blasting extraction method in the early stages.

The hill strip will be done in slices or flitches probably around 2 to 2.5 metres in height with suitable ramp access. The slices will be progressively downwards with the effective bench being the full hill area at that elevation. Thereafter, the ore will be mined in benches with similar slices/flitches. Blasted ore will also be dug in a similar configuration.

The ore material in the pit will be defined by geological mapping and standard grade control processes/procedures and this material will be marked as such if defined as waste or below cut-off. Grade control will be by means of blasthole sampling where blastholes are being drilled and by trenching when ore/waste is being free dug or ripped-dug. The samples will be analysed and the ore and waste zones modelled and delineated in the pit. Pit mapping will also be undertaken by the geology department to help in the ore and waste delineation as well as for structural purposes.

Ore material will be sent to the ROM pad via the appropriate routing with the waste going to the TSF construction area or the waste landform depending upon the material balance and operational requirements. Material below cutoff will either be sent to the low grade stockpile area for blending or go immediately to waste landform/TSF.

Operating bench widths are envisaged to be typically 15-20 metres in width with final bench widths of 5 metres. This allows for adequate equipment operation particularly with respect to the loading of ore and waste. Bench heights are a maximum of 15 metres with face angles as defined in the RMR model and described in the appropriate sections.

Grade control and the handling of ore and waste will be the same for BYG-Krian as is for Jugan.

At this stage it is envisaged that due to the limited reserves and short pit life that the ore will be transported to the Jugan plant facility by road transport. The ore mined from the pit will either be loaded onto a stockpile pad near the pit (historic stockpile area) or then rehandled into road trucks for transport to Jugan. Alternately, the ore could be loaded directly onto road trucks for direct transport. This will be determined nearer the time as there is a significant Inferred Resource below, and adjacent to the pit which would increase the reserve three- to four-fold and extend the mine life to a stage whereby the plant at Jugan may be relocated to the BYG-Krian area. This is a number of years down the track and needs to be reviewed regularly as the project progresses and changes to the situation arise.

16.8. Mining Operations

16.8.1. Drilling & Blasting

Listed below in this section are the detailed calcuations, parameters and tables for drilling and blasting at Jugan. The equipment for drilling and blasting is detailed in a later section on mining equipment. Due to the nature of the deposit and the exposed ore on Jugan Hill it is anticipated that the initial mining will be either free digging and/or rip and dig. As the pit progresses downwards the rock will need to drilled and blasted.

16.8.1.1. Drill & Blast – Background & Calculations

The main reference for the Blast Design is Chapter 13 (Blasting Rock) of the 6th edition of Construction Planning, Equipment and Methods by Dr. Ibrahim Assakaff, A.J. Clark School of Engineering, University of Maryland. Other references include Lilly (1973-1986), Cunningham (1987), Kutznetsuv (1973) and Kuz-Ram (2003).

The objective of this study is to design the blasthole layout and calculate the amount of explosives required for blasting the rock. Blast Design is not an exact science but by considering rock formation and specific properties of explosives, it is possible to produce the desired blast result.

The basic parameters required in the blast design are as follows;

1. Burden – is the distance to the free face of the excavation and it's the most critical dimension in blast design.
2. Stemming – is the adding of an inert material, such as drill cuttings on top of the explosive in a blasthole for the purpose of confining the energy of the explosive.
3. Sub-drilling – a shot will normally not break to the very bottom of the blasthole. To achieve a specified grade, one will need to drill below the desired floor elevation. This portion of the blasthole below the desired final grade is termed `subdrilling'.
4. Blasthole size (diameter) – the diameter of blasthole will affect blast considerations concerning fragmentation, air blast, flyrock and ground vibration.
5. Stiffness Ratio – the second mechanism of rupture is flexural and stiffness ratio (SR) affects several critical blasting considerations, which are fragmentation, air blast, flyrock and ground vibration.
6. Spacing – proper spacing of blastholes is controlled by the initiation timing and the stiffness ratio. When holes are spaced too close and fired instantaneously, venting of the energy will occur with resulting air blast and flyrock. When the spacing is extended, there is a limit beyond which fragmentation will become harsh. Delayed initiation is considered in this study.
7. Powder Column – the powder column length is the total drillhole length less stemming. The amount of explosive required to fracture a cubic meter of rock is relative to the powder column length which is a measure of economy of the blast design.

8. Powder Factor – is the ratio of the total weight of explosive in powder column length to the total volume of rock fractured by one blasthole under the pattern area to a depth of bench (L).
9. Bench Height (L) – a design specification resulting from the open pit design considering geotechnical data and other geological features of the ore deposit.

Empirical Formula

The empirical formulae derived by experts from the various references were used in this study.

1. **BURDEN**

B1 - based on specific gravity of rock and explosive

$$\boldsymbol{B1} = ((2SG_e/SG_r)+1.5)*D_e)$$

B2 - based on relative bulk energy of explosive (REE)

$$\boldsymbol{B2} = (\mathbf{0.67} * \boldsymbol{De} * (\sqrt[3]{(St_v/SG_r)})$$

Bc – Burden Corrected, correction factors applied for specific geological conditions

B1c = B1 x Kd x Ks
B2c = B2 x Kd x Ks

Where;

B	=	burden, ft
SGe	=	specific gravity of explosive, **Table_7.5, Table_7.6 & Table_7.7**
SGr	=	specific gravity of the rock, in **Table_7.1**
De	=	diameter of the explosive (inches), **Tables_7.5, 7.6 & 7.7**
Stv	=	relative bulk strength compared to ANFO, **Table_7.5**
Kd	=	correction factor for rock deposition, in **Table_7.2**
Ks	=	correction factor for rock structure, in **Table_7.2**
L	=	bench height

2. **STEMMING**

T	=	Stemming depth
T	**=**	**0.7 x B**

Under normal conditions and good stemming material (drill cuttings)

3. **SUBDRILLING**

J	=	Subdrilling depth
J	**=**	**0.3 x B**

Subdrilling represents the depth required for explosive placement not a field drilling depth

4. Blasthole Size (diameter)

Blasthole Diameter = 76mm to 89mm (3 to 3.5 inches) for weak/soft rocks like shale and schist
Sandvik recommended 89mm to 127mm for this project

5. STIFFNESS RATIO

SR = Stiffness Ratio, bench height divided by the burden distance
SR = L/B
For control of fragmentation, air blast, flyrock and ground vibration

6. SPACING

S = Spacing
6.1 For instantaneous initiation with SR greater than 1 but less than 4
S = (L + 2B)/3
6.2 For instantaneous initiation with SR equal to or greater than 4
S = 2B
6.3 For delayed initiation with SR greater than 1 but less than 4
S = (L + 7B)/8
6.4 For delayed initiation with SR equal to or greater than 4
S = 1.4B
Spacing in the field should be within plus or minus 15% of the calculated value

7. Powder Column

PC = Powder Column
PC = L + J = T

8. Powder Factor

PF = Powder Factor
PF = Density of explosive x PC/BCM

9. Bench Height

L = Bench Height
= 10 meters and 15 meters from open pit design
= Flitch Height at 5 meters

10. Blastability Index (Lilly 1986)

BI = (RMD + JPS +JPO + SGI + H)/2

Where;

RMD – rockmass description
JPS – joint plane spacing
JPO – joint plane orientation
SGI – specific gravity influence = (25 x SG) -50
H - MOH's hardness
Values are shown in Table 3.4

11. Kutznetsuv Formula (1973) and Kuz-Ram Fragmenatation Model

$$X_m = A * K^{-0.8} * Q^{\frac{1}{6}} * \left[\frac{115}{RWS}\right]^{\frac{19}{20}}$$

Where
X_m = mean particle size of muckpile, cm
A = rock factor
K = powder factor, kg explosive per m^3 of rock
Q =mass of explosive in the hole, kg
RWS = weight strength relative to ANFO

Rosin-Rammler equation:

$$Y_x = \exp\left[-0.693\left(\frac{x}{x_m}\right)^n\right]$$

Y_x = mass fraction retained on screen opening, x
n = Uniformity index

The above formula was originally developed by Kutznetsuv (1973) and further developed by Cunningham (1987).

UNIFORMITY INDEX (Kuz - Ram Model)

$$n = \left[2.2 - \frac{14B}{d}\right]\sqrt{\left(\frac{1+\frac{S}{B}}{2}\right)}\left\{1-\frac{W}{B}\right\}\left[abs\left\{\frac{CCL-BCL}{L}\right\}+0.1\right]^{0.1}\frac{L}{H}$$

B = Burden, m;
S = Spacing, m;
d =hole diameter, mm
W = standard deviation of drilling precision, m;
L = Charge length, m
CCL = column charge length, m;
BCL = bottom charge length, m;
H = bench height, m

16.8.1.2. Drill & Blast – Design

Table 16-36 - Drill & Blast Summary Calculations to determine SMU below is a summary table and calculations for the drilling and blasting design derived for the Standard Mining Unit (SMU). The detailed drilling and blasting calculations and tables are listed in *Appendix A16-7*.

Drilling & Blasting Data from Design	SMU		Annualised	
Diameter of Blasthole	89	mm	89	mm
Designed Burden for ore - using Emulsion expl	3.2	m	3.2	m
Designed Burden for waste - using Emulsion expl	3.3	m	3.3	m
Designed Spacing for ore	4.0	m	4.0	m
Designed Spacing for waste	4.6	m	4.6	m
Ave. powder factor for ore (from Table 3.2)	0.490	kg/m^3	0.490	kg/m^3
Ave. powder factor for waste (from Table 3.2)	0.409	kg/m^3	0.409	kg/m^3
Average BCM per hole - Ore (from Table 3.2)	127.4	m^3	1,114,495.2	m3
Emulsion per Blasthole - Ore	62.43	kgs	546,102.65	kgs
Average BCM per hole - Waste (from Table 3.2)	151.6	m^3	1,326,196.8	m3
Emulsion per blasthole - Waste	62.0	kgs	1,056,493.0	kgs
ANFO - bulk price (from Table 5)	1.48	per kg	1.48	per kg
EMULSION - bulk price (from Table 5)	2.35	per kg	2.35	per kg
ANFO & Emulsion - average price	1.92	per kg	1.92	per kg
Number of holes/drilling round - ore	24	holes	8,748	holes
Number of holes/drilling round - waste	24	holes	17,039	holes
Depth of hole for 10m bench	11	m	11	m
Drilled meters per round - ore	264	m	96,228	m
Drilled meters per round - ore	264	m	187,429	m
DTH/Rotary Drill Rate (average for shale)	105	sec/m	105	sec/m
Total Drilling Time (24 holes)	7.7	hrs	8,273.33	hrs
Average BCM per Round - Ore	3,057.6	bcm ore	1,114,504	bcm ore
Average Tonnes per Round - Ore	8,010.9	tonnes ore	2,920,000	tonnes ore
Average BCM per Round - Waste	3,638.4	bcm waste	2,583,077	bcm waste
Average Tonnes per Round - Waste	9,459.8	tonnes waste	6,716,000	tonnes waste
Total Production (Ore + Waste)			9,636,000	tonnes

Table 16-36 - Drill & Blast Summary Calculations to determine SMU

16.8.1.3. Drill & Blast – SMU

The 24-holes Staggered Pattern (see *Table 16-37 - 24-Holes Staggered Drilling Pattern*) drilling layout would be appropriate as the smallest mining unit. The minimum area of this layout is 10 metres by 24 metres, using a burden design of 3.3 m and spacing of 3.5 m. This area will be sufficient for one loading station to serve a fleet of 1 shovel and 4 to 5 trucks. The truck width is 4.78 metres and 8.8 metres long, whilst the shovel is 5.3 metres by 7.85 metres excluding the boom and dipper.

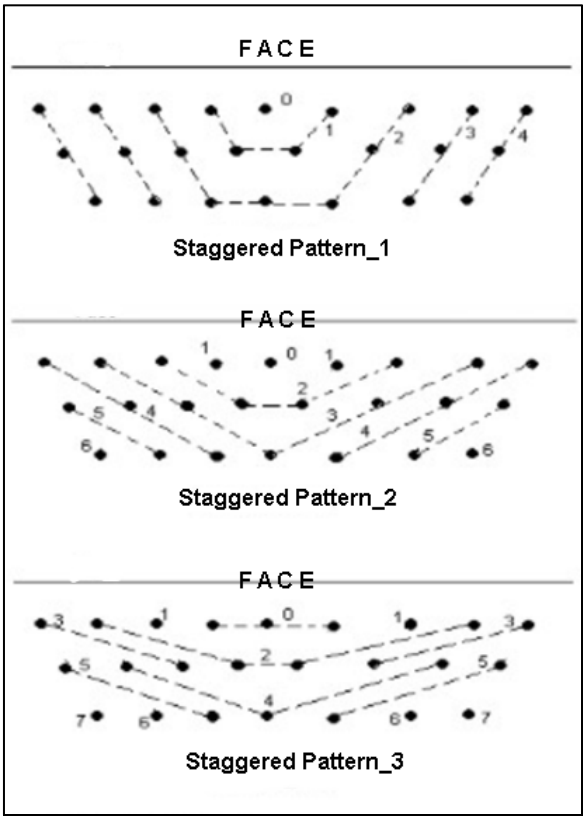


Table 16-37 - 24-Holes Staggered Drilling Pattern

16.8.2. Loading & Hauling

Loading and hauling will be conducted using a fleet of 56-tonne capacity trucks (for both ore and waste) in combination with a 7m^3 capacity hydraulic shovel. It is estimated that the 8,000 tpd operation will require 2 shovels and 9 trucks, with a smaller wheel loader for flexibity in-pit and another loader in the stockpile area.

A comparison and evaluation was undertaken to determine the best truck and shovel combination. The following tables contain the various parameters, including specifications and cycle times used in the calculation of the loader and truck match-up simulation and the resulting productivity data for the selected equipment. *Table 16-38 - Jugan Ore & Waste Specific Gravity & Swell* to *Table 16-52 - Waste Truck Unit Cost and Consumption Rates* contain the aforementioned values and calculations.

Material Properties Jugan Ore & Waste	
Specific Gravity (average)	2.62
Swell in Dipper (Shovel)	1.20
Swell in Tray (Truck)	1.25
Average Haul Distance for Ore	1600 m
Average Haul Distance for Waste	1200 m

Table 16-38 - Jugan Ore & Waste Specific Gravity & Swell

Loader Base Data	CAT 6015 FS		
No. of Units	2		
Standard dipper	7	m^3 (heap)	
Actual dipper	7	m^3 (heap)	
Rated Load Limit	15	t	
Max cut height	11	m	
Dump radius	10.5	m	
Std Fill Factor	100	%	
90° dipper cycle	30	secs	
Indicated Loader Eff	94	%	
Indicated swing cyle	32	secs	
actual swing cycle	32	secs	
		calc	actual
Min panel width for DSL	m	28	22
Panel width	m		30
Wall height (max)	m		15
Fill Factor	%	100	95%

Table 16-39 - Loader Base Data for Base Case

Operating System	
SSL	Single Side Loading
DSL	Double Side Loading
MDB	Modified Drive By

Table 16-40 - Loading System

Bunching Curve Formula Loader Efficiency = c1 + (c2 * MF) + (c3 * MF^2)			
Bunching Character (1, 2, 3)			
1	Severe - multiple loaders/multiple dumps/no dispatch system		
2	Average - multiple loaders/dumps with dispatch		
3	Light - one loader per truck fleet		
Standard Constants			
	1	2	3
c1	-0.0167	-0.0674	-0.2187
c2	1.1315	1.3368	1.7385
c3	-0.3156	-0.4184	-0.6192

Table 16-41 - Bunching Characters for Loader & Truck Match-up

Diggability Modifier	Production
Index	%
1	40%
2	50%
3	70%
4	90%
5	100%

Table 16-42 - Diggability Index & Prodction

Operator Skills Modifier	Production
Skill Index	%
1	40%
2	50%
3	70%
4	90%
5	100%

Table 16-43 - Operator Skills Index & Production

Truck Base Data	CAT 772G	
Standard tray	31.2	m^3 (heap)
As Modified	28.08	m^3
Rated Load Limit	56	t
Operating width	3.69	m
Turning radius	10.05	m

Table 16-44 - Truck Base data for Base Case

Truck Cycle Data		loaded	empty
Ave haul distance	m	1,400	1,400
Ave haul time	secs	232.5	179.5
Ave speeds	kph	21.7	28.1
Ave spot @ loading/dumping	secs	15	30
Ave dump time	secs	18	
Ave wait @ loading/dumping	secs	30	30

Table 16-45 - Truck Cycle Data for Base Case

Annualised Operating time		Loader	Truck
Scheduled Hours	hrs	8,352	8,352
Available Hours - AT	hrs	7,284	7,236
Utilised Hours - UT (or OH)	hrs	5,920	5,906
Possible Operated OT	hrs	4,648	4,634
Net Operated Time OT (DOH)	hrs	4,220	4,142
Equipment waiting	hrs	428	492

Table 16-46 - Annualised Operating Time

Base Case Parameters	By Volume	By weight	Fill Truck
Operating System	DSL		
Bunching character	3		
Dipper load	6.65 cm	14.52 t	Yes
Passes to fill tray to	4.05	3.86	4
Resulting truck load	26.72 cm	56 t	21.37 bcm
Operator Skills Factor	4		
Blast Diggability Factor	4		

Table 16-47 - Parameters for Loader & Truck Match-up

Total Cyle Time	Units	Loader	Truck
Loader 4 passes	secs	128	158
Modified for skill &	secs	158	195

Total cylce time	mins	2.63	11.67
instantaneous rate		22.8	5.1
Production Rate	t/unit/OT	1,275.8	288.0

Table 16-48 - Total Cycle Time for Loading & Hauling

Fleet Productivity							
No. of Loader	2	2	2	2	2	2	2
No. of Trucks	6	7	8	9	10	11	12
Match Factor	0.677	0.790	0.903	1.016	1.129	1.242	1.354
Loader Effy (%)	67.44%	76.82%	84.43%	90.83%	95.46%	97.58%	100.00%
Truck Effy (%)	99.59%	97.23%	93.51%	89.41%	84.58%	78.59%	73.83%
Net Fleet Effy (%)	67.24%	76.59%	84.18%	90.56%	95.18%	97.28%	99.70%
Fleet Cap (Mtpa)	7.97	9.08	9.98	10.74	11.29	11.54	11.82
Annual Loader Capacity per unit				5,369,750			
Annual Truck Capacity per unit - ore				1,044,118			
Annual Truck Capacity per unit -waste				1,392,157			

Table 16-49 - Annual Fleet Capacity for Base Case

Loader (Shovel/Excavator)		
No. of Units for Basecase (CAT_6015FS)	2	units
Capacity	7	m3
Unit Cost	1,476,765	US$
Material Cost	60	US$/UT
Fuel Rate	55	litres
Replace Hours	50000	hours
Ratio: MT/AMT (maintenance time)	1.23	
Ratio: Avail Time/Utilised Time (AT/UT)	1.25	
Utilised Time (machine hours per yr)	5920	UT hours
Maintenance Time	2103	hours
Annual Production/Unit	5,370,000	tonnes
Operating Cost Allocation (+5% losses)	90%	

Table 16-50 - Loader Unit Cost and Consumption Rates

Rigid Dump Truck for Ore		
No. of Units for Basecase (CAT_772G)	4	units
Capacity	30	m3
Unit Cost	662,903.4	US$
Material Cost	30	US$/UT
Fuel Rate	42	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time (per yr)	5,906	UT hours
Maintenance Time	2,237	hours
Average Hauling Distance	1,600	meters

Rigid Dump Truck for Ore		
Annual Production/Unit	1,044,120	tonnes
Operating Cost Allocation	90%	

Table 16-51 - Ore Truck Unit Cost and Consumption Rates

Rigid Dump Truck for Waste		
No. of Units for Basecase (CAT_772G)	5	units
Capacity	30	m3
Unit Cost	662,903.4	US$
Material Cost	30	US$/UT
Fuel Rate	42	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time (per yr)	5,906	UT hours
Maintenance Time (per yr)	2,237	hours
Average Hauling Distance	1,200	meters
Annual Production/Unit	1,392,200	tonnes
Operating Cost Allocation	90%	

Table 16-52 - Waste Truck Unit Cost and Consumption Rates

Additional truck and shovel calculations, tables and parameters can be found in *Appendix A16-6*.

16.9. Haul & Access Roads

Ore and waste will be hauled up 10% grade ramps within the pit stages. The ramps will be 20 metre wide for for each stage. When the pit gets down to the lower part of the pit the ramp will reduce to 10 metre. A safety bund will be formed along the open side of each ramp to one-half of the wheel height of the largest truck. A spoon drain will be formed on the wall side of each ramp.

Ore haulage from the pit to the ROM will be via the east pit exit, with waste being hauled via the other (west) pit exit. Ore will be hauled 548 metres (1,510 metres for alternate option) along the surface haul roads to the ROM pad. Waste will be hauled 545 metres (536 metres) to the TSF position and then within the TSF to the exact construction delivery point as required. TSF return trip is 1,770 metres (1,741 metres). If waste is not going to the TSF for bund construction then it will be hauled 557 metres (1,068 metres) to the waste landform for placement at the appropriate position. Thereafter, the empty trucks will return to the pit entrances as appropriate or as defined by the management or traffic routing system.

Surface haul roads are 25 metres wide i.e. three times (3x) the maximum truck width with allowance for any windrows and drainage. Single roads are 15m wide or two times (2x) the maximum vehicle width including appropriate allowances as defined previously.

Waste will be hauled up a 20-metre wide, 10% grade road to be formed in the east face of the waste dump. The entry to this ramp is on the southeast end of the dump, and the ramp is inside of the operation to limit the impact of truck noise and dust.

In the case of the TSF, particularly as the containment bund is raised, the waste will be hauled up a 15-metre wide ramp at a gradient of 10 % or less. This ramp is formed on the east face of the TSF containment bund. In-pit ramps will be created at a 10 % gradient and maintained and graded. During the wet season the shale can become slippery and it is envisaged that suitable material will need to be laid on the ramps to ensure suitable rolling resistance and to prevent slippage.

Surface haul roads will be constructed using the local limestone, cambered and graded/maintained regularly. Suitable drainage and culvert istalled as required. During the dry season or when dust is an issue, both in-pit ramps and haul roads, will need to be watered using a watering truck to suppress any dust or dust generation.

The in-pit ramps, TSF bund ramp, waste landform ramp and surface haul roads are shown in *Figure 16-42 - Jugan: Road and Ramp Configuration Option* below, with the alternate option shown in *Figure 16-43 - Jugan: Road and Ramp Configuration Alternate Option*.

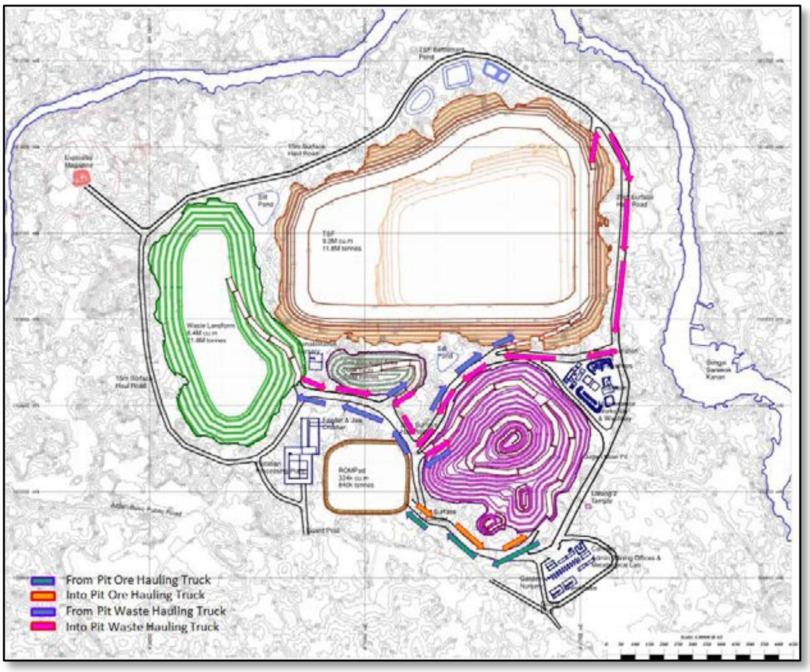


Figure 16-42 - Jugan: Road and Ramp Configuration Option

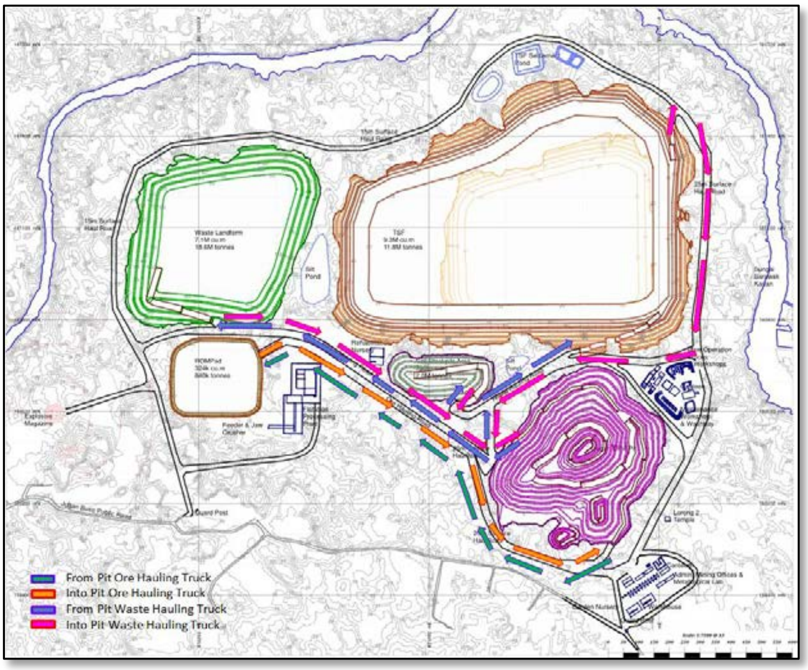


Figure 16-43 - Jugan: Road and Ramp Configuration Alternate Option

For BYG-Krian the ore will be hauled out of pit using the main haul road. Thereafter, the ore will be dumped on a small stockpile for transfer to road trucks for transport to the Jugan processing facility, via the current mine entrance and suitable roads avoiding the town.

Waste will be hauled out-pit and transported the short distance to the waste disposal landform adjacent to the pit. The pit, waste disposal landform and short road network are shown in *Figure 16-43 - BYG-Krian: Road and Ramp Configuration Option* below. A road to the current tailings facility is also included in the configuration.

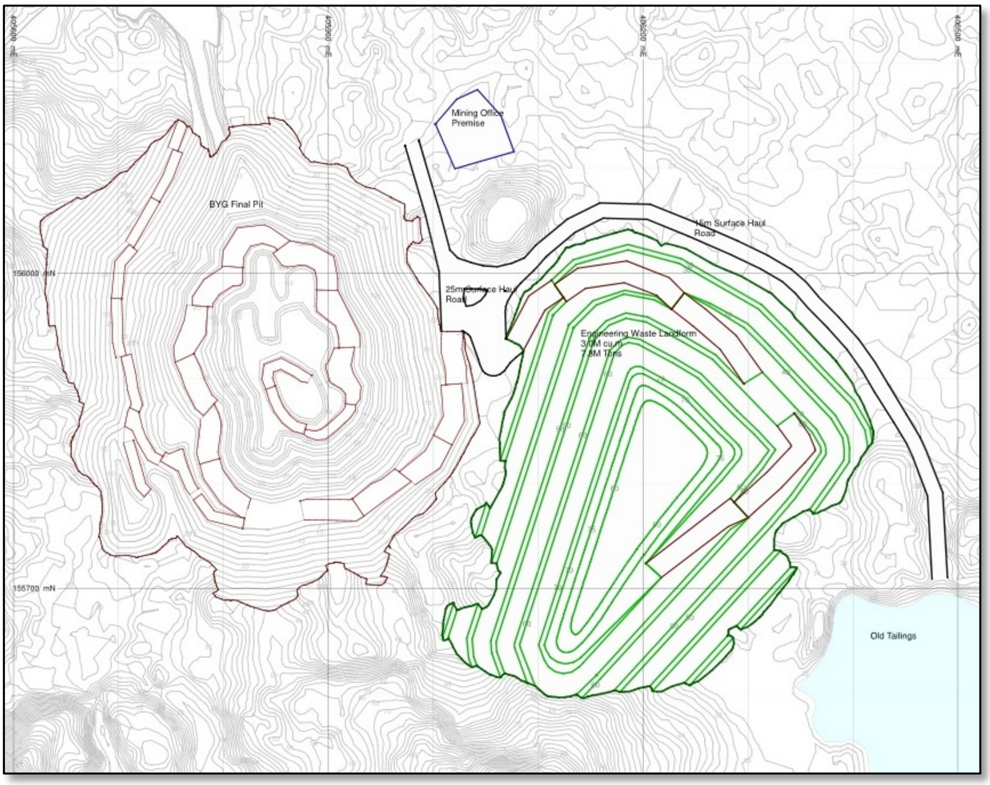


Figure 16-44 - BYG-Krian: Road and Ramp Configuration Option

16.10. Mining Equipment

Depending upon the mining option selected the mining equipment requirement list varies. For the owner-operator the mine will need to purchase the equipment, whereas in the contract-mining option the equipment will be provided by the contractor based on his requirements, but with input from mine management. The equipment configuration has been scoped based on production parameters and associated factors, cycle times, equipment factors and sizing, etc.

The list of equipment required for the 8,000 tpd base case option is listed in *Table 16-53 - Equipment for Base Case (8,000 tpd) Owner-Operator* for the owner-operator option and in *Table 16-54 - Equipment for Base Case (8,000 tpd) Contract-Mining* for the contract-mining option. *Appendix A16-5 Mine Equipment Lists by Production & Mining Type* contains the full lists for each production scenario for owner-operator and contract-mining.

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
2	Production Drill, Sandvik DX800, 76mm to 127mm hole, crawler
2	Hydraulic Shovel, 7m^3, CAT6015/FS
1	Wheel Loader or FEL, 6.4 m^3 for pit operation
1	Wheel Loader or FEL, 6.4 m^3 for Stockpile operation
1	CAT_D10T Dozer with ripper
1	D6W Tractor (CAT_D6R XL)

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
9	Hauling Truck, Rigid Rear Dump CAT_772G
2	Road Grader, CAT_12K
2	Water Truck (10,000 litres)
2	Compactor, CAT CS533E for haul road maintenance
2	Explosive Truck (1000 kg cap) or Mobile Mixing Unit
1	Cable Bolter (Surface Drill + grouting machine combo)
2	Service/Tire Truck (off highway road)
5	4WD LV Toyota Hi-lux
Note: Equipment for Waste Dump operation is not included	
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
4	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
22	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
100	Caplamps with charger
65	Handheld Radios

Table 16-53 - Equipment for Base Case (8,000 tpd) Owner-Operator

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
4	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
22	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
5	Oxy-acetylene Equipment
100	Caplamps with charger
65	Handheld Radios

Table 16-54 - Equipment for Base Case (8,000 tpd) Contract-Mining

It has been assumed that the all activities are incuded in the owner-operator option, except TSF and waste landform construction which are assumed to be contracted out. In the contract-mining option all work is assumed to have been contracted out. The equipment configuration calculations, including haulage, cycle times, availability, utilisation, capacities, etc. are listed in the tables and calculations in *Appendix A16-6 Mine Equipment Selection – Calculations & Parameters*.

16.11. Mining Manpower

Listed in *Table 16-55 - Manpower for Base Case (8,000 tpd) Owner-Operator* below is the planned manpower list for the mining and associated operations (excluding plant labour) for the 8,000 tpd base case scenario.

Manpower or Parameter Description	No.	Unit	Local Staff	Expat Staff
Operating Time Parameters				
Operating Period (Calendar Days - Jan to Dec)	1	year		
Days in Period	365	days		
Days operated	7	days		
Shifts per day	3	shifts		
Operating Hours per day	24	hours		
Manpower for Base Case_8000 TPD Mining				
Direct Labour - Pit Operations:				
Equipment Operators (drivers & operators) Based on 85% equipment availability	72	staff	72	
Shop Mechanics	10	staff	10	
Service Mechanics	4	staff	4	
Shop Electrician	4	staff	4	
Service Electrician	3	staff	3	
Helper/Utility	12	staff	12	
Direct Labour	**105**		**105**	
Manager & Supervision Staff Labour:				
Mine Manager (Expat)	1	staff		1
Mine Shift Foreman	3	staff	3	
Planning Engineer	1	staff	1	
Shift Supervisor	8	staff	8	

Manpower or Parameter Description	No.	Unit	Local Staff	Expat Staff
Pit Geologist	2	staff	2	
Resource/Reserve Geologist	1	staff	1	
Geotech Engineer	1	staff	1	
Chief Surveyor	2	staff	2	
Safety Manager	1	staff	1	
Safety Supervisor	3	staff	3	
Fleet Maintenance Manager	1	staff		1
Mechanical Engineer	1	staff	1	
Maintenance Supervisor	4	staff	4	
Maintenance Planner	1	staff	1	
Electrical Engineer	1	staff	1	
Electrical Supervisor (maint)	3	staff	3	
Warehouse Manager	1	staff	1	
Warehouse Supervisor	2	staff	2	
Environment Engineer	1	staff	1	
Tailings Dam Manager	1	staff	1	
Supervisor (tailings dam)	3	staff	3	
Supervision and Technical Labour	**42**		**40**	**2**
Mine Service Department:				
Safety Officer/Trainer	2	staff	2	
Mine Clerk/Statisticians	2	staff	2	
Grade Control Technician	3	staff	3	
Samplers	6	staff	6	
Surveyor	1	staff	1	
Survey crew	4	staff	4	
Geotech crew	2	staff	2	
Security manager	1	staff	1	
Security guards	12	staff	12	
Mine Services Labour	**33**		**33**	
Engineering Services:				
Engineering Manager (Expat)	1	staff		1
Civil Engineer	1	staff	1	
Mechanical Engineer	1	staff	1	
Electirlcal Engineer	1	staff	1	
Engineering Labour	**4**		**3**	**1**
Admin, PR & HR:				
Mine Admin Manager (Expat)	1	staff	1	
HR Manager	1	staff	1	
PR Manager	1	staff	1	
Office Personnel	9	staff	9	
Admin Labour	**12**		**12**	
Procurement, Accounting & Finance and ICT:				
Procurement Manager (Expat)	1	staff		1

Manpower or Parameter Description	No.	Unit	Local Staff	Expat Staff
Procurement Staff/ Buyer	3	staff	3	
Finance Mgr/Comptroller	1	staff	1	
Accountant	1	staff	1	
Cashier	1	staff	1	
Accounting Staff	2	staff	2	
IT Manager	1	staff	1	
IT Technician	2	staff	2	
PAFI Labour	**12**		**11**	**1**
Tailings Dam Labour:				
Tailings Dam Crew	6	staff	6	
Contractual Expats - Engineers & Geologists:	4			4
TOTAL MANPOWER	**218**			
Local	**210**			
EXPAT	**8**			

Table 16-55 - Manpower for Base Case (8,000 tpd) Owner-Operator

17. Processing & Process Engineering

17.1. Introduction & General

A number of processing methods are being investigated for the Bau Gold Project, as well as a number of plant configurations for the initial mining at Jugan, with BYG-Krian following and future mining in the rest of the extensive goldfield in and around Bau. The gold field stretches for 15 km in a NE to SW direction and is 6-8 km wide perpendicular to this. The initial mining and processing will be at the Jugan Hill deposit which is situated at the NE end of this area approximately 7 kms from Bau town.

The process methods being investigated are Flotation Concentrate production and BIOX, POX or Albion oxidation techniques with the associated tail end gold plant elements. The project has a basecase and a number of alternate options encompassing mining options (owner-operator or contract-mining), production tonnage (4,000 tpd to 12,000 tpd in 2,000 tpd increments), transport options, plant site (near deposit, regional or centralised) and plant type (flotation, BIOX, POX or Albion) options among others.

The basecase is based around the 8,000 tpd (2,920,000 tpa) flotation concentrate plant. All other cases have been investigated and costed as well. This section will cover the elements of the process flow and processing method for the basecase and generic elements (e.g. crushing), and also list and detail the alternate process/processing options, namely the oxidation part of the process.

For the full processing options (BIOX, POX and Albion) the plant tail end aspects (gold room, detoxification, etc.) will also be discussed. There are eighty (80) combinations of mining type (owner-operator or contract-mining), production tonnages (4,000, 6,000, 8,000, 10,000 & 12,000 tpd) and process option (Flotation, BIOX, POX, Albion) for Jugan initially followed by BYG-Krian.

In relation to the basecase option – namely 8,000 tpd (2,920,000 tpa) and flotation concentrate process option – the process plant will likely have the following configuration:

- Crushing
- Grinding/Primary Cyclone
- Cyclone or Continuous Knelson Desliming
- Rougher/Scavenger Flotation
- Regrinding/Secondary Cyclone
- Cleaner Flotation
- Concentrate Filter feed Thickener
- Filter Press
- Reagent mixing, storage and distribution
- Services.
- Control room & Facilities

- o Support Facilities.

The process flow is summarised in the schematic in *Figure 17-1 - 8,000 tpd Flotation Concentrate Process Flow Schematic* below and included in *Enclosure B17-1*.

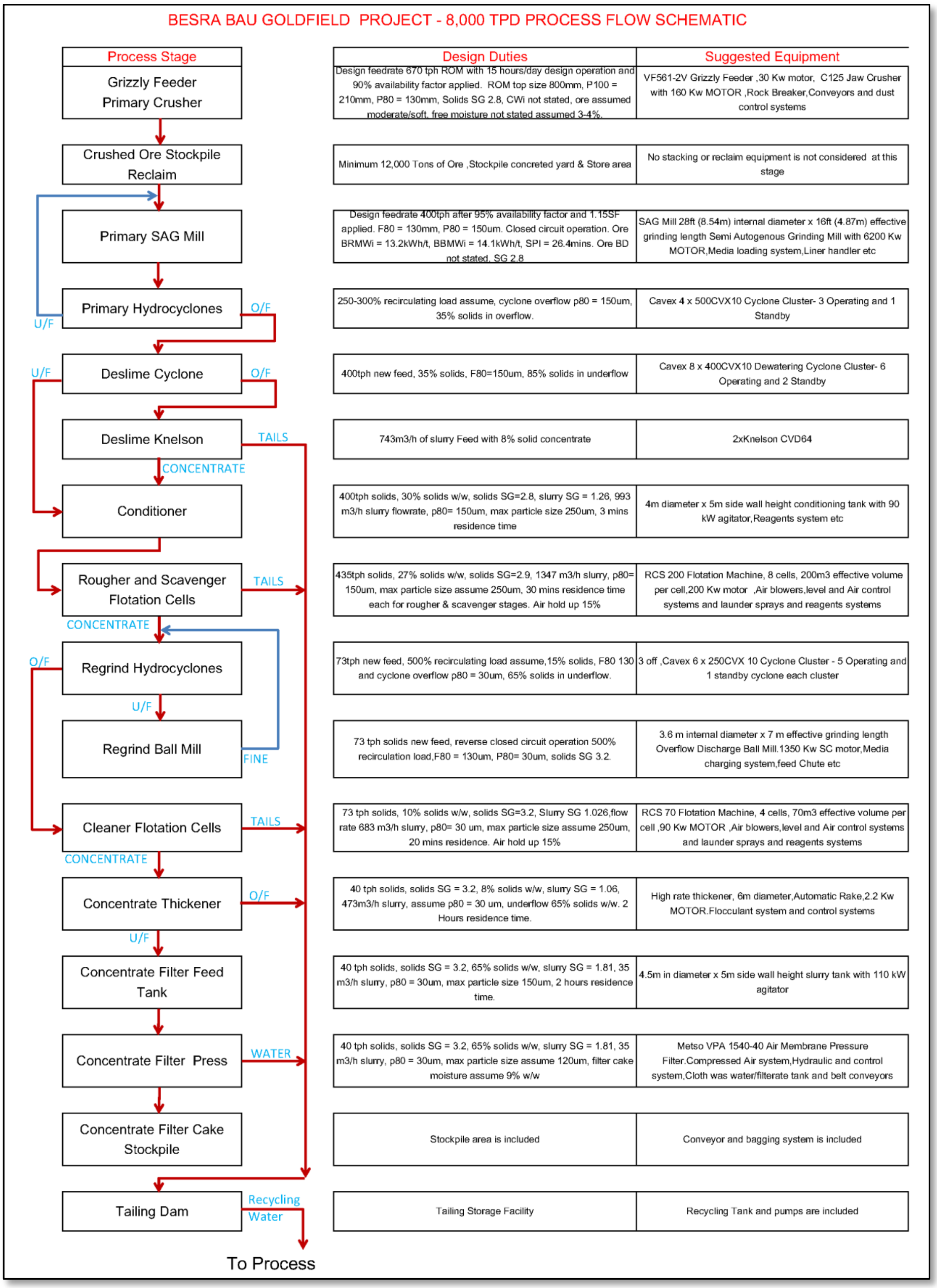


Figure 17-1 - 8,000 tpd Flotation Concentrate Process Flow Schematic

The process flow sheet is also shown in *Figure 17-2 - 8,000 tpd Flotation Concentrate Process Flow Sheet* below and in *Enclosure B17-2*.

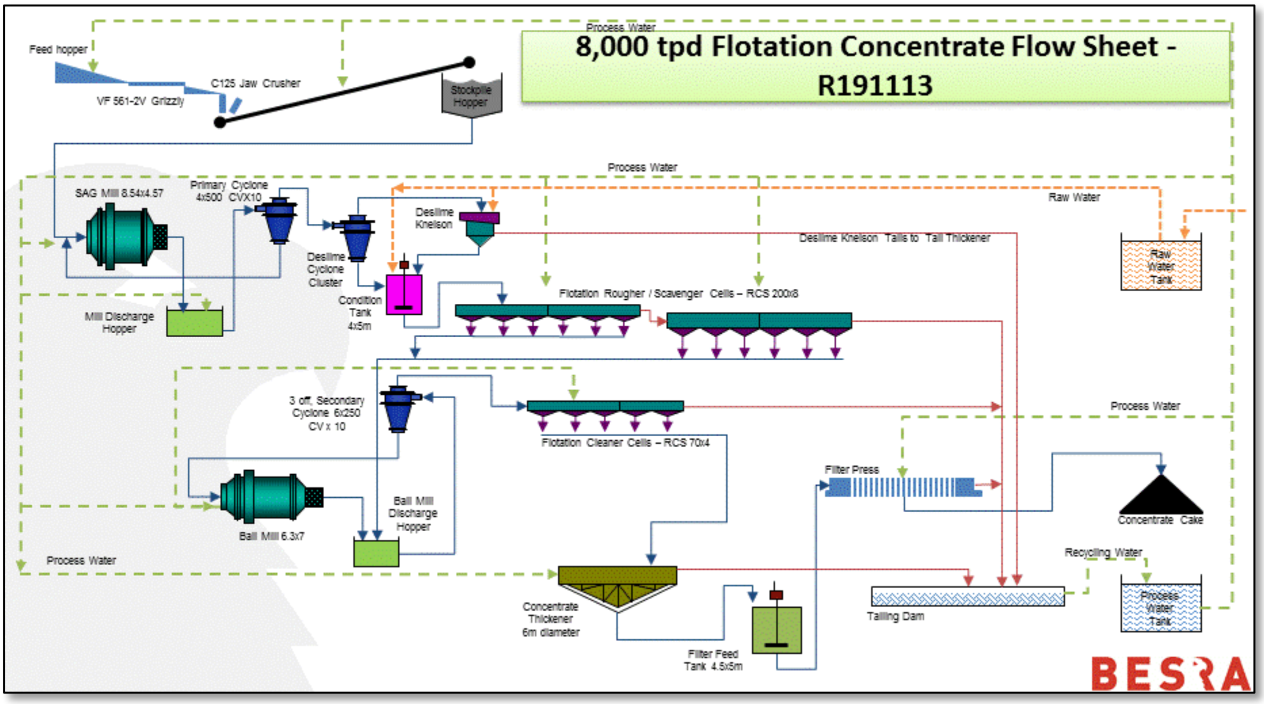


Figure 17-2 - 8,000 tpd Flotation Concentrate Process Flow Sheet

The process plant is planned to produce a filter cake of flotation concentrate and the gold will be recovered from the filter cake in an outside smelting/processing facility. The flotation concentrate filter cake is produced from the ore by a combination of Crushing, Grinding, Cyclone, Desliming, Flotation, Thickening of concentrates and Filter press dewatering. The concentrate will then shipped in suitable containers to the external facility as mentioned previously. The head grade assumed for the process plant is in the 1.5-1.9 g/t Au range (average 1.7 g/t Au). *Figure 17-3 - Jugan: Flotation Plant Layout Configuration* below shows an indicative plant layout configuration.

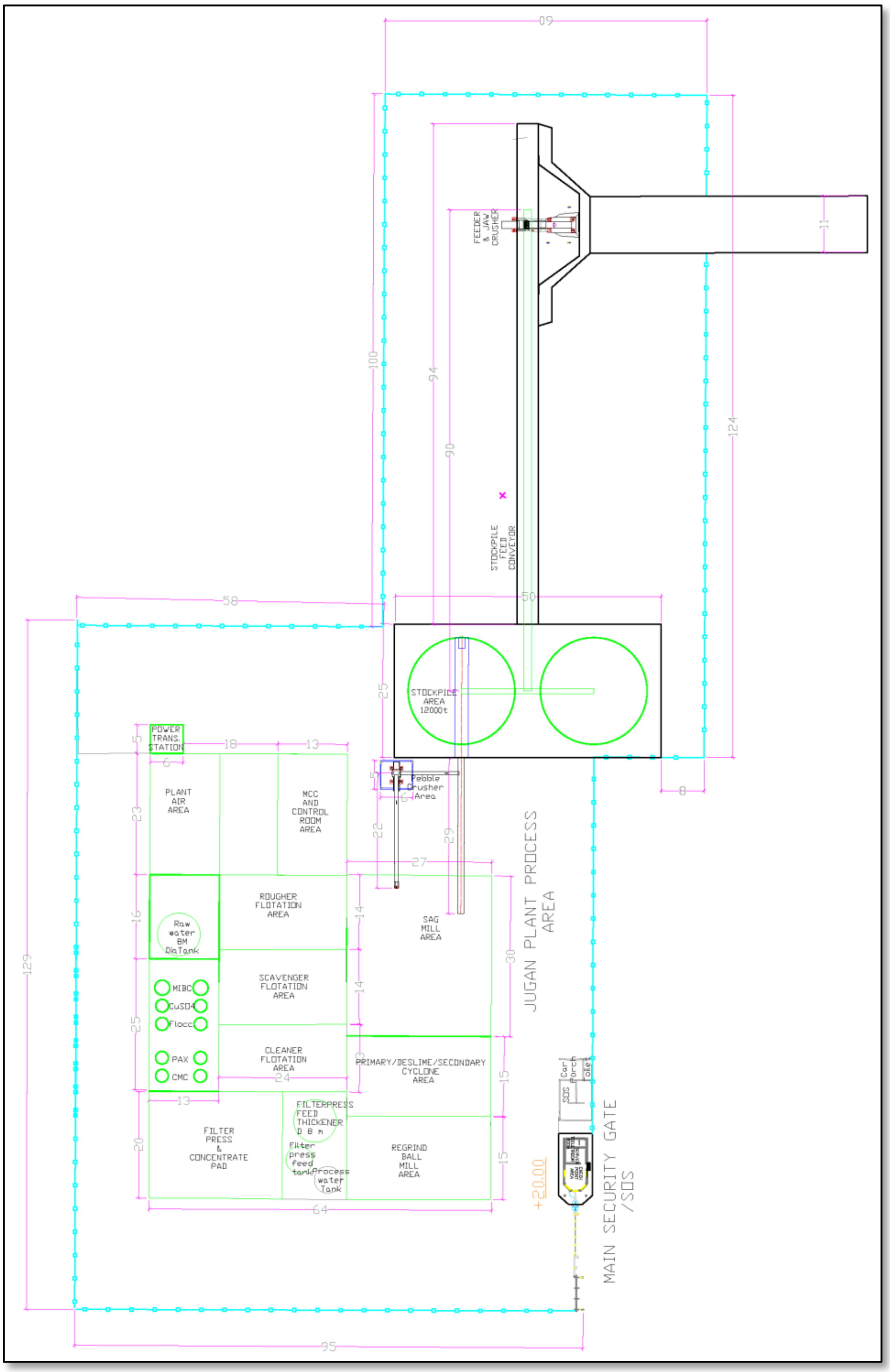


Figure 17-3 - Jugan: Flotation Plant Layout Configuration

The following sections outline the flotation concentrate plant and associated general plant elements. Thereafter, the alternate oxidation process and plant elements will be discussed along with common general plant elements for the tail-end of the process should these be used or implemented.

17.2. Primary Crushing

17.2.1. Area 10: Primary Crusher with Grizzly Feeder

Ore is delivered by truck from the mining operation to a run of mine (ROM) storage pad. A front-end loader OR direct truck transfers ore from the ROM stockpile to a VF 561-2V,30 kW Vibratory Grizzly feeder and +130 mm will fed to a C 125, 160 kW Jaw Crusher to crush the ore to P_{100} = 210 mm ,P_{80} = 130 mm. The bottom chute and the conveyor will convey the crushed ore to the stockpile hopper or on to a concrete pad.

The crushed material is conveyed to the 12,000 tonne capacity mill stockpile.

A static belt magnet is installed along the primary feed conveyor, and is swung away from its position using a hand pulley, to enable operators to remove tramp metal.

The crushing circuit capacity is rated at 670 Mtph and is designed for fourteen (15) hours per day operation at 90% availability. This will have sufficient stock for one (1) day operation.

Dual display HMI workstation will be set up in the crusher control room (20 ft container) for operators to control and monitor crushing system. This HMI workstation will be interconnected through Ethernet IP communications networks to Main Control Rooms in Area 110.

A 20 ft container electrical room will be used to house the motor control centre, PLC system, starters, Variable Speed drive and control power systems.

17.3. Grinding

17.3.1. Area 20: Grinding/Primary Cyclone

The mill stockpile has a capacity of 12,000 tonnes and contains crusher product which is sized at P_{100} of 210 mm. The mill stockpile sits over a concrete tunnel which contains three vibrating pan feeders and a conveyor belt which directs material into the SAG mill retractable feed chute. A single 6,200 kW SAG Mill (8.54 m dia x 4.57 m) is operating in closed circuit with a vibrating screen 1.6 m x 3.6 m which in turn returns the oversize after pebble crushing to the SAG Mill and a Cyclone classification. The SAG Mill will reduce the size F_{80}=130 mm to P_{80}= 150 micron.

A duty and standby mill discharge pump delivers slurry to the closed loop Primary cyclone Cavex 4 x 500 CV x 10 cyclone cluster, three (3) working and one (1) stand by. A 300 % recirculation load is considered. The under flow will flow by gravity to the Mill feed hopper and the overflow P_{80}-150 micron at 35 % solids will report to a thrash screen then to the desliming continuous CVD 64 Knelson units/Deslime Cyclone cluster. The deslimed feed will then gravity flow to the flotation condition tank.

A shark-fin sampling cutter device on the mill feed conveyor, which will allow collecting the representative samples.

The design capacity of the grinding circuit is 400 tph feed rate, closed circuit operation at 95 % availability with a design factor of 1.15.

17.3.2. Area 30: Gravity/Desliming

The cyclone overflow will be deslimed in either continuous CVD 64 Knelson units or a cluster of cyclones from which the deslimed slurry at 35 % solids will flow by gravity to the flotation conditioning tank.

17.4. Flotation

17.4.1. Area 40: Rougher/Scavenger Flotation

The deslime cyclone under flow with P_{80} 150 micron, 35 % solids and maximum 250 micron at a feed rate of 993 m^3/hr will have three (3) minutes retention time in the 90 kW agitated conditioning tank of 4 m diameter and 5 m high.

Conditioned slurry overflow, P_{80}-150 micron at a feed rate of 435 tph, 27 % by solids by weight and a maximum slurry flow rate of 1,347 m^3/hr will report to the Flotation machine comprising four (4) rougher cells and four (4) scavenging cells each 200 m^3 effective volume and fitted with 200 kW agitators. The retention time of 30 minutes each is provided for the roughing and the scavenging cycle. The rougher and scavenger concentrates will be pumped to the Regrind Ball Mill feed hopper operated in a closed loop with a cyclone cluster. The flotation tails will report to the tails thickener or direct to the tailing dam.

17.4.2. Area 50: Regrinding/Secondary Cyclone

The Regrind wet overflow discharge ball mill (3.6 m X 7 m) with a 1,350 kW motor will take the feed F_{80}-130 micron at 73 tph will give us P_{80}-30 micron. The Ball Mill in a close loop with a cyclone cluster and discharge pumps deliver slurry to three (3) sets of Cavex cyclone cluster (6 x 250 CV X 10) with five (5) cyclones running and one (1) on standby. The cyclone cluster will do the duty of 500 % recirculating load and will give us a cyclone overflow P_{80}-30 micron with 10 % solids by weight will feed to a cleaner flotation circuit.

17.4.3. Area 60: Cleaner Flotation

A concentrate slurry flow rate of 683 m^3/hr, 73 tph solids, 10 % solids, and P_{80}-30 micron will be cleaned in a Cleaner flotation machine comprising four (4) cells, each 70 m^3 effective volume with a retention time of twenty (20) minutes. The cleaner tails will report to a tail thickener/tailings dam.

17.5. Post-Flotation (Concentrate Option)

For the basecase flotation concentrate option the plant will consist of the elements listed below, otherwise the next step is the appropriate oxidation step.

17.5.1. Area 70: Concentrate Filter Feed Thickener

At a feed rate of 40 tph, SG (assumed) 3.2, 8 % solids by weight, slurry SG (assumed) 1.06 and a slurry flow rate of 473 m^3/hr, a 6 m diameter thickener will produce an underflow of 65 % solids.

This underflow of 40 tph with 65 % solids will report to a Filter feed slurry agitated tank (4.5 m dia x 5 m height) fitted with 110 kW agitator, providing 2 hours retention time.

17.5.2. Area 80: Filter Press

For a feed rate of 40 tph, SG (assumed) 3.2, 65 % solids by weight, slurry SG (assumed) 1.8, slurry flow rate 52 m^3/hr and P_{80}-30 micron

A Metso VPA 1540 – 40 Air membrane pressure filter is considered, which includes 200 kW oil free compressor, a 35 m^3 air receiver and high/low pressure pumps. The thickened pulp is introduced to the air membrane filter press at high pressure, which will produce a filter cake with assumed moisture of 9 % and maximum particle size of 120 micron.

The filter cake is discharged to a storage area and then loaded in to 2.5 tonne double lined bags. The bags are loaded in to sea containers and shipped by road transport to the port.

17.6. Oxidation Processes

Three flotation concentrate oxidation processes have been considered, namely Pressure Oxidation (POX), Biological Oxidation (BIOX) and Albion. Although these are not part of the base case flotation concentrate option they are included for completeness, allow for the introduction of any of these processes in future, or if the flotation concentrate option no longer viable.

17.6.1. Pressure Oxidation (POX)

The flotation concentrate is transferred to a 24 hour retention surge tank. The POX autoclave feed rate will be up to 200 m^3/h at 15 wt % solids and has been sized for one hour retention (vessel size of about 300 m^3). The autoclave will be operated at 180 ^{o}C with an oxygen overpressure of 500 kPa and total pressure of 1,400 kPa. Oxygen is supplied from an oxygen plant. The oxidized slurry will be cooled in flash towers with water circulation through heat exchangers. The slurry will then be washed in three counter-current thickeners (CCD). The wash solution will be treated in a series of six (6) neutralization tanks with limestone addition to pH 5.5 and lime addition to pH 7. Residual arsenic is precipitated as ferric arsenate in this step. The washed thickened tailings, adjusted to 40 wt % solids, are pH adjusted to 11 with lime and provision is made for a four (4) hour lime boil step to dissolve any basic ferric sulphate in the pulp. The lime boiled slurry constitutes the feed to the CIL described in *Section 17.7* below. The POX simplified flowsheet is shown in *Figure 17-4 - POX Flowsheet* below.

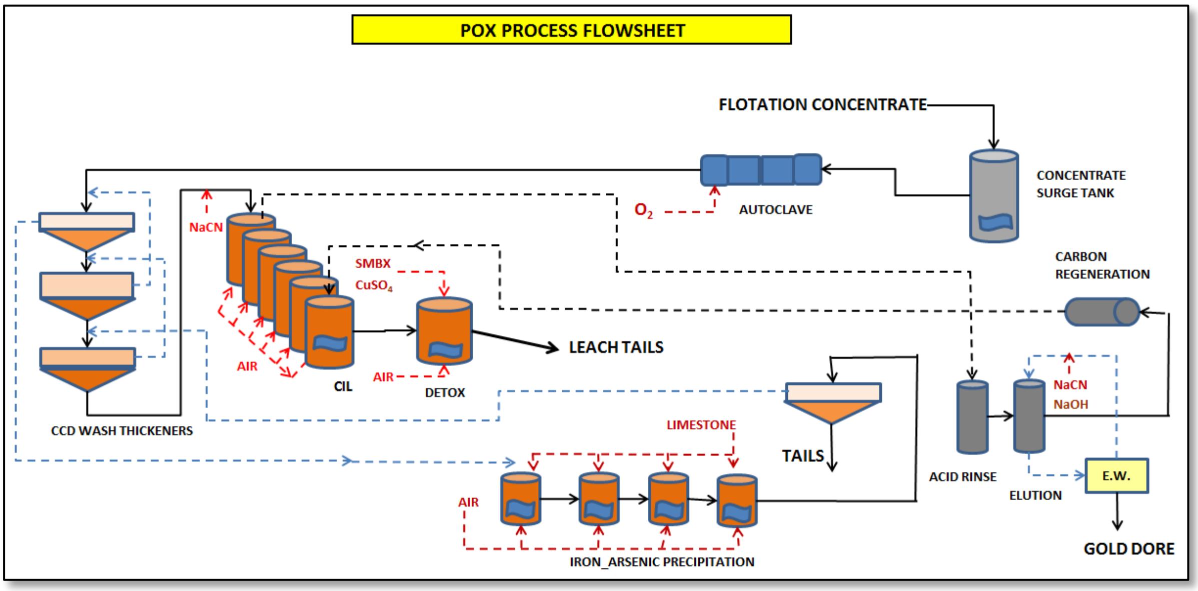


Figure 17-4 - POX Flowsheet

17.6.2. Biological Oxidation (BIOX)

The flotation concentrate is transferred into a 24 hour retention surge tank maintained at 20 wt % solids. Nutrients are added as required to the first stage BIOX tanks. The BIOX circuit will comprise 18 reactors configured in three (3) parallel trains each with three (3) first stage reactors followed by three (3) reactors in series. The slurry will then be washed in three counter-current thickeners (CCD). The wash solution will be treated in a series of 6 neutralization tanks with limestone addition to pH 5.5 and lime addition to pH 7. Residual arsenic is precipitated as ferric arsenate in this step. The washed thickened tailings are pH adjusted to 11 with lime prior to being fed to the CIL at 30 wt % solids described in *Section 17.7 Carbon-in-Leach (CIL)* below. *Figure 17-5 - BIOX Flowsheet* below depicts the flowsheet of the BIOX process prior to the CIL, detox and gold recovery section which is the same as for the POX shown in *Figure 17-4 - POX Flowsheet* above.

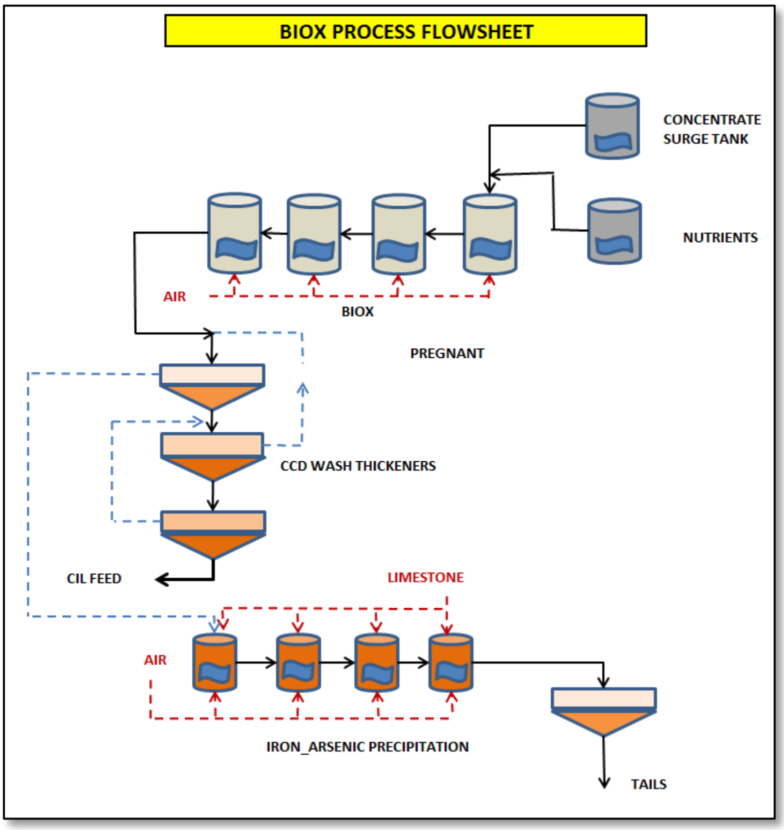


Figure 17-5 - BIOX Flowsheet

17.6.3. Albion Process

The flotation concentrate is reground in an IsaMill to a P_{80} of about 10 µm and transferred to the oxidation surge tank at 40 wt % solids. The ground ore is fed to six (6) Albion oxidation tanks in series controlled slightly below 100 °C by evaporative cooling. Pure oxygen is added to each reactor from an oxygen plant. The pH is controlled at 5.5 with limestone and NaOH is supplemented to promote the oxidation of arsenopyrite (neutral Albion leach or NAL process). During oxidation the arsenic is precipitated as ferric arsenate and excess iron as goethite. Suphuric acid resulting from sulphur oxidation is neutralized by the limestone, precipitating calcium sulphate. As a result there is an increase in mass by about 75 % with respect to the mass of feed flotation concentrate. About 75 % of the sulphur is oxidized. The effluent after oxidation is thickened and overflow recycled to the first oxidation tank. The thickened slurry is neutralized with lime to pH 11 with lime and % solids adjusted to 30 % prior to feeding to the CIL circuit. *Figure 17-6 - Albion Flowsheet* below depicts the Albion process flowsheet prior to the CIL, detox and gold recovery section which is the same as for the POX shown in *Figure 17-4 - POX Flowsheet* above.

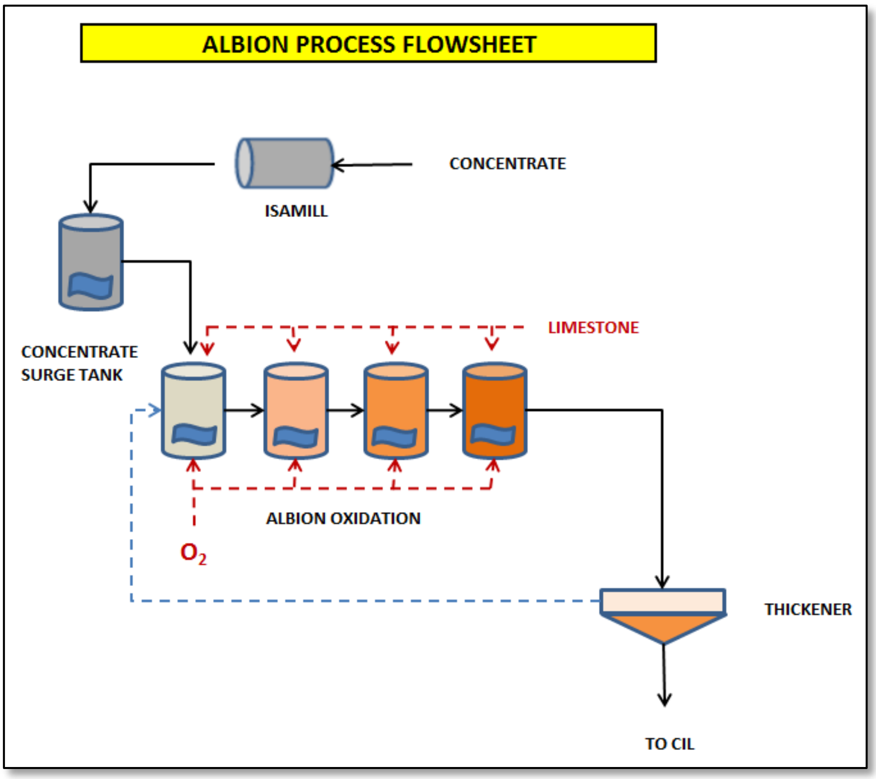


Figure 17-6 - Albion Flowsheet

17.7. Carbon-in-Leach (CIL)

The CIL (6 reactors in series) is according to standard practice with addition of NaCN to the first leach tank to control the NaCN in the last tank at about 500 mg/l. The consumption of NaCN is the lowest for the treatment of the POX product as almost all the sulphur has been converted to sulphate. With the BIOX and the Albion oxidation products the NaCN consumption is up to 2.5 times higher due the presence of partly oxidized sulphide (elemental sulphur and thiosalts) wich react with cyanide to form thiocyanate. The size of the CIL circuit for the Albion process is also more than double that of the BIOX and POX due to the increased solid mass going to the CIL. Carbon is added to the last CIL tank to about 30 g/l and moved forward by air lifting to the preceding tanks, all equilibrating at 30 g/l carbon, and collected from the first CIL tank where it is fully loaded with gold. Interstage screens prevent the carbon from flowing from tank to tank. The carbon adsorbs the gold from solution as cyanidation of the ore proceeds.

17.8. Carbon Desorption and Regeneration

The loaded carbon from the first CIL tank is transferred about five (5) times per week to the acid wash column where base metals are dissolved away from the carbon with a 3 wt % HCl solution, folowed by water rinsing. The carbon is then moved to the gold elution column where a heated electrolyte (about 90 ºC) is circulated in closed loop with electrowinning cells where gold is reduced on steel wool cathodes producing a gold sludge. The stripped carbon is re-activated in a carbon regeneration kiln from where it is returned to the last CIL tank with addition of make-up fresh carbon to compensate for carbon attrition losses.

17.9. Electrowinning and Refining

Electrowwining of gold from the gold containing elution solution is performed in rectangular cells with pervious cathodes packed with steel wool and stainless steel mesh anodes. The gold sludge produced from EW is filtered, calcined and melted in a furnace to produce gold doré bars. The gold doré bars are sent to a refinery for production of gold bullion.

17.10. Detoxification

The tailings from the CIL circuit are treated in an aerated and agitated detox tank using the world leading copper catalysed SO_2/Air cyanide detoxification process. The process is very efficient for the detoxification of cyanide in the presence of solids. Because no solid/liquid separation is required all cyanide leaving the plant is detoxified in this treatment process. The overall reaction is shown below:

$$2CN^{-} + Na_2S_2O_5 + 2O_2 + H_2O = 2OCN^{-} + 2NaHSO_4 \quad (1)$$

The SO_2 is dosed as a solution of sodium metabisulfite ($Na_2S_2O_5$), copper as a solution of copper sulfate and oxygen is provided by air sparging. All forms of cyanide are detoxified in the process and base metals (copper, zinc, nickel) are precipitated as solid hydroxides. Iron cyanide is precipitated in the form of stable cupric-ferrocyanide.

Cyanide is oxidized to cyanate (OCN-). The reaction also produces some acidity which is neutralized to calcium sulphate with lime at a controlled pH of 8.5.

Cyanide is removed from the base metals (copper, nickel, and zinc), which causes the metals to precipitate as hydroxides (base metal Me^{2+}) as shown below:

$$Me^{2+} + 2OH_{-} = Me(OH)_2 \quad (2)$$

The iron cyanide which is present as $Fe(CN)_6^{4-}$ in solution is removed into a stable cupric ferrocyanide precipitated by the following reaction:

$$Fe(CN)_6^{4-} + 2\,Cu^{2+} = Cu_2Fe(CN)_6 \quad (3)$$

17.11. Process Management and Control

17.11.1. General

The instrumentation and control system design for the plant provides for a high level of safety, reliability and availability of the operating plant while minimizing maintenance requirements and allowing simple system faultfinding.

The design and installation of all equipment complies with all statutory regulations and standards but will be subject to approval by the local regulatory authorities, where required.

The general philosophy is as follows:

- The Process Control System (PCS) is a microprocessor-based system composed of distributed controllers, the number of which will be determined by process requirements. A controller comprises of one or more processors, I/O devices and communication devices. Human interaction is provided through operator interfaces, strategically positioned at locations that provide the most effective control of the plant. These components will be interconnected through dedicated communications networks. A listing of communication links to the PCS is presented in *Table 17-1 - Communications Link Listing*.

Equipment Items	Control and Data Connection to the PCS
LV motor starters	Ethernet IP with Hard wired I/O
LV MCC incoming circuit breakers	Hard wired I/O
LV variable speed drives	Ethernet IP with Hard wired I/O
emergency generator controls	Modbus for data collection with Hardwire I/O
Uninterruptible power supplies	Hard wired I/O
Battery charger alarms	Hard wired I/O
HV switchgear	Modbus for data collection with Hardwire I/O
HV motor starters	Modbus for data collection with Hardwire I/O
Secondary resistance starters	Hardwire I/O
HV power factor correction control	Hardwire I/O
Vendor package (without programmable system)	Remote I/O module with Ethernet IP
Vendor package (with programmable system)	Ethernet IP with Hard wired I/O
Local control panels (No programmable system)	Remote I/O module with Ethernet IP
Analogue Instruments	Hard wired I/O
Digital Instruments	Hard wired I/O
Modulating control valves	Hard wired I/O
On/off solenoid operated control valves	Hard wired I/O
Fire alarm system	Industry standard serial communication protocol.

Table 17-1 - Communications Link Listing

- The systems shall be modular and scalable to cater for future expansions
- The level of automation is moderate. Analogue control is extensive within the main areas, but minimal in the batch and manual handling sections of the facility. Sequencing of equipment and the automatic operation of on/off valves are provided where necessary for the safe and effective operation of the relevant areas.

- Integrated control of the process plant via the PCS, for all areas where equipment requires sequencing or process interlocking, including all vendor package items
- Monitoring of all relevant operating conditions via the PCS and recording of selected information for data logging or trending. In addition, the PCS will gather process data to be readily exchanged with enterprise level applications like data historians and statistical analysis packages
- All control loops are via the PCS except where integrated into vendor package equipment
- Individual PLCs for control of packaged equipment are restricted to where specialized control functions are required. Special PLCs will network via communication data links to the plant PCS, using industry Ethernet IP protocols and for the purpose of recording, monitoring and remote operation of the package
- The use of Ethernet IP technology is used for electrical drives to minimize site control cabling and provide the ability to gather comprehensive information for maintenance personnel
- Emergency stops, conveyor pull wire switches and similar safety related items shall be hardwired. In general, all interlocks necessary for personnel safety and equipment protection are also hardwired
- Communications networks will be designed such that no single point of failure will impair operator visibility of the process or compromise safety. This will be achieved by employing ring or dual bus networks where practical. This is generally not available or feasible at the field level and so device failure will be detected and appropriate control actions taken

17.11.2. System Architecture

The PCS and associated instrumentation for the plant are integrated to a plant-wide control and data management system.

The levels of control and data management are:

Level 0: Incorporates all process instruments and equipment that interface directly with the process. Examples are field level networks, pressure, level and flow elements and transmitters, in line analyzers, hand switches, proximity switches, solenoids, control and shutdown valves, drives and VSDs.

Level 1: Includes the process control system elements that directly control the process and advanced process control functions and subsystems. Typical of this level items are PCS controllers, control networks, PLCs, PCs and workstations. Subsystems include analyzers, laboratory data and package unit controlsystems.

Level 2: Incorporates supervisory control and process data management, allowing for on-line and off-line management of the process, recovery improvement, quality control, analysis and environmental monitoring. This level also incorporates tieins to the plant-wide local area network (LAN), thus allowing common PC-based workstations used by various managers and departments to gain controlled access to the process control system database.

Level 3: Incorporates plant wide management of the operation. Includes; maintenance planning and scheduling, spares, consumables and assets management.

Level 4: This level is for corporate management and includes administration, finance and sales, corporate ERP systems or similar. The engineering criteria specified in this document pertain specifically to Levels 0 and 1, with some Level 2 requirements related to remote user access. All equipment selected for Levels 0 and 1 shall be readily capable of future integration to Levels 2, 3 and 4.

17.11.2.1. System Performance

The response time of the system is sufficient to maintain control over the plant processes under all system operating conditions including extreme plant upset conditions with all points in alarm. The response time is the total elapsed time for transmission of data from field state change to the system communication path to the operator interface. This time includes all communication time from processor to processor, I/O scans, nodes, gateways and associated equipment internal to the system. The minimum expected system response time is presented in *Table 17-2 - Minimum Expected System Response Time Summary*.

Function	Nominal Response (msec)
Monitoring/Information	1000
Modulating Control - Slow Loops	500
Modulating Control - Fast Loops	250
Manual Control	500
Motor Control	500
Sequence-of-Events and Alarm Monitoring	N/A

Table 17-2 - Minimum Expected System Response Time Summary

17.11.2.2. Redundancy

To minimize the possible impact of equipment failure, installed redundancy is provided as follows:

- Operator interface stations - Yes
- PCS servers - Yes
- Main system network communications - Yes
- Communication to package PLCs - Yes
- Communication to remote I/O modules - No
- PCS controller processors - Yes
- Processor power supplies - Yes
- I/O – No

17.11.3. Hardware

17.11.3.1. Voltage Utilization

The voltages used by the PCS are as follows:

- PCS control hardware 220 V AC
- Warning sirens 24 V DC or 220 V AC
- PCS operator stations 220 V AC
- Remote I/O modules 24 V DC
- Digital inputs 24 V DC
- Digital outputs 24 V DC
- Solenoid valves 24 V DC
- Four wire instruments 24 V DC

17.11.3.2. Earthing of PCS System

Three earthing systems are provided for the control system as described below:

- Power earth for all 220 V AC systems
- Instrument earth separately cabled from the power and Intrinsically Safe (I.S.) system earth. A connection is made in each substation from the PCS cubicle instrument earth bar to the main electrical power system earth bar
- I.S. earth is separately cabled from the power and instrument system earths. A single connection is made in each substation from the PCS cubicle I.S. earth bar to the main I.S. system earth bar

17.11.3.3. Major Components

The process control system includes the following major equipment:

i) PCS Cabinets

PCS cabinets are provided to house control hardware and for the termination of communication bus and field cabling. The cubicles are located both within the MCC and for remote I/O located in the field (for that area). Control hardware includes the controller units, communication equipment, power supplies, system input/output and interface modules.

The cabinets are divided into separate sections segregating control system hardware from the termination of field cables. PCS network and local field network cables terminate in the hardware panel of the cubicle.

Within the termination panel, separate terminal rails are used for discrete digital and analogue signal cables, with I.S. signal cables also segregated separately where applicable.

Field located remote I/O cabinets achieve an IP65 rating and are strategically located in areas to allow concentration of field equipment or instrumentation.

ii) Control Network

The control network is part of a site wide communication backbone utilizing fiber optic cables between facilities and copper cables within buildings. The shall either be complete redundant cabling via differing routes for each or “spanning tree” type communications ring technology for the main PCS backbone.

The control network design is such that no single point of failure shall degrade the operation of other components within the system.

iii) Operator Interfaces

Operator stations located in the control rooms are each comprised of a color LCD screen, a keyboard and a cursor control mouse to provide the normal interface between the operator, the plant processes and equipment being controlled or monitored. Alarm trend and report functions are also displayed on these stations.

Separate levels of access control are provided for:

- Monitoring only (viewer or remote access)
- Starting and stopping of drives and changes to operating settings (operations)
- Program configuration and loop tuning changes (engineering)

iv) Engineering Workstations

Engineer/programmer’s workstations provide the interface between the plant engineer and the plant processes and equipment for control system tuning, system program modification and graphic display development.

v) Printers

The printer within the control room provides the operator with a hard copy record of logs, reports, system events and graphical displays. A printer is located next to the engineering console to provide the engineer logs, special reports and documentation of system programming changes.

vi) Historian Server

A plant data historian is provided to capture critical data, improved management reporting and interface to corporate systems. Acquisition of data focuses on assisting management with operating to key performance indicators of production, quality and availability. This data can also be used for later plant optimization

17.11.3.4. Field I/O Subsystems

In general, field I/O is interfaced to the PCS in two different ways, i.e. hardwired direct to the substation PCS cubicle or via remote I/O PCS modules located in strategic field locations. There are two different types of I/O, either standard or intrinsically safe. Individual fuses are provided for all inputs and outputs.

The system arrangement permits significant future expansion if required, nominally 20% spare capacity. In most situations, field signals from devices such as flow switches and solenoid valves are marshaled to junction boxes located in the process areas. Depending on distance and the number of signals, the junction boxes may include remote I/O module(s) with a PCS communication link to the main PCS controller. For conventional instrumentation, signal levels to and from the control system is as follows:

- Digital Inputs 24 V DC powered from the interface I/O module
- Digital Output 24 V DC powered from the interface I/O module or isolated voltage free relay contact (as applicable)
- Analogue Inputs 24 V DC, 4-20 mA, loop powered or field powered transmitters are used as appropriate
- Analogue Outputs 24 VD C, 4-20 mA, loop powered and individually isolated

17.11.3.5. Interfaces to MCCs and VSDs in Substations

i) MCCs and Smart Devices
All MCC fixed speed drives, Variable speed drives and protection relays shall use Ethernet IP as the preferred communications protocol to provide indication and control.
ii) Hardwired Signals
Other hardwired signals (where applicable) are connected to remote I/O units located in dedicated cubicles and connected to the PCS via the Ethernet IP communications platform. These include devices such as power supplies, circuit breakers, etc.

17.11.3.6. Safety Interlocks

i) Statutory Equipment and Safety Interlocks
All personnel safety devices (e.g. pull wire switches and emergency stops) are directly wired into the drive control circuit, maintained in the control circuit under all conditions and designed such that de-energizing of the control circuit stops the drive. All other interlocks are soft wired via the PCS.
ii) All software interlocks and control signals sent over the network or via communication system are engineered such that failure of the communication link results in control action that does not affect plant safety.
iii) Safety Instrumented Functions
Hazard identification and hazardous operation analysis is to be carried out during detailed design. Any risk targets that require safety instrumented functions and the associated safety integrity level necessary to meet these targets, will be designed in accordance with relevant international standard. Similarly, insurers or statutory bodies may deem certain functions mandatory and require guaranteed reliability and integrity of operations, particularly related to the fire suppression system, emergency shutdown loops, etc.

17.11.3.7. Interfaces to Vendor Packages or Third Party Systems

i) Minor Vendor Packages
Where vendor packages do not include an integral PLC system, the input and outputs required are wired to remote I/O modules within the package. The modules communicate to the PCS using a Ethernet IP communication link.

ii) Large Vendor Packages or PLCs
For large packaged equipment, the vendor is responsible for supplying the process control equipment for that package with due regard to standardization (e.g. a site standard control system) and applicable acts, codes, standards and client standards. The package control is interfaced to the main plant PCS using Ethernet IP technology allowing remote low-level supervisory control (e.g. individual drive start/stop with appropriate interlocks), monitoring from the area operations centre and remote engineering access. The use of hardwired controls is minimized consistent with the speed and reliability of an Ethernet connection. Safety related signals are hardwired to the vendor package equipment.

17.11.3.8. Power Supplies

Power supply for the PCS is 220 V AC from the relevant instrument Uninterruptible Power Supply (UPS). This supply is sufficient to supply the PCS, including all operator stations, for 30 minutes in the event of loss of power supply.

The UPS is equipped with a bypass switch to allow system maintenance. 24 V DC power supplies for PCS I/O are either dual redundant or N+1 in capacity so as to ensure continuous supply to the nominal load in the event of a single unit failure.

Power supply to remote I/O is provided from dedicated distribution boards or the PCS. Each supply is fed from a separate circuit breaker.

17.11.4. System Software

The sites software systems shall be compatible and shall the same operating environment. Applications for all systems shall be stored centrally and registered to the client. Data for all systems shall be stored centrally where possible and periodically replicated to the central servers where required. Daily backups to the corporate information technology network will provide data security and integrity in the case of catastrophic failures. Software that requires remote support shall be accessible through the corporate network via the use of Virtual Private Networks.

17.11.5. Preferred product

The preferred PCS product is Controllogix 5000 series with FactoryTalk platform. This system is used in Phuoc Son Mining.

17.12. Instrumentation & Valves

17.12.1. Field Instruments

17.12.1.1. General

Instruments are specified and selected to provide uniformity of supply to the extent practical and with a view of minimizing the spare parts inventory. Where practical, field instruments are mounted in areas that offer easy access for maintenance, are free of vibration and do not block walkways or prevent maintenance of other equipment.

Field-mounted instruments are of design suitable for the area in which they are located. All instruments and control equipment have corrosion resistant, weatherproof cases and are suitable for outdoor mounting. The minimum protection rating is IP65.

In general, the equipment types shall be as follows:

i) Flow Instruments
For the measurement of the flowrate of slurries, liquids and gases, the basis of the design is to provide non-intrusive in-line flow measurement devices. Magnetic flow meters are used for measurement of conductive slurries and liquids, whereas vortex type flow meters are employed for low conductivity liquids. For air and gas flowrate measurement, differential pressure type devices are used

ii) Pressure Instruments
All pressure transmitters and gauges are installed with block and drain/vent valves for process isolation and calibration, except. For slurries and where the process conditions require, silicon filled diaphragm seals is used to protect the pressure gauges and transmitters.
Pressure transmitters are capacitance type devices of stainless steel element construction. Local pressure gauges are 100 mm diameter 316 stainless glycerin damped steel bourdon tube type instruments

iii) Level Instruments
Instruments such as ultrasonic, microwave and nucleonic devices have been selected for level measurement. Visual site glasses shall be used where required on boiler drum levels or where visual sighting of level is critical. Level switches of the ball float type will typically be used to start and stop spillage sump pumps

iv) Temperature Instruments
All temperature transmitters and gauges are to be installed with stainless steel thermo wells for maintenance and replacement of elements during operation. In more severe applications, the instrument wetted parts are constructed of identical material as the pipe or vessel the instrument is mounted to. Temperature element types are PT100 RTD devices and temperature gauges are stainless steel bi-metallic type with 125 mm "every angle" type dials.

17.12.1.2. Hazardous Area Instrumentation

The location of all instrumentation equipment, which may be a source of ignition, shall be determined after the plant hazardous areas have been classified. Where possible such equipment shall be located outside of the classified area, or in the zone of least hazard.

All equipment, which must be installed in a hazardous area, shall be selected in accordance with all the applicable statutory codes, standards and procedures. For field-mounted instruments, Intrinsic Safety using electronic barriers will be the preferred protection technique for instrumentation.

17.12.2. Actuated Valves

Discrete actuated and control valve types are selected according to the basis of the design. All valves shall fail to their nominated failure position in the event of an electrical or pneumatic failure. All on-off actuated valves, other than solenoids, shall be provided with open and closed feedback limits.

17.13. Electrical System

17.13.1. Motor Control Centers

Each MCC comprises:

- An incoming section with withdrawable circuit breaker
- Suitably rated busbar system
- Motor control modules to suit the number and ratings of motors and equipment detailed in the equipment list
- Cableways adjacent to each tier of motor control modules

The MCC supply is via single core cables from the associated step-down transformer to the incoming isolator/circuit breaker.

Each MCC is of Form 4 construction with a minimum enclosure protection rating IP42 and suitable for top entry of cables from overhead cable ladder fixed beneath the roof of the switchroom.

Within each vertical modular section, a 300 mm wide common power and control cable zone is provided to house the control terminals for each drive. Power cables are connected directly to the drive overload relay.

The MCC starter modules are of demountable construction and, in general, contain the following equipment, subject to drive size, type and special requirements:

- Padlockable drive isolation via the molded case circuit breaker with door interlock
- Molded case circuit breaker
- Contactor
- “Smart” Electronic Overload Relay incorporating thermistor inputs
- Separate RTD relay for motors 220 kW and above
- Motor heaters on drives of 220 kW and above

17.13.2. Variable Speed Drives

LV variable speed drives are as follows:

- Voltage/frequency control
- Controllers are housed in separate wall-mounted or freestanding cubicles
- Output filters are included as recommended by the drive vendor for the motor size and cable length

17.13.3. MV Motor

The only MV motor is for the SAG mill. The motor will be fitted with two RTD's per winding phase for temperature detection 1 RTD for each bearing. The mill motor will be fitted with an anti-condensation heater.

No aluminum components shall be used due to possible acidic fume environment.

17.13.4. LV Motors

The majority of squirrel cage induction motors are mining industry standard with cast iron frames and cast iron fans and cowls. Motors have Class F insulation and rated for a Class B temperature rise and are from the manufacturer's range of standard/high efficiency motors.

A single thermistor per phase is fitted to all motors of rating 37 kW to 220 kW and on smaller motors is driven from a variable speed drive. Motors 220 kW and larger are fitted with one RTD per phase for temperature detection and an anti-condensation heater.

No aluminum components shall be used due to possible acidic fume environment.

Motors located in hazardous areas will be purchased with the appropriate "Ex" rating.

17.13.5. Plant Cabling

The following cable types are used:

- HV power cables - Copper XLPE/SWA/PVC
- LV motor cables - Copper PVC/SWA/PVC
- LV Variable speed drives - Copper PVC/SWA/PVC or screened depending on the supplier's recommendation
- Control Signals - Copper multi-pair Dekoron twisted pair, overall screened cables
- Analogue Signals - Copper twisted pair, overall screened cables
- PCS communications - Fiber optic cables between controllers.

Cable ladders are installed overhead in the MCC rooms, with cables being distributed around the plant on overhead ladder supported from the plant structure where practical.

The final connecting cables to electrical equipment are supported by conduits or ladder, with as far as practicable cables entering from below.

Cables are rated to suit the circuit load, voltage drop requirements and site conditions, with power cables laid touching on ladders in a single layer, or spaced if required to minimize installed cable cost.

Control and instrument cables are installed multi-layered on separate ladders from power cables. Where it is not practicable to use separate ladder, the cables shall be segregated from power cables by a barrier strip.

Buried cables are protected against termite damage as follows:

- HV cables, major LV feeders and telecom cables nylon coated with sacrificial PVC sheath
- Termidor (or equivalent) termite treatment in the cable trench
- Control, instrumentation and small power cables installed in PVC conducts

17.13.6. Materials of Construction

The standard material of construction for cable support systems is galvanized mild steel. In areas exposed to corrosive liquids or atmospheres the following materials are used:

- 316 stainless steel, fiber glass, ABS or FRP materials
- No aluminum materials or components in outdoor equipment

17.13.7. Earthing

Earthing of the plant electrical power system is the following features:

- Combined HV and LV earthing system – earthed to a common earth bar
- The HV system impedance earthed to limit the earth fault current to 350 amps
- The LV systems solidly earthed at a single point

17.13.8. Lightning Protection System

Due to the lightning risk, additional measures to protect equipment and personnel from lightning. This includes:

- Lightning earthing systems, where provided, shall be bonded to the power earthing system
- Where LV power supplies originate in another facility, such as an administration office supplied from a plant switchboard, shunt surge protection shall be provided in the receiving switchboard\Data and telecommunication cables between facilities, where not fiber optic, shall be protected by series surge protection at the entry to the building/facilities

- Where instruments or control signals originate outside a facility, series surge protection shall be provided at both the remote device and receiving equipment

17.13.9. Lighting and Small Power

General lighting throughout the plant is supplied by high-pressure sodium fittings mounted, where possible, on the steelwork. When structure mounting is not practical, such as along conveyors, 3 m mounting poles are used.

Fluorescent fittings are installed in areas that require immediate re-strike, such as the MCC rooms and safety shower locations. In safety shower locations, green fluorescent fittings are used. For emergency lighting, battery back-up fluorescent fittings are provided in the plant control rooms, switchrooms and on stairways to permit safe access to ground level.

Outdoor lighting, except emergency and safety lighting, is controlled by photoelectric cells.

A manual bypass switch is provided on each distribution board to facilitate maintenance.

Area lighting is by floodlights mounted from the plant structure. Roadway lighting poles have been included where there is no nearby building structure to mount the fittings on.

Roadway light poles will not be arranged to swivel.

Small power switched single-phase 220 V socket outlets are strategically located within the plant area. 380 V three-phase welding outlets are provided at ground level around the perimeter of the plant, on top of tank platforms and in the crushing area.

17.13.10. Hazardous Areas

Design, installation, final inspection and equipment selection shall be determined such that they conform to applicable standards and appropriate hazardous area zoning requirements, as identified by the hazardous area classification study.

17.14. Ancillary & Miscellaneous

For the basecase flotation concentrate option the following elements are applicable.

17.14.1. Area 90: Reagents

This module consists of the mixing tanks, mixers, storage tanks and pumps required for the following reagents:

- $CuSO4$ –Copper Sulphate
- PAX – Potassium Amyl Xanthate or SIBX (Sodium Isobutyl Xanthate)
- Aero Float
- Flocculent

- Anti-foam
- CMC
- Additional (MIBX or other to be determined)

The full module will be arranged in a separate area with proper platform, ladders, safety showers and handling systems for easy and safe working. The reagents module will be fully automated for centralised control and monitoring.

17.14.2. Area 100: Services

This module includes:

- Fresh Water, Process water supply holding tanks;
- Duty and standby water supply pumps and distribution system;
- Duty and standby high pressure air compressors and receivers;
- Duty and standby low pressure air compressors/blowers;
- Diesel Generator & Diesel storage and Daily service tanks.

17.14.3. Area 110: Main Control Room and Electrical Room

17.14.3.1. Main Control Room

The main control room will be located in the grinding area. This room contain Process Control Systems (PCS) and associated peripheral equipment required for controlling, supervising and monitoring all process plant, eg: HMI Servers, operator HMI workstations, engineering workstation, switch, controller hardware and cabinets. Process Control System located in the main control room will interface to the PLC's and will be networked with HMI workstation in the crusher area 10.

Operator HMI workstation will provide real-time process control; motor control; data acquisition, display, and logging; alarming; trending; and reporting for their particular applications. Components of the system will be distributed in remote locations throughout the plant, with the communication network linking all locations.

Engineering workstations in the main control room will support off-line configuration, data entry, application program development and documentation, custom graphics generation, software backup, and system diagnostics of all devices connected to the system networks. Each workstation will operate independently but will have access to common data and displays, including current and historical data.

17.14.3.2. Main Electrical Room

The main electrical room located in the grinding area will house the motor control centre, PLC system, starters, Variable Speed drive and control power systems.

Power is fed from the main substation will utilize 11 kV for the larger motor loads and at 380 V for smaller loads.

Medium voltage and low voltage transformers will be installed in the available area next to the electrical rooms to provide the required voltage levels to each MCC.

17.14.4. Area 120: Support Facilities

The following elements in *Table 17-3 - Plant Infrastructure Components & Area Covered* are for associated support facilities and buildings/infrastructure for the plant and the area covered.

Facility	Area
Metallurgical Laboratory	240 m^2
Plant Office	240 m^2
Plant Workshop	200 m^2
Chemical Storage Warehouse	200 m^2
Plant Warehouse	150 m^2
Dress/Change Room & Security	108 m^2
Generator House	450 m^2
Crusher Building	300 m^2
Crusher Control Room	14 m^2
MCCB & Control Room	240 m^2
SAG/Ball Mill Building	1,400 m^2
Process Plant Building	3,200 m^2
Stockpile Building	2,000 m^2
Reagent Building	200 m^2
Plant Compressor Building	108 m^2
Medical Facility Building	200 m^2
Main Security & Server Room	60 m^2
Kitchen & Dining Facilities	200 m^2

Table 17-3 - Plant Infrastructure Components & Area Covered

17.15. Process Manpower

Listed in *Table 17-4 – Planned Metallurgical Manpower List* below is the planned manpower list (per plant area) for the processing operations for the 8,000 tpd base case scenario.

Position or Plant Area	Staffing
Manager	2
Supervisior	3
HR Staff	1
Crusher	10
Ball Mill	5
Flotation	6
Filter Press	3
Cyclone	1

Position or Plant Area	Staffing
Reagents	2
Control Room	3
Power	2
Tailings Dam	3
Maintenance	30
Office	4
Metallurgical Lab	5
Contingency	6
Total:	***86***

Table 17-4 – Planned Metallurgical Manpower List

18. Project Infrastructure & Ancillary Services

18.1. Site Access

The site can be accessed from the old main road from Kuching to Bau via two (2) roads, namely the Jugan-Siniawan Road and the Jugan-Buso Road. Entrance to the site is within 1-2 kms via sealed roads. These roads will need some minor upgrading with some upgrading of two (2) small bridges required to handle heavy traffic vehicles. The main and access roads are shown in *Figure 18-1 - Aerial Image with Site Access Roads* below.

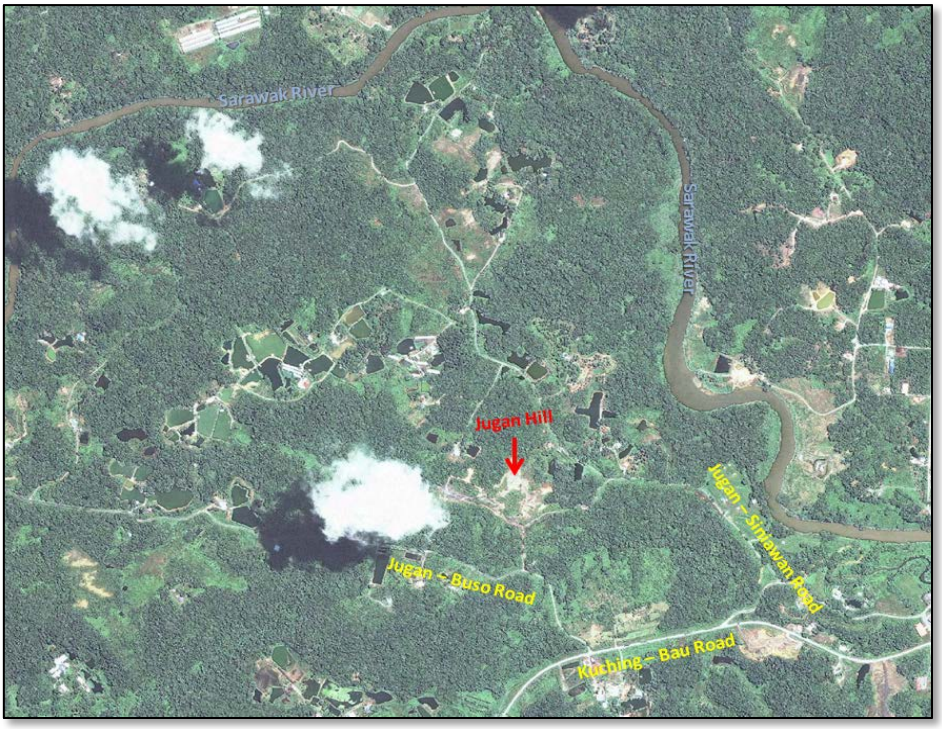


Figure 18-1 - Aerial Image with Site Access Roads

18.2. Site Development & Facilities

The pit, process plant, TSF, waste disposal landform and support facilities will be built within the ML's covering the Jugan Hill project. The overall site plan showing the infrastructure and facilities is shown in the following diagram – *Figure 18-2 - Jugan: Mine Infrastructure & MC/ML Boundaries*. The mining leases (ML's) are also shown on the attached plan.

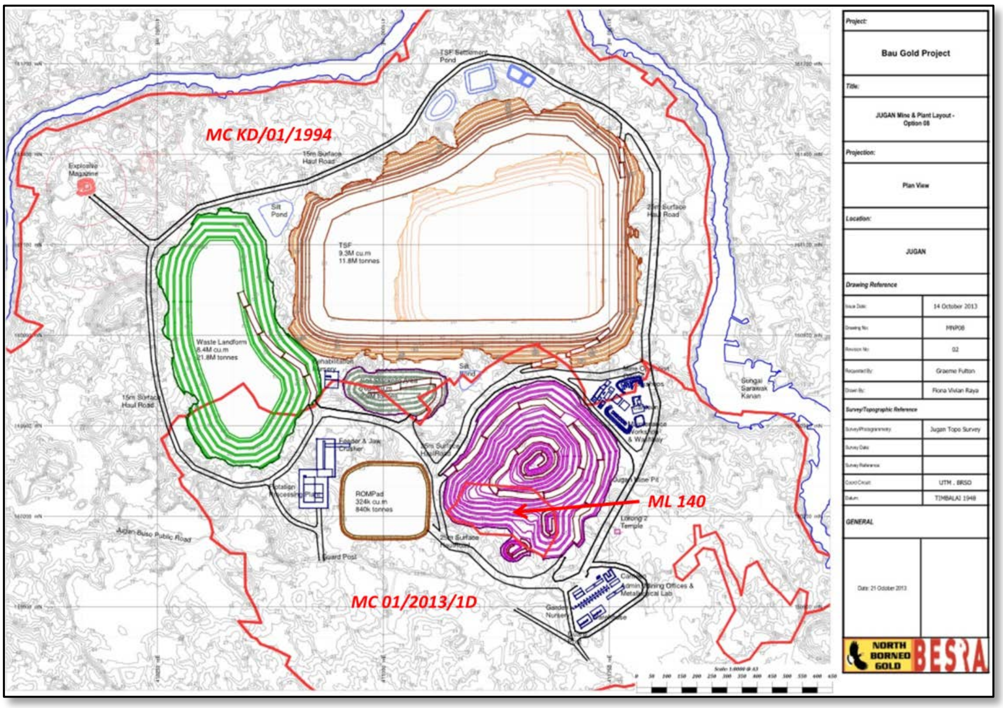


Figure 18-2 - Jugan: Mine Infrastructure & MC/ML Boundaries

18.3. Existing Facilities

The only existing facilities currently present at the site are the roads, power and communications (telephone) supplying the houses of the local residents.

18.4. Tailings Facilities & Management

18.4.1. Design Basis & Criteria

The 20 m high Jugan TSF has been designed to internationally and nationally acceptable standards to provide a facility for safe and environmentally acceptable containment of tailings. An internationally recognised tailings dam designer, reviewer, and constructor has been engaged in the design of the TSF, and continue to be involved in the design development for future raising of the TSF and the progressive rehabilitations of the TSF throughout its operational life and deactivation. The designer will also be contracted in the actual construction of the dam in terms of site supervision of the entire earthworks activity and in the QA/QC aspect of it. The designer will also be commissioned to conduct the annual auditing of the facility.

The sizing is based on 8,000 tonne per day ore mining over a period of 3-4 years. The beneficiation of gold ore is via flotation process with an assumed and approximate 10% mass pull-out as concentrate. The remaining 90% will constitute the tailings to be contained into the TSF pond at 35% solids during transport and deposition.

The design criteria for the TSF focused on the following aspects:

- Its long-term structural integrity throughout its operational life, and thereafter;
- Maximised use of mine waste as build-up material in a cost-effective manner;
- Maximised use of cut materials from the TSF basin as part of the borrowed build-up materials; this undertaking will also serve as part of the pond's base grading design to its desired bottom elevation;
- Maximised the possible extent of desiccation after tailings deposition through a combination of sub-aerial deposition technique, the use of effective cut-off blanket drain system located between the upstream and downstream embankment zones and an open spillway system designed in handling both the discharge of tailings supernatant and run-off derived from a peak 100-year rainfall event;
- The TSF should be able to withstand nominal 0.10 g peak ground acceleration from any earthquake generators. This PGA value is taken from a nearby local reference – the earthquake attenuation value adopted in the newly-built Bengoh Dam, a 63.2 m high concrete hydro dam located in Bau along the Bengoh Range. Moreover, the Global Seismic Hazard Map (GSHAP) categorises Sarawak as a low hazard seismic area; it has a maximum 8 % PGA, or 0.08 g, or 0.8 m/s^2 that may most certainly happen at least once in 475 years;
- Protection of the groundwater and receiving water bodies from any contamination to the highest extent practicable;
- Protection against overtopping where the spillway was designed to handle the peak run-off from a 100-year rainfall event.

18.4.2. Tailings Site Selection

The Jugan TSF is sited according to a number of factors that we have considered necessary to optimise both the construction of the dam and the operational aspects of the whole mining complex, among which the following are:

- The geology in the area is shale, which is relatively good for construction material;
- The topographic features in the area on which the TSF will stand are lower and relatively flatter than most of the areas near the Jugan Pit; and the plant site will be sited in a hill which gives advantage to the conveyance of the tailings slurry to the tailings pond via gravity (up to a certain point);
- There is an abundance of small hills within the footprint of the basin, which will serve as one of the sources of borrow material – cutting these hills down to RL 20m is part of the levelling within the basin
- In addition, the presence of small hills along the footprint of the embankment would allow the abutment or key-in of shorter length dikes into the sidewalls of these hills thus creating more surface contact (other than the base foundation key-in) between the embankment and original ground
- Its relative proximity to the Jugan Pit, hence the processing plant is also nearby allowing better management, monitoring, control and foreseeing of the overall mine operations;

- Abundance of adjacent source of fill material; mine waste will be used to build-up the dam apart from the borrowed materials from the TSF basin hence the nearer the TSF that is functioning also as a repository site of mine waste, the favourable it is in operational and economic sense for the mine operation;
- The site allows for a bigger TSF where there is more than enough tailings impoundment and containment both for Jugan and Bukit Young pits.

18.4.3. Tailings Characteristics

In terms of the chemical characteristics of the ore prior to mill beneficiation by flotation process, it is categorised as potential acid-forming (PAF). However, during flotation, much of the sulphides present (>90 %) in the ore will be recovered in the concentrate. Thus, the tailings will contain only about 0.3 % sulphides and is categorised as benign non-acid forming (NAF). The series of recent metallurgical tests confirmed that after flotation, the rougher tails exhibit low sulphide content. Although no signs of acid generation are observed, the potential for the mixed NAF and PAF embankment and the shale bedrock to be reactive with the tailings deposited in the long-term is not discounted.

Hence the TSF floor will be lined entirely with HDPE liner (to prevent seepage into the bedrock), while the upstream embankment side exposed to the beached tailings will be provided with a blanket drain acting as a cut-off behind to promote seepage through the embankment's blanket drain and aid in the drying of tailings, other than via exposure to sunlight and natural decantation of supernatant by provision of a spillway. This will also protect the downstream embankment from getting saturated). The HDPE will be anchored or skirted to a nominal height of 4 m above the toeline of the upstream embankment, or a metre above the level of the blanket drain (underdrain) provided in the floor.

In terms of the tailings' geotechnical and hydraulic properties, it is assumed that the predominantly silt-sized tailings will have low densities when discharged. However, the density of the contained tailings will increase substantially as loading is applied from succeeding tailings deposition.

For the TSF closure, non-reactive clay mixed with NAF mine waste will be provided over the tailings to form an inert cover.

18.4.4. Tailings Storage Facility Design

The tailings will be deposited using sub-aerial technique via multiple perimeter spigotting. The aim is to remove as much water as possible from the tailings at the time of deposition and to increase the in-situ density to the maximum extent possible by desiccation/drying. Moreover, this approach further ensures the long-term structural soundness of the TSF in addition to the design basis safety factor intended during its operating life.

Section 19.2.3.2, on the other hand, provides the list of combined field and laboratory geotechnical tests, geological and topographical requirements pertinent to the construction.

The Stage 1 and partial of Stage 2 embankment will be built using borrowed material from the basin, which is shale in general, up to a height of 10 metres (20 mRL to 30 mRL). The rest, the remaining of Stage 2 not built from borrowed basin materials, and the rest of Stage 3, will be raised up to 40 mRL using mine waste. The TSF design has accounted for quarterly infilling of the pond and the availability of materials to support the build-up that is substantial enough to ensure containment volume. The pond water will be maintained 2 metres below the crest at any one time through the provision of a transient spillway for each stage.

The TSF is located west of the Jugan Pit and plant site. The site was chosen primarily to provide sufficient storage for tailings, for water management, the availability of borrowed material, its proximity to the pit and plant site, and for its topographical advantage over other areas within the Jugan site.

As part of the final design, a thorough review was made in-house on the constructability of the dam in terms of storage capacity and availability of nearby materials for the build-up. The upstream embankment is basically a combination of suitable borrow material from the basin and NAF-mine waste. The downstream embankment, on the other hand, will be built using both PAF and NAF mine waste. The PAF materials will be encapsulated by NAF in layers while the entire side slopes will be covered in clay. The minimum 1 m thick blanket drain comprising of free-draining materials of about D50 = 5 cm is placed in between the two (2) embankment regions. The filter drain material will be sourced strictly from limestone quarry sites where it will be free of fines.

Each stage, namely Stages 1 through 3, will be provided with a temporary 10 m wide flank spillway along ridges where the embankment is hinged or keyed-in at the side walls, to drain freely excess tailings supernatant and for rainwater run-off. Each series of flank spillway until the permanent spillway at the designed final height, are designed to handle a 100-year flood return period. The nominal figure of 100-year event rainfall intensity around Bau is about 4.08 inches of rain per hour, or 103.56 mm of rain per hour. The catchment area of the TSF is just the pond itself as it is designed to be a ring dam and therefore no external sources of run-off will compromise the TSF from overtopping failure due to insufficient spillway sizing.

Initial seepage analyses were carried out for the upstream embankment and blanket drain at every stage build-up to estimate the seepage through the dam. The purpose of which is to arrive at the sizing and quantity of the weep holes or internal drains and to assess the effectiveness of the perimeter concrete cut-off drain in handling the seepage. The seepage through the foundation of the dam is assumed to be draining freely since the blanket drain will extend to this level and that no pore pressure is believed to affect the dam's overall stability. It is also assumed that no seepage is directed to the dam's floor basin since the entire section will be covered with welded HDPE liner. The seepage from the pond water itself will therefore be passing through the upstream embankment until it reaches the blanket drain section where eventually is conveyed freely to the perimeter concrete cut-off drain.

The proposed TSF in Jugan is sized taking into account the flotation concentrate mass pull-out of 10 % for both Jugan and BYG-Krian pits. Out of a total 9.94 Mt ore grading 1.56 g/t Au from Jugan and 0.94 Mt ore grading 3.10 g/t Au from BYG-Krian, around 10.73 Mt will end up as tailings. The derived concentrate from the processing plant will be treated elsewhere as a gold concentrate.

For the build-up of the TSF, around 2.48 Mm^3 of mine waste and 1.25 Mm^3 derived from the cut materials in the containment pond will be required. This will be done in three (3) stages throughout the current mine life.

The proposed TSF's manner of tailings deposition is planned as a beach-type during its operating years. The tailings dam will be built from 20 mRL to 40 mRL and will be provided with a final spillway at 38 mRL to naturally drain out the supernatant or the tailings water fraction up to the spillway invert elevation and to handle any excess run-off water since the region is known for frequent rains. The final spillway is sized at 10 m wide and the design is based on a peak 100-year flood. A local reference nearby (newly built, 63.2 m high water supply Bengoh Dam situated along the Bengoh Range some 25 km or so from Bau) is used to determine the design parameters.

The 20 m high zoned TSF with combined clay and borrowed sulphide-free material as upstream material and mine rock waste as downstream material will be provided with a 1 m thick blanket drain (D_{50} = 5cm) in between the upstream and downstream. A concrete cut-off drain at the downstream toe to handle seepage from the pond passing through the clay zone and into the blanket drain will also be part of the structure. The function of the blanket drain is to bring down the phreatic head passing through the upstream embankment zone such that disallowing excessive pore pressure from occurring at this side, to deny any form of seepage from the pond water to pass through the downstream embankment, and to prevent the downstream from getting saturated.

Provided at the next page (*Figure 18-2 - Final TSF Configuration along with Other Mine Infrastructure*) is the proposed lay-out of the TSF at final height relative to the Jugan Pit, and an estimate on tailings production after flotation process. A 50 m buffer is given between the final toe of the dam to the final crest level of the pit.

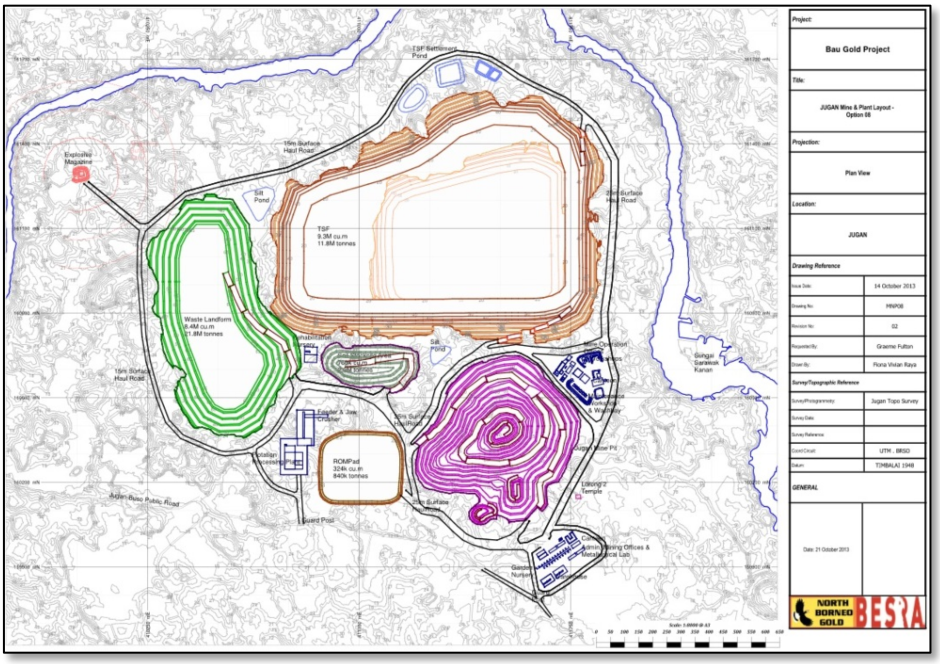


Figure 18-3 - Final TSF Configuration along with Other Mine Infrastructure

The 10.73 Mt combined flotation tails from the Jugan and Bukit-Young pits will be slurried at 35% solids before pumping the tailings to the TSF pond. At 35 % solids it will have an initial estimated density of 1.28 t/m^3. The 20 m high ring dam to contain it has a total capacity of 9.29 Mm^3 or 11.9 Mt. The natural removal of the tailings water fraction at some point will be through the 15 m wide spillway, while desiccation will be achieved by beaching the tailings at the time of deposition. Beaching is accomplished by perimeter spigotting of the tailings. The estimated final settling density of the tailings is 1.70 t/m^3 at 65.4 % solids, which will amount to about 8.38 Mm^3.

The upstream clay zone will maintain a slope of 2.5H : 1.0V or 21.8° while the downstream side made entirely of mine waste will maintain a local slope 2.0H : 1.0V but will be mated with 5 m wide berms at every 5 m lift interval. The resulting flatter downstream slope will be around 18.4° or about 3.0H : 1.0V average.

18.4.5. Planned Construction

Granting the issuance and securement of required government permits and approvals, the construction will commence with an initial phase of preparation. This will start about three (3) months in advance of the scheduled actual construction and the following activities to be carried out are as follows:

- A temporary field office for the contractor and laydown area for earthmoving equipment maintenance;
- Temporary roads to access the site including the borrow area;

- Clearing of the TSF basin, including salvaging and stockpiling of merchantable timbers. Vegetation grubbing and stripping will be done to expose suitable fill material in the borrow areas;
- Topsoil stripping from within the work areas and stockpiling outside the basin in a nearby designated area for future recovery as reclamation material;
- Excavation of the initial key-in (for Stage 1 build up) and re-grading and proof rolling of the dam foundation surface. The nominal key-in depth will be about 2m deep;
- Setting up of temporary environmental control measures for site runoff water management during construction;
- All survey work required for laying out the work and control of lines and grades;
- A pre-construction trial pad will be carried out prior to the full-scale construction operation. It is important to conduct such test pad first to understand the geotechnical properties of the shale fill and its response in the actual field conditions. This will provide first-hand experience that will be useful during the full-blown construction. It will allow an assessment of the best gradation of fill material to use, the optimum moisture requirement to achieve optimum field compaction, and the strength/stability behaviour of the platform;
- Stockpiling of gravel for blanket drain material in a free area within the TSF basin and the supply of gravel will come from nearby limestone quarry operators, which will be delivered to site.

A construction quality assurance and control (QA/QC) programme will be developed specific to the construction of the upstream embankment of Stage 1. It will serve as a guide in the dam construction manual, including the operations and maintenance of the succeeding lifts. This programme will be performed by a qualified 3rd party engineering company.

The full-blown construction phase will be in three (3) stages. The ideal months of construction and build-up for each stage are March to first (1^{st}) half of November, or during the relative dry season.

The Stage 1 will be constructed using borrow materials from the TSF basin. It will be raised from 20 mRL to 30 mRL to store 3.06 Mt of tailings over the next fourteen (14) months of tailings containment while Stage 2 is being completed. Construction of the dam is planned to start ahead by eight and half (8.5) months prior to the actual flotation schedule, where there is already enough initial ore stockpile. Materials to be used in raising Stage 1 will be sourced from the TSF basin, which is also a part of the cut-and-fill philosophy to increase capacity and for cost effectiveness at the same time. Normally, where the geology and the topographic features on the selected site support material availability for borrow sourcing, it is always cost effective to employ cut-and-fill over the sourcing of materials outside for embankment build-up. Plan of Stage 1 is shown in *Figure 18-3 - Plan View of Stage 1 of TSF at Jugan* below.

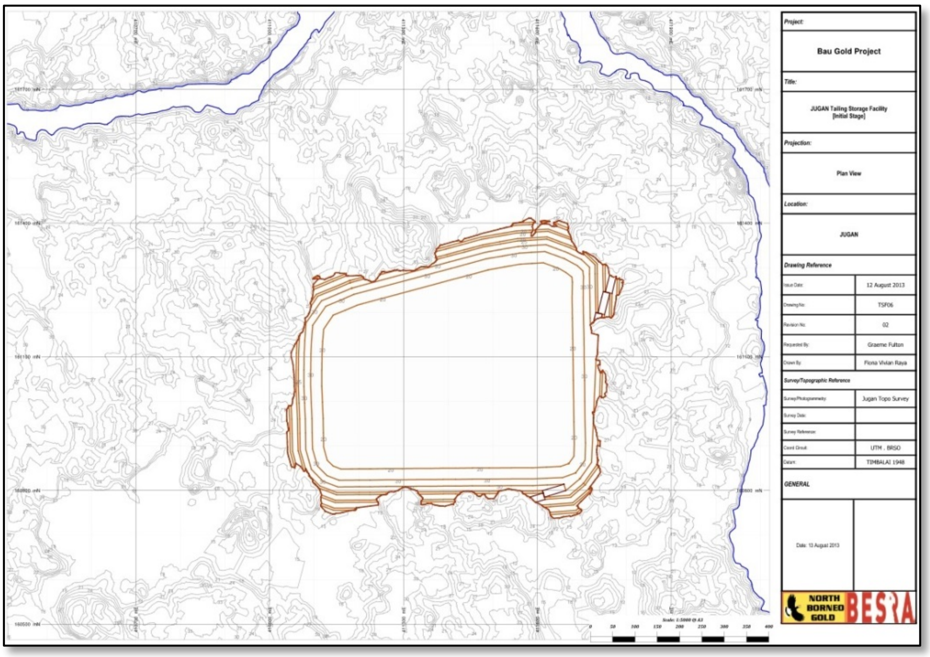


Figure 18-4 - Plan View of Stage 1 of TSF at Jugan

The lithology within the TSF basin is shale, which is similar to the host rock in Jugan Pit. From the aimed cut volume of about 1.23 Mm^3, around 0.86 Mm^3 will be used for the Stage 1 build-up. This may require twelve (12) units of 20-tonner trucks and two (2) units of 3.30 heap m^3 capacity excavators operating on a 6-day per week cycle, while assuming above 90 % efficiency truck-bin loading capacity. Moreover, around 0.64 Mm^3 of the cut materials from the basin, rendered unsuitable for construction since much of it is topsoil will be stored in the topsoil storage dump for future rehabilitation use.

Stage 2 will store around 1.64 Mt of tailings storage and will be built by expanding Stage 1 within the same elevation 20 mRL to 30 mRL. This will require about 1.34 Mm^3 mine waste rock augmented by around 0.39 Mm^3 of fill taken from the borrow area within the footprint of the TSF basin that has not been quarried yet in the build-up of Stage 1. The duration of the construction will be around 10 months. The hills inside the basin will be levelled down to 20 mRL, which is the base elevation of the TSF. Around 0.43 Mm^3 of the cut materials rendered unsuitable, will be stored in the topsoil storage dump for future rehabilitation use. The plan view of Stage 2 is shown in *Figure 18-4 - Plan View of Stage 2 of TSF at Jugan* below)

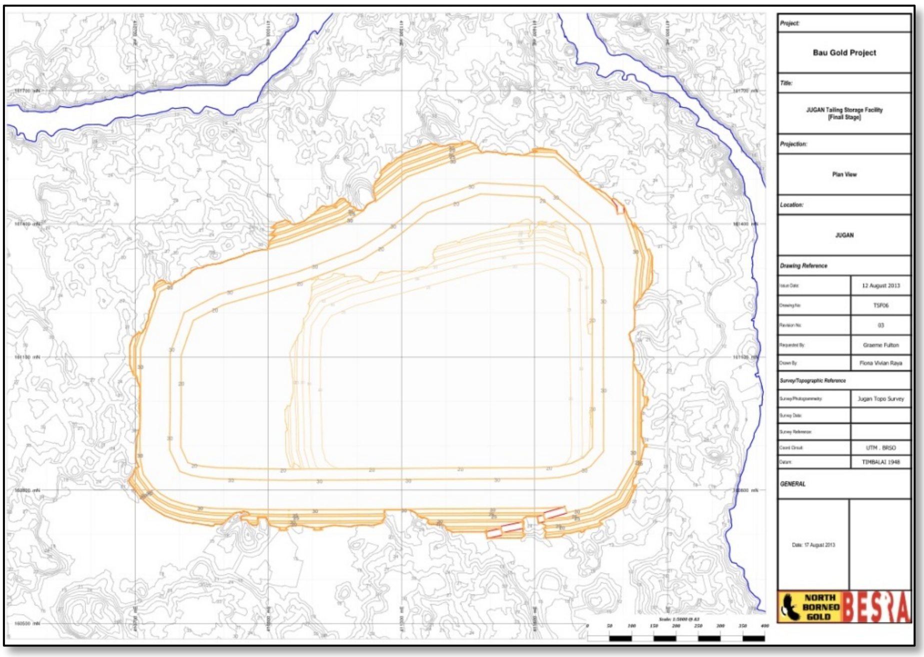


Figure 18-5 - Plan View of Stage 2 of TSF at Jugan

Stage 3, or the final stage, will store an extra 6.03 Mt of tailings and will be built from elevation by raising 30 mRL to 40 mRL over 8 month duration. This will be built solely with 1.53 Mm^3 mine waste rock. The 3^{rd} and final stage of the TSF is shown in *Figure 18-5 - Plan View of Stage 3 of TSF at Jugan*.

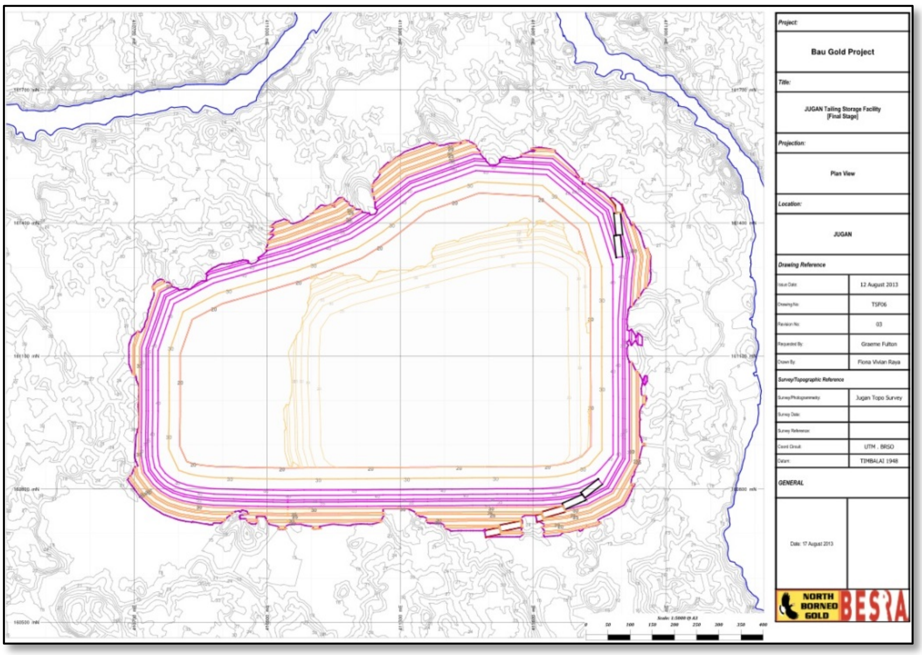


Figure 18-6 - Plan View of Stage 3 of TSF at Jugan

The three TSF stages are also shown in 3D perspective views in *Figure 18-6 - 3D View Stage 1 of TSF at Jugan* to *Figure 18-8 - 3D View Stage 1 of TSF at Jugan* below.

Figure 18-7 - 3D View Stage 1 of TSF at Jugan

Figure 18-8 - 3D View Stage 2 of TSF at Jugan

Figure 18-9 - 3D View Stage 1 of TSF at Jugan

The dam will be built much the same way that the Jugan waste dump is to be engineered but with the compaction slightly higher at a Proctor density of around 97 % compaction. This usually requires six (6) to eight (8) passes with the sheeps-foot roller combined with vibro-rolling through an initial 0.5 m high layer of placed material. Soil testing will also be more rigid, especially in the part where the TSF is to be keyed-in through a 4 m wide x 1.5 m nominal depth throughout the perimeter of the upstream side during the 1st stage construction. Additionally, along the footprints of the upstream and downstream sides of Stage 2, the foundation would have to be slotted in also inside a trench to form the base foundation of the Jugan TSF. The slopes are also gentler on both sides (upstream and downstream) of the TSF compared to the 37° slopes on the Jugan waste dump.

The placement of the PAF and NAF materials in the downstream embankment will have the same manner of handling described in *Section 19.2.3.3.1*.

For Stages 1 and 2, a 10 m wide transitory spillway for each will be provided. Each spillway will be founded into an original ground, in an area where a part of the embankment is abutted to a hill. As mentioned earlier in Section 20.4, the 10 m wide spillway will be more than enough to handle the runoff from a 103.56 mm/h 100 year event rainfall magnitude. The final 10 m wide spillway will be at 38 mRL. The catchment area at final spillway level will be 0.60 Mm^2 for an expected 100-year peak runoff volume of 17.37 m^3/s and 0.12 m^3/s from the tailings supernatant.

Estimated amount of materials needed for the construction are as follows:

- Ring Dam Embankment – 3.73 Mm^3
- Clay as foundation key-in – 35,900 m^3

- Clay as lining – 0.45 Mm^3
- Lime as clay additive – 1,800 m^3
- Blanket drain – 0.37 Mm^3
- Geofabric – 0.77 Mm^2
- Concrete drain
    - 620 m^3 cement
    - 1,760 m^3 sand
    - 4,250 pcs of 4 m long 50 mm dia. PVC pipes to be perforated
    - 2,430 pcs 2 m long x 1 m high x 1 m width rock gabion
    - 4,860 m^3 gabion boulders
- HDPE liner – 0.58 Mm^2
- Stage 1 spillway (14 m top by 10 m bottom width, concrete lined 0.10 m)
    - 60 m^3 cement
    - 90 m^3 sand
- Stage 2 spillway (14 m top by 10 m bottom width, concrete lined 0.10 m)
    - 85 m^3 cement
    - 125 m^3 sand
- Stage 3 spillway (14 m top by 10 m bottom width, concrete lined 0.10 m)
    - 165 m^3 cement
    - 250 m^3 sand

18.4.6. Planned Operations

18.4.6.1. Tailings Deposition Technique

For conventional storage, the tailings are generally discharged from spigots/outfalls located along the TSF embankment. The tailings will be discharged using sub-aerial techniques, i.e. the tailings will be impounded above the water line to form tailings beaches allowing faster drying or desiccation than when the tailings are deposited underwater. Generally during tailings deposition, natural segregation occurs. The degree of this segregation essentially depends on the particle size range of the tailings and the pulp density of the slurry. Normally, for low pulp density deposition, the rougher tails settle closest to the discharge point with the slimes being carried furthest away. The higher density coarse fraction will build up and rest next to the upstream slope while the slimes will stay afloat in the water. The deposition flow rate from the spigot greatly influences this segregation or if flows from adjacent spigots combine as a single stream further down the beach. For sub-aerial deposition this results in a beach sloping downwards from the spigot outfall towards the supernatant pond.

For the Jugan tailings, however, de-sliming will be carried out before tailings impoundment. The plant will have separate system for de-sliming the fines before combining with the coarse fraction in the TSF. The general concept is to de-slime by adding small amounts of ferric sulphate, then sequentially pipe them first for deposition into the TSF prior to the impoundment of the rougher tails above it. Such approach enhances the tailings characteristics by pinning the fines under the heavier rougher tailings.

The application of multiple outfalls for tailings deposition, or spigots, is the most common method used to fill a TSF and this technique will be adopted for Jugan. For conventional storage facilities, multiple spigots help to control the geometry and location of the supernatant pond within the facility. This helps to prevent the pond water encroaching into the embankment and reduces the risk of losing the freeboard. Moreover, pushing the pond water as far as possible from the upstream earth embankment brings down the head, thus the flow line and the pressure of the seepage pathways into the embankment. Multiple spigotting also has the advantage to control the layer thickness of tailings.

To promote faster drying and maximise desiccation as much as possible through the separation of the supernatant, the tailings discharge onto the beaches should form shallow low velocity braided streams that allow the tailings settlement and segregation. Sub-aerial impoundment is generally practiced at TSF sites the world over with the use of multiple discharge points. This allows the deposition of tailings to be rotated between different locations around the facility to allow newly deposited tailings to bleed, dry and consolidate while tailings can continue to be discharged to other zones of the facility. The frequency of discharge point rotation and the number of deposition zones is dependent on the climate, tailings production rate, tailings drying characteristics and the tailings facility shape. The company will employ a cyclic programme of discharge and drying of zones within the TSF to promote in-situ density gain in the tailings and thus maximise storage volume available. Desiccation and shrinkage of the tailings will occur as water is expelled and will become completely unsaturated if allowed to dry for a sufficient amount of time.

The cut-off gravel blanket drain system will intercept seepage and convey it out of the facility promoting faster drying. A new layer of tailings is then placed over the dried area in time corresponding to the TSF discharge zoning and cyclic plan.

On the other hand, sub-aerial deposition exposes the beached tailings to oxygen and water ingress and may allow oxidation of the remaining plus 0.2 % to 0.3 % sulphides. However, the production of any acidic drainage from this low sulphide value (non-acid forming) should never be detrimental to the receiving water environment. Moreover, the exposure of tailings in the sunlight will maximise evaporation and will dissipate or dispel the effects from such impact, if any at all.

18.4.6.2. Embankment Raising Method

Crest raise-up will be by zoned downstream method. This approach does not involve fill placement on top of tailings deposited (newly-deposited and desiccated tailings beaches) while building-up of the upstream embankment zone. This type of earth dam is the most stable engineered landform for tailings containment even in pseudo-static loading, since it is not seated on top of a liquefiable material – the tailings. Apart from its open spillway system, the gravel blanket drain system from the zoned embankment approach provides an effective desiccating and a cut-off medium by letting the supernatant flow through it easily and away from the structure; hence protection of the downstream embankment against saturation is achieved from a design and engineering standpoint. The blanket drain also relieves the upstream embankment of any pore water pressure build-up since as soon as the seepage gets through the drain filter, seepage flows through it easily and pressure is dropped substantially. Moreover, due to desiccation, the tailings will consolidate and strengthen in time compared to when in slurry state during deposition.

The tailings dam, as mentioned, will be raised in stages using a combination of borrow material from the basin and mine waste. By Stage 2 going through the final stage, the raise-up was established from the mine production schedule. The pit development schedule has incorporated these quantity requirements to ensure the supply of suitable fill material. The ultimate crest elevation of the dam provides enough storage for the projected 10.7 Mt plant tailings. The plant at this stage is already into production and Stage 1 is in commission for tailings containment.

The TSF is designed so that the supernatant, or the tailings effluent, and run-off from the catchment pond are draining out freely through the open spillway. At the same time, the water can also be recycled for use in the mill process at any one-time. The plant's minimum daily operating water requirement of 13,400 m^3 water can be augmented sufficiently from the tailings water fraction. The maximum normal operating volume in the TSF pond is based on the seasonal fluctuations in precipitation and the water treatment plant capacity. In average conditions, the maximum water level will occur in the tailings pond during the rainy months of November to February. The excess water will be handled by an open spillway provided for each stage. In a high precipitation region like Sarawak, it is always important that a TSF is provided with a proper spillway system to handle run-off volume in the event of unexpected climatic conditions or operational constraints.

18.4.6.3. Dam Safety & Satisfactory Performance

Aside from the suite of environmental monitoring procedures that is in the programme, to ensure the dam safety and satisfactory performance of the TSF, field instrumentation such as piezometers and prism monitors will be placed in the dam embankment. The piezometers will be slotted through the embankment down to a depth of a metre above the gravel blanket drain. The instrumentation will allow for monitoring of any movements in the embankment, monitoring of seepage water level, assess sudden changes in flow rates along the seepage concrete cut-off drain, and trace clogging along the blanket drain system, if any, or perched water during operation, deactivation, and post-closure. The concrete-lined open spillway system will also be inspected on a regular basis for any form of deterioration, whether by corrosion of concrete, clogging or damage along the entire length of the spillway for appropriate action. The same inspection arrangement will also extend to the concrete blanket drain. The frequency of such inspections on the spillways and concrete drain will be more often during the wet season, 2nd half of November through end of February.

Moreover, the operation of the TSF will undergo annual technical auditing by an internationally recognised tailings dam design and engineering consulting company.

18.4.7. Tailings Closure

As part of the closure plan, the tailings pond will be capped with clay materials at the end of mine life. The clay will be sourced from the stockpile topsoil material where the clayey part is segregated from the topsoil. If the stockpiled clay happens to be not enough, as a contingency, the additional clay required will be sourced nearby; the Jugan area has abundant clayey type soils derived from the weathering of the Pedawan Formation, which is basically shale.

Moreover, as part of the dam's rehabilitation programme, progressive re-vegetation will be carried out during its operating years. With the completion of each stage, planting of grasses along the slopes will be initiated while the next stage is being raised. This measure is also to prevent any erosion of the dam slopes during the rainy season, thus planting of grasses outright after completion of each stage is crucial in ensuring the structural integrity and aesthetic of the dam.

In terms of the final TSF closure, it is also determined that the clay cover of the tailings pond will encourage some water ponding and vegetation growth which thus offer a technically viable yet cost effective closure scheme. The final spillway of the tailings dam will be concreted and will serve a permanent structure in addressing any run-off volume coming from the pond. The water or effluent will be sampled and analysed for water quality and either release the water to the receiving environment, or treat it prior to release.

A continuing post-closure monitoring programme, instrumentation, inspection and audit will be provided for the TSF embankment. This will ensure that the embankment integrity is maintained.

Detailed closure and rehabilitation plan is covered in the Environmental Section of this report.

18.5. Power Supply & Distribution

Grid power is proposed as the permanent power supply for Jugan process plant and mine. Sarawak Energy will provide a 3 phase, 132KV transmission line to the plant site which will be available before the commissioning of the plant installation. Provision of the line would take in the order of 4-5 months upon signing of power agreements.

The incoming 132 KV line will be stepped down to 6,600 volts at the main receiving plant substation which is suitable for SAG Mill and Ball Mill and further stepped down to 415 volts, 3 phase, 50 hertz for the mill process plant downstream equipment's, open pit mine, support facilities, utilities ,tailing dam, warehouse and plant & mine lighting. It is estimated that under normal conditions, the mill process plant, mine and other support facilities will draw power of around 15,840 KW or 101,801,532 KWh per year broken down in *Table 18-1 - Power Requirement Breakdown for Plant & Mine Site* as follows:

Area	Item	Installed Power kW	Utilisation	Power Draw	Annual kWh
			%	%	
10	Crusher	450	70	80	2,207,520
20	Grinding	6232	95	80	41,490,163
30	Knelson-Cyclone	1315	95	85	9,301,916
40	Flotation	3875	90	85	25,967,925
50	Regrinding	1398	90	85	9,368,557
60	Compressor Air	375	80	85	2,233,800
70	Filter press thickener/Tank	206	90	85	1,380,488
80	Filter Press	499	90	85	3,343,999
90	Reagent	45	95	85	318,317
100	Utilities	720	80	80	4,036,608
110	Others	250	85	80	1,489,200
120	Mine	475	80	80	2,663,040
		15,840			**103,801,532**

Table 18-1 - Power Requirement Breakdown for Plant & Mine Site

The site will be reticulated by overhead and underground lines to mine , mill process plant, pant & mine mechanical/electrical maintenance repair shops, warehouse, camp housing, tailings dam pump house, industrial water pump house, domestic water pump house, water head tanks and to the recreational facilities. A transformer substation will be constructed as per power regulatory authority standard to address the line losses and prevent overloading and grounding problems. The full plant and support facilities will be well protected by lightning and earthing protection systems as per approved standards. All electrical equipment will have adequate locked-out mechanism to prevent inadvertent start-up during maintenance and repair activities. Lock-out procedures will be an important part of the safety training program.

It should be noted that the future requirement of 100% of the total connected load, which is 16 MW or more, is included in the determination of the size of the power line and the main transformer receiving station in preparation for any potential plant expansion for the downstream processing of flotation concentrate.

Based on the case studies, it shows that the main grid power is the most economical and the cheapest. However, to maintain flexibility and safe operation of SAG Mill & Ball Mill, a minimum standby source of power is hereby recommended for installation even after the grid power has been stringed to the plant/mine site. 4 units of 2 MW skid mounted portable synchronized diesel generators are hereby recommended for installation which will provide a 50% back-up source of power in anticipation to the grid abnormalities during operations.

Until the power grid is connected, the power requirement will be provided by the company from single unit skid mounted 2 MW, 415 volts, 3 phase, 50 hertz, diesel generator to be placed near the site temporary electrical control room. This generator will then become one of the backup diesel generators in the event of a grid power loss.

Operating cost of power as indicated by Sarawak Energy is 0.07 USD/KWh. This cost makes grid power the cheapest source of power, even after considering the temporary power and the provision of backup generators.

18.6. Site Water Supply

Water will be sourced from the main water pipeline running along the old Kuching-Bau road and piped a short distance (±1 km) to the mine site. Potable water will be supplied either in bottled form or from bulk coolers. Water sourced from the treated ponds and Sarawak River can be used to water rehabilitation plantings or dust suppression during dry weather.

Based on the milling rate of 8,000 tpd at 35 % solids, the clear water impoundment at the TSF is at the rate of 14,857 m^3/day or 14,857,000 l/day equivalent to 619m^3/hr. About 70 % TSF water will be recycled to the mill process plant; pumps which are capable of pumping 433 m^3/hr will be installed. The remaining water will be sourced from the main water pipeline, the mine pit and the bore wells.

A submersible pump is preferred for use. This will be mounted on a floating barge for mobility, which can be moved anywhere within the dam as necessary. To be more flexible, there will be two units of tails pumps to operate alternately mounted on the 2.5-ton capacity-floating barge.

The reclaimed process water and the raw water will be stored in separate closed top steel tanks which then supply water to the modules by identified pumps.

18.7. Offices, Workshops & Ancillary Buildings

The mine site will include a number of offices and structures in addition to the process plant setup and other mining landforms (TSF, Waste landform, pit and stockpiles), and these include:

- Mine operation offices
- Administration offices
- Mine technical offices
- Canteen and change rooms
- Dressing station and first aid facilties
- Computer and equipment control rooms
- Parking
- Process plant control office
- Metallurgical and assay laboratory
- Guard houses
- Storage areas/buildings
- Workshops, truck bays and associated buildings
- Refuelling station
- Special chemical and dangerous goods storage areas
- Other workshops

- Open storage areas (fenced)
- Etc.

18.8. Communications

External Communications

Internet and email services will be linked to a leased line from Telecom Malaysia as per the current situation. The compay server will be situated on site with a backup situated at the old mine site in Bau. All servers will be linked to the corporate server in Singapore. The leased line runs along the old Bau-Kuching road and is close to site with easy connection. Telephone and cell phone reception are available. A conferencing (Polycom) site will also be available on-site as is per current situation.

Internal Communications

A cable network (fibre-optic) will provide the infrastructure for the various operating systems around the mine site, and include:

- WAN/LAN
- VoIP
- Process monitoring and control
- Fire detection
- Security and surveillance (CCTV)
- Office computers
- Fire detection & emergency response
- Etc.

Also available will be radio system both hand-held and on-board mobile equipment.

18.9. Explosives Storage

Explosives will be stored in the designated explosives magazine, which will be constructed and maintained as per international standards and in line with Malaysian government requirements, as defined in the appropriate mining codes. It is envisaged that the magazine and exposives handling will be undertaken by a reputable company or operator or supplier of the explosives (e.g. Orica). The designed position of the magazine is as per distance regulations and it will be secured in accordance with mine security, safety and operational procedures and in accordance with all regulations.

18.10. Site Drainage

Pit water (either ground or runoff) will be collected in the pit sump at the base of the then current floor level. This water will be pumped to sets of water treatment and silt retention ponds. The treated and settled water will then be re-cycled or discharged appropriately based on requirements at that point in time.

Water from the TSF will be collected, transported and treated in the TSF ponds and wetland before discharge or usage.

Surface run off from the waste landform, roads and other areas within the mine site will be collected by a suitable network of drainage channels, culverts and other water flow structures to appropriate areas for collection, treatment and discharge (or re-use).

Runoff and water management will be minimised where possible by use of revegetation, silt retention geo-fabrics and other minimisation methods. This is particularly important during the construction phase where large amounts of ground will be exposed. All discharges will be subject to strict environmental standards and procedures.

18.11. Sewage & Solid Waste Disposal

Solid waste that cannot be re-cycled or is not of an organic nature will be collected in suitable waste skips and containers. This material will then be removed by the council waste collection contractor or a private contractor hired by the company. The refuse will then be transferred to the appropriate council refuse disposal facility.

Organic material will be collected, shredded and composted for use in the ongoing and final rehabilitation works or used for plant propagation in the plant nursery.

Sewage will be either collected from toilets and sewage collection points by mobile sewage contractors or the mine site sewage will be linked to the nearby sewage network.

18.12. Fire Protection & Emergency Response

Standard fire warning and response systems will be in place throughout the mine site with specialist equipment where dangerous gases and chemicals, or electrical equipment are present. Fire suppression systems will be present and will include fire extinguishers (suitable type for hazard), water hoses, sprinkler systems, etc.

A comprehensive fire and emergency response plan will be implemented in conjunction with the local Malaysian Fire and Rescue Department and local Malaysian Police Department. Mines rescue teams will be implemented and trained for on-site situations and will be available to support the above government departments for any fire or emergency response required at the mine site.

18.13. Security

The site will be surrounded by a continuous fence with gates at suitable points. Fencing will also be placed around facilities that required to be fenced off for safety reasons (e.g. pit) or for restricted access (e.g. plant). Security systems will include card access controllers and door mechanisms. Security personnel will be employed to prevent illegal access to the site or to restricted areas. The security systems and personnel will be supplemented by visual surveillance systems (e.g. CCTV) and preventative fencing (barbed or razor wire or electric fencing) where required.

All staff will have photo-ID access cards with the appropriate entry level authorisations which will be carried at all times. Other tagging systems may be used is required. Also standard locks and locking mechanisms will also be applied.

19. Market Studies & Contracts

Marketing of the gold concentrate has been investigated by NBG/Besra and preliminary offers by Chinese processing facilities have been used in the base case financial modelling. These are not disclosed here for commercial reasons.

Transport and shipping of concentrate is discussed in *Section 24.3*.

20. Environmental & Socio-Economic

20.1. Introduction

20.1.1. Project Overview and Concept

The Bau Gold project lies approximately 35 kilometres from Sarawak's administrative capital, Kuching and 7 kilometres from Bau Town, and is linked with a decently laid out network of sealed road. This plays an important role in terms of developing the project into a successful and sustainable mining venture.

The success of a mining operation does not lies with the amount of tonnage of ore a mine produces but rather the successful integration of responsible mining practices and restoring the landscape into a sustainable ecosystem post mining. By identifying any potential environmental and social impacts throughout various stages of project progression, strategies to manage and mitigate impacts related to mining can be implemented such as techniques of remediation and reclamation, including best practice of land management planning and monitoring.

The realistic nature of restoring a mine site to its pristine natural state may be an unlikely scenario. However, with diligent progressive rehabilitation and landscape stabilization of potential physical and chemical hazards, it is possible to establish a diverse and functional ecosystem in accordance to local and international standard of environmental awareness in creating a sustainable post-mining venture.

North Borneo Gold Sdn Bhd is actively pursuing the development of economical mineral resources in the Bau goldfield and at the same time, the company's is committed to undertake an environmental impact assessment to identify issues and data gaps to develop an environmental management and rehabilitation plan in compliance to international standard and local regulatory requirement, in order to create a sustainable environment during post-mining operation and closure. The aim is to create a post-mining environment that is at least equal in environmental value or better.

20.1.2. Key Environmental Aspects

The key environmental aspects that require consideration and attention for the future development of a mine and its associated infrastructure in the Bau area are:

- Identification of environmental impact and constraints associated with exploration and mining activities to ecology, socio-economic including historical / cultural sites in the area;
- The extent of land acquisition and loss of existing land use due to mining and exploration activities, and the likely cost of compensation to landowners;
- The potential for alteration of the existing environment such as elevated levels of suspended sediments or the presence of chemicals including metals (e.g. trace metals

such as Hg, As, and Fe) in streams emanating from the mine site, and the downstream ecological effects;

- The ecological impacts due to the alteration of pre-existing environment, such as land clearing and landscape modification affecting indigenous wildlife species in the area;
- The potential for ore, overburden, and tailings to generate Acid Mine Drainage due to the oxidation of the sulphide minerals and the mitigation measures associated with the rehabilitation and regulatory compliance;
- Socio-economic effects (both positive and negative) on local communities associated with or affected by the mining development, and the costs involved in maximising positive and minimising negative socio-economic effects;
- The scale and nature of rehabilitation scope for the eventual mine closure and post closure monitoring required for all deactivated mined-out areas and associated auxiliary structures and facilities; and
- Post mining environmental monitoring to ensure the success of rehabilitation and the preservation of a sustainable post-mining ecosystem.

20.2. Regulatory Framework

As stipulated in Malaysia's Environmental Quality Act (Prescribed Activities) 1987, Schedule No.11, under mining states that for:

- *Mining of minerals in new areas where the mining lease covers a total area in excess of 250 hectares; and;*
- *Ore processing, including concentrating for aluminium, copper, gold or tantalum;*

….an Environmental Impact Assessment is required to identify potential environmental risks and impact associated with the mining operation. The act stipulates that specific environmental risks and impact should be mitigated through the employment of an Environmental Management System.

North Borneo Gold is a strong proponent of creating a sustainable environment and pursuant of all aspects of environmental compliancy. The list of statutory regulations to be adhered to includes:

- *Sarawak Mining Ordinance 2004*
- *National Mineral Policy 2; and*
- *Environmental Quality (Environmental Impact Assessment) Order 1987 under the Environmental Quality Act (EQA) 1974*

Regulatory requirements for ***Sarawak Mining Ordinance 2004 under section 45*** stated that no development work or mining shall commence for which the lease has been granted until after the approval of:

a) Mine feasibility study under ***Section 55***, if such a study is required by the Authority;
b) Mine rehabilitation plan if so required, under ***Section 108***; and

c) An Environmental Impact Assessment (EIA) if so required under the Natural Resources and Environment (Prescribed Activities)

The National Mineral Policy 2 sets out the principles leading towards sustainable mining. It emphasizes sustainable development of mineral resources, environmental stewardship, and progressive and post- mining rehabilitation. The objectives are:

- To ensure sustainable development and optimum utilization of mineral resources;
- To promote environmental stewardship that will ensure the nation's mineral resources are developed in an environmentally sound, responsible and sustainable manner;
- To enhance the nation's mineral sector competitiveness and advancement in the global arena;
- To ensure the use of local minerals and promote the further development of mineral-based products; and
- To encourage the recovery, recycling and reuse of metals and minerals

Environmental Quality (Environmental Impact Assessment) Order 1987 under the Environmental Quality Act (EQA) 1974

The Environmental Quality Act addresses the prevention, abatement and control of pollution, and enhancement of the environment. The Act places restrictions on

- Atmospheric pollution
- Noise pollution
- Soil pollution
- Pollution of water-courses and water-bodies.

20.3. Existing Environment

20.3.1. Physical Environment

The Jugan prospect, amongst other deposits being explored at the moment, is generally low-lying with some very distinct topographic relief such as steep-sided hills formed by limestone pinnacles. The deposit in Jugan is of Carlin type model which generally consisted of interbedded sedimentary sequence of shale, siltstone, mudstone and sandstone of the Pedawan Formation of Lower Cretaceous age.

These areas are also a mixture of dispersed swampy areas and undulating hills. The typified drainage of the area is best characterized by dendritic system of creeks and riverine directed downstream towards Sungai Sarawak Kanan.

The surrounding steep terrain and high-rainfall eventually causes high rates of runoff into streams creating alternate low suspended sediment carrying loads during base flow conditions to considerably elevated loads following heavy rainfall. It can be surmised that indigenous aquatic fauna in these local fluvial environment are resilient and has adapted to frequent and wide variation in streams flows and fluctuation in turbidity.

The local economy is based on cash products such as rubber, cocoa, poultry and fish farming. Some residents also commute to the nearby city of Kuching for work. Other locally-produced cash crops include plantations of corn, cassava, oil palms and various fruit trees. Food sources are also supplemented by hunting for spotted birds, cats and other animals by the local people.

Apart from areas subjected to mining, agriculture and urban development, the project area, namely Jugan consisted of largely secondary or regrowth forest.

20.3.2. Land Usage

Other than nearby state land and road reserves, the surrounding land project area consisted of dispersed alienated land held by various lease holders.

Scattered and sporadic houses can be found in the immediate surrounds of Jugan prospect whilst 3 permanent settlements of sizeable communities are located within a 3 km radius from the prospect area namely Kg. Buso, Kg. Siniawan and Kg. Buso.

The current land status is comprised of cultivated and un-cultivated land, with the latter being in the majority. Many land parcels have absentee lease holders with no activity occurring on this land. Large portions of the land in and around Jugan comprised of secondary forest or re-growth scrubs and bushes.

Cultivation activities appear to be limited to small scale fish ponds and minor fruit growing, pepper & rubber trees supplemented by poultry farming and swine rearing including subsistence cash crops cultivation of corn and cassava.

The Bau District has had long association with mining and has been part of the district legacy since 1800's. Current and historical land use of the area is gold mining, which has occurred at least since the early 19th century. Evidences of both past and current mining activities can be found throughout the district, e.g. the deactivated Tai Parit pit and Lake Bekajang that served in the past as a tailings storage facility including several limestone quarries for aggregate production currently active.

20.3.3. Climate

Long-term rainfall data available from Kuching indicates an annual rainfall value of around 3100 mm minimum to 5100 mm maximum. Although significant rainfall may occur at any time of the year, the highest rainfall months are December and January, with the highest monthly rainfall occurring in January. The driest months are from April to September. Average monthly rainfall exceeds average monthly evaporation rates for all months of the year.

20.3.4. Biological Environment

Recently completed ecological survey of Jugan presented a diverse ecosystem through identification of approximately 20 known floras and at least 7 species of mammals including 5 species of reptiles and 7 species of birds whereas 7 species of aquatic fauna was identified comprising of mostly fresh water fishes and variety of crustaceans.

Amongst the flora and fauna identified at Jugan, 2 species of flora were categorized as endangered and 1 species of reptiles is listed as vulnerable. At least 1 type of amphibians is listed as a threaten species. None of bird species and aquatic faunas was listed as endangered.

20.3.5. Human Environment

The Bau District is approximately 884 km^2, with a municipal administrative centre located in Bau town. Bau is approximately 35 km southeast of Sarawak State capital Kuching. While the population is multi-racial, the main ethnic-group consist of Bidayuh (68.8 %), followed by Chinese (17.4 %) and Malays (7.8 %). The remaining 4 % constitute of other races and non-Malaysian.

Based on the Socio-economic study, statistics shows a significant proportion of the populations of Bau tend to migrate outside of Bau, particularly to Kuching for reasons of employment and improved standard of living. Such opportunities were possible due to:

- An improved road system which facilitates movement of people between rural and urban areas;
- Poor employment opportunities in Bau area relative to Kuching or other developing areas; and
- A tendency for government officials and white-collar employees to prefer living in urban rather than rural locations.

20.3.6. Socio-Economic Environment

The ethnic background of Bau Town and its surrounding areas comprised of the majority Bidayuh from the Dayak ethnic group followed by Chinese who are mainly descendants of early miners brought in the mid to late 19th Century to exploit the gold and antimony deposits at Bau and Sarawakian's Malays.

The known industries in the Bau district are limestone quarrying, small scale fish, poultry and swine rearing, rice farming, palm oil and rubber production.

The Jugan Project generally has good infrastructural aspects both within Bau Township towards Kuching. The main infrastructural features are:

- Regular and reliable international air services to Kuching from Kuala Lumpur, Singapore and Indonesia and the Airport is only 40 minute drive from the project area;
- Two ports with good dock and storage facilities;
- Two main sealed trunk roads from Kuching for delivery of supplies, heavy plant and equipment to the plant site;
- Excellent labour and engineering support services;

- Easy Accessibility – project extremities are less than 20 minutes' drive from the exploration base. All important mines and gold prospects are linked by road;
- Area is serviced with municipal power and water;
- The official language in Sarawak is Bahasa Malaysia, but most local communities have an understandable communication knowledge of English;
- Well educated workforce;
- An active quarrying industry focused mainly on limestone and marble for roading aggregates and agricultural purposes;
- Ready supply of earthmoving equipment that supports the quarrying industry; and
- A local labour source with mining experience gained from the quarrying industry and previous mining and exploration companies previously active in the area.

Based on the socio-economic study conducted, majority of respondent indicated farming does not constitute as a major source of income but rather as a supplementary means to acquire additional financial enrichment. Other income generating source comes from small scale business enterprise and self-employment such as carpentry and masonry including public and private sector executives. *Table 20-1 - Percentage Split of Economic Activities* shows the economic activities and employment percentage:

Economic Activities	Percentage of Employment
Private Sector	34%
Self-employed	34%
Farming	23%
Public Sector	11%

Table 20-1 - Percentage Split of Economic Activities

20.3.7. Community Development, Liaison & Public Relations

Stakeholders identified through socio-economic study and upcoming Environmental Impact Assessment (EIA), associated with the proposed Jugan prospect, will be encourage to participate in the planning and decision making in regards to issues relating to socio-economic development and environmental awareness in order to:

- Understand the likely environmental, social and economic impacts of mine closure on affected communities;
- Take into consideration the interests and expectations of the respective stakeholders;
- Ensure the process of closure occurs in an orderly, cost-effective and timely manner;
- Establish a set of indicators which will demonstrate the successful completion of the closure process;
- Establish completion criteria to the satisfaction of the responsible authority;
- Ensure support for closure decisions; and
- Enhance public image and reputation.

Stakeholder involvement and engagement process has been initiated and will continue through the feasibility study, EIA and throughout mining operations effectively to reduced misunderstanding and increased awareness via regular meetings, educational presentations, information releases, website updates as a mean to address any arising issues relating to socio-economic and environmental concern.

20.4. Environmental Assessment of Major Elements, Impacts & Mitigation

The nature of mining operations in transforming a landscape is a common occurrence. Although the restoration of a particular landscape to its natural state may not be a realistic solution, the next logical alternative would be to implement best mining practice and environmental rehabilitation framework to formulate an achievable outcome to restore the landscape and establish diverse functional and sustainable ecosystem post-mining.

The realisation of past mining practices and their lack of enthusiasm to identify and appropriately plan for a rehabilitative mine closure. This has since prompted an ever increasing awareness and the need to include environmental, social and economic issues into mine development planning.

As such, some of the main mining elements which require specific attention are:

- Open Pit
- Waste Disposal
- Tailing Storage Facility
- Mining Infrastructure (Processing Plant & Buildings)

The Environmental aspects relating to the above are:

- Acid Mine Drainage (AMD)
- Landform Stability (Slope stability and Erosional Control)
- Land Rehabilitation (Re-vegetation & Conservation)
- Dust and Noise
- Ecological

A successful mining venture legacy will be measure through the implementation and integration method whereby environmental degradation aspects has been properly addressed and avoid a costly mine closure.

The major mining elements and their related environmental aspects are covered in the following sections.

20.4.1. Major Elements

20.4.1.1. Open Pit

Overburden removal of topsoil and non-mineralised rock in conjunction with the extraction of economic ore is a significant part of the mining operation. The overall configuration and extent of the open pit depends on the geological setting, size of the deposit and related economic aspects.

The proposed Jugan mine pit is approximately 24 ha. This size of pit is at the lower end of the scale when compared with other open pit mining around the world. The open pit is estimated to produce approx. 9 Mt of ore and 22 Mt of non-mineralized rock (waste) material over the entire mine's operation. The final depth is provisionally estimated at approx. 200 metres depth below surface. This is based on the current feasibility study and subject to change depending on future mine planning and possible discovery of new extension of mineralized ore.

Rehabilitation & Post Mining Options

The open pit will be part of an altered landform throughout a mine's life until cessation of mining. There are a number of conceptual rehabilitation options such as transforming it into a recreational lake to support local tourism and recreation, or for agricultural and commercial activities such as re-stocking with common fresh aquatic species for use in supply to local fish farmers. An alternate option would be to utilise the open pit as a disposal site for residential and/or industrial waste with each layer being capped with clay liner to prevent seepage and act as containment for waste.

20.4.1.2. Waste Disposal Landform

The waste disposal landform is anticipated to be a visible landform feature at the end of mining. The desired goal would be to integrate and blend the landform into the surrounding topography and environment.

Progressive rehabilitative schemes can be conducted concurrently with regular mine operations. These include slope grading to minimize erosion and surface runoff and progressive re-vegetation.

A diversion drainage channel or spillway will be constructed surrounding the waste disposal landform to regulate and divert runoff to designated silt and settling ponds to contain possible runoff. This surface runoff water will be subsequently sampled and sent for laboratory analysis to ensure compliance to local and international environmental standards.

The waste disposal landform will be constructed in a series of lifts with 5m catch berms. Currently, the projected final height of the waste disposal landform will be approximately 70m in height. It will be built in a bottom-up construction method where thin layers will be compacted and loose spoils on the slope, will be trimmed at every designated height interval.

All potentially acid producing mine waste (PAF) will be overlain by a metre thick non-acid producing mine waste (NAF) and will be encapsulated with clay-lining and covered with topsoil for eventual re-vegetation.

Rehabilitation & Post Mining Options

There is various reclamation and rehabilitation options available for consideration for the waste disposal landform, some examples are:

- Return land to natural state as part of a park area (in itself or as part of the overall rehabilitation strategy), which could include planting and re-introduction of native species; or
- Create recreational facilities, by itself, or in conjunction with the rehabilitation of the other areas, e.g. hiking and mountain biking trails; or
- Possibility of transforming the land into some form of agriculture usage e.g. for grazing of livestock; or
- Usage of the land as part of a tourism venture, e.g. Bidayuh longhouse and village; or
- Combinations of the above options.

20.4.1.3. Tailings Storage Facility

The tailing storage facility (TSF) will be an integral part of mining operations and contains the processed material after the ore has been extracted. As such, the TSF needs to be a stable structure to properly hold processed material. This structure will be rehabilitated as part of the overall plan.

The current proposed TSF is projected to cover an area of approximately 75ha with an estimated height of approximately 20m. It will comprise an impoundment bund constructed from the same waste rock / un-mineralized overburden at Jugan Hill and contain the processed tailings. Waste rock of will be compacted and stacked-up in 1m lifts. Localized clay will be utilised as lining at the bottom of the TSF and acting as “Capping” to seal any PAF material in-place and to prevent seepage. Progressive re-vegetation of slopes at every suitable lift interval will be conducted, after the clay-lining.

The embankment will rise or expand overtime to accommodate tailings production. Aspects such as slope stability, erosional control and vegetative cover will be some of the elements to be addressed in the progressive rehabilitation framework, which can be initiated concurrently with the expansion of the TSF embankment.

Rehabilitation & Post Mining Options

One conceptual option for environmental sustainability would be to convert the TSF into wetlands and transform it into a wildlife sanctuary with proper rehabilitation programmes and reclamation design. Alternately, the rehabilitated land may have some agricultural or commercial usage, e.g. livestock farming, or some other land use may be devised to fit in with the overall rehabilitation.

20.4.1.4. Infrastructure, Building & Processing Plant

Buildings and other infrastructure elements will be decommissioned and demolished at the end of mining operations. A few structures will remain post closure to be utilized as monitoring and treatment facilities e.g. pump stations to regulate drainage. Vacant areas will be graded to ensure land form stability and reclaimed/rehabilitated through various stages of the re-vegetation scheme.

Rehabilitation & Post Mining Options

Operational and re-useable mechanical component including building material will be salvage and may be potentially re-used in other mining projects owned by the company or re-sale to potential buyers. Scraps metals and other recyclable leftover shall be delivered to gazetted recycling facility for proper handling.

Vacant areas are likely to be re-contoured, graded and re-vegetated. Some potential uses may be:

- Convertion back into agricultural land or for commercial enterprises;
- Recreational facilities either individually or in conjunction with the other elements;
- As a park and associated nursery for re-introduction of local flora and fauna, etc.

20.4.2. Environmental Impacts & Mitigation

20.4.2.1. Acid Mine Drainage

Acid mine drainage (or acid rock drainage) is a process whereby minerals containing sulphides are oxidized due to exposure to atmospheric conditions. Contributing factors to AMD can be classified into 3 categories, namely Primary, Secondary and Tertiary. The Primary factors involved sulphide oxidation, where sulphide bearing rocks are exposed to atmospheric oxygen, which subsequently initiates chemical reactions producing sulphate and acidic conditions. Secondary factors are those involving minerals with acidity neutralizing characteristics, which altered the chemical composition. Tertiary factors involved physical conditions such as topography and climate, which accelerates sulphides oxidation and subsequently the migration of oxidized products.

The extraction of mineral ores and associated sulphide bearing minerals potentially increase the rate of AMD. Earth moving operations such as stripping of waste rock may also contribute to AMD due to close geological association to sulphide wall rocks being exposed in the mine pit and within the waste dump. The most cost effective solution, is to minimize AMD through proper mine planning, suitbale containment or treatment and the integration of hydrological controls covering all stages of mining.

It is essential to understanding the source, pathway and receptors in order to properly address AMD effectively. By understanding the site specific mechanism for acid generation, suitable control strategies can be develop to address AMD. Specific and detailed analysis of this has been undertaken for NBG, by SGS Environmental in Perth, Australia.

The common laboratory tests, conducted by SGS, to determine the acid generation or neutralization capacity of a rock or minerals are (i) Static (ii) Kinetic Tests.

i. Static Test
- Total Sulphur (in Sulphides)
- Net Acid Generation (NAG)
- Net Acid Producing Potential (NAPP)
- Acid Neutralization Capacity (ANC)

ii. Kinetic Test
- Sulphides Oxidation Rates
- Rates of Metal and Mineral leachate Generation
- Biological Acid Production Potential

Mitigation Options

There are a number of commonly utilised strategies available to prevent or mitigate the impact of AMD. These include avoidance of disturbance, dry covers, clay lining, underwater storage, neutralization, collection and treatment. Avoidance of disturbance of a potential acid forming material is always the preferred option. PAF materials are inert as long as they are not exposed to oxygen and water. If disturbance is inevitable, minimisation of acid drainage requires control of oxygen diffusion, water infiltration and neutralisation of existing and potential acidity.

Preventive measures in dealing with AMD comprise of the following mechanisms:

- Exclusion of oxygen from sulphidic mine wastes;
- Control of water influx and hydrological management to minimized migration and transport of oxidized products;
- Neutralization of AMD with alkaline materials; or
- Encapsulation by utilising “Wet or Dry” covers.

As part of the Feasibility Study and the baseline work for the EIA, samples were collected and sent to an internationally accredited laboratory (SGS Environmental in Perth, Australia) for testing to assess potentially acid forming (PAF) or non-acid forming (NAF) properties on non-mineralized rocks (waste) to be utilize as buffer or barrier to reduce and minimize the effect of AMD.

Recent static geochemistry laboratory test results shows majority of the waste rock or un-mineralized rocks indicates these have strong NAF (non-acid forming) properties or at least acid consuming.

The incorporation and placement of NAF to encapsulate any potential PAF in alternate layers has the capacity to reduce or at least moderate acid drainage. The layered NAF will act as “cap” of dry cover material in AMD seepage mitigation. The design strategy is to prevent generation of acid leachate and prevent un-control seepage into the surrounding environment by constructing alternating layers of NAF and PAF with clay acting as barrier or seal. These will be followed up with re-vegetation of types of resilient and plants with acid consuming ability (like the local ferns on the natural exposures) to limit AMD as well as to enhance slope stability and erosion prevention.

Another solution would be to utilize water as a cover (wet cover). Sulphide material would be rendered unreactive due to reduced availability of oxygen. By combination of passive and active methods, including the incorporation of inert material such as quart sand, fly-ash and calcium carbonate; or application of alkaline material (e.g. limestone), will increase the AMD stabilisation or neutralisation impacts.

20.4.2.2. Landform Stability (Slope Stability & Erosional Control)

The factors affecting land form stability depend upon the landform slope gradient, drainage or surface run-off control and erosion prevention. Changes to elevation, slope angles and lengths brought about by excavation, dumping and reshaping may possibly render the new land surface susceptible to erosion if suitable measures are not undertaken. The desired goal should be a stable and non-erosive landform that conforms and blends into the surrounding topography and environment

When considering slope stability, geological characterization, design parameters and slope geometry play an important role. Another important attribute in slope stability is the drainage pattern. Since the natural drainage pattern has potentially achieved equilibrium with the natural surroundings over time, changes brought by excavation, alteration of landform in terms of elevation and geometry will render newly exposed land surface susceptible to erosion unless handled or mitigated properly.

Three major elements when dealing with slope stability aspects are, the:

- Jugan Mine Pit;
- Waste Disposal Area; and
- Tailing Storage Facility.

Mitigation Options

There are three (3) major criteria for determining slope stability and design. The first being the structural domain of lithological contact boundaries such as faults, shear zones and planes of weakness. The second is mining pit wall orientation as rock within the same structural domain exhibit different degrees of instability at different rock face angles. The final and third criterion is the operational factor whereby the slope of an area, within the open pit and waste dump, is interactive and revolves around mine planning.

Key mitigation steps are:

- Bench design in the open pit – final benches are left with suitable final widths to assist with pit slope stability and act as rock catchments for any material that is dislodged;

- Slope angles – all slope angles (pit, TSF, waste landform and others) are designed, based on their constituent materials, to prevent slope failure, in conjunction with appropriate drainage and suitable coverage;
- Slope management – particularly in the pit it is important to identify any potential failure areas such as faults, joints and other weak rock elements, and introduce or apply observation and rectification strategies to ensure these potential failure areas are managed adequately; these will include visual mapping, use of measurement devices (e.g. inclinometers), slope design and orientation and support strategies (e.g. cable bolts, shotcreting, etc.);
- Water management – an integrated water management and drainage design and implementation to control water flow in such a manner as to limit the erosional or failure impacts; these include drainage elements (surface and sub-surface), vegetation or other surface binding (e.g. geo-fabrics), ponds and silt retention barriers;
- Vegetative and topsoil cover – prevention of erosion is best mitigated using vegetation to ensure a cohesive and physically stable surface, particularly for runoff; the plan is to minimise exposed land surfaces as much as possible and to re-vegetate land at the earliest possible moment after exposure, both finally and in an ongoing manner.

It should be noted that the above are intrinsically linked and will not be implemented individually but holistically in a combied and planned manner.

20.4.2.3. Land Rehabilitation (Re-Vegetation & Conservation)

The ultimate objective of a successful mine closure plan is to “return” the altered landscape as close as practical to its natural condition into a sustainable ecosystem or as a substitute landscape that is natural in form and principle. The re-vegetation scheme will cover a wide range of areas such as mine pit, waste dump, tailing embankment and areas affected by infrastructures and engineer design such as building, plant facilities and access roads.

Mitigation Options

Disturbed areas such as waste dump and excavated overburden will be re-vegetated to further create slope stability and as a means of erosion control. As part of the re-vegetation scheme, native plant species shall be selected where applicable or those that enhance the landscape.

The scope will be to provide sufficient vegetative cover for natural and altered land to create soil retention and slope stabilization. Therefore, re-vegetation framework will encompass various levels in order to achieve a successful closure plan. Re-vegetation will comprise the bulk of the rehabilitation program which will eventually be extended post mining to ensure exposed surfaces are covered and vegetated to reduce soil erosion and create stable landforms.

20.4.2.4. Air and Noise

Dust particulates in the air may be generated from mining activities within and surrounding the open pit (drilling and loading of haul trucks), and from haul truck movement along haul roads, or from transport trucks into and out of the mine site. Activities such mechanical disturbance of rock and soil materials from bulldozing and blasting will also contribute to create dust. Wind blowing over exposed ground or stockpiles of fine aggregate, especially during dry season, may generate additional dust.

Noise levels will primarily be emanating from the operating plant. Additional noise from vehicular movement, other machinery and rock blasting will potentially increase noise levels.

<u>Mitigation Options</u>

Identified noise and dust issues will be mitigated by implementing the measures listed below. Also, by establishing strategically stationed air and noise monitoring points, data can be collected to formulate solutions for ongoing noise and dust issues.

Common methods that are likely to be implemented in some form, to minimize releases of airborne particulate matter (dust), include:

- Selective spraying water on possible dust generation surfaces (e.g. haul roads) to maintain sufficient surface moisture;
- Vehicles to go through a “Wheel Wash” or “Wash Bay” to reduce dust and dirt from the vehicle wheels and undercarriage;
- Using tarpaulin covers for trucks to minimise the release of dust or to prevent material falling out onto roads during the transportation of material to/from the mine site;
- Minimising the amount of exposed soil or un-vegetated surfaces;
- Establishing speed limits on unpaved surfaces that are sufficient to minimise dust from vehicles; and
- Regular and systematic air monitoring to ensure compliance.

Noise mitigation methods include:

- Impose speed limits to minimise vehicle’s noise;
- Equipment with lower sound power levels will be used in preference to more noisy equipment;
- The on-site road network to be maintained to limit body noise from empty trucks travelling on uneven internal roads;
- Implement vehicle muffling or noise reduction equipment to limit/reduce noise emanating from machinery and vehicle exhaust pipes;
- No blasting to be conducted at night time, and only at designated times;
- Regular vehicle and machinery maintenance to ensure proper equipment operation;
- Construct noise bunds, fencing and vegetation planting to act as noise barriers where applicable; and
- Processing plant and other structures with equipment that may potentially generate noise to be covered in, and if applicable to have noise reduction elements installed.

20.4.2.5. Visual Aspects

The mining operation, including the initial construction phase, may possibly have some visual impacts on the surrounding area. These visual impacts need to be addressed and mitigated.

<u>Mitigation Options</u>

Typical visual mitigation measures are:

- Planting of vegetation to screen the pit, plant and other elements from the surrounding areas affected;
- Construction of suitable fencing or walls to block the visual impact;
- Building or creating visual bunds (including vegetation) to further assist in the screening of visual impacts; and
- Vegetation and rehabilitation of mining elements in a progressive manner to reduce any obvious visual impacts.

20.5. Environmental Monitoring & Management

20.5.1. Environmental Management Plan

The main objective of an Environmental Management Plan is to develop control strategies and mitigation measures to deal with potential environmental impacts and restore the land to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety;
- Limiting the production and circulation of substances that could have negative impact on the environment
- Restoring the site to a condition in which it is visually acceptable to the community; and,
- Develop a rehabilitation plan which focuses on land reclamation, solid waste storage, tailings containment, and drainage control to prevent erosion

Rehabilitation is a continual process whereby work can commence at any stage throughout mining operations, provided it is appropriate and physically achievable. It is a dynamic and evolving framework which addresses the various facets of environmental issues.

Essentially, all disturbed earth and "borrowed land" will to be rehabilitated not just for the sake of regulatory compliance and adherence to internationally accepted mining best practice. It is to "return" or transform the land into a sustainable ecosystem post mining. At the onset of mining activities, as certain areas will not be disturbed and rehabilitation activities can occur during the mining phase where applicable, i.e. progressive rehabilitation. However, the bulk of the rehabilitation work will be conducted after mining and associated other activities are complete. In order to achieve a successful and sustainable objective, the following section will further discuss various environmental monitoring measures.

20.5.2. Environmental Baseline Data Management

Environmental baseline monitoring was undertaken to collect background data on the surrounding land, which will further enhance rehabilitation strategies by identifying any pre-existing concerns or situations.

A governmentally accredited and recognized environmental consulting firm (Chemsain Sdn Bhd) has been engaged to conduct laboratoratory testing and environmental monitoring at the Jugan mining project. This consulting firm is also planned to provide additional support and services in the studies and compilation of the EIA report. List of baseline studies include;

- Soil Monitoring
- Surface Water Monitoring
- Ambient Air and Noise Monitoring
- Weather and Rainfall Recording
- Ecological Study
- Social Economic Study

Additional studies and analysis work has been planned in the near term to meet EIA requirement and regulatory compliance. Monitoring program is by no means a short term undertaking but rather a continuous process until cessation of mining to ensure all environmental aspects are properly manage and the land is self sustaining.

As the Jugan mining project expands, and transform into a fully functional mine, other environmental monitoring scope will be incorporated to better evaluate the altered environment and implement site specific rehabilitation parameter in compliance with local and international regulatory standard. Other scope of environmental monitoring includes:

- Groundwater Monitoring
- Soil Monitoring
- Water Discharge and Sedimentation Monitoring
- Vibration Monitoring
- Ecological Monitoring of Flora and Fauna
- Air and Noise Monitoring

20.5.3. Environmental Monitoring

Collection of baseline environmental data will be based on parameters detailed and stated in the EIA as approved by local regulatory governing bodies. Data collection will be a combined effort of mining personnel and appointed environmental consultant recognized and approved by governmental regulatory department. Methods and parameters of environmental data collection are listed below.

The majority of baseline envonronmental work has been completed to date, with only a few minor studies to be undertaken. Ongoing monitoring will also need to e conducted and this is also summarised below.

20.5.3.1. Baseline Surface and Groundwater Monitoring

Parameters to be included are pH, dissolved oxygen, turbidity, conductivity suspended solids, Biological Oxygen Demand (BOD,) Chemical Oxygen Demand (COD), oil and grease, metals (As, Cd, Cr, Cu, Fe, Pb, Ni, Hg, Zn), coliforms and cyanide (specifically weak acid dissociable WAD cyanide).

Locations of sampling points have been and will be positioned to monitor the following:

- Operating plant discharge to TSF;
- Seepage from waste rock and waste disposal;
- Mine discharge water;
- Discharge and seepage from TSF;
- Selected locations upstream and downstream of discharge to surface water.

Surface water monitoring will be supplemented by groundwater monitoring and sampling as stipulated in the EIA and EMP. Accumulated data will be incorporated into the EIA as a baseline for future monitoring.

In order to characterize the groundwater characteristics and flow regime, a detail hydrogeological study will be incorporated as part of the EIA study to determine the water level drawdown and potential ground subsidence effect due to dewatering associated with the mine pit operation. Future proposed ground water well(s) may be installed in the periphery of the mine pit for aquifer test pumping calculations and on water recharge rates.

20.5.3.2. Baseline Soil Monitoring

Prior to the commencement of mining operations, a regional baseline soil sampling program was completed to characterise the geochemical and geotechnical properties of the soil in the project area. Data collected will be incorporated into the EIA to be utilised in subsequent environmental monitoring program and rehabilitation schemes.

Soil geochemical test parameters included:

- pH
- Metals (Fe, Ag, Mo, Mn, Mg, Cd, Pb, As, Al, Fe, Hg, Cu, Zn)
- Nutrients (N, P, Na, K, NO3, S).

Geotechnical properties parameters analysed included:

- Moisture Content
- Specific Gravity / Density
- Atterberg Limit (Liquid Limit & Plasticity Index)
- Constant Head Permeability Test

Data compiled may be utilized for future re-vegetation and rehabilitation undertakings and serve as baseline data for the EIA documentation.

20.5.3.3. Water Discharge & Sedimentation

Sediment samples will be taken at pre-determined location as indicated in the EIA to monitor the metals concentration in the river sediments. This will establish the baseline metal concentrations associated with mineralised rock being deposited into the river from naturally exposed outcrop.

Monitoring location will be set up at strategically determine location as identify in the EIA to monitor effluent discharge outlet such as:

- Processing Plant
- Tailing Storage Facility
- Spillway
- Laboratory

Sediment sampling is necessary to characterise sedimentation profiles on nearby streams and rivers to determine sedimentation overload especially during rainy season which will cause an increase in amount of suspended solids and turbidity, whereby affecting oxygen demand levels for aquatic life. Monitoring stations will be proposed at positions:

- Upstream
- Midstream
- Downstream

20.5.3.4. Air, Noise & Vibration

Parameters monitored and to have ongoing monitoring include, noise level, air such as SO2, NOx, CO, wind direction and including wind speed to predict dispersion effect. Daily rainfall, humidity, evaporation and barometric pressure measurement will be monitor and collected should be established at the mine site offices in order.

The noise at selected work sites and workshops around the mine should be periodically tested at least twice per year and when new major machinery is brought on-line. This should allow the correct level of noise protection equipment to be purchased and issued to staff working in these areas.

Potential adverse noise effects from plant operations especially at night time will need to be identified and dealt with accordingly. Adverse effects from mining operation vehicles should to be considered as well. Air and Noise aspect and mitigation measure had been discussed in previous section.

Blast vibration should be measured around the site and at nearby locations to ensure compliance with vibration levels. It is envisaged that modern blasting methods are expected to produce low vibration levels.

20.5.3.5. Ecological Monitoring of Flora and Fauna

During construction and operational phases, environmental monitoring will be conducted to measure and assess any potential adverse concern related to natural flora and fauna. This will include natural habitat monitoring of the Sarawak Kanan River (Sg. Sarawak Kanan) to identify possible ecological impact in the surrounding vicinity.

Baseline monitoring will be an effective tool to capture baseline data which can be incorporated into mine planning to develop a co-existence scenario with the natural habitat.

An ecological study has been completed for the Jugan project whereby species of flora and fauna were categorized to assist in future conservation effort.

20.6. Site Closure & Rehabilitation

A conceptual Mine Rehabilitation Plan (MRP) has been submitted to governmental departments and agencies for review, has been updated twice, and has been accepted. The current conceptual MRP is subject to amendment pending results of the detailed EIA and Feasibility Study work and actual operations. Any changes will be communicated to the authorities in the form of a revised or updated MRP.

The main focus after completion of a successful mining operation would be to return the “Borrowed Land” and disturbed area to an environmentally sustainable and properly rehabilitated landscape with suitable land usage. Goals set out include:

- Regulatory Compliance – Meet and exceed all regulatory requirements and standards
- Environment Rehabilitation – Develop a sustainable and rehabilitated land
- Stakeholder Agreement - Active stakeholder and community engagement
- Public Safety - Elimination of hazard and return the land safe for future use

Decommissioning and closure of the majority of the facilities and infrastructure will commence only after cessation of mining. However, some ongoing rehabilitation will also be conducted as required. Others facilities such as power station, water pump station, TSF and detox pond need to remain long after closure to facilitate rehabilitation and reclamation. Sequential closure will be based on governmental regulatory institution and stakeholder agreement. Post closure monitoring and maintenance will continue to ensure compliance and achievement of all regulatory criteria successfully.

Figure 20-1 - 3D View: Indicative Sequence Showing Current Topography, Mining Infrastructure & Rehabilitated Site below is an indicative closure option – the sequence of images shows the original topography, the mining infrastructure and landform amendments, and finally an example of the rehabilitated topography post-mining and after closure.

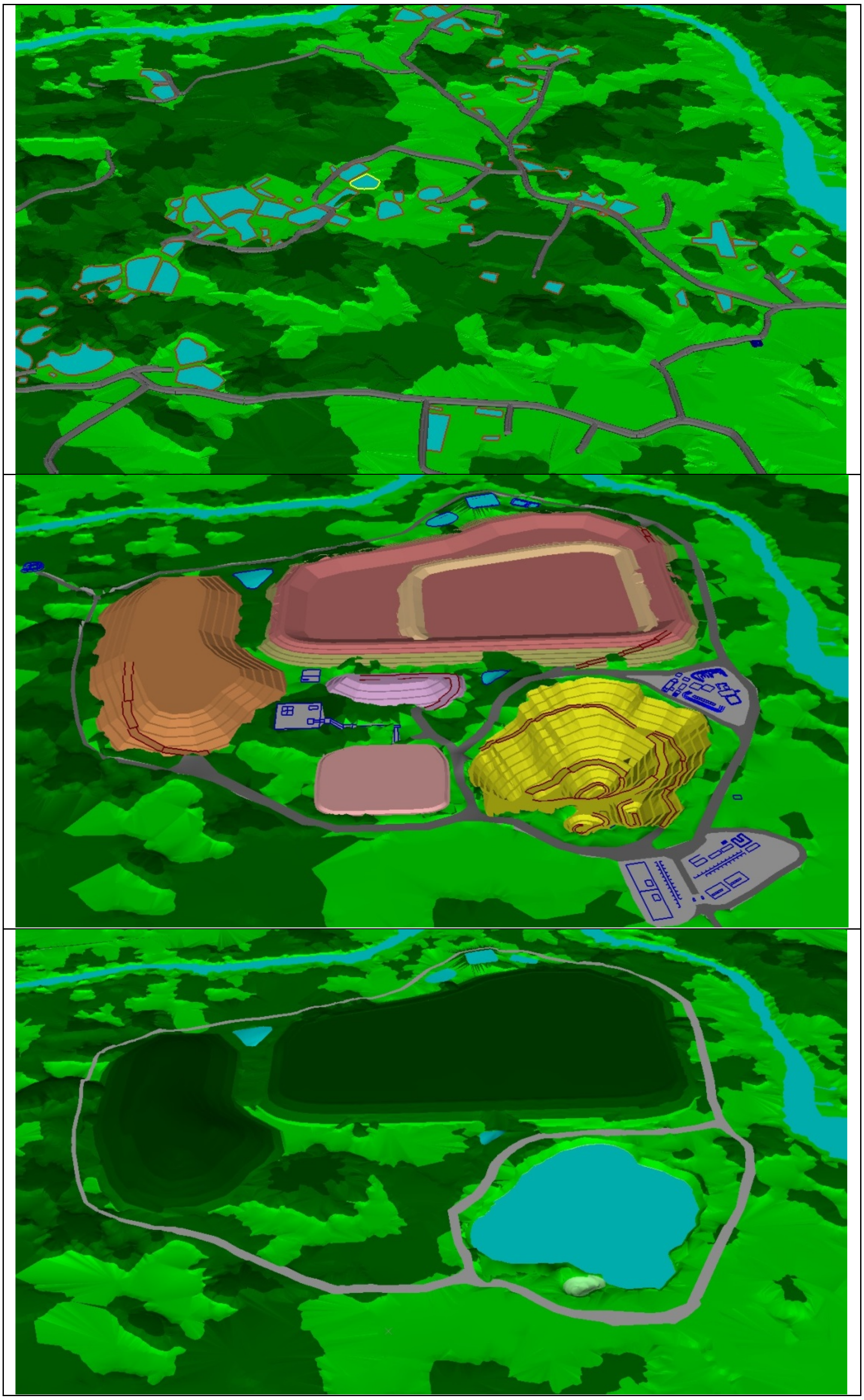


Figure 20-1 - 3D View: Indicative Sequence Showing Current Topography, Mining Infrastructure & Rehabilitated Site

The monitoring program involves data recording of variables associated with the impacts. These includes, air, water, soil, rehabilitation success and public safety.

Surface water monitoring will be a crucial component to ensure the health of the rehabilitated land and detection of any potentially harmful and hazardous substances is not being released to existing water supply (i.e. rivers, streams and lakes). Water sampling will be further supplemented by soil sampling at selected areas which has shown historical impact or potentially new areas which need constant monitoring.

General health and safety issues need to be address post mining as certain areas may be hazardous to the public, namely the open pit, TSF and waste disposal. These areas will require proper signage and fencing where applicable or other measures to properly mitigate any hazards.

20.6.1. Closure Timeframes

Typically post closure activities range from six (6) months to two (2) years in duration depending upon the size and scale of mining, and the amount of on-going rehabilitation work completed to date. A typical conceptualized closure timeframe, detailing work activities and the related timings are detailed in the following table. Completion of the Feasibility Study and EIA will provide information to further undertake detailed scheduling of mine closure activities and help develop the associated costs of rehabilitation.

20.7. Identification & Management of Closure Issues

Closure frameworks, associated with various mining activities, will be identified at the onset of mine planning and the operations stage. This framework will be constantly under review and assessment throughout until mine closure. By identifying important aspects relating to environmental impacts, control measures can be implemented and integrated into mine planning and operations, to prevent long lasting reputational and financial risk. As such, prior to the actual closure, issues such as AMD, erosion, drainage and slope stability control including re-vegetation will be given precedence.

20.7.1. Measures on Public Safety and Health

Mining and exploration sites must be left in a condition that ensures the safety of the Public are well looked after. All appropriate and recognized Health and Safety practices will be followed. The Rehabilitation Plan should cover the security of the site and public safety, following cessation of operations. This may require limiting public access by fencing and barring of vehicular access tracks.

Safety and security objectives include the following:

- Safe and secure environment for humans and wildlife for the long term;
- Stability assessment of remaining mining voids where relevant to the potential for further ground movement and the need for controls such as barriers or fencing;
- Stabilization of slopes (e.g. of pit walls, waste dumps, dams) so that no hazard to the public remains after final closure. Temporary restriction of access to specific areas to

protect remaining equipment or facilities or to ensure undisturbed development of vegetation which needs care and maintenance until the area is deemed safe. Over the long-term, there will be no restrictions of access by the public, as all hazards to safety, property and health will have been removed;
- Site maintenance and security will be on going activities during closure, and will be subject to periodic inspection.

Other Public Health and Safety measures include, but are not limited to:

- Elimination of areas susceptible to water logging and free of stagnant and standing water to reduce and / or eliminate breeding ground of disease borne insects, especially mosquito;
- Proper disposal of waste and erection of Health and Safety signboards with proper barricades at specifically designated waste disposal areas to reduce public contact with waste and prevent public contamination;
- Occupational Health and Safety training for employees to create awareness on methods of prevention of accidents and disease whereby, trained employees will be able to relate and share these knowledge to their respective family member;
- Community engagement and liaison including education for the public to foster cooperation and awareness in creating a clean, sustainable and healthy living environment;
- Environmental monitoring of air, noise and water to ensure public are not exposed to harmful effects and contamination especially water reservoir.

20.8. Summary & Conclusions

North Borneo Gold Sdn Bhd is actively pursuing the development of economical mineral resources in the Bau goldfield and at the same time, the company's is committed to undertake an environmental impact assessment to identify issues and data gaps to develop an environmental management and rehabilitation plan in compliance to international standard and local regulatory requirement, in order to create a sustainable environment during post-mining operation and closure. The aim is to create a post-mining environment that is at least equal in environmental value or better.

Regulatory framework and compliance plays an important role to guarantee the basis for environmental impact assessment is being addressed appropriately to ensure sustainable development of mineral resource and at the same time, promote environmental stewardship to ensure the mining project is being developed in an environmentally sound, responsible and sustainable manner.

Socio-economic undertaking such as stakeholder engagement, local community development, public relation and liaison present a unique opportunity for the company to encourage local community participation and interest for dialog to identify possible concerns and expectation relating to socio-economic development and environmental awareness. This will foster relations and create credibility which will further elevate the company's public image and reputation. As such, some of the key environmental aspects, which required attention as the project progresses into a viable mining operation, are:

- Identification of environmental impacts and constraints associated with exploration and mining activities to ecology, socio-economic including historical / cultural sites in the area;
- The ecological impacts due to alteration of pre-existing environment such as land clearing and landscape modification effecting indigenous wild life species in the area
- The concern of Acid Mine Drainage (AMD) generation due to the oxidation of sulphide minerals from ore and waste and mitigation measure associated with the rehabilitation and regulatory compliance;
- Socio-economic effects (both positive and negative) on local communities associated with or affected by the mining development, and the costs involved in maximising positive and minimising negative socio-economic effects; and
- The scale and nature of rehabilitation scope for the eventual mine closure and post closure monitoring required for all deactivated mined-out areas and associated auxiliary structures and facilities.
- Post mining environmental monitoring to ensure the success of rehabilitation and the preservation of a sustainable ecosystem.

In order to achieve the task of creating a sustainable environment post mining, control measures and mitigation methods need to be in place to counter potential environmental impact. Hence, mine planning isincorporating the following management framework for the integration of site specific mitigation design. These are:

- Environmental Impact Assessment
- Environmental Management Plan
- Mine Rehabilitation Plan
- Monitoring Program
- Alternatives Land Use Planning

Through proper incorporation of the above mentioned management framework, a progressive rehabilitation process can commence to deal with potential long-term environmental impacts due to mining. The objectives of rehabilitation schemes are to develop achievable goals at various stages as mine planning evolves by converting an area of concern to a safe and stable condition, restoring the site to a pre-mining condition as closely as possible in order to ensure sustainability development. Mine rehabilitation is essentially a process whereby the development of mineral resources is being conducted in accordance with the principles of leading sustainable practice. Rehabilitation should be part of an effective integrated program coexisting with mine operation and mine development in all phases.

In summary, mining is a temporary use of the land, the successes of a mining venture lies with the notion that the mining operator has successfully integrated mining best practice with the development of sustainable mining operation and integrated the best mine closure standards by ensuring the future of the land is not compromised but rather in a sustainable manner.

The progression from current land status through mining to a possible rehabilitated state is shown in the modelled 3D views to show the concept, and these are shown in *Figure 20-2 - Jugan - Current Land Situation and Status* to *Figure 20-4 - Indicative Rehabilitated Scenario Option for Jugan* below.

Figure 20-2 - Jugan - Current Land Situation and Status

Figure 20-3 - Jugan - Land Situation During Short Mining Phase

Figure 20-4 - Indicative Rehabilitated Scenario Option for Jugan

21. Capital & Operating Costs

21.1. Introduction & Basis of Estimate

Capital and operating cost estimates are based on a number of processes and techniques, and these are listed below in no particular order:

- Quotes were obtained for major process and mining capital items from Metso, CAT Tractors Malaysia, Sandvik Malaysia Sdn Bhd, Orica, etc;
- Costs where applicable were benchmarked against in-house costs at Beras's other operations, from costing database and from information from similar operations worldwide;
- Where applicable costs have been derived from first principles based on unit costs and derived calculations;
- Costs for locally sourced items obtained from local suppliers;
- Standard engineering rates and principles were applied to anciliary items and associated elements in line with normal engineering costing practice;
- Standard rates and values were applied based on published information, e.g. import tariffs, etc.

21.2. Capital Costs

21.2.1. Mining Capital

Mobile mining equipment capital costs are listed in *Table 21-1 - Mining Capital Costs - Mobile Equipment (Owner-Operator)* below and they are for the owner-operator mining type option at the 8,000tpd base case option. For the contract-mining option it is assumed that similar equipment is used but is supplied by a contractor and not be included in the capital estimate.

The equipment includes an estimate for critical or major spares and ranges fro 10-30 % of the total capital cost. The equipment costs do not include import tariffs or freight costs as they are quoted from within Malaysia with these already included. Mining capital costs for the different options are detailed in the project cost model and summarised in *Section 22* or in the *Appendices*.

No.	***Mining Equipment for Jugan Open Pit***	***Unit Cost (US$)***	***Total Cost (US$)***	***Spares (US$)***	***Total (US$)***
2	Production Drill, Sandvik DX800 or equivalent, 76mm to 127mm hole, crawler	$ 565,920	$ 1,131,840	$ 339,552	$ 1,471,392
2	Hydraulic Shovel, 7m3, CAT6015/FS	$1,476,765	$ 2,953,530	$ 590,706	$ 3,544,236
1	Wheel Loader CAT 988H, 6.4 m3 for pit operation	$ 820,425	$ 820,425	$ 164,085	$ 984,510
1	Wheel Loader or FEL, 6.4 m4 for stockpile operation	$ 820,425	$ 820,425	$ 164,085	$ 984,510
1	CAT_D10T Dozer with ripper	$1,670,385	$ 1,670,385	$ 334,077	$ 2,004,462

No.	*Mining Equipment for Jugan Open Pit*	*Unit Cost (US$)*	*Total Cost (US$)*	*Spares (US$)*	*Total (US$)*
1	D6W Tractor (CAT_D6R XL)	$ 274,022	$ 274,022	$ 54,804	$ 328,826
9	Hauling Truck, Rigid Rear Dump CAT_772G	$ 662,903	$ 5,966,131	$ 1,193,226	$ 7,159,357
2	Road Grader, CAT_12K	$ 308,480	$ 616,960	$ 61,696	$ 678,656
2	Water Truck (10,000 liters)	$ 88,606	$ 177,212	$ 17,721	$ 194,933
2	Compactor, CAT CS533E	$ 95,169	$ 190,339	$ 19,034	$ 209,372
1	Surface Drill Rig for Cable Bolting	$ 565,920	$ 565,920	$ 56,592	$ 622,512
2	Explosive Truck (1000 kg cap) or Mobile Mixing Unit	$ 50,000	$ 100,000	$ 10,000	$ 110,000
2	Service/Tire Truck (off highway road)	$ 90,000	$ 180,000	$ 18,000	$ 198,000
5	4WD LV Toyota Hi-lux	$ 26,254	$ 131,268	$ 13,127	$ 144,395
33	**Totals**		**$15,598,456**	**$3,036,705**	**$18,635,161**

Table 21-1 - Mining Capital Costs - Mobile Equipment (Owner-Operator)

For both the owner-operator and the contract-mining option there is fixed plant capital. This cost is applicable to both options as this would not be within a mobile equipment contract. The fixed plant capital costs are listed in *Table 21-2 - Mining Capital Costs - Fixed Equipment (All Mining Type)* below.

No.	*Fixed Plant & Capital Services*	*Total (US$)*
2	Surface Blaster (i-kon)/Exploder	$ 20,700
4	Mobile Light Plant (13kW)	$ 40,000
1	Pit Dewatering Pump, centrifugal 75 li/sec	$ 20,550
1	Pit Dewatering Pump, centrifugal 40 li/sec	$ 13,700
1	Dewatering Pump, diaphragm type 20 li/sec	$ 34,250
2	Vacuum Pump	$ 6,250
1	Butt Welder for HDPE pipes	$ 22,500
1	Workshop Tools & Equipment	$ 15,000
500	HDPE Pipes (for air & water), 6m length	$ 21,000
22	Fire Fighting Equipment for each machine	$ 2,200
535	**Totals**	**$ 196,150**

Table 21-2 - Mining Capital Costs - Fixed Equipment (All Mining Type)

Capital costs for mining construction work, and in particular the initial construction work, is listed in *Table 21-3 - Mining Capital Costs - Mining Construction (All Mining Types)* below.

No.	*Mining Construction & Contents*	*Total (US$)*
	Building & Infrastructure	
1440 m^2	Heavy Machine & Truck Maintenance Workshop	$ 793,981
180 m^2	Electrical Workshop	$ 82,706
250 m^2	Tyre (repair & maintenance) Shop	$ 114,870
312 m^2	Drilling / Drill Equipment Shop	$ 143,358

No.	Mining Construction & Contents	Total (US$)
180 m^2	Light Vehicle Shop	$ 82,706
600 m^2	Mine Operation Dispatch Cabin	$ 101,138
600 m^2	Equipment Operators & Drivers Cabin	$ 101,138
140 m^2	Coreyard & Core House	$ 23,599
42 m^2	Exploration & Resource Drillers Cabin	$ 5,095
42 m^2	Pit (production) Drilling Crew Cabin	$ 5,095
54 m^2	Survey Crew Cabin	$ 6,550
42 m^2	Explosive Mixers & Blasting Crew Cabin	$ 5,095
45 m^2	Dewatering & Diesel/Oil Crew Cabin	$ 5,459
672 m^2	Operation Washbay	$ 308,771
1,000 m^2	Diesel Tank Area/Depot	$ 82,050
500 m^2	Explosive Magazine	$ 303,585
10,800 m^3	Explosive Magazine Perimeter Bund	$ 42,535
1,188 m^2	Mine & Mill Operation Warehouse	$ 545,862
500 m^2	Reagents & Chemicals Store/Warehouse	$ 229,740
682 m^2	Mining Admin & Management Offices	$ 514,815
650 m^2	Mining Engineers & Geologists Offices	$ 490,659
720 m^2	Training & Conference Room	$ 543,499
363 m^2	Canteen & Food Store	$ 36,694
1,188 m^2	Nursery for Agro-Forest & Rehabilitation	$ 116,970
750 m^2	Mining Contractors Area	$ 73,845
5 m^2	Guard House	$ 1,477
	Total Building & Infrastructure	**$ 4,761,292**
	Workshop Equipment & Tools:	
1	Standby power generator 60 to 70 kW_workshop	$ 21,700
1	Standby power generator 20kW_Office	$ 4,595
1	Medium duty overhead crane - Demag 20t	$ 70,000
2	Mono-rail hoist_5t for workshop & warehouse	$ 17,001
2	Chain Hoist (comealong) - 1ton	$ 295
2	Chain Hoist (comealong) - 2ton	$ 446
1	Lever/Chain Hoist - Ratchet type 3t	$ 246
1	Lever/Chain Hoist - Ratchet type 5t	$ 361
1	Forklift, 10tonne cap	$ 63,000
1	Hydraulic Jack - 50 ton	$ 1,641
2	Battery Charger - 12V	$ 118
2	Welding Machine - professional welder	$ 1,615
1	Portable (inverter) welder	$ 527
1	Inverter Plasma	$ 1,077
2	Oxy Acetylene welding/cutting set	$ 255
1	Bandsaw variable speed	$ 223
1	Air Compressor	$ 256
2	Bench Grinder	$ 257

No.	*Mining Construction & Contents*	*Total (US$)*
3	Disk Grinders	$ 180
1	Lathe Machine complete with cutting tools	$ 2,745
1	Boring Machine	$ 1,053
1	Bench Drill	$ 383
1	Hydraulic Press - floor type 20-tonne	$ 1,148
1	Arbor press - 3 tonne	$ 560
1	Bench Press - 10 tonne	$ 599
1	Combination Spanner - 25pcs per set	$ 71
1	16 to 36mm Spanner set - 12pcs per set	$ 59
1	Socket Set - 40 pcs per set	$ 30
1	Hex Key Set - 30 pcs per set	$ 18
1	Box Wrench - 6 pcs per set	$ 15
2	Adjustable Wrench - heavy duty	$ 16
1	Extension (socket) bars - 3pcs per set	$ 12
1	Metric Socket set - 7 pcs per set	$ 17
4	Pipe Wrench - 4 sizes	$ 39
1	Bunded Diesel Fuel Tank, 12000 liter capacity with metering dispenser (for pit operation)	$ 16,800
1	Bunded Steel Fuel Tank, 5000 liter capacity	$ 5,600
3	Pressure Washers	$ 591
	Total Workshop Equipment & Tools:	**$213,549**
	Office Equipment & Furnitures:	
4	Split type air conditioning unit, 4hP	$ 1,444
10	Window type office aircon unit	$ 1,050
10	Window type aircon unit for cabin	$ 1,050
2	Q-series Office System_Director's Office	$ 1,168
4	V-series Office System_Management Office	$ 1,838
14	Office Tables for sr managers	$ 2,619
12	Office tables for Mine Office staff	$ 1,674
4	Office table for Mechanical/Electrical Office	$ 558
6	Office tables for Mill Office staff	$ 837
6	Office tables for Metallurgical Office	$ 837
10	Office tables for Admin Office staff	$ 1,395
4	Office table for Safety & Environment Office	$ 558
4	Office tables for Engineering Office staff	$ 558
12	Tables for Procurement, Accounting, HR & IT staff	$ 1,674
8	Office tables for Mining Engineers & Geologists	$ 1,116
14	Executive Chairs	$ 1,378
64	Office Chairs	$ 4,621
24	Steel File Cabinets - 4 drawer	$ 4,490
8	HR & Accounting cabinets	$ 1,838
10	Book shelves (2m x 2m)	$ 1,313

No.	Mining Construction & Contents	Total (US$)
15	Full Height Cupboard	$ 2,215
36	Desk Top Computers with Windows Pro	$ 70,891
18	Laptop for Sr Managers with Windows Pro	$ 38,399
14	Laptop for Engineers & Geos with Windows Pro	$ 36,758
16	Laptop for Support Staff with Windows Pro	$ 39,384
1	A0 Roll Printer	$ 2,626
1	Network Printer/Scanner/FAX (FUJI)	$ 7,220
2	A4 Printer for Management Office	$ 2,626
1	High resolution, Lumens=4000+, Projector	$ 2,462
2	Low resolution, Lumens=2000-3000, Projector	$ 1,772
2	Coffee Maker Dispenser	$ 1,641
	Total Office Equipment & Furnitures:	**$238,010**
	Office/Business System & ICT:	
1	Microsoft Office for 84 computer units	$ 100,800
1	CAE Mining System	$ 150,000
1	SCALA Office system (timekeeping & payroll)	$ 150,000
1	Telephone/Communication System	$ 75,000
1	Integrated Plant & Office Security Systems	$ 50,000
	Total Office/Business System & ICT:	**$525,800**
	Mine Roads & Clearing	
29,008 m^3	Sub_grade Fill (0.5m) for 25m Haul Road (4.392 km)	$ 116,032
82,880 m^3	Base Course (1.0m) for 25m Haul Road (4.392 km)	$ 828,800
41,440 m^3	Wearing Course (0.5m) for 25m_wide Haul Road	$ 414,400
18,053 m^3	Sub_grade Fill (0.5m) for 15m service road (4.350 km)	$ 72,212
51,580 m^3	Base Course (0.5m) for 15m service road (4.350 km)	$ 515,800
25,790 m^3	Wearing Layer (0.5m) for 15m_wide service road	$ 257,900
	Total Mine Roads & Clearing	**$2,205,144**
	ROM Pad	
33,641 m^3	Base Course (0.5m)	$ 336,410
	Total ROM Pad	**$336,410**
	Waste Dump	
27 Ha	Clearing/grubbing and drainage	$ 218,700
10 Ha	Downstream Wetland (drain location)	$ 100,000
91,142 m^3	Base Course (0.3m) - Drainage	$ 911,424
	Total Waste Dump	**$1,230,124**
	Mining Construction (incl road & waste dump):	**$9,510,329**

Table 21-3 - Mining Capital Costs - Mining Construction (All Mining Types)

All other or sundry mining capital costs are listed in *Table 21-4 - Mining Capital Costs - Mining Other (All Mining Types)* below.

No.	*Other Mining Capital*	*Total (US$)*
	Health & Safety and Environment	
1	First Aid Equipment & paraphernalias	$ 21,000
10	Fire Fighting Equipment - Fixed	$ 1,050
1	Fire Hydrant System	$ 15,000
1	Ambulance	$ 31,500
	Total Health & Safety:	***$ 68,550***
	Mine Services: (mine planning, survey & geology)	
1	Survey Equipment	$ 35,200
1	GeoMIMS System or GEMS Additional Licences	$ 80,000
5	Computer / Laptops	$ 12,500
	Total Mine Services:	***$ 127,700***
	Communication & Security:	
1	Telephone System	$ 20,000
1	Base Radio for pit operation	$ 10,000
5	Wireless Camera System	$ 25,000
5	Motorbikes for Security personnel	$ 7,500
	Total Communication & Security:	***$ 62,500***
	Sundries:	-
1	Office Furniture (one lot)	$ 1,641
4	Workshop Racks & Storage	$ 3,480
5	Oxy-acetylene Equipment	$ 1,969
100	Caplamps with charger	$ 17,500
65	Handheld Radios	$ 53,328
	Total Sundries:	***$ 77,917***
	Total Mining Other	**$ 336,667**

Table 21-4 - Mining Capital Costs - Mining Other (All Mining Types)

The total mining capital cost for the base case 8,000 tpd, for owner-operator is summarised in *Table 21-5 - Summary of Mining Capital Costs (Owner-Operator)* with the contract-mining costs summarised in *Table 21-6 - Summary of Mining Capital Costs (Contract-Mining)* below.

Mining Capital Group	*Total Cost (US$)*
Mobile Mining Equipment	18,635,200
Fixed Mining Equipment	196,150
Mining Construction	9,510,300
Mining - Other	336,700
Total – Owner-Operator	**28,678,350**

Table 21-5 - Summary of Mining Capital Costs (Owner-Operator)

Mining Capital Group	*Total Cost (US$)*
Mobile Mining Equipment	0
Fixed Mining Equipment	196,150
Mining Construction	9,510,300
Mining - Other	336,700
Total – Contract-Mining	**10,043,150**

Table 21-6 - Summary of Mining Capital Costs (Contract-Mining)

It should be noted that due the the outcropping nature of the orebody and the direct access to ore from day one (1), no pre-stripping capital is applicable.

21.2.2. Process Plant Capital

This section provides the comparative capital costs for the treatment of Jugan ore at 4,000 tpd, 8,000 tpd and 12,000 tpd milling rates. These tonnages are the middle and both ends of the 2,000tpd increment options being considered.

Four process options have been considered, namely production of a flotation concentrate to be sold to a third party in the first option and production of gold dore by further processing of the flotation concentrate on location by either the BIOX (biological oxidation route), the POX (pressure oxidation route) or Albion (ultrafine grinding and atmospheric oxidation route).

Details of these processes have been provided in *Section 17* above. The flotation concentrate is the base case option but the oxidation options and the plant tail end are also calculated should one of these options be selected now or in the future.

The capital costs below have been used in conjunction with operating costs for input in the Feasibility Model discussed in *Section 22* of this report. The costs for the alternate process options are included in the project cost model and included in *Section 22* and in the *Appendices*.

21.2.2.1. Capital – Flotation Concentrate

Table 21-7: Capital Cost for Flotation Concentrate Production from Jugan Ore below summarizes the capital costs for the production of a flotation concentrate from Jugan ore at three daily milling rates. The mass pull in flotation has been assumed to be 10% based on testwork to-date. These are for the base case and both ends of the concentrate production spectrum. Further below is the costing for the possible oxidation options and the tail end of a plant (CIL, electrowinning, etc.) should those options be followed now or at a later stage.

Tonnage Rate	12,000 TPD	8,000 TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
A - Packaged Plant			
Crushing Plant	1,885,000	1,420,000	980,000
Primary SAG Mill	11,279,000	7,779,000	5,585,000
Knelson CVD	1,350,000	950,000	650,000

Tonnage Rate	12,000 TPD	8,000 TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
Primary Cyclone Cluster	375,000	310,000	275,000
Flotation Conditioner Tank	190,000	145,000	105,000
Flotation Cell Unit	10,810,000	8,110,000	4,950,000
Regrind Ball Mill	3,267,000	2,252,000	1,332,000
Regrind Mill Cyclone	402,000	346,000	301,000
Concentrate Filter Feed Thickener	455,000	355,000	295,000
Filter Press Unit	3,507,000	2,705,000	1,965,000
TOTAL - A	**33,520,000**	**24,372,000**	**16,438,000**
B - Out of Packaged Plant			
Reagents	750,000	650,000	500,000
Water Supply System	550,000	450,000	350,000
Low/High Pressure Air System	650,000	450,000	350,000
Buildings/Cranes	1,550,000	1,550,000	1,250,000
Electrical Power System, Generator/Grid	4,800,000	4,300,000	3,800,000
TOTAL - B	**8,300,000**	**7,400,000**	**6,250,000**
TOTAL (A+B)	**41,820,000**	**31,772,000**	**22,688,000**
C - Other Vendor Items			
Structural Steel/Platform etc,8%	3,698,190	2,859,480	2,041,920
Pumps, Piping, Valves, Launders, Chutes, 7%	3,345,600	2,541,760	1,815,040
Civil works, 5%	2,091,000	1,588,600	1,134,400
Concrete works, 10%	4,182,000	3,177,200	2,268,800
Electrical Distribution, 12%	5,018,400	3,812,640	2,722,560
Instrumentation & Control, 4%	1,045,500	1,270,880	907,520
Customs/Taxes & Shipping/Transport, 10%	4,182,000	3,177,200	2,268,800
Engineering Cost, 2.5%	1,045,500	794,300	567,200
Contingency, 10%	4,182,000	3,177,200	2,268,800
First fill cost/Spare, 5 %	2,091,000	1,588,600	1,134,400
TOTAL - C	**30,881,190**	**23,987,860**	**17,129,440**
TOTAL (A+B+C)	**72,701,190**	**55,759,860**	**39,817,440**

Table 21-7: Capital Cost for Flotation Concentrate Production from Jugan Ore

21.2.2.2. Capital – BIOX

Table 21-8: Capital Costs for a Gold Plant with BIOX Treatment of the Jugan Ore below summarises the capital cost for three milling rates applying the same up front flotation concentrate flowsheet as above and further processing of the flotation concentrate by bacterial oxidation in suitably aerated tanks, folowed by counter current washing in three thickeners, carbon-in-leach (CIL) gold extraction, carbon stripping and gold electro-winning, carbon regeneration and gold doré melting. The CIL tailings are detoxified with SO_2 and air in the presence of a copper catalyst.

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
A - Packaged Plant			
Water Cooling Unit	6,500,000	4,000,000	2,160,000
Isa Mill	0	0	0
Air Compressors	5,800,000	3,900,000	2,000,000
Nutrient Module	4,500,000	3,000,000	1,500,000
Oxidation Plant	28,000,000	17,000,000	9,000,000
Oxygen Plant-Installed	0	0	0
Limestone Plant-Installed	18,000,000	12,000,000	6,000,000
Installation, 12%	5,376,000	3,348,000	1,759,200
Freight, 5%	2,240,000	1,395,000	733,000
TOTAL - A	**70,416,000**	**44,643,000**	**23,152,200**
B - Out of Packaged Plant			
Crushing Plant	1,885,000	1,420,000	980,000
Primary SAG Mill	11,279,000	7,779,000	5,585,000
Knelson CVD	1,350,000	950,000	650,000
Primary Cyclone cluster	375,000	310,000	275,000
Flotation Conditioner Tank	190,000	145,000	105,000
Flotation Cell Unit	10,810,000	8,110,000	4,950,000
Regrind Ball Mill	3,250,000	2,240,000	1,325,000
Regrind mill cyclone	365,000	315,000	275,000
Thickeners	4,500,000	3,500,000	2,750,000
CIL	4,500,000	3,100,000	1,400,000
Elution & Gold Room	4,850,000	4,000,000	2,950,000
Detox	1,500,000	700,000	400,000
Nutralization Tanks	1,750,000	1,350,000	1,050,000
Reagents	5,200,000	4,000,000	2,200,000
Water Supply System	650,000	500,000	350,000
Low/High Pressure Air System	650,000	550,000	450,000
Buildings	2,200,000	1,800,000	1,200,000
TOTAL - B	**55,304,000**	**40,769,000**	**26,895,000**
TOTAL (A+B)	**125,720,000**	**85,412,000**	**50,047,200**
C - Other Vendor Items			
Electrical Power Systems, Generator/Grid	5,955,000	5,155,000	4,355,000
Structural Steel/Platform, etc., 8%	10,057,600	6,832,960	4,003,776
Piping, Valves, Launders, 7%	8,800,400	5,978,840	3,503,304
Civil Works, 5%	6,286,000	4,270,600	2,502,360
Concrete Works, 10%	12,572,000	8,541,200	5,004,720
Electrical Distribution, 12%	15,086,400	10,249,440	6,005,664
Instrumentation & Control, 4%	5,028,800	3,416,480	2,001,888
TOTAL - C	**63,786,200**	**44,444,520**	**27,376,712**
TOTAL (A+B+C)	**189,506,200**	**129,856,520**	**77,423,912**

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
D - Other Items			
First Fill & Spares - 5%	9,475,310	6,492,826	3,871,196
Transportation Cost - 2%	3,790,124	2,597,130	1,548,478
Customs & Duties - 8%	15,160,496	10,388,522	6,193,913
Engineering Charges - 2.5%	4,737,655	3,246,413	1,935,598
Contingency, 10%	12,572,000	8,541,200	5,004,720
TOTAL - D	**45,735,585**	**31,266,091**	**18,553,905**
TOTAL (A+B+C+D)	**235,241,785**	**161,122,611**	**95,977,817**

Table 21-8: Capital Costs for a Gold Plant with BIOX Treatment of the Jugan Ore

21.2.2.3. Capital – POX

Table 21-9: Capital Cost for a Gold Plant with POX Treatment of the Jugan Ore summarizes the capital cost for the processing of Jugan ore at three milling rates with treatment of the flotation concentrate by autoclave pressure oxidation followed by counter-current washing of the oxidized concentrate, CIL and gold recovery in the same way as for the BIOX concentrate processing above.

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
A - Packaged Plant			
Water Cooling Unit	0	0	0
Isa Mill	0	0	0
Air Compressors	0	0	0
Nutrient Module	0	0	0
Oxidation Plant	40,000,000	28,000,000	15,000,000
Oxygen Plant-Installed	28,000,000	21,000,000	10,000,000
Limestone Plant-Installed	18,000,000	12,000,000	6,000,000
Installation, 12%	4,800,000	3,360,000	1,800,000
Freight, 5%	2,000,000	1,400,000	750,000
TOTAL - A	**92,800,000**	**65,760,000**	**33,550,000**
B - Out of Packaged Plant			
Crushing Plant	1,885,000	1,420,000	980,000
Primary SAG Mill	11,279,000	7,779,000	5,585,000
Knelson CVD	1,350,000	950,000	650,000
Primary Cyclone Cluster	375,000	310,000	275,000
Flotation Conditioner Tank	190,000	145,000	105,000
Flotation Cell Unit	10,810,000	8,110,000	4,950,000
Regrind Ball Mill	3,250,000	2,240,000	1,325,000
Regrind Mill Cyclone	365,000	315,000	275,000
Thickeners	3,500,000	2,750,000	1,950,000
CIL	3,200,000	2,400,000	1,400,000

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
Elution & Gold Room	4,850,000	4,000,000	2,950,000
Detox	860,000	700,000	400,000
Nutralization Tanks	1,750,000	1,350,000	1,050,000
Reagents	5,200,000	4,000,000	2,200,000
Water Supply System	650,000	500,000	350,000
Low/High Pressure Air System	650,000	550,000	450,000
Buildings	2,200,000	1,800,000	1,200,000
TOTAL - B	**52,364,000**	**39,319,000**	**26,095,000**
TOTAL (A+B)	**145,164,000**	**105,079,000**	**59,645,000**
C - Other Vendor Items			
Electrical Power Systems, Generator/Grid	5,955,000	5,155,000	4,355,000
Structural Steel/Platform, etc., 8%	11,613,120	8,406,320	4,771,600
Piping, Valves, Launders, 7%	10,161,480	7,355,530	4,175,150
Civil Works, 5%	7,258,200	5,253,950	2,982,250
Concrete Works, 10%	14,516,400	10,507,900	5,964,500
Electrical Distribution, 12%	17,419,680	12,609,480	7,157,400
Instrumentation & Control, 4%	5,806,560	4,203,160	2,385,800
TOTAL - C	**72,730,440**	**53,491,340**	**31,791,700**
TOTAL (A+B+C)	**217,894,440**	**158,570,340**	**91,436,700**
D - Other Items			
First Fill & Spares - 5%	10,894,722	7,928,517	4,571,835
Transportation Cost - 2%	4,357,889	3,171,407	1,828,734
Customs & Duties - 8%	17,431,555	12,685,627	7,314,936
Engineering Charges - 2.5%	5,447,361	3,964,259	2,285,918
Contingency, 10%	14,516,400	10,507,900	5,964,500
TOTAL - D	**52,647,927**	**38,257,710**	**21,965,923**
TOTAL (A+B+C+D)	**270,542,367**	**196,828,050**	**113,402,623**

Table 21-9: Capital Cost for a Gold Plant with POX Treatment of the Jugan Ore

21.2.2.4. Capital – Albion

Table 21-10: Capital Cost for a Gold Plant with Albion Treatment of the Jugan Ore below summarizes the capital cost for a gold plant using the Albion process for the treatment of the flotation concentrate with CIL, carbon stripping and gold electro-winning, carbon regeneration, gold doré melting and CIL tails detoxification with SO_2 and air in the presence of a copper catalyst.

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
A - Packaged Plant			
Water Cooling Unit	0	0	0

Tonnage Rate Costing Elements	12,000 TPD (US$)	8,000TPD (US$)	4,000 TPD (US$)
Isa Mill	14,650,000	10,000,000	6,000,000
Air Compressors	0	0	0
Nutrient Module	0	0	0
Oxidation Plant	18,600,000	13,000,000	7,000,000
Oxygen Plant-Installed	23,000,000	15,000,000	8,000,000
Limestone Plant-Installed	18,000,000	12,000,000	6,000,000
Installation, 12%	4,000,000	2,640,000	1,560,000
Freight, 5%	1,700,000	1,150,000	650,000
TOTAL - A	**79,950,000**	**53,790,000**	**29,210,000**
B - Out of Packaged Plant			
Crushing Plant	1,885,000	1,420,000	980,000
Primary SAG Mill	11,279,000	7,779,000	5,585,000
Knelson CVD	1,350,000	950,000	650,000
Primary Cyclone Cluster	375,000	310,000	275,000
Flotation Conditioner Tank	190,000	145,000	105,000
Flotation Cell Unit	10,810,000	8,110,000	4,950,000
Regrind Ball Mill	3,250,000	2,240,000	1,325,000
Regrind Mill Cyclone	365,000	315,000	275,000
Thickeners	3,500,000	2,750,000	1,950,000
CIL	4,500,000	3,100,000	1,400,000
Elution & Gold Room	4,850,000	4,000,000	2,950,000
Detox	1,500,000	1,150,000	700,000
Nutralization Tanks	0	0	0
Reagents	5,200,000	4,000,000	2,200,000
Water Supply System	650,000	500,000	350,000
Low/High Pressure Air System	350,000	280,000	150,000
Buildings	2,200,000	1,800,000	1,200,000
TOTAL - B	**52,254,000**	**38,849,000**	**25,045,000**
TOTAL (A+B)	**132,204,000**	**92,639,000**	**54,255,000**
C - Other Vendor Items			
Electrical Power Systems, Generator/Grid	6,415,000	5,615,000	4,815,000
Structural Steel/Platform, etc., 8%	10,576,320	7,411,120	4,340,400
Piping, Valves, Launders, 7%	9,254,280	6,484,730	3,797,850
Civil Works, 5%	6,610,200	4,631,950	2,712,750
Concrete Works, 10%	13,220,400	9,263,900	5,425,500
Electrical Distribution, 11%	15,864,480	11,116,680	6,510,600
Instrumentation & Control, 4%	5,288,160	3,705,560	2,170,200
TOTAL - C	**67,228,840**	**48,228,940**	**29,772,300**
TOTAL (A+B+C)	**199,432,840**	**140,867,940**	**84,027,300**
D - Other Items			
First Fill & Spares - 5%	9,971,642	7,043,397	4,201,365

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Costing Elements	**(US$)**	**(US$)**	**(US$)**
Transportation Cost - 2%	3,988,657	2,817,359	1,680,546
Customs & Duties - 8%	15,954,627	11,269,435	6,722,184
Engineering Charges - 2.5%	4,985,821	3,521,699	2,100,683
Contingency, 10%	13,220,400	9,263,900	5,425,500
TOTAL – D	**48,121,147**	**33,915,790**	**20,130,278**
TOTAL (A+B+C+D)	**247,553,987**	**174,783,730**	**104,157,578**

Table 21-10: Capital Cost for a Gold Plant with Albion Treatment of the Jugan Ore

In summary the capital costs for the four process options for the upper, lower and middle tonnage rates are summarised and rounded up (nearest 100) in *Table 21-11 - Summary of Total Capital Costs by Tonnage Option & Process Method* below.

Tonnage Rate	12,000 TPD	8,000TPD	4,000 TPD
Process Methods	**(US$)**	**(US$)**	**(US$)**
Flotation Concentrate	**72,701,190**	**55,759,860**	**39,817,440**
Biological Oxidation	**235,241,800**	**161,122,700**	**95,977,900**
Pressure Oxidation	**270,542,400**	**196,828,100**	**113,402,700**
Albion Process	**247,554,000**	**174,783,800**	**104,157,600**

Table 21-11 - Summary of Total Capital Costs by Tonnage Option & Process Method

The detailed equipment list and costing for the base case process plant (8,000 tpd flotation concentrate option) can be found in *Appendix A21-6*, with the plant buildings list included in *Appendix A21-7*.

21.2.3. Infrastructure & Facilities

Mining infrastructure, including roading, has been included with the mining capital costs above.

One major infrastructure capital item is the TSF and this cost is outlined in *Table 21-12 - Tailings Storage Facility - Capital Cost Breakdown* below for the designed TSF as specified in Section 16 for the 8,000tpd base case scenario.

Amount	TSF Construction Item	Unit Cost	Total Cost
82.0 ha	Clearing/grubbing & removal of horizon 'A'	$ 8,100	$ 664,200
3,726,235 m3	Ring Dam - Placement of fill material from pit & TSF basin	$ 4.00	$ 14,904,940
35,850 m3	Clay as Foundation Key-in	$ 4.80	$ 172,080
370,000 m3	Blanket Drain_clean gravel (installed volume)	$ 10.80	$ 3,996,000
770,000 m2	Geo Fabric in sqm (4m x 225m per roll)	$ 1.38	$ 1,061,399
4,851 m	Concrete Drain - perimeter cut-off drain	$ 11.91	$ 57,775
4,260 m	PVC Pipe 50mm dia	$ 2.00	$ 8,520
2500 pcs	Rock Gabion Wire	$ 3.40	$ 8,500
5,000 m3	Rock Material for Gabion	$ 10.80	$ 54,000

Amount	TSF Construction Item	Unit Cost	Total Cost
45.0 ha	Scarify & Compaction of TSF floor	$ 9,720	$ 437,400
454,910 m3	Placement of additional clay lining-installed volume	$ 4.80	$ 2,183,568
580,000 m2	HDPE Liner - 500 m^2 per roll	$ 1.70	$ 986,000
580,000 m2	HDPE Liner Installation	$ 1.28	$ 739,500
550 m	Spillway (cut & concrete lined)	$ 11.91	$ 6,551
3,500 m	Protection Bund, including cut for key-in	$ 103.70	$ 362,950
1,793 m3	Lime to be mixed with the clay for posolanic effect	$ 164.10	$ 294,149
	Field Testing work & 3rd Party review/sign-off		$ 1,148,689
	Total:		***$27,086,221***

Table 21-12 - Tailings Storage Facility - Capital Cost Breakdown

This capital cost is split into three (3) parts associated with the three (3) stages of the TSF construction. The split of costs are:

- Stage 1 = $8,125,866
- Stage 2 = $13,543,111
- Stage 3 = $5,417,244

A capital cost has been calculated for general offices, car park and warehousing and is $492,300 for 800$m^2$ area. This excludes the mine offices and workshops which are included in the mining costs.

Fencing and gates to surround the mining site and other facilities represents a total of 7,138 m to 8,123 m depending upon the mine site configuration. The plant and other key sites within the mine site will also have fencing, which is approximately 1,921 m and 1,786 m in length, respectively. The total fencing cost, including installation and gates, has been costed at $2,402,700.

Water and drainage infrastructure will be provided to handle the TSF water requirements, roading drainage and general drainage around the facilities. The construction of the water and drainage infrastructure including ponds, drainage channels and wetland costing will be $1,154,575. The breakdown of the costing is shown in *Table 21-13 - Water Infrastructure Cost Breakdown* below.

Water Infrastructure Item	Cost
Wetlands	$ 400,000
TSF Ponds (x5)	$ 125,000
Drainage channels for ponds & wetlands (1,725 m)	$ 91,325
Other ponds (x3)	$ 75,000
General site drainage channels (8,750 m)	$ 463,250
Total	***$1,154,575***

Table 21-13 - Water Infrastructure Cost Breakdown

21.2.4. Other Capital

Land valuation costing for the purchase land (approx. 340 ha) affected by the mining operations is $21,847,200. The *Table 21-14 - Land Valuation Summary* below summarises the land valuation totals and land improvement amounts that make up the total land valuation cost. The table also includes a contingency factor and the MYR : USD ROE is 3.2 : 1.

Land Valuation & Improvement Description	Total Amount	Total Amount
Land Value	RM 51,600,200	$ 16,125,060
Land Improvement – Fishponds	RM 547,700	$ 171,160
Land Improvement – Concrete Structures	RM 9,009,800	$ 2,815,560
Land Improvement – Wooden & Other Structures	RM 2,397,800	$ 749,310
Total Land Value	**RM 63,555,500**	**$ 19,861,090**
Contingency (@ 10 %)	*RM 6,355,550*	*$ 1,986,110*
Total Land Value (incl. Contingency)	***RM 69,911,050***	***$21,847,200***

Table 21-14 - Land Valuation Summary

Basis for land costs is based on the provision of land costs and land improvement calculations supplied by a qualified chartered surveying company (Williams, Talhar, Wong & Yeo Sdn Bhd) for standard or base land types in the Jugan-Siniawan area. These base or standard land type parameters and costs were then applied to similar land parcels.

This is an estimate for the purposes of the feasibility study with a contingency of 10 % applied to cover minor variations. Prior to actual land purchases a full land valuation will need to be performed.

Where land is not fully required to be purchased, or is peripheral in nature or will only have minor impacts applicable (e.g. road through property) then these land parcels may be leased/rented from the leaseholder/landowner for some ongoing rental amount yet to be negotiated.

This approach may reduce the capital amount required for land purchase. This has not been applied to the costs above but is considered a possible upside to this cost.

Sale of land after completion of mining operations and full rehabilitation has been included with a percentage applied to the costing based on the potential saleable part.

21.2.5. Contingency & Other Factors

A contingency factor of 10 % has been applied to the major mining and processing capital items. Other minor contingencies and conservative costing has been applied throughout.

The following items are excluded from the capital cost calculations:

- Inflation and escalation;

- Costs associated with protection against currency fluctuations;
- Project financing costs.

21.2.6. Sustaining Capital

The sustaining capital is based on two parts; sustaining capital for the mining and sustaining capital for the process plant. For mining the sustaining capital is based on 5% of the fixed mining capital items per annum. For the processing the sustaining capital is based on 5% of the processing Opex cost per tonne (excluding consumable spares) multiplied by the annualised tonnage. Also included is calculation for major mobile plant replacement parts based on operating hours and usage. For the base case this is $1,963,102 ($53,300 mining, $943,500 mobile equipment & $966,300 processing) which is $490,800 per quarter.

This amount caters for such items as equipment upgrades and modifications, pump replacements, ancilliary mining equipment, major spares for plant and mobile equipment, etc. and other deferred capital e.g. future TSF expansion stages.

The sustaining capital also includes TSF extensions, but these have been costed individually and separately and are $18,156,320 for Stage 2 & 3 and are not part of the initial capital requirement.

21.2.7. Reclamation & Closure

The mine site rehabilitation costs have been calculated for mine closure, post mine closure (monitoring) and ongoing rehabilitation during operations (where applicable or possible). The total rehabilitation cost is $6,365,750 broken down into $2,403,780 for pre-closure/ongoing rehabilitation, $3,166,970 for mine closure activities and $795,000 for post closure monitoring.

The costs per major closure component and rehabilitation timing are listed in *Table 21-15 - Mine Rehabilitation Costs by Compnent & Closure Period* below.

Closure Component	Pre-closure Cost	Mine Closure Cost	Post-Closure Cost	Total Cost per Component
TSF	$1,286,310	$1,708,160	$421,840	**$3,416,310**
Waste Disposal	$570,380	$747,280	$176,940	**$1,494,600**
Nursery	$31,750	$31,750	$31,750	**$95,250**
Mine Pit	$ 0	$406,970	$96,880	**$503,850**
Infrastructure	$205,250	$272,810	$67,590	**$545,650**
During Construction	$310,090	$ 0	$ 0	**$310,090**
Total Cost per Timing	***$2,403,780***	***$3,166,970***	***$795,000***	***$6,365,750***

Table 21-15 - Mine Rehabilitation Costs by Component & Closure Period

21.3. Operating Costs

21.3.1. Mining Operating Costs

The open pit operating costs along with the associated mining costs are detailed in *Table 21-13 – OPEX Costs: Direct & Associated Mining Costs* below. The total mining cost (per tonne) is then derived for both ore and waste mining for the owner-operator mining type. The contract-mining cost markups based on current mining contractor rates within Besra are detailed at the bottom of the table.

Mining Cost Item Description	Qty		Unit Cost (US$)		Total Cost (US$)	Cost/Tonne (US$/t)
DRILLING: For Ore						
Blastholes	24	holes			Annualised	
Drill Bit	264	m	$ 2.08	/m	$ 200,156	$ 0.069
Drill Rod	264	m	$ 1.47	/m	$ 141,456	$ 0.048
Diesel	354	liters	$ 0.90	/litre	$ 104,577	$ 0.036
Spare parts, tires & lube	7.7	hrs	$38.00	/UT hr	$ 95,988	$ 0.033
Drilling Labour	23.1	hrs	$ 4.38	/hr	$ 36,880	$ 0.013
Total Drilling Cost_ore					**$ 579,056**	**$ 0.20**
Drilling: For Waste						
Blastholes	24	holes				
Drill Bit	264	m	$ 2.40	/m	$ 323,339	$ 0.067
Drill Rod	264	m	$ 1.47	/m	$ 203,558	$ 0.041
Diesel	354	liters	$ 0.90	/litre	$ 150,487	$ 0.030
Spare parts, tires & lube	7.7	hrs	$38.00	/UT hr	$ 138,129	$ 0.028
Drilling Labour	23.1	hrs	$ 4.38	/hr	$ 53,070	$ 0.011
Total Drilling Cost_Waste					**$ 877,584**	**$ 0.18**
BLASTING: For Ore	24	holes				
Emulsion (wet/dewatered holes) using Fortis bulk system	1,498	kgs	$ 2.35	/kg	$ 1,283,351	$ 0.440
Package Explosive	-	kgs	$ 3.08	/kg	$ -	$ -
Non electric detonators Exel MS 7.3m	24.0	pcs	$14.82	each	$ 129,668	$ 0.044
Pentex Booster H (120g) as primer/initiator	24.0	pcs	$13.04	/piece	$ 114,053	$ 0.039
Exel Lead Line	150.0	m	$ 1.02	/m	$ 55,496	$ 0.019
Mixing Emulsion, delivery & charging (3-men)	144.0	m	$ 3.50	/hr	$ 183,709	$ 0.063
Dewatering & stemming (3-men)	6.0	hrs	$ 3.50	/hr	$ 7,655	$ 0.003
Total Blasting Cost_Ore					**$ 1,773,932**	**$ 0.61**
Blasting: For Waste						
Emulsion (wet/dewatered holes) using Fortis bulk system	1,488	kgs	$ 2.35	/kg	$ 1,846,764	$ 0.370
Package Explosive	-	kgs	$ 3.08	/kg	$ -	$ -
Non electric detonators Exel MS 7.3m	24.0	pcs	$14.82	each	$ 186,595	$ 0.038

Mining Cost Item Description	Qty		Unit Cost (US$)		Total Cost (US$)	Cost/Tonne (US$/t)
Pentex Booster H (120g) as primer/initiator	24.0	pcs	$13.04	/piece	$ 164,124	$ 0.033
Exel Lead Line	150.0	m	$ 1.02	/m	$ 79,859	$ 0.016
Mixing Emulsion, delivery & charging (3-men)	144.0	m	$ 3.50	/hr	$ 264,361	$ 0.053
Dewatering & stemming (3-men)	6.0	hrs	$ 3.50	/hr	$ 11,015	$ 0.002
Total Blasting Cost_Waste					**$ 2,552,718**	**$ 0.51**
Loading: CAT 6015FS Hydraulic Shovel/Excavator	2	units			Annualised	
Diesel	651,200	litres	$ 0.90	/litre	$ 527,472	$ 0.049
Parts, materials/supplies & Lube	11,840	hrs	$60.00	/hr	$ 639,360	$ 0.060
Operating Labour (2man-crew)	23,680	hrs	$ 4.38	/hr	$ 103,600	$ 0.010
Maintenance Labour (3-man crew)	12,618	hrs	$ 4.31	/hr	$ 54,320	$ 0.005
Total Loading Cost - Ore					**$ 1,324,752**	**$ 0.12**
Total Loading Cost - Waste						**$ 0.12**
Hauling Ore: CAT 772G - Rigid Dump Truck	4	units			Annualised	
Diesel	992,208	litres	$ 0.90	/litre	$ 803,688	$ 0.192
Parts, materials/supplies & Lube based on machine hrs	23,624	hrs	$30.00	/hr	$ 637,848	$ 0.153
Operating Labour based on machine hours	23,624	hrs	$ 4.38	/hr	$ 103,355	$ 0.025
Maintenance Labour (2-man crew)	17,896	hrs	$ 4.31	/hr	$ 77,042	$ 0.018
Total ORE Hauling Cost					**$1,621,934**	**$ 0.39**
Hauling Waste: CAT 772G_Rigid Dump Truck	5	units			Annualised	
Diesel	1,240,260	litres	$ 0.90	/litre	$ 1,004,611	$ 0.144
Parts, materials/supplies & Lube	29,530	hrs	$30.00	/hr	$ 797,310	$ 0.115
Operating Labour based on machine hours	29,530	hrs	$ 4.38	/hr	$ 129,194	$ 0.019
Maintenance Labour (2-man crew)	22,370	hrs	$ 4.31	/hr	$ 96,303	$ 0.014
Total Waste Hauling Cost					**$ 2,027,417**	**$ 0.29**
Dozing/Ripping, CAT D10 & CAT D6R	2	units			Annualised	
Diesel	422,928	litres	$ 0.90	/litre	$ 342,572	$ 0.036
Parts, materials/supplies & Lube	10,638	hrs	$41.50	/hr	$ 397,329	$ 0.041
Operating Labour	10,638	hrs	$ 4.38	/hr	$ 46,541	$ 0.005
Maintenance Labour (2man-crew)	9,302	hrs	$ 4.31	/hr	$ 40,045	$ 0.004
Total Dozing /Ripping Cost					**$ 826,487**	**$ 0.09**
Mining Ancilliary:					Annualised	
Wheel Loader for Stockpile operation & back-up pit loader	5520	UT hrs	$56.00	/hr	$ 861,120.00	$ 0.09
Road Grader / Road Maintenance	4,800	UT hrs	$80.00	/hr	$ 384,000.00	$ 0.040
Water Truck / Road Maintenance	3,300	UT hrs	$90.00	/hr	$ 297,000.00	$ 0.031
Mobile Mixing Unit (MMU) or Explosive Truck	6,210	UT hrs	$80.00	/hr	$ 496,800.00	$ 0.052
Service/Tire Truck	8760	UT hrs	$90.00	/hr	$ 788,400.00	$ 0.082
Service Vehicle Leased (4 units)	5	units	$1,500	/month	$ 7,500.00	$ 0.001

Mining Cost Item Description	Qty		Unit Cost (US$)		Total Cost (US$)	Cost/Tonne (US$/t)
Diesel (180 liters per month per unit)	10,800	litre	$ 0.66	/litre	$ 7,088.47	$ 0.001
Power for dewatering pumps (90kW)	345,600	kWhr	$0.068	/kWhr	$ 23,500.80	$ 0.002
Total Mine Ancilliary Cost - ore					**$ 2,865,409**	**$ 0.30**
Total Mine Ancilliary Cost - waste						**$ 0.30**
Waste Dump Maintenance					Annualised	
Grader	4,800	UT hrs	$80.00	/hr	$ 384,000	$ 0.057
Compactor	4,800	UT hrs	$80.00	/hr	$ 384,000	$ 0.057
Tractor/Dozer	4,800	UT hrs	$80.00	/hr	$ 384,000	$ 0.057
Water Truck	3300	UT hrs	$90.00	/hr	$ 297,000	$ 0.044
Waste Dump Maintenance Cost					**$1,449,000**	**$ 0.22**
Grade Control					Annualised	
Blasthole samples by AAS	11664	samples	$ 7.50	/sample	$87,480.00	$ 0.030
Blasthole samples by Fire Assay	2916	samples	$12.00	/sample	$34,992.00	$ 0.012
Trench/Channel samples	1458	samples	$12.00	/sample	$17,496.00	$ 0.006
Total Grade Control Cost					**$139,968**	**$ 0.05**
Ground/Slope Support					Annualised	
Cable Bolting - 6m to 16m	20,800	m	$ 5.29	/m	$110,032.00	$ 0.04
Wire Mesh & straps	10,400	set	$ 7.50	/set	$ 78,000.00	$ 0.03
Grout (approx 5kg per hole)	104,000	kg	$ 2.20	/kg	$228,800.00	$ 0.08
Drilling (all in)	20,800	m	$ 6.80	/m	$141,440.00	$ 0.05
Bolting Labour	41,600	hrs	$ 2.80	/hr	$116,480.00	$ 0.04
Total Ground Support Cost					**$ 674,752.00**	**$ 0.23**
Ore Mining – Owner-Operator						**$ 2.02**
Waste Mining – Owner-Operator						**$ 1.72**
	Cost	*Profit*	*Total*			
		Mark-up				
Ore Mining – Mining-Contract	*5%*	*25%*	*30%*			**$ 2.62**
Waste Mining – Mining-Contract	*5%*	*25%*	*30%*			**$ 2.23**

Table 21-16 – OPEX Costs: Direct & Associated Mining Costs

For the contract mining an additional amount of $0.49/tonne is added to the cost for mining equipment costs applied by the contractor.

Mining labour costs are summarised in *Table 21-14 - OPEX Costs - Direct & Indirect Mining Labour* below, both for direct and in-direct mining labour. The detailed breakdown is included in *Appendix 21-3*.

Labour Cost Item Description	Qty	Total Cost (US$/mth)
Direct Labour (pit operations) - costing included in OPEX1_Mining		
Equipment Operators	74	$ 84,996.03
Shop Mechanics	10	$ 11,485.95

Labour Cost Item Description	Qty	Total Cost (US$/mth)
Service Mechanics	4	$ 4,594.38
Shop Electrician	4	$ 4,594.38
Service Electrician	3	$ 3,445.79
Helper/Utility	12	$ 7,876.08
Direct Labour	**107**	**$116,992.61**
Manager & Supervision Staff Labour:		
Mine Manager Expat	1	$20,000.00
Mine Shift Foreman	3	$19,690.20
Planning Engineer	1	$5,907.06
Shift Supervisor	6	$29,535.30
Pit Geologist	2	$9,845.10
Resource/Reserve Geologist	1	$5,907.06
Geotech Engineer	1	$5,907.06
Chief Surveyor	1	$3,281.70
Safety Manager	1	$3,281.70
Safety Supervisor	3	$4,922.55
Fleet Maintenance Manager	1	$4,922.55
Mechanical Engineer	1	$3,281.70
Maintenance Supervisor	3	$4,922.55
Maintenance Planner	1	$1,969.02
Electrical Engineer	1	$3,281.70
Electrical Supervisor (maint)	3	$4,922.55
Warehouse Manager	1	$3,281.70
Warehouse Supervisor	2	$3,281.70
Environment Engineer	1	$3,281.70
Tailings Dam Manager	1	$3,281.70
Supervisor (tailings dam)	3	$4,922.55
Mine Overhead Labour	**38**	**$149,627.15**
Mine Service Department		
Safety Officer/Trainer	2	$3,281.70
Mine Clerk/Statisticians	2	$3,281.70
Grade Control Technician	3	$2,658.18
Samplers	6	$5,316.35
Surveyor	1	$1,640.85
Survey crew	4	$3,544.24
Geotech crew	2	$1,772.12
Security manager	1	$3,281.70
Security guards	12	$10,632.71
Mine Services Labour	**33**	**$35,409.55**
Engineering Services		
Engineering Manager	1	$15,000.00
Civil Engineer	1	$3,281.70
Mechanical Engineer	1	$3,281.70

Labour Cost Item Description	Qty	Total Cost (US$/mth)
Electirlcal Engineer	1	$3,281.70
Engineering Labour	**4**	**$24,845.10**
Admin, PR & HR		
Mine Admin Manager	1	$15,000.00
HR Manager	1	$3,281.70
PR Manager	1	$3,281.70
Office Personnel	9	$7,383.83
Admin Labour	**12**	**$28,947.23**
Procurement, Accounting & Finance & ICT		
Procurement Manager	1	$15,000.00
Procurement Staff/ Buyer	3	$2,953.53
Finance Mgr/Comptroller	1	$3,938.04
Accountant	1	$1,969.02
Cashier	1	$984.51
Accounting Staff	2	$1,969.02
IT Manager	1	$3,938.04
IT Technician	2	$2,625.36
PAFI Labour	**12**	**$33,377.52**
Tailings Dam Labour:		
Tailings Dam Crew	6	$ 4,922.55
Total Labour Costs:		**$277,129.10**
Overhead Labour:		
Labour_Staff Onsite Costs	15	$41,569
Labour_Travel & Accommodation	15	$41,569
Contractual Expats/Consultants		$50,000
Total Overhead Costs		**$133,138.00**
Grand Total Labour/Overhead		**$410,267.10**
Total Annual Labour Costs:		**$4,923,205**
Personnel with PPEs	*188*	
Labour Cost per tonne (for MCAF)		**$ 0.62**

Table 21-17 - OPEX Costs - Direct & Indirect Mining Labour

Additional mining related operating costs are summarised in *Table 21-17 - OPEX Costs - Mine Engineering Services Costing* covering the engineering services related to mining; and Table 21-18 - OPEX Costs - Technical Services, Health & Safety and Sundry Costing covering the mining services, health & safety and sundry mining costs both at the 8,000 tpd rate. The detailed tables are shown in *Appendix A21-4* and *Appendix A21-5* respectively.

Engineering Cost Item Description	Total Cost (US$)	Cost/Tonne (US$/t)
Services:		
Water Pipe - Service Water	$ 29,160.00	$ 0.010

Engineering Cost Item Description	Total Cost (US$)	Cost/Tonne (US$/t)
HDPE Pipe - pit dewatering pipes	$ 29,160.00	$ 0.010
Water Pipe Clamps	$ 8,323.20	$ 0.003
Water Pipe - Bends	$ 106.70	$ 0.000
Water Pipe - Valves	$ 416.65	$ 0.000
LT Equipment	$ 7,700.00	$ 0.003
LT Equipment - Frames	$ 905.44	$ 0.000
Pipe Support	$ 25,044.00	$ 0.009
Electric Cable - 70mm XLPE	$ 39,660.00	$ 0.014
Elec Cable - 70mm XLPE 1000/600V	$	$ -
Electric Cable - 4C/16mm	$ 5,920.00	$ 0.002
Luminaires	$ 1,435.00	$ 0.000
Bulbs	$ 260.00	$ 0.000
Dewatering pump consumables	$ 8,000.00	$ 0.003
Total Services:	***$ 156,090.99***	***$ 0.05***
Electricity		
Workshop & equipment (70kW)	$ 29,245.44	$ 0.010
Mobile Light Plant for pit (6 x 13kW)	$ 16,292.80	$ 0.006
Offices & accommodation (20kW)	$ 8,355.84	$ 0.003
Electricity for pumps (in opex1)		
Total Electricity	***$ 53,894.08***	***$ 0.02***
Sundries		
Potable Water	$ 1,575.22	$ 0.001
Water for Workshop	$ 2,362.82	$ 0.001
Cleaners - Degreasing	$ 50,112.00	$ 0.017
Total Sundries:	***$ 54,050.04***	***$ 0.02***
TOTAL ENGINEERING COSTS	**$ 264,035.11**	**$ 0.09**

Table 21-18 - OPEX Costs - Mine Engineering Services Costing

Cost Item Description	Total Cost (US$)	Cost/Tonne (US$/t)
Health & Safety:		
Boots	$ 16,287.73	$ 0.004
Hard Hats	$ 3,701.76	$ 0.001
Overalls	$ 6,367.02	$ 0.002
Gloves	$ 96.25	$ 0.000
Belts	$ 26,652.65	$ 0.007
Ear Muffs	$ 6,663.16	$ 0.002
Glasses	$ 4,145.97	$ 0.001
First Aid Materials	$ 5,119.45	$ 0.001
Reflector Jackets	$ 5,922.81	$ 0.002
Danger Tape	$ 2,047.78	$ 0.001

Cost Item Description	Total Cost (US$)	Cost/Tonne (US$/t)
Hand Torches	$ 1,122.34	$ 0.000
Safety Signage	$ 6,563.40	$ 0.002
Total Health & Safety:	***$ 84,690.33***	***$ 0.02***
Mining Services:		
Sampling Materials		
Sample Bags	$ 15,789.57	$ 0.005
Hammers	$ 59.07	$ 0.000
Spray Paint	$ 1,575.22	$ 0.001
Measuring Tape	$ 23.63	$ 0.000
Survey Materials		$ -
Survey Pegs	$ 1,260.17	$ 0.000
Spray Paint	$ 1,575.22	$ 0.001
Measuring Tape	$ 23.63	$ 0.000
Geology Materials		$ -
Sample Bags	$ 1,181.41	$ 0.000
Geology Hammers	$ 39.38	$ 0.000
Spray Paint	$ 1,575.22	$ 0.001
Measuring Tape	$ 23.63	$ 0.000
Office Items/Supplies		$ -
Software Licenses/ Maintenance	$ 30,000.00	$ 0.010
Office Supplies	$ 1,969.02	$ 0.001
Total Mining Services:	***$ 55,095.16***	***$ 0.02***
Sundries:		
Paint	$ 393.80	$ 0.0001
Spray Paint	$ 2,100.29	$ 0.0007
Measuring Tapes	$ 118.14	$ 0.0000
Hand Tools	$ 1,969.02	$ 0.0007
Pad Locks	$ 393.80	$ 0.0001
Shovels & Picks	$ 708.85	$ 0.0002
Hammers	$ 590.71	$ 0.0002
Heavy Duty Plastic	$ 11,814.12	$ 0.0040
Cement	$ 10,553.95	$ 0.0036
Nails, Nuts & Bolts	$ 220.53	$ 0.0001
Battery Fluid	$ 656.34	$ 0.0002
Oxygen	$ 2,629.35	$ 0.0009
Acetylene	$ 2,192.70	$ 0.0008
Washers	$ 656.34	$ 0.0002
Gaskets	$ 1,640.85	$ 0.0006
Total Sundries:	***$ 36,638.79***	***$ 0.01***
TOTAL GENERAL:	**$ 176,424.28**	**$ 0.05**

Table 21-19 - OPEX Costs - Technical Services, Health & Safety and Sundry Costing

21.3.2. Process Plant Operating Costs

This section provides the comparative operating costs for the treatment of Jugan ore at 4,000 tpd, 8,000 tpd and 12,000 tpd milling rates. Four options have been considered, namely production of a flotation concentrate to be sold to a third party in the first option (base case scenario) and production of gold dore by further processing of the flotation concentrate on location by either the BIOX (biological oxidation route), the POX (pressure oxidation route) or Albion (ultrafine grinding and atmospheric oxidation route). Details of these processes have been provided in *Section 17* above.

The operating costs have been incorporated in conjunction with the capital costs in the feasibility model for each process and option. The feasibility model is discussed in *Section 22* of this report.

21.3.2.1. Operating – Flotation Concentrate

Listed below in *Table 21-13 - OPEX Costs - 8,000 tpd Flotation Concentrate Option* is a breakdown of the operating costs for the 8,000 tpd flotation concentrate option.

Item	Unit Cost	Flotation Concentrate	
		Consumption	Cost
	US$/kg	(kg/t)	(US$/t)
Power	0.07/kWh	35kW/t	2.45
Steel Balls	1.6	0.74	1.18
Grinding Media		0	0.00
CuSO4	2.45	0.2	0.49
CMC	2.00	0.2	0.40
PAX	2.13	0.15	0.32
Frother,MIBC	3.2	0.04	0.13
Promoter	3.2	0.035	0.11
Nutrients	0.7	0	0.00
Floculent	5	0.015	0.08
Coagulent	2.19	0	0.00
Oxygen	0.02	0	0.00
Limestone	0.035	0	0.00
Lime	0.2	0.5	0.10
NaOH	0.7	0	0.00
NaCN	3.8	0	0.00
Carbon	2.8	0	0.00
Na2S2O5	0.8	0	0.00
LPG	0.58	0	0.00
HCl	0.47	0	0.00
Manpower		80	0.67
Maintenance	(4% CAPEX/yr)	$50.4M	0.69
Total:			**6.62**
Spares		5.5%	0.95
Total Operating Cost:			**7.57**

Table 21-20 - OPEX Costs - 8,000 tpd Flotation Concentrate Option

21.3.2.2. Operating – BIOX

Listed below in *Table 21-14 - OPEX Costs - 8,000 tpd Biological Oxidation Option* is a breakdown of the operating costs for the 8,000 tpd biological oxidation (BIOX) option.

Item	Unit Cost	Biological Oxidation	
		Consumption	Cost
	US$/kg	(kg/t)	(US$/t)
Power	0.07/kWh	70kW/t	4.90
Steel Balls	1.6	0.74	1.18
Grinding Media		0	0.00
CuSO4	2.45	0.35	0.86
CMC	2.00	0.2	0.40
PAX	2.13	0.15	0.32
Frother,MIBC	3.2	0.04	0.13
Promoter	3.2	0.035	0.11
Nutrients	0.7	0.75	0.53
Floculent	5	0.112	0.56
Coagulent	2.19	0.135	0.30
Oxygen	0.02	0	0.00
Limestone	0.035	60	2.10
Lime	0.2	8.25	1.65
NaOH	0.7	0	0.00
NaCN	3.8	1.95	7.41
Carbon	2.8	0.05	0.14
Na2S2O5	0.8	2	1.60
LPG	0.58	1L/t	0.58
HCl	0.47	0.037	0.02
Manpower		119	1.00
Maintenance	(4% CAPEX/yr)	$156M	2.83
Total:			**26.62**
Spares		5.5%	3.89
Total Operating Cost:			**30.49**

Table 21-21 - OPEX Costs - 8,000 tpd Biological Oxidation Option

21.3.2.3. Operating – POX

Listed below in *Table 21-15 - OPEX Costs - 8,000 tpd Pressure Oxidation Option* is a breakdown of the operating costs for the 8,000 tpd pressure oxidation (POX) option.

Item	Unit Cost	Pressure Oxidation	
		Consumption	Cost
	US$/kg	(kg/t)	(US$/t)
Power	0.07/kWh	67kW/t	4.69
Steel Balls	1.6	0.74	1.18
Grinding Media		0	0.00
CuSO4	2.45	0.235	0.58
CMC	2.00	0.2	0.40
PAX	2.13	0.15	0.32
Frother,MIBC	3.2	0.04	0.13

Item	Unit Cost	Pressure Oxidation	
		Consumption	Cost
	US$/kg	(kg/t)	(US$/t)
Promoter	3.2	0.035	0.11
Nutrients	0.7	0	0.00
Floculent	5	0.112	0.56
Coagulent	2.19	0.135	0.30
Oxygen	0.02	69	1.38
Limestone	0.035	80	2.80
Lime	0.2	1.8	0.36
NaOH	0.7	0	0.00
NaCN	3.8	0.75	2.85
Carbon	2.8	0.05	0.14
Na2S2O5	0.8	0.8	0.64
LPG	0.58	1L/t	0.58
HCl	0.47	0.037	0.02
Manpower		119	1.00
Maintenance	(4% CAPEX/yr)	$191.7M	3.32
Total:			**21.36**
Spares		7.5%	6.22
Total Operating Cost:			**27.56**

Table 21-22 - OPEX Costs - 8,000 tpd Pressure Oxidation Option

21.3.2.4. Operating – Albion

Listed below in *Table 21-16 - OPEX Costs - 8,000 tpd Albion Process Option* is a breakdown of the operating costs for the 8,000 tpd Albion process option.

Item	Unit Cost	Albion Process	
		Consumption	Cost
	US$/kg	(kg/t)	(US$/t)
Power	0.07/kWh	83kW/t	5.81
Steel Balls	1.6	0.74	1.18
Grinding Media		0	0.50
CuSO4	2.45	0.375	0.92
CMC	2.00	0.2	0.40
PAX	2.13	0.15	0.32
Frother,MIBC	3.2	0.04	0.13
Promoter	3.2	0.035	0.11
Nutrients	0.7	0	0.00
Floculent	5	0.112	0.56
Coagulent	2.19	0.135	0.30
Oxygen	0.02	47	0.94
Limestone	0.035	60	2.10
Lime	0.2	9.75	1.95
NaOH	0.7	5.25	3.68
NaCN	3.8	2.1	7.98
Carbon	2.8	0.05	0.14
Na2S2O5	0.8	2	1.60
LPG	0.58	1L/t	0.58

Item	Unit Cost	Albion Process	
		Consumption	Cost
	US$/kg	(kg/t)	(US$/t)
HCl	0.47	0.037	0.02
Manpower		119	1.00
Maintenance	(4% CAPEX/yr)	$169.2M	3.01
Total:			**33.23**
Spares		6.0%	4.51
Total Operating Cost:			**37.28**

Table 21-23 - OPEX Costs - 8,000 tpd Albion Process Option

21.3.3. Transportation & Infrastructure Costs

In the base case option the concentrate transportation costs are applicable from site to the concentrate processing facility in China. The concentrate transport costs are $32.76/concentrate tonne, and are inclusive of transport from mine site to port/warehouse, port/warehouse rehandling and sea freight, etc.

If gold is produced on-site then the transport and refining costs are listed below under Non-OPEX items.

21.3.4. Overheads, G&A Costs

Accounts have provided a rate of $0.55 per tonne for overheads and this is in line with the current rate for Besra operations in Vietnam.

21.4. Non-OPEX Items

21.4.1. Transportation & Refining

The refining, including transport, is only applicable if the POX, BIOX or Albion process options are selected. The base case option would see the gold refined by the concentrate processing company. Therefore, costs are applied only for these options and the costs are based on our current costs as per our Vietnam operations. Currently Besra's gold is refined in Switzerland but options for refining in Singapore are being investigated with the option to reduce these costs.

For the concentrate option the costs for transport from mine-site to processing/smelting facility are calculated based on standard transport and shipping rates (see above). These costs are not applicable when the full processing options on-site are selected.

If the full processing option is selected then the cost to transport and ship gold for refining is $4.50/oz and $2.50/oz respectively.

21.4.2. Royalties, Taxes, Tariffs & Tax Incentives

21.4.2.1. Royalties, Taxes & Tariffs

In Malaysia the corporate income tax is 24 % of net taxable profits. Other taxes are GST (10 %) and where applicable a service tax (6 %) – where services only are provided.

Import duties are applicable at a rate of bewtween 10-30 % for most standard goods; however, drilling and mining equipment are subject to nil import tariffs based on the individual item and related part numbers.

Employees and company contribute to the Malaysian Employee Provident Fund (EPF) and to employee insurance (SOCSO). A training levy will be imposed starting 2014 and is a 1 % levy on the local employees salary amount. These amounts are included in the labour rates.

There is no royalty (0 %) on gold produced in Sarawak, and there is no export duty or tariff for gold concentrate.

21.4.2.2. Tax Incentives

Pioneer Status is a standard concession available under which companies can apply for this status which allows 70 % of the net income of the project to be tax free for the first five years. That can also be extended for a second five year term under certain circumstances. Pioneer status must be applied for prior to project commencement.

Pioneer status is not automatically available and must be applied for. It is generally granted for companies within industries that the government wishes to encourage. It is noted that the committee will favour the submission by a company that intends to use local labour predominantly, and will source inputs locally as well as having a unique product or processing facility.

Investment Tax Allowance (ITA) is a capital expenditure-based incentive which is given by way of an exemption of income. ITA is a 'once only' allowance which is given at the standard rate of 60% of qualifying capital expenditure for the basis period in which the capital expenditure is incurred. Eligibility lasts for five years from the date of approval. The allowance is used to exempt statutory income, with a limit of 70% on that income.

Any allowance not used may be carried forward indefinitely. Where the 70% restriction applies, the balance of 30%, as under pioneer status, becomes liable to tax. Capital expenditure refers to capital expenditure incurred on a factory or on any plant and machinery used in Malaysia in connection with, and for the purposes of, the promoted activity or product. It does not include buildings used as living accommodation.

Income is computed in the normal way down to statutory income level, at which point the eligibility for exemption can be determined. Normal capital allowances may also be claimed but there is no compulsion to do so. Loss relief is also given in the normal way. Unlike pioneer status, there is no requirement to offset losses against exempt income.

Exploration and prospecting costs are eligible for special tax allowances to which we will be entitled.

22. Economic, Sensitivity & Risk Analysis

22.1. Economic Analysis

A cost model was derived to be able to analyse each of the 650+ possible scenarios. These were based on the main factors making up all the possible options available. These have been subsequently refined down to ±40 options. The remaining options are still built into the cost model and can be re-visited if this option becomes available or more information/data comes to hand. The model is built around the selection of a scenario number which then calculates or updates the costing worksheets, this information is then extracted and summarised in the cost model worksheet. *Figure 22-1 – Extract from Cost Model Scenario Options List* below shows an extract from the options table showing some examples of the option parameters.

Option	Ore Source	Production Rate (tpd)			Production Options			Process Rate (tpd)			Crush/Grind	Flotation	Oxidation-CIL	Oxidation	Secondary	CIL	Au Recovery			Transport	Transport	Contractor
		1st	2nd	3rd	1st	2nd	3rd	1st	2nd	3rd	Location	Location	Location	Process	Process	Float Tails	Primary	Direct CIL	Heap Leach	Option 1	Option 2	Option
445	Jugan+BYG	10,000	10,000		10_ALBN_C2			10,000	10000		On-Site	Central	Central	ALBION	HL1	Y2	80%		50%	Conveyor		Y
446	Jugan+BYG	10,000	10,000		10_ALBN_C2			10,000	10000		On-Site	Central	Central	ALBION	HL1	Y2	80%		50%	Highway-Truck		Y
447	Jugan+BYG	10,000	10,000		10_BIOX_C2			10,000	10000		On-Site	Central	Central	BIOX	HL1	Y2	80%		50%	Conveyor		Y
448	Jugan+BYG	10,000	10,000		10_BIOX_C2			10,000	10000		On-Site	Central	Central	BIOX	HL1	Y2	80%		50%	Highway-Truck		Y
449	Jugan+BYG	8,000	8,000		8_POX_B2			8,000	8000		On-Site	On-Site	On-Site	POX		Y2	85%			Site-Truck		N
450	Jugan+BYG	8,000	8,000		8_ALBN_B2			8,000	8000		On-Site	On-Site	On-Site	ALBION		Y2	80%			Site-Truck		N
451	Jugan+BYG	8,000	8,000		8_BIOX_B2			8,000	8000		On-Site	On-Site	On-Site	BIOX		Y2	80%			Site-Truck		N
452	Jugan+BYG	8,000	8,000		8_FLOT_B2			8,000	8000		On-Site	On-Site	Overseas	FLOTATION		Y2	72%			Site-Truck	Truck-Shipping	N
453	Jugan+BYG	8,000	8,000		8_POX_B2			8,000	8000		On-Site	On-Site	On-Site	POX	HL1	Y2	80%		50%	Site-Truck		N
454	Jugan+BYG	8,000	8,000		8_ALBN_B2			8,000	8000		On-Site	On-Site	On-Site	ALBION	HL1	Y2	80%		50%	Site-Truck		N
455	Jugan+BYG	8,000	8,000		8_BIOX_B2			8,000	8000		On-Site	On-Site	On-Site	BIOX	HL1	Y2	80%		50%	Site-Truck		N
456	Jugan+BYG	8,000	8,000		8_FLOT_B2			8,000	8000		On-Site	On-Site	Overseas	FLOTATION	HL1	Y2	77%		50%	Site-Truck	Truck-Shipping	N
457	Jugan+BYG	8,000	8,000		8_POX_B2			8,000	8000		On-Site	On-Site	Central	POX		Y2	85%			Pipeline		N
458	Jugan+BYG	8,000	8,000		8_POX_B2			8,000	8000		On-Site	On-Site	Central	POX		Y2	85%			Slurry-Truck		N
459	Jugan+BYG	8,000	8,000		8_ALBN_B2			8,000	8000		On-Site	On-Site	Central	ALBION		Y2	80%			Pipeline		N
460	Jugan+BYG	8,000	8,000		8_ALBN_B2			8,000	8000		On-Site	On-Site	Central	ALBION		Y2	80%			Slurry-Truck		N
461	Jugan+BYG	8,000	8,000		8_BIOX_B2			8,000	8000		On-Site	On-Site	Central	BIOX		Y2	80%			Pipeline		N
462	Jugan+BYG	8,000	8,000		8_BIOX_B2			8,000	8000		On-Site	On-Site	Central	BIOX		Y2	80%			Slurry-Truck		N

Figure 22-1 – Extract from Cost Model Scenario Options List

22.1.1. Pre-Tax Basis

Some assumptions were used in the cost models and the main assumptions for the base case options, are listed below:

- Gold price fixed at $1,300, though sensitivities have been investigated (see below);
- Based on a discounted cash flow model on a pre-tax basis;
- Production levels are fixed for each production option, except in build up and end;
- High grading with higher production initially and lower production has been investigated, though processing was maintained at constant level – this was only applicable in limited cases but is an option to be investigated further if required;
- NPV was fixed at 8% and calculated based on the net cash flow generated from the Project;
- No escalation or inflation factors were taken into account (constant 2013);
- The IRR on total investment was calculated based on 100% equity financing;
- Production schedules for Jugan and BYG-Krian are linear/serial and were not done in parallel, though this option is possible and may need investigation if required;
- Pre-mining occurs in all options and a six-month build up applied;
- Processing is offset by one quarter to allow for commissioning, build up and throughput lag;
- Phased capital was applied at the appropriate time ahead of requirements.

Table 22-1 - Cashflow Model - Option 484 (8,000tpd Contractor-Mining) and *Table 22-2 - Cashflow Model - Option 452 (8,000tpd Owner-Operator)* below presents the project cost model (before tax) for both base case options (owner-operator & contract-mining). *Enclosure B22-1* lists the cost models for both base case scenario options for comparative purposes. These are at A3 size for ease of reading.

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Mined Ore Tonnes	10,927,500	240,900	489,100	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,100	730,100	730,100	730,100	729,400	721,700	716,000	-	-	-	-	-
Mined Au Grade	1.70	1.53	1.53	1.53	1.53	1.53	1.58	1.58	1.58	1.58	1.56	1.56	1.56	1.56	1.59	2.11	3.24	-	-	-	-	-
Mined Au Ounces	598,830	11,870	24,090	35,960	35,960	35,960	36,990	36,990	36,990	36,990	36,600	36,600	36,600	36,600	37,190	48,940	74,500	-	-	-	-	-
Cumulative Mined Ore Tonnes		240,900	730,000	1,460,000	2,190,000	2,920,000	3,650,000	4,380,000	5,110,000	5,840,000	6,570,100	7,300,200	8,030,300	8,760,400	9,489,800	10,211,500	10,927,500	-	-	-	-	-
Cumulative Mined Au Grade		1.53	1.53	1.53	1.53	1.53	1.54	1.55	1.55	1.55	1.55	1.56	1.56	1.56	1.56	1.60	1.70	-	-	-	-	-
Cumulative Mined Au Ounces		11,870	35,960	71,920	107,880	143,840	180,830	217,820	254,810	291,800	328,400	365,000	401,600	438,200	475,390	524,330	598,830	-	-	-	-	-
Processed Ore Tonnes	10,927,500		240,900	489,100	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,100	730,100	730,100	730,100	729,400	721,700	716,000	-	-	-	-
Recovered Au Grade	1.32		1.19	1.19	1.19	1.19	1.19	1.22	1.22	1.22	1.22	1.21	1.21	1.21	1.21	1.23	1.63	2.51	-	-	-	-
Recovered Au Ounces	463,650		9,190	18,650	27,840	27,840	27,840	28,640	28,640	28,640	28,640	28,340	28,340	28,340	28,340	28,800	37,890	57,680	-	-	-	-
Cumulative Processed Ore Tonnes		-	240,900	730,000	1,460,000	2,190,000	2,920,000	3,650,000	4,380,000	5,110,000	5,840,000	6,570,100	7,300,200	8,030,300	8,760,400	9,489,800	10,211,500	10,927,500	-	-	-	-
Cumulative Recovered Au Grade		-	1.19	1.19	1.19	1.19	1.19	1.19	1.20	1.20	1.20	1.20	1.20	1.20	1.20	1.21	1.24	1.32	-	-	-	-
Cumulative Recovered Au Ounces		-	9,190	27,840	55,680	83,520	111,360	140,000	168,640	197,280	225,920	254,260	282,600	310,940	339,280	368,080	405,970	463,650	-	-	-	-
Waste Volume			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Tonnes	18,569,000	118,100	239,900	358,000	358,000	358,000	922,600	922,600	922,600	922,600	1,650,100	1,650,100	1,650,100	1,650,100	1,813,000	2,320,900	2,712,300	-	-	-	-	-
Cumulative Waste Tonnes		118,100	358,000	716,000	1,074,000	1,432,000	2,354,600	3,277,200	4,199,800	5,122,400	6,772,500	8,422,600	10,072,700	11,722,800	13,535,800	15,856,700	18,569,000	-	-	-	-	-
Strip Ratio	1.70	0.49	0.49	0.49	0.49	0.49	1.26	1.26	1.26	1.26	2.26	2.26	2.26	2.26	2.49	3.22	3.79	-	-	-	-	-
Cumulative Strip Ratio		0.49	0.49	0.49	0.49	0.49	0.65	0.75	0.82	0.88	1.03	1.15	1.25	1.34	1.43	1.55	1.70	-	-	-	-	-
Capital Costs:																						
Capital Development/Pre-stripping																						
Capital Costs Relative to Option																						
Capital Cost - Mining	$ 4,304,495	$ 4,304,495																				
Capital Cost - Processing (Main)	$ 58,547,853	$ 58,547,853																				
Capital Cost - Processing (Addn CIL Circuit)	$ -																					
Capital Cost - Processing (Heap Leach)	$ -																					
Capital Cost - Transport	$ -	$																				
Capital Cost - Other	$ 60,205,555	$ 29,087,341	$ -	$ -	$ 11,356,641	$ 8,469,245	$ 5,646,164	$ -	$ -	$ -	$ -	$ -	$ 2,823,082	$ 2,823,082	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Rehabilitation (Stage 1&2)	$ 7,160,750		$ -	$ -	$ -	$ -	$ -	$ -	$ 2,403,780	$ -	$ -	$ -	$ -	$ -	$ 3,166,970	$ -	$ 795,000	$ 795,000	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Process	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Land Acqisition	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - TSF & Other	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Rehabilitation	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Condemnation/Resource Drilling	$ 180,000	$ 180,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Annual Sustaining Capital	$ 4,479,262		$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ -	$ -	$ -	$ -	$ -
Total Capital Costs	$ 134,877,915	$ 92,119,690	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 3,121,699	$ 3,465,587	$ 298,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -	$ -
Cumulative Capital Costs		$ 92,119,690	$ 92,418,307	$ 92,716,925	$ 104,372,183	$ 113,140,046	$ 119,084,827	$ 119,383,444	$ 122,085,842	$ 122,384,459	$ 122,683,077	$ 122,981,694	$ 126,103,394	$ 129,225,093	$ 132,690,680	$ 132,989,298	$ 134,082,915	$ 134,877,915	$ -	$ -	$ -	$ -
Operating Costs:																						
Mining - Waste	$ 50,525,000	$ 321,342	$ 652,752	$ 974,094	$ 974,094	$ 974,094	$ 2,510,333	$ 2,510,333	$ 2,510,333	$ 2,510,333	$ 4,489,811	$ 4,489,811	$ 4,489,811	$ 4,489,811	$ 4,933,051	$ 6,315,013	$ 7,379,986	$ -	$ -	$ -	$ -	$ -
Mining - Ore	$ 33,862,504	$ 746,509	$ 1,515,640	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,458	$ 2,262,458	$ 2,262,458	$ 2,262,458	$ 2,260,289	$ 2,236,428	$ 2,218,765	$ -	$ -	$ -	$ -	$ -
Labour (Mine Overhead)	$ 18,872,319	$ 410,268	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ -	$ -	$ -	$ -	$ -
General Costs	$ 584,299	$ 12,881	$ 26,152	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,039	$ 39,039	$ 39,039	$ 39,039	$ 39,001	$ 38,590	$ 38,285	$ -	$ -	$ -	$ -	$ -
Engineering Costs	$ 988,097	$ 21,783	$ 44,226	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,018	$ 66,018	$ 66,018	$ 66,018	$ 65,955	$ 65,258	$ 64,743	$ -	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Main)	$ 196,325,629	$ -	$ 4,307,431	$ 8,745,221	$ 13,052,652	$ 13,052,652	$ 13,052,652	$ 13,068,652	$ 13,068,652	$ 13,068,652	$ 13,068,652	$ 13,064,363	$ 13,064,363	$ 13,064,363	$ 13,064,363	$ 13,061,581	$ 13,111,576	$ 13,409,805	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Addn CIL)	$ -																					
Metallurgical/Processing Costs (Heap Leach)	$ -																					
Transport Cost to Central Plant/Port	$ 35,796,851	$ -	$ 789,152	$ 1,602,218	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,698	$ 2,391,698	$ 2,391,698	$ 2,391,698	$ 2,389,405	$ 2,364,181	$ 2,345,509	$ -	$ -	$ -	$ -
General Overhead (BESRA)	$ 6,010,125	$ 132,495	$ 269,005	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,555	$ 401,555	$ 401,555	$ 401,555	$ 401,170	$ 396,935	$ 393,800	$ -	$ -	$ -	$ -	$ -
Total Operating Costs	$ 342,964,825	$ 1,645,278	$ 8,835,161	$ 15,321,027	$ 20,417,610	$ 20,417,610	$ 21,953,849	$ 21,969,849	$ 21,969,849	$ 21,969,849	$ 23,949,707	$ 23,945,746	$ 23,945,746	$ 23,945,746	$ 24,386,331	$ 25,734,013	$ 26,802,138	$ 15,755,314	$ -	$ -	$ -	$ -
Operating Cost per Tonne Ore	$ 31.39	$ 6.83	$ 18.06	$ 20.99	$ 27.97	$ 27.97	$ 30.07	$ 30.10	$ 30.10	$ 30.10	$ 32.80	$ 32.80	$ 32.80	$ 32.80	$ 33.43	$ 35.66	$ 37.43	$ -	$ -	$ -	$ -	$ -
Cumulative Operating Costs		$ 1,645,278	$ 10,480,439	$ 25,801,466	$ 46,219,077	$ 66,636,687	$ 88,590,536	$ 110,560,385	$ 132,530,234	$ 154,500,083	$ 178,449,789	$ 202,395,536	$ 226,341,282	$ 250,287,028	$ 274,673,359	$ 300,407,372	$ 327,209,511	$ 342,964,825	$ -	$ -	$ -	$ -
Total Costs:																						
Total Costs	$ 477,842,740	$ 93,764,968	$ 9,133,779	$ 15,619,644	$ 32,072,869	$ 29,185,473	$ 27,898,630	$ 22,268,466	$ 24,672,246	$ 22,268,466	$ 24,248,324	$ 24,244,364	$ 27,067,445	$ 27,067,445	$ 27,851,919	$ 26,032,631	$ 27,895,756	$ 16,550,314	$ -	$ -	$ -	$ -
Total Cumulative Costs		$ 93,764,968	$ 102,898,746	$ 118,518,391	$ 150,591,260	$ 179,776,733	$ 207,675,363	$ 229,943,829	$ 254,616,076	$ 276,884,542	$ 301,132,866	$ 325,377,230	$ 352,444,675	$ 379,512,121	$ 407,364,039	$ 433,396,670	$ 461,292,426	$ 477,842,740	$ -	$ -	$ -	$ -
Total Cost per Tonne Ore	$ 43.73	$ 389.23	$ 18.67	$ 21.40	$ 43.94	$ 39.98	$ 38.22	$ 30.50	$ 33.80	$ 30.50	$ 33.21	$ 33.21	$ 37.07	$ 37.07	$ 38.18	$ 36.07	$ 38.96	$ -	$ -	$ -	$ -	$ -
Total Cost per Ounce	$ 1,030.61	$ -	$ 993.88	$ 837.51	$ 1,152.04	$ 1,048.33	$ 1,002.11	$ 777.53	$ 861.46	$ 777.53	$ 846.66	$ 855.48	$ 955.10	$ 955.10	$ 982.78	$ 903.91	$ 736.23	$ 286.93	$ -	$ -	$ -	$ -
Total Cost per Tonne Ore (incl. Resale)	$ 41.29																					
Total Cost per Ounce (incl. Resale)	$ 973.14																					
Revenue:																						
Mine Call Factor	1.00	-	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	-	-	-	-
Capital Equipment Resale/Salvage	$ 8,291,104		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8,291,104	$ -	$ -	$ -	$ -
Asset Resale	$ 18,354,167	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 18,354,167	$ -	$ -	$ -
Gold Revenue	$ 602,745,000	$ -	$ 11,947,000	$ 24,245,000	$ 36,192,000	$ 36,192,000	$ 36,192,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 37,440,000	$ 49,257,000	$ 74,984,000	$ -	$ -	$ -	$ -
Marketing & Royalties																						
Freight/Transport	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Refining	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Royalties	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Marketing Costs	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Revenue Before Tax	$ 629,390,272	$ -	$ 11,947,000	$ 24,245,000	$ 36,192,000	$ 36,192,000	$ 36,192,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 37,440,000	$ 49,257,000	$ 83,275,104	$ 18,354,167	$ -	$ -	$ -
Cumulative Revenue		$ -	$ 11,947,000	$ 36,192,000	$ 72,384,000	$ 108,576,000	$ 144,768,000	$ 182,000,000	$ 219,232,000	$ 256,464,000	$ 293,696,000	$ 330,538,000	$ 367,380,000	$ 404,222,000	$ 441,064,000	$ 478,504,000	$ 527,761,000	$ 611,036,104	$ 629,390,272	$ -	$ -	$ -
Annual Revenue before Tax	$ 629,390,272	$ -				108,576,000				147,888,000				147,758,000				206,814,104				18,354,167
Annualised Cumulative Revenue		$ -				108,576,000				256,464,000				404,222,000				611,036,104				629,390,272
Quarterly Cash Flow	$ 151,547,532	($ 93,764,968)	$ 2,813,221	$ 8,625,356	$ 4,119,131	$ 7,006,527	$ 8,293,370	$ 14,963,534	$ 12,559,754	$ 14,963,534	$ 12,983,676	$ 12,597,636	$ 9,774,555	$ 9,774,555	$ 8,990,081	$ 11,407,369	$ 21,361,244	$ 66,724,791	$ 18,354,167	$ -	$ -	$ -
Cumulative Cash Flow		($ 93,764,968)	($ 90,951,746)	($ 82,326,391)	($ 78,207,260)	($ 71,200,733)	($ 62,907,363)	($ 47,943,829)	($ 35,384,076)	($ 20,420,542)	($ 7,436,866)	$ 5,160,770	$ 14,935,325	$ 24,709,879	$ 33,699,961	$ 45,107,330	$ 66,468,574	$ 133,193,364	$ 151,547,532	$ -	$ -	$ -
Annualised Cashflow	$ 151,547,532	($ 93,764,968)				22,564,235				50,780,191				45,130,421				108,483,485				18,354,167
Annualised Cumulative Cashflow		($ 93,764,968)				71,200,733)				20,420,542)				24,709,879				133,193,364				151,547,532
Yearly NPV @	$ 91,407,216																					
8%																						
Yearly IRR	38.0%																					
Estimated Dates for Project (Assuming Start Date)		30/06/2014	30/09/2015	31/12/2015	31/03/2016	30/06/2016	30/09/2016	31/12/2016	31/03/2017	30/06/2017	30/09/2017	31/12/2017	31/03/2018	30/06/2018	30/09/2018	31/12/2018	31/03/2019	30/06/2019	30/09/2019	31/12/2019	31/03/2020	30/06/2020
Period NPV @	$ 88,666,062																					
8%																						
Period IRR	29.5%																					

Table 22-1 - Cashflow Model - Option 484 (8,000tpd Contractor-Mining)

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Mined Ore Tonnes	11,209,300	241,100	489,400	730,500	730,500	730,500	729,900	729,900	729,900	729,900	730,300	730,300	730,300	730,300	726,700	668,500	728,300	323,000	-	-	-	-
Mined Au Grade	1.69	1.53	1.53	1.53	1.53	1.53	1.56	1.56	1.56	1.56	1.55	1.55	1.55	1.55	1.55	1.55	2.74	3.83	-	-	-	-
Mined Au Ounces	609,900	11,870	24,110	35,980	35,980	35,980	36,680	36,680	36,680	36,680	36,390	36,390	36,390	36,390	36,310	33,400	64,180	39,810	-	-	-	-
Cumulative Mined Ore Tonnes		241,100	730,500	1,461,000	2,191,500	2,922,000	3,651,900	4,381,800	5,111,700	5,841,600	6,571,900	7,302,200	8,032,500	8,762,800	9,489,500	10,158,000	10,886,300	11,209,300	-	-	-	-
Cumulative Mined Au Grade		1.53	1.53	1.53	1.53	1.53	1.54	1.54	1.55	1.55	1.55	1.55	1.55	1.55	1.55	1.55	1.63	1.69	-	-	-	-
Cumulative Mined Au Ounces		11,870	35,980	71,960	107,940	143,920	180,600	217,280	253,960	290,640	327,030	363,420	399,810	436,200	472,510	505,910	570,090	609,900	-	-	-	-
Processed Ore Tonnes	11,209,300		241,100	489,400	730,500	730,500	730,500	729,900	729,900	729,900	729,900	730,300	730,300	730,300	730,300	726,700	668,500	728,300	323,000	-	-	-
Recovered Au Grade	1.31		1.19	1.19	1.19	1.19	1.19	1.21	1.21	1.21	1.21	1.20	1.20	1.20	1.20	1.20	1.20	2.12	2.97	-	-	-
Recovered Au Ounces	472,240		9,190	18,670	27,860	27,860	27,860	28,400	28,400	28,400	28,400	28,180	28,180	28,180	28,180	28,110	25,860	49,690	30,820	-	-	-
Cumulative Processed Ore Tonnes		-	241,100	730,500	1,461,000	2,191,500	2,922,000	3,651,900	4,381,800	5,111,700	5,841,600	6,571,900	7,302,200	8,032,500	8,762,800	9,489,500	10,158,000	10,886,300	11,209,300	-	-	-
Cumulative Recovered Au Grade		-	1.19	1.19	1.19	1.19	1.19	1.19	1.19	1.20	1.20	1.20	1.20	1.20	1.20	1.20	1.20	1.26	1.31	-	-	-
Cumulative Recovered Au Ounces		-	9,190	27,860	55,720	83,580	111,440	139,840	168,240	196,640	225,040	253,220	281,400	309,580	337,760	365,870	391,730	441,420	472,240	-	-	-
Waste Volume			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Tonnes	20,926,100	126,500	256,900	383,400	383,400	383,400	983,200	983,200	983,200	983,200	1,713,500	1,713,500	1,713,500	1,713,500	2,066,600	1,901,200	3,542,000	1,095,900	-	-	-	-
Cumulative Waste Tonnes		126,500	383,400	766,800	1,150,200	1,533,600	2,516,800	3,500,000	4,483,200	5,466,400	7,179,900	8,893,400	10,606,900	12,320,400	14,387,000	16,288,200	19,830,200	20,926,100	-	-	-	-
Strip Ratio	1.87	0.52	0.52	0.52	0.52	0.52	1.35	1.35	1.35	1.35	2.35	2.35	2.35	2.35	2.84	2.84	4.86	3.39	-	-	-	-
Cumulative Strip Ratio		0.52	0.52	0.52	0.52	0.52	0.69	0.80	0.88	0.94	1.09	1.22	1.32	1.41	1.52	1.60	1.82	1.87	-	-	-	-
Capital Costs:																						
Capital Development/Pre-stripping																						
Capital Costs Relative to Option																						
Capital Cost - Mining	$ 24,499,502	$ 24,499,502																				
Capital Cost - Processing (Main)	$ 58,547,853	$ 58,547,853																				
Capital Cost - Processing (Addn CIL Circuit)	$ -																					
Capital Cost - Processing (Heap Leach)	$ -																					
Capital Cost - Transport	$ -	$ -																				
Capital Cost - Other	$ 60,205,555	$ 29,087,341	$ -	$ -	$ 11,356,641	$ 8,469,245	$ 5,646,164	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,823,082	$ 2,823,082	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Rehabilitation (Stage 1&2)	$ 7,955,750		$ -	$ -	$ -	$ -	$ -	$ -	$ 2,403,780	$ -	$ -	$ -	$ -	$ -	$ 3,166,970	$ -	$ 795,000	$ 795,000	$ 795,000	$ -	$ -	$ -
Capital Cost - Stage 3 - Process	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Land Acqisition	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - TSF & Other	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Rehabilitation	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Condemnation/Resource Drilling	$ 180,000	$ 180,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Annual Sustaining Capital	$ 4,777,880		$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ -	$ -	$ -	$ -
Total Capital Costs	$ 156,166,540	$ 112,314,696	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 6,288,669	$ 298,617	$ 1,093,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -
Cumulative Capital Costs		$ 112,314,696	$ 112,613,314	$ 112,911,931	$ 124,567,190	$ 133,335,053	$ 139,279,834	$ 139,578,451	$ 142,280,849	$ 142,579,466	$ 142,878,084	$ 143,176,701	$ 143,475,319	$ 146,597,018	$ 152,885,687	$ 153,184,305	$ 154,277,922	$ 155,371,540	$ 156,166,540	$ -	$ -	$ -
Operating Costs:																						
Mining - Waste	$ 33,880,177	$ 204,808	$ 415,931	$ 620,740	$ 620,740	$ 620,740	$ 1,591,839	$ 1,591,839	$ 1,591,839	$ 1,591,839	$ 2,774,224	$ 2,774,224	$ 2,774,224	$ 2,774,224	$ 3,345,907	$ 3,078,117	$ 5,734,637	$ 1,774,305	$ -	$ -	$ -	$ -
Mining - Ore	$ 21,229,027	$ 456,614	$ 926,863	$ 1,383,477	$ 1,383,477	$ 1,383,477	$ 1,382,340	$ 1,382,340	$ 1,382,340	$ 1,382,340	$ 1,383,098	$ 1,383,098	$ 1,383,098	$ 1,383,098	$ 1,376,280	$ 1,266,056	$ 1,379,310	$ 611,722	$ -	$ -	$ -	$ -
Labour (Mine Overhead)	$ 20,103,123	$ 410,268	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ -	$ -	$ -	$ -
General Costs	$ 599,367	$ 12,892	$ 26,168	$ 39,060	$ 39,060	$ 39,060	$ 39,028	$ 39,028	$ 39,028	$ 39,028	$ 39,050	$ 39,050	$ 39,050	$ 39,050	$ 38,857	$ 35,745	$ 38,943	$ 17,271	$ -	$ -	$ -	$ -
Engineering Costs	$ 1,013,578	$ 21,801	$ 44,253	$ 66,054	$ 66,054	$ 66,054	$ 66,000	$ 66,000	$ 66,000	$ 66,000	$ 66,036	$ 66,036	$ 66,036	$ 66,036	$ 65,710	$ 60,448	$ 65,855	$ 29,207	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Main)	$ 201,321,170	$ -	$ 4,310,855	$ 8,750,756	$ 13,061,610	$ 13,061,610	$ 13,061,610	$ 13,062,140	$ 13,062,140	$ 13,062,140	$ 13,062,140	$ 13,064,587	$ 13,064,587	$ 13,064,587	$ 13,064,587	$ 13,001,564	$ 11,960,319	$ 13,460,552	$ 6,145,386	$ -	$ -	$ -
Metallurgical/Processing Costs (Addn CIL)	$ -																					
Metallurgical/Processing Costs (Heap Leach)	$ -																					
Transport Cost to Central Plant/Port	$ 36,719,985	$ -	$ 789,807	$ 1,603,201	$ 2,393,008	$ 2,393,008	$ 2,393,008	$ 2,391,043	$ 2,391,043	$ 2,391,043	$ 2,391,043	$ 2,392,353	$ 2,392,353	$ 2,392,353	$ 2,392,353	$ 2,380,560	$ 2,189,906	$ 2,385,802	$ 1,058,100	$ -	$ -	$ -
General Overhead (BESRA)	$ 6,165,115	$ 132,605	$ 269,170	$ 401,775	$ 401,775	$ 401,775	$ 401,445	$ 401,445	$ 401,445	$ 401,445	$ 401,665	$ 401,665	$ 401,665	$ 401,665	$ 399,685	$ 367,675	$ 400,565	$ 177,650	$ -	$ -	$ -	$ -
Total Operating Costs	$ 321,031,543	$ 1,238,988	$ 8,013,851	$ 14,095,866	$ 19,196,528	$ 19,196,528	$ 20,166,075	$ 20,164,639	$ 20,164,639	$ 20,164,639	$ 21,348,058	$ 21,351,816	$ 21,351,816	$ 21,351,816	$ 21,914,182	$ 21,420,969	$ 23,000,338	$ 19,687,311	$ 7,203,486	$ -	$ -	$ -
Operating Cost per Tonne Ore	$ 28.64	$ 5.14	$ 16.37	$ 19.30	$ 26.28	$ 26.28	$ 27.63	$ 27.63	$ 27.63	$ 27.63	$ 29.23	$ 29.24	$ 29.24	$ 29.24	$ 30.16	$ 32.04	$ 31.58	$ 60.95	$ -	$ -	$ -	$ -
Cumulative Operating Costs		$ 1,238,988	$ 9,252,839	$ 23,348,705	$ 42,545,232	$ 61,741,760	$ 81,907,835	$ 102,072,474	$ 122,237,113	$ 142,401,751	$ 163,749,810	$ 185,101,625	$ 206,453,441	$ 227,805,257	$ 249,719,439	$ 271,140,408	$ 294,140,746	$ 313,828,057	$ 321,031,543	$ -	$ -	$ -
Total Costs:																						
Total Costs	$ 477,198,083	$ 113,553,684	$ 8,312,469	$ 14,394,483	$ 30,851,786	$ 27,964,391	$ 26,110,856	$ 20,463,256	$ 22,867,036	$ 20,463,256	$ 21,646,676	$ 21,650,433	$ 21,650,433	$ 24,473,515	$ 28,202,852	$ 21,719,586	$ 24,093,955	$ 20,780,929	$ 7,998,486	$ -	$ -	$ -
Total Cumulative Costs		$ 113,553,684	$ 121,866,153	$ 136,260,636	$ 167,112,422	$ 195,076,813	$ 221,187,669	$ 241,650,925	$ 264,517,961	$ 284,981,218	$ 306,627,893	$ 328,278,327	$ 349,928,760	$ 374,402,275	$ 402,605,126	$ 424,324,713	$ 448,418,668	$ 469,199,597	$ 477,198,083	$ -	$ -	$ -
Total Cost per Tonne Ore	$ 42.57	$ 470.98	$ 16.99	$ 19.70	$ 42.23	$ 38.28	$ 35.77	$ 28.04	$ 31.33	$ 28.04	$ 29.64	$ 29.65	$ 29.65	$ 33.51	$ 38.81	$ 32.49	$ 33.08	$ 64.34	$ -	$ -	$ -	$ -
Total Cost per Ounce	$ 1,010.50	$ -	$ 904.51	$ 771.00	$ 1,107.39	$ 1,003.75	$ 937.22	$ 720.54	$ 805.18	$ 720.54	$ 762.21	$ 768.29	$ 768.29	$ 868.47	$ 1,000.81	$ 772.66	$ 931.71	$ 418.21	$ 259.52	$ -	$ -	$ -
Total Cost per Tonne Ore (incl. Resale)	$ 39.83																					
Total Cost per Ounce (incl. Resale)	$ 945.52																					
Revenue:																						
Mine Call Factor	1.00	-	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	-	-	-
Capital Equipment Resale/Salvage	$ 12,330,106		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,330,106	$ -	$ -	$ -
Asset Resale	$ 18,354,167	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 18,354,167	$ -	$ -
Gold Revenue	$ 613,912,000	$ -	$ 11,947,000	$ 24,271,000	$ 36,218,000	$ 36,218,000	$ 36,218,000	$ 36,920,000	$ 36,920,000	$ 36,920,000	$ 36,920,000	$ 36,634,000	$ 36,634,000	$ 36,634,000	$ 36,634,000	$ 36,543,000	$ 33,618,000	$ 64,597,000	$ 40,066,000	$ -	$ -	$ -
Marketing & Royalties																						
Freight/Transport	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Refining	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Royalties	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Marketing Costs	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Revenue Before Tax	$ 644,596,273	$ -	$ 11,947,000	$ 24,271,000	$ 36,218,000	$ 36,218,000	$ 36,218,000	$ 36,920,000	$ 36,920,000	$ 36,920,000	$ 36,920,000	$ 36,634,000	$ 36,634,000	$ 36,634,000	$ 36,634,000	$ 36,543,000	$ 33,618,000	$ 64,597,000	$ 52,396,106	$ 18,354,167	$ -	$ -
Cumulative Revenue		$ -	$ 11,947,000	$ 36,218,000	$ 72,436,000	$ 108,654,000	$ 144,872,000	$ 181,792,000	$ 218,712,000	$ 255,632,000	$ 292,552,000	$ 329,186,000	$ 365,820,000	$ 402,454,000	$ 439,088,000	$ 475,631,000	$ 509,249,000	$ 573,846,000	$ 626,242,106	$ 644,596,273	$ -	$ -
Annual Revenue before Tax	$ 644,596,273	$ -				108,654,000				146,978,000				146,822,000				171,392,000				70,750,273
Annualised Cumulative Revenue		$ -				108,654,000				255,632,000				402,454,000				573,846,000				644,596,273
Quarterly Cash Flow	$ 167,398,190	($ 113,553,684)	$ 3,634,531	$ 9,876,517	$ 5,366,214	$ 8,253,609	$ 10,107,144	$ 16,456,744	$ 14,052,964	$ 16,456,744	$ 15,273,324	$ 14,983,567	$ 14,983,567	$ 12,160,485	$ 8,431,148	$ 14,823,414	$ 9,524,045	$ 43,816,071	$ 44,397,620	$ 18,354,167	$ -	$ -
Cumulative Cash Flow		($ 113,553,684)	($ 109,919,153)	($ 100,042,636)	($ 94,676,422)	($ 86,422,813)	($ 76,315,669)	($ 59,858,925)	($ 45,805,961)	($ 29,349,218)	($ 14,075,893)	$ 907,673	$ 15,891,240	$ 28,051,725	$ 36,482,874	$ 51,306,287	$ 60,830,332	$ 104,646,403	$ 149,044,023	$ 167,398,190	$ -	$ -
Annualised Cashflow	$ 167,398,190	($ 113,553,684)				27,130,871				57,073,595				57,400,943				76,594,678				62,751,787
Annualised Cumulative Cashflow		($ 113,553,684)				(86,422,813)				(29,349,218)				28,051,725				104,646,403				167,398,190
Yearly NPV @ 8%	$ 97,289,637																					
Yearly IRR	34.3%																					
Estimated Dates for Project (Assuming Start Date)		30/06/2014	30/09/2015	31/12/2015	31/03/2016	30/06/2016	30/09/2016	31/12/2016	31/03/2017	30/06/2017	30/09/2017	31/12/2017	31/03/2018	30/06/2018	30/09/2018	31/12/2018	31/03/2019	30/06/2019	30/09/2019	31/12/2019	31/03/2020	30/06/2020
Period NPV @ 8%	$ 94,912,428																					
Period IRR	27.4%																					

Table 22-2 - Cashflow Model - Option 452 (8,000tpd Owner-Operator)

The base case options summary results are shown in the following tables – *Table 22-3 - Key Summary Results from Cashflow Model - Option 484 (8,000tpd Contractor-Mining)* to *Table 22-4 - Key Summary Results from Cashflow Model - Option 452 (8,000tpd Owner-Operator).*

Key Summary Results			
Mined Ore Tonnes			10,928,000
Waste Tonnes			18,569,000
Gold Price		$	1,300.00
Strip Ratio			1.70
Total Recovered Ounces			463,700
Average Ounces/Annum			116,000
Recovery Percentage			0.77
Total Capital		$	134,878,000
Initial Capital		$	92,119,690
Stage 3 Capital		$	-
Ongoing Capital		$	42,758,310
Operating Cost/ Ore Tonne		$	31.39
Cost per Ounce		$	1,030.61
Cost per Ounce (incl. Resale)		$	973.14
Mine Life (Years)			4.00
Mine Life (Quarters)			16.00
Pre-Mine Period (Years)			1.00
Yearly NPV @	8%	$	91,407,216
Yearly IRR			38.0%

Table 22-3 - Key Summary Results from Cashflow Model - Option 484 (8,000tpd Contractor-Mining)

Key Summary Results			
Mined Ore Tonnes			11,210,000
Waste Tonnes			20,927,000
Gold Price		$	1,300.00
Strip Ratio			1.87
Total Recovered Ounces			472,300
Average Ounces/Annum			111,200
Recovery Percentage			0.77
Total Capital		$	156,167,000
Initial Capital		$	112,314,696
Stage 3 Capital		$	-
Ongoing Capital		$	43,852,304
Operating Cost/ Ore Tonne		$	28.64
Cost per Ounce		$	1,010.50
Cost per Ounce (incl. Resale)		$	945.52
Mine Life (Years)			4.25
Mine Life (Quarters)			17.00
Pre-Mine Period (Years)			1.00
Yearly NPV @	8%	$	97,289,637
Yearly IRR			34.3%

Table 22-4 - Key Summary Results from Cashflow Model - Option 452 (8,000tpd Owner-Operator)

The payback period is 2.25 years (9 quarters) from the start of production (excluding any pre-mining).

22.1.2. After Tax Basis

The taxes and royalties are outlined in detail in Section 21.4.2, and the key items are summarised below:

- Zero percent (0%) royalty on gold produced;
- No export duty or tariffs applicable to gold concentrate exports;
- Corporate income tax is 24 % of net taxable profits.

Import duties are applicable at a rate of bewtween 10-30 % for most standard goods; however, drilling and mining equipment are subject to nil import tariffs based on the individual item and related part numbers. These duties are included in the capital costs where applicable.

Two tax incentive schemes are available in Malaysia; these are Pioneer Status and Investment Tax Allowance (ITA).

Pioneer Status is a standard concession available under which companies can apply for this status which allows 70 % of the net income of the project to be tax free for the first five years. That can also be extended for a second five year term under certain circumstances.

Investment Tax Allowance (ITA) is a capital expenditure-based incentive which is given by way of an exemption of income. ITA is a 'once only' allowance which is given at the standard rate of 60% of qualifying capital expenditure for the basis period in which the capital expenditure is incurred. Eligibility lasts for five years from the date of approval. The allowance is used to exempt statutory income, with a limit of 70% on that income.

These tax incentives are not included in the cashflow model, but are currently being investigated to assess their impact on the project.

Building in the taxes and royalty rates listed at the beginning of this section the after tax cash flow model is determined. The after-tax cash flow model is shown in *Table 22-5 - After-Tax Cashflow Model - Option 484 (8,000tpd Contractor-Mining)* and *Table 22-6 - After-Tax Cashflow Model - Option 452 (8,000tpd Owner-Operator)* below, this first part of the cash flow models are as above and the figures below only show the subsequent after tax calculations. *Enclosure B22-1* lists the cost models for both base case scenario options for comparative purposes. These are at A3 size for ease of reading.

The key summary results, for each base option, are shown in *Table 22-7 - After-Tax Summary Results from Cashflow Model - Option 484 (8,000tpd Contractor-Mining)* and *Table 22-8 - After-Tax Summary Results from Cashflow Model - Option 452 (8,000tpd Owner-Operator)*.

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Stage		*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*2*	*2*	*0*	*0*	*0*	*0*	*0*
Tax - No Incentives																						
Opening Capital/Deferred Exploration		$ 7,500,000																				
Capital Expenditure (Stage 1&2)	*$ 134,877,915*	$ 92,119,690	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 3,121,699	$ 3,465,587	$ 298,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -	$ -
Capital Expenditure (Stage 3)	*$ -*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Non-Depreciable Capital	*$ 22,942,709*	$ 22,942,709	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Depreciable Capital		$ 99,619,690	$ 99,918,307	$ 93,972,030	$ 99,362,487	$ 101,033,029	$ 99,206,039	$ 91,237,486	$ 85,645,567	$ 77,379,628	$ 69,080,509	$ 60,744,063	$ 55,188,039	$ 49,111,731	$ 42,754,973	$ 32,364,847	$ 22,670,182	$ 12,130,091	$ -	$ -	$ -	$ -
Depreciation	142,377,915	$ -	$ 6,244,894	$ 6,264,802	$ 7,097,320	$ 7,771,771	$ 8,267,170	$ 8,294,317	$ 8,564,557	$ 8,597,736	$ 8,635,064	$ 8,677,723	$ 9,198,006	$ 9,822,346	$ 10,688,743	$ 10,788,282	$ 11,335,091	$ 12,130,091	$ -	$ -	$ -	$ -
Depreciation Quarters			16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-	-	-	-
Closing Depreciable Capital		**$ 99,619,690**	**$ 93,673,413**	**$ 87,707,228**	**$ 92,265,166**	**$ 93,261,258**	**$ 90,938,869**	**$ 82,943,169**	**$ 77,081,010**	**$ 68,781,891**	**$ 60,445,445**	**$ 52,066,339**	**$ 45,990,032**	**$ 39,289,385**	**$ 32,066,229**	**$ 21,576,565**	**$ 11,335,091**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**
Disposal of Plant & Equipment	*$ 8,291,104*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 8,291,104*	$ -	$ -	$ -	$ -
Tax Profit/(loss) from Disposal of P&E	*$ 8,291,104*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 8,291,104*	$ -	$ -	$ -	$ -
Proceeds from Disposal of Non-Depreciable Capital (land)	*$ 18,354,167*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 18,354,167*	$ -	$ -	$ -
Write-Off @ Project End			$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 4,588,542*	$ -	$ -
Balance of Non-Depreciable Capital (after disposal)			**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 4,588,542**	**$ -**	**$ -**	**$ -**
Debt Drawdowns	*$ 64,483,783*	$ 64,483,783																				
Interest Counter		1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Quarters Production		-	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	-	-	-	-
Principal Repayment Sweep %			0.00%	0.00%	0.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Principal Repayments	*($ 64,483,783)*		$ -	$ -	$ -	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Debt Outstanding		$ 64,483,783	$ 64,483,783	$ 64,483,783	$ 64,483,783	$ 58,035,404	$ 51,587,026	$ 45,138,648	$ 38,690,270	$ 32,241,891	$ 25,793,513	$ 19,345,135	$ 12,896,757	$ 6,448,378	$ 0	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Interest	*($ 13,283,659)*	($ 3,611,092)	($ 1,289,676)	($ 1,289,676)	($ 1,289,676)	($ 1,160,708)	($ 1,031,741)	($ 902,773)	($ 773,805)	($ 644,838)	($ 515,870)	($ 386,903)	($ 257,935)	($ 128,968)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
EBITDA	*$ 259,780,175*	($ 1,645,278)	$ 3,111,839	$ 8,923,973	$ 15,774,390	$ 15,774,390	$ 14,238,151	$ 15,262,151	$ 15,262,151	$ 15,262,151	$ 13,282,293	$ 12,896,254	$ 12,896,254	$ 12,896,254	$ 12,455,669	$ 11,705,987	$ 22,454,862	$ 59,228,686	$ -	$ -	$ -	$ -
Depreciation	*($ 142,377,915)*	$ -	($ 6,244,894)	($ 6,264,802)	($ 7,097,320)	($ 7,771,771)	($ 8,267,170)	($ 8,294,317)	($ 8,564,557)	($ 8,597,736)	($ 8,635,064)	($ 8,677,723)	($ 9,198,006)	($ 9,822,346)	($ 10,688,743)	($ 10,788,282)	($ 11,335,091)	($ 12,130,091)	$ -	$ -	$ -	$ -
Interest	*($ 13,283,659)*	($ 3,611,092)	($ 1,289,676)	($ 1,289,676)	($ 1,289,676)	($ 1,160,708)	($ 1,031,741)	($ 902,773)	($ 773,805)	($ 644,838)	($ 515,870)	($ 386,903)	($ 257,935)	($ 128,968)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of P&E	*$ 8,291,104*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8,291,104	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of Land	*$ -*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Earnings	*$ 112,409,705*	($ 5,256,370)	($ 4,422,731)	$ 1,369,495	$ 7,387,394	$ 6,841,910	$ 4,939,241	$ 6,065,061	$ 5,923,789	$ 6,019,577	$ 4,131,359	$ 3,831,628	$ 3,440,312	$ 2,944,940	$ 1,766,926	$ 917,704	$ 11,119,771	$ 55,389,699	$ -	$ -	$ -	$ -
Carry Forward Losses (if applicable)		($ 5,256,370)	($ 9,679,101)	($ 8,309,606)	($ 922,212)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Adjusted Taxable Earnings		**$ -**	**$ -**	**$ -**	**$ -**	**$ 5,919,698**	**$ 4,939,241**	**$ 6,065,061**	**$ 5,923,789**	**$ 6,019,577**	**$ 4,131,359**	**$ 3,831,628**	**$ 3,440,312**	**$ 2,944,940**	**$ 1,766,926**	**$ 917,704**	**$ 11,119,771**	**$ 55,389,699**	**$ -**	**$ -**	**$ -**	**$ -**
Tax with No Incentives	*($ 26,978,329)*	$ -	$ -	$ -	$ -	($ 1,420,728)	($ 1,185,418)	($ 1,455,615)	($ 1,421,709)	($ 1,444,698)	($ 991,526)	($ 919,591)	($ 825,675)	($ 706,786)	($ 424,062)	($ 220,249)	($ 2,668,745)	($ 13,293,528)	$ -	$ -	$ -	$ -
After Tax Cashflow	*111,285,543*	*($ 32,892,277)*	*$ 1,523,546*	*$ 7,335,680*	*$ 2,829,455*	*($ 2,023,287)*	*($ 372,167)*	*$ 6,156,768*	*$ 3,915,861*	*$ 6,425,619*	*$ 5,027,901*	*$ 4,842,765*	*$ 2,242,566*	*$ 2,490,423*	*$ 2,117,641*	*$ 11,187,120*	*$ 18,692,499*	*$ 53,431,263*	*$ 18,354,167*	*$ -*	*$ -*	*$ -*
Cumulative After Tax Cashflow		*($ 32,892,277)*	*($ 31,368,731)*	*($ 24,033,051)*	*($ 21,203,596)*	*($ 23,226,883)*	*($ 23,599,049)*	*($ 17,442,282)*	*($ 13,526,421)*	*($ 7,100,802)*	*($ 2,072,901)*	*$ 2,769,864*	*$ 5,012,430*	*$ 7,502,853*	*$ 9,620,494*	*$ 20,807,614*	*$ 39,500,113*	*$ 92,931,376*	*$ 111,285,543*	*$ -*	*$ -*	*$ -*
Annualised Cashflow	*111,285,543*	*($ 32,892,277)*	*$*			*9,665,394*	*$*			*16,126,081*	*$*			*14,603,655*	*$*			*85,428,523*	*$*			*18,354,167*
		-$ 93,764,968				*$ 21,143,507*				*$ 45,272,751*				*$ 41,686,844*				*$ 91,876,901*				*$ 18,354,167*
Yearly NPV: Tax No Incentive	$ 71,983,893																					
8%																						
Yearly IRR: Tax No Incentive	32.6%																					

Table 22-5 - After-Tax Cashflow Model - Option 484 (8,000tpd Contractor-Mining)

Cashflow Item	Totals	Yr -1 Pre-Mining	Yr 1 Qtr1	Yr 1 Qtr2	Yr 1 Qtr3	Yr 1 Qtr4	Yr 2 Qtr1	Yr 2 Qtr2	Yr 2 Qtr3	Yr 2 Qtr4
Stage		*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*
Tax - No Incentives										
Opening Capital/Deferred Exploration		$ 7,500,000								
Capital Expenditure (Stage 1&2)	*$ 156,166,540*	$ 112,314,696	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617
Capital Expenditure (Stage 3)	*$ -*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Non-Depreciable Capital	*$ 22,942,709*	$ 22,942,709	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Depreciable Capital		$ 119,814,696	$ 120,113,314	$ 113,346,442	$ 117,917,548	$ 118,824,241	$ 116,281,576	$ 107,635,457	$ 101,368,233	$ 92,451,557
Depreciation	163,666,540	$ -	$ 7,065,489	$ 7,084,153	$ 7,861,170	$ 8,487,446	$ 8,944,737	$ 8,969,621	$ 9,215,294	$ 9,245,156
Depreciation Quarters			17	16	15	14	13	12	11	10
Closing Depreciable Capital		**$ 119,814,696**	**$ 113,047,825**	**$ 106,262,290**	**$ 110,056,378**	**$ 110,336,795**	**$ 107,336,840**	**$ 98,665,836**	**$ 92,152,939**	**$ 83,206,401**
Disposal of Plant & Equipment	*$ 12,330,106*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Tax Profit/(loss) from Disposal of P&E	*$ 12,330,106*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Proceeds from Disposal of Non-Depreciable Capital (land)	*$ 18,354,167*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Write-Off @ Project End			$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Balance of Non-Depreciable Capital (after disposal)			**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**
Debt Drawdowns	*$ 78,620,288*	$ 78,620,288								
Interest Counter		1	4	4	4	4	4	4	4	4
Quarters Production		-	1	2	3	4	5	6	7	8
Principal Repayment Sweep %			0.00%	0.00%	0.00%	9.09%	9.09%	9.09%	9.09%	9.09%
Principal Repayments	*($ 78,620,288)*		$ -	$ -	$ -	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)
Debt Outstanding		$ 78,620,288	$ 78,620,288	$ 78,620,288	$ 78,620,288	$ 71,472,989	$ 64,325,690	$ 57,178,391	$ 50,031,092	$ 42,883,793
Interest	*($ 16,981,982)*	($ 4,402,736)	($ 1,572,406)	($ 1,572,406)	($ 1,572,406)	($ 1,429,460)	($ 1,286,514)	($ 1,143,568)	($ 1,000,622)	($ 857,676)
EBITDA	*$ 292,880,457*	($ 1,238,988)	$ 3,933,149	$ 10,175,134	$ 17,021,472	$ 17,021,472	$ 16,051,925	$ 16,755,361	$ 16,755,361	$ 16,755,361
Depreciation	*($ 163,666,540)*	$ -	($ 7,065,489)	($ 7,084,153)	($ 7,861,170)	($ 8,487,446)	($ 8,944,737)	($ 8,969,621)	($ 9,215,294)	($ 9,245,156)
Interest	*($ 16,981,982)*	($ 4,402,736)	($ 1,572,406)	($ 1,572,406)	($ 1,572,406)	($ 1,429,460)	($ 1,286,514)	($ 1,143,568)	($ 1,000,622)	($ 857,676)
Taxable Profits/(Losses) on Disposal of P&E	*$ 12,330,106*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of Land	*$ -*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Earnings	*$ 124,562,041*	($ 5,641,724)	($ 4,704,746)	$ 1,518,576	$ 7,587,897	$ 7,104,567	$ 5,820,675	$ 6,642,172	$ 6,539,445	$ 6,652,530
Carry Forward Losses (if applicable)		($ 5,641,724)	($ 10,346,470)	($ 8,827,894)	($ 1,239,997)	$ -	$ -	$ -	$ -	$ -
Adjusted Taxable Earnings		**$ -**	**$ -**	**$ -**	**$ -**	**$ 5,864,569**	**$ 5,820,675**	**$ 6,642,172**	**$ 6,539,445**	**$ 6,652,530**
Tax with No Incentives	*($ 29,894,890)*	$ -	$ -	$ -	$ -	($ 1,407,497)	($ 1,396,962)	($ 1,594,121)	($ 1,569,467)	($ 1,596,607)
After Tax Cashflow	*120,521,318*	*($ 39,336,133)*	*$ 2,062,126*	*$ 8,304,111*	*$ 3,793,808*	*($ 1,730,646)*	*$ 276,370*	*$ 6,571,756*	*$ 4,335,576*	*$ 6,855,162*
Cumulative After Tax Cashflow		*($ 39,336,133)*	*($ 37,274,007)*	*($ 28,969,896)*	*($ 25,176,088)*	*($ 26,906,734)*	*($ 26,630,365)*	*($ 20,058,609)*	*($ 15,723,033)*	*($ 8,867,871)*
Annualised Cashflow	*120,521,318*	*($ 39,336,133)*	*$*			*12,429,399*	*$*			*18,038,863*
		-$ 113,553,684				*$ 25,723,374*				*$ 50,916,438*
Yearly NPV: Tax No Incentive	$ 76,106,036									
8%										
Yearly IRR: Tax No Incentive	29.4%									

Cashflow Item	Yr 3 Qtr1	Yr 3 Qtr2	Yr 3 Qtr3	Yr 3 Qtr4	Yr 4 Qtr1	Yr 4 Qtr2	Yr 4 Qtr3	Yr 4 Qtr4	Yr 5 Qtr1	Yr 5 Qtr2	Yr 5 Qtr3	Yr 5 Qtr4
Stage	*1*	*1*	*1*	*1*	*1*	*2*	*2*	*2*	*0*	*0*	*0*	*0*
Tax - No Incentives												
Opening Capital/Deferred Exploration												
Capital Expenditure (Stage 1&2)	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 6,288,669	$ 298,617	$ 1,093,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -
Capital Expenditure (Stage 3)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Non-Depreciable Capital	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Depreciable Capital	$ 83,505,019	$ 74,525,301	$ 65,508,256	$ 59,271,633	$ 55,681,696	$ 44,843,975	$ 34,726,598	$ 24,244,683	$ 12,917,342	$ -	$ -	$ -
Depreciation	$ 9,278,335	$ 9,315,663	$ 9,358,322	$ 9,878,605	$ 11,136,339	$ 11,210,994	$ 11,575,533	$ 12,122,342	$ 12,917,342	$ -	$ -	$ -
Depreciation Quarters	9	8	7	6	5	4	3	2	1	-	-	-
Closing Depreciable Capital	**$ 74,226,683**	**$ 65,209,638**	**$ 56,149,933**	**$ 49,393,027**	**$ 44,545,357**	**$ 33,632,981**	**$ 23,151,066**	**$ 12,122,342**	**$ -**	**$ -**	**$ -**	**$ -**
Disposal of Plant & Equipment	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 12,330,106*	$ -	$ -	$ -
Tax Profit/(loss) from Disposal of P&E	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 12,330,106*	$ -	$ -	$ -
Proceeds from Disposal of Non-Depreciable Capital (land)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 18,354,167*	$ -	$ -
Write-Off @ Project End	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	*$ 4,588,542*	$ -
Balance of Non-Depreciable Capital (after disposal)	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 4,588,542**	**$ -**	**$ -**
Debt Drawdowns												
Interest Counter	4	4	4	4	4	4	4	4	4	4	4	4
Quarters Production	9	10	11	12	13	14	15	16	17	-	-	-
Principal Repayment Sweep %	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Principal Repayments	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	$ -	$ -	$ -	$ -	$ -	$ -
Debt Outstanding	$ 35,736,494	$ 28,589,195	$ 21,441,897	$ 14,294,598	$ 7,147,299	$ 0	$ -	$ -	$ -	$ -	$ -	$ -
Interest	($ 714,730)	($ 571,784)	($ 428,838)	($ 285,892)	($ 142,946)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -
EBITDA	$ 15,571,942	$ 15,282,184	$ 15,282,184	$ 15,282,184	$ 14,719,818	$ 15,122,031	$ 10,617,662	$ 44,909,689	$ 32,862,514	$ -	$ -	$ -
Depreciation	($ 9,278,335)	($ 9,315,663)	($ 9,358,322)	($ 9,878,605)	($ 11,136,339)	($ 11,210,994)	($ 11,575,533)	($ 12,122,342)	($ 12,917,342)	$ -	$ -	$ -
Interest	($ 714,730)	($ 571,784)	($ 428,838)	($ 285,892)	($ 142,946)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of P&E	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,330,106	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of Land	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Earnings	$ 5,578,876	$ 5,394,738	$ 5,495,024	$ 5,117,687	$ 3,440,532	$ 3,911,038	($ 957,871)	$ 32,787,347	$ 32,275,278	$ -	$ -	$ -
Carry Forward Losses (if applicable)	$ -	$ -	$ -	$ -	$ -	$ -	($ 957,871)	$ -	$ -	$ -	$ -	$ -
Adjusted Taxable Earnings	**$ 5,578,876**	**$ 5,394,738**	**$ 5,495,024**	**$ 5,117,687**	**$ 3,440,532**	**$ 3,911,038**	**$ -**	**$ 31,829,476**	**$ 32,275,278**	**$ -**	**$ -**	**$ -**
Tax with No Incentives	($ 1,338,930)	($ 1,294,737)	($ 1,318,806)	($ 1,228,245)	($ 825,728)	($ 938,649)	$ -	($ 7,639,074)	($ 7,746,067)	$ -	$ -	$ -
After Tax Cashflow	*$ 6,072,365*	*$ 5,969,747*	*$ 6,088,624*	*$ 3,499,049*	*$ 315,176*	*$ 6,737,466*	*$ 9,524,045*	*$ 36,176,997*	*$ 36,651,553*	*$ 18,354,167*	*$ -*	*$ -*
Cumulative After Tax Cashflow	*($ 2,795,506)*	*$ 3,174,241*	*$ 9,262,866*	*$ 12,761,915*	*$ 13,077,091*	*$ 19,814,557*	*$ 29,338,601*	*$ 65,515,598*	*$ 102,167,151*	*$ 120,521,318*	*$ -*	*$ -*
Annualised Cashflow	*$*			*21,629,786*	*$*			*52,753,683*	*$*			*55,005,720*
				$ 52,220,225				*$ 67,191,227*				*$ 55,005,720*

Table 22-6 - After-Tax Cashflow Model - Option 452 (8,000tpd Owner-Operator)

Comparison of Pre-Tax, After Tax & After Tax Allowances			
Pre-Tax:			
Yearly NPV @	8%	**$**	**91,407,216**
Yearly IRR			**38.0%**
After Tax:			
Yearly NPV: Tax No Incentive	8%	**$**	**71,983,893**
Yearly IRR: Tax No Incentive			**32.6%**
After ITA Allowance:			
Yearly NPV: ITA Allowance	8%	**$**	**-**
Yearly IRR: ITA Allowance			**N/A**
After Pioneer Status:			
Yearly NPV: Pioneer Status	8%	**$**	**-**
Yearly IRR: Pioneer Status			**N/A**

Table 22-7 - After-Tax Summary Results from Cashflow Model - Option 484 (8,000tpd Contractor-Mining)

Comparison of Pre-Tax, After Tax & After Tax Allowances			
Pre-Tax:			
Yearly NPV @	8%	**$**	**97,289,637**
Yearly IRR			**34.3%**
After Tax:			
Yearly NPV: Tax No Incentive	8%	**$**	**76,106,036**
Yearly IRR: Tax No Incentive			**29.4%**
After ITA Allowance:			
Yearly NPV: ITA Allowance	8%	**$**	**-**
Yearly IRR: ITA Allowance			**N/A**
After Pioneer Status:			
Yearly NPV: Pioneer Status	8%	**$**	**-**
Yearly IRR: Pioneer Status			**N/A**

Table 22-8 - After-Tax Summary Results from Cashflow Model - Option 452 (8,000tpd Owner-Operator)

22.2. Sensitivity Analysis

A sensitivity analysis has been performed on some of the key factors that will or may have a financial impact on the project performance. The following financial and non-financial elements were assessed along with their variability values and the results are presented in the tables and graphs shown below in the list.

- Gold price: $1,100 to $2,000 ($100 increments)

Gold Price	Contract-Mining		Owner-Operator	
	NPV	IRR	NPV	IRR
$1,100	$ 21,529,886	15.3%	$ 26,575,389	15.4%
$1,200	$ 56,468,551	26.9%	$ 61,932,513	25.0%
$1,300	$ 91,407,216	38.0%	$ 97,289,637	34.3%
$1,400	$ 126,345,880	48.8%	$ 132,646,761	43.3%
$1,500	$ 161,284,545	59.3%	$ 168,003,885	52.2%
$1,600	$ 196,223,210	69.6%	$ 203,361,009	60.9%
$1,700	$ 231,161,874	79.7%	$ 238,718,133	69.4%
$1,800	$ 266,100,539	89.7%	$ 274,075,257	77.8%
$1,900	$ 301,039,204	99.6%	$ 309,432,381	86.2%
$2,000	$ 335,977,869	109.3%	$ 344,789,505	94.4%

Table 22-9 – NPV & IRR – Gold Price Sensitivity

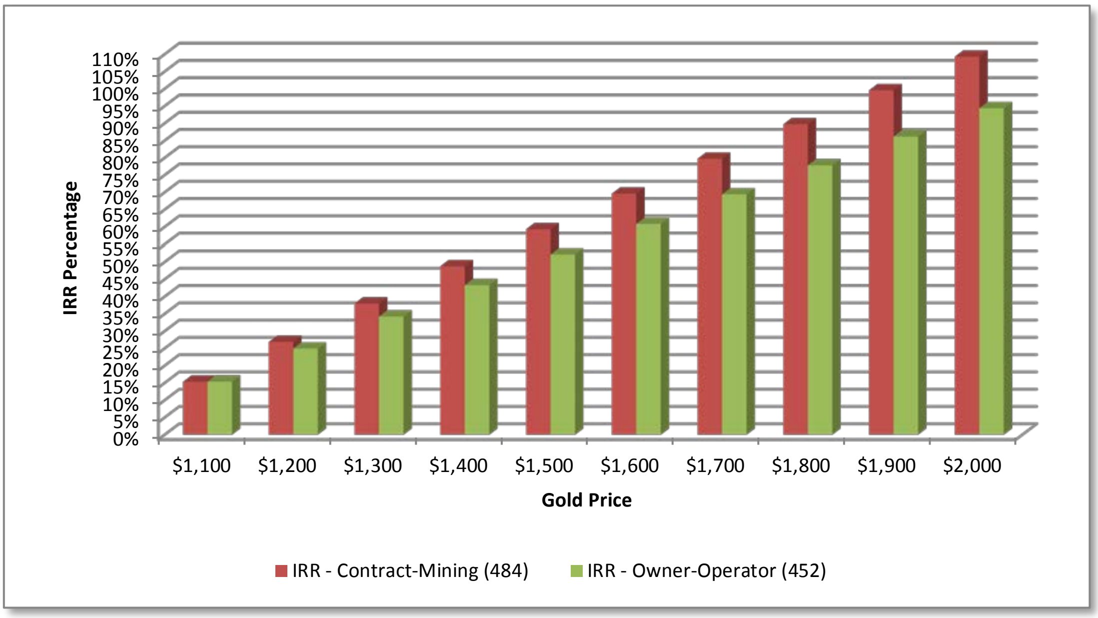


Figure 22-2 - Graph of IRR - Gold Price Sensitivity

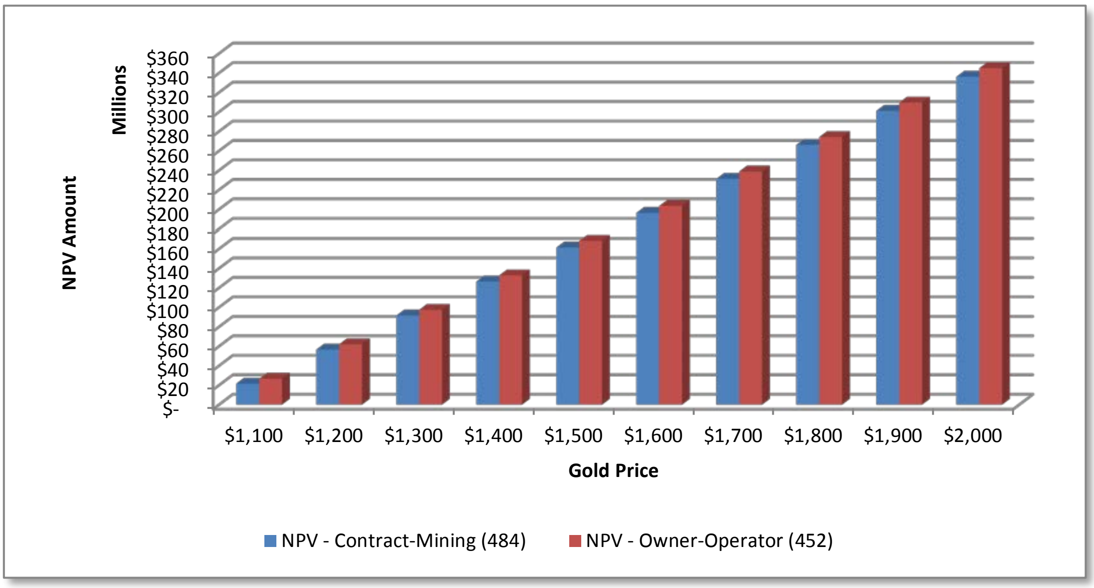


Figure 22-3 - Graph of MPV - Gold Price Sensitivity

- Mining costs: -20%, -10, 0%, 10%, 20% difference in mining costs

Mining Cost	Contract-Mining		Owner-Operator	
	NPV	IRR	NPV	IRR
-20%	$ 104,092,520	41.9%	$ 105,533,935	36.4%
-10%	$ 97,749,868	40.0%	$ 101,411,786	35.3%
0%	$ 91,407,216	38.0%	$ 97,289,637	34.3%
10%	$ 85,064,563	36.0%	$ 93,167,488	33.2%
20%	$ 78,721,911	34.0%	$ 89,045,338	32.1%

Table 22-10 - NPV & IRR – Mining Cost Sensitivity

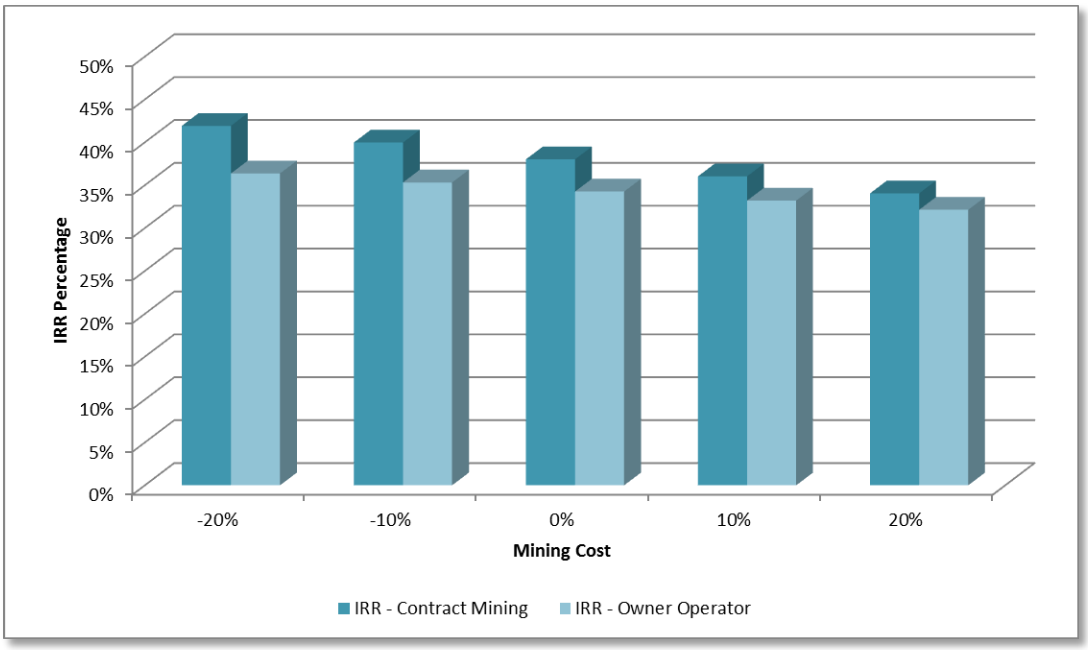


Figure 22-4 - Graph of IRR - Mining Cost Sensitivity

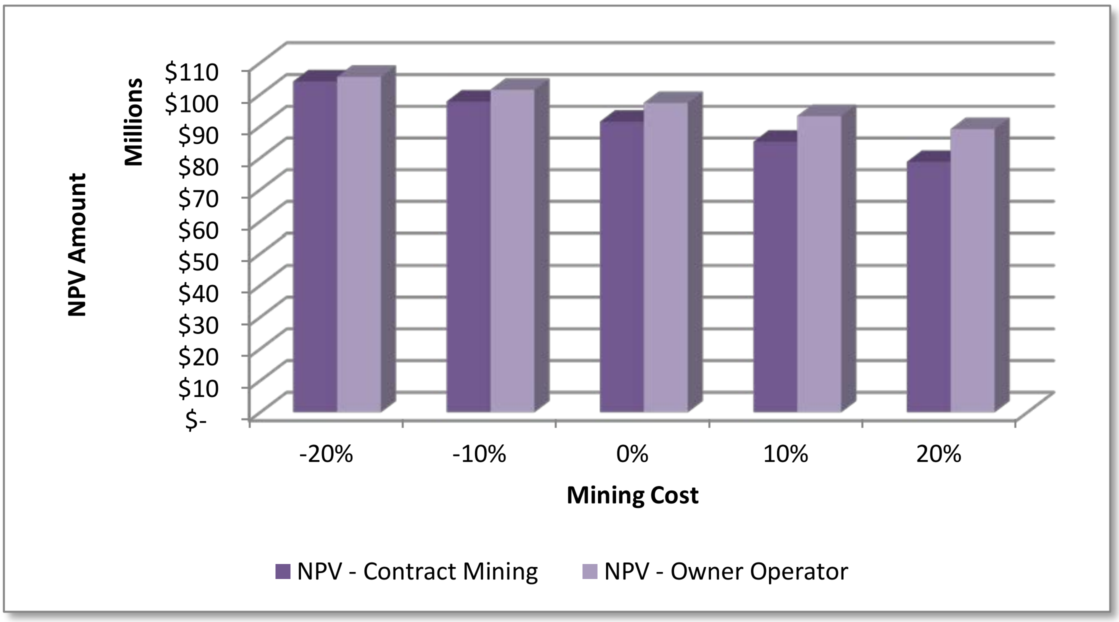


Figure 22-5 - Graph of NPV - Mining Cost Sensitivity

- Processing costs: -20%, -10, 0%, 10%, 20% difference in processing costs

Process Cost	Contract-Mining		Owner-Operator	
	NPV	IRR	NPV	IRR
-20%	$ 104,281,395	42.1%	$ 110,436,077	37.8%
-10%	$ 97,844,305	40.1%	$ 103,862,857	36.0%
0%	$ 91,407,216	38.0%	$ 97,289,637	34.3%
10%	$ 84,970,126	35.9%	$ 90,716,416	32.5%
20%	$ 78,533,036	33.8%	$ 84,143,196	30.7%

Table 22-11 - NPV & IRR – Process Cost Sensitivity

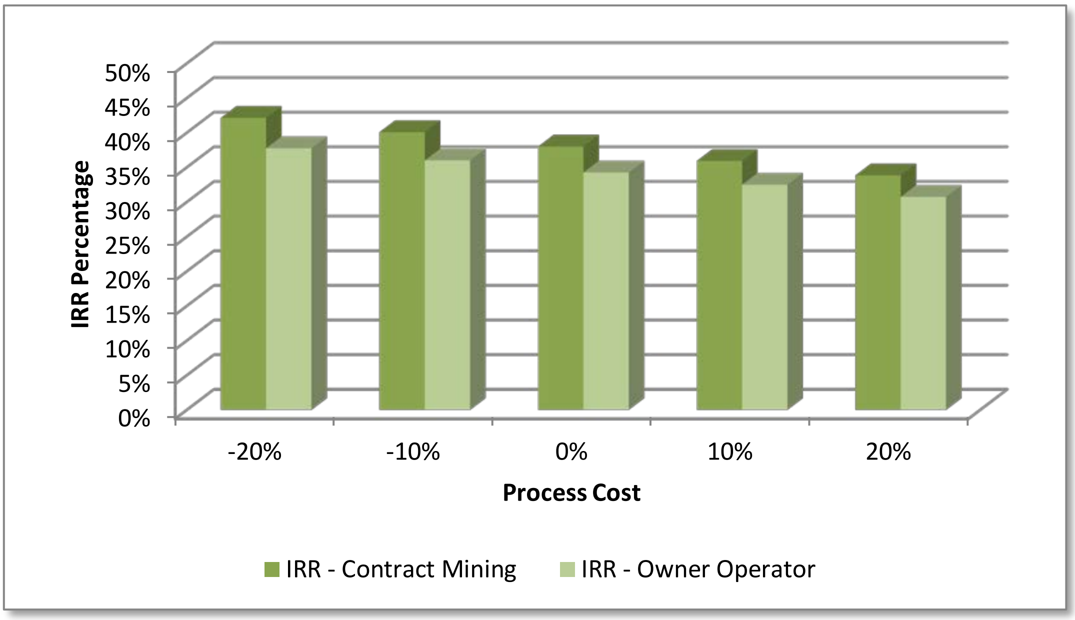


Figure 22-6 - Graph of IRR - Process Cost Sensitivity

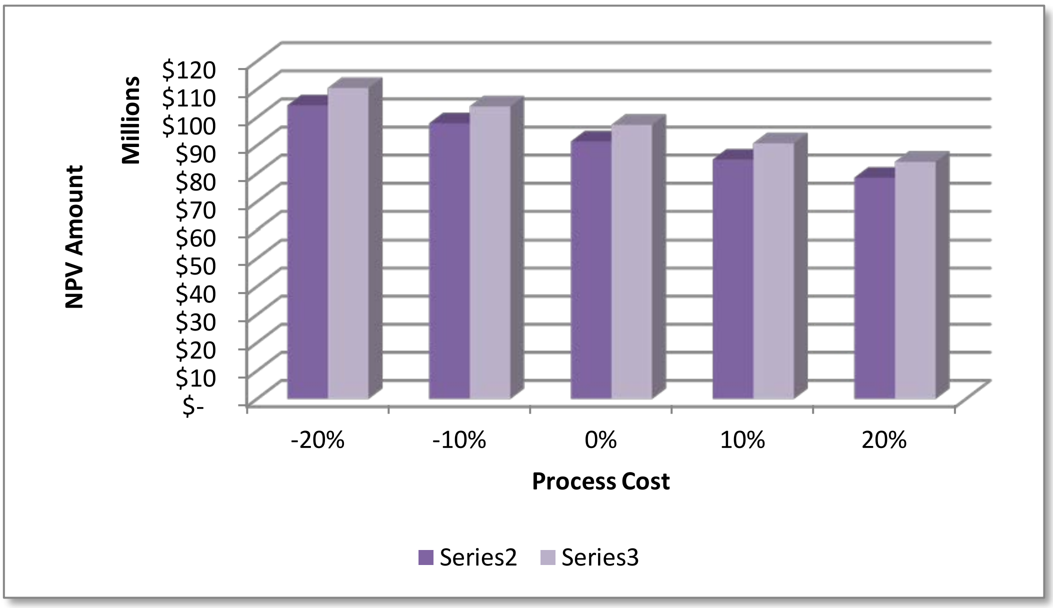


Figure 22-7 - Graph of NPV - Mining Cost Sensitivity

- Capital costs: -20%, -10, 0%, 10%, 20% difference in capital costs

Capital Cost	Contract-Mining		Owner-Operator	
	NPV	IRR	NPV	IRR
-20%	$ 111,611,728	51.1%	$ 121,203,223	46.6%
-10%	$ 101,509,472	44.0%	$ 109,246,430	39.9%
0%	$ 91,407,216	38.0%	$ 97,289,637	34.3%
10%	$ 81,304,959	32.9%	$ 85,332,843	29.5%
20%	$ 71,202,703	28.5%	$ 73,376,050	25.3%

Table 22-12 - NPV & IRR – Capital Cost Sensitivity

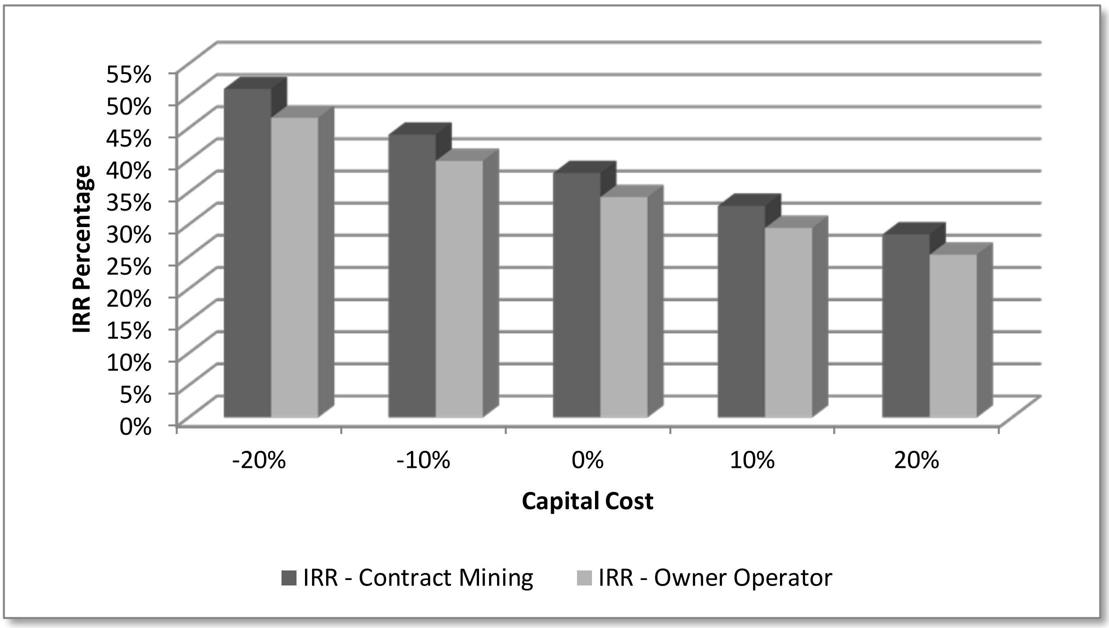


Figure 22-8 - Graph of IRR - Capital Cost Sensitivity

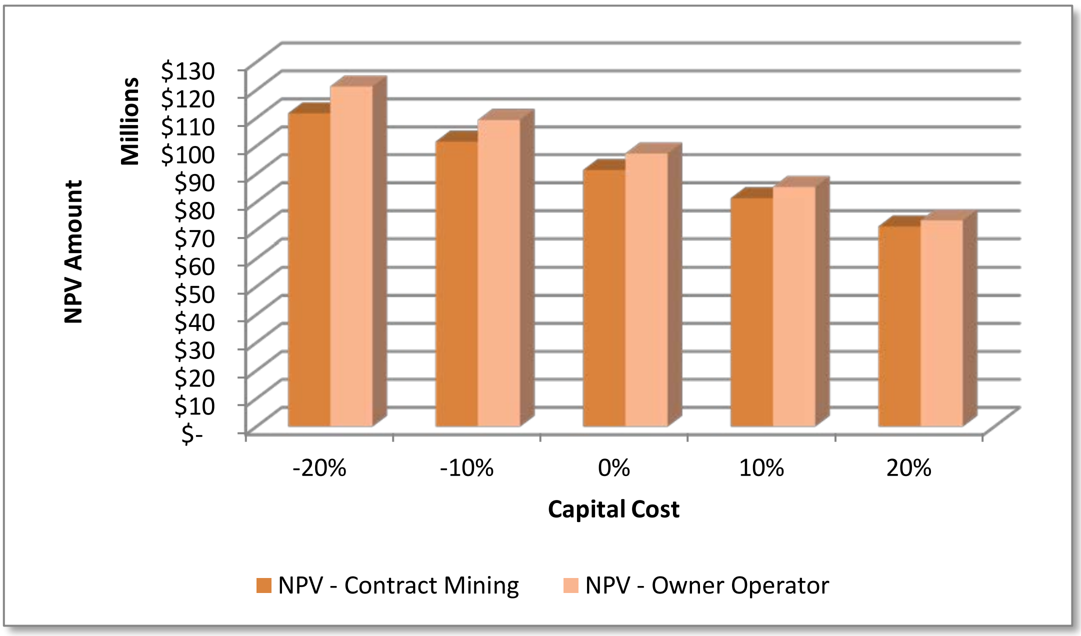


Figure 22-9 - Graph of NPV - Capital Cost Sensitivity

- Grade: -20%, -10, 0%, 10%, 20% difference in average grade

Grade	Contract-Mining		Owner-Operator	
	NPV	IRR	NPV	IRR
-20%	$ 1,964,234	8.7%	$ 6,775,399	9.9%
-10%	$ 46,685,725	23.7%	$ 52,032,518	22.3%
0%	$ 91,407,216	38.0%	$ 97,289,637	34.3%
10%	$ 136,128,706	51.8%	$ 142,546,755	45.8%
20%	$ 180,850,197	65.1%	$ 187,803,874	57.1%

Table 22-13 - NPV & IRR – Grade Sensitivity

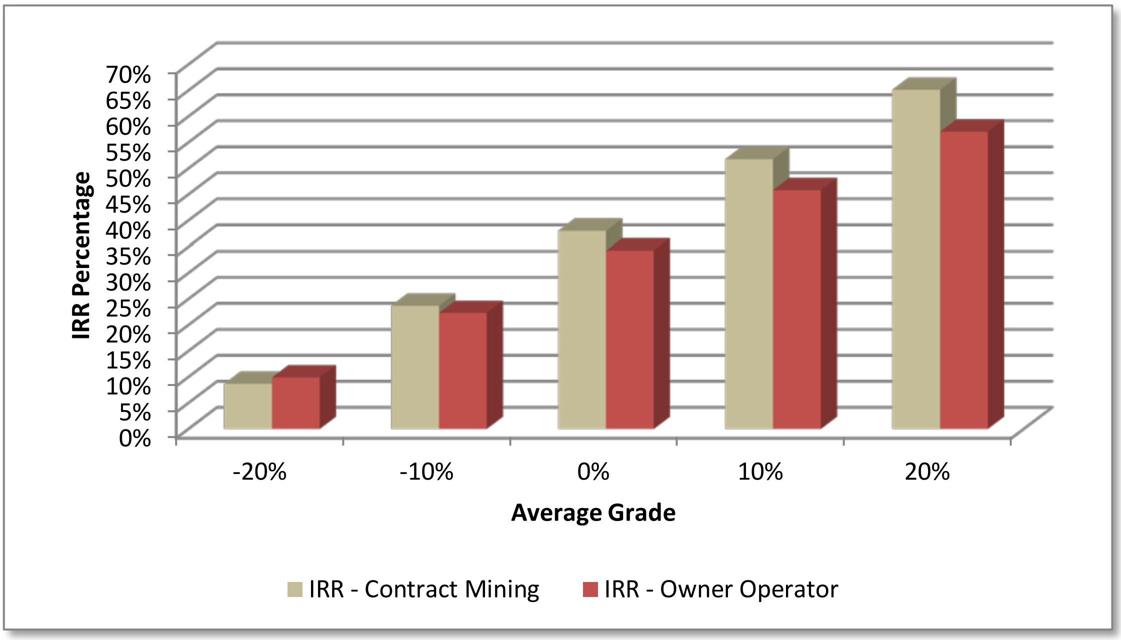


Figure 22-10 - Graph of IRR - Grade Sensitivity

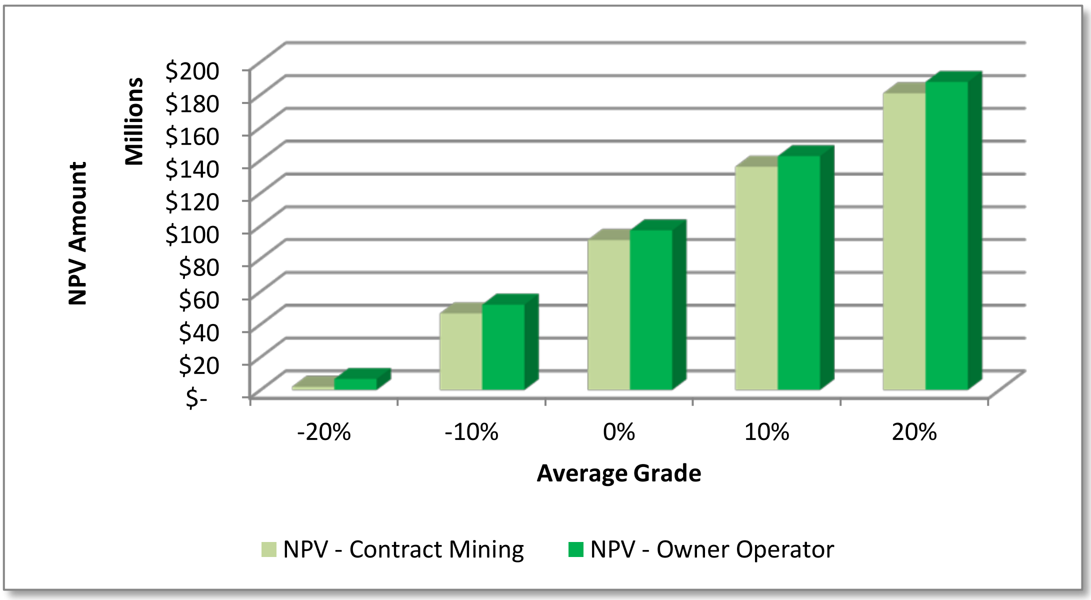


Figure 22-11 - Graph of NPV - Grade Sensitivity

- Process Recovery: -20%, -10, 0%, 10%, 20% difference in overall recovery around the current 72%:

Recovery	Contract-Mining		Owner-Operator	
	NPV	IRR	NPV	IRR
-20%	$ 1,952,448	8.7%	$ 6,763,236	9.9%
-10%	$ 46,694,139	23.7%	$ 52,014,738	22.3%
0%	$ 91,407,216	38.0%	$ 97,289,637	34.3%
10%	$ 136,166,330	51.8%	$ 142,538,230	45.8%
20%	$ 180,853,444	65.1%	$ 187,813,130	57.1%

Table 22-14 - NPV & IRR – Recovery Sensitivity

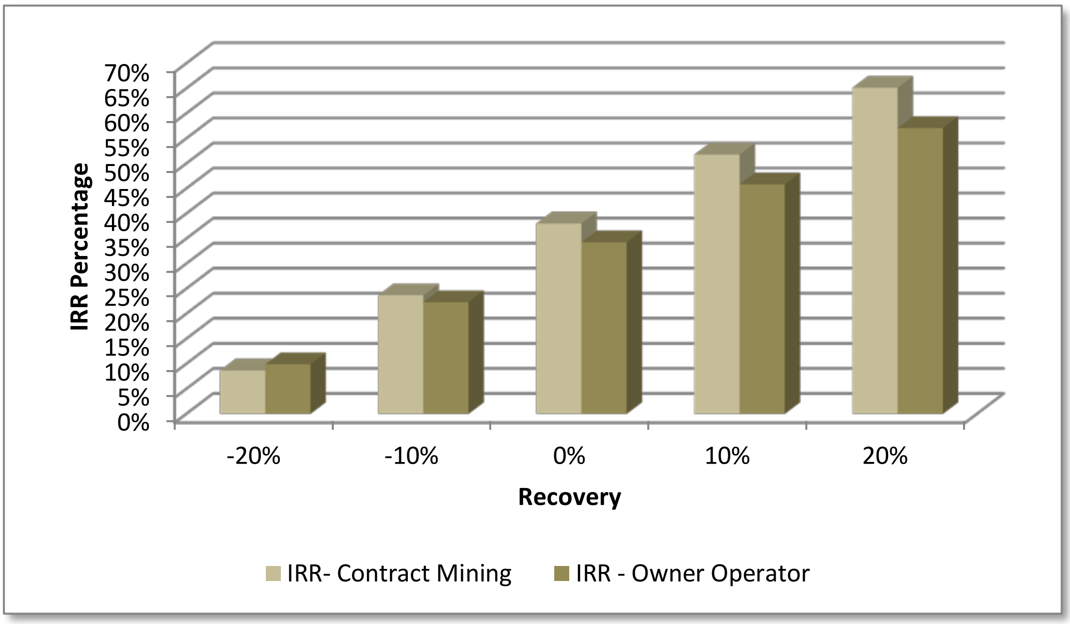


Figure 22-12 - Graph of IRR - Recovery Sensitivity

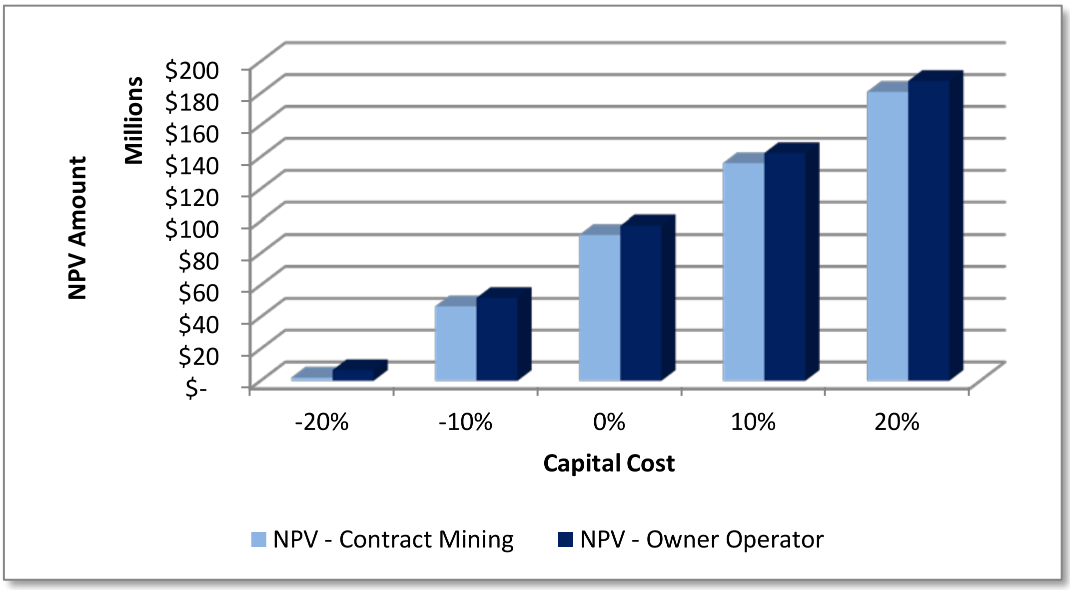


Figure 22-13 - Graph of NPV - Recovery Sensitivity

Some key effects can be seen in the above tables and graphs, and these are:

- Although mining and process cost variations, and in particular in the positive direction, show drops in IRR/NPV but these do not go negative within the variations tested, showing a little less sensitivity.
- Similarly the capital cost variation, shows similar trend although the +20% takes the project negative.
- Other than gold price the grade and recovery analysis shows much more sensitivity to the factors.
- Although the sensitivities show the negative impact on higher costs and lower grade, recovery and gold price they also show a large upside for small increments.
- These sensitivities have been reviewed individually. Combined they have a compounded impact.
- The negative impacts are of lower value than the equivalent positive impact.

22.3. Risk Analysis

22.3.1. Introduction & Methodology

Various key areas of the project have been examined and reviewed in terms of their risk profile. The risk identification and documentation of these critical elements is identified below, along with their potential impact, probability, manageability and any mitigation measures.

Each risk within each Risk Group is identified, along with which area it impacts (e.g. schedule) and it's possible consequence or possible impact(s). Thereafter the probability of occurrence is assigned as High, Medium or Low. Then the mitigation level is assessed in terms of the impact or effect of the mitigation and this is also qualitatively assessed as High, Medium or Low.

Associated with the mitigation level is the potential mitigation measure(s) that can or may be applied to the identified risk. Also, any related comments are also added if any.

The "probability of occurrence" levels are associated with a probability description and these are listed in *Table 22-15 - Risk Anaysis: Probability of Occurrence Levels* below.

Assigned Level	Probability Description
HIGH	Likely to happen or high probability of occurrence
MEDIUM	May happen or moderate probability of occurrence
LOW	Unlikely to happen or low probability of occurrence

Table 22-15 - Risk Anaysis: Probability of Occurrence Levels

The "consequence levels" are associated with the potential impact or consequence that the risk might have, and these are listed below in *Table 22-16 - Risk Anaysis: Consequence Levels*.

Assigned Level	Consequence Description
HIGH	High impact or major consequence
MEDIUM	Medium level impact or moderate consequence,
LOW	Low level of impact or little/minor consequence

Table 22-16 - Risk Anaysis: Consequence Levels

Therefore, any risk with a HIGH "probability of occurrence" and a HIGH "consequence level" has the greatest risk impact on the project. Conversely, the opposite combination has the lowest impact.

22.3.2. Project Risk Analysis

Table 22-17 - Project Risk Register (Part 1) and *Table 22-18 - Project Risk Register (Part 2)* below, presents a summary of the project risks identified to date, and the risk categories and mitigation measures associated with that risk. This is a live document and should be visited regularly during the project and updated as progress is made. Table is also shown in *Enclosure B22-2* at A3 size.

A risk matrix has also been developed to display the relationship between the "probability" and "consequences". The key risks are displayed in the top right of the matrix (above the brown line and shaded area). This matrix is shown in *Table 22-19 - Consequence vs. Probability Risk Matrix* following the risk register lists.

No.	Risk Group	Risk Description	Description of Consequence or Impact	Probability	Consequence	Score	Estimated Cost Impact	Mitigation Measures	Comments
1	Processing/Plant	Low Concentrate Grade	Concentrate grade too low	HIGH	HIGH	6	>$5million	Test existing and new metallurgical processes with a focus on slimes removal and flotation technologies (flash flotation, arsenopyrite/pyrite separation, ultrasonics). Develop testwork programme to gain understanding of deposit geometallurgy. Utilise information in design of plant to optimise concentrate grade.	
2	Processing/Plant	Concentrate Specs Not Met	Concentrate produced does not meet the required specs of processor/smelter	HIGH	HIGH	6	>$5million	As above.	
3	Processing/Plant	Clay in Ore	Clay affecting mining, crushing and processing of ore; do we require roll crusher before jaw crusher	HIGH	HIGH	6	>$5million	Investigate existing and new metallurgical processes for slimes removal and clay mitigation. Plant design to account for high clay content. Develop testwork programme to gain understanding of deposit geometallurgy.	
4	Processing/Plant	More Metallurgical Testwork Required for Economics	More detailed testwork required help define requirements for plant and associated costs or economics	HIGH	MEDIUM	5	>$1million	Develop project testwork programme to gain understanding of deposit geometallurgy using both in-house and external expertise. Assign budget to geometallurgical programme. Utilise test results in plant design and project economics.	
5	Processing/Plant	Metallurgical Characteristics Incomplete for Design	Incomplete understanding of metallurgical charactersitics	HIGH	MEDIUM	5	>$5million	Design project testwork programme to gain understanding of deposit geometallurgy and incorporate results into plant design. Assign budget to geometallurgical programme. Utilise test results in plant design and project economics.	
6	Construction & Implementation	Construction/Commission Delays	Delays in construction and/or commissioning schedule	MEDIUM	HIGH	5	>$5million	Incorporate both penalties and bonuses into construction contracts to discourage delays.	
7	Geotechnical	Pit Slope Instability/Failure(s)	Pit instability or failures affecting pit production	MEDIUM	HIGH	5	>$5million	Measure geotechnical properties of orebody. Incorporate these measurements into mine design.	
8	Geotechnical	Landform/Slope Stability or Failure	Landform instability or failures affecting the TSF, waste dump and plant/infrastructure	MEDIUM	HIGH	5	>$1million	As above.	
9	Processing/Plant	Low Concentrate Recovery	Concentrate recovery too low	MEDIUM	HIGH	5	>$1million	Test existing and new metallurgical processes with a focus on slimes removal and flotation technologies (flash flotation, arsenopyrite/pyrite separation, ultrasonics). Develop testwork programme to gain understanding of deposit geometallurgy. Utilise information in design of plant to maximise concentrate recovery.	
10	Processing/Plant	Plant Design Specifications Not Met	Plant operation not meet design specifications	MEDIUM	HIGH	5	>$5million	Design project testwork programme to gain understanding of deposit geometallurgy and incorporate results into plant design. Assign budget to geometallurgical programme. Utilise test results in plant design.	
11	Procurement & Capital Items	Delivery Schedule Delay	Delay in capital item delivery or delays due to other impacts (customs, shipping, etc.)	LOW	HIGH	4	>$5million	Order critical items asap. Incorporate penalties and bonuses into delivery contract. Track delivery status on a regular basis.	
12	Permits/Approvals	Mining Certificate/Lease Delays	Inability or delays of Gladioli to obtain MC/ML renewal covering part of the mine operational area (mainly TSF and waste landform)	LOW	HIGH	4		Ensure regular and ongoing liaison with Gladioli. Track progress of permits and ensure deadlines are met.'	
13	External Factors	Political Change/Government Interference	Changes in the current political situation or interference from government officials	LOW	HIGH	4		Communicate regularly with all parties and promote the project to ensure positive views. Monitor political communications for any negative communications	
14	Geology/Resource	Missing or Incomplete Resource Data	Possible missing elements affecting process; Zonation of mineralogical characteristics unknown; Incomplete key element data (particularly S & Fe)	MEDIUM	MEDIUM	4		Ensure data is captured in future drilling and if applicable in any grade control work.	
15	Geology/Resource	Oxidised Layer	Impact of partial oxidative layer - amount and volume of oxidated material	MEDIUM	MEDIUM	4		Geological mapping and grade control to monitor the oxidised layer. Track plant performance and recovery.	
16	Environmental & Rehab	EIA Delayed/Rejected	Process of obtaining EIA delayed/rejected	LOW	HIGH	4		Ensure EIA basline work is comprehensive enough. That the EIA report and EIA consultant have clearly identified the effects and applied suitable mitigation measures. Track the EIA schedule and timeline closely. Ensure open and clear communications with all parties.	

Table 22-17 - Project Risk Register (Part 1)

No.	Risk Group	Risk Description	Description of Consequence or Impact	Probability	Consequence	Score	Estimated Cost Impact	Mitigation Measures	Comments
17	Environmental & Rehab	MRP Delayed/Rejected	Process of obtaining or acceptance of MRP is delayed or rejected	LOW	HIGH	4		As above.	
18	General	Inflationary Impacts	Inflationary effects on pricing due to delays	HIGH	LOW	4	>$1million	Use of hedge instruments. Incorporate inflationary estimates into economic model.	
19	Permits/Approvals	Building/Construction Permit Delays	Delays in building/construction permits issued by local government	LOW	HIGH	4		As per EIA, MRP and other government processes	
20	Geology/Resource	Lower Average Grade	Resource grade lower on average than in model	MEDIUM	MEDIUM	4		Monitor through geological investigations and grade control	
21	Processing/Plant	Plant Operational/Throughput Problems	Problems affecting the plant throughput - bottlenecks, breakdowns, under-performance	MEDIUM	MEDIUM	4	>$5million	Design project testwork programme to gain understanding of deposit geometallurgy and incorporate results into plant design. Assign budget to geometallurgical programme. Utilise test results in plant design.	
22	External Factors	Gold Export Rule Change	Increase in current export rates for gold concentrate > 0%	LOW	MEDIUM	3			Current 0%
23	Environmental & Rehab	Acid Mine Drainage	Leakage or levels above permitted	LOW	MEDIUM	3	>$1million	Containment of PAF material and control of site drainage. Incorporation of lime dosage.	
24	Environmental & Rehab	Mine Closure Rehab Delayed/Rejected	Non-acceptance of mine closure rehab or delays due to rectification	LOW	MEDIUM	3		As per EIA, MRP and other government processes	
25	Hydrology & Water Management	Severe Weather Events	Impact of severe weather events on the mining operations or other operations (power disruption, flooding preventing staff getting to work, etc.)	MEDIUM	LOW	3	>$100,000	Incorporate weather forecasts in routine operational planning.	
26	Finance/Costs	Operating Cost Increases	Increase in some or all of operating costs	LOW	MEDIUM	3	>$1million	Maintain tight control on contract negotiations/costs; minimise unit costs and usage.	
27	Mining/Operations	Production Delays	Delays in reaching full/ongoing production	LOW	MEDIUM	3		Regular and detailed project schedule to ensure no delays. Develop alternate options list should delaying events occur ahead of tme to ensure quick remedy	
28	General	Major Negative Event	Major event e..g fire, loss of power supply, etc.	LOW	MEDIUM	3		Regular monitoring of all hazards, regular checks and detailed H&S training. Develop a H&S strategy to deal with any incident	
29	External Factors	Royalty Rate Increase	Increase in current royalty rate >0%	LOW	MEDIUM	3		Communicate benefits of no increase and constantly monitor government opinion. Develop strategies to mitigate	Current 0%
30	Tailings Facility	Insufficient Waste Material	Insufficient construction material at point in time	LOW	LOW	2		Develop alternate plans and sources of material. Ensure detailed and regular short term planning to ensure no problem with waste aterial balance and supply	
31	External Factors	Illegal Miners	Illegal miners stealing gold/ore or impacting operations	LOW	LOW	2	>$10,000	Employ security team to keep deposit secure; regular contact with local police; physical barriers to exclude miners (fence).	
32	External Factors	Anti-Mining & Environmental Disruption	Protests or other interference from anti-mining groups or environmental groups	LOW	LOW	2	>$10,000	Regular monitoring of these groups. Good communications strategy to government and local residents. Good security and regular communications with the police.	
33	Environmental & Rehab	Excessive Rehabilitation Bond	Excessive rehabilitation bond and restrictive rehab conditions	LOW	LOW	2	>$1million	Design closure plan in accordance with best practice; use of reasonable examples in bond application.	
34	Contracts	Contract Conditions Not Met	Contract conditions with service provider not met on consistent basis	LOW	LOW	2	>$100,000	Close contract management - penalties and bonuses to encourage contract obeyance.	
35	Contracts	Poor Contractual Terms	Poor, inconsistent or vague contract terms	LOW	LOW	2	>$100,000	Legal review of contract conditions.	
36	Transport	Transport Security Issues	Security issues with concentrate transport - theft	LOW	LOW	2	>$100,000	Monitoring of vehicles, personnel and concentrate bags. Good security measures and plans	
37	Transport	Transport Disruption	Disruption due to ship unavailability, road issues, truck unavailability, etc.	LOW	LOW	2	>$100,000	Develop a strategy and plan to deal with any disruptions. Ensure suitable equipment, transport, personnel and other options to meet any problems	
38	Mining/Operations	Low Mine Production	Various factors impacting the mine production	LOW	LOW	2		Regular planning and operational monitoring to ensure no impacts on mine production	
39	General	Labour Issues	Insufficient labour, skills level and training	LOW	LOW	2		Develop a detailed labour, training and HR policy plan. Ensure good instructors and training material available	

Table 22-18 - Project Risk Register (Part 2)

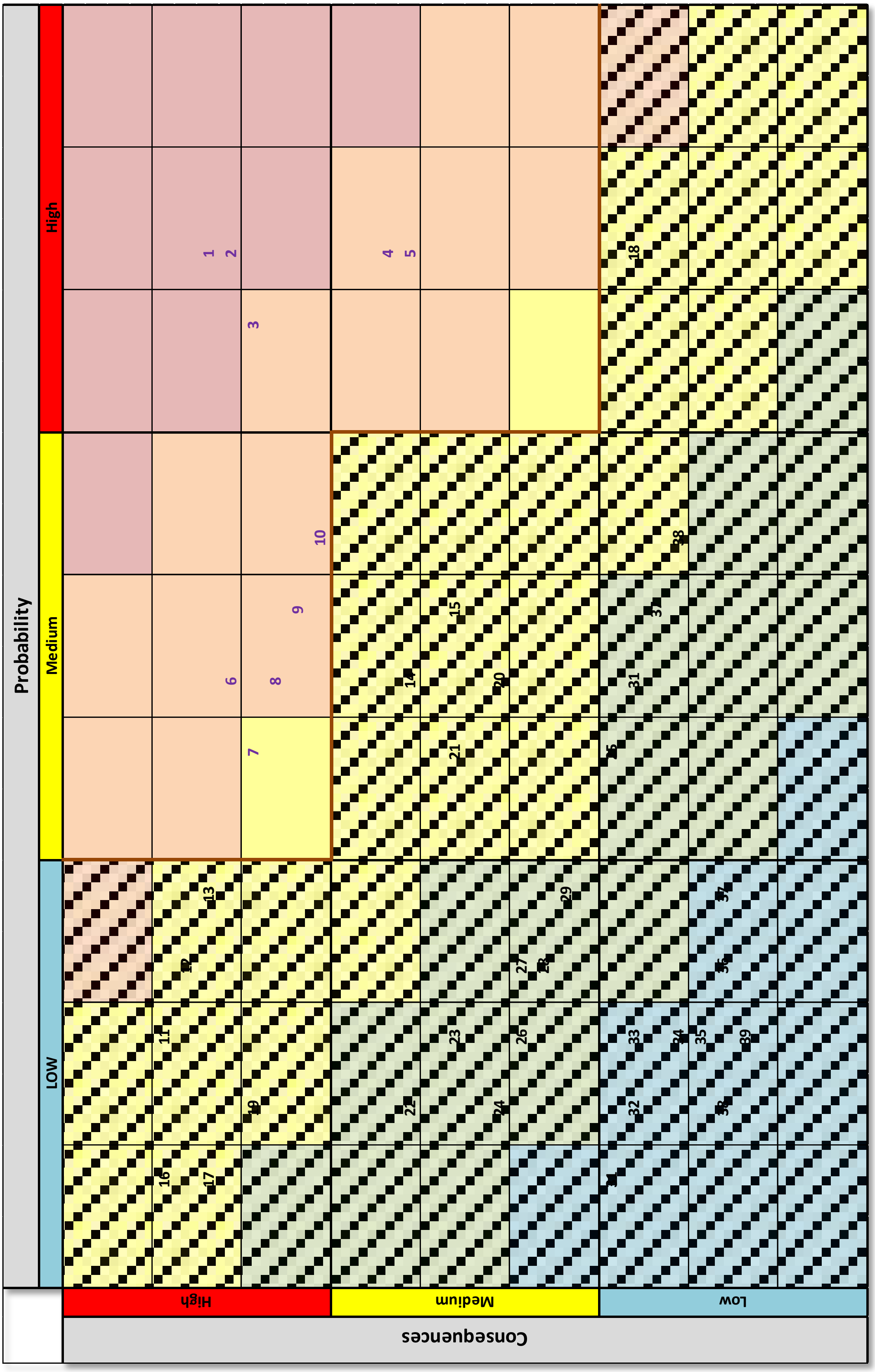


Table 22-19 - Consequence vs. Probability Risk Matrix

23. Adjacent Properties

There are no known significant producing properties adjacent to or near the Bau Gold property. North Borneo Gold Sdn Bhd (NBG) is the only significant explorer in the Bau Goldfield.

The most significant adjoining mine not under the control of NBG, is the now abandoned Lucky Hill Mine which was mined primarily for antimony but with reported high gold. There are no known production records available for this deposit which is part of the vein systems in the Krian area, near Bau.

The nearest properties with significant gold production history are in Kalimantan. These include the now closed Kelian Gold Mine, mined by CRA which produced approximately 176 tonnes of gold from and inventory of 245 tonnes, and the Mt. Muro Mine in central Kalimantan which is operated by Straits Resources and has a gold resource inventory of approximately 2 Moz (2009 Annual report, Straits Resources Limited).

Further to the north in Sabah, Malaysia's largest copper mine, the Mamut Porphyry Copper Deposit operated from 1975 to 1999 and had a reported production of 600,000 tonnes of copper, 45 tonnes of gold and 294 tonnes of silver. (Crimsonant.com, 2013).

24. Other Relevant Data & Information

24.1. Geotechnical Studies

24.1.1. Introduction

The geotechnical investigations conducted in Jugan and Bekajang areas were aimed at obtaining data for the two open pit mine sites and in the existing deactivated tailings storage facility (TSF) of the decommissioned mining pits of Bukit-Young and Tai Parit. The geotechnical study forms part of the feasibility study for the project being carried out by the company.

The study began in the Jugan area during the drilling period from September 2011 to September 2012 followed by a surface structural mapping from June 2012 to September 2012. Seventy-five (75) holes were drilled by the company in the area from JUDDH-6 to JUDDH-81 and these serve the basis for geologic logging and geomechanical rating of the rocks. The geomechanical logging with RMR rating and structural interpretation from the drill cores went together with the geological logging.

In the Bekajang area, forty-two (42) holes were drilled during the period April to December 2011. Geomechanical logging and RMR rating was also carried out on all the drill cores.

The geotechnical investigation on the deactivated Bekajang TSF is still on going up to date. Since April 2011, a total of thirteen (13) standard penetration test (SPT) have be carried out to form part of the in-situ measurement, including installation of nine (9) piezometers and five (5) inclinometers all around the TSF site. Some field vane shear tests were also done in identified soft grounds. The cone penetration test (CPT) is still on schedule to commence and around thirteen (13) CPT will also be done in the site.

24.1.2. Field Investigations & Findings

24.1.2.1. Jugan Sector

24.1.2.1.1. Jugan – Drillhole Geomechanical Logging

The drillhole geomechanical logging was done together with the geological logging of drillholes from JUDDH-6 to JUDDH-81. While the geological logging was largely based on the lithology, alteration and mineralisation, and veining and structures, the geomechanical logging was done based on the drill run at a maximum length of 3.0 metres per run. The geomechanical logging takes into account the several features of the rock, namely, the mechanical, structural and the mineralogical properties of the rocks and rates them according to the Rock Mass Rating (RMR) criteria.

The parameters in measuring the RMR are the following:

1. Rock Quality Designation (RQD) based on:

   a. Recovered length
   b. Length of run
2. Discontinuity per metre based on:
   a. Total number of discontinuities
   b. Recovered length of run
3. Discontinuity roughness
4. Discontinuity alteration and fill based on:
   a. Infill and mineralisation in the infill
   b. Alteration of the discontinuity walls
   c. Minerals present in the discontinuity walls
5. Weathering state of discontinuities
6. Aperture of the discontinuities
7. R-values taken from the intact samples of each lithology units
8. Intact Rock Strengths (IRS) derived from the weighted R-values of intercepted lithologies in the run

For the purpose of this study, the measured RMR values are used to develop the block model RMR for Jugan. Together with the structural model that was created, a slope design was developed for a planned 15m high with 5m bench face slope. The slopes from the optimised pit design generated using an initial overall 45-degree pit slope were adjusted based on the recommended slopes from the RMR block model.

As an example a geomechanical logsheet looks is shown below in *Figure 24-1: Geomechanical Logging Example - JUDDH-77* below.

HOLE ID	FROM	TO	RQD	DEFECT TYPE	NO. OF SET	COUNT	JN/M	JC ROUGHNESS	R FACTOR	JC FILL/ ALT.	ALT. FACTOR	JC WEATHERING	WEATHERING FACTOR	JC OPENING	APERTURE RATING	WATER	LITHO	R VALUE	IRS	RATING RQD	RATING IRS	RATING JN/M	RATING JC	RMR	RMR CLASS
JUDDH-77	0.00	1.50	0.00		3	9999	>30	ps	0.65	cp-ox	0.50	D	0.00	Vo	0.00		Overburden	0.00	2.85	5	0.13	2	0.00	7.13	Very poor
JUDDH-77	1.50	1.80	0.00		3	9999	>30	ps	0.65	cp-ox	0.50	D	0.00	Vo	0.00		Overburden	0.00	2.85	5	0.13	2	0.00	7.13	Very poor
JUDDH-77	1.80	3.30	60.00		3	9999	>30	ps	0.65	cp-ox	0.50	Hw	0.17	Mo-T	0.75		Overburden-Shale	43.33	29.31	12	2.27	2	1.66	17.92	Very poor
JUDDH-77	3.30	4.80	0.00		3	9999	>30	ps	0.65	cyc	0.67	Mw	0.50	Vo	0.00		Shale	50.00	41.96	5	3.20	2	0.00	10.20	Very poor
JUDDH-77	4.80	6.30	13.33		3	9999	>30	ps	0.65	cyc	0.67	Sw	0.83	Vo-O	0.09		Shale	50.00	41.96	5	3.20	2	1.23	11.43	Very poor
JUDDH-77	6.30	6.80	92.00	j	1	1	2	ps	0.65	cyc	0.67	Uw	1.00	T-Vt	0.92		Shale	50.00	41.96	19	3.20	24	15.94	62.14	Good
JUDDH-77	6.80	7.80	0.00	j	3	15	19	ps	0.65	cyc	0.67	Uw	1.00	Vo	0.00		Shale	50.00	41.96	5	3.20	8	0.00	16.20	Very poor
JUDDH-77	7.80	8.80	13.00	j	3	16	16	ps	0.65	cyc	0.67	Uw	1.00	Vo-O	0.09		Shale	50.00	41.96	5	3.20	10	1.48	19.68	Very poor
JUDDH-77	8.80	9.60	0.00	j	3	12	15	ps	0.65	cyc	0.67	Uw	1.00	Vo	0.00		Shale	50.00	41.96	5	3.20	11	0.00	19.20	Very poor
JUDDH-77	9.60	10.80	10.83	j	3	10	8	ps	0.65	cyc	0.67	Uw	1.00	Vo-O	0.09		Shale	50.00	41.96	5	3.20	16	1.48	25.68	Poor
JUDDH-77	10.80	12.30	25.33	j	2	9	6	ps	0.65	cyc	0.67	Uw	1.00	O-Mo	0.42		Shale	50.00	41.96	5	3.20	19	7.32	34.52	Poor
JUDDH-77	12.30	13.80	50.67	j	2	8	5	ps	0.65	cyc	0.67	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	10	3.20	20	13.07	46.27	Fair
JUDDH-77	13.80	15.30	26.67	j	3	14	9	ps	0.65	cyc	0.67	Uw	1.00	O-Mo	0.42		Shale	50.00	41.96	6	3.20	15	7.32	31.52	Poor
JUDDH-77	15.30	16.20	0.00	j	3	14	16	ps	0.65	cyc	0.67	Uw	1.00	Vo	0.00		Shale	50.00	41.96	5	3.20	10	0.00	18.20	Very poor
JUDDH-77	16.20	16.80	50.00	j	1	4	7	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	10	3.20	21	15.45	49.65	Fair
JUDDH-77	16.80	18.30	8.67	j	3	12	12	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Vo	0.00		Shale	50.00	41.96	5	3.20	13	0.00	21.20	Poor
JUDDH-77	18.30	19.80	60.00	j	2	8	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	20	15.45	50.65	Fair
JUDDH-77	19.80	21.30	52.67	j	2	8	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	11	3.20	20	15.45	49.65	Fair
JUDDH-77	21.30	22.70	59.29	j	2	7	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	20	15.45	50.65	Fair
JUDDH-77	22.70	24.30	63.75	j	2	5	3	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	13	3.20	21	15.45	52.65	Fair
JUDDH-77	24.30	25.80	23.33	j	3	12	8	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	O-Mo	0.42		Shale	50.00	41.96	5	3.20	16	8.65	32.85	Poor
JUDDH-77	25.80	27.30	0.00	f	3	9999	>30	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Vo	0.00		Shale-Fault	16.67	6.98	5	0.40	2	0.00	7.40	Very poor
JUDDH-77	27.30	28.90	0.00	f	3	9999	>30	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Vo	0.00		Fault-Shale	6.25	3.99	5	0.27	2	0.00	7.27	Very poor
JUDDH-77	28.90	30.30	75.00	j	2	6	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-Vt	0.84		Shale	50.00	41.96	15	3.20	21	17.21	56.41	Fair
JUDDH-77	30.30	31.80	16.00	j	2	9	6	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Vo-O	0.09		Shale	50.00	41.96	5	3.20	19	1.75	28.95	Poor
JUDDH-77	31.80	33.30	82.00	j	2	8	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-Vt	0.84		Shale	50.00	41.96	17	3.20	20	17.21	57.41	Fair
JUDDH-77	33.30	34.80	84.00	j	2	6	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-Vt	0.84		Shale	50.00	41.96	17	3.20	21	17.21	58.41	Fair
JUDDH-77	34.80	35.80	95.00	j	1	2	2	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	T-Vt	0.92		Shale	50.00	41.96	19	3.20	24	18.85	65.05	Good
JUDDH-77	35.80	36.30	42.00	j	1	2	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo	0.67		Shale	50.00	41.96	9	3.20	23	13.81	49.01	Fair
JUDDH-77	36.30	39.30	73.33	j	3	12	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-Vt	0.84		Shale	50.00	41.96	15	3.20	19	17.21	54.41	Fair
JUDDH-77	39.30	42.30	56.67	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	18	15.45	48.65	Fair
JUDDH-77	42.30	45.30	44.67	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo	0.67		Shale	50.00	41.96	9	3.20	18	13.81	44.01	Fair
JUDDH-77	45.30	48.40	69.68	j	3	13	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	14	3.20	19	15.45	51.65	Fair
JUDDH-77	48.40	51.30	57.59	j	3	18	6	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	17	15.45	47.65	Fair
JUDDH-77	51.30	54.30	56.67	j	3	12	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	19	15.45	49.65	Fair
JUDDH-77	54.30	57.30	57.33	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	18	15.45	48.65	Fair
JUDDH-77	57.30	60.10	62.14	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	13	3.20	18	15.45	49.65	Fair
JUDDH-77	60.10	63.20	67.74	j	3	13	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	14	3.20	19	15.45	51.65	Fair
JUDDH-77	63.20	66.30	35.48	j	3	19	6	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	O-Mo	0.42		Shale	50.00	41.96	7	3.20	17	8.65	35.85	Poor
JUDDH-77	66.30	69.30	30.67	j	3	9999	>30	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	O-Mo	0.42		Shale	50.00	41.96	6	3.20	2	8.65	19.85	Very poor
JUDDH-77	69.30	72.30	60.00	j	3	12	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	19	15.45	49.65	Fair
JUDDH-77	72.30	75.30	71.00	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-Vt	0.84		Shale	50.00	41.96	15	3.20	18	17.21	53.41	Fair
JUDDH-77	75.30	78.00	53.33	j	3	14	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	11	3.20	18	15.45	47.65	Fair
JUDDH-77	78.00	81.10	55.48	j	3	13	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	11	3.20	19	15.45	48.65	Fair
JUDDH-77	81.10	83.90	27.14	j	3	17	6	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	O-Mo	0.42		Shale	50.00	41.96	6	3.20	17	8.65	34.85	Poor
JUDDH-77	83.90	87.00	78.39	j	3	13	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-Vt	0.84		Shale	50.00	41.96	16	3.20	19	17.21	55.41	Fair
JUDDH-77	87.00	87.50	60.00	j	1	2	4	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	23	15.45	53.65	Fair
JUDDH-77	87.50	90.50	61.33	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	13	3.20	18	15.45	49.65	Fair
JUDDH-77	90.50	93.40	58.97	j	3	15	5	ps	0.65	no-cyc-ca-cb	0.79	Uw	1.00	Mo-T	0.75		Shale	50.00	41.96	12	3.20	18	15.45	48.65	Fair
JUDDH-77	93.40	96.50	45.16	j	3	16	5	ps	0.65	no-ca-cb	0.83	Uw	1.00	Mo	0.67		Shale-Sandstone	50.81	43.82	9	3.47	18	14.52	44.90	Fair
JUDDH-77	96.50	99.50	94.33	j	2	9	3	ps	0.65	no-ca-cb	0.83	Uw	1.00	T-Vt	0.92		Sandstone-Shale	54.83	54.42	19	4.00	21	19.83	63.83	Good

Figure 24-1: Geomechanical Logging Example - JUDDH-77

24.1.2.1.2. Jugan – Surface & Subsurface Structural Mapping & Interpretation

Details of the structural mapping were taken from the "Report on the Detailed Geological Mapping of Jugan" dated 9 November 2013. The detailed geological mapping was conducted by the company between June to September 2012. The objective of the geological mapping programme was to create an updated detailed map of Jugan to better understand the controls of mineralisation. It obtained an enhanced interpretation of the apparent trend of the deposit that is based largely on the structural expressions. This also provided the reasons that influenced heavily the mechanical properties of the rock mass aside from the different lithology units in the area.

The Jugan deposit is hosted by the Pedawan formation that consists of shale with interbedded siltstone and sandstone units. This sedimentary sequence is intruded by several post-mineral NW and WNW trending dacite porphyry dikes.

The majority of the bedding planes NE-SW direction while conjugate fractures and strike-slip faults generally trend NW. The presence of folds, thrust/reverse faults and strike-slip faults in the area indicates a compressional regime. Development of NE-trending folds, thrust faults and NW-trending strike-slip faults indicates that the principal stress ($\sigma 1$) is coming from the northwest heading towards the southeast. These NE and NW structures were interpreted to have formed during WNW event in the mid-Eocene. The event is comprised of EW to NW-SE compression. The NE-trending thrust faults recorded on the east side of the hill which was interpreted as part of footwall thrust and selected NW-trending faults matched to the structures interpreted in the drillholes.

The several exploration activities such as trenching and drilling that time made it difficult to recognise and map the structures. All minor and major thrust/reverse faults recorded on the central to the east part of Jugan hill are all NE trending and moderately to steeply dipping to the NW. Only few fold axes were recorded and all are NE trending and slightly plunging (<10°) to the NE.

On the west part of the hill, the fracturing, shearing and deformation is more intense compared to the east part. There are three (3) sets of bedding planes recorded on the west part, first is trending E-W, second is NE and the third is trending NW forming highly deformed structures. An ENE trending isolated fold and a localised NNW trending listric fault is also noted in this part with strike-slip and dip-slip movement along certain fault planes. Structures recorded in Jugan surface mapping and trenching are displayed in *Figure 24-2: Surface and Trench Structural Mapping at the Jugan Hill Deposit* below.

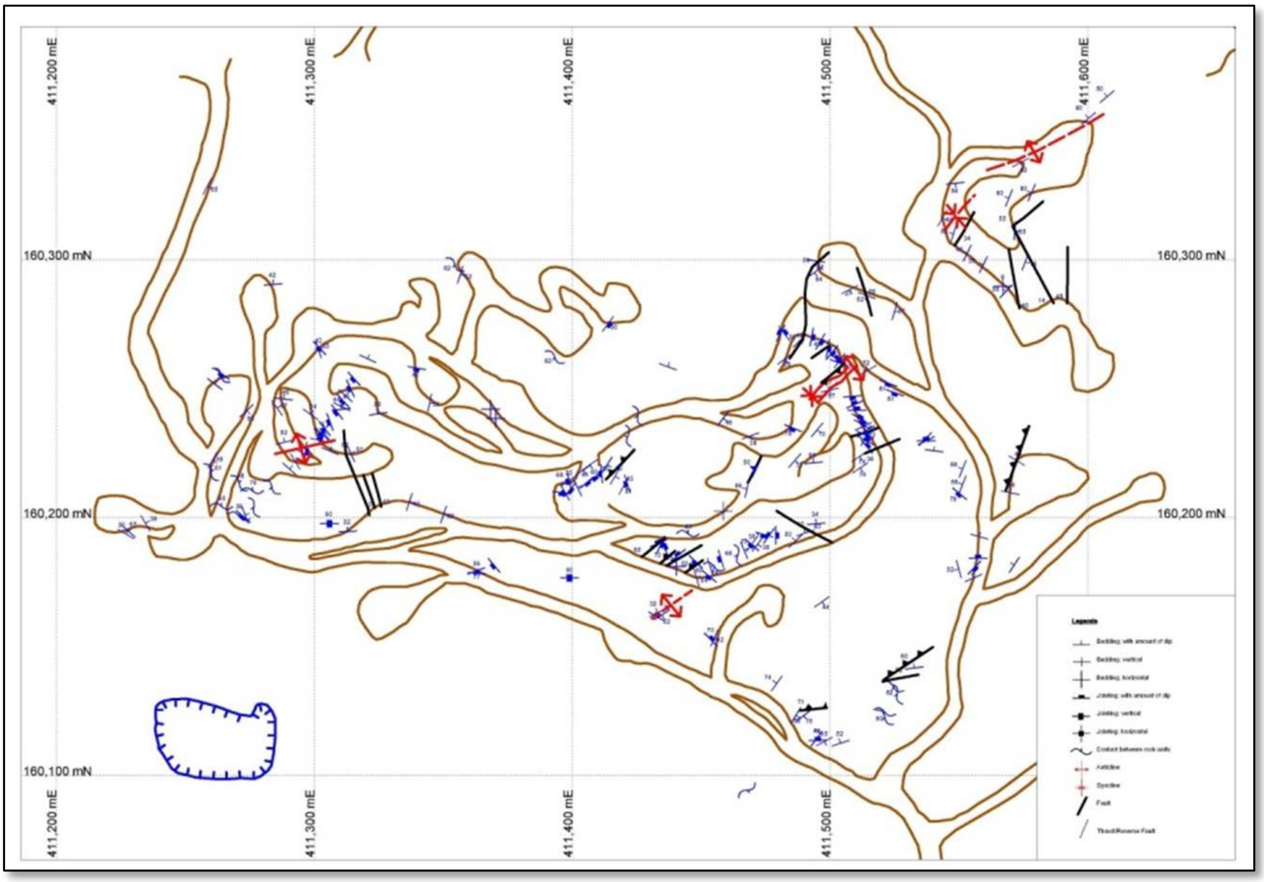


Figure 24-2: Surface and Trench Structural Mapping at the Jugan Hill Deposit

On the subsurface structural mapping, the findings at depth are as follows. Based on recorded structural data from two hundred and fifty three (253) drillholes in Jugan, which the included the seventy-nine (79) drilled by the company, several series of faults were identified and interpreted to provide control on the geometry and possible extension of the known Jugan orebody.

There are two (2) general set of fault trends: first is the pre to syn-mineral, ENE-trending and NW-dipping thrust/reverse-faults. Generally, these structures bound the mineralisation along the footwall and hanging wall. However they do not strictly confine and limit it as mineralisation was observed to extend or come-short from the thrust-fault contact.

The second set is the syn to post-mineral, steeply-dipping, NW-trending, conjugate strike-slip, oblique and scissor faults. These NW-faults cut across the earlier ENE faults and are the result of differential movement from compression and thrusting. These structures are thought to be responsible for the offset of the mineralisation to the ENE and tapering of the geometry of the orebody in the SW.

The complexity of structures in Jugan can be correlated to the NNE and WNW deformation events. The WNW event has more impact being the most recent event.

Based on the surface mapping and structural evidence collected from the drillholes, it is concluded that there are two major fault trends controlling the geometry and limits of the Jugan orebody. Firstly is the NE-trending thrust/reverse fault that confines the ore body and secondly the NW-trending strike-slip faults that cut and displaced the earlier NE-trending structures as well as the orebody. The NW faults are younger and are observed to cut and displaced the older NE thrust.

In terms of geotechnical consideration other than the strength of the rockmass based on its RMR rating and the observed surficial degradation of shale upon exposure of the expansive clays in the atmosphere, the presence of these major structures, including the dike intrusive trending NNW were also considered into the final pit design after the pit optimisation process.

The attached plan view in *Figure 24-3: Sub-Surface Structural Modelling at -100 mRL* shows the intertwining major structures at depth of -100 mRL. Those ones in yellow colour are the series of conjugate strike-slip-and-oblique NW-trending faults. Those blue ENE-trending thrust faults that flank the orebody in the north represent the upthrown hanging wall while the other flanking at the south is the footwall side. The one highlighted in magenta is the NNE-trending orebody dipping NW.

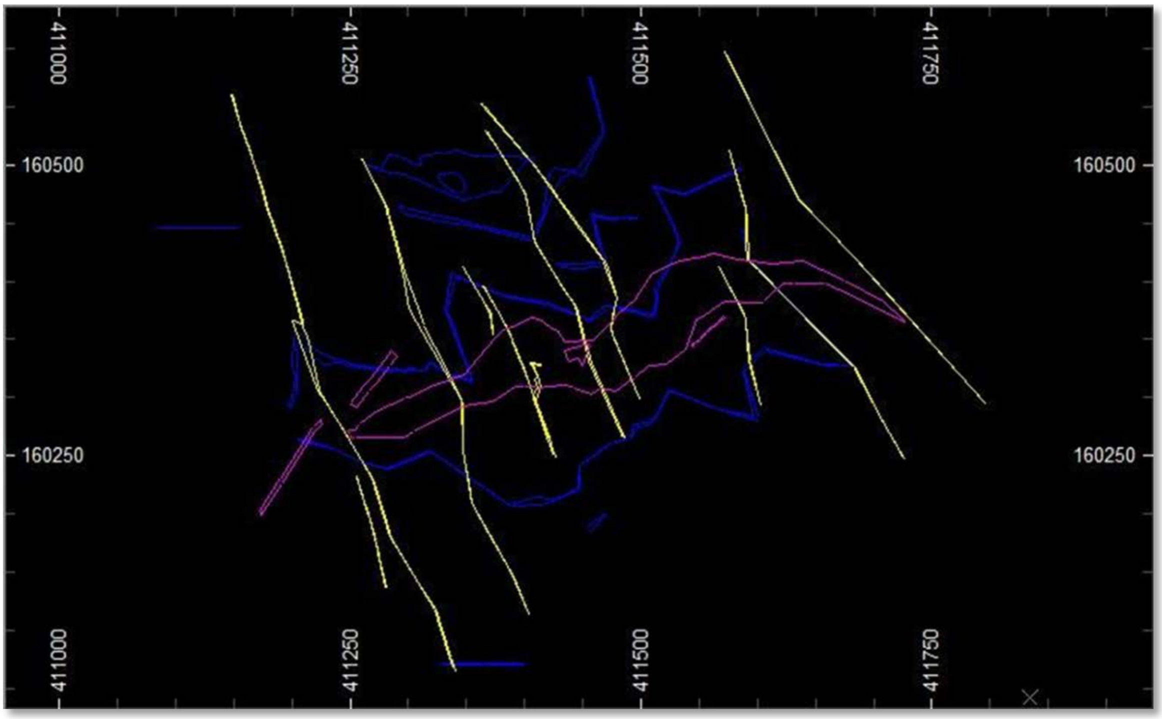


Figure 24-3: Sub-Surface Structural Modelling at -100 mRL

24.1.2.1.3. Jugan – Geotechnical Modelling

The geotechnical modelling is based on the block model derived from the RMR ratings of the drillholes and the surface and subsurface structural interpretations at Jugan.

In terms of the structural complexity in the area, it needs to be accepted that some berm losses will occur along with some instabilities from the local slopes. The rapid variations in structural conditions should also limit the scale of the failures provided that the major faults, the folds, shear zones, and the NW-trending clay-altered dike are not undercut. These features would however be evident while the pit develops and appropriate measures must be taken when observed. The orientation and continuity of the structural features control the pit wall stability.

Upon exposure, the shale rocks are prone to disintegration. This is evidenced by the breaking down of the cores, surface exposures of the clays (smectite and illite) and fracturing. It is however expected that this will only be a surface feature within the proposed open pit, with slopes exposed in the long term having small talus slopes forming at the toe.

The walls of the open pit are designed consistent with the economic factors, the stability during the life of mine and the consequence of any failure, for example pit access will require a higher safety factor than for some other areas of the pit. The final wall design is a result of the interaction between the orebody geometry, the pit access and the stability factor. It is observed and recorded that due to the silicified nature of the orebody; it is relatively competent compared to the surrounding host rock.

The proposed mining envisages an open pit mining method over an approximate area of 240,000 m^2. The mine design is currently proceeding as part of the feasibility which was based on the optimised pit using a 56° cut slope with bench height of 15 m and 5 m berm arriving at 45° overall pit slope. The face slopes were later on adjusted using the RMR block model.

From 30 mRL to -85 mRL, the rock mass approximate friction angle varies from 15° to slightly above 30° while cohesion varies from less than 100 kPa to slightly above 250 kPa. Down to -85 mRL, the rocks are rated between poor to fair, and from RL -85 mRL to pit bottom they are generally fair to good with friction angle ranging from above 30° to 40° and cohesion ranging above 250 kPa to 400 kPa.

Rosettes were applied in the slope design beginning from approximately 25 mRL down to -85 mRL to provide a varying pit slope designs. The pit slope at this sector is between 40-48°. From -85 mRL to the pit bottom at -145 mRL, the pit slope throughout is 48°.

A plan view *(Figure 238: Plan View of Jugan Orebody Wireframe & Pit Design*) of the open pit design relative to the orebody wireframe and a section view with the colour-coded RMR blocks together with the orebody (in magenta) and the modelled major structures (in orange) are provided for in the succeeding pages as examples on how the interpretation and modelling was done using CAE Mining Studio 3. In the section view (*Figure 24-5: Section View of Jugan Orebody, RMR Model and Pit Design*) looking N45E along section line 135_02, the graphics display on the RMR block model, the orebody and structure wireframes are shown as intersections.

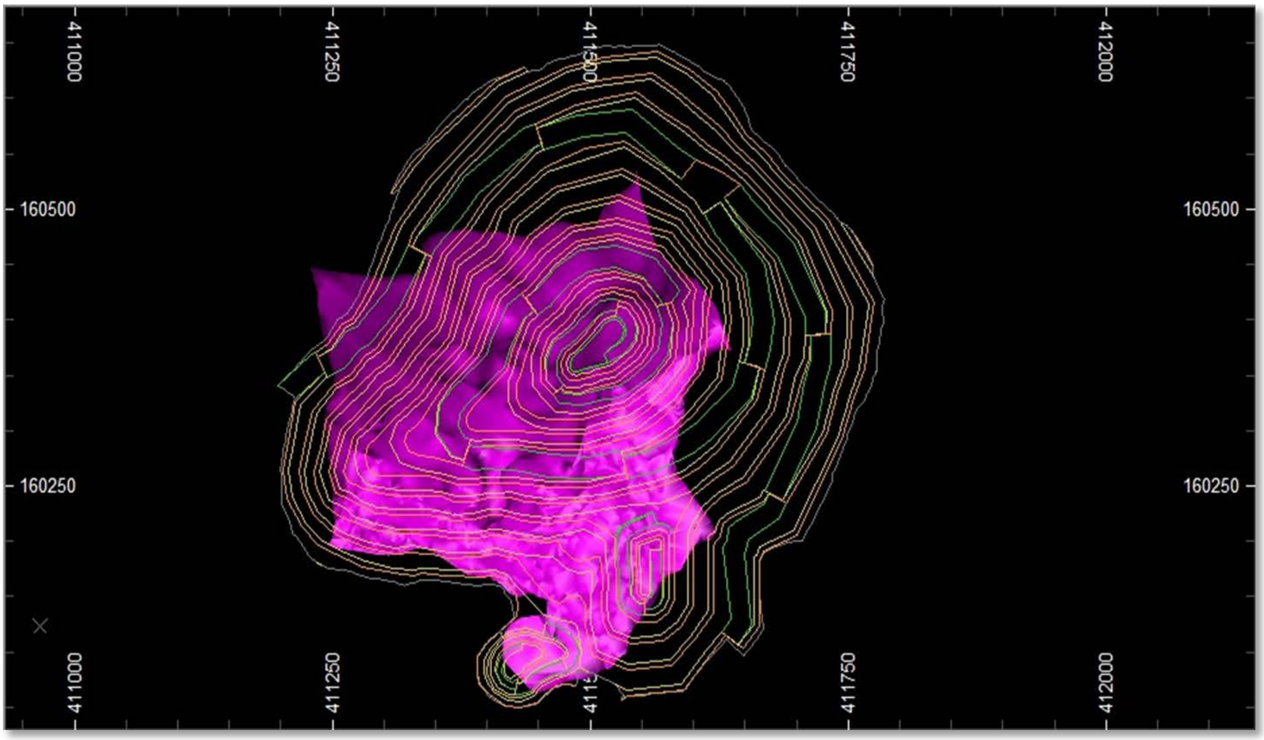


Figure 24-4: Plan View of Jugan Orebody Wireframe & Pit Design

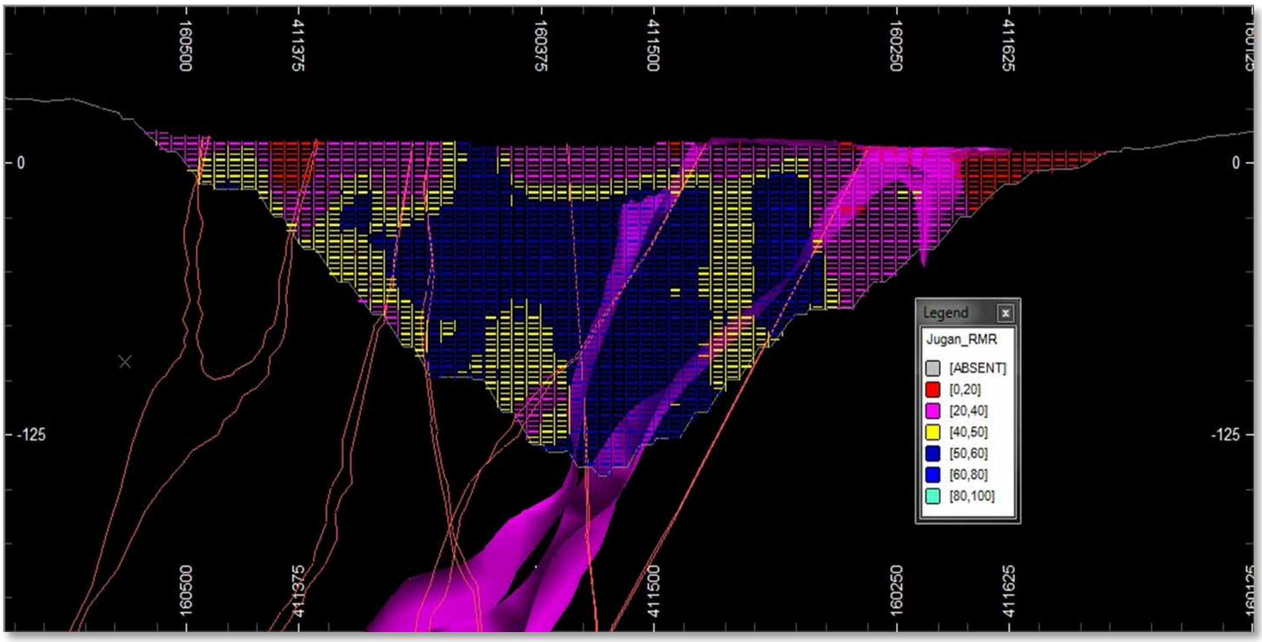


Figure 24-5: Section View of Jugan Orebody, RMR Model and Pit Design

As stated earlier, it is observed and recorded that due to the silicified nature of the orebody; it is relatively competent compared to the surrounding host rock. Hence, it has a relatively higher RMR than the host rock/waste rock. *Table 24-1: Jugan Orebody RMR Values in Group Ranges* and *Table 24-2: Jugan Host Rock/Waste RMR Values in Group Ranges* below show a summary of the RMR values of the ore and the waste rock.

RMR Group		Population (%)	Est. Friction Angle *(φ)*	Est. Cohesion (kPa)
Very poor	0-20	3.10	<15	<100
Poor	20-30	10.55	15-20	100-150
Poor	30-40	19.34	20-25	150-200
Fair	40-50	23.93	25-30	200-250
Fair	50-60	39.67	30-35	250-300
Good	60-80	3.41	35-45	300-400
Very good	80-100	-	>45	>400

Table 24-1: Jugan Orebody RMR Values in Group Ranges

RMR Group		Population (%)	Est. Friction Angle *(φ)*	Est. Cohesion (kPa)
Very poor	0-20	8.43	<15	<100
Poor	20-30	18.56	15-20	100-150
Poor	30-40	25.51	20-25	150-200
Fair	40-50	22.19	25-30	200-250
Fair	50-60	24.61	30-35	250-300
Good	60-80	0.69	35-45	300-400
Very good	80-100	-	>45	>400

Table 24-2: Jugan Host Rock/Waste RMR Values in Group Ranges

Other than the established RMR values and the corresponding designed pit angles from 25 mRL to -85 mRL (40-48°) and -85 mRL down to -145 mRL (48°), the additional procedures should be observed:

- The disintegration of the shale domain from 25 mRL down to -85 mRL upon exposure. As stated earlier, due to the presence of smectite and illite along the laminations, the shale rocks are prone to disintegration. This may impact on the stability of the pit slopes. Field mapping and additional investigation as the pit develops will be carried out.
- Drying of the clays during summer season may result into development of tension cracks, hence further degradation of excavated surfaces. Tension cracks may be present behind steep excavations faces. Intermittent rain water may percolate and create high pore water pressure that will destabilise the slope. Adequate run-off diversion is required and routine, on-going inspections and monitoring are recommended.
- Drainage measures, such as horizontal drains, may be required at localised areas where preferential seepage is observed to maintain slope stability.
- A detailed kinematic slope stability assessment by wedge analysis should be carried out as soon as the pit is developed to the point wherein structural mapping can be conducted either by digital photogrammetry or field mapping. Through this, the

existing structural geology model based on the previous field mapping of the Jugan hill and exploration boreholes can be updated.

- Where the pit is advanced to the presumed fault locations at depth, or where the faults daylight/ meet with the surface, or where the faults change in dip directions (as most of them are conjugate faults especially those at the northwest side of the pit), additional investigations should be done every now and then, or at need basis as part of the update on the structural model.
- Pit development should include assessment of slope performance that allows adjustment to the slope geometry.
- Piezometers will be needed in deep wells that are scheduled to be drilled within the peripheries of the pit as part of the hydrogeological study for Jugan Pit. Details of the study are explained in *Section 21.1*. The water level variations relative to the pit development and seasonality would have to be monitored on a monthly basis. This will enable modelling and prediction of water inflow to the pit.
- Inclinometers may have to be established in strategic locations to monitor pit slope movements, e.g. east wall, SE wall, NW wall and safeguard mine operations.
- Slope alarms or radar monitors, the latter if budget permits, may also be established along the pit walls in conjunction with the inclinometers.
- An operational manual is needed for the safe development and operation of the open pit.

24.1.2.2. Bekajang/Krian Sector

24.1.2.2.1. BYG – Drillhole Geotechnical Logging

The drilling in the area was focused mainly in the deactivated Bukit-Young Pit, which is a part of the Bekajang/Krian Sector. The objective was to find the mineralised extension of the orebody that was once mined by the previous company Bukit-Young Gold Sdn Bhd.

The drillhole geomechanical logging on the drillholes BYDDH-01 to BYDDH-42 was done together with the geological logging. Like the same methodologies adopted on the Jugan drillholes, the same set of procedures was employed on the BYDDH holes. While the geologic logging was largely based on the lithology, alteration and mineralisation, and veining and structures, the geomechanical logging was done based on the drill run at a maximum length of 3.0 metres per run. The geomechanical logging takes into account the several features of the rock, namely, the mechanical, structural and the mineralogical properties of the rocks and rates them according to the Rock Mass Rating (RMR) criteria. The parameters in measuring the RMR are the same as those explained in *Section 19.2.1.1*.

The measured RMR values are used to develop the RMR block model for Bukit-Young. Together with the projections from the initial structural model that was created, a slope design was developed for a planned 15m high with 5m bench face slope. The slopes from the optimised pit design generated using an initial overall 45° pit slope were adjusted based on the recommended slopes from the RMR block model.

An example of the geomechanical logsheet for BYG drillholes is shown in *Figure 24-6: Geomechanical Logging Example - BYDDH-34* below.

HOLE ID	FROM	TO	RQD	DEFECT TYPE	NO. OF SET	COUNT	JN-M	JC ROUGHNESS	R FACTOR	JC FILL/ ALT	ALT. FACTOR	JC WEATHERING	WEATHERING FACTOR	JC OPENING	APERTURE RATING	WATER	LITHO	R VALUE	IRS	RATING RQD	RATING IRS	RATING JN-M	RATING JC	RMR	RMR CLASS
BYDDH-34	0	1.5	10.67	j	2	8	10	Ps	0.65	cgt	0.67	Sw	0.83	Vo-O	0.09		Soil, Clay, Mullock, Rock Fill or Tailings	0.00	2.85	5.0	0.12	16.0	1.23	22.36	Poor
BYDDH-34	1.5	1.9	0.00	j	2	8	20	Ps	0.65	cgt	0.67	Sw	0.83	Vo	0.00		Soil, Clay, Mullock, Rock Fill or Tailings Intrusive	0.00	2.85	5.0	0.13	10.0	0.00	15.13	Very poor
BYDDH-34	1.9	3.4	0.00	-	3	9999	>30	Pr	0.70	cp-ox	0.50	D	0.00	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	3.4	4.9	0.00	-	3	9999	>30	Pr	0.70	cp-ox	0.50	D	0.00	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	4.9	6.4	0.00	-	3	9999	>30	Pr	0.70	cp-ox	0.50	Mw	0.50	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	6.4	7.9	0.00	-	3	9999	>30	Pr	0.70	cp-ox	0.50	Mw	0.50	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	7.9	9.4	0.00	-	3	9999	>30	Pr	0.70	cp-ox	0.50	Mw	0.50	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	9.4	10.9	0.00	-	3	9999	>30	Pr	0.70	qt-scyc-c	0.65	Sw	0.83	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	10.9	12.2	0.00	-	3	9999	>30	Pr	0.70	qt-scyc-c	0.65	Sw	0.83	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	12.2	13	0.00	-	3	9999	>30	Pr	0.70	qt-scyc-c	0.65	Sw	0.83	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	13	13.8	0.00	-	3	9999	>30	Pr	0.70	qt-scyc-c	0.65	Uw	1.00	Vo	0.00		Intrusive	65.00	94.03	5.0	6.40	2.0	0.00	13.40	Very poor
BYDDH-34	13.8	14.6	55.00	j	1	2	3	Pr	0.70	qt-scyc-c	0.65	Uw	1.00	Mo-T	0.75		Intrusive-Andesite	66.50	101.94	11.0	7.00	23.0	13.65	54.65	Fair
BYDDH-34	14.6	15.7	95.45	j	1	2	2	Pr	0.70	qt-ca-c-sl	0.77	Uw	1.00	T-Vt	0.92		Andesite	67.00	104.72	19.0	7.00	24.0	19.73	69.73	Good
BYDDH-34	15.7	16.8	100.00	j	1	1	1	Pr	0.70	qt-ca-c-sl	0.77	Uw	1.00	Vt	1.00		Andesite	67.00	104.72	20.0	7.00	25.0	21.56	73.56	Good
BYDDH-34	16.8	18.4	100.00	j	1	1	1	Pr	0.70	qt-ca-c-sl	0.77	Uw	1.00	Vt	1.00		Andesite	67.00	104.72	20.0	7.00	25.0	21.56	73.56	Good
BYDDH-34	18.4	19.1	97.14	j	1	1	1	Pr	0.70	qt-ca-c-sl	0.77	Uw	1.00	T-Vt	0.92		Andesite	67.00	104.72	20.0	7.00	25.0	19.73	71.73	Good
BYDDH-34	19.1	19.9	100.00	v	1	2	5	Pr-Us	0.75	qt-ca-c	0.79	Uw	1.00	Vt	1.00		Andesite	67.00	104.72	20.0	7.00	22.0	23.70	72.70	Good
BYDDH-34	19.9	21.4	100.00	j	1	1	1	Us	0.80	qt-ca-c	0.79	Uw	1.00	Vt	1.00		Andesite	67.00	104.72	20.0	7.00	25.0	25.28	77.28	Good
BYDDH-34	21.4	22.9	98.67	j	1	2	1	Us	0.80	qt-ca	0.85	Uw	1.00	T-Vt	0.92		Andesite-Intrusive	67.00	104.72	20.0	7.00	25.0	24.89	76.89	Good
BYDDH-34	22.9	24.4	94.67	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	19.0	6.40	25.0	20.22	70.62	Good
BYDDH-34	24.4	25.9	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	25.9	26.8	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	26.8	27.5	95.71	j	1	1	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	19.0	6.40	25.0	20.22	70.62	Good
BYDDH-34	27.5	30.5	97.00	j	1	3	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	30.5	33.5	96.33	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	33.5	36.5	98.33	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	36.5	37.9	78.57	j	1	3	2	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Mo-Vt	0.84		Intrusive	65.00	94.03	16.0	6.40	24.0	18.45	64.85	Good
BYDDH-34	37.9	39.5	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	39.5	42.5	95.00	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	19.0	6.40	25.0	20.22	70.62	Good
BYDDH-34	42.5	43.5	89.00	j	1	1	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Mo-Vt	0.84		Intrusive	65.00	94.03	18.0	6.40	25.0	18.45	67.85	Good
BYDDH-34	43.5	45.5	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	45.5	48.5	98.67	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	48.5	51.5	95.33	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	19.0	6.40	25.0	20.22	70.62	Good
BYDDH-34	51.5	54.5	98.33	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	54.5	57.5	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	57.5	60.5	99.33	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	60.5	63.5	98.33	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	63.5	66.5	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	66.5	69.5	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	69.5	72.5	100.00	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	72.5	75.5	95.67	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	19.0	6.40	25.0	20.22	70.62	Good
BYDDH-34	75.5	78.5	97.67	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	78.5	81.5	100.00	-	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	81.5	84.5	100.00	0	1	0	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	84.5	87.5	96.67	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	87.5	90.5	100.00	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	90.5	93.5	96.67	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	93.5	96.5	100.00	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	96.5	99.5	96.33	j	1	2	1	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good
BYDDH-34	99.5	103	99.67	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	Vt	1.00		Intrusive	65.00	94.03	20.0	6.40	25.0	22.10	73.50	Good
BYDDH-34	102.5	106	97.33	j	1	1	0	Ur	0.85	qt-scyc-c	0.65	Uw	1.00	T-Vt	0.92		Intrusive	65.00	94.03	20.0	6.40	25.0	20.22	71.62	Good

Figure 24-6: Geomechanical Logging Example - BYDDH-34

24.1.2.2.2. BYG – Surface & Subsurface Structural Mapping & Interpretation

The Bekajang/Krian project is three (3) plus years down the line. A surface structural mapping in the area is planned into the near future.

The initial subsurface structural interpretation, however, were initially established based on the result from the airborne geophysical data done in the late 1990s. There were some sharp contrasts between the low and high resistivities in some areas within the Bekajang/Krian sector giving the impression that these are major structures. The aim of studying these structures recently is to understand the mineralisation based on its ore controls. It served also as a guide why recent initial drilling activities were conducted at the old deactivated Bukit-Young Pit.

In terms of geotechnical consideration other than the strength of the rockmass based on its RMR rating, the presence of these major structures trending NNE dipping steeply to the NW were also considered into the interim pit design after the pit optimisation process.

The semi- to detailed surface structural mapping and subsurface structural modelling will be part of the Bekajang/Krian project schedule into the near future.

24.1.2.2.3. BYG – Geotechnical Modelling

The geotechnical modelling on the Bukit Young Pit is based on the block model derived from the RMR ratings of the drillholes and the preliminary subsurface structural interpretations in the area.

In terms of the structural complexity in the area based entirely on aeromagnetics, there seems to be less structural variations in the major structures that trends NNE. These features, however, will be more evident when future exploration drilling commences together with the structural mapping. Also, when the pit develops, the better it will be to fully understand the visual imprints of the structures and how they influence pit wall stability. The overall geotechnical soundness of the walls other than the established mechanical properties of the rock fabric by RMR rating will then be thoroughly established.

The proposed mining envisages an open pit mining method over an approximate area of 85,530 m^2. The mine design is currently proceeding as part of the feasibility which was based on the optimised pit using a >65°cut slope with bench height of 10 m and 5 m berm width arriving at approx. 47° overall pit slope. The slopes were designed fixed at those angles from 39 mRL down to -50mRL.

It is observed that the orebody is relatively competent compared to the surrounding host rock. It has a relatively higher RMR than the host rock/waste rock. However, the orebody occurs in vughy sheared breccias rich in quartz, jasperiod and sulphides. *Table 24-3: BYG Ore & Waste RMR Values in Group Ranges* below shows a summary of the RMR values within the ore and waste. It should be noted that it is difficult to separate the two (2) materials in terms of their RMR values because of the patchy nature of the shear breccia type orebody.

RMR Group		Population (%)	Est. Friction Angle *(φ)*	Est. Cohesion (kPa)
Very poor	0-20	13.02	<15	<100
Poor	20-30	10.42	15-20	100-150
Poor	30-40	14.71	20-25	150-200
Fair	40-50	20.74	25-30	200-250
Fair	50-60	21.71	30-35	250-300
Good	60-80	19.37	35-45	300-400
Very good	80-100	0.04	>45	>400

Table 24-3: BYG Ore & Waste RMR Values in Group Ranges

A plan view (*Figure 24-7: Plan View of BYG Orebody Wireframe & Pit Design*) of the open pit design relative to the orebody wireframe and a section view of the pit with the colour-coded RMR blocks together with the orebody (in magenta) and the modelled major structures (in orange), are provided for in the succeeding pages. In the section view (*Figure 24-8: Section View of BYG Orebody, RMR Model and Pit Design*) looking north along section line BYC 2.5N, the RMR block model, the orebody and the structure wireframes are shown as intersections.

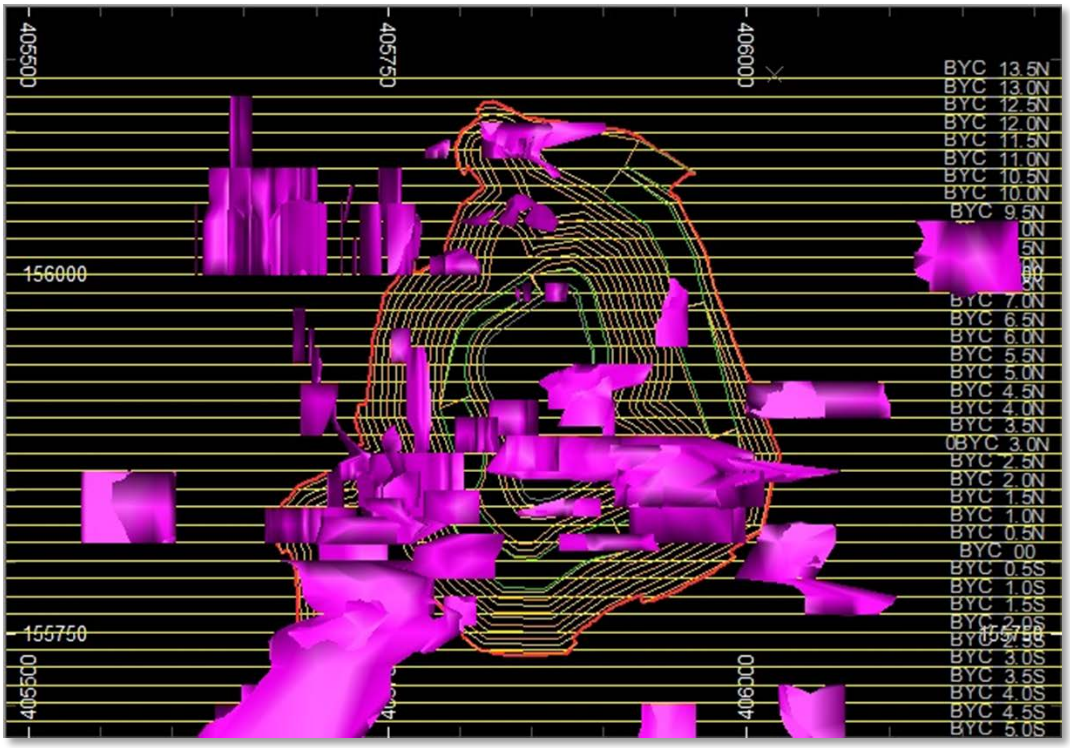


Figure 24-7: Plan View of BYG Orebody Wireframe & Pit Design

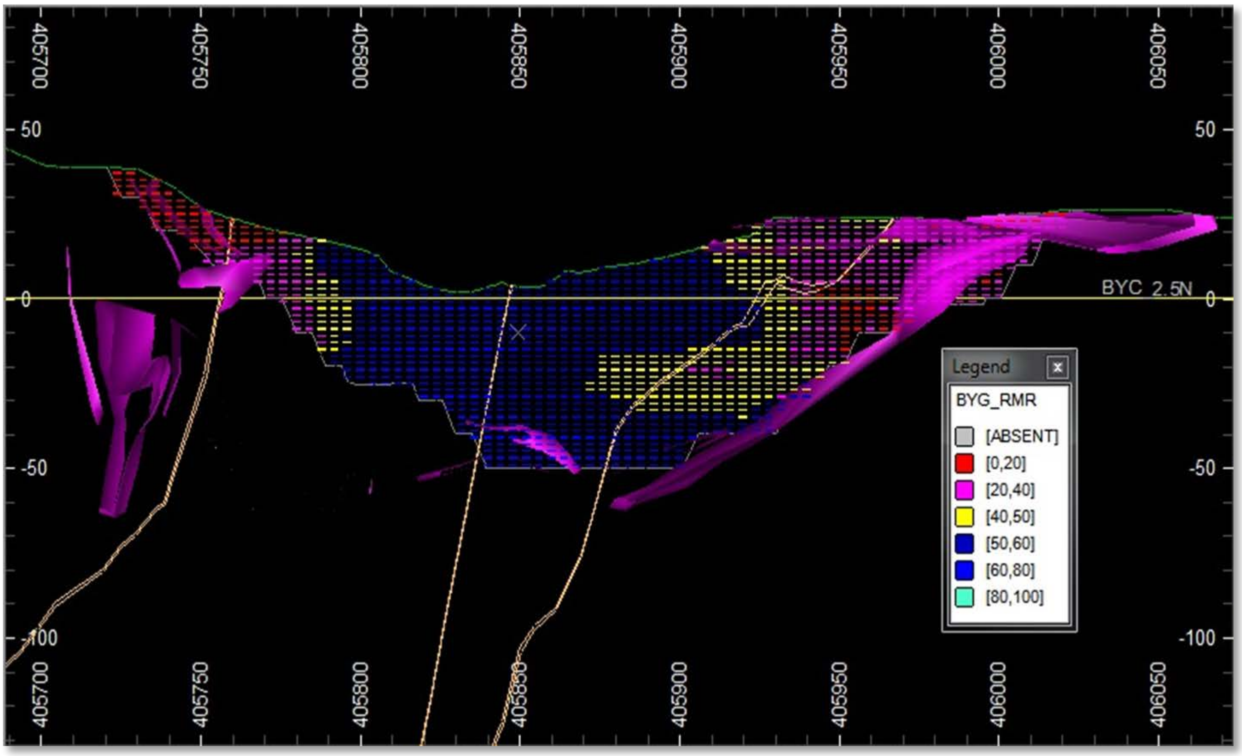


Figure 24-8: Section View of BYG Orebody, RMR Model and Pit Design

Other than the established RMR values and the corresponding designed 47° pit angle, the additional procedures should be observed:

- Drainage measures, such as horizontal drains, may be required at localised areas where preferential seepage is observed to maintain slope stability.
- A detailed kinematic slope stability assessment by wedge analysis should be carried out as soon as the pit is developed to the point wherein structural mapping can be conducted either by digital photogrammetry or field mapping. Through this, the existing structural geology model based on the previous aerial geophysical results, may be verified and updated.
- Where the pit is advanced to the presumed fault locations at depth, or where the faults daylight/ meet with the surface, or where the faults change in dip directions, additional investigations should be done every now and then, or at need basis as part of the update on the structural model .
- Pit development should include assessment of slope performance that allows adjustment to the slope geometry.
- Piezometers will be needed in deep wells that are scheduled to be drilled within the peripheries of the pit as part of the hydrogeological study for Bukit-Young Pit. Details of the study are explained below in *Section 21.2*. The water level variations relative to the pit development and seasonality would have to be monitored on a monthly basis. This will enable modelling and prediction of water inflow to the pit.
- Inclinometers may have to be established in strategic locations to monitor pit slope movements, e.g. NW wall and safeguard mine operations.
- Slope alarms or radar monitors, the latter if budget permits, may also be established along the pit walls in conjunction with the inclinometers.
- An operational manual is needed for the safe development and operation of the open pit.

24.1.2.3. Other

Other work that has been conducted, in progress or planned for other elements that have a geotechnical or geomechanical input are listed in the following sub-sections.

24.1.2.3.1. TSF – Proposed

The proposed TSF in Jugan is sized taking into account the flotation concentrate mass pull-out of 10%. Out of the total 9.71 Mt ore grading 1.56 g/t Au average that will be milled around 8.74 Mt at 0.40 g/t Au average will end up as tailings. The rest will be treated elsewhere (possibly in China) as gold concentrate.

For the build-up of the TSF, around 6.4 Mm^3 of mine waste and 1.9 Mm^3 derived from the cut materials in the containment pond will be required. This will be done in three (3) stages throughout the current mine life.

The proposed TSF is planned as a beach-type TSF during its operating years. The tailings dam will be built from RL 25m to RL 45m and will be provided with a final spillway at RL 43m to naturally drain out the supernatant or the tailings water fraction up to the spillway invert elevation and to handle any excess run-off water since the region is known for frequent rains. The final spillway is initially sized at 15m wide but this figure will be confirmed later on based on the updated historical rain figures and catchment size within the pond as the feasibility is still progressing. The design will be based on 72-hour PMF (Probable Maximum Flood) period.

The 20m high zoned TSF with combined clay and borrowed sulphide-free material as upstream material and mine rock waste as downstream material will be provided with a 1m thick blanket drain (D_{50} = 5cm) in between the upstream and downstream. A concrete cut-off drain at the downstream toe to handle seepage from the pond passing through the clay zone and into the blanket drain will also be part of the structure. The function of the blanket drain is to bring down the phreatic head passing through the upstream embankment zone such that disallowing excessive pore pressure from occurring at this side, to deny any form of seepage from the pond water to pass through the downstream embankment, and to prevent the downstream from getting saturated.

Figure 24-9: Proposed Jugan Tailings Impoundment Design in Relation to Pit below displays the layout of the proposed TSF relative to the Jugan Pit, and an estimate on tailings production after flotation process in *Table 24-4: Tailings Production Estimate - Jugan Pit*, thereafter. A 100 m buffer is given between the final toe of the dam to the final crest level of the pit.

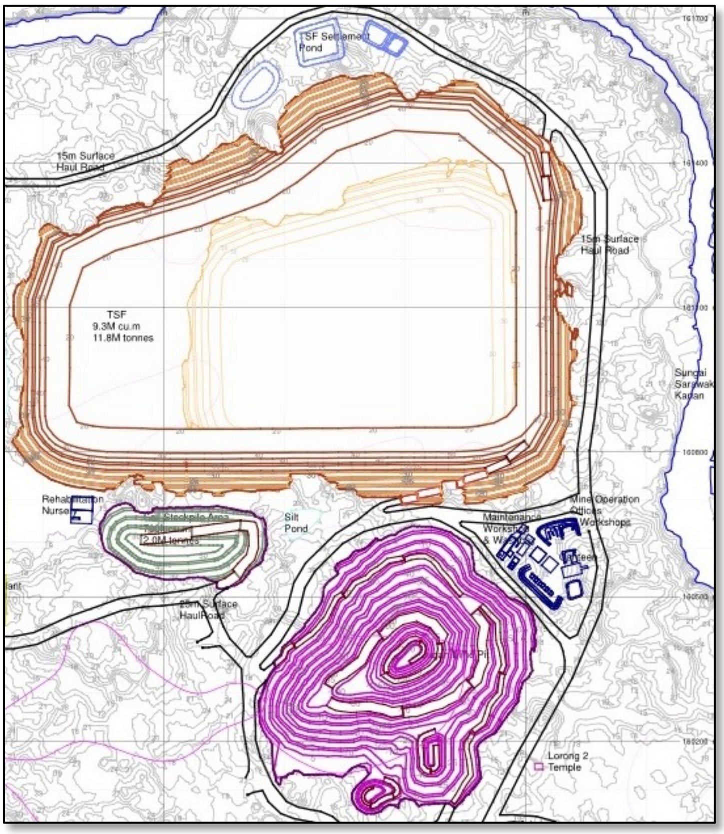


Figure 24-9: Proposed Jugan Tailings Impoundment Design in Relation to Pit

Year	Units	Year 1	Year 2	Year 3	Year 4	Total Average
Working days	days	365	365	365	119	1,213.91
Milling rate	t/d	8,000.00	8,000.00	8,000.00	8,000.00	8,000.00
	t/y	2,920,000.00	2,920,000.00	2,920,000.00	951,243.72	9,711,243.72
Mill grade	gAu/t	1.56	1.56	1.56	1.56	1.56
Recovery %	Au	77.00%	77.00%	77.00%	77.00%	77.00%
Flotation concentrate	t/d	800.00	800.00	800.00	800.00	800.00
	t/y	292,000.00	292,000.00	292,000.00	95,124.37	971,124.37
Production	oz Au/y	112,768.90	112,768.90	112,768.90	36,736.54	375,043.23
To tailings pond	flotation tailings, t/d	7,200.00	7,200.00	7,200.00	7,200.00	7,200.00
	flotation tailings, t/y	2,628,000.00	2,628,000.00	2,628,000.00	856,119.35	8,740,119.35
	gAu/t	0.40	0.40	0.40	0.40	0.40
	tails slurry, m^3	5,853,978.98	5,853,978.98	5,853,978.98	1,907,041.35	19,468,978.30
	tails solids, m^3	973,344.98	973,344.98	973,344.98	317,085.04	3,237,119.99
	tails slurry, m^3 (final settled volume)	2,364,306.34	2,364,306.34	2,364,306.34	770,216.29	7,863,135.31
	ore density, t/m^3	2.70	2.70	2.70	2.70	2.70
	flotation tails density, t/m^3	1.28	1.28	1.28	1.28	1.28
	% solids by weight	35.00%	35.00%	35.00%	35.00%	35.00%

Table 24-4: Tailings Production Estimate - Jugan Pit

The 8.74 Mt flotation tails from the Jugan Pit will be slurried at 35 % solids before pumping the tailings to the TSF pond. At 35 % solids it will have an initial estimated density of 1.3 t/m^3. The 20 m high ring dam to contain it has a total capacity of 8.6 Mm^3. The natural removal of the tailings water fraction at some point will be through the 15 m wide spillway, while desiccation will be achieved by beaching the tailings at the time of deposition. Beaching is accomplished by perimeter spigotting of the tailings. The estimated final settling density of the tailings is 1.70 t/m^3 at 65.4% solids, which will amount to about 7.9 Mm^3.

The upstream clay zone will maintain a slope of 2.5H : 1.0V or 21.8° while the downstream side made entirely of mine waste will maintain a local slope 2.0H : 1.0V but will be mated with 5m wide berms for every 5 m lifts. The resulting flatter downstream slope will be around 18.4° or about 3.0H : 1.0V average.

Estimated amount of materials needed for the construction are as follows:

- Upstream clay zone – 0.36 M lcm
- Downstream mine waste plus clay as cover – 11.10 M lcm
- Blanket drain – 0.37 Mm^3
- Geofabric – 0.77 Mm^2
- Concrete drain
    - 620 m^3 cement
    - 1760 m^3 sand
    - 4,250 pcs of 4m long 50mm dia. PVC pipes to be perforated
    - 2,430 pcs 2 m long x 1 m high x 1 m width rock gabion
- HDPE liner – 0.58 Mm^2
- Final spillway (20 m wide on top, 15 m wide bottom lined with 0.10 m thick concrete)
    - 60 m^3 cement
    - 90 m^3 sand

24.1.2.3.1.1. Soil Geotechnical Investigation Tests Required

The following tests are central to any TSF detailed design and construction and are precursor to any foundation engineering design. These are deemed to be the most applicable for our objective of constructing a tailings dam in Jugan that is idealised to be also a water-retention structure throughout its operational stage.

Soil Field Tests:

- Standard Penetrometer Test (SPT) up to zone of refusal
- Collection of undisturbed samples by SPT
- Cone Penetration Test or CPT (in selected areas following the SPT)
- Field vane shear test on soft soils encountered by SPT
- Field permeability test on soil
- Test pitting in selected areas
- Trenching – to observe preferential seepage pathway since the groundwater is believed to be shallow at this floodplain area
- Rock coring up to 4m deep, if intercepted
- Rock Quality Designation (RQD)
- Field permeability test on rockmass intercepted (Packer Test)

Laboratory Test for Soil, Drainage Blanket, and Mine Waste:

- Atterburg limits (liquid limit, plastic limit, Plasticity Index)
- Permeability of clay
- Grain size distribution
- Permeability of drainage blanket material where D50 = 5mm
- Density of clay
- Density of sand
- Density of mine waste
- Sulphate soundness test for blanket drain material
- Pinhole test for clay
- X-ray diffraction for clay
- Void ratio for clay
- Tri-Axial with pore pressure for clay
- Direct shear test for clay
- Unconfined Compressive Strength (UCS) for clay
- Absorption test for mine waste
- X-ray diffraction for mine waste
- Petrographic analysis for mine waste

Laboratory Test for Rock Cores:

- Specific gravity
- Unconfined Compressive Strength

As explained above, the idea of a water retention tailings dam instead of a typical beach-type is to prevent the oxidation of tailings by oxygen ingress. When the tailings are stored under water, the reduction in oxidation is about 25,000 times less and the production of sulphate (SO_4) in the water (supernatant coming out of the spillway and as seepage caught by the blanket drain) is maintained at a minimum. Also since our available filter drain material on site may most likely be sourced from the limestone or limestone-marblelised quarries, the production of precipitates that may blind the carbonate-type underdrain is minimised significantly.

24.1.2.3.2. TSF - Existing

24.1.2.3.2.1. Re-Activiation of Existing TSF at BYG

The possible resumption of the old TSF to accommodate any future tailings from the operation of BYG Pit, the Bekajang and Krian deposits is subject to the result of the on-going validation of its structural integrity at current state (deactivated state).

A preliminary report summary will be prepared in-house after all the tests needed have been concluded. The in-house report will then undergo 3rd party review for any gaps, errors, corrections up to validation and certification. A discussion with the 3rd party may also lead to further stability analysis extending to an increased embankment height scenario to accommodate future tailings, both for operational and deactivation stages.

At the current there is only around 283,000 m^3 of remaining volume for the containment of tailings inside the pond while maintaining the present freeboard of about 1.7 m based on the existing pond water level compared to the crest level of SE dike.

Figure 24-10 - Remaining Impoundment areas at Existing TSF and *Table 24-5 - Tailings Production Estimate - BYG Pit* below are the approximate lay-out of the pockets of remaining impoundment areas inside the Bekajang TSF pond, and the estimate of tailings expected to be generated from the resumption of mining at the old BYG Pit. Volume of tailings estimated is based on the latest mining schedule and concentrate mass pull-out percentage from the flotation process.

Figure 24-10 - Remaining Impoundment areas at Existing TSF

Year	Units	Year 1	Total Average
Working days	days	203	202.75
Milling rate	t/d	4,000.00	4,000.00
	t/y	811,000.00	811,000.00
Mill grade	gAu/t	3.34	3.34
Recovery %	Au	77.00%	77.00%
Flotation concentrate	t/d	200.00	200.00
	t/y	40,550.00	40,550.00
Production	oz Au/y	67,057.78	67,057.78
To tailings pond	flotation tailings, t/d	3,800.00	3,800.00
	flotation tailings, t/y	770,450.00	770,450.00
	gAu/t	0.81	0.81
	tails slurry, m^3	1,971,601.54	1,971,601.54
	tails solids, m^3	540,673.14	540,673.14
	tails slurry, m^3 (final settled volume)	870,928.56	870,928.56
	ore density, t/m^3	2.40	2.40
	flotation tails density, t/m^3	1.12	1.12
	% solids by weight	35.00%	35.00%

Table 24-5 - Tailings Production Estimate - BYG Pit

At its current state, the remaining 283,000 m^3 capacity of the TSF pond to accommodate total tailings slurry of 870,930 m^3 at settled density is not enough. Certainly, we have to complete all the in-situ soil geotechnical tests first and validate the dam's structural integrity in terms of slope stability analysis, stress and strain analysis as upstream-type construction dam, seepage analysis if any, and safety factor against sliding prior to any decision to utilise it once more as tails impoundment site.

Once the analysis is complete and the findings are positive the option to increase the TSF capacity will be undertaken. If the analysis is not positive then the tailings will need to be accommodated in another TSF, likely at Jugan.

24.1.2.3.2.2. Current Investigation & Analysis of Existing TSF

The on-going soil geotechnical study on the old and deactivated upstream-type Bekajang tailings storage facility (BYG TSF) is focused on its structural integrity. The sites of concern within the impoundment area are the main embankment located in the north and at the far end to the SE is the dyke located within the once Bekajang lake, which eventually became a part of the whole BYG TSF. The dyke is serving as a cut-off structure that blocked the Bekajang Lake adjoining the Jebong Lake to the east.

The objectives of this undertaking is to establish the long term stability of the dam at its deactivated state, where the tailings are assumed to have consolidated (up to a point) since Bukit-Young Mine ceased operating, and if it can function once again as a tailings impoundment site for the Bukit-Young, Bekajang and Krian pits. Additionally, after the analysis it will be determined if the tailings infrastructure can accommodate the additional tailings requirement.

The work involves mainly the in-situ measurement of the soil used as embankment fill for the embankment and the dike. The in-situ measuring instruments are standard penetration test (SPT) up to rockhead depth, rock coring up to 4 metres depth starting from the rockhead depth, installation of standpipe piezometers and inclinometers as monitoring stations, field vane shear test of intercepted soft clays within the soil horizon, and cone penetrometer test. The various laboratory tests of disturbed and undisturbed soils samples and uni-axial testing of selected rock cores is part of the activity.

As from the last week of April 2013 to present, what has been completed to date is the wash-boring of twelve (12) SPTs at the main embankment and one (1) at the southeast dyke, rock coring at up to 4m depth, some field vane shear tests beside SPT locations that returned very low N values at some sections along their holes, installation of piezometers and inclinometers for the monitoring of the phreatic level and lateral movements in the embankment, if any. Currently on-going are the laboratory tests and the CPT tests.

Below in *Figure 24-11 - Existing TSF - In-Situ Measuring Points for TSF Assessment* is a layout of the proposed in-situ measuring points at the main embankment and SE dyke.

Figure 24-11 - Existing TSF - In-Situ Measuring Points for TSF Assessment

Four (4)of the five (5) inclinometers were slotted and keyed-in below 1.0 – 1.5 m the rockhead inside each of the SPTs located along the 2nd berm of the main embankment , and one (1) at the SE dyke. The objective of the inclinometers is to monitor any lateral movement of the embankment with respect to the rock unit beneath. Eight (8) of the total nine (9) piezometers, on the other hand, were fitted inside the SPTs done at the 1st and bottom berms of the main embankment while the other one (1) is located 2 m away from the inclinometer installed at the SE dyke. The piezometers will serve as monitoring points for phreatic surface passing through the embankment materials, if any.

Once all in-situ tests and laboratory testing have been completely done, there will be a semi-detailed if not a thorough in-house assessment on the structural integrity of the deactivated Bekajang TSF, followed by a 3rd party review.

24.1.2.3.3. Waste Landforms

24.1.2.3.3.1. Jugan – Waste Landform

As indicated above around 6.4 Mm^3 of mine waste will be required in the build-up of Jugan TSF. The remaining mine waste, around 3.4 Mm^3, will be stored in an engineered waste dump. The maximum slope on mine waste landforms required by the Malaysian Government is 37° as the angle of repose and the company will treat this as a local face slope, not as the overall. The waste landform will be constructed in three (3) lifts at 10 m height with 5 m berm for each lift.

It will be built in a bottom-up construction method where thin layers of about 0.5 m thick will be vibro-roll compacted up to 0.3 m in height. The loose spoils on the side slopes, on the other hand, will be trimmed at every 4 m height interval, placed on top of the dump, spread out thinly in layers and re-compacted. Vibro-rolling will be with the use of a smooth drum roller or a sheeps-foot where a maximum 95 % Proctor density is to be achieved. Compaction tests will be conducted at every 1 m lift either with the use of a nuclear density meter (NDM) or sand-cone replacement.

All the potentially acid producing mine waste (PAF) arriving on site will be vibro-roll compacted as well that for every stacked-up 1m height, they will be underlain by a meter thick non-acid producing mine waste (NAF). Clay-lining at the bottom of the waste landform (before the build-up) and during deactivation period (after its operating years) will be provided. There will also be clay-lining on the side slopes of the waste dump after the loose spoils have been trimmed and compacted. Progressive re-vegetation of slopes at every required 10m lift and after the clay-lining, as explained, will be carried out.

24.1.2.3.3.2. Bukit-Young – Waste Landform

Around 1.8 Mm^3 of mine waste will be stored in an engineered waste landform proximate to the SE part of the Bukit-Young Pit. Similar to Jugan waste landform, the Bukit-Young waste landform will follow the government stipulated 37° angle of repose for every 10 m lift.

Construction of the waste landform will be via bottom-up method where vibro-roll compaction and re-trimming of the batter slopes as explained above will also be employed. Compaction will be maintained at 95 % Proctor density.

The host rock in Bukit-Young Pit, or in Bekajang area as a whole, is carbonaceous in nature where it is classed by colour predominantly as medium-to-dark grey, argillaceous, crystalline limestone and pale-to-medium grey, fossiliferous, crystalline limestone.

The host rock is treated as non-PAF waste material in general although there was no static geochemistry test done so far in the samples. The sulphides are existent only within silicified ores found along the lithological contacts of shale and limestone and in limestone with intrusive-shale-limestone combination. Along the vein-type ores, on the other hand, in calcite with microcrystalline quartz and vughy quartz, the sulphides are non-observable in most occasions mainly due to the very fine-grain texture of this ore type. The carbonate, stockworks and quartz veins can measure from less than a centimetre thick to around 5 metres. All the sulphidic veins, except on one occasion, measure between 0.1 to 1.0 centimetres thick with sulphide contents ranging from 1 % to 5 %. The single sulphidic vein intercepted in a drillhole measured around 2.75 metres thick and contains 7 % sulphide. All the different ore types, however, will undergo gold beneficiation at the plant where the tailings will end up under water inside the Bekajang tailings pond.

Although the host rock is treated as non-PAF waste material, the 1.4 % sulphides cut-off on NAF applied in Jugan will also be adopted for the Bukit-Young waste characterisation. The same arrangement of waste placement and management practice to be applied in Jugan waste landform will also be adopted in the Bukit-Young waste landform.

24.1.3. Design Recommendations

Under each of the following sub-sections are the design recommendations as supplied by the geotechnical/geomechanical team. These are either designed and as modelled and analysed or are generic principles until such time as they are modelled, analysed and recommended.

24.1.3.1. Open Pit

24.1.3.1.1. Jugan Open Pit

The Jugan Pit will employ varying pit slopes based on the RMR values and structural orientations per sector, i.e. for the E-SE slope and W-NW slope, and RMR values per elevation as the pit progresses to its final economic mining depth. Moreover, the Jugan Pit will be mined in flitches of every 2.5 m height only thus if the actual situation in the field requires any adjustments in the cut slope early on, which eventually affects the pit slope up to a point, can be carried out immediately before a single bench reaches its full height of 15 m. General details of the pit design based on the results of the geotechnical modelling were previously discussed in *Section 19.2.1.3.*

Anywhere between 30 mRL down to 0 mRL, a 40° pit slope will be maintained all throughout the pit. Starting at 0 mRL down to -25 mRL on the W-NW side of the pit, the pit slope will be around 44° starting midway up to final pit boundary. Between -25 mRL down to -55 mRL, the pit slope will adjust anywhere between 44° and 48°. By -55 mRL down to the bottom of the pit at RL -145m, the 48° pit slope will be maintained up to the W-NW pit limit boundary. The geological structures at this side of the pit are generally steeply dipping towards the slopes, which is favourable for stability, though some of the structures may change in dip-direction at depth since they are labelled as conjugate faults. We expect some slight adjustments in terms of cut slope and direction of the cuts (i.e. slightly acute/obtuse to the strike direction of the structure or ground support by cable-bolting) when the structures dip away from the slopes.

On the E-SE sector, on the other hand, starting from the surface at approximately 30 mRL down to -35 mRL, a pit slope of 40° will be maintained at the final pit limit. The reason for adopting this is mainly due to the very poor RMR values at this sector. Around 0 mRL to -35 mRL though, midway towards the final pit limit, the pit slope varies from 40° to 44° depending on the RMR values of the rockmass encountered. From -35 mRL down to -65 mRL at the pit limit boundary, the pit slope will be highly variable between 40° to 44°. By -65 mRL to -85 mRL, the pit slope will vary from 44° to 48°. From -85 mRL down to the pit bottom, a 48° pit slope will be maintained. The geological structures at this side of the pit are generally steeply dipping away from the slopes, which is “conditionally” unfavourable for stability. When the cut slope is steeper than the structure’s dip and when the slopes are excavated parallel to the strike of the structures then slope stability is compromised. Actual mining at this sector as carried out per 2.5 m high flitch will be slightly acute or obtuse to the strike direction of the structures. Any required slope improvements as we go along, e.g. slope adjustment, ground support by cable-bolting, and provision of weep holes will be carried out accordingly.

24.1.3.1.2. Bukit-Young Open Pit

The current Bukit-Young Pit design is based on a 10 m high slope x 5 m wide berm and maintains a consistent 47° pit slope all throughout from RL 30m down to RL -50m. This design was based on the previous pit optimisation run that uses the overall 66.5° cut slope and 47° pit slope. However, before the economic model was made for the Bukit-Young pit optimisation we already have a general idea on the strength of the rock masses and their RMR ratings thus a steeper 66.5° cut slope was initially adopted.

A design revision using a 15 m high slope x 5 m wide berm is underway. The difference though from the current model will be as follows:

From RL 30m down to RL 0m the pit slope will be at 44° and from RL 0m down to the bottom at RL -50m the pit slope will be 48°.

The major geological structures in the Bekajang/Krian area where the Bukit-Young Pit is situated are striking NNE and steeply dipping towards NW. We expect that the trends of these structures are consistent at depth and appear to be non-conjugate faults. As mentioned above, the Bekajang/Krian project is still three (3) plus years down the line. The semi- to detailed surface structural mapping and subsurface structural modelling in this area will be part of the project schedule into the near future. By then, we would have a better interpretation and more vivid presentation of these geological structures in question and have more confidence in incorporating them in detail into the whole geotechnical modelling.

Similar to the mining in Jugan, actual mining will be carried out per 2.5m high flitch and as such, any required slope improvements as we go along, e.g. slope adjustment, adjustment in direction of the slope cut, ground support by cable-bolting, and provision of weep holes will be carried out accordingly.

24.1.3.2. Tailings Storage Facilities

24.1.3.2.1. TSF – Proposed

The Jugan TSF has been designed in-house to internationally acceptable standards (Australian New Zealand Committee on Large Dams (ANCOLD) Guidelines) to provide a facility for the safe and environmentally acceptable containment of process tailings. To further ensure its structural integrity, the company will engage a 3rd party review. An internationally and nationally recognised dam designer and construction consulting company will be selected to do the review and validation of the proposed TSF. Furthermore, the consultant will be involved during the construction process, providing construction advice along the way, and ensuring adherence of the local dam contractor to QA/QC standards. Thereafter, prior to the operation of the facility, a certification would have to be issued by the consulting company.

The TSF design has also taken into account the requirements for progressive rehabilitation by grassing during its operational years and towards the end of mine life, and its reclamation and further re-greening during its deactivation and closure stage.

The criteria considered in the design of the TSF are the following aspects:

- Structural integrity or physical stability both on static condition and pseudo-static loading coming from the nominal 475-year earthquake event in Sarawak;
- Groundwater protection;
- Maximised drying or desiccation of tailings through an effective use and placement of appropriate blanket drain materials combined with an open spillway system;
- Maximised tailings consolidation;
- Maximised use of borrow materials from the TSF basin;
- Maximised use of mine wastes as added source of borrow materials on a cost-effective manner ; and
- Safety against overtopping based on a peak 100-year rainfall event.

24.1.3.2.2. TSF – Dam Seepage Analysis

Seepage analyses were carried out for each of the three (3) stages of the TSF to estimate the seepage through the dam for the purpose of sizing and positioning of the blanket drain layer, as well as examining the foundation seepage and the effectiveness of the HDPE liner that is intended to be placed from the basin's floor to 3.0 metres above it, i.e. 20 mRL to 23 mRL.

The seepage analyses were based on the hydraulic conductivities of the foundation strata, embankment fill and tailings derived from external references as well as dam engineering experience with relatively quite similar materials used in numerous tailings dam constructions experience in the recent past. Moreover, some auger drilling was done in the area for the purpose of exploration where samples from soil profile B horizon (clay derived from weathered shale) were submitted for soil mechanics laboratory analyses. The depth/thickness of the intercepted weathered B horizon, where A horizon is the topsoil, was about 3.0 metres thick extending towards the transition zone (C horizon) prior to the Shale bedrock.

The seepage analysis and estimates on various stages were done using a finite element application, SEEP/W. The analysis provided a picture of the movement of the seepage starting from the tailings that contain around 65 % water by weight upon deposition through the embankment and finally through the gravel blanket drain conduit. The pore-water or pressure head distribution throughout the discretised area was also provided for each analysis. The seepage analysis indicated that very minimal to nil amounts of seepage may get through the foundation, mainly because of the presence of the non-permeable HDPE liner coupled by the non-permeable nature of the foundation and the placement of the gravel blanket drain relatively near to seepage sources.

For Stage 1, a one (1) metre thick blanket drain extends from the bottom at an idealised elevation of 20 mRL and will branch-out up to 25 mRL where it is just distanced 10 m to the tailings interface with the upstream embankment outline. The same arrangement will be used for Stage 2 since it will have the same height as Stage 1, with volume of impoundment increasing laterally due to the TSF aerial extension. Stage 3, on the other hand, will be provided with an additional 1m thick branch up to 35 mRL, also just 10 m close to the tailings interface. All of the finger drains' branching up are connected to the 1.5 m thick blanket drain main line at 20 mRL. *Figure 24-12 - TSF: Stage 1 - Finite Element Model for Seepage Analysis* to *Figure 24-20 - TSF: Stage 3 - Seepage Flow Vectors* shows the section of the seepage analyses for Stages 1 and 3. In the figures, the flow vectors are magnified several thousand times to show how seepage flows from high-pressure low permeability zones to low-pressure zones and high permeability conduits (gravel blanket drain), and how the seepage or groundwater height goes down thus effectively lowering the pore-water pressure in the embankment to increase the dam's structural stability.

Furthermore, the flow vectors themselves also provide an indication as to how the tailings are desiccated. It should be noted, however, that the presence of an open spillway system at 26 mRL (for Stages 1 and 2) and at 38 mRL (for Stage 3), to handle excess supernatant discharge, and mainly, to prevent dam overtopping from rainfall run-off was not incorporated into the analysis. It is believed that the target settled density of 1.70 t/m3 is achievable based on the results of the seepage analysis. Drying of the tailings from 65 % by weight water content, or about 83 % by volume, down to about 34.6 % by weight water content can be achieved not only because of the presence of a gravel blanket drain but also due to the presence of an open spillway system at each stage in the dam construction and during operation.

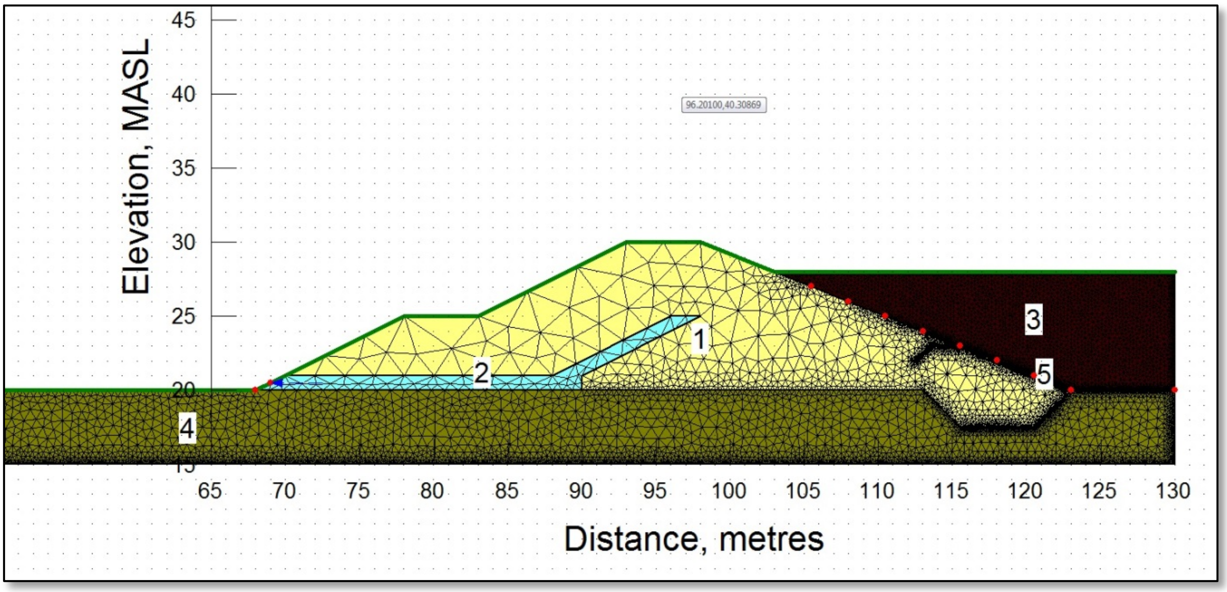


Figure 24-12 - TSF: Stage 1 - Finite Element Model for Seepage Analysis

Figure 24-12 - TSF: Stage 1 - Finite Element Model for Seepage Analysis above shows the different zones; Zone 1 – TSF embankment (light yellow), Zone 2 – Gravel blanket drain (aqua blue), Zone 3 – Stage 1 tailings (dark brown), Zone 4 – Foundation (greenish brown) , Zone 5 – HDPE liner (black). Red dots represent the boundary conditions outlining where varying hydrostatic pressure at depth is applied. The red dots at the left-hand side of the model, located at the toe of the downstream side where the blanket drain discharge section is located indicate the seepage face.

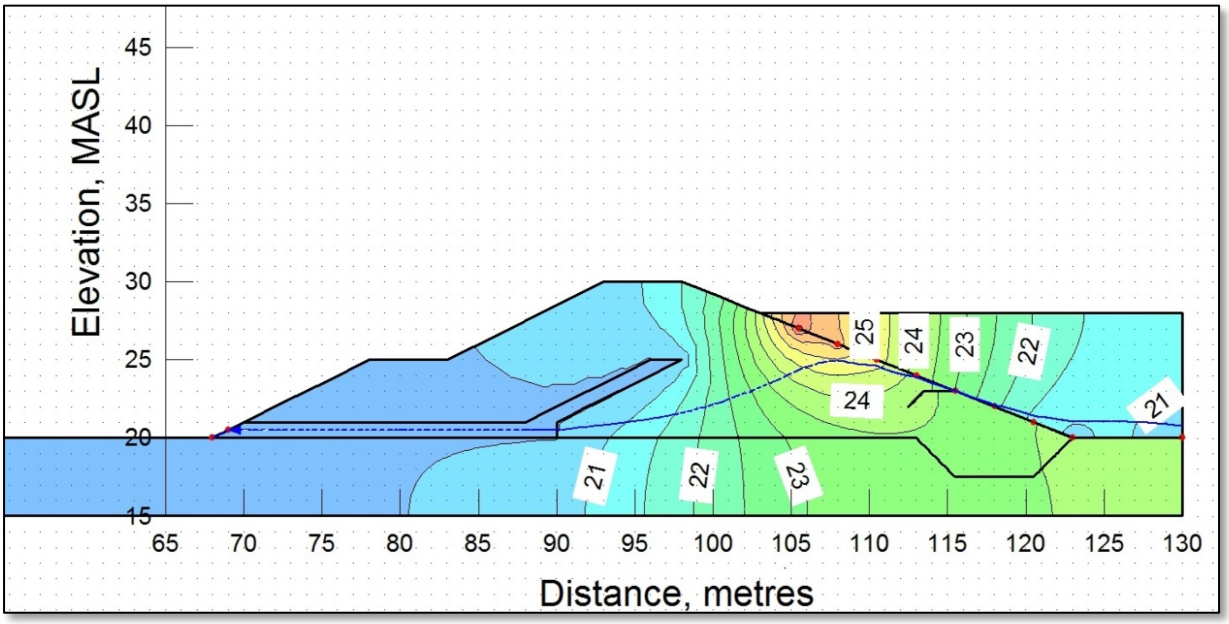


Figure 24-13 - TSF: Stage 1 - Seepage Analysis Results Based on Head

In *Figure 24-13 - TSF: Stage 1 - Seepage Analysis Results Based on Head* above, the phreatic line or groundwater seeping into the embankment towards the gravel filter drain is indicated in dashed outline (blue colour).

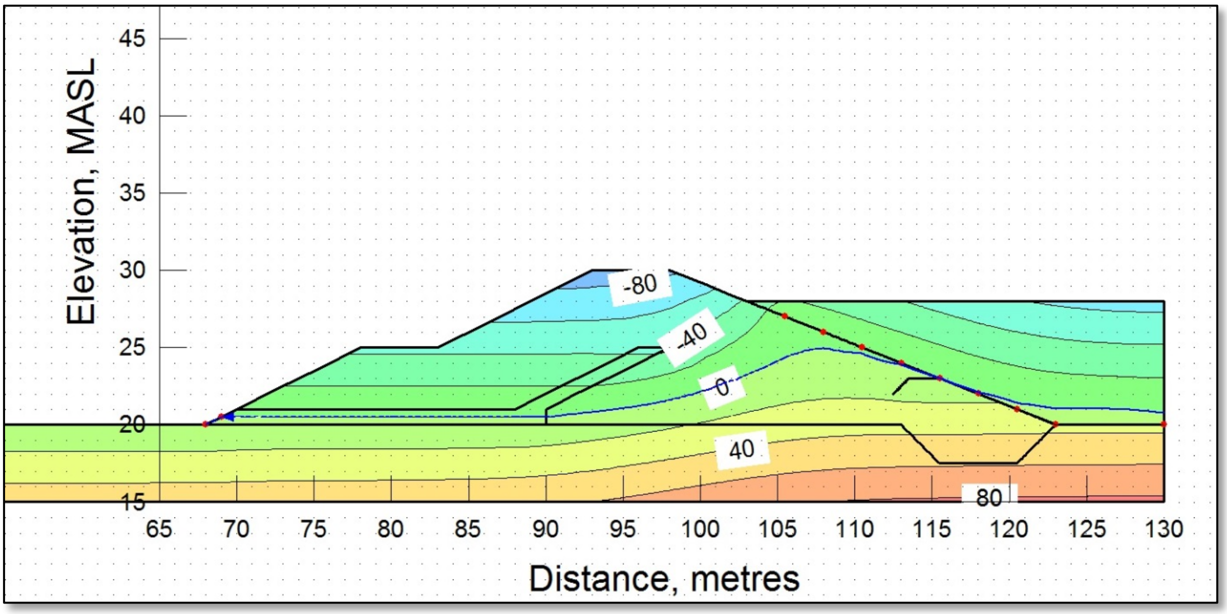


Figure 24-14 - TSF: Stage 1 - Seepage Analysis Results Based on Pressure (kPa)

In *Figure 24-14 - TSF: Stage 1 - Seepage Analysis Results Based on Pressure (kPa)* above, the zero (0) pressure indicates where the phreatic/groundwater level is located.

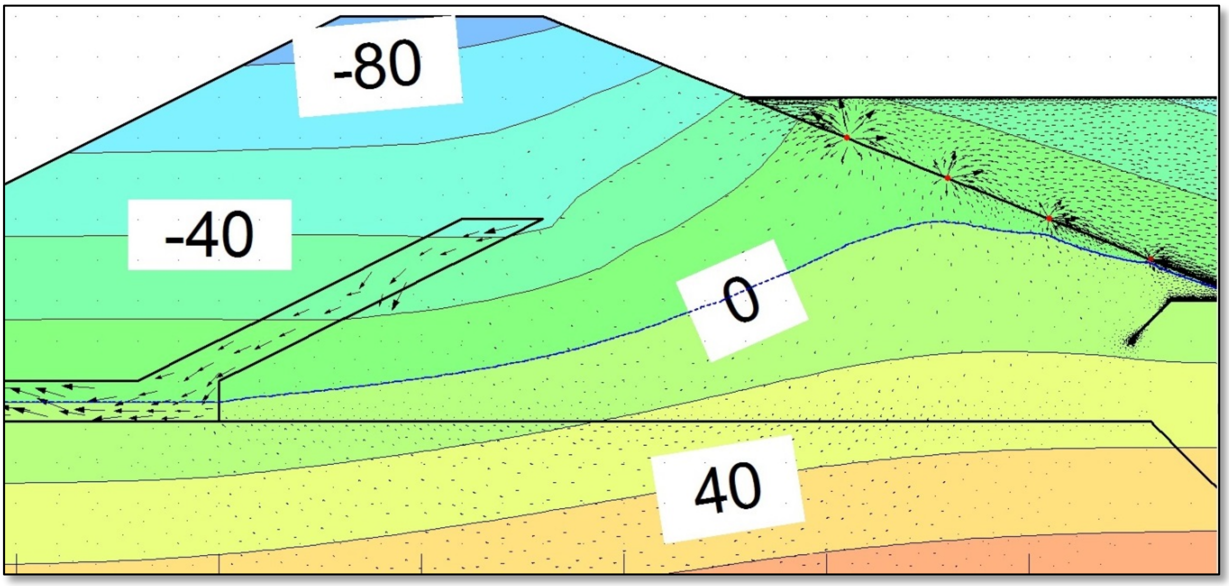


Figure 24-15 - TSF: Stage 1 - Seepage Flow Vectors

In Figure 24-15 - TSF: Stage 1 - Seepage Flow Vectors above, the seepage flow vectors that are directed from high pressure towards the zero (0) pressure where the gravel filter drain is located effectively lowering the groundwater table in the process. The vectors are magnified 300,000x where 1mm = 1.67 x 10^{-6} m/sec.

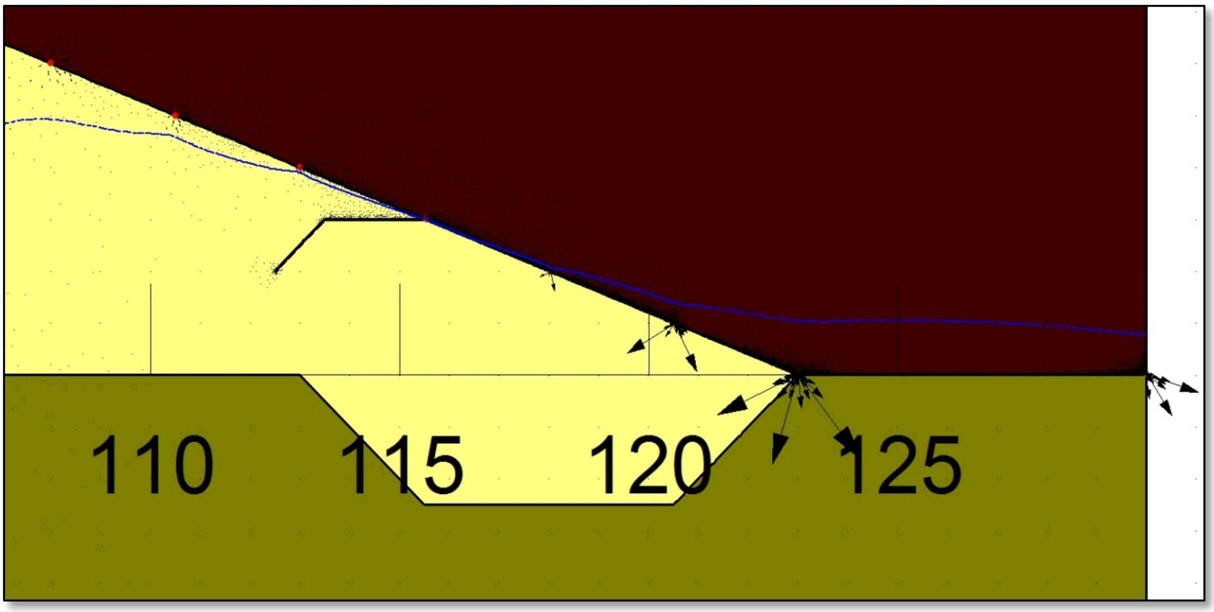


Figure 24-16 - TSF: Stage 1 - Seepage Flow Vectors at HDPE Liner

In *Figure 24-16 - TSF: Stage 1 - Seepage Flow Vectors at HDPE Liner* above, shows the seepage flow vectors supposed to permeate through the HDPE liner. The absence of vectors beneath the liner into the foundation basin and convection thereafter into the gravel blanket drain indicates zero seepage into the basin exactly beneath the liner. The vectors are magnified 100,000x where 1mm represents a seepage rate of 5.00 x 10^{-6} m/sec.

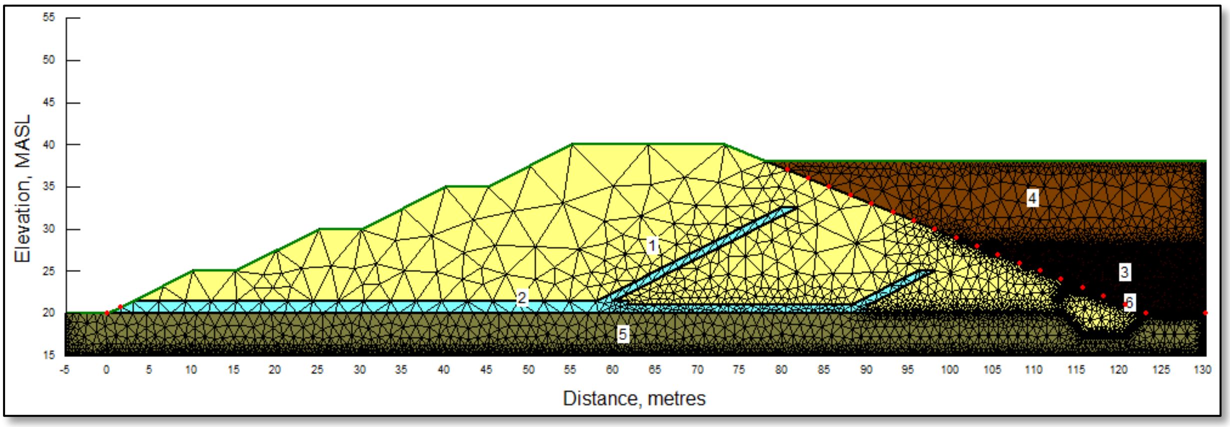


Figure 24-17 - TSF: Stage 3 - Finite Element Model for Seepage Analysis

In *Figure 24-17 - TSF: Stage 3 - Finite Element Model for Seepage Analysis* above, shows the different zones; Zone 1 – TSF embankment (light yellow), Zone 2 – Gravel blanket drain (aqua blue), Zone 3 – Stage 1 tailings (dark brown), Zone 4 – Stage 3 tailings (light brown), Zone (5) – Foundation (greenish brown) , Zone 6 – HDPE liner (black). Red dots represent the boundary conditions outline where varying hydrostatic pressure at depth is applied. The red dots at the left-hand side of the model, located at the toe of the downstream side where the blanket drain discharge section is located indicate the seepage face.

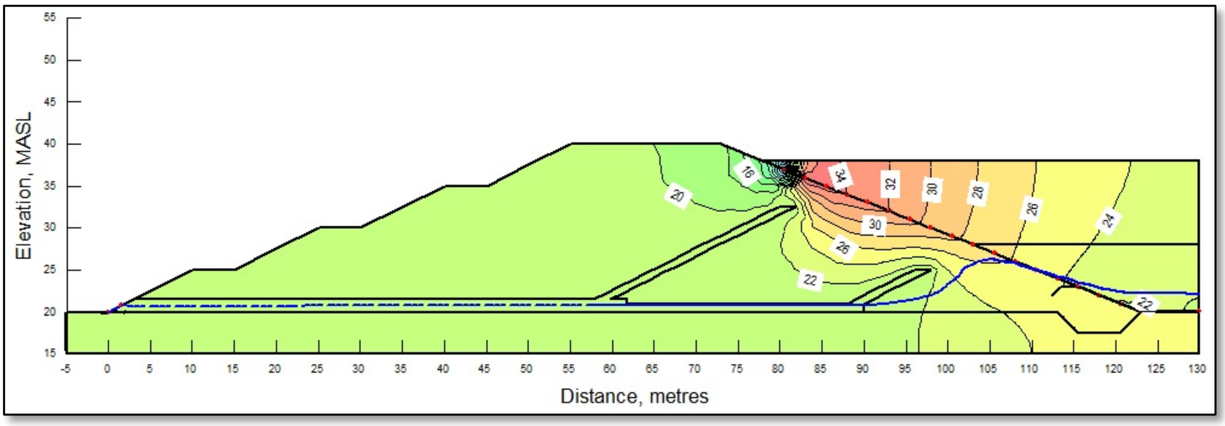


Figure 24-18 - TSF: Stage 3 - Seepage Analysis Results Based on Head

In *Figure 24-18 - TSF: Stage 3 - Seepage Analysis Results Based on Head* above, the phreatic line or groundwater seeping into the embankment towards the gravel filter drain is indicated in dashed outline (blue colour).

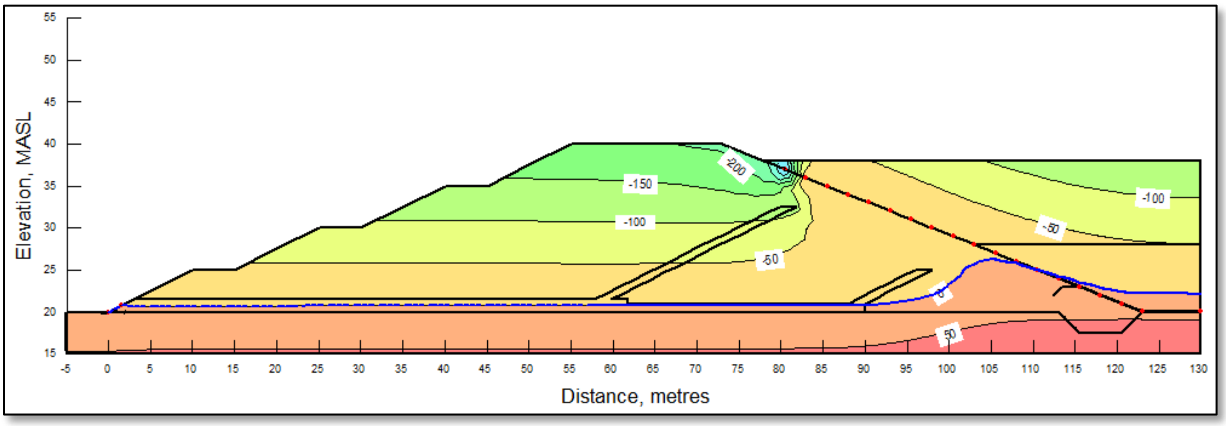


Figure 24-19 - TSF: Stage 3 - Seepage Analysis Results Based on Pressure (kPa)

In Figure 24-19 - TSF: Stage 3 - Seepage Analysis Results Based on Pressure (kPa) above, the zero (0) pressure indicates where the phreatic/groundwater level is located.

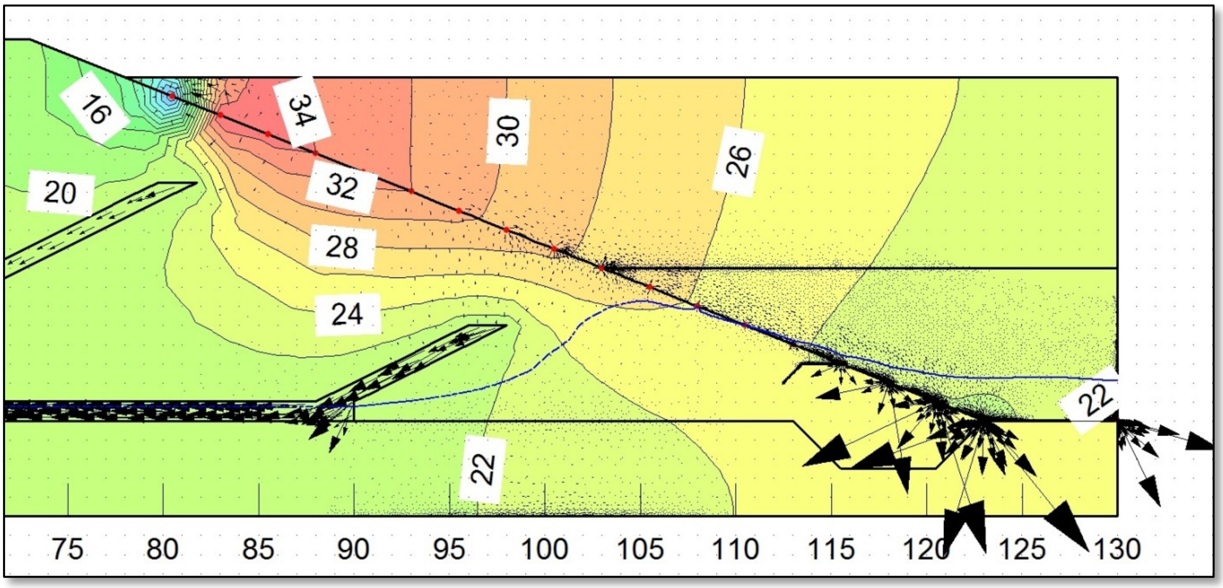


Figure 24-20 - TSF: Stage 3 - Seepage Flow Vectors

In *Figure 24-20 - TSF: Stage 3 - Seepage Flow Vectors* above, the seepage flow vectors directed from high head to the lowest head (20 mRL), where the gravel filter drain is located effectively lowering the groundwater table in the process. The vectors are magnified 250,000x where 1 mm means a seepage rate of 2.0 x 10^{-6} m/sec. The absence of flow vectors beneath the HDPE liner into the foundation basin and absence of convection within that discrete section indicate zero seepage into the basin exactly beneath the liner.

On the other hand, those very slow moving vectors that we see some distance away from the liner and permeating downwards from the tailings and seemingly seeping into the foundation are actually convecting upwards again towards the blanket drain, where the low head (20 m) or zero pressure zone is located. Thus almost all of the seepage from the tailings, if not all, are caught up by the filter drain.

24.1.3.2.3. TSF – Dam Stability Analysis

The total pressure head values and the resulting groundwater level from the SEEP/W were incorporated into the limit-equilibrium slope stability analysis software SLOPE/W.

By examining the results of the slope stability analyses given in the succeeding pages, it may be concluded that the TSF design complies with all the Acceptable Factors of Safety stipulated under the Australian New Zealand Committee on Large Dams (ANCOLD) Guidelines on Tailings Dam Design, Construction and Operation. The recommended minimum factors of safety (F of S) for tailings dams at every perceivable loading condition during dam operation and post-operation are as follows:

- Steady seepage at high pool level – 1.5
- Rapid drawdown from pool level – 1.2 (see Note 3)
- Earthquake (high pool for downstream slope, or at intermediate pool for upstream slope – 1.1 for pseudo-static analysis (see Note 4 and Section 6.12)
- Construction conditions, either slope 1.3 or 1.1 (see Note 5)

Note 1 – Values are quoted by the National Research Council (1983) from the US Corp of Engineer's requirements.

Note 2 – The Bishop Simplified method (or equivalent, i.e. Spencer, Fredlund and Krahn, Janbu Generalised, Morgenstern and Price, and Sarma methods) must be used.

Note 3 – F of S for undrained analysis only. If analysed using pore pressure from transient seepage analysis, use an F of S of at least 1.4.

Note 4 – The pseudo-static analysis is only for a preliminary screening evaluation of the stability condition. US Corp of Engineers suggest F of S of 1.1 post-earthquake using post-liquefaction strength.

Note 5 – If saturated soil parameters are assumed.

Note 6 – Whichever gives the more critical situation.

Under Note 6, particularly in section 6.12, the Operating Basis Earthquake (OBE) for tailings dams should generally be governed by the following criteria in terms of dam size and hazards it pose to the environment, subject to ANCOLD 1998:

- A one (1) in 50 Annual Exceedance Probability (AEP) event for Low Hazard Dams – 2.00 % probability exceedance in any given year
- A one (1) in 100 AEP event for Significant Hazard Dams – 1.00 % probability of exceedance in any given year
- A one (1) in 1,000 AEP event for High Hazard Dams – 0.10 % probability of exceedance in any given year.

The US Global Seismic Hazard Assessment Program (GSHAP) provides expected earthquake acceleration for countries around the world over a return period of 475 years. The expected resulting peak ground acceleration (PGA) in Sarawak originating from any nearby earthquake generators is in the range or 0.40 m/s^2 to 0.80 m/s^2, or between 0.04 g to 0.08 g. The 475-year return period PGA range supplied by GSHAP means that there is a probability of exceedance of 0.21 % in any given year, or i.e. there is a chance that a PGA in any of these values will occur in the area at any given year.

The Maximum Credible Earthquake (MCE) in consonant to the OBE requirement under ANCOLD, or the Design Basis Earthquake (DBE) used for the Jugan TSF is based on the PGA values provided in the GSHAP. For the Jugan TSF, a 0.10 g was adopted for the slope stability analysis which is slightly greater than the maximum 0.08 g in the GSHAP. By statistical and hazard category in the ANCOLD 1998 criteria for OBE, the Jugan TSF's slope stability design assessment is based between the requirements for Significant Hazard Dams and High Hazard Dams.

Moreover, well compacted embankment dams constructed in clayey fill and rockfill are generally resistant to earthquake shaking, especially when constructed as downstream-types like the Jugan TSF will be built upon. It is the upstream-type of dams built or standing on hydraulically-placed sand fills that require careful design and detailing to withstand earthquake loading as they are susceptible to ground shaking because of their inherent less cohesive foundation properties where liquefaction occurs easily during earthquakes.

The soil parameters used in the stability analysis of the Jugan TSF and the F of S results in the upstream and downstream embankment sectors at static and pseudo-static conditions are provided in the next pages, in *Table 24-6 - TSF: Material Properties for Stability Analyses* and *Table 24-7 - TSF: Sectors and Factors of Safety* plus *Figure 24-21 - TSF: Stage 1 Downstream Sector F of S under Steady-State Seepage in Static Conditions* to *Figure 24-30 - TSF: Stage 3 F of S in Rapid Drawdown Condition*. For the analysis on Stage 3, however, the tailings impounded in Stage 1 is assumed to have a settled density of 16.67 kN/m^3 due to its relatively drier state as a result of its self-weight, effects of surface drying and the effects of the draining out of seepage through series of underdrain gravel blanket drains.

Material properties							
Type	Model	Unit weight, kN/m^3	φ vertical	φ horizontal	c vertical, kPa	c horizontal, kPa	τ/σ ratio
Embankment	Mohr-Coloumb	23.30	25.00	25.00	30.00	30.00	
Tailings	Shansep (overburden)	12.55					0.75
Pond water	No strength	9.81					
Foundation	Anisotropic	22.00	25.00	20.75	30.00	24.00	
Blanket drain	Mohr-Coloumb	22.00	34.00	34.00	34.00	34.00	
HDPE liner	Impenetrable	-	-	-	-	-	-

Table 24-6 - TSF: Material Properties for Stability Analyses

Dam stages	Sector	Safety factor
Stage 1 and 2	Downstream	2.093
	Upstream	5.452
	Downstream at 0.1g PGA	1.583
	Upstream with 0.1g PGA	3.133
	Rapid drawdown	2.016
Stage 3	Downstream	1.815
	Upstream	6.133
	Downstream with 0.1g PGA	1.393
	Upstream with 0.1g PGA	3.638
	Rapid drawdown	1.719

Table 24-7 - TSF: Sectors and Factors of Safety

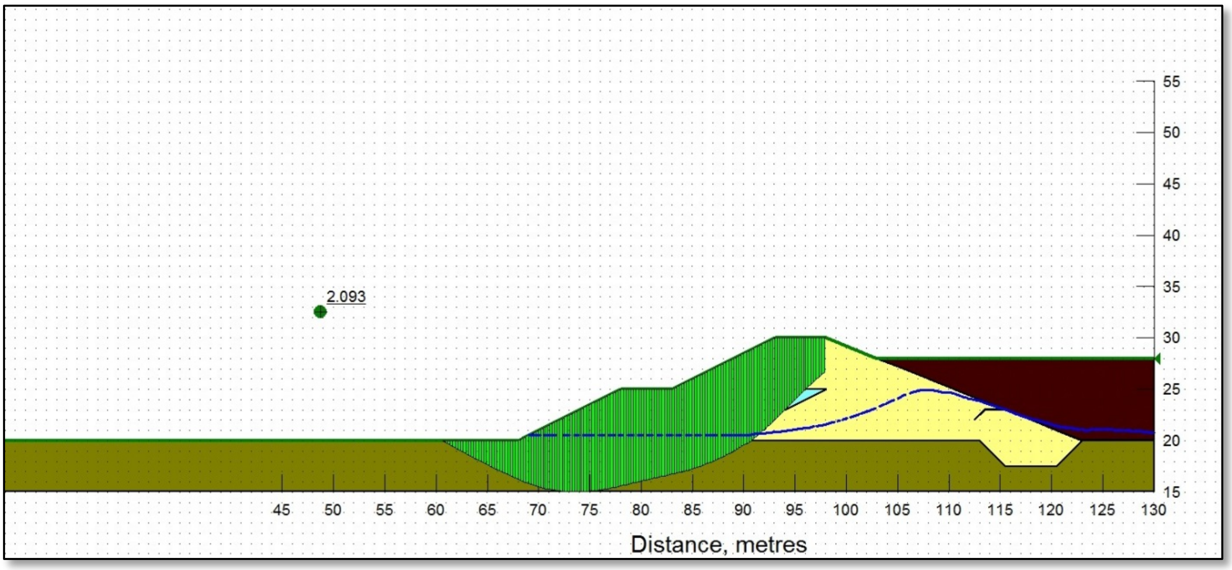


Figure 24-21 - TSF: Stage 1 Downstream Sector F of S under Steady-State Seepage in Static Conditions

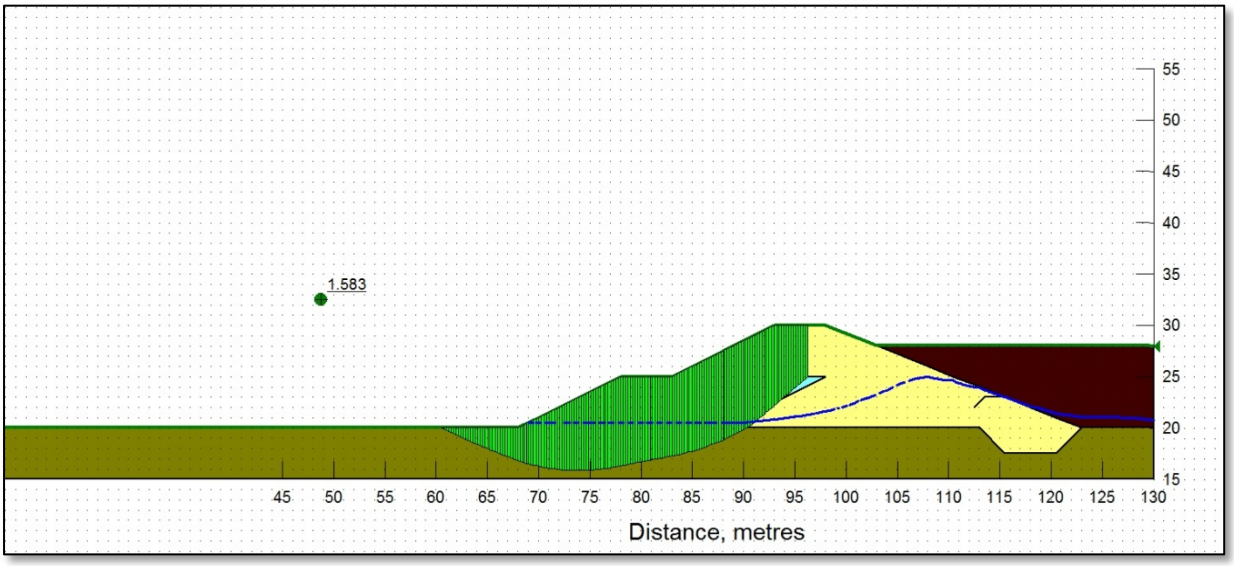


Figure 24-22 - TSF: Stage 1 Downstream Sector F of S under Steady-State Seepage in Pseudo-Static Conditions of 0.1g

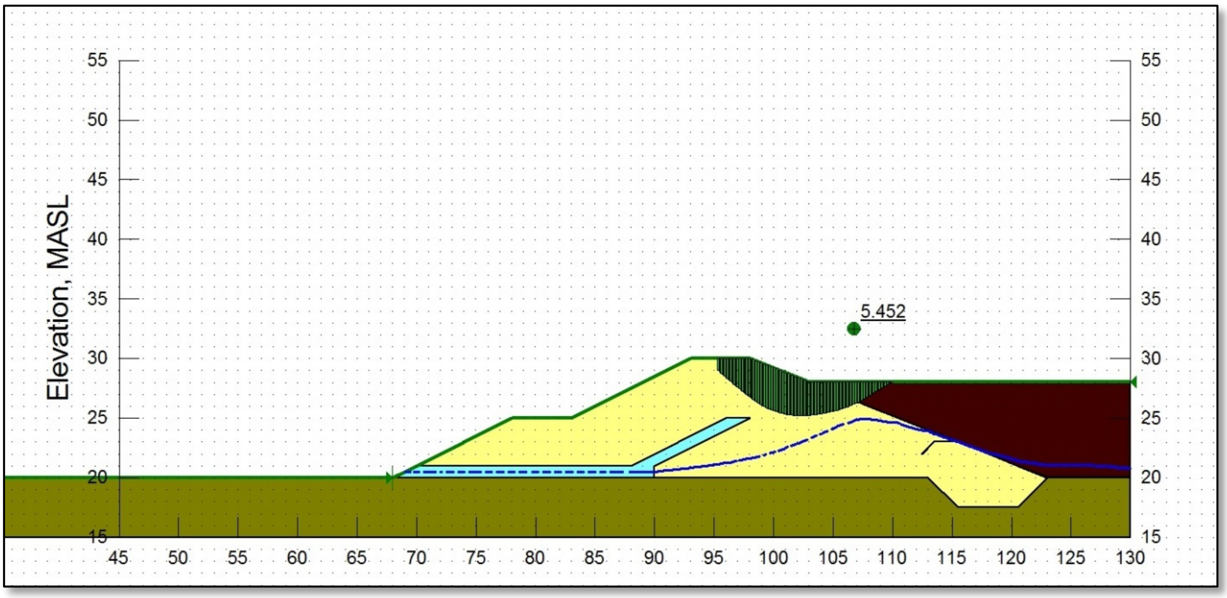


Figure 24-23 - TSF: Stage 1 Upstream Sector F of S under Steady-State Seepage in Static Conditions

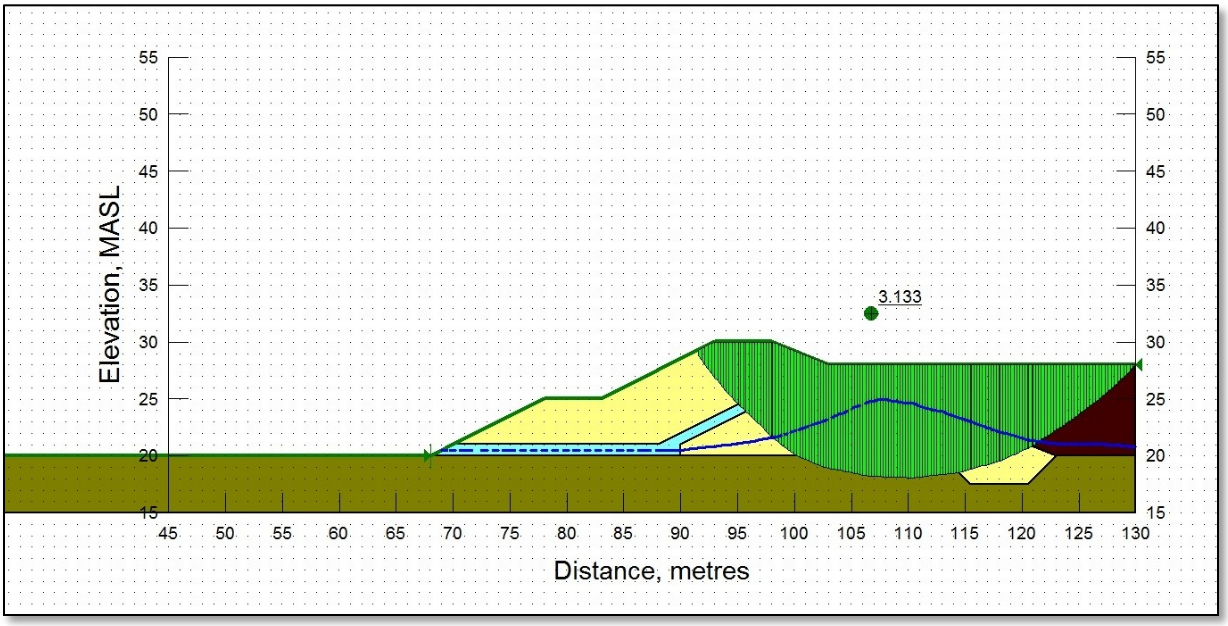


Figure 24-24 - TSF: Stage 1 Upstream Sector F of S under Steady-State Seepage in Pseudo-Static Conditions of 0.1g

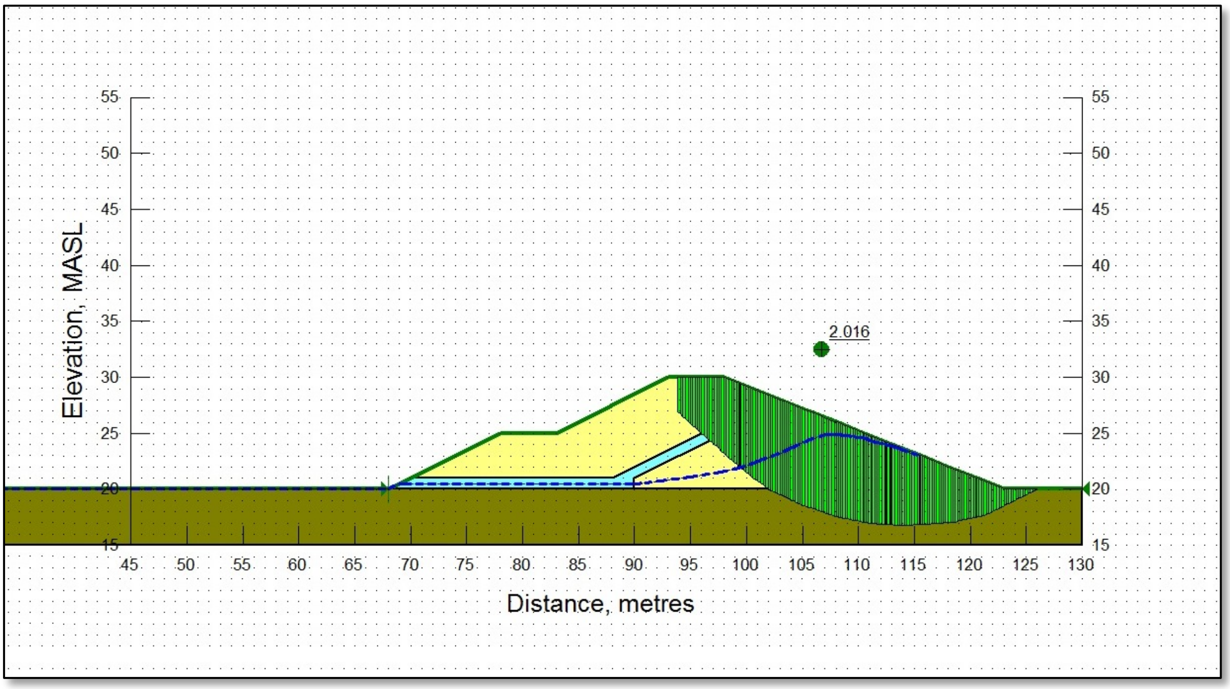


Figure 24-25 - TSF: Stage 1 F of S in Rapid Drawdown Condition

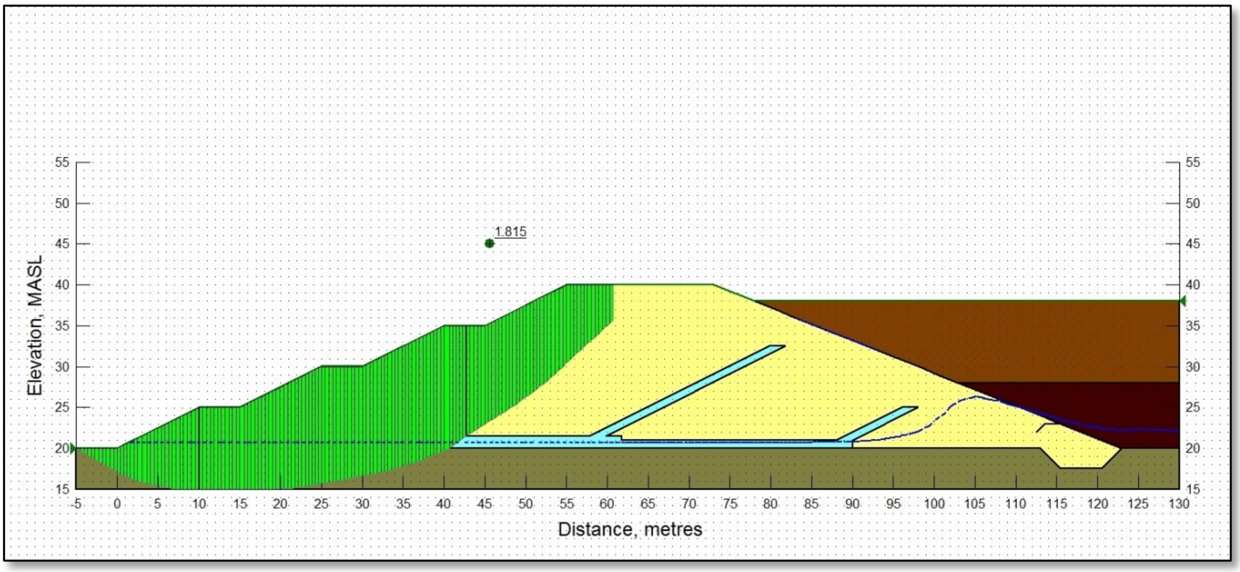


Figure 24-26 - TSF: Stage 3 Downstream Sector F of S under Steady-State Seepage in Static Conditions

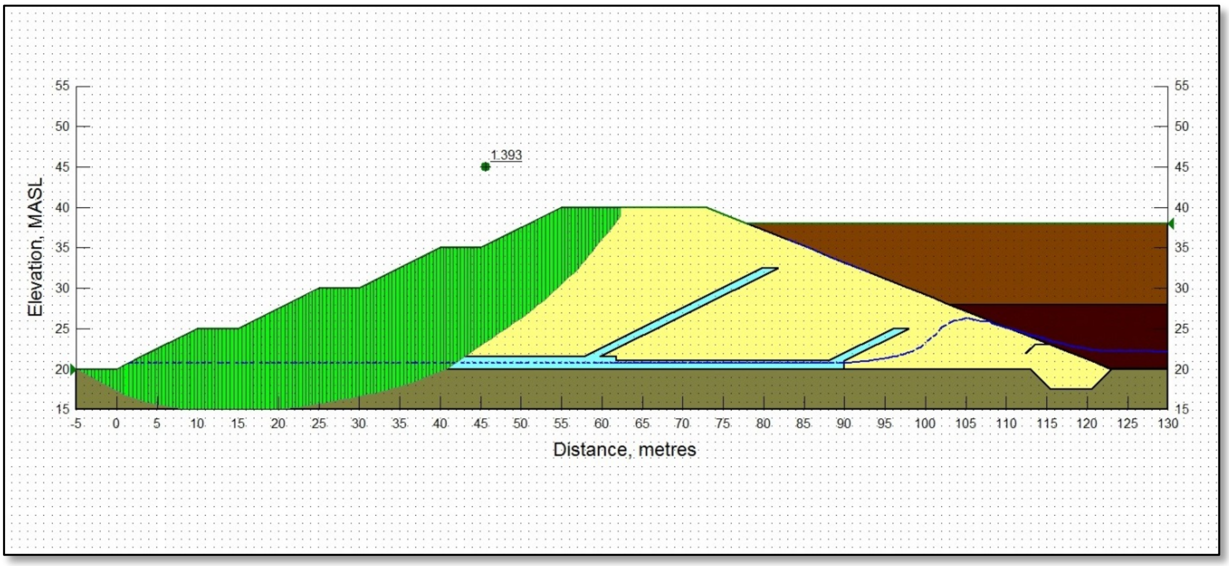


Figure 24-27 - TSF: Stage 3 Downstream Sector F of S under Steady-State Seepage in Pseudo-Static Conditions of 0.1g

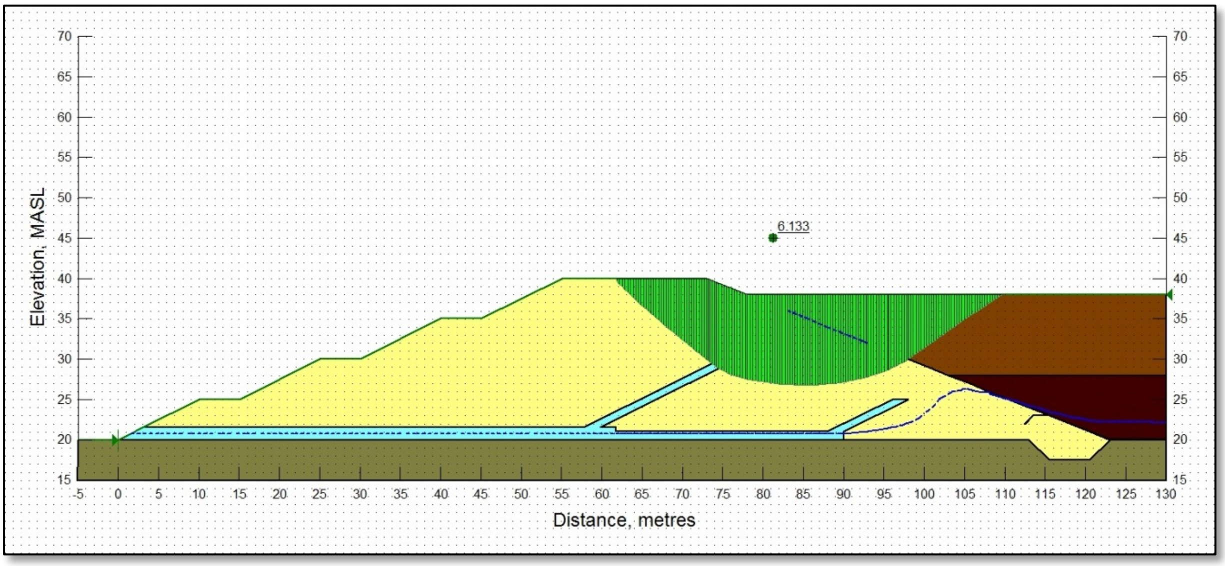


Figure 24-28 - TSF: Stage 3 Upstream Sector F of S under Steady-State Seepage in Static Conditions

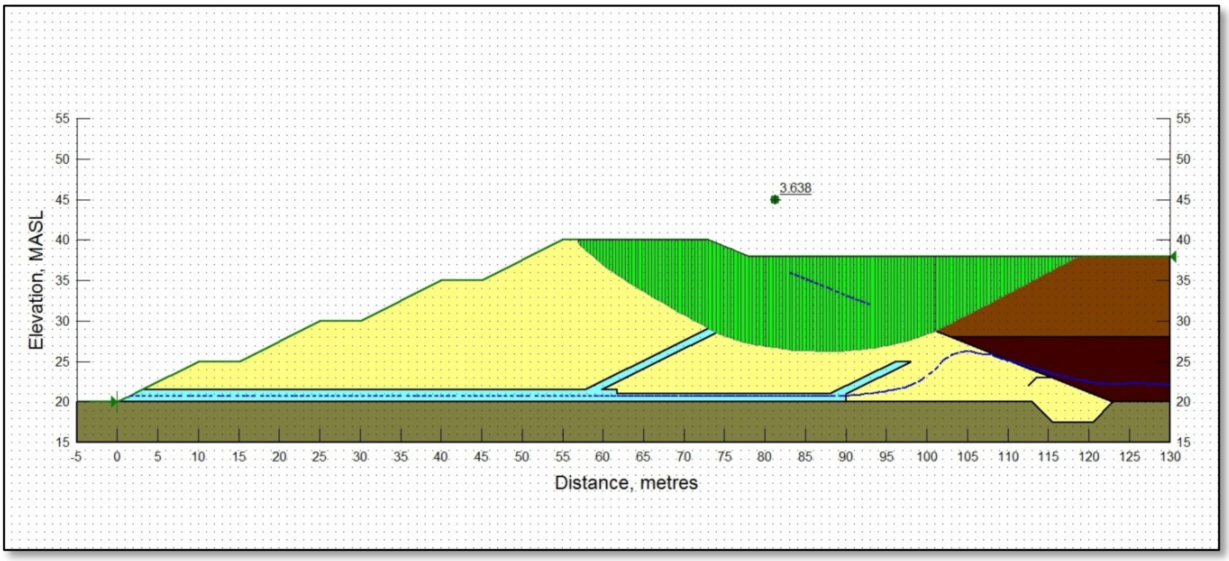


Figure 24-29 - TSF: Stage 3 Upstream Sector F of S under Steady-State Seepage in Pseudo-Static Conditions of 0.1g

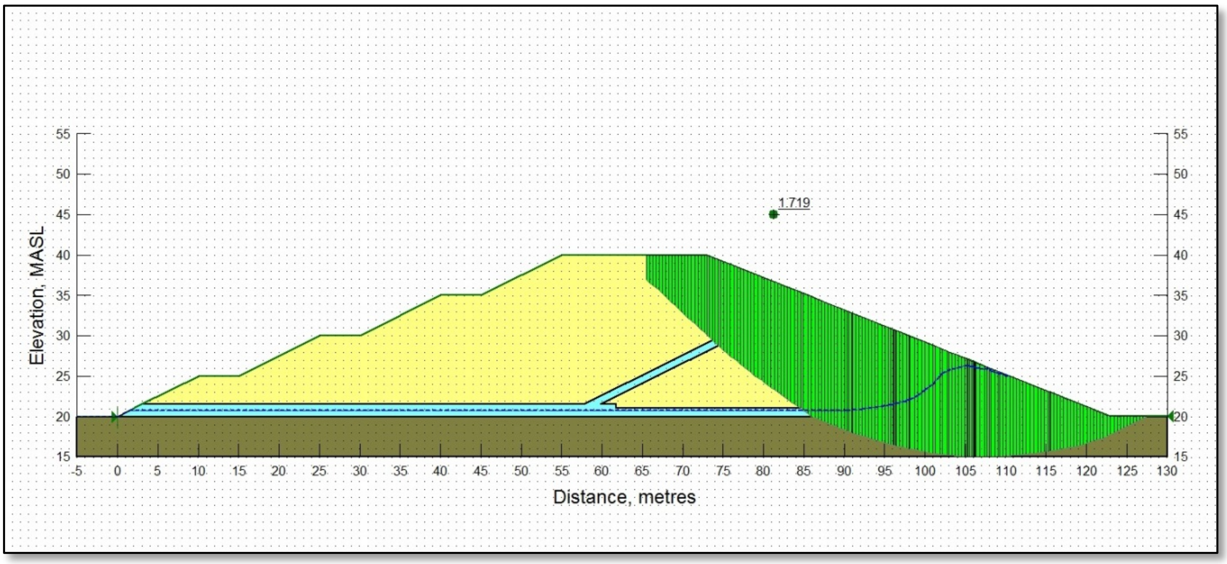


Figure 24-30 - TSF: Stage 3 F of S in Rapid Drawdown Condition

24.1.3.2.4. TSF – Soil & Rock Geotechnical Investigation Required

The following tests are central to any TSF detailed design and construction and are precursor to any foundation engineering design. These are deemed to be the most applicable for our objective of constructing a tailings dam in Jugan that is idealised to be also a water-retention structure throughout its operational stage.

<u>Soil Field Tests:</u>

1. Standard Penetrometer Test (SPT) up to zone of refusal
2. Collection of undisturbed samples by SPT
3. Cone Penetration Test or CPT (in selected areas following the SPT)
4. Field vane shear test on soft soils encountered by SPT
5. Field permeability test on soil
6. Test pitting in selected areas
7. Trenching – to observe preferential seepage pathway, if any, and identify unsuitable materials beneath the TSF, such as soft materials, swelling clays and potential abnormal constituents
8. Rock coring up to 4m deep, if intercepted
9. Rock Quality Designation (RQD)
10. Field permeability test on rockmass intercepted (Packer Test)

Laboratory Test for Soil, Drainage Blanket, and Mine Waste:

a) Atterburg limits (Liquid Limit, Plastic Limit, Plasticity Index)
b) Shrinkage Limit – to establish Shrinkage Index (Liquid Limit – Shrinkage Limit)
c) Void ratios at Liquid Limit and Shrinkage Limit
d) Grain size distribution
e) Permeability of clay and fill

f) Permeability of drainage blanket material where D50 = 50mm
g) Specific gravities of fill, clay and blanket drain materials
h) Sulphate soundness test for blanket drain material
i) Pinhole and Emerson dispersion tests
j) X-ray diffraction
k) Tri-axial with pore pressure
l) Unconfined Compressive Strength (UCS)
m) Absorption test
n) Petrographic analysis

Laboratory Test for Rock Cores:

I. Specific gravity
II. Unconfined Compressive Strength

Others tests:

- Detailed topographic survey
- Condemnation/sterilisation drilling on site
- Mapping of any exposed geological structures

24.1.3.2.5. TSF – Existing

As mentioned above, the possible resumption of the old Bekajang tailings dam to accommodate any future tailings from the operation of Bukit-Young Pit, the Bekajang and Krian deposits is subject to the result of the on-going validation on its structure at its current state (deactivated state). Other than this, there are also issues on the following:

- How stable the dam will be to accommodate more tailings into the future;
- How stable it will be should we raise it up granting that it has no longer enough capacity;
- It was built by the previous company (Bukit-Young Gold Sdn. Bhd.) as an upstream-type where each lifts are sitting on top of the tailings, and it may well remain as such if we raise it up into the future;
- There may be huge land issues should we extend the embankment towards the downstream side, or if we shift to a downstream-type embankment since a community below the facility (a private housing complex or subdivision) already exists;
- Issues on transport of new pollutants in the form of seepage – the old TSF was built without a filter drain to handle any seepage permeating through and underneath the embankment and the foundation is generally underlain by karst topography.

The in-situ soil geotechnical study on this old TSF is still on-going.

24.1.3.3. Other/Miscellaneous

24.1.3.3.1. Plant Site

The proposed plant in Jugan will be situated on top of a hill north of the Jugan Pit. The objective of this being located in a slightly higher elevation is to, among others:

- Lesser requirements in terms of foundation engineering specifications since the topsoil depth is lesser compared to when the plant is sited in flat low-lying area where the same is known to be a floodplain. Moreover, the rock horizon is not too deep at the top of the hill and it makes a better foundation than the usual thick topsoil horizon with some presence of high liquid limit clays found in the low-lying areas around Jugan;
- The groundwater in the hills is relatively deeper compared to the low-lying area within the Jugan mining complex that is just 450m away towards the east where the major river system, Sunggai Sarawak Kanan, channels through;
- Lesser excavation volume of unwanted materials since the topsoil is not as deep compared to the topsoil horizon found usually in low-lying lying areas;
- A relatively higher area provides better security measures in terms of overseeing, monitoring management, and the siting provides added deterrence against pilferage and theft;
- Lesser tails slurry pumping cost to the TSF although there will be some cost for pumping of return water.

Prior to the implementation of any detailed foundation design and structural engineering and construction related to the plant's erection, in conjunction with the plant site preparation by soil stripping, the area has to undergo detailed soil and rock geotechnical in-situ testing in selected sites where foundations are to be laid-out based on the conceptual plant design layout prepared by the plant construction engineers. Emphasis should be given in grounds where there will be vibrating plant equipment assemblies, such as the crushing plant and the ball bill.

Among the most important in-situ tests and observational tests to be done are SPTs with collection of undisturbed sample for laboratory, CPTs, test pitting, and rock coring. The undisturbed samples has to be tested for Atterburg limits, density of the fill, x-ray diffraction, void ratio of fill, tri-axial with pore pressure in soils, UCS on rock core samples, and sulphate soundness which if at a higher percentage will compromise the integrity of the concrete in the foundation in long-term.

The combined field observation by shallow subsurface test pitting, in-situ soil and rock measurements, and laboratory tests would be more than enough for the civil designers to arrive at the bearing capacity estimates,soil compressibility, in predicting total settlement, rates of settlement, differential settlements (the latter if any), and what foundation style will be most appropriate for each of the plant module and for the whole plant's base foundation (mat foundation, spread footing, combined spread footing with slab, or piling - though the latter is most unlikely to be used).

24.2. Hydrogeology

24.2.1. Jugan – Hydrogeological Study & Pit Dewatering Assessment

A series of three to four (3 to 4) deep boreholes, to a depth of ±200 m, for pit dewatering assessment are planned shortly in areas around the pit. This is to investigate the hydraulic condition (k) in the vicinity of Jugan pit with respect to the Sunggai Sarawak Kanan River (SSK) and to evaluate and establish the potential groundwater inflow into the pit for pit dewatering purposes.

The SSK is located anywhere between 450 to 1,600 metres away from the Jugan Pit and meandering relative to pit's northwest to northeast sides before it heads up straight to Kuching where it meets up with the South China Sea.

Piezometers will be installed in selected boreholes so that in-situ hydraulic conductivity tests could be performed to characterise the hydraulic conductivity of the Shale domain.

Although the Jugan orebody is generally a shale-hosted gold deposit, it is expected that the k value will be much higher in the shale-sandstone interbeds, along the contact between the WNW-trending clayey intrusive and shale domain, and in the highly-fractured rock matrix itself from the Jugan hill at 25 mRL down to about -85mRL (a massive rockmass is basically non-existent in these levels).

A hydrogeologist will be contracted to carry-out the job. Furthermore, pumping wells within the perimeter of the proposed pit will be drilled and tested. The pumping test will include both step-test and constant rate tests. The data from the pumping tests would be analysed using methods appropriate to the hydrogeological conditions. Values for transmissivity and aquifer storativity will be determined. All of these once established would be used to define the model input parameters for simulation purposes. This would provide more confidence in the results of the modelling that the hydrogeologist would be doing.

Further to the geotechnical considerations, the established cut slopes and pit slopes at the Jugan pit will be further optimised once the phreatic level, if any, has been established. To incorporate the phreatic level into the overall analysis, a finite element application, Phase 2, will be used to perform a shear strength reduction modelling. This will also enable us to achieve a particular safety factor for an individual sector in the pit while optimising the pit angles. Phase 2 is a numerical modelling stress-strain application package.

24.2.2. BYG – Hydrogeological Study & Pit Dewatering Assessment

Closer to the commencement of the BYG-Krian pit mining, a series of deep boreholes for pit dewatering assessment are planned in the area between the Bukit-Young Pit and Tasik Biru. This is to investigate the hydraulic condition (k) in the vicinity of the pit with respect to the Tasik Biru Lake and to evaluate and establish the potential groundwater inflow into the pit for pit dewatering purposes. The lake is located around 500 m away from the northwest side of the Bukit-Young Pit.

Piezometers will be installed in selected boreholes so that in-situ hydraulic conductivity tests could be performed to characterise the hydraulic conductivity of the limestone and sandstone domains.

Similar work to that at Jugan will be undertaken.

24.3. Ore Concentrate Bagging & Transport

24.3.1. Bulk Bag – 1m x 1m 1m made of Woven Polypropylene Plastic

24.3.1.1. Types of Construction

General types are Circular (Tubular), U Panel, or rectangular. There are various specification variants with top, base, in-let & out-let spouts wherever applicable. Duffle/skirt and/or side skirting is specified by client. Bulk bags are cost effective method of shipping and storing dry goods. Made from woven polypropylene plastic, bulk bags have been estimated to have a lifespan of 400 to 1000 years before completely disintegrating. In most industries they can be used multiple times and when not in use can collapse to 1/50 of their size when filled, minimizing the need for storage space.

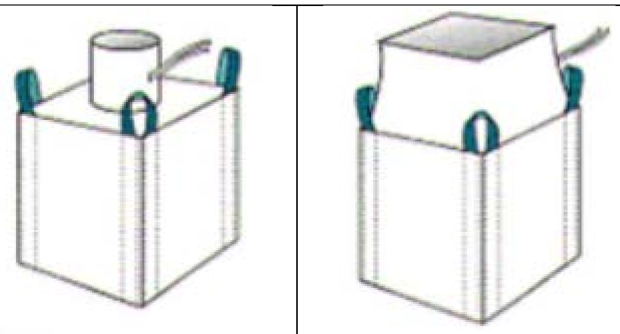
Product Name	Polypropylene (PP) jumbo bag/bulk container bag/PP woven bag/FIBC bag
Materials	Woven Polypropylene Plastic -new pp (made in China or Australia)
Type of bag	U-panel/ tubular/circular/rectangular shape
Fabric	Laminated/plain/vent
Size	100*100*100cm,or any other size is ok
Color	white, or under clients' request
Top	Full open/ with spout/with skirt cover
Bottom	Flat/spout
Liner	yes or as per customers' request, Liner(HDPE,LDPE)
Sewing	Plain/chain/chain lock with optional soft-proof
Lifting loop	2 or 4 belts, cross corner loop/fully loop/loop in loop
Ropes	1 or 2 around the bag body, or under customers' requirement
SWL	2500kg – 3000kg
Safety Factor (std)	5:1
Package	bales or sacks
Characteristics	Breathable and airy, anti-static, UV stabilization, reinforcement, dust-proof, moisture-proof
Delivery time	10-20 workdays

Feature	Breathable
Packing	20pcs/Bundle, 3500pcs/container in bales or pallets or as per customers' demand
Trade Term	CIF, FOB

Table 24-8 - Concentrate Bag Specifications

24.3.1.2. Top Filling Options (2 of 5 options)

Listed below in the *Figure 24-31 - Concentrate Bag – Some Top Filling Options* are 2 examples of bag filling options.



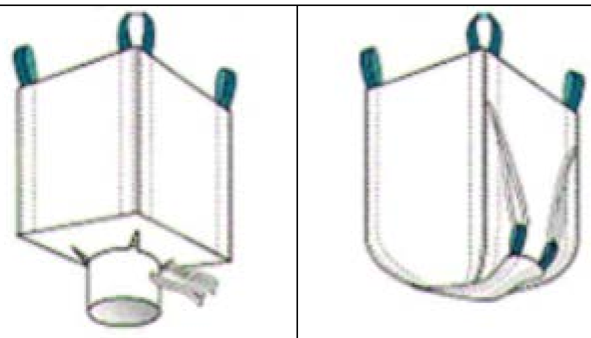
A – Inlet Spout with Tie

B – Skirt Top

Figure 24-31 - Concentrate Bag – Some Top Filling Options

Discharge Options (2 of 5 options)

Listed below in the *Figure 24-32 - Concentrate Bag – Some Discharge Options* are 2 examples of bag filling options.



A – Outlet Spout with Tie

B – Skirt Top

Figure 24-32 - Concentrate Bag – Some Discharge Options

Figure 24-33 - Example of Bulk Bag

24.3.1.3. Filling and Weighing

There is a wide range of bulk bagging-filling systems in the market but has to be custom-designed based on the product or material to be handled.

Some examples of bulk bag filling are shown by the *Figure 24-34 - Bucket Loader Mounted as Bucket of Wheel Loader* to *Figure 24-37 - Base Weight System in Combo with Bagging/Filling Equipment* below.

Figure 24-34 - Bucket Loader Mounted as Bucket of Wheel Loader

Figure 24-35 - Mechanised Fill System

Figure 24-36 - Automated Filling System

Figure 24-37 - Base Weight System in Combo with Bagging/Filling Equipment

24.3.2. Loading at Jugan Plant Site

Probably the most practical loading system at the plant site aside from a Forklift is an overhead hoist and trolley similar to the *Figure 24-34 - Loading / Unloading using Overhead Trolley* below.

An overhead hoist & trolley is more appropriate for loading High-bed Trailer Trucks or Low-bed Trailer Trucks with 20 ft container vans. If possible direct loading into container vans at the plant site is preferable to minimize multiple lifting of the bulk bags.

Figure 24-38 - Loading / Unloading using Overhead Trolley

24.3.3. Hauling – From Jugan Mine Site to Pending Port

Material to be hauled:	Ore Concentrate (density 2.5 tonnes per cubic meter)
Packaging System:	Bulk Bags (1m x 1m x 1m) weighing about 2.5 tonnes
HAULING DISTANCE:	35 kilometers from Jugan (plant site) to Pending (shipping port)
Trucks to be used:	Hi-bed trailer trucks or Low-bed trucks with 20ft container vans
Capacity of truck:	At least 30 tons payload or 12-Bulk bags (2 container vans)
Quantity:	At least 400 tonnes per day (12,000 tonnes per month)
No. of trips:	400 trips/mth (80 trips/truck per mth using 5 units of 30t-truck)
Loading at Jugan:	By overhead hoist & trolley for the bulk bags
Unloading at Pending:	Assume Overhead Crane

24.3.4. Port Handling including Storage & Security – Pending Industrial Estate Kuching

<u>Assumptions:</u>

Bulk Bags containing the merchandise (ore concentrate) will be hauled to Pending Industrial Estate, accumulated and stored in a secured building awaiting shipment schedule.

Assuming that shipment to China is by small dedicated Cargo Vessel (4000 t to 6000 t DW capacity), this means an accumulation period of at least 10 days given a minimum production of 400 tonnes per day ore concentrate.

24.3.5. Shipping

Dedicated small Cargo Vessel (4k – 6k deadweight ton capacity), the type of cargo vessel could be a bulk carrier or small container ship. For bulk carrier, there is no need for container vans as the bulk bags can be loaded directly into the vessel. If shipment is by Container Cargo Ship, 20 ft container vans will be used but due to the payload limit of 28,000 kg, the container van will just be 50 % loaded in terms of volume. Shipping cost is expected to be higher compared to big container ship but has some degree of flexibility in terms of shipment schedule.

24.3.5.1. Bagging & Transport Cost (cost/tonne)

Assumptions:

- Ore Production: 8,000 tpd (basecase)
- Mass Pull: 7% ave. (from 6% to 8% range)
- Concentrate Production: 560 tpd
- Number of Bulk Bags: 224 bags (2.5 capacity)
- Hauling Distance: 35 km (via Jln Bau- Jln Stephen Yong-Jln Batu Kawa)
- Alternate Route: 29 km (Jalan Bau - for night hauling)
- Payload capacity of truck 30 tonnes (12 bulk bags)

24.3.5.2. Cost Items

1. ***Bulk Bag*** – made of woven Polypropylene Plastic with HDPE Liner. Alibaba.com listed a made in China bulk bags with an FOB price of $2.00 to $9.00 per bag. For heavy load of up to 2.5 t, the design needs to be customized, reinforced and tested to make sure it can take the required load and should be able to endure multiple usages.

    - Adjusted Price: US$ 15.00 per bag (2.5t cap with HDPE liner/skirting)
    - Average Usage: 4 times (assume use the bags 4x before proper disposal)
    - Tonnage per bag: 4 x 2.5 = 10.0 tonnes (life span of bag in terms of tonnes)

2. ***Bagging & Weighing*** – The filling and weighing of the bulk bags should be at least semi-automatic. Bagging rate is assumed to be 20-bags per hour using 2-man crew as compared to the advertised 20-bags per hour NBE (1-operator) semi-auto system or 20 bags per minute for FIBC_7000 fully automatic filling/weighing system.

Bagging & Weighing Description	Parameters
Bagging and weighing production line	1 x 2-man crew
Bagging & Weighing Rate	20 bags/hour
Bagging & Weighing Time for 224 bags	11.2 hours
Bagging/weighing man-hours for 224 bags	22.4 hours
Preparation, equipment checks & house keeping (excluding 2hrs allocated for maintenance)	0.5 hour
Lunch break (day shift break)	1 hour

Bagging & Weighing Description	Parameters
Total bagging man-hours for 224 bags	23.9 hours

Table 24-9 - Bagging & Weighing Parameters

3. ***Loading of Haul Trucks*** – assume loading at plant site is by overhead hoist & trolley with 2-man crew. Ideally, the bagging and loading rates should be almost the same in terms of the number of bags filled and loaded per hour.

Haul Truck Loading Description	Parameters
Loading system	Overhead hoist & trolley (5t_cap)
Loading crew (1 operator + 1 spotter)	2-man crew
Loading Cycle Time (open top loading)	2.5 min/bag (including spotting)
Loading Rate	24 bags per hour
Loading Time for 224 bags	9.33 hours
Loading man-hours for 224 bags	18.67 hours
Equipment checks & housekeeping (excluding 2hrs allocated for maintenance)	0.5 hour
Lunch/meal break	1 hour
Truck change & truck spotting	1 min per truck
Truck change & spotting for 19 trucks	0.32 hr (1min per truck)
Total Loading man-hours for 224 bags	20.49 hours

Table 24-10 - Haul Truck Loading Parameters

4. ***Transport Cost***

Contractor Transport quotes:

Price in RM (all-in cost): RM 315 per trip at 30 tonnes per trip via 35km route

Price in US$: $ 96.21 per trip or US$ 3.21 per tonne

5. ***Port Handling Cost*** – including storage & security. Same as previous estimate = $ 3.00/tonne

6. ***Shipping Cost***

- Previous estimate was $ 25.00/tonne
- From World Freight Rates.com = $ 693.61 per container van (20ft) (Reference: Kota Kinabalu to Shanghai)
- Cost per tonne using 28t payload capacity = $ 24.77

Table 24-11 - Concentrate - Bagging and Transport Costing below summarises the costing for the concentrate transport costs.

Production Data	TPD	1 bag	1 bag (used 4 times)	224 bags (used 4 times)
Ore, t	8,000			
Concentrate, t	560			
Bulk Bags	224 bags	2.5	10.0	2,240

Description (Cost Item)	Qty-Unit	Unit Cost (US $)	Total Cost (US $)	Cost/Tonne (concentrate)
Bulk Bag – 1m x 1m x 1m Made in China PP	224 bags	$ 15.00	$ 3,360.00	$ 1.50
Bagging Labour for 224 bags	23.90 hrs	$ 5.12	$ 122.37	$ 0.22
Loading Labour for 224 bags	20.49 hrs	$ 4.44	$ 90.98	$ 0.16
Bagging/Loading supervision	12 hrs	$ 6.25	$ 75.00	$ 0.13
Power	72 KwH	$ 0.07	$ 5.04	$ 0.01
Maintenance (labour & parts)	2 hours	$ 9.00	$ 18.00	$ 0.03
Trucking by Contractor	560 t	$ 3.21	$ 1,797.60	$ 3.21
SUMMARY				
BAGGING AND TRANSPORT COST				**$ 5.27**
Port Handling at Port of Exit & Entry (Same as previous estimate)				**$ 3.00**
SHIPPING COST a. **Previous Estimate** b. **based from World Freight Rates.com**				**$ 25.00**
TOTAL COST a. **using previous estimate of shipping cost** b. **based on WFR.com rates**				**$ 33.27**

Table 24-11 - Concentrate - Bagging and Transport Costing

25. Conclusions

In conclusion, BESRA finds the first stage of the plan to develop and put the Bau Goldfield into production is a lean business case and economically viable strategy with manageable risks.

The region has significant opportunity for growth and by moving into detailed engineering and construction now BESRA can best be setup for a return to higher gold prices and for developing long term partnerships with the smelter customers. Strategically, the concentrate option offers advantages as fuel source for the smelters while leaving BESRA the opportunity for secondary processing on site should a more robust gold market return.

By moving into production now BESRA is able to generate significant cash flow to further improve the gold field resources and reserves as well to take advantage of the opportunity for growth with the site infrastructure built up to then. BESRA has become a stronger operator every year of its existence and the management team are fully aware of the lessons learned from the past while being cautiously optimistic about the next step in our future in Malaysia.

26. Recommendations

The project has the capacity for further optimisation and refinement of the following:

- Process plant equipment and clay/fines handling
- Capital and operating cost optimisation
- Detailed in-house engineering and construction using the existing skills that have developed two mines already – reducing the EPCM costs
- Mining and infrastructure detailed design and optimisation to reduce
- Use of materials and equipment sourced locally (or within Malaysia) to reduce transport and any import costs

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A. Appendices

A2-1. Glossary, Technical Nomenclature & Abbreviations

Symbol/Abbreviation/Nomenclature	Description
>	Greater than
<	Less than
=	Equal
%	Percent
±	Plus/Minus or approximate
>=	Greater than or equal to
<=	Less than or equal to
‘ or ft	Feet (Imperial)
“ or in	Inches (Imperial)
#	Mesh
$	Dollars (US unless specified)
°	Degrees
°C	Degrees Celsius
3D	Three dimensional
AAS	Atomic absorption spectrometer
AI	Abrasion Index
Ag	Silver
Al	Aluminum
As	Arsenic
Au	Gold
AusIMM	Australasian Institute of Mining & Metallurgy
Ba	Barium
BBWI	Bond Ball Mill Work Index
Bi	Bismuth
BLEG	Bulk leach extractable gold
BIOX	Biological Oxidation
BQ	Diamond drill core – 36.4 mm diameter
BRSO	Borneo Rectified Skew Orthomorphic
BRWI	Bond Rod Mill Work Index
BYG	Bukit Young Goldmines

Symbol/Abbreviation/Nomenclature	Description
BYGS	Bukit Young Gold Services (Menzies)
Ca	Calcium
Cd	Cadmium
CIL	Carbon-in-leach
CIMM	Canadian Institute of Mining, Metallurgy & Petroleum
Co	Cobalt
Cr	Chromium
Cu	Copper
CV	Coefficient of variation
DD	Diamond drilling
DDH	Diamond drill hole
DIGHEM	
E	East
EIA	Environmental impact assessment
ELF	Engineered Land Form
EM	Electromagnetic
ENE	East north east
EPCM	Engineering procurement and construction management
EPL	Exclusive Prospecting Licence
ESE	East south east
FA	Fire assay
Fe	Iron
FOB	Free on board
G&A	General and administration
gcm^{-3}	Grams per cubic centimetre
Gladioli	Gladioli Enterprises Sdn Bhd
GPL	General Prospecting Licence
g/t	Grams per tonne
g/t Au	Grams per tonne gold
Ha	Hectare
Hg	Mercury
hr(s)	Hour(s)

Symbol/Abbreviation/Nomenclature	Description
HQ	Diamond drill core – 63.5 mm diameter
HQ-3	Diamond drill core – 61.1 mm diameter
ICP	Inductively coupled plasma
ICP-MS	Inductively coupled plasma mass spectrometer
ICP-OES	Inductively coupled plasma optical emission spectroscopy
IP	Induced potential
IRR	Internal rate of return
IsaMill	Proprietary Mt Isa mill technology
ISO	International Standards Organisation
JORC	Joint Ore Reserves Committee
JV	Joint venture
K	Potassium
kg	kilogramme
km(s)	Kilometre(s)
km^2	Square kilometres
Koz	Thousand ounces
kPa	Kilopascal
Kg/m^3	Kilogramme per cubic metre
Kt	Thousand tonnes
kW	Kilowatt
kWh	Kilowatt hour
kl	Kilolitre
l	Litre
LOM	Life of mine
L&S	Lands & Survey Department
l/s	Litre per second
LSC	Limestone-shale contact
m	Metre
m^2	Square metre
m^3	Cubic metre
M	Million
Ma	Million years

Symbol/Abbreviation/Nomenclature	Description
MC	Mining Certificate
MCAF	Mine cost adjustment factor
Mg	Magnesium
MIK	Multiple Indicator Kriging
MIM	Mount Isa Mines
ML	Mining Licence
mm	Millimetre
µm	Micron
MMI	Mobile metal Ion
Mo	Molybdenum
Moz	Million ounces
Mn	Manganese
mE	Metres East
mN	Metres North
mRL	Metres relative level
Mt	Million tonnes
Mtpa	Million tonnes per annum
MW	Megawatt
MYR	Malaysian ringgit
my	Million years
N	North
Na	Sodium
NAF	Non-acid forming
NAPP	Net acid production potential
NBG	North Borneo Gold Sdn Bhd
NE	North east
Ni	Nickel
NNE	North north east
NNW	North north west
No.	Number
NPV	Net present value
NQ	Diamond drill core – 47.6 mm diameter

Symbol/Abbreviation/Nomenclature	Description
NQ-3	Diamond drill core – 45 mm diameter
NREB	Natural Resources Environment Board
nsg	Non-sulphide grains
NW	North west
OK	Ordinary Kriging
OYM	Olympus Pacific Minerals (Besra predecessor)
oz	Ounce (troy)
pa	Per annum
Pb	Lead
PAF	Potentially acid forming
POX	Pressure Oxidation
ppb	Parts per billion
ppm	Parts per million
PQ	Diamond drill core – 85 mm diameter
PQ-3	Diamond drill core – 83 mm diameter
P_{80}	80% passing
Q-Q	Quantile-Quantile plots
QAQC	Quality assurance, quality control
RC	Reverse circulation (drilling)
RL	Relative level
ROM	Run of mine
RQD	Rock quality designation
RM	Malaysian ringgit (alternate)
RMR	Rock mass rating
S	Sulphur
SAG	Semi-autogenous grinding (mill)
Sb	Stibnite (antimony ore)
SE	South east
SG	Specific gravity
SGS	Société Générale de Surveillance
SHRGD	Sediment-hosted rock gold deposits
SO_2	Sulphur dioxide

Symbol/Abbreviation/Nomenclature	Description
SO_3	Sulphide
SO_4	Sulphate
SO_x	Sulphide oxidation
SPI	SAG power index
SSE	South south east
SSW	South south west
SW	South west
t	Tonnes
Ti	Titanium
Tl	Thallium
TMCSA	Terra Mining Consultants & Stevens & Associates
t/m^3	Tonnes per cubic metre
tpa	Tonnes per annum
tpd	Tonnes per day
UCS	Unconfined compressive strength
UFG	Ultra-fine grinding
US$	United States of America dollar
UTM	Universal Transverse Mercator
V	Volt
V	Vanadium
W	Tungsten
WI	Work index
WNW	West north west
WSW	West south west
wt	weight
yr	Year
Zn	Zinc
4WD	Four-wheel drive

A11-1. Logging Codes & Descriptions

Lithology	Lithology Description
A	Andesite
A-J	Andesite-Tonalite
APV	Andesitic Pyroclastic Volcanics
B	Basalt
BC	Calcite - Black Fine Grained, Sulphide, Pyrite & Organics
BM	Base-Metal Vein
CL	Clay
DAC	Dacite
DE	Dacite-Mudstone Contact
DF	Dacite Porphyry - Fine Grained
DI	Diorite
DK	Endoskarn-calcsilicates
D-LD	Dacite in contact with limestone
D-M	Dacite-Marble Contact Zone
DP	Dacite Porphyry
DPV	Dacitic Pyroclastics
D-SH	Dacite-Shale Contact Zone
DXI	Intrusion Breccia in Dacite Porphyry
F	Hornfels
FM	Hornfelsed Marble
FT	Fault
FTG	Fault Gouge
G	Conglomerate
GD	Granodiorite
G-SL	Interbedded Conglomerate and Siltstone
GT	Grit
GT-L	Grit in contact with Limestome
GTOL	Interbedded Grit, Marl and Limestone
GT-SS	Interbedded Grit and Sandstone
H2O	Water-Drilling of Platform in Tasik Biru
IN	Intrusive

Lithology	Lithology Description
L	Limestone - Undifferentiated
LA	Limestone - Dark Grey, Gritty, Argillaceous
LC	Limestone - Clastic, Grey-Dark Grey
LD	Limestone - Dark Grey-Black, Argillaceous
LD-LG	Limestone-Interbedded Pale Grey and Dark Grey and Gritty
LD-LP	Interbedded Light and Dark Grey Limestone
LD-XT	Limestone-Dark Grey Argillaceous- Brecciated (Tectonic)
LF	Limestone - Fossiliferous
LF-SS	Limestone Fossiliferous wth Sandstone Layers
LG	Limestone - Massive, Pale Grey-Grey
LG-SH	Interbedded Limestone and Shale
LG-W	Limestone - Contact Cavity
LG-WC	Limestone - Contact Calcite Veining
LG-XH	Limestone - Hydrothermlally brecciated
LG-XT	Limestone-Brecciated (Tectonic)
LJ	Jasperoid
LK	Exoskarn-calcsilicates
LL	Calclutite
LOSL	Interbedded Limestone, Marl and Siltstone
LP	Limestone - Pale Grey, Soft-Porous
LP-LG	Limestone - Grey in contact with pale grey
LP-O	Pale Grey Limestone in Contact with Marl
LS	Limestone - Silty
LSHM	Interbedded Limestone, Shale and Mudstone
M	Marble
MD	Microdiorite
MDPV	Micro-Diorite and Pyroclastics
MGD	Micro-granodiorite
MQDP	Microgranodiorite porphyry
MS	Mudstone
MS-SH	Interbedded Mudstone and Shale
NC	No Core or Sample
NL	No Lithology Indicated

Lithology	Lithology Description
O	Marl
OB	Overburden
O-GT	Interbedded Marl and Grit
O-XT	Brecciated Marl (Tectonic)
PV	Pyroclastic Volcanics
QTZ	Quartz Vein
QC	Quartz-Calcite Vein
QX	Brecciated Quartz Vein
R	Radiolarian Siliceous Rock
SC	Calcareous Shale-Marl
SH	Shale
SH-G	Interbedded Shale and Conglomerate
SH-GTL	Shale-Grit and Limestone
SH-LD	Interbedded Shale & Dark Grey-Black Argillaceous Limestone
SH-LG	Shale in contact with pale grey limestone
SHOL	Interbedded Shale, Marl and Limestone
SH-SL	Interbedded Shale-Siltstone
SH-SS	Interbedded Shale-Sandstone
SL	Siltstone
SL-GT	Interbedded Siltstone and Grit
SL-LD	Interbedded Siltstone and Limestone
SL-O	Interbedded Siltstone and Marl
SLOGT	Interbedded Siltstone, Marl and Grit
SL-SHL	Interbedded Siltstone, Shale and Limestone
SL-SS	Interbedded Siltstone-Sandstone
SS	Sandstone
SS-SL	Interbedded Sandstone-Siltstone
ST	Coarse Sandstone
T	Tonalite
TF	Tuff
U	Alluvium
V	Void-Cavity
V-CL	Cavity with Clay Fill

Lithology	Lithology Description
WC	Calcite - White, Sparry with Sulphide
X	Breccia
XC	Breccia - Collapse
XH	Breccia - Hydrothermal
XI	Breccia - Intrusive
XQC	Brecciated Quartz Calcite Vein
XQDP	Xenolithic Quartz Diorite Porphyry
XT	Breccia - Tectonic
Z	Soil, Clay, Mullock, Rock Fill or Tailings

Formation	Formation Description
B	Bau Limestone Formation
I	Intrusive
KR	Krian Member
NF	No Formation
NR	Not Recorded
P	Pedawan Formation
Q	Quaternary and Recent Deposits
S	Serian Volcanics
W	Water-Tasik Biru

Colour	Colour Description
BK	Black
BL	Blue
BR	Brown
CM	Cream
GN	Green
GY	Gray
MV	Mauve
OR	Orange
RD	Red
WH	White

Colour	Colour Description
YW	Yellow

Colour Intensity	Colour Intensity Description
L	Light
M	Medium
D	Dark

Oxidation	Oxidation Description
1	Unoxidised
2	Weakly oxidised
3	Moderately oxidised
4	Strongly oxidised
5	Completely oxidised

Alteration Type	Alteration Description
C	Carbonate
CL	Clay altered
CLX	Clay altered-Oxidized
CS	Carbonate-Silica
CSX	Carbonate-Silica - Oxidised
CX	Carbonate - Oxidised
D	Decalcified
DX	Decalcified-Oxidised
E	Epidote
I	Illite
IS	Illite-Smectite
K	Calc-silicate (skarn)
KS	Calc-silicate (skarn)-Silicified
KX	Calc-silicate (skarn)-Oxidised
L	Recrystallised
M	Marble
ME	Marble-Epidote

Alteration Type	Alteration Description
MX	Marble-Oxidised
O	Chlorite
OI	Chlorite-Illite
ORT	K-Feldspar+Epidote+Calcite
OM	Chlorite-Marble
OS	Chlorite-Silica
OX	Chlorite - Oxidised
Q	Quartz-Sericite-Pyrite
QCO	Quartz-Sericite-Carbonate-Chlorite
QRO	Quartz-Sericite-Chlorite
R	Sericite
RC	Sericite-Carbonate
RCO	Sericite-Carbonate-Chlorite
RCS	Sericite-Carbonate-Silica
RO	Sericite-Chlorite
RX	Sericite-Oxidised
S	Silicified
SC	Sericite-Carbonate
SD	Siderite
SE	Silica-Epidote
SR	Silica-Sericite
SX	Silicified - Oxidized
UN	Unaltered
X	Oxidised
XD	Oxidised-Decalcified
XO	Oxidised-Chloritised
Y	Propyllitic
YX	Propyllitic-Oxidised
Z	Oxidised-Silicified

Alteration Style	Alteration Style Description
FC	Fracture controlled
FG	Fracture coating

Alteration Style	Alteration Style Description
IR	Irregular
PA	Patchy
PER	Pervasive
SP	Semi-pervasive
SR	Selective replacement
ST	Stringer
VN	Veins
VS	Vein Selvedges

Alteration Intensity	Alteration Intensity Description
1	Incipient
2	Weak
3	Moderate
4	Strong
5	Intense

Mineralisation	Mineralisation Description
A	Arsenopyrite
A-R	Arsenopyrite-Orpiment-Realgar
A-S	Arsenopyrite-Stibnite
B	Pyrite-Chalcopyrite-Galena-Sphalerite
BP	Pyrite-Chalcopyrite-Galena
C	Calcite-Pyrite-Native Arsenic
CZ	Quartz-Calcite-Stibnite
D	Dickite-Illite
E	Dickite-Calcite-Pyrite
F	Quartz-Calcite-Pyrite
FK	Quartz-Calcite-Pyrite-Stibnite-Native Arsenic
FS	Quartz-Calcite-Pyrite-Stibnite
FSA	Quartz-Calcite-Pyrite-Stibnite-Native Arsenic-Arsenopyrite
G	Dickite-Pyrite
H	Pyrite-Arsenopyrite

Mineralisation	Mineralisation Description
H-BP	Arsenopyrite-Basemetal
H-D	Pyrite-Arsenopyrite-Dickite-Calcite
H-L	Pyrite-Calcite-Arsenopyrite
H-Q	Pyrite-Arsenopyrite-Quartz-Dickite
H-R	Pyrite-Native Arsenic-Orpiment-Realgar
H-S	Pyrite-Arsenopyrite-Stibnite
H-Z	Pyrite-Arsenopyrite-Quartz Vein
I	Stibnite-Native Arsenic
J	Quartz-Pyrite
J-I	Quartz-Pyrite-Stibnite-Native Arsenic
JP	Jasperoid-Pyrite
J-S	Quartz-Pyrite-Stibnite
KC	Realgar-Native Arsenic-Stibnite
KN	Orpiment-Stibnite-Native Arsenic
L	Calcite Vein
LA	Calcite-Arsenopyrite
LAS	Calcite-Arsenopyrite-Stibnite
L-D	Calcite-Dickite
LN	Calcite-Realgar-Native Arsenic
LNA	Calcite-Realgar-Native Arsenic-Arsenopyrite
L-R	Calcite-Realgar-Orpiment
M	Calcite-Pyrite-Stibnite
N	Native Arsenic
P	Pyrite
PN	Pyrite-Realgar-Orpiment-Native Arsenic
PO	Pyrite-Pyrrhotite
P-Q	Pyrite-Quartz-Dickite
PR	Pyrite-Orpiment-Realgar
Q	Quartz-Dickite
R	Orpiment-Realgar
RN	Realgar-Orpiment-Native Arsenic
RS	Realgar-Orpiment-Stibnite
S	Stibnite

Mineralisation	Mineralisation Description
S-P	Stibnite-Pyrite
SZ	Quartz-Stibnite
T	Quartz-Dickite-Pyrite-Stibnite
U	Quartz-Calcite
UA	Quartz-Calcite-Arsenopyrite

Mineralisation Style	Mineralisation Style Description
AG	Aggregations
BND	Banded
BX	Breccia
DI	Disseminated
FG	Fracture coating
IN	Interstitial
IR	Irregular
MA	Massive
MS	Conc in vein margins-selvedges
PA	Patchy
PV	Pervasive
SC	Specks
SO	Spots
SP	Semi-pervasive
SR	Selective replacement
ST	Stringer
VG	Vughy
VN	Veins
VS	Vein Selvedges

Mineralisation Intensity	Mineralisation Intensity Description
1	Incipient
2	Weak
3	Moderate

Mineralisation Intensity	Mineralisation Intensity Description
4	Strong
5	Intense

Sulphide Type	Sulphide Type Description
A	Arsenopyrite
BO	Bornite
CH	Chalcocite
CP	Chalcopyrite
CS	Cu-sulphides (general)
CV	Covellite
GA	Galena
MO	Molybdenite
OP	Orpiment
PO	Pyrrhotite
PY	Pyrite
RG	Realgar
SB	Stibnite
SP	Sphalerite
SR	Sarabauite

Sulphide Style	Sulphide Style Description
BM	Concentrated within breccia matrix
BND	Banded sub parallel layers
CD	Coarse grained disseminated
FD	Fine grained disseminated
FG	Fracture coating
GN	Granular
IR	Irregular
MD	Medium grained disseminated
MS	Concentrated in vein margins and selvedges
PA	Patchy
RM	Rimming fragments

Sulphide Style	Sulphide Style Description
SC	Specks
SE	Segregations
SO	Spots
VL	Veinlets
VN	Vein

Sulphide Percentage
0.1
0.25
0.5
1.0
1.5
2.0
2.5
3.0
3.5
5.0
7.0
10.0
20.0
30.0
50.0
80.0
100.0

Vein Type	Vein Type Description
VC	Carbonate (undifferentiated)
VD	Dolomite
VF	Crustiform
VG	Vughy
VH	Sheeted
VI	Calcsilicate

Vein Type	Vein Type Description
VL	Crack seal
VO	Colloform
VQ	Quartz
VR	Carbonate Stringers
VS	Sulphidic
VT	Tensional
VW	Stockwork
VX	Brecciated
VY	Cryptocrystalline
VZ	Quartz-calcite

Breccia Type	Breccia Type Description
BC	Collapse Breccia
BE	Milled Breccia
BF	Fault Breccia
BH	Hydrothermal Breccia
BI	Intrusion Breccia
BK	Karst collapse Breccia
BL	Crackled Breccia
BN	Monomictic Breccia
BR	Zone of Brecciation and re-cementation
BS	Shear Breccia
BT	Imbricate Breccia
BU	Undifferentiated Breccia
BV	Vein Breccia
BY	Polymictic Breccia

Breccia Intensity	Breccia Intensity Description
1	Weak
2	Moderate
3	Strong

Structure	Structure Description
BD	Bedding
BR	Broken
BX	Breccia
CG	Cleavage
CN	Contact
CO	Compositional layering
CT	Cataclasites
FB	Fault brittle
FL	Fault brittle-ductile
FO	Foliation
FR	Fracture
FT	Fault
FWC	Vein contact - footwall
HWC	Vein contact - hanging wall
JN	Joint
LI	Lineation
MY	Mylonite
PC	Pug-clayzone
SH	Shear zone
SL	Slickensides
ST	Stringers
SW	Stockwork
UC	Unconformity
VL	Veinlets
VN	Vein

Fill	Fill Description
AS	Arsenopyrite
CA	Calcite
CH	Chlorite
CL	Clay
CS	Calc-Silicate
GO	Gouge

Fill	Fill Description
IC	Iron Carbonate
IO	Iron Oxide
PY	Pyrite
QZ	Quartz
SU	Sulphide

Roughness	Roughness Description
1	Rough or irregular, stepped
2	Smooth, stepped
3	Slickensided, stepped
4	Rough or irregular, undulating
5	Smooth, undulating
6	Slickensided, undulating
7	Rough or irregular, planar
8	Smooth, planar
9	Slickensided, planar

A15-1. Reserves – Pit Optimisation

JUGAN RESERVES – 4,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	3,442,370	1.471
	Contract Mining-POX Processing	2,889,480	1.645
	Contract Mining-BIOX Processing	2,654,580	1.723
	Contract Mining-Albion Processing	2,137,160	1.909
	Owner Mining-Concentrate Production	3,445,955	1.470
	Owner Mining-POX Processing	2,902,270	1.642
	Owner Mining-BIOX Processing	2,667,260	1.720
	Owner Mining-Albion Processing	2,152,050	1.905
Probable	Contract Mining-Concentrate Production	6,471,250	1.607
	Contract Mining-POX Processing	5,300,100	1.753
	Contract Mining-BIOX Processing	5,072,920	1.784
	Contract Mining-Albion Processing	4,041,500	1.909
	Owner Mining-Concentrate Production	6,505,980	1.604
	Owner Mining-POX Processing	5,574,080	1.741
	Owner Mining-BIOX Processing	5,303,440	1.779
	Owner Mining-Albion Processing	4,354,420	1.905

JUGAN RESERVES – 6,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	3,444,390	1.470
	Contract Mining-POX Processing	2,891,390	1.645
	Contract Mining-BIOX Processing	2,661,660	1.721
	Contract Mining-Albion Processing	2,140,970	1.908
	Owner Mining-Concentrate Production	3,446,390	1.470
	Owner Mining-POX Processing	2,905,070	1.642
	Owner Mining-BIOX Processing	2,669,070	1.720
	Owner Mining-Albion Processing	2,152,990	1.905
Probable	Contract Mining-Concentrate Production	6,473,220	1.607
	Contract Mining-POX Processing	5,505,550	1.745
	Contract Mining-BIOX Processing	5,078,980	1.783
	Contract Mining-Albion Processing	4,190,980	1.907

Category	Scenario Description	Tonnes (t)	Grade (g/t)
	Owner Mining-Concentrate Production	6,609,400	1.594
	Owner Mining-POX Processing	5,612,750	1.736
	Owner Mining-BIOX Processing	5,324,130	1.779
	Owner Mining-Albion Processing	4,397,180	1.903

JUGAN RESERVES – 8,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	3,444,580	1.470
	Contract Mining-POX Processing	2,892,650	1.645
	Contract Mining-BIOX Processing	2,664,500	1.721
	Contract Mining-Albion Processing	2,145,010	1.907
	Owner Mining-Concentrate Production	3,452,670	1.469
	Owner Mining-POX Processing	2,912,748	1.640
	Owner Mining-BIOX Processing	2,674,220	1.718
	Owner Mining-Albion Processing	2,153,290	1.905
Probable	Contract Mining-Concentrate Production	6,475,920	1.607
	Contract Mining-POX Processing	5,539,620	1.743
	Contract Mining-BIOX Processing	5,248,330	1.785
	Contract Mining-Albion Processing	4,285,540	1.912
	Owner Mining-Concentrate Production	6,705,100	1.590
	Owner Mining-POX Processing	5,788,262	1.727
	Owner Mining-BIOX Processing	5,476,530	1.774
	Owner Mining-Albion Processing	4,532,840	1.902

JUGAN RESERVES – 10,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	3,445,100	1.470
	Contract Mining-POX Processing	2,899,480	1.643
	Contract Mining-BIOX Processing	2,666,430	1.720
	Contract Mining-Albion Processing	2,150,840	1.906
	Owner Mining-Concentrate Production	3,453,350	1.469
	Owner Mining-POX Processing	2,912,750	1.640

Category	Scenario Description	Tonnes (t)	Grade (g/t)
	Owner Mining-BIOX Processing	2,675,520	1.718
	Owner Mining-Albion Processing	2,156,310	1.904
Probable	Contract Mining-Concentrate Production	6,647,860	1.597
	Contract Mining-POX Processing	5,573,050	1.741
	Contract Mining-BIOX Processing	5,276,740	1.782
	Contract Mining-Albion Processing	4,354,420	1.905
	Owner Mining-Concentrate Production	6,763,260	1.587
	Owner Mining-POX Processing	5,812,680	1.724
	Owner Mining-BIOX Processing	5,481,400	1.773
	Owner Mining-Albion Processing	4,539,940	1.901

JUGAN RESERVES – 12,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #	3,445,960	1.470
	Contract Mining-POX Processing	2,901,480	1.642
	Contract Mining-BIOX Processing	2,667,260	1.720
	Contract Mining-Albion Processing	2,150,840	1.906
	Owner Mining-Concentrate Production	3,459,240	1.467
	Owner Mining-POX Processing	2,913,090	1.640
	Owner Mining-BIOX Processing	2,675,520	1.718
	Owner Mining-Albion Processing	2,156,310	1.904
Probable	Contract Mining-Concentrate Production	6,648,050	1.597
	Contract Mining-POX Processing	5,574,080	1.741
	Contract Mining-BIOX Processing	5,276,900	1.782
	Contract Mining-Albion Processing	4,354,420	1.905
	Owner Mining-Concentrate Production	9,159,660	1.437
	Owner Mining-POX Processing	5,819,500	1.723
	Owner Mining-BIOX Processing	5,486,840	1.773
	Owner Mining-Albion Processing	4,541,170	1.901

BYG-KRIAN RESERVES – 4,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #		

Category	Scenario Description	Tonnes (t)	Grade (g/t)
	Contract Mining-POX Processing		
	Contract Mining-BIOX Processing		
	Contract Mining-Albion Processing		
	Owner Mining-Concentrate Production		
	Owner Mining-POX Processing		
	Owner Mining-BIOX Processing		
	Owner Mining-Albion Processing		
Probable	Contract Mining-Concentrate Production	972,760	3.177
	Contract Mining-POX Processing	869,190	3.482
	Contract Mining-BIOX Processing	814,270	3.628
	Contract Mining-Albion Processing	711,610	3.980
	Owner Mining-Concentrate Production	1,036,980	3.088
	Owner Mining-POX Processing	922,020	3.393
	Owner Mining-BIOX Processing	861,630	3.535
	Owner Mining-Albion Processing	742,300	3.902

BYG-KRIAN RESERVES – 6,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #		
	Contract Mining-POX Processing		
	Contract Mining-BIOX Processing		
	Contract Mining-Albion Processing		
	Owner Mining-Concentrate Production		
	Owner Mining-POX Processing		
	Owner Mining-BIOX Processing		
	Owner Mining-Albion Processing		
Probable	Contract Mining-Concentrate Production	1,005,430	3.137
	Contract Mining-POX Processing	874,160	3.469
	Contract Mining-BIOX Processing	816,920	3.620
	Contract Mining-Albion Processing	716,970	3.967
	Owner Mining-Concentrate Production	1,044,130	3.086
	Owner Mining-POX Processing	924,910	3.388
	Owner Mining-BIOX Processing	870,970	3.520
	Owner Mining-Albion Processing	743,190	3.898

BYG-KRIAN RESERVES – 8,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #		
	Contract Mining-POX Processing		
	Contract Mining-BIOX Processing		
	Contract Mining-Albion Processing		
	Owner Mining-Concentrate Production		
	Owner Mining-POX Processing		
	Owner Mining-BIOX Processing		
	Owner Mining-Albion Processing		
Probable	Contract Mining-Concentrate Production	1,007,380	3.133
	Contract Mining-POX Processing	901,240	3.419
	Contract Mining-BIOX Processing	818,860	3.616
	Contract Mining-Albion Processing	722,380	3.951
	Owner Mining-Concentrate Production	1,051,310	3.077
	Owner Mining-POX Processing	931,535	3.378
	Owner Mining-BIOX Processing	869,050	3.516
	Owner Mining-Albion Processing	770,760	3.854

BYG-KRIAN RESERVES – 10,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #		
	Contract Mining-POX Processing		
	Contract Mining-BIOX Processing		
	Contract Mining-Albion Processing		
	Owner Mining-Concentrate Production		
	Owner Mining-POX Processing		
	Owner Mining-BIOX Processing		
	Owner Mining-Albion Processing		
Probable	Contract Mining-Concentrate Production	1,008,090	3.132
	Contract Mining-POX Processing	906,610	3.405
	Contract Mining-BIOX Processing	828,120	3.592
	Contract Mining-Albion Processing	733,810	3.923
	Owner Mining-Concentrate Production	1,060,490	3.062
	Owner Mining-POX Processing	936,020	3.367
	Owner Mining-BIOX Processing	886,370	3.504
	Owner Mining-Albion Processing	771,910	3.851

BYG-KRIAN RESERVES – 12,000TPD

Category	Scenario Description	Tonnes (t)	Grade (g/t)
Proven	Contract Mining-Concentrate Production #		
	Contract Mining-POX Processing		
	Contract Mining-BIOX Processing		
	Contract Mining-Albion Processing		
	Owner Mining-Concentrate Production		
	Owner Mining-POX Processing		
	Owner Mining-BIOX Processing		
	Owner Mining-Albion Processing		
Probable	Contract Mining-Concentrate Production	1,021,030	3.116
	Contract Mining-POX Processing	907,240	3.404
	Contract Mining-BIOX Processing	861,000	3.537
	Contract Mining-Albion Processing	734,130	3.922
	Owner Mining-Concentrate Production	1,060,720	3.062
	Owner Mining-POX Processing	936,490	3.366
	Owner Mining-BIOX Processing	886,640	3.503
	Owner Mining-Albion Processing	772,020	3.851

A15-2. Ore Reserves – JORC Code Table 1 Checklist

Section 1: Sampling Techniques & Data

Criteria	Section 1 – Commentary
Sampling techniques	• Besra drillholes were sampled and assayed on nominal 1m intervals, except at geological or lithological boundaries. Early historic drillholes were sample at 1.5 and 2m intervals with later historic holes were nominally 1m. These longer lengths only make up approximately 5-10% of the total drilling metres. • Besra drillhole assays were sample prepped and assayed by SGS at their onsite laboratory in Bau (ISO17025 certified); assaying onsite was for Au by fire assay with other elements (23) assayed by ICP at the SGS laboratory in Perth. Umpire assays were done by Mineral Assay & Services Company (MAS) in Bangkok, Thailand. Some selected samples were also checked at SGS Waihi, New Zealand. • Historic assays: Renison Goldfields (RGC) and Gencor/Minsarco used commercial labs and their own QAQC systems; BYGS/Menzies Gold used Assaycorp initially in Australia and then in Kuching, Sarawak with McPhar, Analabs and Inchape for umpire sampling and QAQC. • For Besra assays, the Au grades were determined by 50g Fire Assay with AAS finish at the onsite SGS laboratory. • Channel and trench sampling was extensively carried out across the Jugan orebody/deposit outcropping on the hill. Channels/trenches were excavated across the mapped orebody surface extents to a depth between 1-3 metres. The base of the trench was “cored/slotted” in 1m sample lengths to mimic the same or similar volume as HQ drill core. These channels and trenches were used to delimit the orezone on surface. Samples collected followed the same/similar logging and sample processing procedures as for drillholes. Trench samples were used in the geological and resource modelling. Analyses of channel/trench data in the resource modelling showed little or no difference in results with or without these channels/trenches, and were deemed applicable to use.
Drilling techniques	• For Besra drilling: all drillholes were diamond with triple tube; all drillholes were angled and orientated; standard drill diameter used is HQ3 with PQ3 collars; NQ3 only used when requirement to reduce (e.g. ground conditions); metallurgical drillholes were drilled in PQ3/PQ. • For historic drilling: diamond and RC drilling; diamond drillholes were predominantly NQ diameter with additional holes in HQ/PQ. • At Jugan only 17 of the 82 RC drillholes (±5% of the 252 total drillholes) were used in the geological modelling; some drillholes were drilled at BQ with only 24 of the 252 (9.5%) drillholes used in the geological modelling; a mix of standard and triple tube drilling was used in the historical diamond drillholes. • At BYG-Krian • Where historic drilling was in BQ or RC, these holes were checked by infill drilling or twinned drillholes at PQ/HQ; analysis of drillhole data in the resource modelling showed little or no difference in results with or without these drillholes; this and the low percentage of these holes was deemed not have a material impact.
Drill sample recovery	• For Besra drillholes at Jugan deposit core recovery was good with an average of 98.25% recovered throughout the deposit/orebody.

Criteria	Section 1 – Commentary
	• Some historic drilling recoveries were also recorded at Jugan, and these average 96.42% • Besra BYG-Krian core recoveries averaged 94.73% and slightly lower mainly due to low recoveries near the collar outwith the ore zones. • Where difficult ground was encountered or where the sample recovery could be compromised, controlled drilling and short drilling runs (1.5m triple tube) were used. • There is no observed correlation between core recovery and Au grades, suggesting no apparent bias in the assay grades due to core recovery.
Logging	• Besra logging was done in specifically designed Excel spreadsheets in the core shed, checked and validated and uploaded to master spreadsheet; subsequently the logging sheets have been uploaded to a fully integrated GDMS system with further validation and checking • Spreadsheet uses pick lists and extensive code tables to standardise data capture; codes entered populate description fields used to verify code entry; during upload to master spreadsheet data range checking and further validation was conducted; GDMS system also provides data and code validation. • Historic data is contained in logging sheets and these have been captured in the Excel spreadsheet format, validated and checked. • Besra logging of lithology, alteration, mineralisation, structure and orientation, recovery, geotechnical and density was undertaken as routine data collection; additionally geomechanical logging was also conducted by a geotechnical engineer as routine • Historic core was systematically reviewed and re-logged/re-interpreted where appropriate by the geologists and assigned to the appropriate logging workbook. • All Besra core was photographed (wet and dry) prior to being logged by geologists with each tray clearly marked with drillhole identification and the interval from beginning of the tray to the end of the tray. All photos are collated electronically and indexed. • All drillcore and RC chips are stored at the core shed in Bau, along with sample pulps and coarse rejects. • Observations of historic drill core shows that all previous companies involved systematically geologically logged data onto paper logs with adequate geological descriptions, sample intervals marked, and correlated to assay data, to lead to the conclusion that systematic procedures were followed in most cases to the accepted standard at the time.
Sub-sampling techniques and sample preparation	• Half core samples are taken using a diamond core saw; majority of historic drillholes were done in the same manner, with only a small amount of very early holes done by core splitter. • The core is then delivered to the cutting room where the field technicians under the supervision of the geologist responsible for each drill hole cuts the core in half using one of the four Clipper core saws installed in 2010. • Density determinations have been carried out routinely on drill core with 10 centimetre cylinders of whole core taken between 10 metres and 20 metres downhole or wherever there is a change in lithology. The method used is a displacement method with samples air dried, weighed, and then sprayed with polyurethane to seal them. They are then weighed again in air and then in water and the density determined using the standard formula.
Quality of assay data	• The sample is dried at a temperature of approximately 100°C. The total sample is then put through a jaw crusher (less than 10mm) followed a Rocklabs Boyd crusher (less than 4mm);

Criteria	Section 1 – Commentary
and laboratory tests	the sample is then riffle split twice with ½ sample being pulverized in an LM3 with 90% passing 75µm; 2 x 150g samples are then packaged with one sample going for Fire Assay and the other for ICP analysis; all sample pulps and coarse rejects are bagged and stored for usage as required (period of 3 months), and thereafter returned to Besra for storage at the core shed in Bau. • Assay data quality was determined by Besra through the submission of standards (Rocklabs SE58, SG56, SK52, SN60, SG40 & SG50), field and laboratory duplicates and blanks were inserted at a nominal interval of 1 sample per 10 samples, except for blanks and standards which are inserted at 1 in 30. • SGS also insert their own duplicates and standards and report these in their monthly reporting. Also reported were percentages passing and not passing 75µm with associated duplicate assays in the Au assay return. • Au grades are determined by 50g Fire Assay (FAA505) with an AAS finish with a detection limit of 0.01ppm. • All other elements (23) are determined by ICP (SGS methods ICP12S, IMS12S, AAS12S & CSA06V); where values exceed detection limit these are then analysed by alternate methods with higher upper limits (e.g. AAS42S) • Standards: the majority of the standards have performed reasonably well with a slight tendency to report on the lower side of the expected value based on the 95 percentile values. Most fall within plus or minus 5% of the expected value. • Field & preparation duplicates: Comparison of the field duplicate plots for Jugan and BYG-Krian shows that correlation coefficients for field duplicates are close to one (1), ranging from 0.9923 to 0.9918; for preparation duplicates the correlation coefficient from 0.9867 to 0.9923 • Laboratory duplicates: the log-log plot of SGS duplicates compiled by Besra shows a correlation coefficient of 0.9848 • Historic drillholes: Gencor and RGC used their own protocols of duplicates, standards, blanks and umpires that were to industry standards of the 1990's. BYGS / Menzies Gold had a rigorous QAQC protocols. All historic QAQC values where available have been captured and analysed. • A full summary of the QAQC and associated sample handling is contained in the appropriate section of the Feasibility Study report.
Verification of sampling and assaying	• NBG routinely sends pulps from approximately 10% of all its samples to a separate independent laboratory for umpire analysis and the results compared, with no significant bias that would affect any resource classification • During the audit process during 2010 on historic drillholes a randomly selected group were sent to SGS Waihi, New Zealand for checking. No significant discrepancies were found. • Possible discrepancies in historic data have been re-sampled (quarter core or coarse rejects) and validated/checked with the discrepancies if occurring resolved. These were re-assayed at SGS.
Location of data points	• Drillhole surveying and orientation readings. All drill holes are routinely surveyed using either single shot or multi-shot downhole cameras. For the most part Camteq Proshot multi-shot electronic cameras were the norm. Drillhole surveys were taken every 25 metres downhole for all drillholes. Each hole was also surveyed at its termination. Orientation data was collected electronically using an Orishot orientation device. This was routinely done at

Criteria	Section 1 – Commentary
	the end of each HQ drill run where the driller judged he would be able to appropriate to obtain usable information. Drill runs normally ran with the core barrel length of between 1.5 metres and 3.0 metres. Orientation data was supplied electronically to prevent transcription errors. • All drillhole collars were surveyed by registered surveyors using differential GPS and or total station, and recorded in the database. All surveys are based on registered and recognised survey stations in the area, including the Land & Survey check station on top of the Jugan deposit. • Historic drillholes collars were captured by the then registered surveyors (by theodolite or total station) working on the project with the majority of the drillholes be resurveyed and checked by current surveyors (as per above); majority of the drillholes were within reasonable survey tolerances, with those outside being adjusted to the re-surveyed value. • Downhole surveys are checked mathematically and visually in the database and in 3D in the CAE Mining Studio geological and mining software package. Any surveys with recorded errors of unacceptable deviations were excluded from the downhole desurvey process. • Topographic digital terrain models were created and used to check the drillhole collars, based on a grid point and topographic surveys, with any obvious errors being resurveyed. • Historic drillholes did not have down hole surveys conducted and only had drillhole orientation conducted at the collar; the majority of these holes are shallow (<100m) and vertical, and any deviation is considered minor. • Channels/trenches were surveyed at start and end by registered surveyors and orientation and dip along the channel recorded; channels were checked against the topographic surveys.
Data spacing and distribution	• Besra drilling at Jugan has been undertaken on nominal NW-SE 25m spaced section lines. • Majority of historic drilling at Jugan and BYG-Krian is vertical on a nominal 25-50m grid, with a number of generations of drillholes creating a near surface drillhole spacing of less than 25m • Besra drilling at BYG-Krian was undertaken on nominal W-E 50m spaced section lines, with infill drilling in the main part of the orebody at 25m intervals; drilling of orebody extensions to the W were partially infilled with 25m spaced drillholes. • All Besra drillholes (Jugan and BYG-Krian) are angled and orientated core drilling used – the predominant drillhole angle is 60°, with a few drillholes drilled flatter at 45-55° and steeper up to 70° mainly due to practical and accessibility reasons. • 252 drillholes were drilled on and around the Jugan deposit with 206 drillholes intercepting mineralisation; of this 206 only 17 were RC drillholes • For BYG-Krian 288 drillholes were drilled in and around the deposit; of these 203 drillholes intercepted mineralisation; of these only 59 being RC; these RC holes were only used, in conjunction with diamond holes, to define the inferred zone areas. • 93-94% of all recent Besra drillholes intercepted mineralisation at Jugan and BYG-Krian • 1m assay composites were used, except where ore mineralisation boundaries limit the drillhole length to less than 1m. • Channel/trench was nominally orientated perpendicular the long axis of the hill outcrop at Jugan and spaced at 20-25m laterally; a few ad-hoc trenches were orientated obliquely due to practical, access reasons and orebody outcrop orientation.

Criteria	Section 1 – Commentary
Orientation of data in relation to geological structure	• Besra drilling at Jugan has been undertaken on nominal NW-SE 25m spaced section lines which is perpendicular to the orebody strike; infill holes and twin holes are done on an ad-hoc basis and orientation to check and validate the historic drillholes whilst trying to maintain a NW/SE orientation • All Besra drillholes (Jugan and BYG-Krian) are angled and orientated core drilling used – the predominant drillhole angle is 60°, with a few drillholes drilled flatter at 45-55° and steeper up to 70° mainly due to practical and accessibility reasons. • Majority of historic drilling at Jugan and BYG-Krian is vertical. • There is no expected bias due to the orientation of the drilling and the orebody strike continuity. • The great majority of the drilling is drilled through the orebody/deposit mineralised structures.
Sample security	• All samples are packaged in secure cloth bags and transported to SGS approximately 300 metres to SGS where they are received by SGS staff. The samples are recorded, batch numbers assigned by SGS and they pass into their system. Once samples are prepped the split for Fire Assay is retained at SGS for analysis while the split for ICP is sent via SGS's secure transport systems to SGS Perth or Port Klang via their freight system using DHL in Kuching. • Having the gold analyses carried out at SGS's laboratory on the Bau Mine Site eliminates a lot of security issues. • Only authorized NBG personnel are allowed access to the SGS sample preparation and laboratory areas and release of data only comes from the authorized laboratory manager to specific authorized senior personnel at NBG the Geology Manager, General Manager and Exploration Director. • The geologists fill out standard instruction forms for SGS and the samples are delivered to the SGS lab sample reception area where they pass into the SGS sample preparation and processing system. Besra sample dispatch numbers and SGS lab batch numbers are used to track and cross-check samples.
Audits or reviews	• Lab audits and checks by Besra have shown no material issues • Historic data has been audited in 2010 by Stevens & Associates geological consultant and Terra Mining Consultants Ltd, with no matters that were serious or were likely to impair the validity of the sampling data and any subsequent use in the Mineral Resource estimates or Ore Reserve work. • SGS conduct their own internal audits and reviews which are relayed to Besra. • Previous validation and review of the historic data has been conducted by a number of parties including Snowden & Associates, Australia and Ashby Consultants, New Zealand with no material problems being raised.

Section 2: Reporting of Exploration Results

No exploration results have been reported in this release, and thus, this section is not material to this report on Ore Reserves.

Section 3: Estimation & Reporting of Mineral Resources

No Mineral Resource results or updates have been reported in this release, and thus, this section is not material to this report on Ore Reserves.

Section 4: Estimation & Reporting of Ore Reserves

Criteria	Section 4 – Commentary
Mineral Resource estimate for conversion to Ore Reserves	• Mineral Resources used for conversion to Ore Reserves are from the Measured or Indicated category. Any Inferred material that may fall within the reserve is treated and reported as waste. • The mineral resources used were defined and updated between August 2010 and November 2012 using the JORC 2004 Code. • Mineral Resources are reported inclusive of the Ore Reserves
Site visits	• Competent person is on site on a permanent basis and supervised or undertook directly work on the exploration, resource drilling and Feasibility Study. The competent person has been intricately involved in the project for the past 4 years. The sites are located in flat lying agricultural land with no water or topographic features that may influence the modifying factors of the Ore Reserve.
Study status	• A full detailed Feasibility Study was conducted and released with the announcement. • As part of this feasibility study, a number of mine optimisations and plans were developed with the base case option and various alternates being economically viable. The mine plan considered mining, geotechnical, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental modifying factors which are detailed in the Feasibility Study.
Cut-off parameters	• Cut-off grades were determined using suitable financial parameters, mining parameters, etc. in the pit optimisations. The cutoff values range from 0.39 g/t to 0.44 g/t for Jugan pit and 0.58 g/t to 0.65 g/t for BYG-Krian pit.
Mining factors or assumptions	• The mining method is planned as traditional open pit mining utilising hydraulic excavators and in-pit trucks for haulage to ROM, dump or TSF construction. Rock breaking will be a combination of free digging, rip and dig using dozer and drill-and-blast depending upon the rock characteristics. Two ramps are designed, one carrying ore and the other waste and positioned relative to ROM and dump/TSF. A surface traffic system has been designed to handle traffic flows. • As the deposit is near surface an open pit mining method is selected. The deposit outcrops on a hill with little or no waste cover, therefore no pre-strip is applicable. • Both contract-mining and owner-operator methods investigated, with contract

Criteria	Section 4 – Commentary
	mining the preferred option at this stage. • Both pit optimisation and detailed designs were undertaken, with very small differences occurring between these. Therefore the pit optimisations have been accepted as being suitable due to the block and data resolution. • Detailed geotechnical logging of drillholes and 3D modelling of geotechnical parameters was undertaken and the slope and bench parameters resulting from this were used in the pit design elements and optimisation slope angles. Slope angles, configurations and zoning are based on the geotechnical domains and 3D locations within the designed pit and scheduled extraction. • For open pit inventory, the resource block model estimation methodology incorporates dilution and provides a reasonable estimate of mined tonnage and grades. Due to the nature of the orebody there are small waste zones, which are unable to be modeled discretely, and are incorporated within the overall ore zone. These can be found in the grade model with no or minor Au grade. This internal dilution is included within the overall reserves and would form the highest percentage of dilution. However, an additional 5% dilution is added. • A 95% mining recovery factor is used. • A minimum mining width of 50m was applied, with a minimum volume used in the optimisation to prevent unpractical islands or pit configurations. • It has been assumed industry standard grade control techniques would be used, but these have not been defined in detail. • Strip ratios for Jugan were 1.6/1.47 for owner-operator and contract-mining options, respectively; for BYG-Krian the strip ratios were 4.4/3.9. • 24/7 mining operations assumed • An average gold price of $1,300/oz was used in the cost modelling, with a range of gold prices used from $1,100 - $2,000/oz in the optimisation and cost model analyses. $1,300/oz Au was used as being a conservative value below the 2013 average ($1,415.48). • US$ used in all pricing; where local Malaysian pricing applicable a MYR : USD exchange rate of 3.2 : 1 • Mining costs used are $1.74/t for the base mining cost (overburden stripping) with MCAF of 1.52 for ore and 1.34 for waste • Processing costs used are $7.57/t for the base case concentrate option (processing costs for other process methods were $30.49 for BIOX, $27.56 for POX and $37.28 for Albion) • G&A and other costs were estimated at $0.16/g Au in the optimisation • A variety of production tonnage options were investigated with the base case option of 8,000tpd average used in the schedules and cost models, and the reserves. Suitable ramp up and tail off in production rates were incorporated. • Any minor amounts of inferred material that inadvertently fall within the open pit

Criteria	Section 4 – Commentary
	and reserve model are treated as waste with no content. • Inferred Resources were investigated internally but are not included in the Reserves. Inferred material may be included with further resource definition work to varying degrees. • Infrastructure requirements for the selected mining methods were taken into consideration as part of the feasibility study – include, but not limited to, TSF, haul roads, waste dump, mine offices, pumping requirements, etc., etc.

Metallurgical factors or assumptions	• A number of metallurgical processes were investigated including POX, BIOX and Albion. However, the selected option is the creation of a gold concentrate from a simple crush, grind and flotation process, with a drying/bagging of the concentrate for shipment.
	• These processes and the one selected are not novel in nature and well tested.
	• Detailed metallurgical work has been conducted and detailed optimisation work is still underway. Factors applicable to the metallurgical process have been modelled in 3D in the resource model along with the Au. These are As, Fe and S insitu content. Future metallurgical factors are proposed to be included in the model.
	• Overall recovery is estimated to be 77% for base case flotation option. The concentrate recovery option is based on a flotation recovery, recovery for contract processing facility and their percentage of metal content. Note, contract processing recoveries are not provided as these are commercially sensitive and under negotiation at present.
	• High levels of clay are present and processes to remove this (de-sliming, etc.) before flotation have been incorporated and further optimisation work is ongoing.
	• Bulk samples from near surface and drillcore from resource drilling as well as specific metallurgical drillholes have been used for all the metallurgical testing at recognised laboratories or in-house. Samples used are from across the full strike length and depth of the orebody. Detailed mineralogy and gold deportment studies have been undertaken.
	• Base case flotation summary:
	  o The Jugan ore exhibits a very low abrasion index and moderate bond ball mill work index (12.3 kWh/t).
	  o The assay data for the Jugan ore zones indicate that there is very little difference with respect to mineral distributions in the ore zones apart from minor variations in arsenic and gold contents. The increases in arsenic coincide with increases in gold showing an evident correlation. Based on sulphide sulphur and arsenic assays the ore is estimated to contain between 2 and 2.5 wt % arsenopyrite and 4.5 to 5 wt % pyrite with a combined arsenopyrite-pyrite in the feed in the range 6.5 to 7.5 wt %.
	  o The mineral assemblage is identical for all the Jugan ore zones tested across the deposit. The bulk of the Jugan ore feeds comprise non-sulphide gangue

Criteria	Section 4 – Commentary
	which is dominated by very fine grained illite (mica) and silica. This results in production of excessive slimes after fine grinding. o Gold deportment testing showed that very little gold is leached in whole ore cyanidation (0.6 to 2%). About 70% of the gold is associated with the arsenopyrite, 25% with the pyrite and 5% with silica. o In excess of 95% of the gold can be recovered in rougher – scavenging flotation. Due to varying slime entrainment the mass pull varied between 17 and 33 wt%. To mitigate the effect of feed slimes the flotation feed will be first deslimed by cyclone or a continuous gravity concentration. Flotation feed desliming test work is still in progress. o Bulk rougher-scavenger followed by cleaner flotation without prior desliming has shown that 90% of the gold can be recovered in a mass pull of 10 wt %. This corresponds to a gold upgrading ratio of 9:1 with respect to the feed grade. Mineralogical composition of a cleaner concentrate showed that the arsenopyrite and pyrite account for 67.4 wt % of the cleaner flotation concentrate. o Results indicate that inclusive of a desliming step, the flotation gold upgrade factor in the rougher circuit will be approximately 9 and in the cleaner stage greater than 2, giving an anticipated concentrate grade of +30 g/tAu
Environmental	• Waste rock and ore material have been tested for their NAF/PAF potential – both static and kinetic testing. Their treatment and impoundment (including neutralization, lining and containment) have been considered to prevent any acid mine drainage issues. • Baseline and preliminary EIA studies have been completed and the EIA Report and submission to local government will happen shortly. The only baseline work not completed to date is the geo-hydrology – awaiting drilling completion. • An initial conceptual MRP and ongoing updates have been submitted to the relevant authorities and have been accepted. A detailed MRP based on the Feasibility Study will be submitted along with the EIA.
Infrastructure	• The project area is centred on the township of Bau some 40 km WSW of the state capital and port of Kuching. • The Bau Project generally has good infrastructural aspects both within Bau Township and in Kuching. The main infrastructural features are: o Regular and reliable international air services to Kuching from Kuala Lumpur, Singapore, Hong Kong and Indonesia. Airport is only a thirty-five to forty (35-40) minute drive from the project area; o Two (2) ports with good dock and storage facilities (port has a capacity for vessels up to 17,000 tonnes); o Two (2) main sealed trunk roads from Kuching for delivery of supplies, heavy

Criteria	Section 4 – Commentary
	plant and equipment to the plant site; o Excellent labour and engineering support services; o Easy Accessibility – project extremities are less than a twenty (20) minute drive from the exploration base, and all important mines and gold prospects are linked by road; o Area is serviced with power and water; o The official language in Sarawak is Bahasa Malaysia, but most local communities speak English as a second language and have their own local dialects; o Well educated workforce (90% of population have received a secondary education); o An active quarrying industry focused mainly on limestone and marble for roading aggregates and agricultural purposes; o Ready supply of earthmoving equipment that supports the quarrying industry; o A local labour source with mining experience gained from the quarrying industry and past gold mining activity.
Costs	• Detailed Feasibility Study Capital and Operating costing has been applied. Costs are based on detailed quotes and/or derived from first principles. A full cost model incorporating all capital and operating costs has been compiled and based on the mining schedule(s). Quantities and amounts involved in the costing are derived from detailed designs, equipment configurations, layouts and usage quantities. Suitable factors have been applied to cover practical and reasonable variations to the costing, and where applicable conservative approaches and values have been used. Benchmarking of costs has been undertaken for key cost items. • Initial Capital - $92.1M; ongoing capital - $42.8M; total capital of $134.9M • Operating cost per tonne averages $31.38 & all in sustaining cost per ounce is $1,030.61 • Exchange rates used are as supplied by credible institutions including our current actual exchange rates realised. • Concentrate processing is based on supplied letters form potential processors and refining charges also, including penalties and costs, along with metal content payable. The concentrate processing details are not published here as Besra is awaiting additional offers and negotiating with suppliers of current payables and TC's. This information is commercially sensitive and details are not included for that reason. • Import duties are applied where applicable, or materials sourced (particularly from within Malaysia) are already inclusive of import taxes. There is scope for savings in this area as some imported items (associated with mining) are exempt from import taxes.

Criteria	Section 4 – Commentary
	• All royalties have been catered for – there currently is zero royalty on gold and the export of gold concentrate does incur any export duties. Licence fees for associated tenements have been paid to date.
Revenue factors	• A range of gold prices have been used in the cost modelling and optimisation work to determine the impacts and variances. • An average gold price of $1,300/oz was used in the cost modelling and schedule, with a range of gold prices used from $1,200 - $2,000/oz in the optimisation and cost model analyses. $1,300/oz Au was used as being a conservative value below the 2013 average ($1,415.48). Note, all pricing is in US$
Economic	• A discount rate of 8% has been used in all calculations and pit optimisations. • No inflation rates have been applied to the costing. • A range of sensitivities were conducted – gold price from $1,100 to $2,000/oz; + and – percentage ranges on processing costs, mining costs, capital cost, average mined grade and process recovery – both in terms of the effect on NPV and IRR • Resultant economics for the base case option(s) are NPV_8 of $91.4M and IRR of 38.0% for contract mining, and NPV_8 of $97.3M and IRR of 34.3% for owner-operator • The estimate inputs for the flotation concentrate base case (operating and capital costs) are at ± 15% and is as expected for this study case. Other processing methods were assessed at PFS level (±25%) and primarily used as a comparison to the preferred flotation concentrate option. • Gold price, grade and recovery show the highest level sensitivities, with lower sensitivities for the other elements analysed. • A number of tax incentives are available and these are currently being investigated. No tax incentives were applied to the cost model, and this may provide some upside to the project. • 650 cost model scenarios were developed with the main 40 scenarios investigated in further detail. Sensitivities and impacts were analysed across the main scenarios with the base case options receiving the most detailed analysis and these are outlined in the Feasibility Study report.
Social	• External and internal studies indicate no impediment for a social licence to operate.
Other	• Risk assessments were conducted and a risk matrix developed as part of Feasibility Study, with no major risk determined that is likely to limit or stop the project. • All tenements covering the mining and plant areas are fully granted for the 20 year maximum period. Part of the infrastructure is on a currently granted tenement that expires in Nov 2014. This tenement has been re-applied for a year in advance (Nov 2103) and will be an application renewal after the expiry date, with existing use

Criteria	Section 4 – Commentary
	rights and priority in time status. It is expected that the licence renewal will be issued soon and should be well in place before operations commence in late 2015.
Classification	• Based on the above and the detailed work in the Feasibility Study the Measured Resources have been converted to Proven Reserves and the Indicated Resources to Probable. No downgrading of Measured Resources to Probable Reserves has been done. This competent person considers the result is reflected appropriately in the classification of Reserves.
Audits or reviews	• A high level review and risk assessment has been undertaken by a third party, along with suitable benchmarking with other sites/projects and internal reviews/checks undertaken.

A16-1. Pit Optimisation – Economic Model Parameters

ECONOMIC MODEL PARAMETERS USED FOR PIT OPTIMISATION

- Metal (AU) Price fixed at US$ 1500/oz in all options
- Metallurgical Recovery changes relative to process options
- Dilution & Mining Recovery – fixed relative to mining rates & process options
- Mining Costs (ore & Waste) – changes relative to mining rates
- Processing Costs – changes relative to process options
- Parameters are the same for Jugan and BYG-Krian, except <u>the concentrate shipping cost</u>

<u>Economic Model Parameters – For 4000 TPD</u>

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Base Mining Cost (Owner Mining)	$/tonne	2.366	2.366	2.366	2.366
MCAF – Ore		1.273	1.273	1.273	1.273
MCAF – Waste/Intrusive		1.228	1.228	1.228	1.228
Base Mining Cost (Contractor)	$/tonne	3.056	3.056	3.056	3.056
MCAF – Ore		1.291	1.291	1.291	1.291
MCAF – Waste/Intrusive		1.243	1.243	1.243	1.243
Incremental Cost per Bench	$/tonne	-	-	-	-
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost (Jugan)	$/g	2.91	-	-	-
Concentrate Shipping Cost (BYG)	$/g	1.90	-	-	-

<u>Economic Model Parameters – For 6000 TPD</u>

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Base Mining Cost (Owner Mining)	$/tonne	2.108	2.108	2.108	2.108
MCAF – Ore		1.321	1.321	1.538	1.321
MCAF – Waste/Intrusive		1.286	1.286	1.341	1.286
Base Mining Cost (Contractor)	$/tonne	2.724	2.724	2.293	2.724

Parameters	Units	Flotation	POX	BIOX	ALBION
MCAF – Ore		1.342	1.342	1.576	1.342
MCAF – Waste/Intrusive		1.304	1.304	1.365	1.304
Incremental Cost per Bench	$/tonne	-	-	-	-
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost (Jugan)	$/g	2.91	-	-	-
Concentrate Shipping Cost (BYG)	$/g	1.90	-	-	-

Economic Model Parameters – For 8000 TPD (base case mining rate)

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Base Mining Cost (Owner Mining)	$/tonne	1.787	1.787	1.787	1.787
MCAF – Ore		1.538	1.538	1.538	1.538
MCAF – Waste/Intrusive		1.341	1.341	1.341	1.341
Base Mining Cost (Contractor)	$/tonne	2.293	2.293	2.293	2.293
MCAF – Ore		1.576	1.576	1.576	1.576
MCAF – Waste/Intrusive		1.365	1.365	1.365	1.365
Incremental Cost per Bench	$/tonne	-	-	-	-
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost (Jugan)	$/g	2.91	-	-	-
Concentrate Shipping Cost (BYG)	$/g	1.90	-	-	-

Economic Model Parameters – For 10000 TPD

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Base Mining Cost (Owner Mining)	$/tonne	1.692	1.692	1.692	1.692
MCAF – Ore		1.445	1.445	1.445	1.445

Parameters	Units	Flotation	POX	BIOX	ALBION
MCAF – Waste/Intrusive		1.339	1.339	1.339	1.339
Base Mining Cost (Contractor)	$/tonne	2.168	2.168	2.168	2.168
MCAF – Ore		1.478	1.478	1.478	1.478
MCAF – Waste/Intrusive		1.364	1.364	1.364	1.364
Incremental Cost per Bench	$/tonne	-	-	-	-
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost (Jugan)	$/g	2.91	-	-	-
Concentrate Shipping Cost (BYG)	$/g	1.90	-	-	-

Economic Model Parameters – For 12000 TPD

Parameters	Units	Flotation	POX	BIOX	ALBION
Gold Price	$/oz	1,500	1,500	1,500	1,500
Selling Cost	$/g	0.16	0.45	0.45	0.45
Mining Recovery	%	95	95	95	95
Mining Dilution	%	5	5	5	5
Base Mining Cost (Owner Mining)	$/tonne	1.614	1.614	1.614	1.614
MCAF – Ore		1.495	1.495	1.495	1.495
MCAF – Waste/Intrusive		1.363	1.363	1.363	1.363
Base Mining Cost (Contractor)	$/tonne	2.063	2.063	2.063	2.063
MCAF – Ore		1.532	1.532	1.532	1.532
MCAF – Waste/Intrusive		1.391	1.391	1.391	1.391
Incremental Cost per Bench	$/tonne	-	-	-	-
Rehab Cost	$/tonne	0.10	0.10	0.10	0.10
Process Cost	$/tonne	7.19	26.92	29.38	38.81
Process Recovery	%	77	85	80	80
Concentrate Shipping Cost (Jugan)	$/g	2.91	-	-	-
Concentrate Shipping Cost (BYG)	$/g	1.90	-	-	-

A16-2. Pit Optimisation – Ultimate Pit Results Summary

SUMMARY OF JUGAN (OWNER-OPERATOR) ULTIMATE PITS

Mining Rate/Process Option			ULTIMATE PIT		RESERVES (Meas+Ind)		Measured		Indicated		Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	g/t	tonnes	g/t	tonnes	g/t	tonnes	tonnes	tonnes	tonnes	tonnes	Ratio
Owned	4000	Flotation	**Pit 67**	289,281,216	10,218,951	1.546	3,451,000	1.469	6,767,950	1.586	16,821,544	222,321	47,494	40,442	16,511,287	1.646
Owned	6000	Flotation	**Pit 68**	317,202,906	10,282,949	1.543	3,452,216	1.469	6,830,733	1.580	17,302,983	222,321	48,515	40,442	16,991,705	1.683
Owned	8000 (base)	Flotation	**Pit 68**	335,570,199	12,916,266	1.439	3,458,755	1.468	9,457,511	1.428	42,927,735	222,954	87,093	66,808	42,550,880	3.324
Owned	10000	Flotation	**Pit 67**	352,341,031	13,109,196	1.433	3,459,404	1.467	9,649,792	1.420	44,961,818	223,062	87,941	66,810	44,584,005	3.430
Owned	12000	Flotation	**Pit 67**	361,415,102	13,113,111	1.433	3,459,404	1.467	9,653,707	1.420	45,028,302	223,062	87,941	66,810	44,650,489	3.434
Owned	4000	POX	**Pit 70**	216,723,334	8,672,518	1.700	2,906,871	1.641	5,765,648	1.729	15,043,386	697,058	427,146	37,855	13,881,327	1.735
Owned	6000	POX	**Pit 65**	235,640,250	8,784,612	1.693	2,914,090	1.639	5,870,523	1.720	15,647,253	713,367	454,131	37,855	14,441,900	1.781
Owned	8000 (base)	POX	**Pit 64**	247,869,297	8,851,758	1.688	2,917,673	1.638	5,934,085	1.713	15,933,155	719,303	475,197	37,855	14,700,800	1.800
Owned	10000	POX	**Pit 65**	257,518,385	8,875,793	1.687	2,917,673	1.638	5,958,120	1.710	16,066,146	719,303	483,089	37,855	14,825,899	1.810
Owned	12000	POX	**Pit 64**	262,359,417	8,888,787	1.686	2,918,018	1.638	5,970,769	1.710	16,235,774	719,303	483,089	37,855	14,995,527	1.827
Owned	4000	BIOX	**Pit 58**	176,538,837	7,996,941	1.759	2,667,258	1.720	5,329,684	1.779	13,595,784	892,360	496,698	26,922	12,179,804	1.700
Owned	6000	BIOX	**Pit 61**	191,282,711	8,099,591	1.754	2,673,238	1.718	5,426,353	1.771	14,188,762	908,225	543,031	26,922	12,710,584	1.752
Owned	8000 (base)	BIOX	**Pit 62**	201,131,233	8,164,716	1.755	2,674,847	1.718	5,489,870	1.772	15,042,586	908,717	558,285	36,013	13,539,571	1.842
Owned	10000	BIOX	**Pit 69**	209,263,505	8,191,231	1.753	2,675,521	1.718	5,515,710	1.770	15,182,332	908,941	571,173	37,855	13,664,363	1.853
Owned	12000	BIOX	**Pit 64**	213,083,158	8,204,847	1.752	2,675,521	1.718	5,529,326	1.769	15,325,952	908,941	575,738	37,855	13,803,418	1.868
Owned	4000	ALBION	**Pit 65**	124,307,365	6,556,497	1.904	2,152,051	1.905	4,404,446	1.903	12,330,016	1,362,470	863,828	23,828	10,079,890	1.881
Owned	6000	ALBION	**Pit 62**	133,643,289	6,680,178	1.904	2,152,988	1.905	4,527,191	1.903	13,558,842	1,362,470	921,851	23,847	11,250,674	2.030
Owned	8000 (base)	ALBION	**Pit 59**	140,373,134	6,733,720	1.9015	2,153,289	1.905	4,580,431	1.900	13,905,966	1,365,266	959,351	23,847	11,557,502	2.065
Owned	10000	ALBION	**Pit 63**	146,380,312	6,744,459	1.901	2,156,309	1.904	4,588,150	1.899	13,933,548	1,367,876	963,908	23,847	11,577,917	2.066
Owned	12000	ALBION	**Pit 64**	148,846,492	6,796,850	1.901	2,156,309	1.904	4,640,541	1.899	14,583,515	1,367,876	999,421	26,922	12,189,296	2.146

SUMMARY OF JUGAN (CONTRACT MINING) ULTIMATE PITS

Mining Rates/Process Option			ULTIMATE PIT		RESERVES (Meas+Ind)		Measured		Indicated		Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	g/t	tonnes	g/t	tonnes	g/t	tonnes	tonnes	tonnes	tonnes	tonnes	Ratio
Contract	4000	Flotation	**Pit 68**	272,889,678	10,005,517	1.555	3,442,369	1.471	6,563,148	1.599	15,054,310	222,182	39,842	37,855	14,754,431	1.505
Contract	6000	Flotation	**Pit 62**	299,705,583	10,031,949	1.553	3,444,909	1.470	6,587,040	1.597	15,203,805	222,321	39,905	37,855	14,903,724	1.516
Contract	8000 (base)	Flotation	**Pit 65**	317,428,122	10,114,132	1.552	3,445,104	1.470	6,669,028	1.594	16,047,317	222,321	40,749	37,855	15,746,392	1.587
Contract	10000	Flotation	**Pit 67**	331,365,992	10,201,613	1.548	3,449,679	1.469	6,751,934	1.588	16,742,496	222,321	47,494	40,442	16,432,239	1.641
Contract	12000	Flotation	**Pit 66**	338,503,255	10,234,559	1.546	3,451,000	1.469	6,783,559	1.585	17,002,572	222,321	47,494	40,442	16,692,315	1.661
Contract	4000	POX	**Pit 66**	201,853,276	8,395,704	1.711	2,890,409	1.645	5,505,295	1.745	13,161,062	672,356	360,832	23,847	12,104,027	1.568
Contract	6000	POX	**Pit 67**	219,762,953	8,473,028	1.707	2,897,012	1.643	5,576,016	1.740	13,575,206	680,649	386,411	26,922	12,481,224	1.602
Contract	8000 (base)	POX	**Pit 68**	231,895,164	8,547,655	1.704	2,898,811	1.643	5,648,844	1.735	14,028,218	684,377	400,551	26,922	12,916,368	1.641
Contract	10000	POX	**Pit 69**	242,188,005	8,626,888	1.703	2,901,562	1.642	5,725,326	1.734	14,886,085	686,871	415,186	37,855	13,746,173	1.726
Contract	12000	POX	**Pit 70**	246,964,779	8,672,519	1.700	2,906,871	1.641	5,765,648	1.729	15,043,386	697,058	427,146	37,855	13,881,327	1.735
Contract	4000	BIOX	**Pit 59**	162,338,205	7,790,118	1.761	2,655,727	1.723	5,134,391	1.781	11,863,887	890,603	438,540	23,828	10,510,916	1.523
Contract	6000	BIOX	**Pit 65**	176,285,524	7,858,315	1.759	2,662,806	1.721	5,195,509	1.779	12,284,817	891,374	445,372	23,828	10,924,243	1.563
Contract	8000 (base)	BIOX	**Pit 62**	185,992,988	7,923,629	1.763	2,664,505	1.721	5,259,124	1.784	13,183,729	891,374	462,023	23,847	11,806,485	1.664
Contract	10000	BIOX	**Pit 59**	194,835,041	7,988,857	1.759	2,666,426	1.720	5,322,431	1.779	13,502,549	892,360	493,073	26,922	12,090,194	1.690
Contract	12000	BIOX	**Pit 63**	198,605,464	7,996,942	1.759	2,667,258	1.720	5,329,684	1.779	13,595,784	892,360	496,698	26,922	12,179,804	1.700
Contract	4000	ALBION	**Pit 69**	111,047,932	6,320,734	1.908	2,138,530	1.909	4,182,204	1.908	10,842,626	1,346,828	785,884	12,464	8,697,450	1.715
Contract	6000	ALBION	**Pit 67**	120,055,210	6,440,384	1.910	2,142,633	1.908	4,297,751	1.911	11,784,711	1,351,339	833,755	12,464	9,587,153	1.830
Contract	8000 (base)	ALBION	**Pit 68**	126,634,883	6,462,553	1.908	2,145,015	1.9073	4,317,538	1.909	11,847,101	1,353,327	848,814	12,464	9,632,496	1.833
Contract	10000	ALBION	**Pit 70**	133,253,906	6,545,900	1.904	2,150,843	1.906	4,395,057	1.904	12,266,003	1,362,470	863,828	12,464	10,027,241	1.874
Contract	12000	ALBION	**Pit 66**	135,692,831	6,555,289	1.904	2,150,843	1.906	4,404,446	1.903	12,329,602	1,362,470	863,828	23,828	10,079,476	1.881

SUMMARY OF BYG-KRIAN (OWNER-OPERATOR) ULTIMATE PITS

Mining Rates/Process Option			ULTIMATE PIT		RESERVES (Ind)		Total Waste	Ind (w)	Inf (w)	Waste	Strip
Mining	**TPD**	**PROCESS**	**Pit Shell**	**NPV**	**tonnes**	**(g/t)**	**tonnes**	**tonnes**	**tonnes**	**tonnes**	**Ratio**
Owned	4000	Flotation	**Pit 67**	82,644,747	1,046,785	3.086	4,608,922	8,736	22,264	4,577,922	4.403
Owned	6000	Flotation	**Pit 68**	84,540,929	1,050,274	3.080	4,642,227	8,736	22,267	4,611,224	4.420
Owned	8000 (base)	Flotation	**Pit 65**	86,432,838	1,060,194	3.065	4,807,623	8,736	22,482	4,776,405	4.535
Owned	10000	Flotation	**Pit 67**	87,673,375	1,217,365	2.754	5,648,898	14,098	24,307	5,610,493	4.640
Owned	12000	Flotation	**Pit 66**	88,252,404	1,219,778	2.752	5,699,221	14,098	24,307	5,660,816	4.672
Owned	4000	POX	**Pit 58**	78,946,125	922,578	3.393	4,232,607	49,245	14,544	4,168,818	4.588
Owned	6000	POX	**Pit 53**	80,617,377	929,787	3.382	4,345,149	49,626	15,004	4,280,519	4.673
Owned	8000 (base)	POX	**Pit 59**	82,314,966	934,804	3.373	4,423,212	50,056	19,018	4,354,138	4.732
Owned	10000	POX	**Pit 61**	83,384,318	944,328	3.354	4,531,414	51,261	19,917	4,460,236	4.799
Owned	12000	POX	**Pit 62**	83,849,018	945,094	3.353	4,558,339	51,261	19,917	4,487,161	4.823
Owned	4000	BIOX	**Pit 56**	70,106,874	866,332	3.531	3,912,503	64,846	12,265	3,835,392	4.516
Owned	6000	BIOX	**Pit 58**	71,580,182	874,773	3.516	4,026,913	66,990	12,353	3,947,570	4.603
Owned	8000 (base)	BIOX	**Pit 54**	73,172,040	884,136	3.511	4,301,995	71,802	12,980	4,217,213	4.866
Owned	10000	BIOX	**Pit 60**	74,170,171	888,309	3.501	4,335,007	71,823	13,908	4,249,276	4.880
Owned	12000	BIOX	**Pit 56**	74,593,785	889,040	3.499	4,342,076	72,237	14,433	4,255,406	4.884
Owned	4000	ALBION	**Pit 55**	63,058,614	742,791	3.901	3,441,575	105,033	10,211	3,326,331	4.633
Owned	6000	ALBION	**Pit 51**	64,260,706	744,714	3.898	3,480,024	105,067	10,418	3,364,539	4.673
Owned	8000 (base)	ALBION	**Pit 56**	65,596,076	774,909	3.848	4,120,208	129,469	10,999	3,979,740	5.317
Owned	10000	ALBION	**Pit 64**	66,505,550	776,894	3.844	4,156,575	129,631	11,080	4,015,864	5.350
Owned	12000	ALBION	**Pit 61**	66,883,831	779,400	3.841	4,222,916	129,631	11,532	4,081,753	5.418

SUMMARY OF BYG-KRIAN (CONTRACT MINING) ULTIMATE PITS

Mining Rates/Process Option			ULTIMATE PIT		RESERVES (Ind)		Total Waste	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	(g/t)	tonnes	tonnes	tonnes	tonnes	Ratio
Contract	4000	Flotation	**Pit 63**	78,149,600	1,004,058	3.141	3,947,382	7,199	16,126	3,924,057	3.931
Contract	6000	Flotation	**Pit 64**	80,111,816	1,012,325	3.127	4,030,187	7,199	19,422	4,003,566	3.981
Contract	8000 (base)	Flotation	**Pit 63**	82,234,736	1,026,886	3.109	4,251,086	7,953	21,597	4,221,536	4.140
Contract	10000	Flotation	**Pit 64**	83,650,458	1,046,247	3.087	4,602,848	8,736	22,264	4,571,848	4.399
Contract	12000	Flotation	**Pit 63**	84,253,340	1,046,916	3.086	4,611,456	8,736	22,267	4,580,453	4.405
Contract	4000	POX	**Pit 49**	74,924,926	869,185	3.482	3,498,384	36,840	12,124	3,449,420	4.025
Contract	6000	POX	**Pit 53**	76,595,985	881,141	3.456	3,599,401	38,115	12,795	3,548,491	4.085
Contract	8000 (base)	POX	**Pit 55**	78,477,697	908,996	3.412	3,993,529	46,330	14,349	3,932,850	4.393
Contract	10000	POX	**Pit 54**	79,727,227	921,391	3.395	4,207,701	48,045	14,544	4,145,112	4.567
Contract	12000	POX	**Pit 57**	80,262,467	924,568	3.393	4,294,132	49,245	14,997	4,229,890	4.644
Contract	4000	BIOX	**Pit 56**	66,370,944	820,619	3.619	3,351,712	49,212	11,391	3,291,109	4.084
Contract	6000	BIOX	**Pit 54**	67,897,251	827,037	3.607	3,426,029	53,791	11,391	3,360,847	4.143
Contract	8000 (base)	BIOX	**Pit 50**	69,589,828	836,701	3.585	3,511,167	55,017	11,484	3,444,666	4.196
Contract	10000	BIOX	**Pit 52**	70,698,385	858,778	3.542	3,781,291	62,516	11,646	3,707,129	4.403
Contract	12000	BIOX	**Pit 54**	71,170,612	865,705	3.533	3,911,968	64,846	12,265	3,834,857	4.519
Contract	4000	ALBION	**Pit 56**	59,524,980	717,409	3.973	3,242,051	93,691	8,773	3,139,587	4.519
Contract	6000	ALBION	**Pit 56**	60,860,293	723,490	3.956	3,292,573	94,692	9,149	3,188,732	4.551
Contract	8000 (base)	ALBION	**Pit 51**	62,403,483	731,175	3.934	3,351,646	96,967	9,620	3,245,059	4.584
Contract	10000	ALBION	**Pit 55**	63,415,415	741,147	3.907	3,437,451	105,033	9,620	3,322,798	4.638
Contract	12000	ALBION	**Pit 56**	63,816,104	742,791	3.901	3,441,575	105,033	10,211	3,326,331	4.633

A16-3. Pit Optimisation – Optimal Pit Results Summary

SUMMARY OF JUGAN (OWNER-OPERATOR) OPTIMAL PITS

Mining Rate/Process Option			OPTIMAL PIT		RESERVES (Meas+Ind)		Measured		Indicated		Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	grade (g/t)	tonnes	g/t	tonnes	g/t	tonnes	tonnes	tonnes	tonnes	tonnes	**Ratio**
Owned	4000	Flotation	**Pit 52**	288,557,077	9,951,932	1.557	3,445,955	1.470	6,505,977	1.604	14,751,345	222,321	39,131	36,632	14,453,261	1.482
Owned	6000	Flotation	**Pit 55**	316,517,503	10,055,787	1.552	3,446,387	1.470	6,609,401	1.594	15,331,911	222,321	39,905	37,855	15,031,830	1.525
Owned	8000 (base)	Flotation	**Pit 54**	334,526,740	10,157,774	1.549	3,452,670	1.469	6,705,104	1.590	16,288,378	222,415	40,749	37,860	15,987,354	1.604
Owned	10000	Flotation	**Pit 54**	347,897,970	10,216,606	1.547	3,453,349	1.469	6,763,256	1.587	16,830,913	222,523	47,494	40,442	16,520,454	1.647
Owned	12000	Flotation	**Pit 61**	360,574,282	12,618,903	1.446	3,459,239	1.467	9,159,664	1.437	39,518,338	223,062	72,943	51,982	39,170,351	3.132
Owned	4000	POX	**Pit 56**	216,446,115	8,476,356	1.707	2,902,274	1.642	5,574,082	1.741	13,604,476	686,488	379,605	26,922	12,511,461	1.605
Owned	6000	POX	**Pit 53**	235,143,601	8,517,818	1.704	2,905,073	1.642	5,612,745	1.736	13,751,875	690,672	389,409	26,922	12,644,872	1.614
Owned	8000 (base)	POX	**Pit 56**	247,674,286	8,701,011	1.698	2,912,748	1.640	5,788,262	1.727	15,133,808	706,000	439,463	26,922	13,961,423	1.739
Owned	10000	POX	**Pit 54**	257,196,908	8,725,423	1.696	2,912,748	1.640	5,812,675	1.724	15,255,154	706,000	440,768	26,922	14,081,464	1.748
Owned	12000	POX	**Pit 54**	261,991,821	8,732,592	1.695	2,913,094	1.640	5,819,498	1.723	15,289,681	706,000	441,974	26,922	14,114,785	1.751
Owned	4000	BIOX	**Pit 54**	176,509,263	7,970,702	1.759	2,667,258	1.720	5,303,444	1.779	13,346,172	892,360	477,345	23,847	11,952,620	1.674
Owned	6000	BIOX	**Pit 55**	191,223,426	7,993,195	1.759	2,669,067	1.720	5,324,128	1.779	13,522,084	892,360	493,073	26,922	12,109,729	1.692
Owned	8000 (base)	BIOX	**Pit 59**	201,126,428	8,150,746	1.755	2,674,218	1.718	5,476,528	1.774	14,964,108	908,225	557,029	26,922	13,471,932	1.836
Owned	10000	BIOX	**Pit 62**	209,237,143	8,156,922	1.755	2,675,521	1.718	5,481,401	1.773	15,002,431	908,906	557,160	26,922	13,509,443	1.839
Owned	12000	BIOX	**Pit 57**	213,048,275	8,162,360	1.755	2,675,521	1.718	5,486,839	1.773	15,035,982	908,906	557,172	26,922	13,542,982	1.842
Owned	4000	ALBION	**Pit 59**	124,258,564	6,506,472	1.905	2,152,051	1.905	4,354,421	1.905	11,983,653	1,362,155	856,442	12,464	9,752,592	1.842
Owned	6000	ALBION	**Pit 57**	133,566,243	6,550,169	1.904	2,152,988	1.905	4,397,182	1.903	12,265,811	1,362,470	868,524	12,464	10,022,353	1.873
Owned	8000 (base)	ALBION	**Pit 56**	140,313,292	6,686,129	1.903	2,153,289	1.905	4,532,840	1.902	13,573,728	1,362,470	923,573	23,847	11,263,838	2.030
Owned	10000	ALBION	**Pit 58**	146,261,353	6,696,248	1.902	2,156,309	1.904	4,539,939	1.901	13,591,508	1,365,080	928,129	23,847	11,274,452	2.030
Owned	12000	ALBION	**Pit 57**	148,692,588	6,697,479	1.902	2,156,309	1.904	4,541,170	1.901	13,598,063	1,365,080	928,144	23,847	11,280,992	2.030

SUMMARY OF JUGAN (CONTRACT-MINING) OPTIMAL PITS

Mining Rates/Process Option			OPTIMAL PIT		RESERVES (Meas+Ind)		Measured		Indicated		Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	g/t	tonnes	g/t	tonnes	g/t	tonnes	tonnes	tonnes	tonnes	tonnes	Ratio
Contract	4000	Flotation	**Pit 61**	272,797,845	9,913,621	1.560	3,442,369	1.471	6,471,252	1.607	14,575,626	222,182	39,130	26,922	14,287,392	1.470
Contract	6000	Flotation	**Pit 53**	299,476,065	9,917,605	1.560	3,444,386	1.470	6,473,219	1.607	14,583,646	222,182	39,130	26,922	14,295,412	1.470
Contract	8000 (base)	Flotation	**Pit 52**	316,737,006	9,920,498	1.559	3,444,581	1.470	6,475,917	1.607	14,605,704	222,182	39,130	26,922	14,317,470	1.472
Contract	10000	Flotation	**Pit 61**	331,290,877	10,092,966	1.554	3,445,104	1.470	6,647,862	1.597	15,966,499	222,321	40,749	37,855	15,665,574	1.582
Contract	12000	Flotation	**Pit 59**	338,364,094	10,094,006	1.554	3,445,955	1.470	6,648,051	1.597	15,970,492	222,321	40,749	37,855	15,669,567	1.582
Contract	4000	POX	**Pit 55**	201,588,692	8,189,576	1.715	2,889,480	1.645	5,300,096	1.753	11,733,410	671,390	314,264	12,464	10,735,292	1.433
Contract	6000	POX	**Pit 63**	219,708,634	8,396,940	1.711	2,891,393	1.645	5,505,547	1.745	13,169,131	672,356	360,832	23,847	12,112,096	1.568
Contract	8000 (base)	POX	**Pit 59**	231,726,051	8,432,274	1.709	2,892,655	1.645	5,539,619	1.743	13,387,954	672,356	378,097	26,922	12,310,579	1.588
Contract	10000	POX	**Pit 60**	241,916,116	8,472,537	1.707	2,899,483	1.643	5,573,054	1.741	13,591,234	682,451	379,593	26,922	12,502,268	1.604
Contract	12000	POX	**Pit 58**	246,573,925	8,475,565	1.707	2,901,483	1.642	5,574,082	1.741	13,600,129	682,452	379,605	26,922	12,511,150	1.605
Contract	4000	BIOX	**Pit 53**	162,253,438	7,727,498	1.763	2,654,581	1.723	5,072,917	1.784	11,522,584	883,239	422,049	12,464	10,204,832	1.491
Contract	6000	BIOX	**Pit 54**	176,048,762	7,740,633	1.762	2,661,659	1.721	5,078,974	1.783	11,555,602	884,010	424,001	12,464	10,235,127	1.493
Contract	8000 (base)	BIOX	**Pit 59**	185,986,135	7,912,835	1.763	2,664,505	1.721	5,248,330	1.785	13,103,977	891,374	459,472	23,847	11,729,284	1.656
Contract	10000	BIOX	**Pit 57**	194,787,547	7,943,170	1.761	2,666,426	1.720	5,276,744	1.782	13,211,436	892,360	468,243	23,847	11,826,986	1.663
Contract	12000	BIOX	**Pit 59**	198,529,289	7,944,160	1.761	2,667,258	1.720	5,276,902	1.782	13,218,215	892,360	468,243	23,847	11,833,765	1.664
Contract	4000	ALBION	**Pit 65**	110,891,728	6,178,660	1.909	2,137,162	1.9088	4,041,498	1.909	9,947,341	1,346,828	736,277	12,464	7,851,772	1.610
Contract	6000	ALBION	**Pit 62**	119,881,240	6,331,948	1.907	2,140,966	1.908	4,190,982	1.907	10,859,780	1,349,053	789,096	12,464	8,709,167	1.715
Contract	8000 (base)	ALBION	**Pit 64**	126,612,877	6,430,553	1.910	2,145,015	1.9073	4,285,538	1.912	11,747,257	1,353,327	824,052	12,464	9,557,414	1.827
Contract	10000	ALBION	**Pit 64**	133,216,238	6,505,264	1.905	2,150,843	1.906	4,354,421	1.905	11,983,239	1,362,155	856,442	12,464	9,752,178	1.842
Contract	12000	ALBION	**Pit 60**	135,627,522	6,505,264	1.905	2,150,843	1.906	4,354,421	1.905	11,983,239	1,362,155	856,442	12,464	9,752,178	1.842

SUMMARY OF BYG-KRIAN (OWNER-OPERATOR) OPTIMAL PITS

Mining Rates/Process Option			OPTIMAL PIT		RESERVES (Ind)		Total Waste	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	(g/t)	tonnes	tonnes	tonnes	tonnes	Ratio
Owned	4000	Flotation	**Pit 61**	82,599,543	1,036,976	3.088	4,331,056	8,420	21,597	4,301,039	4.177
Owned	6000	Flotation	**Pit 62**	84,502,745	1,044,129	3.086	4,520,067	8,420	21,606	4,490,041	4.329
Owned	8000 (base)	Flotation	**Pit_56**	86,405,519	1,051,312	3.077	4,637,900	8,736	22,385	4,606,779	4.412
Owned	10000	Flotation	**Pit 58**	87,612,405	1,060,487	3.062	4,773,386	8,736	22,522	4,742,128	4.501
Owned	12000	Flotation	**Pit 56**	88,127,290	1,060,720	3.062	4,777,133	8,736	22,522	4,745,875	4.504
Owned	4000	POX	**Pit 57**	78,945,277	922,020	3.393	4,216,139	48,045	14,544	4,153,550	4.573
Owned	6000	POX	**Pit 49**	80,604,588	924,908	3.388	4,245,780	49,587	14,552	4,181,641	4.590
Owned	8000 (base)	POX	**Pit 52**	82,309,179	931,535	3.378	4,359,797	49,769	15,636	4,294,392	4.680
Owned	10000	POX	**Pit 51**	83,363,218	936,021	3.367	4,379,772	50,395	16,528	4,312,849	4.679
Owned	12000	POX	**Pit 50**	83,819,199	936,487	3.366	4,385,425	50,395	16,528	4,318,502	4.683
Owned	4000	BIOX	**Pit 54**	70,105,382	861,631	3.535	3,800,329	63,151	11,646	3,725,532	4.411
Owned	6000	BIOX	**Pit 55**	71,573,667	870,967	3.520	3,943,342	66,865	12,346	3,864,131	4.528
Owned	8000 (base)	BIOX	**Pit 46**	73,013,175	869,054	3.516	3,844,040	64,482	11,916	3,767,642	4.423
Owned	10000	BIOX	**Pit 55**	74,165,295	886,374	3.504	4,299,970	71,802	13,812	4,214,356	4.851
Owned	12000	BIOX	**Pit 50**	74,586,371	886,642	3.503	4,302,338	72,215	14,337	4,215,786	4.852
Owned	4000	ALBION	**Pit 54**	63,057,848	742,295	3.902	3,428,855	105,033	10,211	3,313,611	4.619
Owned	6000	ALBION	**Pit 48**	64,254,785	743,193	3.898	3,431,289	105,033	10,211	3,316,045	4.617
Owned	8000 (base)	ALBION	**Pit 53**	65,593,587	770,764	3.854	4,025,982	127,944	10,992	3,887,046	5.223
Owned	10000	ALBION	**Pit 58**	66,487,939	771,907	3.851	4,037,092	128,230	11,073	3,897,789	5.230
Owned	12000	ALBION	**Pit 55**	66,854,870	772,022	3.851	4,039,385	128,230	11,073	3,900,082	5.232

SUMMARY OF BYG-KRIAN (CONTRACT-MINING) OPTIMAL PITS

Mining Rates/Process Option			ULTIMATE PIT		RESERVES (Ind)		Total Waste	Ind (w)	Inf (w)	Waste	Strip
Mining	TPD	PROCESS	Pit Shell	NPV	tonnes	(g/t)	tonnes	tonnes	tonnes	tonnes	Ratio
Contract	4000	Flotation	**Pit 49**	77,953,734	972,764	3.177	3,532,824	7,133	15,804	3,509,887	3.632
Contract	6000	Flotation	**Pit 56**	80,099,575	1,005,425	3.137	3,948,698	7,199	16,126	3,925,373	3.927
Contract	8000 (base)	Flotation	**Pit 50**	82,146,335	1,007,380	3.133	3,963,619	7,199	16,282	3,940,138	3.935
Contract	10000	Flotation	**Pit 46**	83,463,536	1,008,087	3.132	3,968,161	7,199	16,282	3,944,680	3.936
Contract	12000	Flotation	**Pit 53**	84,142,459	1,021,034	3.116	4,160,614	7,953	21,593	4,131,068	4.075
Contract	4000	POX	**Pit 49**	74,924,926	869,185	3.482	3,498,384	36,840	12,124	3,449,420	4.025
Contract	6000	POX	**Pit 51**	76,590,008	874,158	3.469	3,522,677	36,840	12,795	3,473,042	4.030
Contract	8000 (base)	POX	**Pit 48**	78,459,678	901,235	3.419	3,832,527	45,394	13,752	3,773,381	4.253
Contract	10000	POX	**Pit 47**	79,671,110	906,610	3.405	3,845,758	46,009	13,940	3,785,809	4.242
Contract	12000	POX	**Pit 47**	80,172,393	907,239	3.404	3,854,196	46,009	13,940	3,794,247	4.248
Contract	4000	BIOX	**Pit 49**	66,339,467	814,266	3.628	3,264,141	47,356	11,301	3,205,484	4.009
Contract	6000	BIOX	**Pit 48**	67,841,783	816,923	3.620	3,273,378	49,212	11,301	3,212,865	4.007
Contract	8000 (base)	BIOX	**Pit 42**	69,467,843	818,861	3.616	3,292,697	49,212	11,394	3,232,091	4.021
Contract	10000	BIOX	**Pit 41**	70,550,082	828,120	3.592	3,341,520	50,624	11,468	3,279,428	4.035
Contract	12000	BIOX	**Pit 51**	71,166,090	861,004	3.537	3,799,794	63,151	11,646	3,724,997	4.413
Contract	4000	ALBION	**Pit 52**	59,509,737	711,614	3.980	3,140,913	90,705	8,637	3,041,571	4.414
Contract	6000	ALBION	**Pit 51**	60,829,628	716,966	3.967	3,197,491	93,574	8,954	3,094,963	4.460
Contract	8000 (base)	ALBION	**pit 45**	62,339,436	722,377	3.951	3,230,046	94,898	9,530	3,125,618	4.471
Contract	10000	ALBION	**Pit 49**	63,397,191	733,809	3.923	3,347,265	100,639	9,620	3,237,006	4.561
Contract	12000	ALBION	**Pit 49**	63,786,568	734,133	3.922	3,344,627	100,639	10,211	3,233,777	4.556

A16-4. Pit Optimisation – Schedule Results

SUMMARY OF JUGAN (OWNER-OPERATOR) SCHEDULES - FLOTATION

Jugan + Flotation + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,119,559	1,460,186	1,460,186	-	659,372	120,554	-	-	538,818	1.557	1.557	-	0.452
	2	2,495,207	1,460,603	1,180,506	280,097	1,034,604	85,045	3,411	-	946,147	1.509	1.457	1.730	0.708
	3	2,855,098	1,461,263	629,465	831,798	1,393,835	13,332	575	9,914	1,370,014	1.510	1.331	1.645	0.954
	4	3,661,857	1,458,024	175,797	1,282,227	2,203,834	3,390	5,726	23,836	2,170,882	1.636	1.332	1.678	1.512
	5	3,703,321	1,461,304	1	1,461,304	2,242,017	-	4,465	2,882	2,234,671	1.592	2.016	1.592	1.534
	6	7,760,554	1,460,800	-	1,460,800	6,299,754	-	6,874	-	6,292,880	1.439	-	1.439	4.313
	7	2,107,680	1,189,752	-	1,189,752	917,929	-	18,079	-	899,849	1.682	-	1.682	0.772
6000 tpd	1	3,176,289	2,192,379	2,134,426	57,954	983,910	187,157	-	-	796,753	1.586	1.561	2.512	0.449
	2	4,484,686	2,188,923	1,180,032	1,008,891	2,295,763	32,072	4,004	10,032	2,249,656	1.448	1.302	1.618	1.049
	3	6,392,179	2,190,787	131,929	2,058,858	4,201,392	3,093	6,795	23,719	4,167,786	1.664	1.489	1.675	1.918
	4	7,059,649	2,188,485	-	2,188,485	4,871,164	-	11,441	4,105	4,855,619	1.501	-	1.501	2.226
	5	4,274,895	1,295,213	-	1,295,213	2,979,682	-	17,665	-	2,962,016	1.563	-	1.563	2.301
8000 tpd	1	4,455,458	2,921,899	2,513,565	408,335	1,533,559	202,508	3,411	-	1,327,639	1.532	1.518	1.614	0.525
	2	6,852,307	2,919,440	936,981	1,982,459	3,932,867	19,906	6,995	32,171	3,873,795	1.563	1.337	1.669	1.347
	3	9,775,375	2,921,238	2,124	2,919,114	6,854,138	-	11,833	5,690	6,836,615	1.550	0.587	1.551	2.346
	4	5,363,011	1,395,196	-	1,395,196	3,967,815	-	18,510	-	3,949,305	1.554	-	1.554	2.844
10000 tpd	1	5,842,881	3,651,816	3,020,957	630,860	2,191,065	217,015	3,411	-	1,970,639	1.489	1.481	1.530	0.600
	2	11,985,794	3,650,235	431,677	3,218,558	8,335,560	5,508	11,460	36,332	8,282,259	1.643	1.383	1.678	2.284
	3	9,218,843	2,914,554	716	2,913,839	6,304,289	-	32,623	4,110	6,267,556	1.498	1.666	1.498	2.163
12000 tpd	1	8,177,570	4,381,607	3,227,909	1,153,698	3,795,963	217,747	879	13,331	3,564,006	1.528	1.477	1.672	0.866
	2	18,547,524	4,379,068	227,447	4,151,621	14,168,456	5,315	25,088	38,005	14,100,047	1.566	1.344	1.578	3.236
	3	25,412,145	3,858,227	3,882	3,854,345	21,553,918	-	46,975	645	21,506,297	1.215	1.141	1.215	5.587

Jugan + Flotation + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,119,559	1,460,186	1,460,186	-	659,372	120,554	-	-	538,818	1.557	1.557	-	0.452
	2	4,614,766	2,920,790	2,640,693	280,097	1,693,976	205,599	3,411	-	1,484,965	1.533	1.512	1.730	0.580
	3	7,469,864	4,382,053	3,270,158	1,111,895	3,087,812	218,931	3,987	9,914	2,854,979	1.525	1.477	1.666	0.705
	4	11,131,721	5,840,076	3,445,955	2,394,121	5,291,645	222,321	9,713	33,750	5,025,861	1.553	1.470	1.672	0.906
	5	14,835,043	7,301,380	3,445,955	3,855,425	7,533,663	222,321	14,177	36,632	7,260,532	1.561	1.470	1.642	1.032
	6	22,595,597	8,762,180	3,445,955	5,316,225	13,833,417	222,321	21,052	36,632	13,553,412	1.540	1.470	1.586	1.579
	7	24,703,277	9,951,932	3,445,955	6,505,977	14,751,345	222,321	39,131	36,632	14,453,261	1.557	1.470	1.604	1.482
6000 tpd	1	3,176,289	2,192,379	2,134,426	57,954	983,910	187,157	-	-	796,753	1.586	1.561	2.512	0.449
	2	7,660,975	4,381,302	3,314,458	1,066,844	3,279,673	219,228	4,004	10,032	3,046,409	1.517	1.469	1.667	0.749
	3	14,053,154	6,572,089	3,446,387	3,125,703	7,481,065	222,321	10,798	33,750	7,214,195	1.566	1.470	1.672	1.138
	4	21,112,803	8,760,574	3,446,387	5,314,187	12,352,229	222,321	22,239	37,855	12,069,814	1.550	1.470	1.602	1.410
	5	25,387,698	10,055,787	3,446,387	6,609,401	15,331,911	222,321	39,905	37,855	15,031,830	1.552	1.470	1.594	1.525
8000 tpd	1	4,455,458	2,921,899	2,513,565	408,335	1,533,559	202,508	3,411	-	1,327,639	1.532	1.518	1.614	0.525
	2	11,307,765	5,841,340	3,450,546	2,390,794	5,466,426	222,415	10,407	32,171	5,201,434	1.547	1.469	1.660	0.936
	3	21,083,140	8,762,577	3,452,670	5,309,908	12,320,563	222,415	22,239	37,860	12,038,049	1.548	1.469	1.600	1.406
	4	26,446,151	10,157,774	3,452,670	6,705,104	16,288,378	222,415	40,749	37,860	15,987,354	1.549	1.469	1.590	1.604
10000 tpd	1	5,842,881	3,651,816	3,020,957	630,860	2,191,065	217,015	3,411	-	1,970,639	1.489	1.481	1.530	0.600
	2	17,828,676	7,302,051	3,452,634	3,849,418	10,526,625	222,523	14,871	36,332	10,252,898	1.566	1.468	1.654	1.442
	3	27,047,519	10,216,606	3,453,349	6,763,256	16,830,914	222,523	47,494	40,442	16,520,454	1.547	1.469	1.587	1.647
12000 tpd	1	8,177,570	4,381,607	3,227,909	1,153,698	3,795,963	217,747	879	13,331	3,564,006	1.528	1.477	1.672	0.866
	2	26,725,094	8,760,676	3,455,357	5,305,319	17,964,419	223,062	25,967	51,336	17,664,053	1.547	1.468	1.599	2.051
	3	52,137,239	12,618,903	3,459,239	9,159,664	39,518,337	223,062	72,943	51,982	39,170,351	1.446	1.467	1.437	3.132

SUMMARY OF JUGAN (OWNER-OPERATOR) SCHEDULES - BIOX

Jugan + BIOX + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,251,142	1,461,923	1,461,923	-	789,219	444,148	-	-	345,071	1.83	1.83	-	0.540
	2	3,448,898	1,458,359	798,452	659,907	1,990,539	153,254	51,637	12,133	1,773,514	1.69	1.59	1.82	1.365
	3	2,991,534	1,462,054	373,381	1,088,673	1,529,481	290,204	72,469	-	1,166,808	1.72	1.60	1.76	1.046
	4	5,769,620	1,459,372	33,501	1,425,871	4,310,248	4,754	78,818	11,695	4,214,980	1.77	1.47	1.78	2.954
	5	5,717,640	1,458,626	-	1,458,626	4,259,014	-	172,827	19	4,086,168	1.66	-	1.66	2.920
	6	1,138,040	670,367	-	670,367	467,672	-	101,594	-	366,078	2.02	-	2.02	0.698
6000 tpd	1	3,708,113	2,191,472	2,009,282	182,190	1,516,641	564,426	12,491	-	939,724	1.727	1.748	1.500	0.692
	2	5,218,759	2,190,725	622,017	1,568,708	3,028,035	323,980	122,322	9,914	2,571,819	1.805	1.638	1.871	1.382
	3	7,110,548	2,188,233	37,768	2,150,465	4,922,315	3,954	145,483	17,008	4,755,870	1.688	1.556	1.690	2.249
	4	5,477,859	1,422,765	-	1,422,765	4,055,094	-	212,778	-	3,842,316	1.846	-	1.846	2.850
8000 tpd	1	5,676,336	2,921,198	2,424,075	497,123	2,755,139	712,857	37,324	-	2,004,958	1.699	1.703	1.682	0.943
	2	7,993,105	2,921,497	250,143	2,671,354	5,071,609	195,368	220,817	26,922	4,628,502	1.790	1.867	1.783	1.736
	3	9,445,413	2,308,052	-	2,308,052	7,137,361	-	298,888	-	6,838,473	1.783	-	1.783	3.092
10000 tpd	1	8,021,558	3,650,584	2,637,753	1,012,831	4,370,974	904,952	64,634	9,914	3,391,474	1.761	1.720	1.866	1.197
	2	11,107,008	3,650,868	37,768	3,613,100	7,456,141	3,954	357,855	17,008	7,077,324	1.697	1.556	1.698	2.042
	3	4,030,786	855,470	-	855,470	3,175,316	-	134,671	-	3,040,646	1.979	-	1.979	3.712
12000 tpd	1	9,776,099	4,381,303	2,675,520	1,705,782	5,394,797	908,906	160,961	14,999	4,309,931	1.741	1.718	1.776	1.231
	2	13,422,243	3,781,057	1	3,781,057	9,641,186	-	396,211	11,923	9,233,052	1.771	2.016	1.771	2.550

Jugan + BIOX + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,251,142	1,461,923	1,461,923	-	789,219	444,148	-	-	345,071	1.829	1.829	-	0.540
	2	5,700,040	2,920,283	2,260,376	659,907	2,779,757	597,402	51,637	12,133	2,118,585	1.761	1.743	1.822	0.952
	3	8,691,574	4,382,336	2,633,757	1,748,580	4,309,238	887,606	124,106	12,133	3,285,393	1.748	1.723	1.785	0.983
	4	14,461,194	5,841,709	2,667,258	3,174,451	8,619,486	892,360	202,924	23,828	7,500,374	1.754	1.720	1.783	1.476
	5	20,178,834	7,300,335	2,667,258	4,633,077	12,878,500	892,360	375,751	23,847	11,586,542	1.735	1.720	1.744	1.764
	6	21,316,874	7,970,702	2,667,258	5,303,444	13,346,172	892,360	477,345	23,847	11,952,620	1.759	1.720	1.779	1.674
6000 tpd	1	3,708,113	2,191,472	2,009,282	182,190	1,516,641	564,426	12,491	-	939,724	1.727	1.748	1.500	0.692
	2	8,926,872	4,382,197	2,631,299	1,750,898	4,544,675	888,406	134,812	9,914	3,511,543	1.766	1.722	1.832	1.037
	3	16,037,420	6,570,430	2,669,067	3,901,363	9,466,990	892,360	280,295	26,922	8,267,413	1.740	1.720	1.754	1.441
	4	21,515,279	7,993,195	2,669,067	5,324,128	13,522,084	892,360	493,073	26,922	12,109,729	1.759	1.720	1.779	1.692
8000 tpd	1	5,676,336	2,921,198	2,424,075	497,123	2,755,139	712,857	37,324	-	2,004,958	1.699	1.703	1.682	0.943
	2	13,669,442	5,842,694	2,674,218	3,168,477	7,826,747	908,225	258,141	26,922	6,633,460	1.745	1.718	1.767	1.340
	3	23,114,854	8,150,746	2,674,218	5,476,528	14,964,108	908,225	557,029	26,922	13,471,932	1.755	1.718	1.774	1.836
10000 tpd	1	8,021,558	3,650,584	2,637,753	1,012,831	4,370,974	904,952	64,634	9,914	3,391,474	1.761	1.720	1.866	1.197
	2	19,128,566	7,301,452	2,675,521	4,625,931	11,827,115	908,906	422,490	26,922	10,468,797	1.729	1.718	1.735	1.620
	3	23,159,352	8,156,922	2,675,521	5,481,401	15,002,431	908,906	557,160	26,922	13,509,443	1.755	1.718	1.773	1.839
12000 tpd	1	9,776,099	4,381,303	2,675,520	1,705,782	5,394,797	908,906	160,961	14,999	4,309,931	1.741	1.718	1.776	1.231
	2	23,198,342	8,162,360	2,675,521	5,486,839	15,035,983	908,906	557,172	26,922	13,542,982	1.755	1.718	1.773	1.842

SUMMARY OF JUGAN (OWNER-OPERATOR) SCHEDULES - POX

Jugan + POX + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,156,234	1,460,973	1,460,973	-	695,262	323,398	-	-	371,864	1.721	1.721	-	0.476
	2	3,141,111	1,460,430	1,020,419	440,011	1,680,680	165,692	26,245	-	1,488,743	1.609	1.580	1.676	1.151
	3	3,470,073	1,461,131	380,917	1,080,214	2,008,943	195,689	61,425	9,914	1,741,914	1.750	1.522	1.830	1.375
	4	3,423,089	1,458,661	39,965	1,418,696	1,964,428	1,708	64,148	16,989	1,881,584	1.703	1.507	1.709	1.347
	5	4,584,089	1,460,217	-	1,460,217	3,123,872	-	98,872	19	3,024,981	1.619	-	1.619	2.139
	6	5,306,236	1,174,945	-	1,174,945	4,131,291	-	128,916	-	4,002,375	1.873	-	1.873	3.516
6000 tpd	1	3,291,890	2,191,309	2,030,933	160,376	1,100,581	422,973	10,043	-	667,564	1.690	1.702	1.546	0.502
	2	5,517,533	2,190,863	834,175	1,356,688	3,326,670	265,991	78,454	9,914	2,972,310	1.696	1.503	1.814	1.518
	3	6,906,884	2,190,075	39,965	2,150,110	4,716,809	1,708	86,818	17,008	4,611,275	1.686	1.507	1.689	2.154
	4	6,553,387	1,945,571	-	1,945,571	4,607,816	-	214,093	-	4,393,723	1.750	-	1.750	2.368
8000 tpd	1	5,316,213	2,920,418	2,600,788	319,630	2,395,796	536,533	23,103	-	1,836,160	1.627	1.633	1.582	0.820
	2	10,004,870	2,921,728	311,960	2,609,768	7,083,142	169,467	156,831	26,903	6,729,941	1.748	1.694	1.755	2.424
	3	8,513,734	2,858,865	-	2,858,865	5,654,870	-	259,529	19	5,395,322	1.717	-	1.717	1.978
10000 tpd	1	7,659,398	3,651,331	2,715,014	936,316	4,008,067	668,504	43,184	9,914	3,286,465	1.697	1.665	1.792	1.098
	2	9,644,551	3,650,652	197,734	3,452,918	5,993,899	37,496	230,576	17,008	5,708,818	1.655	1.295	1.676	1.642
	3	6,676,628	1,423,440	-	1,423,440	5,253,188	-	167,007	-	5,086,181	1.796	-	1.796	3.691
12000 tpd	1	9,421,175	4,381,446	2,912,388	1,469,058	5,039,729	704,582	85,666	9,914	4,239,567	1.680	1.640	1.760	1.150
	2	14,601,097	4,351,146	706	4,350,440	10,249,952	1,418	356,308	17,008	9,875,218	1.711	0.711	1.711	2.356

Jugan + POX + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,156,234	1,460,973	1,460,973	-	695,262	323,398	-	-	371,864	1.721	1.721	-	0.476
	2	5,297,345	2,921,403	2,481,392	440,011	2,375,942	489,090	26,245	-	1,860,607	1.665	1.663	1.676	0.813
	3	8,767,418	4,382,533	2,862,309	1,520,225	4,384,885	684,780	87,669	9,914	3,602,522	1.693	1.644	1.786	1.001
	4	12,190,507	5,841,194	2,902,274	2,938,921	6,349,313	686,488	151,817	26,903	5,484,105	1.696	1.642	1.749	1.087
	5	16,774,596	7,301,411	2,902,274	4,399,137	9,473,185	686,488	250,689	26,922	8,509,086	1.680	1.642	1.706	1.297
	6	22,080,832	8,476,356	2,902,274	5,574,082	13,604,476	686,488	379,605	26,922	12,511,461	1.707	1.642	1.741	1.605
6000 tpd	1	3,291,890	2,191,309	2,030,933	160,376	1,100,581	422,973	10,043	-	667,564	1.690	1.702	1.546	0.502
	2	8,809,422	4,382,172	2,865,108	1,517,064	4,427,250	688,964	88,497	9,914	3,639,874	1.693	1.644	1.786	1.010
	3	15,716,306	6,572,247	2,905,073	3,667,174	9,144,059	690,672	175,316	26,922	8,251,149	1.691	1.642	1.729	1.391
	4	22,269,693	8,517,818	2,905,073	5,612,745	13,751,875	690,672	389,409	26,922	12,644,872	1.704	1.642	1.736	1.615
8000 tpd	1	5,316,213	2,920,418	2,600,788	319,630	2,395,796	536,533	23,103	-	1,836,160	1.627	1.633	1.582	0.820
	2	15,321,084	5,842,146	2,912,748	2,929,398	9,478,938	706,000	179,934	26,903	8,566,101	1.688	1.640	1.736	1.623
	3	23,834,818	8,701,011	2,912,748	5,788,262	15,133,808	706,000	439,463	26,922	13,961,423	1.698	1.640	1.727	1.739
10000 tpd	1	7,659,398	3,651,331	2,715,014	936,316	4,008,067	668,504	43,184	9,914	3,286,465	1.697	1.665	1.792	1.098
	2	17,303,948	7,301,983	2,912,748	4,389,235	10,001,966	706,000	273,761	26,922	8,995,283	1.676	1.640	1.701	1.370
	3	23,980,577	8,725,423	2,912,748	5,812,675	15,255,154	706,000	440,768	26,922	14,081,464	1.696	1.640	1.724	1.748
12000 tpd	1	9,421,175	4,381,446	2,912,388	1,469,058	5,039,729	704,582	85,666	9,914	4,239,567	1.680	1.640	1.760	1.150
	2	24,022,272	8,732,592	2,913,094	5,819,498	15,289,681	706,000	441,974	26,922	14,114,785	1.695	1.640	1.723	1.751

SUMMARY OF JUGAN (OWNER-OPERATOR) SCHEDULES - ALBION

Jugan + Albion + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,675,634	1,460,699	1,456,808	3,891	1,214,936	765,152	-	-	449,784	1.965	1.962	3.066	0.832
	2	4,312,179	1,459,302	682,551	776,751	2,852,877	594,519	149,874	-	2,108,484	1.918	1.792	2.029	1.955
	3	3,902,551	1,462,212	12,691	1,449,520	2,440,339	2,485	291,817	12,464	2,133,573	1.838	1.449	1.842	1.669
	4	4,771,860	1,459,418	-	1,459,418	3,312,442	-	312,650	-	2,999,792	1.799	-	1.799	2.270
	5	2,827,902	664,841	-	664,841	2,163,061	-	102,101	-	2,060,960	2.123	-	2.123	3.254
6000 tpd	1	4,899,514	2,191,777	1,980,236	211,541	2,707,737	1,155,693	35,496	-	1,516,549	1.889	1.890	1.877	1.235
	2	6,721,824	2,190,135	172,752	2,017,383	4,531,689	206,777	387,067	12,464	3,925,381	1.954	2.077	1.943	2.069
	3	7,194,643	2,168,257	-	2,168,257	5,026,385	-	445,962	-	4,580,424	1.868	-	1.868	2.318
8000 tpd	1	7,099,330	2,921,449	2,129,525	791,924	4,177,881	1,359,696	156,069	-	2,662,116	1.941	1.909	2.029	1.430
	2	9,026,380	2,920,194	23,764	2,896,431	6,106,186	2,774	606,513	23,847	5,473,050	1.811	1.577	1.813	2.091
	3	4,134,147	844,485	-	844,485	3,289,662	-	160,990	-	3,128,672	2.090	-	2.090	3.896
10000 tpd	1	8,559,386	3,651,224	2,130,291	1,520,934	4,908,162	1,362,305	307,371	12,133	3,226,352	1.926	1.908	1.952	1.344
	2	11,728,370	3,045,024	26,019	3,019,006	8,683,346	2,774	620,758	11,714	8,048,100	1.874	1.573	1.876	2.852
12000 tpd	1	12,319,452	4,380,612	2,156,309	2,224,303	7,938,840	1,365,080	411,219	19,409	6,143,133	1.920	1.904	1.935	1.812
	2	7,976,091	2,316,867	-	2,316,867	5,659,224	-	516,926	4,439	5,137,860	1.869	-	1.869	2.443

Jugan + Albion + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,675,634	1,460,699	1,456,808	3,891	1,214,936	765,152	-	-	449,784	1.965	1.962	3.066	0.832
	2	6,987,813	2,920,001	2,139,359	780,642	4,067,812	1,359,670	149,874	-	2,558,268	1.942	1.908	2.034	1.955
	3	10,890,364	4,382,213	2,152,051	2,230,162	6,508,151	1,362,155	441,691	12,464	4,691,841	1.907	1.905	1.909	1.669
	4	15,662,224	5,841,631	2,152,051	3,689,580	9,820,593	1,362,155	754,341	12,464	7,691,632	1.880	1.905	1.866	2.270
	5	18,490,125	6,506,472	2,152,051	4,354,421	11,983,653	1,362,155	856,442	12,464	9,752,592	1.905	1.905	1.905	3.254
6000 tpd	1	4,899,514	2,191,777	1,980,236	211,541	2,707,737	1,155,693	35,496	-	1,516,549	1.889	1.890	1.877	1.235
	2	11,621,338	4,381,912	2,152,988	2,228,924	7,239,426	1,362,470	422,563	12,464	5,441,930	1.921	1.905	1.937	1.652
	3	18,815,981	6,550,169	2,152,988	4,397,182	12,265,812	1,362,470	868,524	12,464	10,022,353	1.904	1.905	1.903	1.873
8000 tpd	1	7,099,330	2,921,449	2,129,525	791,924	4,177,881	1,359,696	156,069	-	2,662,116	1.941	1.909	2.029	1.430
	2	16,125,710	5,841,644	2,153,289	3,688,355	10,284,066	1,362,470	762,583	23,847	8,135,166	1.876	1.905	1.859	1.761
	3	20,259,857	6,686,129	2,153,289	4,532,840	13,573,728	1,362,470	923,573	23,847	11,263,838	1.903	1.905	1.902	2.030
10000 tpd	1	8,559,386	3,651,224	2,130,291	1,520,934	4,908,162	1,362,305	307,371	12,133	3,226,352	1.926	1.908	1.952	1.344
	2	20,287,756	6,696,248	2,156,309	4,539,939	13,591,508	1,365,080	928,129	23,847	11,274,452	1.902	1.904	1.901	2.030
12000 tpd	1	12,319,452	4,380,612	2,156,309	2,224,303	7,938,840	1,365,080	411,219	19,409	6,143,133	1.920	1.904	1.935	1.812
	2	20,295,543	6,697,479	2,156,309	4,541,170	13,598,064	1,365,080	928,144	23,847	11,280,992	1.902	1.904	1.901	2.030

SUMMARY OF JUGAN (CONTRACT-MINING) SCHEDULES - FLOTATION

Jugan + Flotation + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,992,212	1,461,713	1,461,713	-	530,499	122,354	-	-	408,145	1.558	1.558	-	0.363
	2	2,475,459	1,460,862	1,272,738	188,123	1,014,598	83,500	2,465	-	928,632	1.492	1.462	1.689	0.695
	3	3,026,723	1,459,588	432,887	1,026,701	1,567,135	12,762	575	9,914	1,543,883	1.570	1.271	1.696	1.074
	4	4,187,639	1,459,075	275,030	1,184,045	2,728,565	3,566	3,683	5,085	2,716,231	1.591	1.360	1.645	1.870
	5	3,937,253	1,460,156	1	1,460,156	2,477,097	-	6,745	11,904	2,458,448	1.603	2.016	1.603	1.697
	6	5,164,331	1,461,101	-	1,461,101	3,703,229	-	8,077	19	3,695,134	1.521	-	1.521	2.535
	7	3,705,630	1,151,125	-	1,151,125	2,554,504	-	17,586	-	2,536,919	1.592	-	1.592	2.219
6000 tpd	1	3,098,347	2,190,020	2,139,513	50,507	908,327	192,535	-	-	715,791	1.593	1.581	2.101	0.415
	2	4,832,699	2,191,136	1,172,078	1,019,058	2,641,563	29,598	3,041	9,914	2,599,010	1.490	1.310	1.696	1.206
	3	4,925,090	2,190,459	132,795	2,057,664	2,734,631	49	9,535	16,989	2,708,058	1.589	1.099	1.621	1.248
	4	9,906,803	2,188,872	-	2,188,872	7,717,931	-	8,969	19	7,708,944	1.534	-	1.534	3.526
	5	1,738,313	1,157,119	-	1,157,119	581,195	-	17,586	-	563,609	1.622	-	1.622	0.502
8000 tpd	1	4,352,256	2,920,168	2,648,616	271,552	1,432,089	209,977	2,465	-	1,219,646	1.532	1.513	1.717	0.490
	2	6,610,469	2,920,096	793,841	2,126,255	3,690,374	12,205	7,247	25,706	3,645,215	1.576	1.329	1.669	1.264
	3	9,521,135	2,920,551	2,124	2,918,427	6,600,584	-	11,832	1,216	6,587,535	1.559	0.587	1.560	2.260
	4	4,042,340	1,159,682	-	1,159,682	2,882,658	-	17,586	-	2,865,073	1.586	-	1.586	2.486
10000 tpd	1	5,753,001	3,651,911	3,118,592	533,319	2,101,091	217,375	3,428	-	1,880,287	1.492	1.486	1.530	0.575
	2	11,402,516	3,649,386	326,512	3,322,873	7,753,131	4,946	10,734	33,750	7,703,700	1.634	1.320	1.664	2.125
	3	8,903,947	2,791,669	-	2,791,669	6,112,278	-	26,587	4,105	6,081,586	1.530	-	1.530	2.190
12000 tpd	1	7,647,834	4,382,336	3,262,906	1,119,430	3,265,499	218,931	4,004	10,749	3,031,815	1.525	1.479	1.659	0.745
	2	13,657,972	4,378,917	183,050	4,195,868	9,279,055	3,390	18,746	27,106	9,229,813	1.575	1.312	1.586	2.119
	3	4,758,692	1,332,753	-	1,332,753	3,425,939	-	18,000	-	3,407,939	1.581	-	1.581	2.571

Jugan + Flotation + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,992,212	1,461,713	1,461,713	-	530,499	122,354	-	-	408,145	1.558	1.558	-	0.363
	2	4,467,671	2,922,574	2,734,451	188,123	1,545,097	205,854	2,465	-	1,336,777	1.525	1.513	1.689	0.529
	3	7,494,394	4,382,163	3,167,338	1,214,824	3,112,231	218,616	3,041	9,914	2,880,660	1.540	1.480	1.695	0.710
	4	11,682,033	5,841,238	3,442,368	2,398,870	5,840,796	222,182	6,723	14,999	5,596,891	1.553	1.471	1.670	1.000
	5	15,619,287	7,301,394	3,442,369	3,859,026	8,317,893	222,182	13,468	26,903	8,055,340	1.563	1.471	1.645	1.139
	6	20,783,617	8,762,496	3,442,369	5,320,127	12,021,122	222,182	21,545	26,922	11,750,473	1.556	1.471	1.611	1.372
	7	24,489,247	9,913,621	3,442,369	6,471,252	14,575,626	222,182	39,130	26,922	14,287,392	1.560	1.471	1.607	1.470
6000 tpd	1	3,098,347	2,190,020	2,139,513	50,507	908,327	192,535	-	-	715,791	1.593	1.581	2.101	0.415
	2	7,931,045	4,381,156	3,311,591	1,069,565	3,549,889	222,133	3,041	9,914	3,314,801	1.541	1.485	1.715	0.810
	3	12,856,135	6,571,615	3,444,386	3,127,229	6,284,520	222,182	12,576	26,903	6,022,859	1.557	1.470	1.653	0.956
	4	22,762,938	8,760,486	3,444,386	5,316,101	14,002,452	222,182	21,545	26,922	13,731,803	1.551	1.470	1.604	1.598
	5	24,501,251	9,917,605	3,444,386	6,473,219	14,583,647	222,182	39,130	26,922	14,295,412	1.560	1.470	1.607	1.471
8000 tpd	1	4,352,256	2,920,168	2,648,616	271,552	1,432,089	209,977	2,465	-	1,219,646	1.532	1.513	1.717	0.490
	2	10,962,725	5,840,264	3,442,457	2,397,807	5,122,462	222,182	9,713	25,706	4,864,862	1.554	1.471	1.674	0.877
	3	20,483,860	8,760,815	3,444,581	5,316,234	11,723,046	222,182	21,545	26,922	11,452,397	1.556	1.470	1.612	1.338
	4	24,526,201	9,920,497	3,444,581	6,475,917	14,605,704	222,182	39,130	26,922	14,317,470	1.559	1.470	1.607	1.472
10000 tpd	1	5,753,001	3,651,911	3,118,592	533,319	2,101,091	217,375	3,428	-	1,880,287	1.492	1.486	1.530	0.575
	2	17,155,517	7,301,296	3,445,104	3,856,193	9,854,221	222,321	14,163	33,750	9,583,987	1.563	1.470	1.646	1.350
	3	26,059,464	10,092,965	3,445,104	6,647,862	15,966,499	222,321	40,749	37,855	15,665,574	1.554	1.470	1.597	1.582
12000 tpd	1	7,647,834	4,382,336	3,262,906	1,119,430	3,265,499	218,931	4,004	10,749	3,031,815	1.525	1.479	1.659	0.745
	2	21,305,806	8,761,253	3,445,955	5,315,298	12,544,554	222,321	22,750	37,855	12,261,628	1.550	1.470	1.601	1.432
	3	26,064,498	10,094,007	3,445,955	6,648,051	15,970,492	222,321	40,749	37,855	15,669,567	1.554	1.470	1.597	1.582

SUMMARY OF JUGAN (CONTRACT-MINING) SCHEDULES - BIOX

Jugan + BIOX + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,249,071	1,460,283	1,460,283	-	788,788	465,118	-	-	323,670	1.844	1.844	-	0.540
	2	3,194,467	1,460,377	805,037	655,340	1,734,089	157,754	49,018	8,353	1,518,965	1.710	1.586	1.864	1.187
	3	2,815,256	1,459,659	368,071	1,091,588	1,355,597	255,613	67,102	-	1,032,882	1.698	1.567	1.742	0.929
	4	5,611,498	1,460,016	21,190	1,438,826	4,151,482	4,754	90,274	4,111	4,052,342	1.759	1.328	1.765	2.843
	5	4,811,742	1,460,728	-	1,460,728	3,351,014	-	168,886	-	3,182,128	1.711	-	1.711	2.294
	6	568,048	426,434	-	426,434	141,613	-	46,769	-	94,844	2.082	-	2.082	0.332
6000 tpd	1	3,747,570	2,190,082	2,048,998	141,084	1,557,487	580,231	10,652	-	966,604	1.751	1.769	1.487	0.711
	2	4,718,556	2,191,595	591,478	1,600,116	2,526,962	299,185	124,690	8,353	2,094,734	1.779	1.569	1.856	1.153
	3	7,001,734	2,190,609	21,183	2,169,426	4,811,125	4,593	139,435	4,111	4,662,986	1.684	1.328	1.687	2.196
	4	3,828,375	1,168,347	-	1,168,347	2,660,027	-	149,224	-	2,510,803	1.897	-	1.897	2.277
8000 tpd	1	5,493,567	2,922,595	2,430,011	492,585	2,570,972	727,853	30,836	-	1,812,282	1.706	1.709	1.689	0.880
	2	7,772,491	2,917,636	234,494	2,683,141	4,854,856	163,521	197,072	23,847	4,470,416	1.798	1.841	1.794	1.664
	3	7,750,754	2,072,604	-	2,072,604	5,678,150	-	231,564	-	5,446,586	1.795	-	1.795	2.740
10000 tpd	1	7,781,156	3,650,734	2,635,481	1,015,253	4,130,422	888,406	55,546	12,133	3,174,337	1.761	1.723	1.862	1.131
	2	12,303,162	3,650,691	30,945	3,619,746	8,652,471	3,954	332,215	11,714	8,304,588	1.714	1.490	1.716	2.370
	3	1,070,288	641,745	-	641,745	428,544	-	80,482	-	348,061	2.025	-	2.025	0.668
12000 tpd	1	7,758,916	4,380,034	2,590,763	1,789,271	3,378,882	887,605	128,973	-	2,362,304	1.751	1.723	1.792	0.771
	2	13,403,459	3,564,125	76,495	3,487,631	9,839,334	4,755	339,270	23,847	9,471,461	1.773	1.611	1.776	2.761

Jugan + BIOX + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,249,071	1,460,283	1,460,283	-	788,788	465,118	-	-	323,670	1.844	1.844	-	0.540
	2	5,443,538	2,920,660	2,265,320	655,340	2,522,878	622,872	49,018	8,353	1,842,635	1.777	1.752	1.864	0.864
	3	8,258,794	4,380,319	2,633,391	1,746,929	3,878,474	878,484	116,120	8,353	2,875,517	1.751	1.726	1.787	0.885
	4	13,870,292	5,840,335	2,654,581	3,185,755	8,029,956	883,239	206,394	12,464	6,927,860	1.753	1.723	1.777	1.375
	5	18,682,034	7,301,064	2,654,581	4,646,483	11,380,970	883,239	375,280	12,464	10,109,987	1.744	1.723	1.756	1.559
	6	19,250,081	7,727,498	2,654,581	5,072,917	11,522,583	883,239	422,049	12,464	10,204,832	1.763	1.723	1.784	1.491
6000 tpd	1	3,747,570	2,190,082	2,048,998	141,084	1,557,487	580,231	10,652	-	966,604	1.751	1.769	1.487	0.711
	2	8,466,126	4,381,677	2,640,477	1,741,201	4,084,449	879,417	135,342	8,353	3,061,338	1.765	1.724	1.826	0.932
	3	15,467,860	6,572,286	2,661,659	3,910,627	8,895,574	884,010	274,777	12,464	7,724,323	1.738	1.721	1.749	1.354
	4	19,296,234	7,740,633	2,661,659	5,078,974	11,555,601	884,010	424,001	12,464	10,235,127	1.762	1.721	1.783	1.493
8000 tpd	1	5,493,567	2,922,595	2,430,011	492,585	2,570,972	727,853	30,836	-	1,812,282	1.706	1.709	1.689	0.880
	2	13,266,059	5,840,231	2,664,505	3,175,726	7,425,828	891,374	227,908	23,847	6,282,698	1.752	1.721	1.778	1.272
	3	21,016,812	7,912,835	2,664,505	5,248,330	13,103,978	891,374	459,472	23,847	11,729,284	1.763	1.721	1.785	1.656
10000 tpd	1	7,781,156	3,650,734	2,635,481	1,015,253	4,130,422	888,406	55,546	12,133	3,174,337	1.761	1.723	1.862	1.131
	2	20,084,318	7,301,425	2,666,426	4,634,999	12,782,893	892,360	387,761	23,847	11,478,925	1.738	1.720	1.748	1.751
	3	21,154,606	7,943,169	2,666,426	5,276,744	13,211,437	892,360	468,243	23,847	11,826,986	1.761	1.720	1.782	1.663
12000 tpd	1	7,758,916	4,380,034	2,590,763	1,789,271	3,378,882	887,605	128,973	-	2,362,304	1.751	1.723	1.792	0.771
	2	21,162,375	7,944,159	2,667,258	5,276,902	13,218,216	892,360	468,243	23,847	11,833,765	1.761	1.720	1.782	1.664

SUMMARY OF JUGAN (CONTRACT-MINING) SCHEDULES - POX

Jugan + POX + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,178,540	1,462,498	1,462,498	-	716,042	317,869	-	-	398,173	1.722	1.722	-	0.490
	2	3,287,342	1,459,616	1,074,663	384,953	1,827,726	256,005	27,050	-	1,544,671	1.592	1.574	1.645	1.252
	3	3,094,619	1,459,526	331,517	1,128,009	1,635,093	95,946	55,068	8,353	1,475,726	1.813	1.562	1.887	1.120
	4	3,387,017	1,459,221	20,801	1,438,419	1,927,796	1,569	55,103	4,111	1,867,012	1.685	1.265	1.691	1.321
	5	4,241,091	1,460,396	-	1,460,396	2,780,695	-	105,751	-	2,674,944	1.614	-	1.614	1.904
	6	3,734,376	888,318	-	888,318	2,846,057	-	71,291	-	2,774,766	1.961	-	1.961	3.204
6000 tpd	1	3,241,133	2,190,972	2,027,163	163,810	1,050,160	421,900	9,172	-	619,088	1.692	1.705	1.532	0.479
	2	5,359,006	2,190,829	831,088	1,359,741	3,168,177	248,748	76,005	12,133	2,831,291	1.702	1.506	1.821	1.446
	3	6,811,063	2,188,203	33,142	2,155,061	4,622,860	1,708	88,930	11,714	4,520,507	1.686	1.436	1.690	2.113
	4	6,154,870	1,826,936	-	1,826,936	4,327,934	-	186,724	-	4,141,210	1.773	-	1.773	2.369
8000 tpd	1	5,190,587	2,920,620	2,471,773	448,847	2,269,967	474,958	24,983	-	1,770,026	1.669	1.666	1.686	0.777
	2	7,163,921	2,920,591	420,882	2,499,709	4,243,330	197,398	132,446	26,922	3,886,563	1.712	1.521	1.744	1.453
	3	9,465,719	2,591,062	-	2,591,062	6,874,657	-	220,667	-	6,653,990	1.753	-	1.753	2.653
10000 tpd	1	6,781,840	3,650,430	2,615,749	1,034,682	3,131,410	533,987	47,647	9,914	2,539,862	1.668	1.634	1.753	0.858
	2	12,418,319	3,650,810	283,734	3,367,076	8,767,509	148,464	162,916	17,008	8,439,121	1.698	1.728	1.696	2.402
	3	2,863,613	1,171,297	-	1,171,297	1,692,316	-	169,031	-	1,523,286	1.860	-	1.860	1.445
12000 tpd	1	9,211,667	4,380,477	2,901,482	1,478,995	4,831,190	682,452	60,997	9,914	4,077,827	1.696	1.642	1.801	1.103
	2	12,864,027	4,095,088	1	4,095,088	8,768,939	-	318,609	17,008	8,433,323	1.719	2.016	1.719	2.141

Jugan + POX + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,178,540	1,462,498	1,462,498	-	716,042	317,869	-	-	398,173	1.722	1.722	-	0.490
	2	5,465,882	2,922,114	2,537,161	384,953	2,543,768	573,875	27,050	-	1,942,844	1.657	1.659	1.645	0.871
	3	8,560,502	4,381,640	2,868,679	1,512,962	4,178,862	669,821	82,118	8,353	3,418,570	1.709	1.648	1.825	0.954
	4	11,947,518	5,840,861	2,889,480	2,951,381	6,106,658	671,390	137,222	12,464	5,285,582	1.703	1.645	1.760	1.046
	5	16,188,609	7,301,257	2,889,480	4,411,777	8,887,352	671,390	242,972	12,464	7,960,526	1.685	1.645	1.712	1.217
	6	19,922,985	8,189,575	2,889,480	5,300,096	11,733,409	671,390	314,264	12,464	10,735,292	1.715	1.645	1.753	1.433
6000 tpd	1	3,241,133	2,190,972	2,027,163	163,810	1,050,160	421,900	9,172	-	619,088	1.692	1.705	1.532	0.479
	2	8,600,139	4,381,801	2,858,251	1,523,550	4,218,338	670,648	85,178	12,133	3,450,379	1.697	1.647	1.790	0.963
	3	15,411,202	6,570,004	2,891,393	3,678,612	8,841,197	672,356	174,107	23,847	7,970,886	1.693	1.645	1.732	1.346
	4	21,566,071	8,396,940	2,891,393	5,505,547	13,169,131	672,356	360,832	23,847	12,112,096	1.711	1.645	1.745	1.568
8000 tpd	1	5,190,587	2,920,620	2,471,773	448,847	2,269,967	474,958	24,983	-	1,770,026	1.669	1.666	1.686	0.777
	2	12,354,508	5,841,211	2,892,655	2,948,556	6,513,297	672,356	157,429	26,922	5,656,589	1.690	1.645	1.735	1.115
	3	21,820,227	8,432,274	2,892,655	5,539,619	13,387,954	672,356	378,097	26,922	12,310,579	1.709	1.645	1.743	1.588
10000 tpd	1	6,781,840	3,650,430	2,615,749	1,034,682	3,131,410	533,987	47,647	9,914	2,539,862	1.668	1.634	1.753	0.858
	2	19,200,159	7,301,240	2,899,483	4,401,757	11,898,918	682,451	210,562	26,922	10,978,983	1.683	1.643	1.709	1.630
	3	22,063,772	8,472,537	2,899,483	5,573,054	13,591,235	682,451	379,593	26,922	12,502,268	1.707	1.643	1.741	1.604
12000 tpd	1	9,211,667	4,380,477	2,901,482	1,478,995	4,831,190	682,452	60,997	9,914	4,077,827	1.696	1.642	1.801	1.103
	2	22,075,693	8,475,565	2,901,483	5,574,082	13,600,129	682,452	379,605	26,922	12,511,150	1.707	1.642	1.741	1.605

SUMMARY OF JUGAN (CONTRACT-MINING) SCHEDULES - ALBION

Jugan + Albion + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,553,331	1,461,210	1,461,210	-	1,092,121	720,404	-	-	371,718	1.980	1.980	-	0.747
	2	3,512,471	1,458,881	522,436	936,446	2,053,590	439,102	147,815	8,353	1,458,320	1.821	1.639	1.923	1.408
	3	4,734,871	1,460,920	153,516	1,307,404	3,273,950	187,322	229,240	-	2,857,387	1.986	2.152	1.967	2.241
	4	4,877,434	1,459,680	-	1,459,680	3,417,754	-	310,593	4,111	3,103,050	1.799	-	1.799	2.341
	5	447,894	337,969	-	337,969	109,925	-	48,628	-	61,297	2.123	-	2.123	0.325
6000 tpd	1	4,274,784	2,190,797	1,778,956	411,841	2,083,987	1,012,364	76,116	-	995,507	1.932	1.935	1.921	0.951
	2	6,885,038	2,190,429	362,011	1,828,418	4,694,610	336,689	352,424	12,464	3,993,033	1.896	1.776	1.919	2.143
	3	6,031,905	1,950,723	-	1,950,723	4,081,182	-	360,555	-	3,720,627	1.893	-	1.893	2.092
8000 tpd	1	6,907,578	2,921,256	2,128,945	792,312	3,986,321	1,346,127	178,333	-	2,461,861	1.917	1.911	1.931	1.365
	2	10,393,807	2,919,110	16,070	2,903,040	7,474,696	7,201	568,708	12,464	6,886,324	1.855	1.402	1.858	2.561
	3	876,426	590,186	-	590,186	286,240	-	77,011	-	209,229	2.153	-	2.153	0.485
10000 tpd	1	8,408,144	3,652,192	2,134,773	1,517,419	4,755,953	1,354,955	296,463	8,353	3,096,182	1.930	1.909	1.960	1.302
	2	10,080,360	2,853,073	16,070	2,837,002	7,227,287	7,201	559,979	4,111	6,655,996	1.873	1.402	1.876	2.533
12000 tpd	1	11,883,268	4,381,227	2,150,843	2,230,384	7,502,041	1,362,155	437,135	12,464	5,690,286	1.906	1.906	1.906	1.712
	2	6,605,236	2,124,037	-	2,124,037	4,481,199	-	419,307	-	4,061,892	1.904	-	1.904	2.110

Jugan + Albion + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,553,331	1,461,210	1,461,210	-	1,092,121	720,404	-	-	371,718	1.980	1.980	-	0.747
	2	6,065,802	2,920,091	1,983,645	936,446	3,145,711	1,159,505	147,815	8,353	1,830,038	1.901	1.890	1.923	1.077
	3	10,800,673	4,381,011	2,137,162	2,243,850	6,419,662	1,346,828	377,056	8,353	4,687,425	1.929	1.909	1.948	1.465
	4	15,678,107	5,840,692	2,137,162	3,703,530	9,837,415	1,346,828	687,649	12,464	7,790,475	1.897	1.909	1.889	1.684
	5	16,126,001	6,178,660	2,137,162	4,041,498	9,947,341	1,346,828	736,277	12,464	7,851,772	1.909	1.909	1.909	1.610
6000 tpd	1	4,274,784	2,190,797	1,778,956	411,841	2,083,987	1,012,364	76,116	-	995,507	1.932	1.935	1.921	0.951
	2	11,159,823	4,381,226	2,140,966	2,240,259	6,778,597	1,349,053	428,540	12,464	4,988,540	1.914	1.908	1.920	1.547
	3	17,191,728	6,331,948	2,140,966	4,190,982	10,859,779	1,349,053	789,096	12,464	8,709,167	1.907	1.908	1.907	1.715
8000 tpd	1	6,907,578	2,921,256	2,128,945	792,312	3,986,321	1,346,127	178,333	-	2,461,861	1.917	1.911	1.931	1.365
	2	17,301,384	5,840,367	2,145,015	3,695,352	11,461,018	1,353,327	747,041	12,464	9,348,185	1.886	1.907	1.873	1.962
	3	18,177,810	6,430,553	2,145,015	4,285,538	11,747,258	1,353,327	824,052	12,464	9,557,414	1.910	1.907	1.912	1.827
10000 tpd	1	8,408,144	3,652,192	2,134,773	1,517,419	4,755,953	1,354,955	296,463	8,353	3,096,182	1.930	1.909	1.960	1.302
	2	18,488,504	6,505,264	2,150,843	4,354,421	11,983,240	1,362,155	856,442	12,464	9,752,178	1.905	1.906	1.905	1.842
12000 tpd	1	11,883,268	4,381,227	2,150,843	2,230,384	7,502,041	1,362,155	437,135	12,464	5,690,286	1.906	1.906	1.906	1.712
	2	18,488,504	6,505,264	2,150,843	4,354,421	11,983,240	1,362,155	856,442	12,464	9,752,178	1.905	1.906	1.905	1.842

SUMMARY OF BYG-KRIAN (OWNER-OPERATOR) SCHEDULES - FLOTATION

Jugan + Flotation + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,942,377	364,268		364,268	1,578,110		5,963	13,087	1,559,060	2.050		2.050	4.332
	2	1,969,337	364,176		364,176	1,605,161		2,457	7,387	1,595,317	3.653		3.653	4.408
	3	1,456,317	308,533		308,533	1,147,784		-	1,122	1,146,662	3.648		3.648	3.720
6000 tpd	1	2,841,784	546,079		546,079	2,295,704		5,963	14,051	2,275,691	2.857		2.857	4.204
	2	2,722,412	498,049		498,049	2,224,362		2,457	7,555	2,214,350	3.337		3.337	4.466
8000 tpd	1	4,270,238	728,268		728,268	3,541,970		8,736	20,570	3,512,664	2.741		2.741	4.864
	2	1,418,974	323,043		323,043	1,095,930		-	1,815	1,094,115	3.833		3.833	3.393
10000 tpd	1	5,251,037	910,471		910,471	4,340,566		8,736	21,289	4,310,541	3.024		3.024	4.767
	2	582,836	150,016		150,016	432,820		-	1,233	431,587	3.297		3.297	2.885
12000 tpd	1	5,837,853	1,060,720		1,060,720	4,777,133		8,736	22,522	4,745,875	3.062		3.062	4.504

Jugan + Flotation + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,942,377	364,268		364,268	1,578,110		5,963	13,087	1,559,060	2.050		2.050	4.332
	2	3,911,715	728,444		728,444	3,183,271		8,420	20,475	3,154,377	2.851		2.851	4.370
	3	5,368,031	1,036,976		1,036,976	4,331,055		8,420	21,597	4,301,039	3.088		3.088	4.177
6000 tpd	1	2,841,784	546,079		546,079	2,295,704		5,963	14,051	2,275,691	2.857		2.857	4.204
	2	5,564,195	1,044,129		1,044,129	4,520,066		8,420	21,606	4,490,041	3.086		3.086	4.329
8000 tpd	1	4,270,238	728,268		728,268	3,541,970		8,736	20,570	3,512,664	2.741		2.741	4.864
	2	5,689,211	1,051,312		1,051,312	4,637,900		8,736	22,385	4,606,779	3.077		3.077	4.412
10000 tpd	1	5,251,037	910,471		910,471	4,340,566		8,736	21,289	4,310,541	3.024		3.024	4.767
	2	5,833,873	1,060,487		1,060,487	4,773,386		8,736	22,522	4,742,128	3.062		3.062	4.501
12000 tpd	1	5,837,853	1,060,720		1,060,720	4,777,133		8,736	22,522	4,745,875	3.062		3.062	4.504

SUMMARY OF BYG-KRIAN (OWNER-OPERATOR) SCHEDULES - BIOX

Jugan + BIOX + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,909,115	364,559		364,559	1,544,556		42,751	10,115	1,491,691	3.216		3.216	4.237
	2	2,305,247	363,496		363,496	1,941,751		16,327	424	1,925,000	3.917		3.917	5.342
	3	447,597	133,576		133,576	314,021		4,074	1,107	308,840	3.370		3.370	2.351
6000 tpd	1	3,056,880	546,393		546,393	2,510,486		29,432	7,351	2,473,702	3.943		3.943	4.595
	2	1,757,429	324,574		324,574	1,432,855		37,432	4,994	1,390,429	2.808		2.808	4.415
8000 tpd	1	3,863,990	728,863		728,863	3,135,127		39,624	8,924	3,086,579	3.785		3.785	4.301
	2	849,104	140,191		140,191	708,913		24,858	2,992	681,063	2.118		2.118	5.057
10000 tpd	1	5,186,344	886,374		886,374	4,299,969		71,802	13,812	4,214,356	3.504		3.504	4.851
12000 tpd	1	5,188,980	886,642		886,642	4,302,338		72,215	14,337	4,215,786	3.503		3.503	4.852

Jugan + BIOX + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,909,115	364,559		364,559	1,544,556		42,751	10,115	1,491,691	3.216		3.216	4.237
	2	4,214,363	728,055		728,055	3,486,307		59,077	10,538	3,416,691	3.566		3.566	4.789
	3	4,661,960	861,631		861,631	3,800,329		63,151	11,646	3,725,532	3.535		3.535	4.411
6000 tpd	1	3,056,880	546,393		546,393	2,510,486		29,432	7,351	2,473,702	3.943		3.943	4.595
	2	4,814,309	870,967		870,967	3,943,342		66,865	12,346	3,864,131	3.520		3.520	4.528
8000 tpd	1	3,863,990	728,863		728,863	3,135,127		39,624	8,924	3,086,579	3.785		3.785	4.301
	2	4,713,093	869,054		869,054	3,844,039		64,482	11,916	3,767,642	3.516		3.516	4.423
10000 tpd	1	5,186,344	886,374		886,374	4,299,969		71,802	13,812	4,214,356	3.504		3.504	4.851
12000 tpd	1	5,188,980	886,642		886,642	4,302,338		72,215	14,337	4,215,786	3.503		3.503	4.852

SUMMARY OF BYG-KRIAN (OWNER-OPERATOR) SCHEDULES - POX

Jugan + POX + Owner-Operator (Incremental Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	Ratio
4000 tpd	1	2,339,397	364,465		364,465	1,974,932		19,583	7,562	1,947,787	2.998		2.998	5.419
	2	1,557,365	363,797		363,797	1,193,568		5,523	2,744	1,185,300	4.339		4.339	3.281
	3	1,241,397	193,758		193,758	1,047,639		22,938	4,238	1,020,462	2.362		2.362	5.407
6000 tpd	1	3,199,491	546,574		546,574	2,652,917		21,156	7,456	2,624,306	3.819		3.819	4.854
	2	1,971,197	378,333		378,333	1,592,863		28,431	7,096	1,557,336	2.764		2.764	4.210
8000 tpd	1	3,932,798	728,180		728,180	3,204,619		24,511	9,660	3,170,448	3.664		3.664	4.401
	2	1,358,533	203,355		203,355	1,155,178		25,258	5,976	1,123,944	2.353		2.353	5.681
10000 tpd	1	5,236,760	910,111		910,111	4,326,649		48,393	15,947	4,262,309	3.351		3.351	4.754
	2	79,032	25,910		25,910	53,122		2,002	581	50,540	3.900		3.900	2.050
12000 tpd	1	5,321,911	936,487		936,487	4,385,424		50,395	16,528	4,318,502	3.366		3.366	4.683

Jugan + POX + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	Ratio
4000 tpd	1	2,339,397	364,465		364,465	1,974,932		19,583	7,562	1,947,787	2.998		2.998	5.419
	2	3,896,762	728,262		728,262	3,168,500		25,106	10,307	3,133,087	3.668		3.668	4.351
	3	5,138,159	922,020		922,020	4,216,139		48,045	14,544	4,153,550	3.393		3.393	4.573
6000 tpd	1	3,199,491	546,574		546,574	2,652,917		21,156	7,456	2,624,306	3.819		3.819	4.854
	2	5,170,688	924,908		924,908	4,245,780		49,587	14,552	4,181,641	3.388		3.388	4.591
8000 tpd	1	3,932,798	728,180		728,180	3,204,619		24,511	9,660	3,170,448	3.664		3.664	4.401
	2	5,291,331	931,535		931,535	4,359,796		49,769	15,636	4,294,392	3.378		3.378	4.680
10000 tpd	1	5,236,760	910,111		910,111	4,326,649		48,393	15,947	4,262,309	3.351		3.351	4.754
	2	5,315,792	936,021		936,021	4,379,771		50,395	16,528	4,312,849	3.367		3.367	4.679
12000 tpd	1	5,321,911	936,487		936,487	4,385,424		50,395	16,528	4,318,502	3.366		3.366	4.683

SUMMARY OF BYG-KRIAN (OWNER-OPERATOR) SCHEDULES - ALBION

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip Ratio
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	
4000 tpd	1	2,267,688	364,071		364,071	1,903,617		53,320	7,295	1,843,003	3.971		3.971	5.229
	2	1,861,995	364,494		364,494	1,497,502		51,713	2,218	1,443,571	3.845		3.845	4.108
	3	41,467	13,731		13,731	27,736		-	698	27,038	3.573		3.573	2.020
6000 tpd	1	3,083,610	546,206		546,206	2,537,403		58,608	6,728	2,472,067	4.256		4.256	4.646
	2	1,090,872	196,987		196,987	893,885		46,425	3,482	843,978	2.907		2.907	4.538
8000 tpd	1	4,465,733	728,120		728,120	3,737,614		122,321	10,740	3,604,552	3.907		3.907	5.133
	2	331,013	42,645		42,645	288,369		5,623	252	282,494	2.952		2.952	6.762
10000 tpd	1	4,808,999	771,907		771,907	4,037,092		128,230	11,073	3,897,789	3.851		3.851	5.230
12000 tpd	1	4,811,407	772,022		772,022	4,039,385		128,230	11,073	3,900,082	3.851		3.851	5.232

Jugan + Albion + Owner-Operator (Cumulative Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip Ratio
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	
4000 tpd	1	2,267,688	364,071		364,071	1,903,617		53,320	7,295	1,843,003	3.971		3.971	5.229
	2	4,129,683	728,565		728,565	3,401,119		105,033	9,512	3,286,573	3.908		3.908	4.668
	3	4,171,150	742,295		742,295	3,428,855		105,033	10,211	3,313,611	3.902		3.902	4.619
6000 tpd	1	3,083,610	546,206		546,206	2,537,403		58,608	6,728	2,472,067	4.256		4.256	4.646
	2	4,174,482	743,193		743,193	3,431,289		105,033	10,211	3,316,045	3.898		3.898	4.617
8000 tpd	1	4,465,733	728,120		728,120	3,737,614		122,321	10,740	3,604,552	3.907		3.907	5.1
	2	4,796,747	770,764		770,764	4,025,982		127,944	10,992	3,887,046	3.854		3.854	5.2
10000 tpd	1	4,808,999	771,907		771,907	4,037,092		128,230	11,073	3,897,789	3.851		3.851	5.230
12000 tpd	1	4,811,407	772,022		772,022	4,039,385		128,230	11,073	3,900,082	3.851		3.851	5.232

SUMMARY OF BYG-KRIAN (CONTRACT-MINING) SCHEDULES - FLOTATION

Jugan + Flotation + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	Ratio
4000 tpd	1	1,702,026	364,423		364,423	1,337,604		6,296	14,323	1,316,985	1.405		1.405	3.671
	2	1,966,275	363,761		363,761	1,602,514		837	647	1,601,030	4.605		4.605	4.405
	3	837,287	244,581		244,581	592,706		-	834	591,872	3.695		3.695	2.423
6000 tpd	1	2,935,457	546,037		546,037	2,389,420		7,199	14,490	2,367,731	1.986		1.986	4.376
	2	2,018,665	459,388		459,388	1,559,277		-	1,635	1,557,642	4.506		4.506	3.394
8000 tpd	1	3,856,619	728,432		728,432	3,128,187		7,199	14,806	3,106,182	2.882		2.882	4.294
	2	1,114,380	278,948		278,948	835,432		-	1,477	833,956	3.791		3.791	2.995
10000 tpd	1	4,701,831	910,139		910,139	3,791,692		7,199	15,185	3,769,308	3.096		3.096	4.166
	2	274,417	97,948		97,948	176,469		-	1,097	175,372	3.464		3.464	1.802
12000 tpd	1	5,181,648	1,021,034		1,021,034	4,160,614		7,953	21,593	4,131,068	3.116		3.116	4.075

Jugan + Flotation + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	Ratio
4000 tpd	1	1,702,026	364,423		364,423	1,337,604		6,296	14,323	1,316,985	1.405		1.405	3.671
	2	3,668,301	728,183		728,183	2,940,118		7,133	14,970	2,918,015	3.003		3.003	4.038
	3	4,505,588	972,764		972,764	3,532,824		7,133	15,804	3,509,887	3.177		3.177	3.632
6000 tpd	1	2,935,457	546,037		546,037	2,389,420		7,199	14,490	2,367,731	1.986		1.986	4.376
	2	4,954,123	1,005,425		1,005,425	3,948,697		7,199	16,126	3,925,373	3.137		3.137	3.927
8000 tpd	1	3,856,619	728,432		728,432	3,128,187		7,199	14,806	3,106,182	2.882		2.882	4.294
	2	4,970,999	1,007,380		1,007,380	3,963,619		7,199	16,282	3,940,138	3.133		3.133	3.935
10000 tpd	1	4,701,831	910,139		910,139	3,791,692		7,199	15,185	3,769,308	3.096		3.096	4.166
	2	4,976,248	1,008,087		1,008,087	3,968,161		7,199	16,282	3,944,680	3.132		3.132	3.936
12000 tpd	1	5,181,648	1,021,034		1,021,034	4,160,614		7,953	21,593	4,131,068	3.116		3.116	4.075

SUMMARY OF BYG-KRIAN (CONTRACT-MINING) SCHEDULES - BIOX

Jugan + BIOX + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,972,480	364,659		364,659	1,607,822		17,559	6,191	1,584,073	3.522		3.522	4.409
	2	1,766,737	363,391		363,391	1,403,347		29,197	4,305	1,369,845	3.802		3.802	3.862
	3	339,189	86,217		86,217	252,972		601	805	251,566	3.340		3.340	2.934
6000 tpd	1	2,781,133	546,061		546,061	2,235,072		20,874	6,728	2,207,470	4.058		4.058	4.093
	2	1,309,168	270,862		270,862	1,038,306		28,339	4,573	1,005,395	2.738		2.738	3.833
8000 tpd	1	3,605,913	728,421		728,421	2,877,492		31,377	9,443	2,836,672	3.825		3.825	3.950
	2	505,644	90,439		90,439	415,205		17,835	1,950	395,419	1.938		1.938	4.591
10000 tpd	1	4,169,641	828,120		828,120	3,341,520		50,624	11,468	3,279,428	3.592		3.592	4.035
12000 tpd	1	4,660,798	861,004		861,004	3,799,794		63,151	11,646	3,724,997	3.537		3.537	4.413

Jugan + BIOX + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	1,972,480	364,659		364,659	1,607,822		17,559	6,191	1,584,073	3.522		3.522	4.409
	2	3,739,217	728,049		728,049	3,011,168		46,755	10,496	2,953,917	3.662		3.662	4.136
	3	4,078,407	814,266		814,266	3,264,141		47,356	11,301	3,205,484	3.628		3.628	4.009
6000 tpd	1	2,781,133	546,061		546,061	2,235,072		20,874	6,728	2,207,470	4.058		4.058	4.093
	2	4,090,301	816,923		816,923	3,273,378		49,212	11,301	3,212,865	3.620		3.620	4.007
8000 tpd	1	3,605,913	728,421		728,421	2,877,492		31,377	9,443	2,836,672	3.825		3.825	3.950
	2	4,111,558	818,861		818,861	3,292,697		49,212	11,394	3,232,091	3.616		3.616	4.021
10000 tpd	1	4,169,641	828,120		828,120	3,341,520		50,624	11,468	3,279,428	3.592		3.592	4.035
12000 tpd	1	4,660,798	861,004		861,004	3,799,794		63,151	11,646	3,724,997	3.537		3.537	4.413

SUMMARY OF BYG-KRIAN (CONTRACT-MINING) SCHEDULES - POX

Jugan + POX + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip Ratio
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	
4000 tpd	1	2,077,792	365,149		365,149	1,712,642		17,604	9,334	1,685,704	2.627		2.627	4.690
	2	1,506,422	363,300		363,300	1,143,121		259	712	1,142,151	4.796		4.796	3.147
	3	783,356	140,735		140,735	642,620		18,977	2,078	621,566	2.306		2.306	4.566
6000 tpd	1	2,924,835	546,469		546,469	2,378,366		17,605	10,543	2,350,218	3.639		3.639	4.352
	2	1,472,000	327,689		327,689	1,144,311		19,236	2,252	1,122,823	3.186		3.186	3.492
8000 tpd	1	3,800,462	728,086		728,086	3,072,376		19,384	7,970	3,045,022	3.780		3.780	4.220
	2	933,300	173,149		173,149	760,151		26,010	5,782	728,359	1.905		1.905	4.390
10000 tpd	1	4,752,369	906,610		906,610	3,845,758		46,009	13,940	3,785,809	3.405		3.405	4.242
12000 tpd	1	4,761,435	907,239		907,239	3,854,196		46,009	13,940	3,794,247	3.404		3.404	4.248

Jugan + POX + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock	Total Ore	Meas	Ind	Total Waste	Meas (w)	Ind (w)	Inf (w)	Waste	Ave_Grade	Meas_Grade	Ind_Grade	Strip Ratio
		(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	(t)	g/t	g/t	g/t	
4000 tpd	1	2,077,792	365,149		365,149	1,712,642		17,604	9,334	1,685,704	2.627		2.627	4.690
	2	3,584,213	728,449		728,449	2,855,764		17,863	10,046	2,827,854	3.709		3.709	3.920
	3	4,367,569	869,185		869,185	3,498,384		36,840	12,124	3,449,420	3.482		3.482	4.025
6000 tpd	1	2,924,835	546,469		546,469	2,378,366		17,605	10,543	2,350,218	3.639		3.639	4.352
	2	4,396,835	874,158		874,158	3,522,677		36,840	12,795	3,473,042	3.469		3.469	4.030
8000 tpd	1	3,800,462	728,086		728,086	3,072,376		19,384	7,970	3,045,022	3.780		3.780	4.220
	2	4,733,762	901,235		901,235	3,832,527		45,394	13,752	3,773,381	3.419		3.419	4.253
10000 tpd	1	4,752,369	906,610		906,610	3,845,758		46,009	13,940	3,785,809	3.405		3.405	4.242
12000 tpd	1	4,761,435	907,239		907,239	3,854,196		46,009	13,940	3,794,247	3.404		3.404	4.248

SUMMARY OF BYG-KRIAN (CONTRACT-MINING) SCHEDULES - ALBION

Jugan + Albion + Contract-Mining (Incremental Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,371,656	364,366		364,366	2,007,290		81,000	7,432	1,918,857	3.591		3.591	5.509
	2	1,480,870	347,248		347,248	1,133,623		9,705	1,204	1,122,713	4.387		4.387	3.265
6000 tpd	1	2,969,564	546,188		546,188	2,423,376		58,192	7,438	2,357,746	4.241		4.241	4.437
	2	944,893	170,778		170,778	774,115		35,383	1,516	737,217	3.092		3.092	4.533
8000 tpd	1	3,952,423	722,377		722,377	3,230,046		94,898	9,530	3,125,618	3.951		3.951	4.471
10000 tpd	1	4,081,073	733,809		733,809	3,347,264		100,639	9,620	3,237,006	3.923		3.923	4.562
12000 tpd	1	4,078,759	734,133		734,133	3,344,626		100,639	10,211	3,233,777	3.922		3.922	4.556

Jugan + Albion + Contract-Mining (Cumulative Schedules)

Production Level	Year	Rock (t)	Total Ore (t)	Meas (t)	Ind (t)	Total Waste (t)	Meas (w) (t)	Ind (w) (t)	Inf (w) (t)	Waste (t)	Ave_Grade g/t	Meas_Grade g/t	Ind_Grade g/t	Strip Ratio
4000 tpd	1	2,371,656	364,366		364,366	2,007,290		81,000	7,432	1,918,857	3.591		3.591	5.509
	2	3,852,527	711,614		711,614	3,140,913		90,705	8,637	3,041,571	3.980		3.980	4.414
6000 tpd	1	2,969,564	546,188		546,188	2,423,376		58,192	7,438	2,357,746	4.241		4.241	4.437
	2	3,914,456	716,966		716,966	3,197,491		93,574	8,954	3,094,963	3.967		3.967	4.460
8000 tpd	1	3,952,423	722,377		722,377	3,230,046		94,898	9,530	3,125,618	3.951		3.951	4.471
10000 tpd	1	4,081,073	733,809		733,809	3,347,264		100,639	9,620	3,237,006	3.923		3.923	4.562
12000 tpd	1	4,078,759	734,133		734,133	3,344,626		100,639	10,211	3,233,777	3.922		3.922	4.556

A16-5. Mine Equipment Lists by Production & Mining Type

Equipment for Base Case_8000 TPD Owner-Operator

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
2	Production Drill, Sandvik DX800, 76mm to 127mm hole, crawler
2	Hydraulic Shovel, 7m3, CAT6015/FS
1	Wheel Loader or FEL, 6.4 m3 for pit operation
1	Wheel Loader or FEL, 6.4 m3 for Stockpile operation
1	CAT_D10T Dozer with ripper
1	D6W Tractor (CAT_D6R XL)
9	Hauling Truck, Rigid Rear Dump CAT_772G
2	Road Grader, CAT_12K
2	Water Truck (10,000 liters)
2	Compactor, CAT CS533E for haul road maintenance
2	Explosive Truck (1000 kg cap) or Mobile Mixing Unit
1	Cable Bolter (Surface Drill + grouting machine combo)
2	Service/Tire Truck (off highway road)
5	4WD LV Toyota Hi-lux
Note: Equipment for Waste Dump operation are not included	
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
4	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
22	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
100	Caplamps with charger
65	Handheld Radios

Equipment for 4000 TPD Owner-Operator

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
1	Production Drill, Sandvik DX800, 76mm to 127mm hole, crawler
1	Hydraulic Shovel, 7m3, CAT6015/FS
1	Wheel Loader or FEL, 6.4 m3 for pit operation
1	Wheel Loader or FEL, 6.4 m3 for Stockpile operation
1	CAT_D10T Dozer with ripper
1	D6W Tractor (CAT_D6R XL)
5	Hauling Truck, Rigid Rear Dump CAT_772G
1	Road Grader, CAT_12K
1	Water Truck (10,000 liters)
1	Compactor, CAT CS533E for haul road maintenance
1	Explosive Truck (1000 kg cap) or Mobile Mixing Unit
1	Cable Bolter (Surface Drill + grouting machine combo)
1	Service/Tire Truck (off highway road)
4	4WD LV Toyota Hi-lux
Note: Equipment for Waste Dump operation are not included	
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
3	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
1	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
500	HDPE Pipes with fittings (for air & water), 6m length
12	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
50	Caplamps with charger
35	Handheld Radios

Equipment for 6000 TPD Owner-Operator

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
2	Production Drill, Sandvik DX800, 76mm to 127mm hole, crawler
2	Hydraulic Shovel, 7m3, CAT6015/FS
1	Wheel Loader or FEL, 6.4 m3 for Stockpile operation
1	CAT_D10T Dozer with ripper
1	D6W Tractor (CAT_D6R XL)
8	Hauling Truck, Rigid Rear Dump CAT_772G
2	Road Grader, CAT_12K
2	Water Truck (10,000 liters)
2	Compactor, CAT CS533E for haul road maintenance
2	Explosive Truck (1000 kg cap) or Mobile Mixing Unit

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
1	Cable Bolter (Surface Drill + grouting machine combo)
2	Service/Tire Truck (off highway road)
4	4WD LV Toyota Hi-lux
Note: Equipment for Waste Dump operation are not included	
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
4	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
18	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
60	Caplamps with charger
50	Handheld Radios

Equipment for 10000 TPD Owner-Operator

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
3	Production Drill, Sandvik DX800, 76mm to 127mm hole, crawler
3	Hydraulic Shovel, 7m3, CAT6015/FS
1	Wheel Loader or FEL, 6.4 m3 for pit operation
1	Wheel Loader or FEL, 6.4 m3 for Stockpile operation
2	CAT_D10T Dozer with ripper
2	D6W Tractor (CAT_D6R XL)
11	Hauling Truck, Rigid Rear Dump CAT_772G
3	Road Grader, CAT_12K
3	Water Truck (10,000 liters)
3	Compactor, CAT CS533E for haul road maintenance
3	Explosive Truck (1000 kg cap) or Mobile Mixing Unit
1	Cable Bolter (Surface Drill + grouting machine combo)
2	Service/Tire Truck (off highway road)
6	4WD LV Toyota Hi-lux
Note: Equipment for Waste Dump operation are not included	
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
5	Mobile Light Plant (13kW)
2	Pit Dewatering Pump, centrifugal 75 li/sec
2	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
26	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
115	Caplamps with charger
75	Handheld Radios

Equipment for 12000 TPD Owner-Operator

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
3	Production Drill, Sandvik DX800, 76mm to 127mm hole, crawler
3	Hydraulic Shovel, 7m3, CAT6015/FS
1	Wheel Loader or FEL, 6.4 m3 for pit operation
1	Wheel Loader or FEL, 6.4 m3 for Stockpile operation
2	CAT_D10T Dozer with ripper
2	D6W Tractor (CAT_D6R XL)
13	Hauling Truck, Rigid Rear Dump CAT_772G
3	Road Grader, CAT_12K
3	Water Truck (10,000 liters)
3	Compactor, CAT CS533E for haul road maintenance
3	Explosive Truck (1000 kg cap) or Mobile Mixing Unit
2	Cable Bolter (Surface Drill + grouting machine combo)
3	Service/Tire Truck (off highway road)
6	4WD LV Toyota Hi-lux
Note: Equipment for Waste Dump operation are not included	
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
6	Mobile Light Plant (13kW)
2	Pit Dewatering Pump, centrifugal 75 li/sec
2	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
30	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2

Open Pit Equipment (Mining Fleet for Owner-Operator Option)	
No. of Units	**Description and Specification**
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
120	Caplamps with charger
80	Handheld Radios

Equipment for Base Case_8000 TPD Contract Mining Option

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
4	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
22	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
100	Caplamps with charger
65	Handheld Radios

Equipment for 4000 TPD Contract Mining Option

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
3	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
1	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
12	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
50	Caplamps with charger
35	Handheld Radios

Equipment for 6000 TPD Contract Mining Option

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
4	Mobile Light Plant (13kW)
1	Pit Dewatering Pump, centrifugal 75 li/sec
1	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
18	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
1	Set of First Aid Equipment & paraphernalias
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
60	Caplamps with charger
50	Handheld Radios

Equipment for 10000 TPD Contract Mining Option

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
5	Mobile Light Plant (13kW)
2	Pit Dewatering Pump, centrifugal 75 li/sec
2	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
26	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
115	Caplamps with charger
75	Handheld Radios

Equipment for 12000 TPD Contract Mining Option

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
Fixed Plant & Capital Services	
2	Surface Blaster (i-kon)/Exploder
6	Mobile Light Plant (13kW)
2	Pit Dewatering Pump, centrifugal 75 li/sec
2	Pit Dewatering Pump, centrifugal 40 li/sec
1	Dewatering Pump, diaphragm type 20 li/sec
2	Vacuum Pump for blasthole dewatering
1	Butt Welder for HDPE pipes
1	Set of Workshop Tools & Equipment
500	HDPE Pipes with fittings (for air & water), 6m length
30	Fire Fighting Equipment for each mobile machine
Offices, Workshops and Stores	
2	Field Office (mine & maintenance)
1	Workshop, 1000 m2
1	Explosive Magazine & ANFO Bin
4	Portable Stores (container vans)
Health-Safety and Environment	
1	Set of First Aid Equipment & paraphernalias

Open Pit Equipment (No Mining Fleet for Contract Mining Option)	
No. of Units	**Description and Specification**
10	Fire Fighting Equipment - Fixed
1	Fire Hydrant System
1	Ambulance
Mine Services: (mine planning, survey & geology)	
1	Survey Equipment
1	GeoMIMS System for pit operation
5	Computer / Laptops
Communication & Security	
1	Telephone System
1	Base Radio for pit operation
5	Wireless Camera System
5	Motorbikes for Security personnel
Sundries	
1	Office Furniture (one lot)
4	Workshop Racks & Storage
5	Oxy-acetylene Equipment
120	Caplamps with charger
80	Handheld Radios

A16-6. Mine Equipment Selection – Calculations & Parameters

Single Period Operating Parameters - For Shovel & Truck Match-up Simulation

For Brand New Equipment	Period	Loader	Trucks
Period = 1 year		1	1
Days in Period	days	365	365
Days operated	days/wk	7	7
Shifts per day	shifts/day	3	3
Hours operating	hrs/day	24	24
Holidays (shutdowns)	days in period	16	16
Not scheduled - days	days in period	0	0
Not scheduled - shifts	days in period	0	0
Other	days in period	1	1
SCHEDULED TIME	**hrs in period**	**8,352**	**8,352**
Working days in 1 year period	days in period	348	348
Major overhaul or Repair	days in period	3	3
AVAILABLE DAYS	**days in period**	**345**	**345**
Maintenance - Planned	hrs in period	336	384
Planned - Out of Scheduled time	hrs in period	192	192
Maintenance - Breakdown	hrs in period	312	312
Breakdown - Out of Scheduled Time	hrs in period	156	156
AVAILABLE TIME (AT)	**hrs in period**	**7,284**	**7,236**
Industrial Delay	days in period	1	1
Weather Delay	days in period	5	5
Not manned	hrs/day	0.3	0.3
Safety	hrs/day	0.5	0.5
No power	hrs/day	0.1	0.1
Shift changes	hrs/day	0.6	0.5
Total meal break losses	hrs/day	1.5	1.5
Blasting	hrs/day	0.5	0.5
Other	hrs/day	0.1	0.1
UTILISED TIME - UT (OH)	**UT hrs in period**	**5,919.6**	**5,905.5**
Wait	hrs/day	0.5	0.5
Prestart checks	hrs/day	0.25	0.25
Daily service	hrs/day	0.5	0.75
Refuel	hrs/day	0.5	0.5
Tyres Check	hrs/day	0.25	0.25
Clean-up	hrs/day	0.5	0.5
Loader Move	hrs/day	0.75	0.5
Other	hrs/day	0.5	0.5
OPERATED HRS - OT (DOH)	OT hrs in period	**4,648**	**4,634**
Equipment Availability	%	87.2	86.6
Use of Availability	%	81.3	81.6

Shovel and Truck Specifications Relative to Productivity

Loader Type (hydraulic)	Dipper (heap) cm	Rated load limit (RSL) t	Max Cut Height m	Dump Radius m	Fill Factor %	Swing 90° cycle secs	Fuel Rate lit/UT
SHOVEL CAT6015FS	7.0	15.0	11	10.5	95	30	55
BACK-HOE CAT390DL	6.0	13.0	12	8	90	30	51
TRUCK TYPE		**Rated Tray (heap)**	**Load limit**	**Operating Width**	**Turning Radius**	**Fuel Rate**	
RIGID	CAT 772G	31.2	56	3.7	10.5	40	
ARTICULATED	CAT 740B	24	40	3.5	8.14	32	
Transmission			**CAT 772G**	**CAT 740B**			
Gear1	Forward	km/hr	12.9	8.90			
Gear2	Forward	km/hr	17.7	12.10			
Gear3	Forward	km/hr	24.0	16.40			
Gear4	Forward	km/hr	32.2	22.00			
Gear5	Forward	km/hr	43.6	30.00			
Gear6	Forward	km/hr	58.7	40.00			
Gear7	Forward	km/hr	79.7	54.70			
Gear8	Reverse1	km/hr	16.9	8.40			
Gear9	Reverse2	km/hr		11.60			

Loader and Truck Productivity Modifier - Using CAT 6015FS Shovel & CAT 772G Truck

Periodic Targets Single Period	1 yr	Indicated loader efficiency	%	94
Loading Method (Double Side Loading)	DSL	Actual Dipper Fill Factor	%	95
Dipper Capacity, cm	7	Actual swing cycle time	secs	32
Modified Fill Factor (Shovel)	95%	Volume in dipper	cm	6.7
Modified Fill Factor (Backhoe)	90%	Weight in the dipper	t	14.2
Modified Tray Cap._CAT 772G	28.08	Percent of dipper load limit	%	94
Modified Tray Cap._CAT 740B	21.6	Nom passes by volume		4.12
Bench Height, m	15	Nom passes by weight		3.86
Panel Width, m	30	Limited by		Weight
Density (SG)	2.60	Actual passes		4.0
Swell in Dipper	1.20	Last pass load factor	%	86
Swell in Tray	1.25	Truck load volume	cm	26
Adjusted Swing Cycle, sec	32.00	Truck load weight	t	55
		Percent of tray load limit	%	98
Operator Skills Modifier		Truck load time (loader)	secs	128

Operator	Prodn	Truck load time (truck)	secs	96
Skill	%	Operator skills factor		4
1	45%	Diggability factor		4
2	50%	Loader time modified (op & dig)	secs	158
3	75%	Truck load time modified (op/dig)	secs	119
4	90%	Loaded haul distance	m	1,400
5	100%	Empty haul distance	m	1,400
		Truck Dump Time	secs	18
Diggability Modifier		Ave spot at dump	secs	15
Diggability	Prodn	Ave wait at dump	secs	30
Index	%	Ave loaded cycle (excl spot & dump)	secs	232
1	45%	Ave spot at loader	secs	30
2	50%	Ave empty cycle (travel)	secs	179
3	75%	Ave wait at loader	secs	30
4	90%	Bunching character		3
5	100%	Nominal loaders allocated		2.0
		Nominal trucks allocated		9.0
Bunching Character		**Fleet Productivity by period**		
Severe	1	Calculated Match Factor		1.14
Average	2	Instantaneous loader capacity	t/OThr	2,490
Light: 1 loader/truck fleet	3	Calculated loader capacity	Mt	11.6
			Mbcm	4.5
		Indicated loader efficiency from MF	%	95.8
		Indicated truck efficiency from MF	%	84.0
		Efficiency adjusted for truck OT	%	95.6
		Resultant fleet capacity	Mt	11.06

Cycle Time of Shovel and CAT_772G Off-Highway Truck (RIGID)

Truck Hauling Speed					
Jugan Ore Hauling	**Haul Distance (m)**	**Haul Speed (km/h)**		**Transmission**	
From Pit to ROM Area	**1400**	**Loaded**	**Empty**	**Loaded**	**Empty**
From Load/Dump	20	12.9	12.9	1st gear	1st gear
Flat Road	900	24	32.2	3rd gear	4th gear
Ramp	480	17.7	21	2nd gear	2&3 gear
Average Speed		**21.7**	**28.1**		
TOP SPEED SPECS		71.7	79.7		7th gear
Hauling Time in Minutes					
Haul Distance	1.4				
Average Haul Time		3.9	3.0		

Haul Time (Loaded + Empty)		6.9		
TOTAL CYCLE TIME	**Truck**	**Loader**		
Loading				
Actual Swing Cycle time		32		
No. of Dipper Passes		4		
Dipper Cyle (Loading)	128	128		
Average Spotting	30		before loading	
Total Load Cycle	158	128		
Loading Cyle Adjusted	**195**	**158**	adjusted relative to operator skill and diggability	
Haul Time	411.9			
Average Spotting	15		before dumping	
Ave. Dump Time	18			
Ave. Waiting Time	30		before loading	
Ave. Waiting Time	30		before dumping	
Total Cycle Time (sec)	**699.98**	**158.02**		
Cycle Time in minutes	**11.67**	**2.63**		

Cycle Time For Shovel and CAT_740B Articulated Off-Highway Truck

Average Speed in km/hr					
Jugan Ore Hauling	**Haul Distance (m)**	**Haul Speed (km/h)**		**Transmission**	
From Pit to ROM Area	**1400**	**Loaded**	**Empty**	**Loaded**	**Empty**
From Load/Dump	20	8.9	8.9	1st gear	1st gear
Flat Road	900	22	30	4th gear	5th gear
Ramp	480	17	21	2-4 gear	2-5 gear
Average Speed		**20.10**	**26.61**		
TOP SPEED SPECS		54.7	54.7		7th gear
Hauling Time in Minutes					
Haul Distance	1.4				
Average Haul Time		4.2	3.2		
Haul Time (Loaded + Empty)		7.3			
TOTAL CYCLE TIME	Truck	Loader			
Loading					
Actual Swing Cycle time		31			
No. of Dipper Passes		4			

Average Speed in km/hr					
Jugan Ore Hauling	**Haul Distance (m)**	**Haul Speed (km/h)**		**Transmission**	
From Pit to ROM Area	**1400**	**Loaded**	**Empty**	**Loaded**	**Empty**
Dipper Cyle (Loading)	126	125.61			
Average Spotting	30		before loading		
Total Load Cycle	156	125.61			
Loading Cyle Adjusted	192	155.07	adjusted relative to operator skill & diggability		
Haul Time	440.1				
Average Spotting	15		before dumping		
Ave. Dump Time	18				
Ave. Waiting Time	30		before loading		
Ave. Waiting Time	30		before dumping		
Total Cycle Time (sec)	**725.25**	**155.07**			
Cycle Time in minutes	**12.09**	**2.585**			

Shovel & Truck Match-up Data Based on Hauling Distance

Hauling Distance	meters	800	1000	1200	1400	1600
Shovel_CAT6015	**units**	**2**	**2**	**2**	**2**	**2**
Truck_CAT 772G	**units**	**7**	**8**	**8**	**9**	**10**
Loader Efficiency	%	92.7	93.9	89.3	90.8	92.0
Truck Efficiency	%	87.7	86.5	90.6	89.4	88.4
Net Fleet Efficiency	%	92.4	93.6	89.0	90.5	91.7
Fleet Capacity	Mtonnes	10.96	11.1	10.56	10.74	10.88
Match Factor (MF)	%	1.06	1.09	0.99	1.02	1.04
Modified MF	%	1.24	1.25	1.12	1.14	1.16
Loader Capacity	Mtonnes	11.6	11.6	11.6	11.6	11.6
Loader Eff from MF	%	98.4	98.7	95.1	95.8	96.4
Truck Eff from MF	%	79.5	78.9	85.0	84.0	83.2
Fleet Efficiency	%	98.1	98.4	94.8	95.6	96.1
Resultant Fleet Capacity	Mtonnes	11.36	11.39	10.98	11.06	11.12

Back-hoe/Excavator & Truck Match-up Data Based on Hauling Distance

Hauling Distance	m	800	1000	1200	1400	1600
Loader_CAT390 DL	**units**	**3**	**3**	**3**	**3**	**3**
Truck_CAT 740B	**units**	**10**	**11**	**13**	**14**	**15**
Loader Efficiency	%	88.1	87.3	90.7	89.9	89.2
Truck Efficiency	%	91.5	92	89.4	90.1	90.7
Net Fleet Eff.	%	87.9	87	90.5	89.6	88.9

Hauling Distance	m	800	1000	1200	1400	1600
Fleet Capacity	Mtonnes	11.23	11.13	11.57	11.46	11.37
Match Factor (MF)	%	0.96	0.95	1.01	1.00	0.98
Modified MF	%	1.12	1.09	1.15	1.11	1.09
Loader Eff from MF	%	95.2	93.9	96.0	94.9	94
Truck Eff from MF	%	84.9	86.4	83.8	85.2	86.3
Fleet Efficiency	%	94.9	93.6	95.7	94.7	93.7
Resultant Fleet Capacity	Mtonnes	11.85	11.68	11.95	11.81	11.69

CAT_6015 Shovel & CAT_772G Truck Productivity Data for Base Case (8000 TPD)

Jugan Open Pit		Yr 0	Yr 1	Yr 2	Yr3	Yr4
Mining Days		60	365	365	365	180
Hauling Distance		800	1000	1400	1400	1600
ORE	tpd		8,000	8,000	8,000	8,000
STRIP RATIO	W:O		0.90	2.60	2.60	2.50
WASTE	tpd	9,000	7,200	20,800	20,800	20,000
ORE + WASTE	tpd	9,000	15,200	28,800	28,800	28,000
Annual Production	Mtonnes	**0.54**	**5.55**	**10.51**	**10.51**	**5.04**
Loader_CAT6015	units	**1**	**1**	**2**	**2**	**2**
Truck_CAT 772G	units	**4**	**4**	**9**	**9**	**9**
Loader Efficiency	%		94	90.8	90.8	90.8
Truck Efficiency	%		86	89.4	89.4	89.4
Net Fleet Efficiency	%		94	90.5	90.5	90.5
Fleet Capacity	**Mtonnes**	**5.55**	**5.55**	**10.74**	**10.74**	**5.05**
Match Factor (MF)	%		1.09	1.02	1.02	1.02

CAT_390DL Loader & CAT_740B Truck Productivity Data for Base Case (8000 TPD)

Jugan Open Pit		Yr 0	Yr 1	Yr 2	Yr3	Yr4
Mining Days		60	365	365	365	180
Hauling Distance		800	1000	1400	1400	1600
ORE	tpd		8,000	8,000	8,000	8,000
STRIP RATIO	W:O		0.90	2.60	2.60	2.50
WASTE	tpd	9,000	7,200	20,800	20,800	20,000
ORE + WASTE	tpd	9,000	15,200	28,800	28,800	28,000
Annual Production	**Mtonnes**	**0.54**	**5.55**	**10.51**	**10.51**	**5.04**
Loader_CAT390 DL	**units**	**2**	**2**	**3**	**3**	**3**
Truck_CAT 740B	**units**	**7**	**7**	**14**	**14**	**15**
Loader Efficiency	%		84.7	89.9	89.9	89.2
Truck Efficiency	%		93.6	90.1	90.1	90.7
Net Fleet Efficiency	%		84.5	89.6	89.6	88.9

Jugan Open Pit		Yr 0	Yr 1	Yr 2	Yr3	Yr4
Fleet Capacity	**Mtonnes**	**7.20**	**7.2**	**11.46**	**11.46**	**5.61**
Match Factor (MF)	%		0.91	1.00	1.00	0.98

A16-7. Drill & Blast Tables & Calculations

A. Drill and Blast Design – Calculation of Burden, B (Table 1.1, Table 1.2 & Table 1.3)

Where; B – Burden
B1 - based on specific gravity of rock and explosive
B2 - based on relative bulk energy of explosive (REE)

Calculated Burden (B) Using Anfo

Blasthole Diameter		**BURDEN** (in meters)		**B1** (corrected), meters		**B2** (corrected), meters		**B**
mm	inches	B1	B2	For RMR =	For RMR=	For RMR =	For RMR =	**Ave.**
		m	m	(fair/good)	(poor)	(fair/good)	(poor)	**M**
76	3.0	1.94	2.06	2.14	2.53	2.27	2.68	**2.4**
89	3.5	2.27	2.41	2.49	2.95	2.65	3.13	**2.8**
89	3.5		2.41			2.65	3.13	**2.9**
102	4.0	2.59	2.75	2.85	3.37	3.03	3.58	**3.2**
115	4.5	2.92	3.09	3.21	3.79	3.40	4.02	**3.6**
127	5.0	3.24	3.44	3.56	4.21	3.78	4.47	**4.0**

Calculated Burden (B) Using Bulk Emulsion Explosive (Orica Fortis)

Blasthole Diameter		**BURDEN** in meters		**B1** (corrected), meters		**B2** (corrected), meters		**B**
mm	inches	B1	B2	For RMR =	For RMR =	For RMR =	For RMR =	**Ave.**
		m	m	(fair/good)	(poor)	(fair/good)	(poor)	**m**
76	3.0	2.17	2.35	2.39	2.83	2.59	3.06	**2.7**
89	3.5	2.54	2.74	2.79	3.30	3.02	3.57	**3.2**
89	3.5		2.74			3.02	3.57	**3.3**
102	4.0	2.90	3.13	3.19	3.77	3.45	4.07	**3.6**
115	4.5	3.26	3.53	3.59	4.24	3.88	4.58	**4.1**
127	5.0	3.62	3.92	3.99	4.71	4.31	5.09	**4.5**

Calculated Burden (B) Using Packaged Emulsion Explosive (PowerFrag/Powerpac)

Blasthole Diameter		**BURDEN** in meters		**B1** (corrected), meters		**B2** (corrected),meters		**B**
mm	inches	B1	B2	For RMR = (fair/good)	For RMR = (poor)	For RMR = (fair/good)	For RMR = (poor)	**Ave.**
76	3.0	1.89	2.52	2.08	2.46	2.78	3.28	**2.6**
89	3.5	2.33	2.94	2.56	3.02	3.24	3.83	**3.2**
89	3.5		2.94			3.24	3.83	**3.5**
102	4.0	2.91	3.36	3.20	3.78	3.70	4.37	**3.8**
115	4.5	3.20	3.65	3.52	4.16	4.01	4.74	**4.1**
127	5.0	3.52	4.05	3.87	4.57	4.46	5.27	**4.5**

B. Drill and Blast Design – Calculation of Stemming, Subdrill, Stiffness Ratio & Spacing

Where; T – Stemming

J – Subdrill
SR – Stiffness ratio
S – Spacing
L – Bench or Flitch Height

Stemming (T), Subdrill (J), Stiffness Ratio (SR) & Spacing (S) - Using ANFO

Blasthole Diameter		Burden, B	Stemming	Subdrill	SR, if L=10m	SR, if L=5m	SPACING S=(L+7B)/8	
mm	inches	average	T=0.7B	J=0.3B	SR=L/B	SR=L/B	L=10	L=5
		m	m	m	m	m	m	m
76	3.0	2.4	1.7	0.72	4.2	2.1	3.4	3.4
89 (o)	3.5	2.8	2.0	0.84	3.6	1.8	3.7	3.1
89 (w)	3.5	2.9	2.0	0.87	3.5	1.7	4.0	3.2
102	4.0	3.2	2.2	0.96	3.1	1.6	4.1	3.4
115	4.5	3.6	2.5	1.08	2.8	1.4	4.4	3.8
127	5.0	4.0	2.8	1.20	2.5	1.2	4.8	4.1

Stemming (T), Subdrill (J), Stiffness Ratio (SR) & Spacing (S) - For Bulk Emulsion

Blasthole Diameter		Burden, B	Stemming	Subdrill	SR, if L=10m	SR, if L=5m	SPACING S=(L+7B)/8	
mm	inches	average	T=0.7B	J=0.3B	SR=L/B	SR=L/B	L=10	L=5
		m	m	m	m	m	m	m
76	3.0	2.7	1.9	0.81	3.7	1.8	3.6	3.0
89 (o)	3.5	3.2	2.2	0.95	3.2	1.6	4.0	3.4
89 (w)	3.5	3.3	2.3	0.99	3.0	1.5	4.6	4.6
102	4.0	3.6	2.5	1.09	2.8	1.4	4.4	3.8
115	4.5	4.1	2.9	1.22	2.5	1.2	4.8	4.2
127	5.0	4.5	3.2	1.36	2.2	1.1	5.2	4.6

Stemming (T), Subdrill (J), Stiffness Ratio (SR) & Spacing (S)-For Packaged Emulsion

Blasthole Diameter		Burden, B	Stemming	Subdrill	SR, if L=10m	SR, if L=5m	SPACING S=(L+7B)/8	
mm	inches	average	T=0.7B	J=0.3B	SR=L/B	SR=L/B	L=10	L=5
		m	m	m	m	m	m	m
76	3.0	2.6	1.9	0.79	3.8	1.9	3.6	2.9
89 (o)	3.5	3.2	2.2	0.95	3.2	1.6	4.0	3.4
89 (w)	3.5	3.5	2.5	1.06	2.8	1.4	4.3	3.7
102	4.0	3.8	2.6	1.13	2.7	1.3	4.5	3.9
115	4.5	4.1	2.9	1.23	2.4	1.2	4.8	4.2
127	5.0	4.5	3.2	1.36	2.2	1.1	5.2	4.6

C. Drill and Blast Design - Calculation of Powder Column, BCM and Powder Factor

Where;
PC – Powder Column
BCM – volume in bcm
PF – Powder Factor

Calculated Powder Column (PC) and Powder Factor (PF) Using ANFO

Blasthole Diameter		Powder Column		Volume, BCM		Powder Factor (PF)	
		PC=L+J-T		V = B*S*L		Density*(PC/BCM)	
mm	inches	L=10	L=5	L=10	L=5	L=10	L=5
		m	m	bcm	bcm	kg/m³	kg/m³
76	3.0	9.0	4.0	80.9	40.5	0.407	0.364
89 (o)	3.5	8.9	3.9	103.9	43.2	0.425	0.446
89 (w)	3.5	8.8	3.8	116.7	45.5	0.377	0.420
102	4.0	8.7	3.7	130.0	55.0	0.435	0.439
115	4.5	8.6	3.6	158.8	68.2	0.443	0.429
127	5.0	8.4	3.4	190.5	82.7	0.447	0.416

Calculated Powder Column (PC) & Powder Factor (PF) For Bulk Emulsion

Blasthole Diameter		Powder Column		Volume, BCM		Powder Factor (PF)	
		PC=L+J-T		V = B*S*L		Density*(PC/BCM)	
mm	inches	L=10	L=5	L=10	L=5	L=10	L=5
		m	m	bcm	bcm	kg/m³	kg/m³
76	3.0	8.9	3.9	98.4	40.7	0.475	0.504
89 (o)	3.5	8.7	3.7	127.4	53.8	0.490	0.496
89 (w)	3.5	8.7	3.7	151.6	75.8	0.409	0.347
102	4.0	8.6	3.6	159.9	68.6	0.499	0.483
115	4.5	8.4	3.4	196.0	85.3	0.504	0.466
127	5.0	8.2	3.2	235.7	103.7	0.506	0.448

Calculated Powder Column (PC) and Powder Factor (PF) for Packaged Emulsion

Blasthole Diameter		Powder Column		Volume, BCM		Powder Factor (PF)	
		PC=L+J-T		V = B*S*L		Density*(PC/BCM)	
mm	inches	L=10	L=5	L=10	L=5	L=10	L=5
		m	m	bcm	bcm	kg/m³	kg/m³
76	3.0	8.9	3.9	94.5	39.0	0.359	0.384
89 (o)	3.5	8.7	3.7	127.0	53.6	0.379	0.383
89 (w)	3.5	8.6	3.6	153.3	65.6	0.308	0.301
102	4.0	8.5	3.5	171.0	73.7	0.481	0.458
115	4.5	8.4	3.4	199.1	86.7	0.487	0.449
127	5.0	8.2	3.2	237.5	104.5	0.449	0.396

Blastability Index (by Lilly and Powder Factor

Blastability Index, BI	From Table 7.10	
BI= (RMD + JPS + JPO + SGI + H)/2	case1	case2
RMD	15	15
JPS	10	10
JPO	20 dip out of face	40 dip into face
SGI	15.5	15.5

Blastability Index, BI	From Table 7.10	
BI= (RMD + JPS + JPO + SGI + H)/2	case1	case2
H	3	3
BI	**31.75**	**41.75**
Powder Factor relative to BI (from Figure 3.1)		
PF in kg/tonne	0.13	0.17
PF in kg/m³	**0.341**	**0.445**

D. Drill and Blast Design - Calculation of Cost per Tonne of Explosives

Calculated Explosive Costs Based on Powder Factor (Bench Height, L=10)

Blasthole Diameter		ANFO (at $1.48/kg)		FORTIS (at $2.35/kg)		Powerfrag (at $3.08/kg)		
mm	inches	PF	$/tonne	PF	$/tonne	PF	$/tonne	
76	3.0	0.407	0.23	0.475	0.43	0.359	0.42	
89	3.5	0.425	0.24	0.490	0.44	0.379	0.44	ore
89	3.5	0.377	0.21	0.409	0.37	0.308	0.36	waste
102	4.0	0.435	0.25	0.499	0.45	0.481	0.56	
115	4.5	0.443	0.25	0.504	0.45	0.487	0.57	
127	5.0	0.447	0.25	0.506	0.45	0.449	0.53	

Calculated Explosive Costs Based on Powder Factor (Flitch Height, L=5)

Blasthole Diameter		ANFO (at $1.48/kg)		FORTIS (at $2.35/kg)		Powerfrag (@$3.08/kg)		
mm	inches	PF	$/tonne	PF	$/tonne	PF	$/tonne	
76	3.0	0.364	0.20	0.504	0.45	0.384	0.45	
89	3.5	0.446	0.25	0.496	0.44	0.383	0.45	ore
89	3.5	0.420	0.24	0.347	0.31	0.301	0.35	waste
102	4.0	0.439	0.25	0.483	0.43	0.458	0.54	
115	4.5	0.429	0.24	0.466	0.42	0.449	0.53	
127	5.0	0.416	0.23	0.448	0.40	0.396	0.47	

E. Drill and Blast Design – Price of Three (3) Major Explosive Products

Exlosive Products		FOB	Freight	Mixing	Fuel Oil	Others	Total Cost
Name	Type	(by Orica)	20%	(mmu)		30%	
		$/kg	$/kg	$/kg	$/kg	$/kg	$/kg
PPAN	Porous Prilled AN	0.77	0.15			0.23	1.16
ANFO	Mixed	0.77	0.15	0.25	0.07	0.23	1.48
FORTIS	Bulk Emulsion	1.40	0.28	0.25		0.42	2.35
POWER FRAG	Packaged Emulsion	2.05	0.41			0.62	3.08
Others – include cost of permits, storage/magazine, security and land transport							

F. Drill and Blast Design – Powder Factor Relative to the Desired Fragmentation

Kutznetsov Formula (1973) and Kuz-Ram Fragmentation Model

Where; **Xm** - Desired Fragmentation in cm
A - Rock Factor = 8.5 from Table 7.11
Q - Mass of explosive in the hole = BCM x PF
WS - Weight Strength relative to ANFO
K-0.8 =(Xm/(A*(Q1/6)*(115/RWS(19/20)))
K - Powder Factor relative to desired fragmentation, kg/m3
K = K-0.8^(1/-0.8)

Powder Factor K, Relative to the Desired Fragmentation Xm - Using ANFO

Blasthole Diameter	BCM V=BxSxL	PF in PC	Xm Desired	Kutznetsov Formula (1973) & Ram-Kuz Model				
				A	Q	RWS	$K^{-0.8}$	K
mm	m^3	kg/m^3	X_m in cm		kg			kg/m^3
76	80.90	0.407	50	8.5	32.95	100.00	2.88	0.27
89	103.88	0.425	50	8.5	44.12	100.00	2.74	0.28
89	116.74	0.377	50	8.5	43.96	100.00	2.74	0.28
102	129.96	0.435	50	8.5	56.56	100.00	2.63	0.30
115	158.84	0.443	50	8.5	70.29	100.00	2.54	0.31
127	190.54	0.447	50	8.5	85.09	100.00	2.46	0.33
76	80.90	0.407	40	8.5	32.95	100.00	2.30	0.35
89	103.88	0.425	40	8.5	44.12	100.00	2.19	0.37
89	116.74	0.377	40	8.5	43.96	100.00	2.19	0.37
102	129.96	0.435	40	8.5	56.56	100.00	2.10	0.39
115	158.84	0.443	40	8.5	70.29	100.00	2.03	0.41
127	190.54	0.447	40	8.5	85.09	100.00	1.96	0.43
76	80.90	0.407	30	8.5	32.95	100.00	1.73	0.51
89	103.88	0.425	30	8.5	44.12	100.00	1.64	0.54
89	116.74	0.377	30	8.5	43.96	100.00	1.65	0.54
102	129.96	0.435	30	8.5	56.56	100.00	1.58	0.57
115	158.84	0.443	30	8.5	70.29	100.00	1.52	0.59
127	190.54	0.447	30	8.5	85.09	100.00	1.47	0.62

Powder Factor K, Relative to the Desired Fragmentation Xm - Using Fortis Emulsion

Blasthole Diameter	BCM V=BxSxL	PF in PC	Xm Desired	Kutznetsov Formula (1973) & Ram-Kuz Model				
				A	Q	RWS	$K^{-0.8}$	K
mm	m^3	kg/m^3	X_m in cm		kg			kg/m^3
76	98.43	0.475	50	8.5	46.76	103.0	2.79	0.28
89	127.37	0.490	50	8.5	62.37	103.0	2.66	0.29
89	151.61	0.409	50	8.5	62.02	103.0	2.66	0.29
102	159.90	0.499	50	8.5	79.79	103.0	2.55	0.31

Blasthole Diameter	BCM V=BxSxL	PF in PC	Xm Desired	Kutznetsov Formula (1973) & Ram-Kuz Model				
				A	Q	RWS	$K^{-0.8}$	K
mm	m^3	kg/m^3	X_m in cm		kg			kg/m^3
115	196.01	0.504	50	8.5	98.71	103.0	2.46	0.32
127	235.71	0.506	50	8.5	119.31	103.0	2.39	0.34
76	98.43	0.475	40	8.5	46.76	103.0	2.23	0.37
89	127.37	0.490	40	8.5	62.37	103.0	2.13	0.39
89	151.61	0.409	40	8.5	62.02	103.0	2.13	0.39
102	159.90	0.499	40	8.5	79.79	103.0	2.04	0.41
115	196.01	0.504	40	8.5	98.71	103.0	1.97	0.43
127	235.71	0.506	40	8.5	119.31	103.0	1.91	0.45
76	98.43	0.475	30	8.5	46.76	103.0	1.67	0.52
89	127.37	0.490	30	8.5	62.37	103.0	1.60	0.56
89	151.61	0.409	30	8.5	62.02	103.0	1.60	0.56
102	159.90	0.499	30	8.5	79.79	103.0	1.53	0.59
115	196.01	0.504	30	8.5	98.71	103.0	1.48	0.61
127	235.71	0.506	30	8.5	119.31	103.0	1.43	0.64

Powder Factor K, Relative to Desired Fragmentation Xm – For Powerfrag Emulsion

Blasthole Diameter	BCM V=BxSxL	PF in PC	Xm Desired	Kutznetsov Formula (1973) & Ram-Kuz Model				
				A	Q	RWS	$K^{-0.8}$	K
mm	m^3	kg/m^3	X_m in cm		kg			kg/m^3
76	94.45	0.359	50	8.5	33.92	121.0	3.43	0.21
89	127.01	0.379	50	8.5	48.09	121.0	3.24	0.23
89	153.31	0.308	50	8.5	47.28	121.0	3.25	0.23
102	170.96	0.481	50	8.5	82.16	121.0	2.96	0.26
115	199.06	0.487	50	8.5	96.86	121.0	2.88	0.27
127	237.45	0.449	50	8.5	106.53	121.0	2.84	0.27
76	94.45	0.359	40	8.5	33.92	121.0	2.74	0.28
89	127.01	0.379	40	8.5	48.09	121.0	2.59	0.30
89	153.31	0.308	40	8.5	47.28	121.0	2.60	0.30
102	170.96	0.481	40	8.5	82.16	121.0	2.37	0.34
115	199.06	0.487	40	8.5	96.86	121.0	2.30	0.35
127	237.45	0.449	40	8.5	106.53	121.0	2.27	0.36
76	94.45	0.359	30	8.5	33.92	121.0	2.06	0.41
89	127.01	0.379	30	8.5	48.09	121.0	1.94	0.44
89	153.31	0.308	30	8.5	47.28	121.0	1.95	0.43
102	170.96	0.481	30	8.5	82.16	121.0	1.78	0.49
115	199.06	0.487	30	8.5	96.86	121.0	1.73	0.50
127	237.45	0.449	30	8.5	106.53	121.0	1.70	0.51

G. References and Specifications

Specific Gravity by Nominal Rock Classification

Rock	SPECIFIC GRAVITY		
Classification	Min	Max	Jugan
Basalt	1.8	3.0	
Dibase	2.6	3.0	
Diorite	2.8	3.0	
Dolomite	2.8	2.9	
Gneiss	2.6	2.9	
Granite	2.6	2.9	
Gypsum	2.3	2.8	
Hematite	4.5	5.3	
Limestone	2.4	2.9	
Marble	2.1	2.9	
Quatzite	2.0	2.8	
Sandstone	2.0	2.8	
SHALE	2.4	2.8	2.62
Slate	2.5	2.8	
Trap Rock	2.6	3.0	

Correction Factors for Burden Distance

Rock Deposition	Kd	
Bedding steeply dipping into cut	1.18	
Bedding steeply dipping into face	0.95	
Other cases of deposition	1	
Rock Structure	**Ks**	**RMR**
Heavily cracked, frequent weak joints, weakly cemented layers	1.3	POOR
Thin, well-cemented layers with joints	1.1	FAIR to good
Massive intact rocks	0.95	very good & >

Stiffness Ratio's Effect on Blasting

Stiffness Ratio (RS)	1	2	3	4 & >4
Fragmentation	POOR	FAIR	GOOD	EXCELLENT
Air Blast	SEVERE	FAIR	GOOD	EXCELLENT
Flyrock	SEVERE	FAIR	GOOD	EXCELLENT
Ground Vibration	SEVERE	FAIR	GOOD	EXCELLENT

Blasthole Size/Diameter

Rock Type	Diameter	Diameter	Drill Type	Machine Option
	mm	inches		Sandvik or CAT
SHALE (Jugan)	76	3.0	Top Hammer	DX800 or MD5075
SHALE (Jugan)	89	3.5	Top Hammer	DX800 or MD5075
SHALE (Jugan)	102	4.0	Top Hammer	DX800 or MD5075

Rock Type	Diameter	Diameter	Drill Type	Machine Option
	mm	inches		Sandvik or CAT
SHALE (Jugan)	115	4.5	Top Hammer	DX800 or MD5075
SHALE (Jugan)	127	5.0	Rotary	Sandvik DR Series
If problem of collapsing hole persists, consider 127mm drill hole by Rotary Drill				

Technical Data of ANFO

Explosive Property	**ANFO**
Density (g/cm^3)	**0.82**
Minimum Blasthole Diameter (mm)	**76**
Maximum Blasthole Depth(m)	**80**
Maximum Charge Length (m)	**75**
Diameter of explosive (De) = Diameter of Blasthole	
Hole Type	**DRY**
Delivery System	**augured/blowloaded**
Recommended booster for 76 – 102mm hole dia	**Pentex H**
Recommended booster for >102mm hole dia	**Pentex PPP**
Typical VOD (km/s)	**2.5 - 4.8**
Relative Effective Energy (REE)(3)	
Relative Weight Energy	**100**
Relative Bulk Strength	**100**
CO2 Output (kg/tonne)	**182**
Sleep Time	**42 days**

Technical Data of FORTIS (Bulk) Explosive - usually mixed at site

Explosive Property	**Fortis™ Advantage System**			
Density (g/cm3)	1.10	1.15	1.20	1.25
Minimum Blasthole Dia. (mm)	89	89	89	89
Maximum Blasthole Depth (m)	30	30	30	30
Maximum Charge Length (m)	25	25	25	25
Hole Type	Dry, Wet or Dewatered			
Delivery System	Pumped			
Booster for min hole diameter	Pentex H	Pentex H	Pentex H	Pentex H
Typical VOD (km/s)-Fortis S	3.7-5.9	3.7-6.1	3.7-6.3	3.7-6.5
Relative Effective Energy (REE)				
Relative Weight Strength	97	100	104	107
Relative Bulk Strength	133	144	156	167
CO2 output (kg/tonne)	137	145	133	133
Typical VOD (km/s) - **Fortis**	3.7-5.9	3.7-6.1	3.7-6.3	3.7-6.5
Relative Effective Energy (REE)				

Explosive Property	Fortis™ Advantage System			
Relative Weight Strength	100	103	107	110
Relative Bulk Strength	137	148	160	172
CO2 output (kg/tonne)	142	140	136	135
Fortis™ Advantage H				
Typical VOD (km/s) - Fortis H	3.7-5.9	3.7-6.1	3.7-6.3	3.7-6.5
Relative Effective Energy (REE)				
Relative Weight Strength	103	107	110	113
Relative Bulk Strength	142	154	165	177
CO2 output (kg/tonne)	154	151	149	148
Sleep Time	21 days			

Technical Data of Senatel TM POWERFRAG (packaged emulsion explosive)

Explosive Property		Nominal	Nominal	Nominal
Senatel POWERFRAG	Diameter	Density g/cc	Length (mm)	Mass (g)
Powerfrag (65mm)	65	1.21	300	1175
Powerfrag (80mm)	80	1.21	400	2275
PowerPac		1.18	250	1000
	PowerFrag		PowerPac	
Relative Effective Energy	REE		REE	
Relative Weight Strength	121		111	
Relative Bulk Strength				
to ANFO @ 0.82g/cc	183		164	
to ANFO @ 0.95g/cc	139		124	
Min. Velocity of Detonation	3.4km/s		5.5	
CO_2[3]	184 kg/t		181	

Explosive Loading Density Chart (Given: Explosive Specific Gravity)

Column Diameter Note: For ANFO & Emulsion, column dia = blasthole dia	Explosive Specific Gravity (ANFO & FORTIS) 0.80	1.10	1.15	1.20	1.25
	Density	Density	Density	Density	Density
3.0"	2.45	3.37	3.525	3.68	3.83
76mm	3.65	5.01	5.25	5.48	5.70
3.5"	3.34	4.59	4.8	5.01	5.215
89mm	4.97	6.83	7.14	7.45	7.76
4.0"	4.36	6	6.27	6.54	6.81
102mm	6.49	8.93	9.33	9.73	10.13
4.5"	5.52	7.58	7.925	8.27	8.615
115mm	8.21	11.28	11.79	12.31	12.82
5.0"	6.81	9.36	9.79	10.22	10.645
127mm	10.13	13.93	14.57	15.21	15.84

Explosive Loading Density Chart (Given: Explosive Specific Gravity)

Column Diameter Note: For Packaged Emulsion	**Explosive Specific Gravity (Powerfrag, Powerpac)**				
	0.80	1.10	1.15	1.20	
Explosive diameter in mm				Density	
2.56"				2.55	lb/ft
65				3.79	kg/m
3.15"				3.70	lb/ft
80				5.51	kg/m
3.94"				6.50	lb/ft
100				9.67	kg/m
4.33"				7.79	lb/ft
110				11.59	kg/m
4.75"				8.75	lb/ft
121				13.02	kg/m

Ratings for the Blastability Index Parameters (after Lilly (1986))

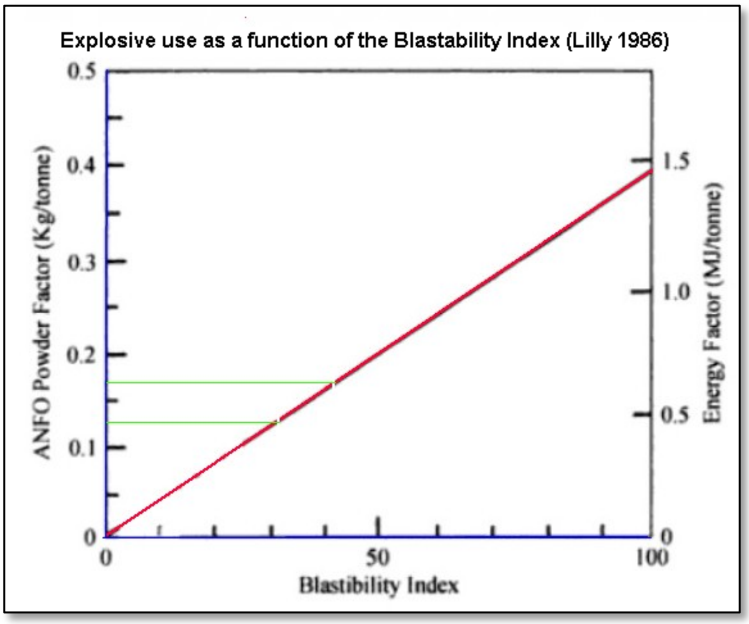
Parameter			**Rating**
1. Rock mass description (RMD)			
	1.1	Powdery/Friable	10
	1.2	Blocky	20
	1.3	Totally Massive	50
2. Joint Plane Spacing (JPS)			
	2.1	Close (<0.1 m)	10
	2.2	Intermediate ().1 to 1.0 m)	20
	2.3	Wide (>1.0 m)	50
3. Joint Plane Orientation (JPO)			
	3.1	Horizontal	10
	3.2	Dip out of face	20
	3.3	Strike normal to face	30
	3.4	Dip into face	40
4. Specific Gravity Influence (SGI)			
	SGI =	(25xSG)-50, wwhere SG is specific gravity of rock	
5. Hardness (H) = 3		H value from Moh's hardness scale (max for shale)	

Rock Factor per Rock Type

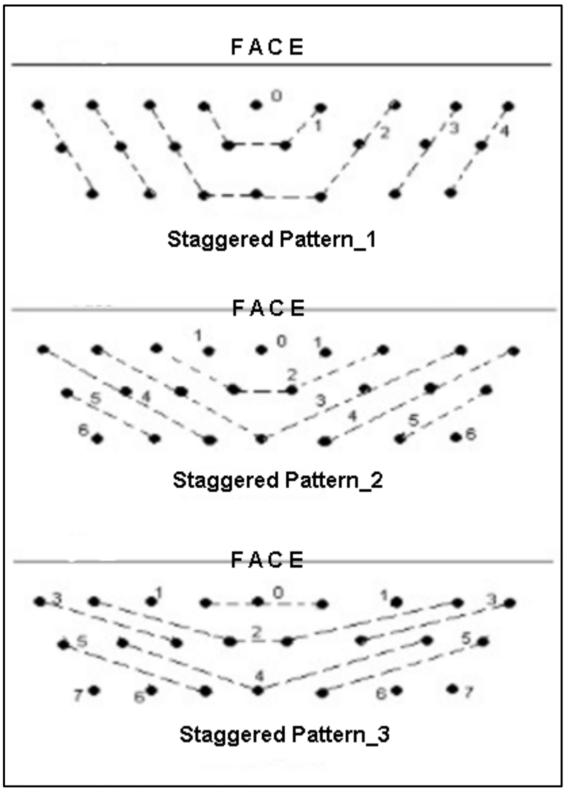
Table 1: Guide to powder factors and rock factors for various rock types

General Category	Rock type	Powder factor (kg/m^3)	Rock factor A
Hard (+200)	Andesite Dolerite Granite Ironstone Silcrete	0.70	12 -14
Medium (100 – 200)	Dolomite Hornfels Quartzite Serpentinite Schist	0.45	10 -11
Soft (50 – 100)	Sandstone Calcrete Limestone Shale	0.30	8 - 9
Very soft (-50)	Coal	0.15 – 0.25	6

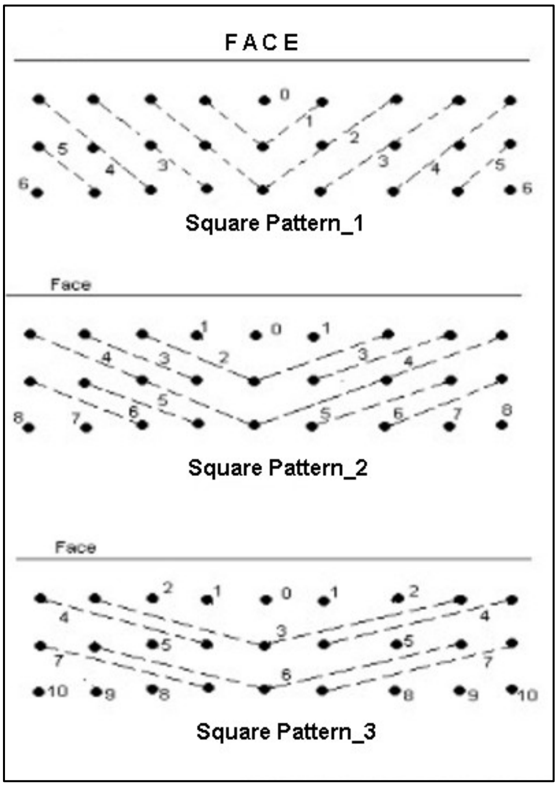
Blastability Index Graph



24-Holes Staggered Drilling Pattern



27-Holes Square Drilling Pattern



A21-1. Calculation Parameters – CAPEX/OPEX Costing

Unit/Factor Description	Value	Units
Mining / Milling Days	365	days
Mining / Milling Days per Quarter	91.25	days
Mining Rate_Base Case	8000	tonnes/day
Density, Ore	2.62	t/m^3
Density, Waste	2.60	t/m^3
Stripping Ratio (average)	2.3	W:O
Cut-off Grade (mining)		g/t
Mining Call Factor	100%	
Pit Optimization: Pit Slope Angle	from RMR model	
Mining Dilution_Jugan Pit	5%	
Mining Dilution_Bukit Young Pit	5%	
Metallurgical Recovery (Ave: Flotation & CIL)	**85%**	
Gold Price Used	**$ 1,500**	US$/Oz
Payable Gold (Sales)	99%	
Refining Charges	2.50	US$/Oz
Freight/Shipment Cost	4.50	US$/Oz
Royalties	0.00	
Income Tax	-	
Depreciation Rate	8%	
No. of shift/mining crew	3	
No. of Shift for hauling on public road	2	
Additional Consumable Usage Factor	1.2	
General Spares Factor	0.2	
US$-RM Rate-of-Exchange	0.3282	
US$-A$ Rate-of-Exchange	1.0319	
US$-**€** Rate-of-Exchange	1.2940	
Contingency (Capex and Opex)	**0%**	
Annual Cost Escalation	0%	
Equipment Resale %	25%	
Equipment Resale Value		
Resale Value	$	
International Freight Dbn-Malaysia - 20'	800.00	US$
International Freight Dbn-Malaysia - 40'	1,050.00	US$
Local Freight & Customs - 20'	1,450.00	RM
Local Freight & Customs - 40'	1,950.00	RM

Unit/Factor Description	Value	Units
Fuel Price_Subsidised	$ 0.66	per liter
Fuel_No subsidy	$ 0.90	per liter
Power Cost	$ 0.068	per kWhr
Operator & Maintenance Labour	$ 3.50	per hour
Labour based on machine hours	$ 4.375	per UT hour
Labour based on maintenance hours	$ 4.32	per MT hour

A21-2. Equipment Data Parameters – CAPEX/OPEX Costing

Unit/Factor Description	Value	Units
Equipment Data Inputs		
Total Mobile Machinery (for 8000 tpd)	40	units
Capital Replacement Factor	15%	
Average Equipment Capacity	90%	
Net Fleet Efficiency	90.5%	
Operating Cost Allocation (+5% losses)	90.0%	
Allowance for extra equipment capacity	10%	
Other Equipment		
Road Grader	US $ 80	US$ /UT
Water Truck	US $ 90	US$ /UT
Service/Tyre Truck	US $ 80	US$ /UT
Explosive (Bulk) Truck	US $ 90	US$ /UT
Compactor	US $ 80	US$ /UT
Complete Mining Equipment - For Base Case, 8000 TPD		
Loader (Shovel/Excavator)		
No. of Units for Basecase (CAT_6015FS)	2	units
Capacity	7	m3
Unit Cost	$ 1,476,765	US$
Material Cost	60	US$/UT
Fuel Rate	55	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time (machine hours) from Table 5	5,920	UT hours
Maintenance Time	2,103	hours
Annual Production/Unit	5,370,000	tonnes
Rigid Dump Truck for Ore		
No. of Units for Basecase (CAT_772G)	4	units
Capacity	30	m3
Unit Cost	$ 662,903	US$
Material Cost	30	US$/UT
Fuel Rate	42	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time	5,906	UT hours
Maintenance Time	2,237	hours
Average Hauling Distance	1,600.0	meters
Annual Production/Unit	1,044,167	tonnes
Rigid Dump Truck for Waste		

Unit/Factor Description	Value	Units
No. of Units for Basecase (CAT_772G)	5	units
Capacity	30	m3
Unit Cost	$ 662,903	US$
Material Cost	30	US$/UT
Fuel Rate	42	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time	5,906	UT hours
Maintenance Time	2,237	hours
Average Hauling Distance	1,200.0	meters
Annual Production/Unit	1,392,222	tonnes
Dozer/Ripper CAT_D10		
No. of Units for Basecase (CAT_772G)	1	unit
Capacity	18	m3
Unit Cost	$ 1,670,385	US$
Material Cost	62	US$/UT
Fuel Rate	56	litres
Replace Hours	60000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time	5,838	UT hours
Maintenance Time	2,151	hours
Production/Unit	9,636,000	tonnes
Tractor D6R		
No. of Units for Basecase (CAT_D6R)	2	units
Capacity	6	m3
Unit Cost	$ 274,022	US$
Material Cost	21	US$/UT
Fuel Rate	20	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time	4800	UT hours
Maintenance Time	2500	hours
Production/Unit		tonnes
Production Drill		
No. of Units for Basecase (Sandvik DX800)	2	units
Hole Dia. min 76mm, max 127mm	89	mm
Unit Cost	$ 565,920	US$
Material Cost	60	US$/UT
Fuel Rate	46	litres
Replace Hours	50000	hours
MT/AMT	1.23	

Unit/Factor Description	Value	Units
Ratio: AT/UT	1.25	
Utilised Time	5,920	UT hours
Maintenance Time	2,103	hours
Production/Unit	5,370,000	tonnes
Road Grader	2	units
Blade width	2	m
Unit Cost	$ 308,480	US$
Material Cost	15	US$/UT
Fuel Rate	20	litres
Replace Hours	40000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time	4,800	UT hours
Maintenance Time	2,193	hours
Wheel Loader	2	units
Capacity	6.4	m3
Unit Cost	$ 820,425	US$
Material Cost	59	US$/UT
Fuel Rate	54	litres
Replace Hours	50000	hours
MT/AMT	1.23	
Ratio: AT/UT	1.25	
Utilised Time		UT hours
Maintenance Time		
Compactor	2	units
Capacity		m3
Unit Cost	$ 95,169	US$
Fuel Rate		litres
Replace Hours	4800	hours
Utilised Time		UT hours
Maintenance Time		
Water Truck	2	units
Capacity	10	m3
Unit Cost	$ 88,606	US$
Fuel Rate		litres
Replace Hours		hours
Utilised Time	1650	UT hours
Service/Tyre Truck	2	units
Capacity		m3

Unit/Factor Description	Value	Units
Unit Cost	$ 90,000	US$
Fuel Rate		litres
Replace Hours	5595	hours
Utilised Time	4380	UT hours
Cable Bolter (identical with production drill)	1	unit
Capacity		m3
Unit Cost	$ 565,920	US$
Fuel Rate		litres
Replace Hours		hours
Utilised Time		UT hours
Explosive Truck	2	units
Capacity		m3
Unit Cost	$ 50,000	US$
Fuel Rate		litres
Replace Hours		hours
Utilised Time	3105	UT hours
Maintenance Time		
Arsenic Smelting Penalty	$ 2.00	/per 0.1% > 0.2%
Aluminium Smelting Penalty	$ 1.25	/per 1% > 3%
Au Feed - Au Concentrate Ratio	7.6	

A21-3. Mining Labour Costing – Direct & Indirect

Labour Cost Item Description	Qty		Unit Cost (MYR)		Unit Cost (US$/mth)		Total Cost (US$/mth)
Direct Labour (pit operations) - costing included in OPEX1_Mining							
Equipment Operators	74	staff	RM 3,500	/mth	$ 1,148.60	/mth	$84,996.03
Shop Mechanics	10	staff	RM 3,500	/mth	$ 1,148.60	/mth	$11,485.95
Service Mechanics	4	staff	RM 3,500	/mth	$ 1,148.60	/mth	$4,594.38
Shop Electrician	4	staff	RM 3,500	/mth	$ 1,148.60	/mth	$4,594.38
Service Electrician	3	staff	RM 3,500	/mth	$ 1,148.60	/mth	$3,445.79
Helper/Utility	12	staff	RM 2,000	/mth	$ 656.34	/mth	$7,876.08
Direct Labour	**107**						**$116,992.61**
Manager & Supervision Staff Labour:							
Mine Manager Expat	1	staff			$20,000.00	/mth	$20,000.00
Mine Shift Foreman	3	staff	RM 20,000	/mth	$6,563.40	/mth	$19,690.20
Planning Engineer	1	staff	RM 18,000	/mth	$5,907.06	/mth	$5,907.06
Shift Supervisor	6	staff	RM 15,000	/mth	$4,922.55	/mth	$29,535.30
Pit Geologist	2	staff	RM 15,000	/mth	$4,922.55	/mth	$9,845.10
Resource/Reserve Geologist	1	staff	RM 18,000	/mth	$5,907.06	/mth	$5,907.06
Geotech Engineer	1	staff	RM 18,000	/mth	$5,907.06	/mth	$5,907.06
Chief Surveyor	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Safety Manager	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Safety Supervisor	3	staff	RM 5,000	/mth	$1,640.85	/mth	$4,922.55
Fleet Maintenance Manager	1	staff	RM 15,000	/mth	$4,922.55	/mth	$4,922.55
Mechanical Engineer	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Maintenance Supervisor	3	staff	RM 5,000	/mth	$1,640.85	/mth	$4,922.55
Maintenance Planner	1	staff	RM 6,000	/mth	$1,969.02	/mth	$1,969.02
Electrical Engineer	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Electrical Supervisor (maint)	3	staff	RM 5,000	/mth	$1,640.85	/mth	$4,922.55
Warehouse Manager	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Warehouse Supervisor	2	staff	RM 5,000	/mth	$1,640.85	/mth	$3,281.70
Environment Engineer	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Tailings Dam Manager	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Supervisor (tailings dam)	3	staff	RM 5,000	/mth	$1,640.85	/mth	$4,922.55
Mine Overhead Labour	**38**						**$149,627.15**
Mine Service Department							
Safety Officer/Trainer	2	staff	RM 5,000	/mth	$1,640.85	/mth	$3,281.70
Mine Clerk/Statisticians	2	staff	RM 5,000	/mth	$1,640.85	/mth	$3,281.70
Grade Control Technician	3	staff	RM 2,700	/mth	$886.06	/mth	$2,658.18
Samplers	6	staff	RM 2,700	/mth	$886.06	/mth	$5,316.35
Surveyor	1	staff	RM 5,000	/mth	$1,640.85	/mth	$1,640.85
Survey crew	4	staff	RM 2,700	/mth	$886.06	/mth	$3,544.24

Labour Cost Item Description	Qty		Unit Cost (MYR)		Unit Cost (US$/mth)		Total Cost (US$/mth)
Geotech crew	2	staff	RM 2,700	/mth	$886.06	/mth	$1,772.12
Security manager	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Security guards	12	staff	RM 2,700	/mth	$886.06	/mth	$10,632.71
Mine Services Labour	**33**						**$35,409.55**
Engineering Services							
Engineering Manager	1	staff			$15,000.00	/mth	$15,000.00
Civil Engineer	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Mechanical Engineer	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Electirlcal Engineer	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Engineering Labour	**4**						**$24,845.10**
Admin, PR & HR							
Mine Admin Manager	1	staff			$15,000.00	/mth	$15,000.00
HR Manager	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
PR Manager	1	staff	RM 10,000	/mth	$3,281.70	/mth	$3,281.70
Office Personnel	9	staff	RM 2,500	/mth	$820.43	/mth	$7,383.83
Admin Labour	**12**						**$28,947.23**
Procurement, Accounting & Finance and ICT							
Procurement Manager	1	staff			$15,000.00	/mth	$15,000.00
Procurement Staff/ Buyer	3	staff	RM 3,000	/mth	$984.51	/mth	$2,953.53
Finance Mgr/Comptroller	1	staff	RM 12,000	/mth	$3,938.04	/mth	$3,938.04
Accountant	1	staff	RM 6,000	/mth	$1,969.02	/mth	$1,969.02
Cashier	1	staff	RM 3,000	/mth	$984.51	/mth	$984.51
Accounting Staff	2	staff	RM 3,000	/mth	$984.51	/mth	$1,969.02
IT Manager	1	staff	RM 12,000	/mth	$3,938.04	/mth	$3,938.04
IT Technician	2	staff	RM 4,000	/mth	$1,312.68	/mth	$2,625.36
PAFI Labour	**12**						**$33,377.52**
Tailings Dam Labour:							
Tailings Dam Crew	6	staff	RM 2,500	/mth	$820.43	/mth	$4,922.55
Total Labour Costs:							**$277,129.10**
Labour_Staff Onsite Costs	15	%					$41,569
Labour_Travel & Accommodation	15	%					$41,569
Contractual Expats/Consultants							$50,000
Grand Total Labour/Overhead							**$410,267**

Labour Cost Item Description	Qty		Unit Cost (MYR)		Unit Cost (US$/mth)		Total Cost (US$/mth)
Total Annual Labour Costs:							$4,923,205
Personnel with PPEs	***188***						
Labour Cost per tonne (for MCAF)							**$ 0.62**

A21-4. Mine Engineering Services Costing

Engineering Cost Item Description	Qty		Unit Cost (US$)		Total Cost (US$)	Cost/Tonne (US$/t)
Services:						
Water Pipe - Service Water	2,400	m	$ 12.15	/m	$ 29,160.00	$ 0.010
HDPE Pipe - pit dewatering pipes	2,400	m	$ 12.15	/m	$ 29,160.00	$ 0.010
Water Pipe Clamps	960	units	$ 8.67	each	$ 8,323.20	$ 0.003
Water Pipe - Bends	10	units	$ 10.67	each	$ 106.70	$ 0.000
Water Pipe - Valves	5	units	$ 83.33	each	$ 416.65	$ 0.000
LT Equipment	4	units	$ 1,925	/unit	$ 7,700.00	$ 0.003
LT Equipment - Frames	4	units	$ 226.36	/unit	$ 905.44	$ 0.000
Pipe Support	1,200	m	$ 20.87	/m	$ 25,044.00	$ 0.009
Electric Cable - 70mm XLPE	2,000	m	$ 19.83	/m	$ 39,660.00	$ 0.014
Elec Cable - 70mm XLPE 1000/600V		m	$ 260.87	/m	$	$ -
Electric Cable - 4C/16mm	2,000	m	$ 2.96	/m	$ 5,920.00	$ 0.002
Luminaires	500	units	$ 2.87	each	$ 1,435.00	$ 0.000
Bulbs	500	units	$ 0.52	each	$ 260.00	$ 0.000
Dewatering pump consumables	1.0	lot	$ 8,000	/lot	$ 8,000.00	$ 0.003
Total Services:					***$ 156,090.99***	***$ 0.05***
Electricity						
Workshop & equipment (70kW)	430,080	kWhr	$ 0.068		$ 29,245.44	$ 0.010
Mobile Light Plant for pit (6 x 13kW)	239,600	kWhr	$ 0.068		$ 16,292.80	$ 0.006
Offices & accommodation (20kW)	122,880	kWhr	$ 0.068		$ 8,355.84	$ 0.003
Electricity for pumps (in opex1)						
Total Electricity	***53.76***				***$ 53,894.08***	***$ 0.02***
Sundries						
Potable Water	3,200	m^3	$ 0.49	/m3	$ 1,575.22	$ 0.001
Water for Workshop	4,800	m^3	$ 0.49	/m3	$ 2,362.82	$ 0.001
Cleaners - Degreasing	24	mths	$ 2,088.00	/mth	$ 50,112.00	$ 0.017
Total Sundries:					***$ 54,050.04***	***$ 0.02***
TOTAL ENGINEERING COSTS					**$ 264,035.11**	**$ 0.09**

A21-5. Technical Services, Health & Safety and Sundry Mining Costing

Cost Item Description	Qty		Unit Cost (MYR)		Unit Cost (US$)		Total Cost (US$)	Cost/Tonne (US$/t)
Health & Safety:								
Boots	451	units	RM 110.00	/pair	$ 36.10	/pair	$ 16,287.73	$ 0.004
Hard Hats	451	units	RM 25.00	/unit	$ 8.20	/unit	$ 3,701.76	$ 0.001
Overalls	451	units	RM 43.00	/unit	$ 14.11	/unit	$ 6,367.02	$ 0.002
Gloves	451	units	RM 0.65	/pair	$ 0.21	/pair	$ 96.25	$ 0.000
Belts	451	units	RM 180.00	/unit	$ 59.07	/unit	$ 26,652.65	$ 0.007
Ear Muffs	451	units	RM 45.00	/unit	$ 14.77	/unit	$ 6,663.16	$ 0.002
Glasses	451	units	RM 28.00	/unit	$ 9.19	/unit	$ 4,145.97	$ 0.001
First Aid Materials	78	units	RM 200.00	/unit	$ 65.63	/unit	$ 5,119.45	$ 0.001
Reflector Jackets	451	units	RM 40.00	/unit	$ 13.13	/unit	$ 5,922.81	$ 0.002
Danger Tape	320	/20m	RM 19.50	/100m	$ 6.40	/100m	$ 2,047.78	$ 0.001
Hand Torches	76	units	RM 45.00	/unit	$ 14.77	/unit	$ 1,122.34	$ 0.000
Safety Signage	200	units	RM 100.00	/unit	$ 32.82	/unit	$ 6,563.40	$ 0.002
Total Health & Safety:							***$ 84,690.33***	***$ 0.02***
Mining Services:								
Sampling Materials								
Sample Bags	19,246	bag	RM 2.50	/bag	$ 0.82	/bag	$ 15,789.57	$ 0.005
Hammers	12.0	unit	RM 15.00	/unit	$ 4.92	/unit	$ 59.07	$ 0.000
Spray Paint	600.0	can	RM 8.00	/can	$ 2.63	/can	$ 1,575.22	$ 0.001
Measuring Tape	12.0	tape	RM 6.00	/tape	$ 1.97	/tape	$ 23.63	$ 0.000
Survey Materials								$ -
Survey Pegs	800	each	RM 4.80	each	$ 1.58	each	$ 1,260.17	$ 0.000
Spray Paint	600.0	can	RM 8.00	/can	$ 2.63	/can	$ 1,575.22	$ 0.001
Measuring Tape	12.0	tape	RM 6.00	/tape	$ 1.97	/tape	$ 23.63	$ 0.000
Geology Materials								$ -
Sample Bags	1440.0	pcs	RM 2.50	/pcs	$ 0.82	/pcs	$ 1,181.41	$ 0.000
Geology Hammers	6.0	unit	RM 20.00	/unit	$ 6.56	/unit	$ 39.38	$ 0.000
Spray Paint	600.0	can	RM 8.00	/can	$ 2.63	/can	$ 1,575.22	$ 0.001
Measuring Tape	12.0	tape	RM 6.00	/tape	$ 1.97	/tape	$ 23.63	$ 0.000
Office Items/Supplies								$ -
Software Licenses/ Maintenance	3	sets		/set	$ 10,000	/set	$ 30,000.00	$ 0.010
Office Supplies	60	reams	RM 100.00	/ream	$ 32.82	/ream	$ 1,969.02	$ 0.001
Total Mining Services:							***$ 55,095.16***	***$ 0.02***
Sundries:								
Paint	60.0	litre	RM 20.00	/litre	$ 6.56	/litre	$ 393.80	$ 0.0001
Spray Paint	800	cans	RM 8.00	/can	$ 2.63	/can	$ 2,100.29	$ 0.0007
Measuring Tapes	60.0	tapes	RM 6.00	/tape	$ 1.97	/tape	$ 118.14	$ 0.0000
Hand Tools	120.0	pcs	RM 50.00	/pc	$ 16.41	/pcs	$ 1,969.02	$ 0.0007
Pad Locks	120.0	locks	RM 10.00	/lock	$ 3.28	/lock	$ 393.80	$ 0.0001
Shovels & Picks	120.0	units	RM 18.00	/unit	$ 5.91	/unit	$ 708.85	$ 0.0002

Cost Item Description	Qty		Unit Cost (MYR)		Unit Cost (US$)		Total Cost (US$)	Cost/Tonne (US$/t)
Hammers	120.0	units	RM 15.00	/unit	$ 4.92	/unit	$ 590.71	$ 0.0002
Heavy Duty Plastic	60.0	units	RM 600.00	/20m	$ 196.90	/20m	$ 11,814.12	$ 0.0040
Cement	2400.0	bags	RM 13.40	/25kg	$ 4.40	/25kg	$ 10,553.95	$ 0.0036
Nails, Nuts & Bolts	60.0	kgs	RM 11.20	/kg	$ 3.68	/kg	$ 220.53	$ 0.0001
Battery Fluid	2000	litre	RM 1.00	litre	$ 0.33	litre	$ 656.34	$ 0.0002
Oxygen	144	tank	RM 55.64	/10.7m³	$ 18.26	/10.7m³	$ 2,629.35	$ 0.0009
Acetylene	72.0	tank	RM 92.80	/6.4m³	$ 30.45	/6.4m³	$ 2,192.70	$ 0.0008
Washers	10	box	RM 200.00	/box	$ 65.63	/box	$ 656.34	$ 0.0002
Gaskets	10	box	RM 500.00	/box	$ 164.09	/box	$ 1,640.85	$ 0.0006
Total Sundries:							***$ 36,638.79***	***$ 0.01***
TOTAL GENERAL:							**$ 176,424.28**	**$ 0.05**

A21-6. Process Plant Equipment List

Item No	Op	S'by	Title	Inst. kW ea	Total. kW	VSD/ Fixed	Description	Manufacturer	Supplier	Model №	Total Cost
					14,128					TOTAL	$29,895,500
			CRUSHING MODULE								
01-BN-01	1		Primary crusher feeder Bin				Mild steel hopper with hardox liner,	Metso	Metso		$ 175,000
	1		Crusher main Feeder	30	30	Fixed	Length 6100 x W 1300,Max rock 900 mm,VF 561-2V	Metso	Metso	VF 561-2V	
	1		Apron Feeder Spillage Chute					Local	Local		$ 12,000
	1		Jaw Crusher	160	160	Fixed	C 125,Jaw portal-1250x950,Max rock 800mm,Cr capacity 670Ton/hr,Screen130-200mm	Metso	Metso	C 125	$ 490,000
			Main support structure crusher				Heay duty steel suuport structure with working platform	Local	Local		$ 162,000
	1		Hanging Electro Magnet	10	10	Fixed	1500X1450X1360 mm,Wt 3100kg,Exciting Power 8.4kw,Oil cooled,700Gs at 300mm gap Mag Intensity	Metso	Metso		$ 45,000
	1		Primary Crusher Maintenance Hoist				10t SWL Electric hoist-Over head crane	Demag,Germ/Viet	Demag		$ 58,000
	1		Conveyor Jaw CR to Stockpile	75	75	Fixed	90 meter,1400 mm wide, 1.2 M/s,1400x5x4x2 mm belt	Metso	Metso		$ 102,000
	1		Ore Bin Feed Conveyor from grizzly	30	30	Fixed	30 meter,1200mm width,1.2m/s,1200x5x4x2mm belt	Metso	Metso		$ 68,000
	1		Jaw Crusher Building				Pree fabricated building	PEB	PEB		$ 56,000
	1		Stockpile Ore Bin				Mild steel hopper with epoxy paint,size	Local	Local		$ 85,000
	1		Stockpile Building				Pree fabricated building	PEB	PEB		$ 58,000
	1	3	Ore Bin Tail Gate				Air actuated tail gate box	Metso	Metso		$ 62,000
	1	3	Vibratory small feeder	1.5	6		Vibratory feeder for tunnel feeder conveyor	Metso	Metso		$ 84,000
	1	3	Feeder Conveyor	4.0	16	VSD	2 meter,1000mm wide,1.0 m/s,1000x5x4x2 mm	Metso	Metso		$ 56,000
	1		Tunnel Conveyor	45	45	Fixed	35 meter,1400mm width,1.2m/s,1400x5x4x2mm belt	Metso	Metso		$ 52,000
			Conveyoor Supports				Schedule 40 pipe supports with suppoirt brackets	Local	Local		$ 75,000
	4		Belt Scrapper Tunnel CV	1.5	6.0	Fixed	Motor connected rotary brush	Metso	Metso		$ 32,000
	1		Weighto meter				Caliberated weightometer.				$ 52,000
	1		Dust Collector	75	75	Fixed	Hopper,blower with full control and connected pipings	Metso	Metso		$ 82,000
					453						$ 1,806,000
			GRINDING MODULE								
	1		SAG Mill Feed Conveyor	30	30	Fixed	29 meter,1400mm width,1.0 m/s,1400x5x4x2mm belt	Metso	Metso		$ 36,000
	1		Belt Scrapper SAG Mill Feed CV	1.5	1.5	Fixed	Motor connected rotary brush	Metso	Metso		$ 8,000
	1		SAG Mill Feed Retractive Chute				Mild steel chute with motorised diversion plate	Metso	Metso		$ 48,000
	1		SAG Mill - complete set	6200	6200	Fixed	8.54 m diameter x 4.87 m long,400 tph feed rate,	Metso	Metso		$ 7,230,000
	1		SAG Mill discharge coarse vibrating screen				Step deck,1.6x3.2M	Metso	Metso		$ 87,000
	1		Pebble Crusher	75	75		60 Tons/hr	Metso	Metso		$ 125,000
	1		Oversize CV to Pebble crusher	15	15		25 meter,800 x 5x 4x 2 conveyor	Metso	Metso		$ 35,000
	1		Gound return CV from Pebble crusher	15	15		16 meter ,800 width,5x4x2 conveyor	Metso	Metso		$ 25,000
	1		SAG Mill Discharge hopper				MD Steel, Epoxy coated, 18m3	Local	Local		$ 41,000
	1		SAG Mill Area Platform				Frame structure with coloums and cross beams.Top is fully secured with GI Gratings.	Local	Local		$ 325,000
	1		Media Loading system				Mechanised ball charger system	Metso	Metso		$ 68,000
	1		Liner handler				Hydraulic Liner handling system	Metso	Metso		$ 165,000
	1		Overhead Crane 15T				15t SWL Electric hoist-Over head crane	Demag,Malaysia	Demag,Malaysia		$ 74,000
					6337						$ 8,267,000
			GRAVITY/DESLIME MODULE								
	1	1	SAG Mill Discharge Pump	150	300	VSD	12/10 D-AH,Centrifugal metal liner pump with gland water,1600 rpmY 250M,6/37 KW,IP54	Warman	Warman		$ 105,000
	1	1	BM Discharge Pump,Gland water system					Local	Local		$ 1,200
	1	1	10" Tech Taylor Valve				10" Tech-Taylor Valve connected to SAG Mill discharge pumps.	Fluid smith,Australia	Fluid Smith		$ 48,000
	1	1	Pump Gland water system					Local	Local		$ 1,200
	1		Cyclone Cavex 4 X 500CV X10				Primary 500mm Cyclone for T/Hr,3 Operating,1 standby	Metso	Metso	Cavex 4x500CVx10	$ 132,000
	6		Control knifegate VALVE				Air actuated lina knife gate valves	Metso			
	1		Cyclone pressure gauge				150 kpa readable pressure gauge	Metso			
	1		Cyclone feed flow meter				10" Pipe Size slurry flow meter,Yokogawa	Yokogawa	Yokogawa		$ 17,500
	1		Cyclone feed Density gauge				10" Pipe size,Nuclear Density Gauge	Australia			$ 38,000
	1		300 T/Hr CVD 64 Knelson Concentrator	150	150	VSD	CVD 64 ,300T/Hr capacity,Cast Urethane G5 Inner Bowl (Desliming)	Knelson	Knelson		$ 475,000
	1		300 T/Hr CVD 64 Knelson Concentrator	150	150	VSD	CVD 64,300 T/Hr capacity,Cast Urethane G5 Inner Bowl (Desliming)	Knelson	Knelson		$ 475,000
	1		Gravity Sump Pump	22	22	VSD	4/4 Vertical sump pump,warman	Warman			$ 28,000
	1		Plant Air receiver				LV 1011L 11BAR CE, 1 M3 Capacity hot dipped air receiver	Atlaas Copco			$ 5,100
	1		Gravity equipment structure				Fabricated stell structure based on supplier drawing for cyclone and Knelson	Local	Local		$ 175,000
	1		BallMill/Gravity Building				Pree fabricated building	PEB Steel			$ 240,000
					622						$ 1,741,000

Item No	Op	S'by	Title	Inst. kW ea	Total. kW	VSD/ Fixed	Description	Manufacturer	Supplier	Model №	Total Cost
					14,128					TOTAL	$29,895,500
			FLOTATION[Rougher/Scavenger]MODULE								
	1		Flotation Condition Tank,100m3				4 m Dia x 5 m height,100 m3 open topped cylidrical mild steel epoxy painted	Metso	Metso		$ 170,000
	1		Rake assembly	90	90	Fixed	Axial flow type agitator	Metso	Mixtec		
	1		Flotation Cell,				RCS 200,8 Cells WITH AGITATOR [FB-Feed Box,PV-Pinch Valve]	Metso	Metso		$ 5,800,000
	8		Flotation cell drive unit Teco motor	200	1600	Fixed	8 Units of agitator on each cell	Metso			
	1		Flow level control Pinch valve					Metso	Metso		
	1	1	Rougher Concentrate pump metso VF250	45	90	Fixed	VF 250 Metso Sala,fixed speed pump	Metso			$ 92,000
	1	1	Scavenger Concentrate pump Metso VF200	30	60	Fixed	VF 200 Metso Sala,fixed speed pump	Metso			$ 88,000
	1		Flotation Tail Sump				15 m3 Open topped consep mild steel painted,inside rubber lined Hopper	Local	Local		$ 42,000
	1		Level Transmitter				Seimens Milltronics Ultrasonic level transmitter	Seimens	Seimens		$ 12,000
	1	1	Flotation Tails pump	150	300	VSD	10/8 Warman,150 kW,Slurry pump	Warman	Warman		$ 105,000
	1		Flotation Floor sump pump	22	22	VSD	4/4 vertical sump pump Warman	Warman			$ 28,000
	1		Flotation Tails Flow meter				10" Pipe Size slurry flow meter,Yokogawa	Yokogawa	Yokogawa		$ 17,500
	1		Floation Tails Desity gauge				10" Pipe size,Nuclear Density Gauge	Australia			$ 38,000
	1		Tech Taylor Valve				10" Tech-Taylor Valve connected to 2 Cyclone feed hopper lina pumps.	Fluid smith,Australia	Fluid Smith		$ 48,000
	5		Flotation Air Blower	200	1000	Fixed	Blower,200 Kw Positive displacement Blower with blow off valve and silencer	Denvar			
			Overflow launders and sumps				Rubber lined steel fabricated sumps and launders with epoxy cpaint coated	Local	Local		$ 46,000
	1		Support structure and working Platform				Steel structure as per supplier drawing with working platform and hand rail	Local	Local		$ 162,000
	1		Building				Pree fabricated building	PEB Steel	PEB		$ 250,000
			Overhead Crane 15T				15t SWL Electric hoist-Over head crane	Demag,Malaysia	Demag,Malaysia		$ 74,000
					3162						$ 6,972,500
			REGRIND MODULE								
	1		Regrind BM	1350	1350		3.6 Dia x 7 m Long Wet Over Flow Ball Mill	Metso	Metso		$ 1,970,000
	1		Regrind BM Slow speed Drive	15	15	Fixed		Metso	Metso		
	1		Regrind BM Air Clutch					Metso	Metso		
	4		Regrind BM Low/High Lube System	7.7	31	Fixed	Drawing	Metso	Metso		
	1		Regrind BM Spray Lube system	2.2	2	Fixed	Air spary and Oil sray pumps	Metso	Metso		
	1		Regrind BM Air Clutch Air receiver					Metso	Metso		
	1		Regrind BM Trommel Screen				Mild Steel circular with 8mm aperature SS mesh fitted	Metso	Metso		$ 52,000
	1		Regrind BM Trash Bin					Local	Local		$ 3,500
	1		Regrind BM Discharge Hopper				12 m3 Mild steel rectangular ,epoxy painted hopper	Local	Local		$ 41,000
	1		Regrind BM Steel Ball Charger Hopper				Circular Mild steel hopper with supporting stand and opening arrangements.	Metso	Metso		$ 65,000
			Regrind Floor sump pump	15	15	Fixed	3/3 vertical sump pump Warman	Warman			$ 23,000
	1		Regrind Ball Mill Platform/ladder				Frame structure with coloums and cross beams.Top is fully secured with GI Gratings.	Local	Local		$ 165,000
					1413						$ 2,319,500
			SECONDARY CYCLONE								
	1	1	Cyclone feed pump	75.0	150		6/4" Warman slurry pump	Warman	Warman		$ 85,000
	1		Cyclone feed flow meter				8" Pipe Size slurry flow meter,Yokogawa	Yokogawa	Yokogawa		$ 17,500
	1		Cyclone feed Density gauge				8" Pipe size,Nuclear Density Gauge	Australia			$ 38,000
	1		Tech Taylor Valve				8" Tech-Taylor Valve connected to 2 Cyclone feed hopper PUMPS	Fluid smith,Australia	Fluid Smith		$ 42,000
	1		Cyclone feed distribution box				Distribution box to 3 cyclone cluster	Metso	Metso		$ 49,500
	3		Cyclone Cavex 6 X 250CV X10				250mm Cyclone for T/Hr,5 Operating,1 standby	Metso	Metso	Cavex6x250 CVx10	$ 194,000
	1		Cyclone pressure gauge				150 kpa readable pressure gauge	Metso			
	1		Cyclone structure				Steel structure as per supplier drawing	Local	Local		$ 115,000
					150						$ 541,000

Item No	Op	S'by	Title	Inst. kW ea	Total. kW	VSD/ Fixed	Description	Manufacturer	Supplier	Model №	Total Cost
					14,128					TOTAL	$29,895,500
			CLEANER FLOTATION MODULE								
	1		Flotation Cell,				RCS 70,4 Cells WITH AGITATOR [FB-Feed Box,PV-Pinch Valve]	Metso	Metso		$ 1,290,000
	4		Flotation cell drive unit Teco motor	90	360	Fixed	4 Units of agitator on each cell	Metso			
	1		Flow level control Pinch valve					Metso	Metso		
	1	1	Cleaner Concentrate pump metso VF250	30	60	Fixed	VF 200 Metso Sala,fixed speed pump	Metso			$ 88,000
	1		Flotation Tail Sump				8 m3 Open topped consep mild steel painted,inside rubber lined Hopper	Local	Local		$ 24,000
	1		Level Transmitter				Seimens Milltronics Ultrasonic level transmitter	Seimens	Seimens		$ 12,000
	1	1	Flotation Tails pump	75	150	VSD	8/6 Warman,75 kW,Slurry pump	Warman	Warman		$ 85,000
	1		Flotation Floor sump pump	11	11	VSD	3/3 vertical sump pump Warman	Warman	Warman		$ 23,000
	1		Flotation Tails Flow meter				8" Pipe Size slurry flow meter,Yokogawa	Yokogawa	Yokogawa		$ 16,500
	1		Floation Tails Desity gauge				8" Pipe size,Nuclear Density Gauge	Australia			$ 38,000
	2		Flotation Air Blower				Blower,100 Kw Positive displacement Blower with blow off valve and silencer	Metso	Metso		
	1		Support structure				Steel support structure ,working platform and hand rail	Local	Local		$ 80,000
			Flotation/Reggrinding Building				Pree fabricated building	PEB Steel	PEB Steel		$ 190,000
					581						$ 1,846,500
			FILTER PRESS THICKENER MODULE								
	1		Thickener	4.0	4	Fixed	8 meter Dia Tank,with Rake mechanism,Hydraulic lift system	Metso	Metso		$ 181,000
	1	1	Overflow pump	22.0	22	VSD	4/3D-AH Centrifugal pump	Metso	Metso		$ 58,000
	1	1	Discharge Pump	25.0	50	VSD	VF 125 Hose Pump,55 m3/hr @65% solids	Metso	Metso		$ 116,000
	1		Flow Meter				6" Size slurry flow meter	Yokogawa	Yokogawa		$ 17,500
	1		Density Gauge				6" Pipe size,Nuclear Desity GAUGE	Australia			$ 38,000
	1		Thickener Areal Sump Pump	30	30	VSD	4/4 Vertical sump pump,warman	Warman	Warman		$ 28,000
	1		Platform and ladder and hand rail				Steel support structure ,working platform and hand rail				$ 46,000
					106						$ 484,500
			FILTER PRESS								
	1		Concentrate feed hopper				4.5 m Diameter x 5 m height vertical tank with Agitator	Metso	Metso		$ 191,000
	1		Rake assembly	132	132	Fixed	Axial flow type agitator	Metso	Metso		
	1	1	Feed Pump	45.0	90	VSD	Heavy duty slurry pump ,6/4 Warman	Warman	Warman		$ 81,000
	1		Filter Press				Metso VPA 1540-40 Air Membrane Filter	Metso	Metso		$ 1,700,000
	1	1	Low pressure [pump	45.0	45	Fixed		Metso	Metso		
	1	1	High pressure pump	22.0	22	Fixed		Metso	Metso		
	1		Oil free compressor	200.0	200	Fixed		Metso	Atlascopco		
	1		Air receiver				35 m3 tested high pressure air receiver	Metso	Atlascopco		
	1		Cloth wash system					Metso	Metso		$ 289,000
	1		Pumps/Coveyors	7.5	7.5						$ 310,000
			Bag packaging system				Custamized packaging system	Metso	Metso		$ 525,000
	1		Support structure				Steel support structure ,working platform and hand rail	Local	Local		$ 72,500
	1	1	Filter press Area Sump Pump	22	22	VSD	4/4 Vertical sump pump,warman	Warman	Warman		$ 56,000
			Filter press area Building				Pree fabricated building	PEB Steel	PEB Steel		$ 185,000
			Overhead Crane 10T				10t SWL Electric hoist-Over head crane	Demag,Malaysia	Demag,Malaysia		$ 58,000
					519						$ 3,467,500
			TAIL THICKENER MODULE								
	1		Thickener Tank	7.5	8	Fixed	16 meter Dia Tank,with Rake mechanism,Hydraulic lift system	Metso	Metso		$ 475,000
	1		Overflow pump	22.0	22	VSD	4/3D-AH Centrifugal pump	Metso	Metso		$ 72,000
	1	1	Discharge Pump	45.0	90	VSD	Hose Pump,35 m3/hr	Metso	Metso		$ 116,000
	1		Flow Meter				8" Size slurry flow meter	Yokogawa	Yokogawa		$ 17,500
	1		Density Gauge				8" Pipe size,Nuclear Desity GAUGE	Australia			$ 38,000
	1		Thickener Area Sump Pump	11	11	VSD	4/4 Vertical sump pump,warman	Warman	Warman		$ 28,000
	1		Platform and ladder and hand rail				Steel support structure ,working platform and hand rail				$ 42,000
					131						$ 788,500

Item No	Op	S'by	Title	Inst. kW ea	Total. kW	VSD/ Fixed	Description	Manufacturer	Supplier	Model №	Total Cost
					14,128					TOTAL	$29,895,500
			REAGENT MODULE								
	1		CuSO4 Mixing Tank				18 m3 SS Circular agitated tank with top and loading arrangement,size 2.8 M Dia x 3.5 M High				$ 28,500
	1		CuSO4 Storage Tank				25 m3 SS Circular tank with top				$ 37,500
	1		CuSO4 Agitator	2.25	2.25		Mixtec,motor 2.25kw,double blade agitator				$ 8,750
	1		CuSO4 Feed box				Mild steel box with loading arrangements				$ 3,800
	1		CuSO4 Transffer Pump	1.5	1.5		Close loop piping system				$ 6,500
	1		CuSO4 Dozing Pump	1.5	1.5						$ 4,800
	1		CuSO4 Flow meter								$ 5,200
					5.3						$ 95,050
	1		CMC Mixing Tank				18 m3 SS Circular tank with top and loading arrangement,size 2.8 M Dia x 3.5 M High				$ 28,500
	1		CMC Storage Tank				25 m3 SS Circular tank with top				$ 37,500
	1		CMC Feed box								$ 8,750
	1		CMC Agitator	1.50	1.5		Mixtec,motor 1.5kw				$ 3,800
	1		CMC Transffer Pump	3.0	3.0		Close loop piping system				$ 6,500
	1		CMC Dozing Pump	1.5	1.5						$ 4,800
	1		CMC Flow meter								$ 5,200
					6.0						$ 95,050
	1		PAX Mixing Tank								$ 28,500
	1		PAX Storage Tank				18 m3 SS 304 Circular top closed with opening agitated Tank,,size 2.8 M Dia x 3.5 M High				$ 37,500
	1		PAX Feed system				25 m3 SS 304 Circular top closed with opening Tank,3.4 M Dia x 3.6 M High				$ 8,750
	1	1	PAX Agitator	1.50	1.5		Mixtec				$ 3,800
	1		PAX Transfer Pump	3.0	3.0		Close loop piping system				$ 6,500
	1		PAX Dozing Pump	1.50	1.50						$ 4,800
	1		PAX Flow meter								$ 5,200
					6.0						$ 95,050
	1		CMC Mixing Tank				18 m3 SS Circular agitated tank with top and loading arrangement,size 2.8 M Dia x 3.5 M High				$ 28,500
	1		CMC Storage Tank				30 m3 SS Circular tank with top ,size 3.4 M Dia x 4 M high				$ 45,000
	1		CMC Feed box								$ 8,750
	1		CMC Agitator	1.50	1.5		Mixtec,motor 2 kw,Double blade agitator				$ 3,800
	1		CMC Transffer Pump	3.0	3.0		Close loop piping system				$ 6,500
	1		CMC Dozing Pump	1.50	1.50						$ 4,800
	1		CMC Flow meter								$ 5,200
					6.0						$ 102,550
	1		Flocculant Mixing Tank				10 m3 SS Circular agitated tank with top and loading arrangement,size 2.8 M Dia x 3.5 M High				$ 19,500
	1		Flocculant Storage Tank				20 m3 SS Circular tank with top ,size 3.4 M Dia x 4 M high				$ 37,500
	1		Flocculant Feed system								$ 8,750
	1		Flocculant Agitator	1.50	1.5		Mixtec,motor 2 kw,Double blade agitator				$ 3,800
	1		Flocculant Transffer Pump	3.0	3.0		Close loop piping system				$ 6,500
	1		Flocculant Dozing Pump	1.50	1.50						$ 4,800
	1		Flocculant Flow meter								$ 5,200
	1		Floor sump pump	11.0	11.0		3/3 vertical sump pump Warman				$ 4,850
					17.0						$ 86,050
			Reagent Platform & Handrail				Galvanized steel support structure with safety hand rail and GI gratings				$ 150,000
			Exhaust sustem				Exhaust and ventitation system				$ 26,250
			Reagent are Building				Pree fabricated building	PEB Steel	PEB Steel		$ 81,500
											$ 257,750

Item No	Op	S'by	Title	Inst. kW ea	Total. kW	VSD/ Fixed	Description	Manufacturer	Supplier	Model №	Total Cost
					14,128					TOTAL	$29,895,500
			PLANT AIR MODULE								
	1	1	High Pressure Air Compressor	75	150		GA 75+ Air cooled compressor,7.5 bar,519 cfm,Oil injected Rotary Screw compressor,IP 55	Atlas Copco	Atlas Copco		$ 150,000
	1		Refrigerant Dryer air cooled				FD 185	Atlas Copco	Atlas Copco		$ 7,400
	1		Main line Filter				DD 520	Atlas Copco	Atlas Copco		$ 1,400
	1		After Filter				PD 520	Atlas Copco	Atlas Copco		$ 950
	3		Automatic Drain Valve				WD 80	Atlas Copco	Atlas Copco		$ 750
	1		Vertical Air Receiver,4m3				LV 4011 L ,11 Bar CE,4 m3,Both side hot dip galvanized,with safety valve,Pr gauge etc	Atlas Copco	Atlas Copco		$ 17,500
	2		Vertical Air Receiver,1m3				LV 1011 L ,11 Bar, CE,1 m3,,Both side hot dip galvanized,with safety valve,Pr gauge etc	Atlas Copco	Atlas Copco		$ 4,500
	1	1	Low Pressure Air Compressor	90	180		ZE3L-2-50,Max pr 2 bar,735 cfm air delivery,Oil free screw compressor	Atlas Copco	Atlas Copco		$ 250,000
	1		Vertical Air Receiver,4 m3				Both side dish end painted 4 m3 compressor with pressure gauge,safety valve and drain	HGPT,Vietnam	HGPT		$ 17,500
			Plant area compressor area building				Pree fabricated building	PEB Steel	PEB Steel		$ 32,000
					330						$ 482,000
			WATER TANK								
	2		Water Tank,350 m3				8 Dia x 7.5 meter height,top closed steel tank with epoxy painting	Local			$ 300,000
	2	1	Discharge Pump	75	225		NBG 200-150,500M3/Hr,Head 20M,2900 RPM	Grundfos,Singapore			$ 147,000
	1	1	Discharge Pump	30	60		NBG 150-100,190M3/Hr,Head 25.6M,1450 RPM	Grundfos,Singapore			$ 75,750
	4		Water pressure gauge				Connected to two Discharge lines.[2 at pump side & 2 at plant main line]				$ 3,000
	1		Platfor,ladder and handrail				steel painted structure with galvanised gratings and pipe hand rails	Local			$ 85,000
	1		Pipings/valves								$ 95,000
					285						$ 705,750

A21-7. Process Plant Building List

Buildings	Description
Met Lab & Office Building	2 Story civil building,First floor Met lab and second floor all offices with false ceiling.Outside buiding is fully cladded with Aluminium cladding sheet.
Plant Work Shop	Preefabricated PEB building ,Concrete floor and all sides constructed 1.2 meter plasered brick wall.Installed 3 Ton mono rail crane.One rolling door.
Plant Warehouse	Preefabricated PEB building ,Concrete floor and all sides constructed,one rolling shutter and one single man door.
Plant Chemical Store Building	Prefabricated PEB building,1.2 meter plastered brick wall,2 sumps at corners,concrete floor,4 motorised rolling shutters are installed.
Dress Change Room & Security	Civil building with concrete columns and brick wall construction,Roof with sandwich panel sheet ,False ceilings are also provided.1 AC,Bath rooms with fittings.
Generator House	
Crusher Building	
Crusher Control Room	
MCCB & Control Room Building	
SAG/Ball Mill Plant Building	
Process Flotation Plant Building	
Filter press/Reagent Building	
Stockpile Building	
New warehouse building	
Plant Compressor Building	
SOS Building	
Main Security Building	
Workers Queue Shed Building.	
New Kitchen & Office Building	

A21-8. Process Plant – Ancilliary Equipment Lists

Description	Drawing	Qty	Units	Unit Rate	Total (US$)	Details
Bau-8,000 TPD Flotation Concentrate Plant						
SCHECDULE OF CONCRETE WORK						
General						
Miscellaneous Concrete,wall,trench etc		980.0	m3	$ 180.00	$ 176,400	
Miscellaneous Earth works		17500.0	m3	$ 12.00	$ 210,000	
					$ 386,400	
Raw Water Pump Station						
Earthworks		387.5	m3	$ 13.00	$ 5,038	15x15x1.5
Back filling & Compaction		245.0	m3	$ 15.00	$ 3,675	15x15x1
Installation of Concrete		38.8	m3	$ 180.00	$ 6,984	156+38 X.2
Install DN200 pipe		36.0	m	$ 105.00	$ 3,780	
Installation of concrete		28.0	m3	$ 180.00	$ 5,040	
Installation of reinforcing bar / mesh		4,000.0	kg	$ 1.65	$ 6,600	Added rebar work
HD Bolts		1.0	LS	$ 1,000.00	$ 1,000	
					$ 32,117	
Piperacks						
Earthworks		50.0	m3	$ 13.00	$ 650	
Installation of Concrete		35.0	m3	$ 180.00	$ 6,300	
Installation of reinforcing bar / mesh		2,500.0	kg	$ 1.65	$ 4,125	
Holding Down Bolts		1.0	LS	$ 1,000.00	$ 1,000	
					$ 12,075	
Fuel Storage						
Earthworks		331.0	m3	$ 13.00	$ 4,303	
Back filling & Compaction		128.0	m3	$ 15.00	$ 1,920	14x14x.5
Install concrete ring beams		37.0	m3	$ 312.00	$ 11,544	
Install concrete slab/sump		224.0	m3	$ 180.00	$ 40,320	20x14x.8
Install bund walls		16.8	m3	$ 245.00	$ 4,116	14x14x.2x1.5
Install U drain		30.0	m	$ 148.00	$ 4,440	
Install DN 150 u/g pipe		20.0	m	$ 105.00	$ 2,100	
Install concrete vehicle slab		17.0	m3	$ 180.00	$ 3,060	
Install concrete pump slab/stair plinth		9.5	m3	$ 180.00	$ 1,710	
Supply and install bollards		6.0	ea	$ 180.00	$ 1,080	
Install concrete sumps		2.0	ea	$ 180.00	$ 360	
Install oil separator pit		1.0	Ls	$ 2,025.00	$ 2,025	
Installation of reinforcing bar / mesh		14,250.0	kg	$ 1.65	$ 23,513	
					$ 100,491	
Power Station						
Earthworks		510.0	m3	$ 13.00	$ 6,630	30x17x1
Back filling & Compaction		408.0	m3	$ 15.00	$ 6,120	30x17x0.8
Generator pedestals		120.0	m3	$ 312.00	$ 37,440	4x0.5.x10x6
Column pedestals		35.0	m3	$ 312.00	$ 10,920	
Slab and kerb, including cable pits		50.0	m3	$ 180.00	$ 9,000	
Install oil separator pit		1.0	Ls	$ 2,025.00	$ 2,025	
Installation of reinforcing bar / mesh		16,000.0	kg	$ 1.65	$ 26,400	
					$ 98,535	
Service Sub Station						
Earthwork		225.0	m3	$ 13.00	$ 2,925	15x15x1
Selective soil Back filling & Compaction		180.0	m3	$ 15.00	$ 2,700	15x15x0.8
Install concrete footings		42.0	m3	$ 180.00	$ 7,560	
Installation of reinforcing bar / mesh		2,500.0	kg	$ 1.65	$ 4,125	
HD Bolts		1.0	LS	$ 1,000.00	$ 1,000	
					$ 18,310	
Reinforced Earthwall						
Install reinforced earthwall		225.0	m3	$ 180.00	$ 40,500	400/m3
Supply, place and compact fill behind wall		15,000.0	m3	$ 6.00	$ 90,000	
					$ 130,500	
Crushing Station						
Earthwork		4,000.0	m3	$ 13.00	$ 52,000	50x20x4
Install retaining wall concrete		285.0	m3	$ 180.00	$ 51,300	
Selective soil Back filling & Compaction		1,000.0	m3	$ 15.00	$ 15,000	50x2.5x8
Install concrete footings/beams		52.5	m3	$ 245.00	$ 12,863	
Install concrete for crushing slabs		80.0	m3	$ 245.00	$ 19,600	10x10x0.8
Install run on slab		12.5	m3	$ 180.00	$ 2,250	
Installation of reinforcing bar / mesh		36,000.0	kg	$ 1.40	$ 50,400	
Floor concrete		240.0	m2	$ 42.00	$ 10,080	24x10
					$ 213,493	

Bau-8,000 TPD Flotation Concentrate Plant						
SCHECDULE OF CONCRETE WORK						
Description	**Drawing**	**Qty**	**Units**	**Unit Rate**	**Total (US$)**	**Details**
Crushing Station Control Room						
Earthwork		50.0	m3	$ 13.00	$ 650	
Selective soil Back filling & Compaction		40.0	m3	$ 24.00	$ 960	
Install concrete footings/floor		16.0	m3	$ 180.00	$ 2,880	10x5x.2
Installation of reinforcing bar / mesh		1,200.0	kg	$ 1.40	$ 1,680	
					$ 6,170	
Stockpile Conveyor CV 01						
Earthwork		64.0	m3	$ 13.00	$ 832	
Install Rebar		350.0	kg	$ 1.40	$ 490	
Install concrete footings		24.0	m3	$ 180.00	$ 4,320	
					$ 5,642	
Stockpile Reclaim Tunnel						
Earthwork		3,675.0	m3	$ 13.00	$ 47,775	[50x25x1.5]+[30x12x5]
Install multi plate tunnel form work		468.0	m2	$ 24.00	$ 11,232	25x6x2x2+200
Place and compact stabilized sand mix		1,288.0	m3	$ 25.00	$ 32,200	30x12x1
Install tunnel concrete encasement		375.0	m3	$ 275.00	$ 103,125	top slab also
Install concrete floor slab		285.0	m3	$ 180.00	$ 51,300	38x25x0.3
Install concrete end walls		18.0	m3	$ 275.00	$ 4,950	
Place and compact fill around tunnel		2,025.0	m3	$ 15.00	$ 30,375	
Installation of reinforcing bar / mesh		84,000.0	kg	$ 1.40	$ 117,600	
					$ 398,557	
SAG Mill Feed Conveyor CV02/CV03						
Earthwork		621.0	m3	$ 13.00	$ 8,073	27x23x1
Place and compact stabilized sand mix		497.0	m3	$ 38.00	$ 18,886	
Install concrete footings		68.0	m3	$ 180.00	$ 12,240	
Floor concrete		532.0	m2	$ 42.00	$ 22,344	
Installation of reinforcing bar / mesh		6,500.0	kg	$ 1.40	$ 9,100	
					$ 70,643	
Grinding Area						
Earthwork		6,075.0	m3	$ 13.00	$ 78,975	[30x27x5]+[15x27x5]
Place and compact fill around		1,815.0	m3	$ 15.00	$ 27,225	
Place and compact stabilized sand mix		815.0	m3	$ 38.00	$ 30,970	
Install concrete SAG Mill foundation		575.0	m3	$ 245.00	$ 140,875	
Install concrete Ball Mill foundation		385.0	m3	$ 245.00	$ 94,325	
Install concrete footings and plinths		117.0	m3	$ 180.00	$ 21,060	
Install concrete floor slab		160.0	m3	$ 210.00	$ 33,600	90+70
Install Pump Pit concrete		68.0	m3	$ 180.00	$ 12,240	
Install concrete bund wall		32.0	m3	$ 225.00	$ 7,200	
Install concrete sumps		3.0	ea	$ 525.00	$ 1,575	
Installation of reinforcing bar / mesh		265,000.0	kg	$ 1.40	$ 371,000	
HD Bolts		1.0	LS	$ 4,500.00	$ 4,500	
					$ 823,545	
Thickener Area						
Earthwork		240.0	m3	$ 13.00	$ 3,120	20x12x1
Place and compact stabilized sand mix		192.0	m3	$ 38.00	$ 7,296	
Install concrete raft footings		37.0	m3	$ 180.00	$ 6,660	
Install pedestals/plinths		30.0	m3	$ 184.00	$ 5,520	
Install concrete bund wall		17.0	m3	$ 225.00	$ 3,825	
Install concrete sump		2.0	ea	$ 525.00	$ 1,050	
Installation of reinforcing bar / mesh		31,000.0	kg	$ 1.40	$ 43,400	
Floor concrete		210.0	m2	$ 42.00	$ 8,820	
HD Bolts		1.0		$ 3,800.00	$ 3,800	
					$ 83,491	
Cyclone Area						
Earthwork		405.0	m3	$ 13.00	$ 5,265	15x27x1
Place and compact stabilized sand mix		324.0	m3	$ 38.00	$ 12,312	
Install concrete raft footings		35.0	m3	$ 180.00	$ 6,300	
Install pedestals/plinths		28.0	m3	$ 184.00	$ 5,152	
Install concrete bund wall		18.0	m3	$ 225.00	$ 4,050	
Install concrete sump		2.0	ea	$ 525.00	$ 1,050	
Installation of reinforcing bar / mesh		28,000.0	kg	$ 1.40	$ 39,200	
Floor concrete		325.0	m2	$ 42.00	$ 13,650	
HD Bolts		1.0	LS	$ 3,200.00	$ 3,200	
					$ 90,179	

Bau-8,000 TPD Flotation Concentrate Plant						
SCHEDULE OF CONCRETE WORK						
Description	Drawing	Qty	Units	Unit Rate	Total (US$)	Details
Flotation Area						
Earthwork		984.0	m3	$ 13.00	$ 12,792	41x24x1
Place and compact stabilized sand mix		787.0	m3	$ 38.00	$ 29,906	41x24x.8
Install concrete ring beam		482.0	m3	$ 225.00	$ 108,450	
Install concrete raft footings/slab		135.0	m3	$ 180.00	$ 24,300	
Install concrete bund wall		14.0	m3	$ 225.00	$ 3,150	
Install concrete sump		26.0	m3	$ 184.00	$ 4,784	
Install pedestals/plinths		34.0	m3	$ 225.00	$ 7,650	
Installation of reinforcing bar / mesh		192,000.0	kg	$ 1.40	$ 268,800	
Floor concrete		984.0	m2	$ 42.00	$ 41,328	
HD Bolts		1.0	Ls	$ 5,150.00	$ 5,150	
					$ 506,310	
Filter Press Area						
Earthwork		423.0	m3	$ 13.00	$ 5,499	25x13x1
Place and compact stabilized sand mix		260.0	m3	$ 38.00	$ 9,880	
Install concrete raft footings/slab		285.0	m3	$ 166.00	$ 47,310	
Install pedestals/plinths		53.0	m3	$ 184.00	$ 9,752	
Install concrete bund wall		26.0	m3	$ 225.00	$ 5,850	
Install concrete sump		2.0	ea	$ 525.00	$ 1,050	
Floor concrete		275.0	m2	$ 42.00	$ 11,550	
Installation of reinforcing bar / mesh		24,000.0	kg	$ 1.40	$ 33,600	
HD Bolts		1.0	LS	$ 3,800.00	$ 3,800	
					$ 128,291	
Reagents Area						
Earthwork		325.0	m3	$ 13.00	$ 4,225	25x13x1
Place and compact stabilized sand mix		260.0	m3	$ 38.00	$ 9,880	
Install concrete footings		145.0	m3	$ 180.00	$ 26,100	
Install all other concrete		85.0	m3	$ 180.00	$ 15,300	
Install concrete sumps		3.0	ea	$ 525.00	$ 1,575	
Install epoxy coating		325.0	m2	$ 53.00	$ 17,225	
Install concrete bund wall		5.4	m3	$ 225.00	$ 1,215	
Installation of reinforcing bar / mesh		8,500.0	kg	$ 1.40	$ 11,900	
					$ 87,420	
Water Area						
Earthwork		380.0	m3	$ 13.00	$ 4,940	38x10
Place and compact stabilized sand mix		100.0	m3	$ 38.00	$ 3,800	
Install concrete ring beams		94.0	m3	$ 284.00	$ 26,696	
Install pump footings		24.0	m3	$ 180.00	$ 4,320	
Installation of reinforcing bar / mesh		6,150.0	kg	$ 1.40	$ 8,610	
Concrete Base slab		90.0	m3	$ 180.00	$ 16,200	
Install concrete bund wall		8.0	m3	$ 225.00	$ 1,800	
HD Bolts		1.0	LS	$ 750.00	$ 750	
					$ 67,116	
Plant Air						
Earthwork		448.0	m3	$ 13.00	$ 5,824	23x13x1.5
Place and compact stabilized sand mix		240.0	m3	$ 38.00	$ 9,120	
Install concrete fondation		57.0	m3	$ 180.00	$ 10,260	
Installation of reinforcing bar / mesh		3,950.0	kg	$ 1.40	$ 5,530	
Floor concrete		285.0	m2	$ 42.00	$ 11,970	
Install concrete bund wall		8.0	m3	$ 225.00	$ 1,800	
Sump and Oil trap pit		2.0	ea	$ 225.00	$ 450	
HD Bolts		1.0	LS	$ 500.00	$ 500	
					$ 45,454	

Bau-8,000 TPD Flotation Concentrate Plant						
SCHECDULE OF CONCRETE WORK						
Description	**Drawing**	**Qty**	**Units**	**Unit Rate**	**Total (US$)**	**Details**
Control Room & Plant Office						
Earthwork		335.0	m3	$ 13.00	$ 4,355	
Place and compact stabilized sand mix		125.0	m3	$ 58.00	$ 7,250	
Install concrete		89.0	m3	$ 180.00	$ 16,020	
Installation of reinforcing bar / mesh		5,670.0	kg	$ 1.40	$ 7,938	
Two story concrete building		360.0	m3	$ 275.00	$ 99,000	
Cable trenches at first floor		80.0	m	$ 180.00	$ 14,400	
					$ 148,963	
Other Construction Details						
Concrete & Granite Retaining Wall		756.0	m3	$ 165.00	$ 124,740	90*12*.7
Measonary Granite Retaining Wall		450.0	m2	$ 82.00	$ 36,900	
Gabion Basket granite retaing wall		400.0	m2	$ 62.00	$ 24,800	
Open Trench		660.0	m	$ 42.00	$ 27,720	
Concrete Cover Trench		280.0	m	$ 106.00	$ 29,680	
Grating coverd double trench		125.0	m	$ 145.00	$ 18,125	
Plant surronding floor concrete,200mm thick		10,114.0	m2	$ 42.00	$ 424,788	
Plant area security Fence		660.0	m	$ 95.00	$ 62,700	
GI Safety Barrier		380.0	m	$ 85.00	$ 32,300	
Tailing trench to the Tailing dam		520.0	m	$ 42.00	$ 21,840	
Concrete pipe supports and Inspection box		18.0	m3	$ 180.00	$ 3,240	
Return Water Pump station Concrete Platform		12.0	m3	$ 225.00	$ 2,700	
Concrete intermediate Storage tank		24.0	m3	$ 225.00	$ 5,400	
Metal Clamp		50.0	Nos	$ 5.50	$ 275	
					$ 815,208	
Total cost for all area					$ 4,268,909	
Overheads/Miscellaneous,10%					$ 426,891	
Total Concrete & Civil cost					$ 4,695,799	
						Variance
Cost as per CAPEX R5					$ 4,765,800	$ 70,001

Total Earth Works	m3	64,493
Total Concrete	m3	9,806
Total Steel reinforce bar	Ton	731

Bau-8,000 TPD Flotation Concentrate Plant									
SCHEDULE OF STRUCTURAL WORK									
Area/ Item	Description	Qty	Units	Material Rate	Total Mtl	Fabrication	Total Fab	Install Rate	Total Rate
				US$	US$	US$	US$	US$	US$
20	**Pipe rack**								
B1-20-1	Steelwork	62	tonnes	$ 1,300.00	$ 80,600	$ 1,050.00	$ 65,100	$ 725.00	$ 44,950
B1-20-2	GI Ready made rack	1200	m	$ 31.00	$ 37,200	$ 24.00	$ 28,800	$ 4.00	$ 4,800
B1-20-3	Grating	350	m2	$ 63.00	$ 22,050	$ 72.00	$ 25,200	$ 9.00	$ 3,150
B1-20-4	Stair Treads	68	ea	$ 25.00	$ 1,700	$ 27.00	$ 1,836	$ 7.50	$ 510
B1-20-5	Handrail	700	m	$ 49.00	$ 34,300	$ 38.00	$ 26,600	$ 13.50	$ 9,450
					$ 175,850		**$ 147,536**		**$ 62,860**
27	**Substation**								
B1-27-1	Steelwork	8	tonnes	$ 1,300.00	$ 10,400	$ 1,050.00	$ 8,400	$ 725.00	$ 5,800
B1-27-2	Grating	12	m2	$ 63.00	$ 756	$ 72.00	$ 864	$ 9.00	$ 108
B1-27-3	Fencing	80	m	$ 65.00	$ 5,200	$ 27.00	$ 2,160	$ 18.00	$ 1,440
					$ 16,356		**$ 11,424**		**$ 7,348**
30	**Crushing Station**								
B1-30-1	Steelwork	54	tonnes	$ 1,300.00	$ 70,200	$ 1,050.00	$ 56,700	$ 725.00	$ 39,150
B1-30-2	ROM Bin Platework	32	tonnes	$ 1,400.00	$ 44,800	$ 1,175.00	$ 37,600	$ 850.00	$ 27,200
B1-30-3	ROM Liner	9.5	tonnes	$ 3,500.00	$ 33,250	$ 1,250.00	$ 11,875	$ 850.00	$ 8,075
B1-30-4	Grating	320	m2	$ 63.00	$ 20,160	$ 72.00	$ 23,040	$ 9.00	$ 2,880
B1-30-5	Stair Treads	100	ea	$ 25.00	$ 2,500	$ 27.00	$ 2,700	$ 7.50	$ 750
B1-30-6	Handrail	340	m	$ 49.00	$ 16,660	$ 38.00	$ 12,920	$ 13.50	$ 4,590
					$ 187,570		**$ 144,835**		**$ 82,645**
30	**Conveyor CV01**								
B1-30-10	Steelwork	12	tonnes	$ 1,300.00	$ 15,600	$ 1,050.00	$ 12,600	$ 725.00	$ 8,700
B1-30-11	Grating	42	m2	$ 63.00	$ 2,646	$ 72.00	$ 3,024	$ 9.00	$ 378
B1-30-12	Stair Treads	18	ea	$ 25.00	$ 450	$ 27.00	$ 486	$ 7.50	$ 135
B1-30-13	Handrail	35	m	$ 49.00	$ 1,715	$ 38.00	$ 1,330	$ 13.50	$ 473
					$ 20,411		**$ 17,440**		**$ 9,686**
30	**Conveyor CV02**								
B1-30-20	Steelwork	48	tonnes	$ 1,300.00	$ 62,400	$ 1,050.00	$ 50,400	$ 725.00	$ 34,800
B1-30-21	Grating	132	m2	$ 63.00	$ 8,316	$ 72.00	$ 9,504	$ 9.00	$ 1,188
B1-30-22	Stair Treads	24	ea	$ 25.00	$ 600	$ 27.00	$ 648	$ 7.50	$ 180
B1-30-23	Handrail	60	m	$ 49.00	$ 2,940	$ 38.00	$ 2,280	$ 13.50	$ 810
					$ 74,256		**$ 62,832**		**$ 36,978**
31	**Reclaim Tunnel - Platforms**								
B1-31-1	Steelwork	14	tonnes	$ 1,300.00	$ 18,200	$ 1,050.00	$ 14,700	$ 725.00	$ 10,150
B1-31-2	Grating	32	m2	$ 63.00	$ 2,016	$ 72.00	$ 2,304	$ 9.00	$ 288
B1-31-3	Stair Treads	12	ea	$ 25.00	$ 300	$ 27.00	$ 324	$ 7.50	$ 90
B1-31-4	Handrail	25	m	$ 25.00	$ 625	$ 27.00	$ 675	$ 13.50	$ 338
					$ 21,141		**$ 18,003**		**$ 10,866**
31	**Conveyor CV02**								
B1-31-10	Steelwork	28	tonnes	$ 1,300.00	$ 36,400	$ 1,050.00	$ 29,400	$ 725.00	$ 20,300
B1-31-11	Grating	60	m2	$ 63.00	$ 3,780	$ 72.00	$ 4,320	$ 9.00	$ 540
B1-31-12	Stair Treads	3	ea	$ 25.00	$ 75	$ 27.00	$ 81	$ 7.50	$ 23
B1-31-13	Handrail	75	m	$ 49.00	$ 3,675	$ 38.00	$ 2,850	$ 13.50	$ 1,013
					$ 43,930		**$ 36,651**		**$ 21,875**
40	**Grinding**								
B1-32-1	Steelwork	82	tonnes	$ 1,300.00	$ 106,600	$ 1,050.00	$ 86,100	$ 725.00	$ 59,450
B1-32-2	Grating	343	m2	$ 63.00	$ 21,609	$ 72.00	$ 24,696	$ 9.00	$ 3,087
B1-32-3	Stair Treads	85	ea	$ 25.00	$ 2,125	$ 27.00	$ 2,295	$ 7.50	$ 638
B1-32-4	Handrail	232	m	$ 49.00	$ 11,368	$ 38.00	$ 8,816	$ 13.50	$ 3,132
					$ 141,702		**$ 121,907**		**$ 66,307**
41	**Milling& Cyclone**								
B1-40-1	Steelwork	38	tonnes	$ 1,300.00	$ 49,400	$ 1,050.00	$ 39,900	$ 725.00	$ 27,550
B1-40-2	Grating	180	m2	$ 63.00	$ 11,340	$ 72.00	$ 12,960	$ 9.00	$ 1,620
B1-40-3	Stair Treads	82	ea	$ 25.00	$ 2,050	$ 27.00	$ 2,214	$ 7.50	$ 615
B1-40-4	Handrail	156	m	$ 49.00	$ 7,644	$ 38.00	$ 5,928	$ 13.50	$ 2,106
					$ 70,434		**$ 61,002**		**$ 31,891**
50	**Rougher/Scavenger Flotation**								
B1-50-1	Steelwork	41	tonnes	$ 1,300.00	$ 53,300	$ 1,050.00	$ 43,050	$ 725.00	$ 29,725
B1-50-2	Grating	380	m2	$ 63.00	$ 23,940	$ 72.00	$ 27,360	$ 9.00	$ 3,420
B1-50-3	Stair Treads	96	ea	$ 25.00	$ 2,400	$ 27.00	$ 2,592	$ 7.50	$ 720
B1-50-4	Handrail	190	m	$ 49.00	$ 9,310	$ 38.00	$ 7,220	$ 13.50	$ 2,565
					$ 88,950		**$ 80,222**		**$ 36,430**

Bau-8,000 TPD Flotation Concentrate Plant

SCHEDULE OF STRUCTURAL WORK

Area/ Item	Description	Qty	Units	Material Rate	Total Mtl	Fabrication	Total Fab	Install Rate	Total Rate
				US$	US$	US$	US$	US$	US$
60	**Regrind/Cyclone**								
B1-60-1	Steelwork	34	tonnes	$ 1,300.00	$ 44,200	$ 1,050.00	$ 35,700	$ 725.00	$ 24,650
B1-60-2	Grating	128	m2	$ 63.00	$ 8,064	$ 72.00	$ 9,216	$ 9.00	$ 1,152
B1-60-3	Stair Treads	75	ea	$ 25.00	$ 1,875	$ 27.00	$ 2,025	$ 7.50	$ 563
B1-60-4	Handrail	190	m	$ 49.00	$ 9,310	$ 38.00	$ 7,220	$ 13.50	$ 2,565
					$ 63,449		**$ 54,161**		**$ 28,930**
70	**Cleaner Flotation**								
B1-70-1	Steelwork	28	tonnes	$ 1,300.00	$ 36,400	$ 1,050.00	$ 29,400	$ 725.00	$ 20,300
B1-70-2	Grating	290	m2	$ 63.00	$ 18,270	$ 72.00	$ 20,880	$ 9.00	$ 2,610
B1-70-3	Stair Treads	84	ea	$ 25.00	$ 2,100	$ 27.00	$ 2,268	$ 7.50	$ 630
B1-70-4	Handrail	150	m	$ 49.00	$ 7,350	$ 38.00	$ 5,700	$ 13.50	$ 2,025
					$ 64,120		**$ 58,248**		**$ 25,565**
80	**Cocentrate Thickener**								
B1-80-1	Steelwork	12	tonnes	$ 1,300.00	$ 15,600	$ 1,050.00	$ 12,600	$ 725.00	$ 8,700
B1-80-2	Grating	72	m2	$ 63.00	$ 4,536	$ 72.00	$ 5,184	$ 9.00	$ 648
B1-80-3	Stair Treads	24	ea	$ 25.00	$ 600	$ 27.00	$ 648	$ 7.50	$ 180
B1-80-4	Handrail	72	m	$ 49.00	$ 3,528	$ 38.00	$ 2,736	$ 13.50	$ 972
					$ 24,264		**$ 21,168**		**$ 10,500**
90	**Filter Press/Packaging**								
B1-90-1	Steelwork	28	tonnes	$ 1,300.00	$ 36,400	$ 1,050.00	$ 29,400	$ 725.00	$ 20,300
B1-90-2	Grating	162	m2	$ 63.00	$ 10,206	$ 72.00	$ 11,664	$ 9.00	$ 1,458
B1-90-3	Stair treads	45	ea	$ 25.00	$ 1,125	$ 27.00	$ 1,215	$ 7.50	$ 338
B1-90-4	Handrail	104	m	$ 49.00	$ 5,096	$ 38.00	$ 3,952	$ 13.50	$ 1,404
					$ 52,827		**$ 46,231**		**$ 23,500**
100	**Reagents**								
B1-100-1	Steel work	29	tonnes	$ 1,300.00	$ 37,700	$ 1,950.00	$ 56,550	$ 725.00	$ 21,025
B1-100-2	Grating	175	m2	$ 63.00	$ 11,025	$ 72.00	$ 12,600	$ 9.00	$ 1,575
B1-100-3	Stair treads	30	ea	$ 25.00	$ 750	$ 27.00	$ 810	$ 7.50	$ 225
B1-100-4	Handrail	88	m	$ 49.00	$ 4,312	$ 38.00	$ 3,344	$ 13.50	$ 1,188
					$ 53,787		**$ 73,304**		**$ 24,013**
110	**Tailing Area**								
B1-110-1	Structural steel	7	tonnes	$ 1,300.00	$ 9,100	$ 1,050.00	$ 7,350	$ 725.00	$ 5,075
B1-110-2	Ponton plate work	4	tonnes	$ 1,400.00	$ 5,600	$ 1,250.00	$ 5,000	$ 9.00	$ 36
B1-110-3	Grating	40	m2	$ 63.00	$ 2,520	$ 72.00	$ 2,880	$ 12.00	$ 480
B1-110-4	Stair Treads	18	ea	$ 25.00	$ 450	$ 27.00	$ 486	$ 7.50	$ 135
	Handrails	40	m	$ 49.00	$ 1,960	$ 38.00	$ 1,520	$ 13.50	$ 540
					$ 19,630		**$ 17,236**		**$ 6,266**
120	**Raw/Process/Fire Water**								
B1-120-1	Steel work	25	tonnes	$ 1,300.00	$ 32,500	$ 1,050.00	$ 26,250	$ 725.00	$ 18,125
B1-120-2	Grating	140	m2	$ 63.00	$ 8,820	$ 72.00	$ 10,080	$ 9.00	$ 1,260
B1-120-3	Stair treads	72	ea	$ 25.00	$ 1,800	$ 27.00	$ 1,944	$ 7.50	$ 540
B1-120-4	Handrail	126	m	$ 49.00	$ 6,174	$ 38.00	$ 4,788	$ 13.50	$ 1,701
					$ 49,294		**$ 43,062**		**$ 21,626**
130	**Plant Air**								
B1-130-1	Steelwork	16	tonnes	$ 1,300.00	$ 20,800	$ 1,050.00	$ 16,800	$ 725.00	$ 11,600
B1-130-2	Grating	32	m2	$ 63.00	$ 2,016	$ 72.00	$ 2,304	$ 9.00	$ 288
B1-130-3	Stair treads	12	ea	$ 25.00	$ 300	$ 27.00	$ 324	$ 7.50	$ 90
B1-130-4	Handrail	40	m	$ 49.00	$ 1,960	$ 38.00	$ 1,520	$ 13.50	$ 540
					$ 25,076		**$ 20,948**		**$ 12,518**
	TOTAL AMOUNT				**$ 1,193,047**		**$ 1,036,210**		**$ 519,802**
							GRAND TOTAL		**$ 2,749,059**
							BUDGET		**$ 2,859,480**
							VARIANCE		**$ 110,422**

Bau-8,000 TPD Flotation Concentrate Plant

PIPING WORK WITH ACCESSORIES

Description	Value
Pipe G01	$ 58,119
Pipe S01	$ 43,260
Pipe C01	$ 131,718
HDPE P01	$ 59,358
Mine Hose	$ 149,660
Moulded Elbows P02	$ 5,936
Sweep Bends C01	$ 4,749
HDPE Tailing Spigotting	$ 4,155
HDPE Pipe fittings/Flange	$ 5,936
Fire Protection System	$ 35,000
Stainers	$ 7,500
Muff coupling for mine hose	$ 27,760
	$ 533,150

Sl No	Details	Value as per Calculation	Value Included in Capex	Net Value after Adjustment with Capex
1	Piping Work	$ 533,150	$ -	$ 533,150
2	Valves	$ 733,130	$ -	$ 733,130
3	Pumps	$ 1,735,000	$ 852,000	$ 883,000
		$ 3,001,280	**$ 852,000**	**$ 2,149,280**

Bau-8,000 TPD Flotation Concentrate Plant - Pumps						
Sl No	Area	Details	Qty	Unit Price	US$ Total Cost	Remarks
A	Crusher Module					
1	Sump Pump	4/4 Vertical sump pump,warman	1	$ 28,000	$ 28,000	Included in CR Capex
					$ -	
B	SAG Mill Area				$ -	
1	SAG Mill Discharge Pump	12/10 D-AH,Centrifugal metal liner pump with gland water,1600 rpmY 250M,6/37 KW,IP54	2	$ 52,500	$ 105,000	
2	Sump Pump	4/4 Vertical sump pump,warman	1	$ 28,000	$ 28,000	
					$ -	
C	SAG Mill Cyclone area				$ -	
1	Cyclone feed pump	12/10 D-AH,Centrifugal metal liner pump with gland water,1600 rpmY 250M,6/37 KW,IP54	2	$ 52,500	$ 105,000	Included in Cyclone Capex
2	Sump pump	4/4 Vertical sump pump,warman	1	$ 28,000	$ 28,000	
D	Rougher/Scavenger Flotation				$ -	
1	Concentrate discharge pump	VF 250 Metso Sala,fixed speed pump	2	$ 46,000	$ 92,000	Included in Flotation Capex
2	Sump Pump	4/4 Vertical sump pump,warman	1	$ 28,000	$ 28,000	included
3	Flotation Tails Pump	12/10 D-AH,Centrifugal metal liner pump with gland water,1600 rpmY 250M,6/37 KW,IP54	2	$ 52,500	$ 105,000	Included
E	Regrind Ball Mill/Cyclone area				$ -	
1	Ball Mill Discharge pump	6/4" Warman slurry pump	2	$ 42,500	$ 85,000	
2	Sump Pump	3/3 vertical sump pump Warman	1	$ 23,000	$ 23,000	Included in RGM Capex
					$ -	
F	Cleaner Flotation				$ -	
1	Cleaner Flotation concentrate discharge pump	VF 200 Metso Sala,fixed speed pump	2	$ 44,000	$ 88,000	
2	Sump Pump	3/3 vertical sump pump Warman	1	$ 23,000	$ 23,000	
3	Flotation Tails Pump	6/4" Warman slurry pump	2	$ 42,500	$ 85,000	
					$ -	
G	Thickener Area					
1	Thickener Discharge Pump	VF 125 Hose Pump,55 m3/hr @65% solids	2	$ 58,000	$ 116,000	
2	Sump Pump	3/3 vertical sump pump Warman	1	$ 23,000	$ 23,000	
H	Filter press Feed Tank/Filter press					
	Feed Discharge pump	VF 125 Hose Pump,55 m3/hr @65% solids	2	$ 58,000	$ 116,000	Included in Filter Press Capex
	Filter press discharge pump	6/4" Warman slurry pump	2	$ 42,500	$ 85,000	Included in Filter Press Capex
	Sump Pump	4/4 Vertical sump pump,warman	2	$ 28,000	$ 56,000	Included in Filter Press Capex
I	RAW/Process water area					
1	Raw water pump	NBG 150-100,190M3/Hr,Head 25.6M,1450 RPM	2	$ 37,500	$ 75,000	
2	Process water pump	NBG 200-150,500M3/Hr,Head 20M,2900 RPM	2	$ 49,000	$ 98,000	Included in Water Tank Capex
					$ -	
J	Tailings Discharge pump					
1	Tails discharge pump	12/10 D-AH,Centrifugal metal liner pump with gland water,1600 rpmY 250M,6/37 KW,IP54	2	$ 52,500	$ 105,000	
2	Sump Pump	4/4 Vertical sump pump,warman	1	$ 28,000	$ 28,000	
					$ -	
K	Tailing water Return pump					
1	Pontoon submercible pump	Grundfos submercible Pump	3	$ 24,500	$ 73,500	
2	High capacity process water return pump	NBG 250-200,500M3/Hr,Head 50M,2900 RPM	2	$ 57,000	$ 114,000	
					$ -	
L	Fire & Safety					
1	Fire & Safety Pump	Grundfos High Pressure Fire Pump	1	$ 22,500	$ 22,500	
					$ -	
					$ -	
					$ -	$ 852,000
	TOTAL		42		$ 1,735,000	
			Net Ammount		$ 883,000	

Bau-80,000 TPD Flotation Concentrate Plant

SCHEDULE OF PIPING WORK RATE

Item	Description	Size NB	Qty	Units	Unit Rate US$	Total Rate US$
B3-C01	Carbon Steel - Spec C01	15	42	m	$ 9.00	$ 378.00
B3-C02		20	18	m	$ 4.00	$ 72.00
B3-C03		25	480	m	$ 6.00	$ 2,880.00
B3-C04		32	92	m	$ 85.00	$ 7,820.00
B3-C05		40	80	m	$ 51.00	$ 4,080.00
B3-C06		50	440	m	$ 23.00	$ 10,120.00
B3-C07		80	318	m	$ 46.00	$ 14,628.00
B3-C08		100	312	m	$ 92.00	$ 28,704.00
B3-C09		150	240	m	$ 103.00	$ 24,720.00
B3-C10		200	66	m	$ 145.00	$ 9,570.00
B3-C11		250	62	m	$ 195.00	$ 12,090.00
B3-C12		300	48	m	$ 347.00	$ 16,656.00
SUB TOTAL - I						$ 131,718.00
B3-G01	Galvanised Spec G01	15	165	m	$ 3.00	$ 495.00
B3-G02		25	560	m	$ 9.00	$ 5,040.00
B3-G03		40	240	m	$ 16.00	$ 3,840.00
B3-G04		50	480	m	$ 18.00	$ 8,640.00
B3-G05		80	36	m	$ 59.00	$ 2,124.00
B3-G06		100	120	m	$ 114.00	$ 13,680.00
B3-G07		150	180	m	$ 135.00	$ 24,300.00
SUB TOTAL - II						$ 58,119.00
B3-R01	Mine hose Spec R01	50	500	m	$ 35.00	$ 17,500.00
B3-R02		75	325	m	$ 56.00	$ 18,200.00
B3-R03		100	280	m	$ 86.00	$ 24,080.00
B3-R04		125	120	m	$ 120.00	$ 14,400.00
B3-R05		150	280	m	$ 136.00	$ 38,080.00
B3-R06		200	50	m	$ 220.00	$ 11,000.00
B3-R07		400	20	m	$ 1,320.00	$ 26,400.00
SUB TOTAL - III						$ 149,660.00
B3-P01	HDPE Pipe - Spec P01	32	50	m	$ 3.00	$ 150.00
B3-P02		63	1800	m	$ 12.00	$ 21,600.00
B3-P03		75	60	m	$ 14.00	$ 840.00
B3-P04		90	360	m	$ 24.00	$ 8,640.00
B3-P05		110	120	m	$ 30.00	$ 3,600.00
B3-P06		160	144	m	$ 58.00	$ 8,352.00
B3-P07		200	96	m	$ 74.00	$ 7,104.00
B3-P08		250	84	m	$ 108.00	$ 9,072.00
SUB TOTAL - IV						$ 59,358.00
B3-S01	Stainless - Spec S01	15	90	m	$ 27.00	$ 2,430.00
B3-S02		25	72	m	$ 30.00	$ 2,160.00
B3-S03		50	240	m	$ 93.00	$ 22,320.00
B3-S04		80	78	m	$ 175.00	$ 13,650.00
B3-S05		1"	120	m	$ 15.00	$ 1,800.00
B3-S06		1/2"	90	m	$ 10.00	$ 900.00
SUB TOTAL - V						$ 43,260.00
TOTAL AMOUNT						$ 442,115.00

Bau-8,000 TPD Flotation Concentrate Plant

Sl No	Location		KNIFE GATE VALVE-AIR ACTUATED								PINCH VALVE					BALL VALVE					BUTTERFLY					TECH- TAYLOR				
	Description		250	200	150	100	80	50	40	25	200	150	100	80	50	100	80	50	25	15	150	100	80	50	25	250	200	150	100	80
1	SAG/Ball Mill Module																													
	SAG/Ball Mill		4	4	6	8											5	10	12	8	2	4	4			2	2	1	1	
2	Cyclone &Knelson																													
	Cyclone			4	4	2						1				3	4	8			2	4						1	1	
	Knelson				2																		2							
3	Rougher/Scavenger Flotation																													
				4	4	6	6	4								1	8	10	12		2	8	8					1		
4	To high capacity thickener																													
					4	4	6					1					6	8	8	4	2	4	4						1	1
5	Cleaner Flotation /Cyclone																													
					4	4	6	3									6	8	8	8		2	4	6				1	1	1
6	Filter Press Module																													
					2	4	6	4				1					8	10	12	8		4	4	4					1	
7	Reagent area																													
																	8	10	12	12		2	3							
8	Raw & Process water tanks/Crusher area																													
					4	6										4	12	14	18	13										
		Qty	4	12	30	34	24	11	0	0	0	3	0	0	0	8	57	78	82	53	8	28	29	10	0	2	2	4	5	2
		Spare	1	1	1	1	1	1	0	0		1	0	0	0	1	1	3	5	4	1	2	3	4	0	0	0	0	0	0
		Total	**5**	**13**	**31**	**35**	**25**	**12**	**0**	**0**	**0**	**4**	**0**	**0**	**0**	**9**	**58**	**81**	**87**	**57**	**9**	**30**	**32**	**14**	**0**	**2**	**2**	**4**	**5**	**2**
		Unit Price	4550	3600	3050	2700	2350	2150	0	0		4350	3625	2450	0	1250	950	690	250	135	495	350	275	210	150	21050	16750	11500	8500	6300
		Total Cost	22,750	46,800	94,550	94,500	58,750	25,800	-	-	-	17,400	-	-	-	11,250	55,100	55,890	21,750	7,695	4,455	10,500	8,800	2,940	-	42,100	33,500	46,000	42,500	12,600
	Total Valve Cost	715,630																												
	Non Return Valve & Reducer	17,500																												
	Total Valve+Reducer+Non return valve	733,130																												

B. Enclosures

B17-1. 8,000 tpd Flotation Concentrate Flow Schematic

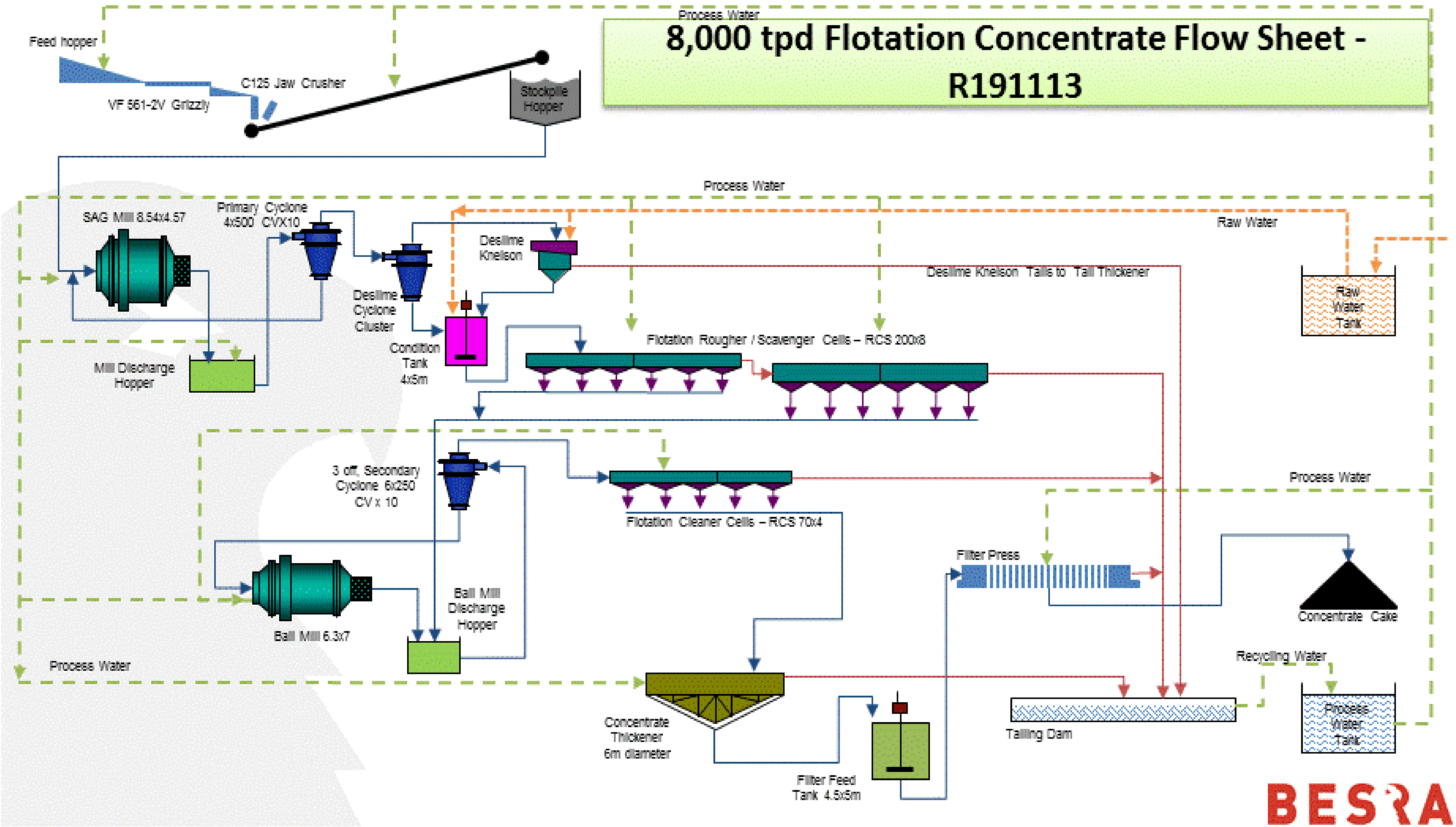
BESRA BAU GOLDFIELD PROJECT - 8,000 TPD PROCESS FLOW SCHEMATIC

Process Stage	Design Duties	Suggested Equipment
Grizzly Feeder Primary Crusher	Design feedrate 670 tph ROM with 15 hours/day design operation and 90% availability factor applied. ROM top size 800mm, P100 = 210mm, P80 = 130mm, Solids SG 2.8, CWi not stated, ore assumed moderate/soft, free moisture not stated assumed 3-4%	VF561-2V Grizzly Feeder ,30 Kw motor, C125 Jaw Crusher with 160 Kw MOTOR ,Rock Breaker,Conveyors and dust control systems
Crushed Ore Stockpile Reclaim	Minimum 12,000 Tons of Ore ,Stockpile concreted yard & Store area	No stacking or reclaim equipment is not considered at this stage
Primary SAG Mill	Design feedrate 400tph after 95% availability factor and 1.15SF applied. F80 = 130mm, P80 = 150um. Closed circuit operation. Ore BRMWi = 13.2kWh/t, BBMWi = 14.1kWh/t, SPI = 26.4mins. Ore BD not stated, SG 2.8	SAG Mill 28ft (8.54m) internal diameter x 16ft (4.87m) effective grinding length Semi Autogenous Grinding Mill with 6200 Kw MOTOR,Media loading system,Liner handler etc
Primary Hydrocyclones	250-300% recirculating load assume, cyclone overflow p80 = 150um, 35% solids in overflow.	Cavex 4 x 500CVX10 Cyclone Cluster- 3 Operating and 1 Standby
Deslime Cyclone	400tph new feed, 35% solids, F80=150um, 85% solids in underflow	Cavex 8 x 400CVX10 Dewatering Cyclone Cluster- 6 Operating and 2 Standby
Deslime Knelson	743m3/h of slurry Feed with 8% solid concentrate	2xKnelson CVD64
Conditioner	400tph solids, 30% solids w/w, solids SG=2.8, slurry SG = 1.26, 993 m3/h slurry flowrate, p80= 150um, max particle size 250um, 3 mins residence time	4m diameter x 5m side wall height conditioning tank with 90 kW agitator,Reagents system etc
Rougher and Scavenger Flotation Cells	435tph solids, 27% solids w/w, solids SG=2.9, 1347 m3/h slurry, p80= 150um, max particle size assume 250um, 30 mins residence time each for rougher & scavenger stages. Air hold up 15%	RCS 200 Flotation Machine, 8 cells, 200m3 effective volume per cell,200 Kw motor ,Air blowers,level and Air control systems and launder sprays and reagents systems
Regrind Hydrocyclones	73tph new feed, 500% recirculating load assume,15% solids, F80 130 and cyclone overflow p80 = 30um, 65% solids in underflow.	3 off ,Cavex 6 x 250CVX 10 Cyclone Cluster - 5 Operating and 1 standby cyclone each cluster
Regrind Ball Mill	73 tph solids new feed, reverse closed circuit operation 500% recirculation load,F80 = 130um, P80= 30um, solids SG 3.2.	3.6 m internal diameter x 7 m effective grinding length Overflow Discharge Ball Mill.1350 Kw SC motor,Media charging system,feed Chute etc
Cleaner Flotation Cells	73 tph solids, 10% solids w/w, solids SG=3.2, Slurry SG 1.026,flow rate 683 m3/h slurry, p80= 30 um, max particle size assume 250um, 20 mins residence. Air hold up 15%	RCS 70 Flotation Machine, 4 cells, 70m3 effective volume per cell ,90 Kw MOTOR ,Air blowers,level and Air control systems and launder sprays and reagents systems
Concentrate Thickener	40 tph solids, solids SG = 3.2, 8% solids w/w, slurry SG = 1.06, 473m3/h slurry, assume p80 = 30 um, underflow 65% solids w/w. 2 Hours residence time.	High rate thickener, 6m diameter,Automatic Rake,2.2 Kw MOTOR.Flocculant system and control systems
Concentrate Filter Feed Tank	40 tph solids, solids SG = 3.2, 65% solids w/w, slurry SG = 1.81, 35 m3/h slurry, p80 = 30um, max particle size 150um, 2 hours residence time.	4.5m in diameter x 5m side wall height slurry tank with 110 kW agitator
Concentrate Filter Press	40 tph solids, solids SG = 3.2, 65% solids w/w, slurry SG = 1.81, 35 m3/h slurry, p80 = 30um, max particle size assume 120um, filter cake moisture assume 9% w/w	Metso VPA 1540-40 Air Membrane Pressure Filter.Compressed Air system,Hydraulic and control system,Cloth was water/filterate tank and belt conveyors
Concentrate Filter Cake Stockpile	Stockpile area is included	Conveyor and bagging system is included
Tailing Dam	Tailing Storage Facility	Recycling Tank and pumps are included

Primary Hydrocyclones: O/F, U/F
Deslime Cyclone: O/F, U/F
Deslime Knelson: TAILS, CONCENTRATE
Rougher and Scavenger Flotation Cells: TAILS, CONCENTRATE
Regrind Hydrocyclones: O/F, U/F
Regrind Ball Mill: FINE
Cleaner Flotation Cells: TAILS, CONCENTRATE
Concentrate Thickener: O/F, U/F
Concentrate Filter Press: WATER
Tailing Dam: Recycling Water

To Process

B17-2. 8,000 tpd Flotation Concentrate Flow Sheet



B22-1. Cost Model Cashflow Worksheets

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Mined Ore Tonnes	10,927,500	240,900	489,100	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,100	730,100	730,100	730,100	729,400	721,700	716,000	-	-	-	-	-
Mined Au Grade	1.70	1.53	1.53	1.53	1.53	1.53	1.58	1.58	1.58	1.58	1.56	1.56	1.56	1.56	1.59	2.11	3.24	-	-	-	-	-
Mined Au Ounces	598,830	11,870	24,090	35,960	35,960	35,960	36,990	36,990	36,990	36,990	36,600	36,600	36,600	36,600	37,190	48,940	74,500	-	-	-	-	-
Cumulative Mined Ore Tonnes		240,900	730,000	1,460,000	2,190,000	2,920,000	3,650,000	4,380,000	5,110,000	5,840,000	6,570,100	7,300,200	8,030,300	8,760,400	9,489,800	10,211,500	10,927,500	-	-	-	-	-
Cumulative Mined Au Grade		1.53	1.53	1.53	1.53	1.53	1.54	1.55	1.55	1.55	1.55	1.56	1.56	1.56	1.56	1.60	1.70	-	-	-	-	-
Cumulative Mined Au Ounces		11,870	35,960	71,920	107,880	143,840	180,830	217,820	254,810	291,800	328,400	365,000	401,600	438,200	475,390	524,330	598,830	-	-	-	-	-
Processed Ore Tonnes	10,927,500		240,900	489,100	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,100	730,100	730,100	730,100	729,400	721,700	716,000	-	-	-	-
Recovered Au Grade	1.32		1.19	1.19	1.19	1.19	1.19	1.22	1.22	1.22	1.22	1.21	1.21	1.21	1.21	1.23	1.63	2.51	-	-	-	-
Recovered Au Ounces	463,650		9,190	18,650	27,840	27,840	27,840	28,640	28,640	28,640	28,640	28,340	28,340	28,340	28,340	28,800	37,890	57,680	-	-	-	-
Cumulative Processed Ore Tonnes		-	240,900	730,000	1,460,000	2,190,000	2,920,000	3,650,000	4,380,000	5,110,000	5,840,000	6,570,100	7,300,200	8,030,300	8,760,400	9,489,800	10,211,500	10,927,500	-	-	-	-
Cumulative Recovered Au Grade		-	1.19	1.19	1.19	1.19	1.19	1.19	1.20	1.20	1.20	1.20	1.20	1.20	1.20	1.21	1.24	1.32	-	-	-	-
Cumulative Recovered Au Ounces		-	9,190	27,840	55,680	83,520	111,360	140,000	168,640	197,280	225,920	254,260	282,600	310,940	339,280	368,080	405,970	463,650	-	-	-	-
Waste Volume			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Tonnes	18,569,000	118,100	239,900	358,000	358,000	358,000	922,600	922,600	922,600	922,600	1,650,100	1,650,100	1,650,100	1,650,100	1,813,000	2,320,900	2,712,300	-	-	-	-	-
Cumulative Waste Tonnes		118,100	358,000	716,000	1,074,000	1,432,000	2,354,600	3,277,200	4,199,800	5,122,400	6,772,500	8,422,600	10,072,700	11,722,800	13,535,800	15,856,700	18,569,000	-	-	-	-	-
Strip Ratio	1.70	0.49	0.49	0.49	0.49	0.49	1.26	1.26	1.26	1.26	2.26	2.26	2.26	2.26	2.49	3.22	3.79	-	-	-	-	-
Cumulative Strip Ratio		0.49	0.49	0.49	0.49	0.49	0.65	0.75	0.82	0.88	1.03	1.15	1.25	1.34	1.43	1.55	1.70	-	-	-	-	-
Capital Costs:																						
Capital Development/Pre-stripping																						
Capital Costs Relative to Option																						
Capital Cost - Mining	$ 4,304,495	$ 4,304,495																				
Capital Cost - Processing (Main)	$ 58,547,853	$ 58,547,853																				
Capital Cost - Processing (Addn CIL Circuit)	$ -																					
Capital Cost - Processing (Heap Leach)	$ -																					
Capital Cost - Transport	$ -	$ -																				
Capital Cost - Other	$ 60,205,555	$ 29,087,341	$ -	$ -	$ 11,356,641	$ 8,469,245	$ 5,646,164	$ -	$ -	$ -	$ -	$ -	$ 2,823,082	$ 2,823,082	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Rehabilitation (Stage 1&2)	$ 7,160,750		$ -	$ -	$ -	$ -	$ -	$ -	$ 2,403,780	$ -	$ -	$ -	$ -	$ -	$ 3,166,970	$ -	$ 795,000	$ 795,000	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Process	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Land Acqisition	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - TSF & Other	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Rehabilitation	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Condemnation/Resource Drilling	$ 180,000	$ 180,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Annual Sustaining Capital	$ 4,479,262		$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ -	$ -	$ -	$ -	$ -
Total Capital Costs	$ 134,877,915	$ 92,119,690	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 3,121,699	$ 3,465,587	$ 298,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -	$ -
Cumulative Capital Costs		$ 92,119,690	$ 92,418,307	$ 92,716,925	$ 104,372,183	$ 113,140,046	$ 119,084,827	$ 119,383,444	$ 122,085,842	$ 122,384,459	$ 122,683,077	$ 122,981,694	$ 126,103,394	$ 129,225,093	$ 132,690,680	$ 132,989,298	$ 134,082,915	$ 134,877,915	$ -	$ -	$ -	$ -
Operating Costs:																						
Mining - Waste	$ 50,525,000	$ 321,342	$ 652,752	$ 974,094	$ 974,094	$ 974,094	$ 2,510,333	$ 2,510,333	$ 2,510,333	$ 2,510,333	$ 4,489,811	$ 4,489,811	$ 4,489,811	$ 4,489,811	$ 4,933,051	$ 6,315,013	$ 7,379,986	$ -	$ -	$ -	$ -	$ -
Mining - Ore	$ 33,862,504	$ 746,509	$ 1,515,640	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,458	$ 2,262,458	$ 2,262,458	$ 2,262,458	$ 2,260,289	$ 2,236,428	$ 2,218,765	$ -	$ -	$ -	$ -	$ -
Labour (Mine Overhead)	$ 18,872,319	$ 410,268	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ -	$ -	$ -	$ -	$ -
General Costs	$ 584,299	$ 12,881	$ 26,152	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,039	$ 39,039	$ 39,039	$ 39,039	$ 39,001	$ 38,590	$ 38,285	$ -	$ -	$ -	$ -	$ -
Engineering Costs	$ 988,097	$ 21,783	$ 44,226	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,018	$ 66,018	$ 66,018	$ 66,018	$ 65,955	$ 65,258	$ 64,743	$ -	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Main)	$ 196,325,629	$ -	$ 4,307,431	$ 8,745,221	$ 13,052,652	$ 13,052,652	$ 13,052,652	$ 13,068,652	$ 13,068,652	$ 13,068,652	$ 13,068,652	$ 13,064,363	$ 13,064,363	$ 13,064,363	$ 13,064,363	$ 13,061,581	$ 13,111,576	$ 13,409,805	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Addn CIL)	$ -																					
Metallurgical/Processing Costs (Heap Leach)	$ -																					
Transport Cost to Central Plant/Port	$ 35,796,851	$ -	$ 789,152	$ 1,602,218	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,698	$ 2,391,698	$ 2,391,698	$ 2,391,698	$ 2,389,405	$ 2,364,181	$ 2,345,509	$ -	$ -	$ -	$ -
General Overhead (BESRA)	$ 6,010,125	$ 132,495	$ 269,005	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,555	$ 401,555	$ 401,555	$ 401,555	$ 401,170	$ 396,935	$ 393,800	$ -	$ -	$ -	$ -	$ -
Total Operating Costs	$ 342,964,825	$ 1,645,278	$ 8,835,161	$ 15,321,027	$ 20,417,610	$ 20,417,610	$ 21,953,849	$ 21,969,849	$ 21,969,849	$ 21,969,849	$ 23,949,707	$ 23,945,746	$ 23,945,746	$ 23,945,746	$ 24,386,331	$ 25,734,013	$ 26,802,138	$ 15,755,314	$ -	$ -	$ -	$ -
Operating Cost per Tonne Ore	$ 31.39	$ 6.83	$ 18.06	$ 20.99	$ 27.97	$ 27.97	$ 30.07	$ 30.10	$ 30.10	$ 30.10	$ 32.80	$ 32.80	$ 32.80	$ 32.80	$ 33.43	$ 35.66	$ 37.43	$ -	$ -	$ -	$ -	$ -
Cumulative Operating Costs		$ 1,645,278	$ 10,480,439	$ 25,801,466	$ 46,219,077	$ 66,636,687	$ 88,590,536	$ 110,560,385	$ 132,530,234	$ 154,500,083	$ 178,449,789	$ 202,395,536	$ 226,341,282	$ 250,287,028	$ 274,673,359	$ 300,407,372	$ 327,209,511	$ 342,964,825	$ -	$ -	$ -	$ -
Total Costs:																						
Total Costs	$ 477,842,740	$ 93,764,968	$ 9,133,779	$ 15,619,644	$ 32,072,869	$ 29,185,473	$ 27,898,630	$ 22,268,466	$ 24,672,246	$ 22,268,466	$ 24,248,324	$ 24,244,364	$ 27,067,445	$ 27,067,445	$ 27,851,919	$ 26,032,631	$ 27,895,756	$ 16,550,314	$ -	$ -	$ -	$ -
Total Cumulative Costs		$ 93,764,968	$ 102,898,746	$ 118,518,391	$ 150,591,260	$ 179,776,733	$ 207,675,363	$ 229,943,829	$ 254,616,076	$ 276,884,542	$ 301,132,866	$ 325,377,230	$ 352,444,675	$ 379,512,121	$ 407,364,039	$ 433,396,670	$ 461,292,426	$ 477,842,740	$ -	$ -	$ -	$ -
Total Cost per Tonne Ore	$ 43.73	$ 389.23	$ 18.67	$ 21.40	$ 43.94	$ 39.98	$ 38.22	$ 30.50	$ 33.80	$ 30.50	$ 33.21	$ 33.21	$ 37.07	$ 37.07	$ 38.18	$ 36.07	$ 38.96	$ -	$ -	$ -	$ -	$ -
Total Cost per Ounce	$ 1,030.61	$ -	$ 993.88	$ 837.51	$ 1,152.04	$ 1,048.33	$ 1,002.11	$ 777.53	$ 861.46	$ 777.53	$ 846.66	$ 855.48	$ 955.10	$ 955.10	$ 982.78	$ 903.91	$ 736.23	$ 286.93	$ -	$ -	$ -	$ -
Total Cost per Tonne Ore (incl. Resale)	$ 41.29																					
Total Cost per Ounce (incl. Resale)	$ 973.14																					
Revenue:																						
Mine Call Factor	1.00	-	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	-	-	-	-
Capital Equipment Resale/Salvage	$ 8,291,104		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8,291,104	$ -	$ -	$ -	$ -
Asset Resale	$ 18,354,167	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 18,354,167	$ -	$ -	$ -
Gold Revenue	$ 602,745,000	$ -	$ 11,947,000	$ 24,245,000	$ 36,192,000	$ 36,192,000	$ 36,192,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 37,440,000	$ 49,257,000	$ 74,984,000	$ -	$ -	$ -	$ -
Marketing & Royalties																						
Freight/Transport	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Refining	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Royalties	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Marketing Costs	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Revenue Before Tax	$ 629,390,272	$ -	$ 11,947,000	$ 24,245,000	$ 36,192,000	$ 36,192,000	$ 36,192,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 37,440,000	$ 49,257,000	$ 83,275,104	$ 18,354,167	$ -	$ -	$ -
Cumulative Revenue		$ -	$ 11,947,000	$ 36,192,000	$ 72,384,000	$ 108,576,000	$ 144,768,000	$ 182,000,000	$ 219,232,000	$ 256,464,000	$ 293,696,000	$ 330,538,000	$ 367,380,000	$ 404,222,000	$ 441,064,000	$ 478,504,000	$ 527,761,000	$ 611,036,104	$ 629,390,272	$ -	$ -	$ -
Annual Revenue before Tax	$ 629,390,272	$ -	$			108,576,000	$			147,888,000	$			147,758,000	$			206,814,104	$			18,354,167
Annualised Cumulative Revenue		$ -	$			108,576,000	$			256,464,000	$			404,222,000	$			611,036,104	$			629,390,272
Quarterly Cash Flow	$ 151,547,532	($ 93,764,968)	$ 2,813,221	$ 8,625,356	$ 4,119,131	$ 7,006,527	$ 8,293,370	$ 14,963,534	$ 12,559,754	$ 14,963,534	$ 12,983,676	$ 12,597,636	$ 9,774,555	$ 9,774,555	$ 8,990,081	$ 11,407,369	$ 21,361,244	$ 66,724,791	$ 18,354,167	$ -	$ -	$ -
Cumulative Cash Flow		($ 93,764,968)	($ 90,951,746)	($ 82,326,391)	($ 78,207,260)	($ 71,200,733)	($ 62,907,363)	($ 47,943,829)	($ 35,384,076)	($ 20,420,542)	($ 7,436,866)	$ 5,160,770	$ 14,935,325	$ 24,709,879	$ 33,699,961	$ 45,107,330	$ 66,468,574	$ 133,193,364	$ 151,547,532	$ -	$ -	$ -
Annualised Cashflow	$ 151,547,532	($ 93,764,968)	$			22,564,235	$			50,780,191	$			45,130,421	$			108,483,485	$			18,354,167
Annualised Cumulative Cashflow		($ 93,764,968)	($			71,200,733)	($			20,420,542)	$			24,709,879	$			133,193,364	$			151,547,532
Yearly NPV @	$ 91,407,216																					
8%																						
Yearly IRR	38.0%																					
Estimated Dates for Project (Assuming Start Date)		30/06/2014	30/09/2015	31/12/2015	31/03/2016	30/06/2016	30/09/2016	31/12/2016	31/03/2017	30/06/2017	30/09/2017	31/12/2017	31/03/2018	30/06/2018	30/09/2018	31/12/2018	31/03/2019	30/06/2019	30/09/2019	31/12/2019	31/03/2020	30/06/2020
Period NPV @	$ 88,666,062																					
8%																						
Period IRR	29.5%																					

Figure B-1 - Cashflow Model - Option 484 (8,000tpd Contractor-Mining)

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Mined Ore Tonnes	10,927,500	240,900	489,100	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,100	730,100	730,100	730,100	729,400	721,700	716,000	-	-	-	-	-
Mined Au Grade	1.70	1.53	1.53	1.53	1.53	1.53	1.58	1.58	1.58	1.58	1.56	1.56	1.56	1.56	1.59	2.11	3.24	-	-	-	-	-
Mined Au Ounces	598,830	11,870	24,090	35,960	35,960	35,960	36,990	36,990	36,990	36,990	36,600	36,600	36,600	36,600	37,190	48,940	74,500	-	-	-	-	-
Cumulative Mined Ore Tonnes		240,900	730,000	1,460,000	2,190,000	2,920,000	3,650,000	4,380,000	5,110,000	5,840,000	6,570,100	7,300,200	8,030,300	8,760,400	9,489,800	10,211,500	10,927,500	-	-	-	-	-
Cumulative Mined Au Grade		1.53	1.53	1.53	1.53	1.53	1.54	1.55	1.55	1.55	1.55	1.56	1.56	1.56	1.56	1.60	1.70	-	-	-	-	-
Cumulative Mined Au Ounces		11,870	35,960	71,920	107,880	143,840	180,830	217,820	254,810	291,800	328,400	365,000	401,600	438,200	475,390	524,330	598,830	-	-	-	-	-
Processed Ore Tonnes	10,927,500		240,900	489,100	730,000	730,000	730,000	730,000	730,000	730,000	730,000	730,100	730,100	730,100	730,100	729,400	721,700	716,000	-	-	-	-
Recovered Au Grade	1.32		1.19	1.19	1.19	1.19	1.19	1.22	1.22	1.22	1.22	1.21	1.21	1.21	1.21	1.23	1.63	2.51	-	-	-	-
Recovered Au Ounces	463,650		9,190	18,650	27,840	27,840	27,840	28,640	28,640	28,640	28,640	28,340	28,340	28,340	28,340	28,800	37,890	57,680	-	-	-	-
Cumulative Processed Ore Tonnes		-	240,900	730,000	1,460,000	2,190,000	2,920,000	3,650,000	4,380,000	5,110,000	5,840,000	6,570,100	7,300,200	8,030,300	8,760,400	9,489,800	10,211,500	10,927,500	-	-	-	-
Cumulative Recovered Au Grade		-	1.19	1.19	1.19	1.19	1.19	1.19	1.20	1.20	1.20	1.20	1.20	1.20	1.20	1.21	1.24	1.32	-	-	-	-
Cumulative Recovered Au Ounces		-	9,190	27,840	55,680	83,520	111,360	140,000	168,640	197,280	225,920	254,260	282,600	310,940	339,280	368,080	405,970	463,650	-	-	-	-
Waste Volume			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Tonnes	18,569,000	118,100	239,900	358,000	358,000	358,000	922,600	922,600	922,600	922,600	1,650,100	1,650,100	1,650,100	1,650,100	1,813,000	2,320,900	2,712,300	-	-	-	-	-
Cumulative Waste Tonnes		118,100	358,000	716,000	1,074,000	1,432,000	2,354,600	3,277,200	4,199,800	5,122,400	6,772,500	8,422,600	10,072,700	11,722,800	13,535,800	15,856,700	18,569,000	-	-	-	-	-
Strip Ratio	1.70	0.49	0.49	0.49	0.49	0.49	1.26	1.26	1.26	1.26	2.26	2.26	2.26	2.26	2.49	3.22	3.79	-	-	-	-	-
Cumulative Strip Ratio		0.49	0.49	0.49	0.49	0.49	0.65	0.75	0.82	0.88	1.03	1.15	1.25	1.34	1.43	1.55	1.70	-	-	-	-	-
Capital Costs:																						
Capital Development/Pre-stripping																						
Capital Costs Relative to Option																						
Capital Cost - Mining	$ 4,304,495	$ 4,304,495																				
Capital Cost - Processing (Main)	$ 58,547,853	$ 58,547,853																				
Capital Cost - Processing (Addn CIL Circuit)	$ -																					
Capital Cost - Processing (Heap Leach)	$ -																					
Capital Cost - Transport	$ -	$ -																				
Capital Cost - Other	$ 60,205,555	$ 29,087,341	$ -	$ -	$ 11,356,641	$ 8,469,245	$ 5,646,164	$ -	$ -	$ -	$ -	$ -	$ 2,823,082	$ 2,823,082	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Rehabilitation (Stage 1&2)	$ 7,160,750		$ -	$ -	$ -	$ -	$ -	$ -	$ 2,403,780	$ -	$ -	$ -	$ -	$ -	$ 3,166,970	$ -	$ 795,000	$ 795,000	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Process	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Land Acqisition	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - TSF & Other	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Stage 3 - Rehabilitation	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Cost - Condemnation/Resource Drilling	$ 180,000	$ 180,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Annual Sustaining Capital	$ 4,479,262		$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ -	$ -	$ -	$ -	$ -
Total Capital Costs	$ 134,877,915	$ 92,119,690	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 3,121,699	$ 3,465,587	$ 298,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -	$ -
Cumulative Capital Costs		$ 92,119,690	$ 92,418,307	$ 92,716,925	$ 104,372,183	$ 113,140,046	$ 119,084,827	$ 119,383,444	$ 122,085,842	$ 122,384,459	$ 122,683,077	$ 122,981,694	$ 126,103,394	$ 129,225,093	$ 132,690,680	$ 132,989,298	$ 134,082,915	$ 134,877,915	$ -	$ -	$ -	$ -
Operating Costs:																						
Mining - Waste	$ 50,525,000	$ 321,342	$ 652,752	$ 974,094	$ 974,094	$ 974,094	$ 2,510,333	$ 2,510,333	$ 2,510,333	$ 2,510,333	$ 4,489,811	$ 4,489,811	$ 4,489,811	$ 4,489,811	$ 4,933,051	$ 6,315,013	$ 7,379,986	$ -	$ -	$ -	$ -	$ -
Mining - Ore	$ 33,862,504	$ 746,509	$ 1,515,640	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,149	$ 2,262,458	$ 2,262,458	$ 2,262,458	$ 2,262,458	$ 2,260,289	$ 2,236,428	$ 2,218,765	$ -	$ -	$ -	$ -	$ -
Labour (Mine Overhead)	$ 18,872,319	$ 410,268	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ 1,230,803	$ -	$ -	$ -	$ -	$ -
General Costs	$ 584,299	$ 12,881	$ 26,152	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,033	$ 39,039	$ 39,039	$ 39,039	$ 39,039	$ 39,001	$ 38,590	$ 38,285	$ -	$ -	$ -	$ -	$ -
Engineering Costs	$ 988,097	$ 21,783	$ 44,226	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,009	$ 66,018	$ 66,018	$ 66,018	$ 66,018	$ 65,955	$ 65,258	$ 64,743	$ -	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Main)	$ 196,325,629	$ -	$ 4,307,431	$ 8,745,221	$ 13,052,652	$ 13,052,652	$ 13,052,652	$ 13,068,652	$ 13,068,652	$ 13,068,652	$ 13,068,652	$ 13,064,363	$ 13,064,363	$ 13,064,363	$ 13,064,363	$ 13,061,581	$ 13,111,576	$ 13,409,805	$ -	$ -	$ -	$ -
Metallurgical/Processing Costs (Addn CIL)	$ -																					
Metallurgical/Processing Costs (Heap Leach)	$ -																					
Transport Cost to Central Plant/Port	$ 35,796,851	$ -	$ 789,152	$ 1,602,218	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,371	$ 2,391,698	$ 2,391,698	$ 2,391,698	$ 2,391,698	$ 2,389,405	$ 2,364,181	$ 2,345,509	$ -	$ -	$ -	$ -
General Overhead (BESRA)	$ 6,010,125	$ 132,495	$ 269,005	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,500	$ 401,555	$ 401,555	$ 401,555	$ 401,555	$ 401,170	$ 396,935	$ 393,800	$ -	$ -	$ -	$ -	$ -
Total Operating Costs	$ 342,964,825	$ 1,645,278	$ 8,835,161	$ 15,321,027	$ 20,417,610	$ 20,417,610	$ 21,953,849	$ 21,969,849	$ 21,969,849	$ 21,969,849	$ 23,949,707	$ 23,945,746	$ 23,945,746	$ 23,945,746	$ 24,386,331	$ 25,734,013	$ 26,802,138	$ 15,755,314	$ -	$ -	$ -	$ -
Operating Cost per Tonne Ore	$ 31.39	$ 6.83	$ 18.06	$ 20.99	$ 27.97	$ 27.97	$ 30.07	$ 30.10	$ 30.10	$ 30.10	$ 32.80	$ 32.80	$ 32.80	$ 32.80	$ 33.43	$ 35.66	$ 37.43	$ -	$ -	$ -	$ -	$ -
Cumulative Operating Costs		$ 1,645,278	$ 10,480,439	$ 25,801,466	$ 46,219,077	$ 66,636,687	$ 88,590,536	$ 110,560,385	$ 132,530,234	$ 154,500,083	$ 178,449,789	$ 202,395,536	$ 226,341,282	$ 250,287,028	$ 274,673,359	$ 300,407,372	$ 327,209,511	$ 342,964,825	$ -	$ -	$ -	$ -
Total Costs:																						
Total Costs	$ 477,842,740	$ 93,764,968	$ 9,133,779	$ 15,619,644	$ 32,072,869	$ 29,185,473	$ 27,898,630	$ 22,268,466	$ 24,672,246	$ 22,268,466	$ 24,248,324	$ 24,244,364	$ 27,067,445	$ 27,067,445	$ 27,851,919	$ 26,032,631	$ 27,895,756	$ 16,550,314	$ -	$ -	$ -	$ -
Total Cumulative Costs		$ 93,764,968	$ 102,898,746	$ 118,518,391	$ 150,591,260	$ 179,776,733	$ 207,675,363	$ 229,943,829	$ 254,616,076	$ 276,884,542	$ 301,132,866	$ 325,377,230	$ 352,444,675	$ 379,512,121	$ 407,364,039	$ 433,396,670	$ 461,292,426	$ 477,842,740	$ -	$ -	$ -	$ -
Total Cost per Tonne Ore	$ 43.73	$ 389.23	$ 18.67	$ 21.40	$ 43.94	$ 39.98	$ 38.22	$ 30.50	$ 33.80	$ 30.50	$ 33.21	$ 33.21	$ 37.07	$ 37.07	$ 38.18	$ 36.07	$ 38.96	$ -	$ -	$ -	$ -	$ -
Total Cost per Ounce	$ 1,030.61	$ -	$ 993.88	$ 837.51	$ 1,152.04	$ 1,048.33	$ 1,002.11	$ 777.53	$ 861.46	$ 777.53	$ 846.66	$ 855.48	$ 955.10	$ 955.10	$ 982.78	$ 903.91	$ 736.23	$ 286.93	$ -	$ -	$ -	$ -
Total Cost per Tonne Ore (incl. Resale)	$ 41.29																					
Total Cost per Ounce (incl. Resale)	$ 973.14																					
Revenue:																						
Mine Call Factor	1.00	-	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	-	-	-	-
Capital Equipment Resale/Salvage	$ 8,291,104		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8,291,104	$ -	$ -	$ -	$ -
Asset Resale	$ 18,354,167	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 18,354,167	$ -	$ -	$ -
Gold Revenue	$ 602,745,000	$ -	$ 11,947,000	$ 24,245,000	$ 36,192,000	$ 36,192,000	$ 36,192,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 37,440,000	$ 49,257,000	$ 74,984,000	$ -	$ -	$ -	$ -
Marketing & Royalties																						
Freight/Transport	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Refining	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Royalties	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Marketing Costs	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Revenue Before Tax	$ 629,390,272	$ -	$ 11,947,000	$ 24,245,000	$ 36,192,000	$ 36,192,000	$ 36,192,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 37,232,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 36,842,000	$ 37,440,000	$ 49,257,000	$ 83,275,104	$ 18,354,167	$ -	$ -	$ -
Cumulative Revenue		$ -	$ 11,947,000	$ 36,192,000	$ 72,384,000	$ 108,576,000	$ 144,768,000	$ 182,000,000	$ 219,232,000	$ 256,464,000	$ 293,696,000	$ 330,538,000	$ 367,380,000	$ 404,222,000	$ 441,064,000	$ 478,504,000	$ 527,761,000	$ 611,036,104	$ 629,390,272	$ -	$ -	$ -
Annual Revenue before Tax	$ 629,390,272	$ -				$ 108,576,000				$ 147,888,000				$ 147,758,000				$ 206,814,104				$ 18,354,167
Annualised Cumulative Revenue		$ -				$ 108,576,000				$ 256,464,000				$ 404,222,000				$ 611,036,104				$ 629,390,272
Quarterly Cash Flow	$ 151,547,532	($ 93,764,968)	$ 2,813,221	$ 8,625,356	$ 4,119,131	$ 7,006,527	$ 8,293,370	$ 14,963,534	$ 12,559,754	$ 14,963,534	$ 12,983,676	$ 12,597,636	$ 9,774,555	$ 9,774,555	$ 8,990,081	$ 11,407,369	$ 21,361,244	$ 66,724,791	$ 18,354,167	$ -	$ -	$ -
Cumulative Cash Flow		($ 93,764,968)	($ 90,951,746)	($ 82,326,391)	($ 78,207,260)	($ 71,200,733)	($ 62,907,363)	($ 47,943,829)	($ 35,384,076)	($ 20,420,542)	($ 7,436,866)	$ 5,160,770	$ 14,935,325	$ 24,709,879	$ 33,699,961	$ 45,107,330	$ 66,468,574	$ 133,193,364	$ 151,547,532	$ -	$ -	$ -
Annualised Cashflow	$ 151,547,532	($ 93,764,968)				$ 22,564,235				$ 50,780,191				$ 45,130,421				$ 108,483,485				$ 18,354,167
Annualised Cumulative Cashflow		($ 93,764,968)				($ 71,200,733)				($ 20,420,542)				$ 24,709,879				$ 133,193,364				$ 151,547,532
Yearly NPV @ 8%	$ 91,407,216																					
Yearly IRR	38.0%																					
Estimated Dates for Project (Assuming Start Date)		30/06/14	30/09/15	31/12/15	31/03/16	30/06/16	30/09/16	31/12/16	31/03/17	30/06/17	30/09/17	31/12/17	31/03/18	30/06/18	30/09/18	31/12/18	31/03/19	30/06/19	30/09/19	31/12/19	31/03/20	30/06/20
Period NPV @ 8%	$ 88,666,062																					
Period IRR	29.5%																					

Figure B-2 - Cashflow Model - Option 452 (8,000tpd Owner-Operator)

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Stage		*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*2*	*2*	*0*	*0*	*0*	*0*	*0*
Tax - No Incentives																						
Opening Capital/Deferred Exploration		$ 7,500,000																				
Capital Expenditure (Stage 1&2)	*$ 134,877,915*	$ 92,119,690	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 3,121,699	$ 3,465,587	$ 298,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -	$ -
Capital Expenditure (Stage 3)	*$ -*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Non-Depreciable Capital	*$ 22,942,709*	$ 22,942,709	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Depreciable Capital		$ 99,619,690	$ 99,918,307	$ 93,972,030	$ 99,362,487	$ 101,033,029	$ 99,206,039	$ 91,237,486	$ 85,645,567	$ 77,379,628	$ 69,080,509	$ 60,744,063	$ 55,188,039	$ 49,111,731	$ 42,754,973	$ 32,364,847	$ 22,670,182	$ 12,130,091	$ -	$ -	$ -	$ -
Depreciation	142,377,915	$ -	$ 6,244,894	$ 6,264,802	$ 7,097,320	$ 7,771,771	$ 8,267,170	$ 8,294,317	$ 8,564,557	$ 8,597,736	$ 8,635,064	$ 8,677,723	$ 9,198,006	$ 9,822,346	$ 10,688,743	$ 10,788,282	$ 11,335,091	$ 12,130,091	$ -	$ -	$ -	$ -
Depreciation Quarters			16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-	-	-	-
Closing Depreciable Capital		**$ 99,619,690**	**$ 93,673,413**	**$ 87,707,228**	**$ 92,265,166**	**$ 93,261,258**	**$ 90,938,869**	**$ 82,943,169**	**$ 77,081,010**	**$ 68,781,891**	**$ 60,445,445**	**$ 52,066,339**	**$ 45,990,032**	**$ 39,289,385**	**$ 32,066,229**	**$ 21,576,565**	**$ 11,335,091**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**
Disposal of Plant & Equipment	*$ 8,291,104*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 8,291,104*	*$ -*	*$ -*	*$ -*	*$ -*
Tax Profit/(loss) from Disposal of P&E	*$ 8,291,104*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 8,291,104*	*$ -*	*$ -*	*$ -*	*$ -*
Proceeds from Disposal of Non-Depreciable Capital (land)	*$ 18,354,167*		*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 18,354,167*	*$ -*	*$ -*	*$ -*
Write-Off @ Project End			*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 4,588,542*	*$ -*	*$ -*
Balance of Non-Depreciable Capital (after disposal)			**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 4,588,542**	**$ -**	**$ -**	**$ -**
Debt Drawdowns	*$ 64,483,783*	$ 64,483,783																				
Interest Counter		1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Quarters Production		-	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	-	-	-	-
Principal Repayment Sweep %			0.00%	0.00%	0.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	10.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Principal Repayments	*($ 64,483,783)*		$ -	$ -	$ -	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	($ 6,448,378)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Debt Outstanding		$ 64,483,783	$ 64,483,783	$ 64,483,783	$ 64,483,783	$ 58,035,404	$ 51,587,026	$ 45,138,648	$ 38,690,270	$ 32,241,891	$ 25,793,513	$ 19,345,135	$ 12,896,757	$ 6,448,378	$ 0	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Interest	*($ 13,283,659)*	($ 3,611,092)	($ 1,289,676)	($ 1,289,676)	($ 1,289,676)	($ 1,160,708)	($ 1,031,741)	($ 902,773)	($ 773,805)	($ 644,838)	($ 515,870)	($ 386,903)	($ 257,935)	($ 128,968)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
EBITDA	*$ 259,780,175*	($ 1,645,278)	$ 3,111,839	$ 8,923,973	$ 15,774,390	$ 15,774,390	$ 14,238,151	$ 15,262,151	$ 15,262,151	$ 15,262,151	$ 13,282,293	$ 12,896,254	$ 12,896,254	$ 12,896,254	$ 12,455,669	$ 11,705,987	$ 22,454,862	$ 59,228,686	$ -	$ -	$ -	$ -
Depreciation	*($ 142,377,915)*	$ -	($ 6,244,894)	($ 6,264,802)	($ 7,097,320)	($ 7,771,771)	($ 8,267,170)	($ 8,294,317)	($ 8,564,557)	($ 8,597,736)	($ 8,635,064)	($ 8,677,723)	($ 9,198,006)	($ 9,822,346)	($ 10,688,743)	($ 10,788,282)	($ 11,335,091)	($ 12,130,091)	$ -	$ -	$ -	$ -
Interest	*($ 13,283,659)*	($ 3,611,092)	($ 1,289,676)	($ 1,289,676)	($ 1,289,676)	($ 1,160,708)	($ 1,031,741)	($ 902,773)	($ 773,805)	($ 644,838)	($ 515,870)	($ 386,903)	($ 257,935)	($ 128,968)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of P&E	*$ 8,291,104*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8,291,104	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of Land	*$ -*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Earnings	*$ 112,409,705*	($ 5,256,370)	($ 4,422,731)	$ 1,369,495	$ 7,387,394	$ 6,841,910	$ 4,939,241	$ 6,065,061	$ 5,923,789	$ 6,019,577	$ 4,131,359	$ 3,831,628	$ 3,440,312	$ 2,944,940	$ 1,766,926	$ 917,704	$ 11,119,771	$ 55,389,699	$ -	$ -	$ -	$ -
Carry Forward Losses (if applicable)		($ 5,256,370)	($ 9,679,101)	($ 8,309,606)	($ 922,212)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Adjusted Taxable Earnings		**$ -**	**$ -**	**$ -**	**$ -**	**$ 5,919,698**	**$ 4,939,241**	**$ 6,065,061**	**$ 5,923,789**	**$ 6,019,577**	**$ 4,131,359**	**$ 3,831,628**	**$ 3,440,312**	**$ 2,944,940**	**$ 1,766,926**	**$ 917,704**	**$ 11,119,771**	**$ 55,389,699**	**$ -**	**$ -**	**$ -**	**$ -**
Tax with No Incentives	*($ 26,978,329)*	$ -	$ -	$ -	$ -	($ 1,420,728)	($ 1,185,418)	($ 1,455,615)	($ 1,421,709)	($ 1,444,698)	($ 991,526)	($ 919,591)	($ 825,675)	($ 706,786)	($ 424,062)	($ 220,249)	($ 2,668,745)	($ 13,293,528)	$ -	$ -	$ -	$ -
After Tax Cashflow	*111,285,543*	*($ 32,892,277)*	*$ 1,523,546*	*$ 7,335,680*	*$ 2,829,455*	*($ 2,023,287)*	*($ 372,167)*	*$ 6,156,768*	*$ 3,915,861*	*$ 6,425,619*	*$ 5,027,901*	*$ 4,842,765*	*$ 2,242,566*	*$ 2,490,423*	*$ 2,117,641*	*$ 11,187,120*	*$ 18,692,499*	*$ 53,431,263*	*$ 18,354,167*	*$ -*	*$ -*	*$ -*
Cumulative After Tax Cashflow		*($ 32,892,277)*	*($ 31,368,731)*	*($ 24,033,051)*	*($ 21,203,596)*	*($ 23,226,883)*	*($ 23,599,049)*	*($ 17,442,282)*	*($ 13,526,421)*	*($ 7,100,802)*	*($ 2,072,901)*	*$ 2,769,864*	*$ 5,012,430*	*$ 7,502,853*	*$ 9,620,494*	*$ 20,807,614*	*$ 39,500,113*	*$ 92,931,376*	*$ 111,285,543*	*$ -*	*$ -*	*$ -*
Annualised Cashflow	*111,285,543*	*($ 32,892,277)*				*$ 9,665,394*				*$ 16,126,081*				*$ 14,603,655*				*$ 85,428,523*				*$ 18,354,167*
		-$ 93,764,968				*$ 21,143,507*				*$ 45,272,751*				*$ 41,686,844*				*$ 91,876,901*				*$ 18,354,167*
Yearly NPV: Tax No Incentive	**$ 71,983,893**																					
8%																						
Yearly IRR: Tax No Incentive	32.6%																					

Table B-3 - After-Tax Cashflow Model - Option 484 (8,000tpd Contractor-Mining)

Cashflow Item	Totals	Yr -1	Yr 1				Yr 2				Yr 3				Yr 4				Yr 5			
		Pre-Mining	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4	Qtr1	Qtr2	Qtr3	Qtr4
Stage		*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*1*	*2*	*2*	*2*	*0*	*0*	*0*	*0*
Tax - No Incentives																						
Opening Capital/Deferred Exploration		$ 7,500,000																				
Capital Expenditure (Stage 1&2)	*$ 156,166,540*	$ 112,314,696	$ 298,617	$ 298,617	$ 11,655,259	$ 8,767,863	$ 5,944,781	$ 298,617	$ 2,702,397	$ 298,617	$ 298,617	$ 298,617	$ 298,617	$ 3,121,699	$ 6,288,669	$ 298,617	$ 1,093,617	$ 1,093,617	$ 795,000	$ -	$ -	$ -
Capital Expenditure (Stage 3)	*$ -*	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Non-Depreciable Capital	*$ 22,942,709*	$ 22,942,709	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Depreciable Capital		$ 119,814,696	$ 120,113,314	$ 113,346,442	$ 117,917,548	$ 118,824,241	$ 116,281,576	$ 107,635,457	$ 101,368,233	$ 92,451,557	$ 83,505,019	$ 74,525,301	$ 65,508,256	$ 59,271,633	$ 55,681,696	$ 44,843,975	$ 34,726,598	$ 24,244,683	$ 12,917,342	$ -	$ -	$ -
Depreciation	163,666,540	$ -	$ 7,065,489	$ 7,084,153	$ 7,861,170	$ 8,487,446	$ 8,944,737	$ 8,969,621	$ 9,215,294	$ 9,245,156	$ 9,278,335	$ 9,315,663	$ 9,358,322	$ 9,878,605	$ 11,136,339	$ 11,210,994	$ 11,575,533	$ 12,122,342	$ 12,917,342	$ -	$ -	$ -
Depreciation Quarters			17	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-	-	-
Closing Depreciable Capital		**$ 119,814,696**	**$ 113,047,825**	**$ 106,262,290**	**$ 110,056,378**	**$ 110,336,795**	**$ 107,336,840**	**$ 98,665,836**	**$ 92,152,939**	**$ 83,206,401**	**$ 74,226,683**	**$ 65,209,638**	**$ 56,149,933**	**$ 49,393,027**	**$ 44,545,357**	**$ 33,632,981**	**$ 23,151,066**	**$ 12,122,342**	**$ -**	**$ -**	**$ -**	**$ -**
Disposal of Plant & Equipment	*$ 12,330,106*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 12,330,106*	*$ -*	*$ -*	*$ -*
Tax Profit/(loss) from Disposal of P&E	*$ 12,330,106*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 12,330,106*	*$ -*	*$ -*	*$ -*
Proceeds from Disposal of Non-Depreciable Capital (land)	*$ 18,354,167*		*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 18,354,167*	*$ -*	*$ -*
Write-Off @ Project End			*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ -*	*$ 4,588,542*	*$ -*
Balance of Non-Depreciable Capital (after disposal)			**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 22,942,709**	**$ 4,588,542**	**$ -**	**$ -**
Debt Drawdowns	*$ 78,620,288*	$ 78,620,288																				
Interest Counter		1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Quarters Production		-	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	-	-	-
Principal Repayment Sweep %			0.00%	0.00%	0.00%	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	9.09%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Principal Repayments	*($ 78,620,288)*		$ -	$ -	$ -	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	($ 7,147,299)	$ -	$ -	$ -	$ -	$ -	$ -
Debt Outstanding		$ 78,620,288	$ 78,620,288	$ 78,620,288	$ 78,620,288	$ 71,472,989	$ 64,325,690	$ 57,178,391	$ 50,031,092	$ 42,883,793	$ 35,736,494	$ 28,589,195	$ 21,441,897	$ 14,294,598	$ 7,147,299	$ 0	$ -	$ -	$ -	$ -	$ -	$ -
Interest	*($ 16,981,982)*	($ 4,402,736)	($ 1,572,406)	($ 1,572,406)	($ 1,572,406)	($ 1,429,460)	($ 1,286,514)	($ 1,143,568)	($ 1,000,622)	($ 857,676)	($ 714,730)	($ 571,784)	($ 428,838)	($ 285,892)	($ 142,946)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -
EBITDA	*$ 292,880,457*	($ 1,238,988)	$ 3,933,149	$ 10,175,134	$ 17,021,472	$ 17,021,472	$ 16,051,925	$ 16,755,361	$ 16,755,361	$ 16,755,361	$ 15,571,942	$ 15,282,184	$ 15,282,184	$ 15,282,184	$ 14,719,818	$ 15,122,031	$ 10,617,662	$ 44,909,689	$ 32,862,514	$ -	$ -	$ -
Depreciation	*($ 163,666,540)*	$ -	($ 7,065,489)	($ 7,084,153)	($ 7,861,170)	($ 8,487,446)	($ 8,944,737)	($ 8,969,621)	($ 9,215,294)	($ 9,245,156)	($ 9,278,335)	($ 9,315,663)	($ 9,358,322)	($ 9,878,605)	($ 11,136,339)	($ 11,210,994)	($ 11,575,533)	($ 12,122,342)	($ 12,917,342)	$ -	$ -	$ -
Interest	*($ 16,981,982)*	($ 4,402,736)	($ 1,572,406)	($ 1,572,406)	($ 1,572,406)	($ 1,429,460)	($ 1,286,514)	($ 1,143,568)	($ 1,000,622)	($ 857,676)	($ 714,730)	($ 571,784)	($ 428,838)	($ 285,892)	($ 142,946)	($ 0)	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of P&E	*$ 12,330,106*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 12,330,106	$ -	$ -	$ -
Taxable Profits/(Losses) on Disposal of Land	*$ -*		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Taxable Earnings	*$ 124,562,041*	($ 5,641,724)	($ 4,704,746)	$ 1,518,576	$ 7,587,897	$ 7,104,567	$ 5,820,675	$ 6,642,172	$ 6,539,445	$ 6,652,530	$ 5,578,876	$ 5,394,738	$ 5,495,024	$ 5,117,687	$ 3,440,532	$ 3,911,038	($ 957,871)	$ 32,787,347	$ 32,275,278	$ -	$ -	$ -
Carry Forward Losses (if applicable)		($ 5,641,724)	($ 10,346,470)	($ 8,827,894)	($ 1,239,997)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	($ 957,871)	$ -	$ -	$ -	$ -	$ -
Adjusted Taxable Earnings		**$ -**	**$ -**	**$ -**	**$ -**	**$ 5,864,569**	**$ 5,820,675**	**$ 6,642,172**	**$ 6,539,445**	**$ 6,652,530**	**$ 5,578,876**	**$ 5,394,738**	**$ 5,495,024**	**$ 5,117,687**	**$ 3,440,532**	**$ 3,911,038**	**$ -**	**$ 31,829,476**	**$ 32,275,278**	**$ -**	**$ -**	**$ -**
Tax with No Incentives	*($ 29,894,890)*	$ -	$ -	$ -	$ -	($ 1,407,497)	($ 1,396,962)	($ 1,594,121)	($ 1,569,467)	($ 1,596,607)	($ 1,338,930)	($ 1,294,737)	($ 1,318,806)	($ 1,228,245)	($ 825,728)	($ 938,649)	$ -	($ 7,639,074)	($ 7,746,067)	$ -	$ -	$ -
After Tax Cashflow	*120,521,318*	*($ 39,336,133)*	*$ 2,062,126*	*$ 8,304,111*	*$ 3,793,808*	*($ 1,730,646)*	*$ 276,370*	*$ 6,571,756*	*$ 4,335,576*	*$ 6,855,162*	*$ 6,072,365*	*$ 5,969,747*	*$ 6,088,624*	*$ 3,499,049*	*$ 315,176*	*$ 6,737,466*	*$ 9,524,045*	*$ 36,176,997*	*$ 36,651,553*	*$ 18,354,167*	*$ -*	*$ -*
Cumulative After Tax Cashflow		*($ 39,336,133)*	*($ 37,274,007)*	*($ 28,969,896)*	*($ 25,176,088)*	*($ 26,906,734)*	*($ 26,630,365)*	*($ 20,058,609)*	*($ 15,723,033)*	*($ 8,867,871)*	*($ 2,795,506)*	*$ 3,174,241*	*$ 9,262,866*	*$ 12,761,915*	*$ 13,077,091*	*$ 19,814,557*	*$ 29,338,601*	*$ 65,515,598*	*$ 102,167,151*	*$ 120,521,318*	*$ -*	*$ -*
Annualised Cashflow	*120,521,318*	*($ 39,336,133)*				*$ 12,429,399*				*$ 18,038,863*				*$ 21,629,786*				*$ 52,753,683*				*$ 55,005,720*
		-$ 113,553,684				*$ 25,723,374*				*$ 50,916,438*				*$ 52,220,225*				*$ 67,191,227*				*$ 55,005,720*
Yearly NPV: Tax No Incentive	**$ 76,106,036**																					
8%																						
Yearly IRR: Tax No Incentive	29.4%																					

Table B-4 - After-Tax Cashflow Model - Option 452 (8,000tpd Owner-Operator)

B22-2. Project Risk Register

No.	Risk Group	Risk Description	Description of Consequence or Impact	Probability	Consequence	Score	Estimated Cost Impact	Mitigation Measures
1	Processing/Plant	Low Concentrate Grade	Concentrate grade too low	HIGH	HIGH	6	>$5million	Test existing and new metallurgical processes with a focus on slimes removal and flotation technologies (flash flotation, arsenopyrite/pyrite separation, ultrasonics). Develop testwork programme to gain understanding of deposit geometallurgy. Utilise information in design of plant to optimise concentrate grade.
2	Processing/Plant	Concentrate Specs Not Met	Concentrate produced does not meet the required specs of processor/smelter	HIGH	HIGH	6	>$5million	As above.
3	Processing/Plant	Clay in Ore	Clay affecting mining, crushing and processing of ore; do we require roll crusher before jaw crusher	HIGH	HIGH	6	>$5million	Investigate existing and new metallurgical processes for slimes removal and clay mitigation. Plant design to account for high clay content. Develop testwork programme to gain understanding of deposit geometallurgy.
4	Processing/Plant	More Metallurgical Testwork Required for Economics	More detailed testwork required help define requirements for plant and associated costs or economics	HIGH	MEDIUM	5	>$1million	Develop project testwork programme to gain understanding of deposit geometallurgy using both in-house and external expertise. Assign budget to geometallurgical programme. Utilise test results in plant design and project economics.
5	Processing/Plant	Metallurgical Characteristics Incomplete for Design	Incomplete understanding of metallurgical charactersitics	HIGH	MEDIUM	5	>$5million	Design project testwork programme to gain understanding of deposit geometallurgy and incorporate results into plant design. Assign budget to geometallurgical programme. Utilise test results in plant design and project economics.
6	Construction & Implementation	Construction/Commission Delays	Delays in construction and/or commissioning schedule	MEDIUM	HIGH	5	>$5million	Incorporate both penalties and bonuses into construction contracts to discourage delays.
7	Geotechnical	Pit Slope Instability/Failure(s)	Pit instability or failures affecting pit production	MEDIUM	HIGH	5	>$5million	Measure geotechnical properties of orebody. Incorporate these measurements into mine design.
8	Geotechnical	Landform/Slope Stability or Failure	Landform instability or failures affecting the TSF, waste dump and plant/infrastructure	MEDIUM	HIGH	5	>$1million	As above.
9	Processing/Plant	Low Concentrate Recovery	Concentrate recovery too low	MEDIUM	HIGH	5	>$1million	Test existing and new metallurgical processes with a focus on slimes removal and flotation technologies (flash flotation, arsenopyrite/pyrite separation, ultrasonics). Develop testwork programme to gain understanding of deposit geometallurgy. Utilise information in design of plant to maximise concentrate recovery.
10	Processing/Plant	Plant Design Specifications Not Met	Plant operation not meet design specifications	MEDIUM	HIGH	5	>$5million	Design project testwork programme to gain understanding of deposit geometallurgy and incorporate results into plant design. Assign budget to geometallurgical programme. Utilise test results in plant design.
11	Procurement & Capital Items	Delivery Schedule Delay	Delay in capital item delivery or delays due to other impacts (customs, shipping, etc.)	LOW	HIGH	4	>$5million	Order critical items asap. Incorporate penalties and bonuses into delivery contract. Track delivery status on a regular basis.
12	Permits/Approvals	Mining Certificate/Lease Delays	Inability or delays of Gladioli to obtain MC/ML renewal covering part of the mine operational area (mainly TSF and waste landform)	LOW	HIGH	4		Ensure regular and ongoing liaison with Gladioli. Track progress of permits and ensure deadlines are met.'
13	External Factors	Political Change/Government Interference	Changes in the current political situation or interference from government officials	LOW	HIGH	4		Communicate regularly with all parties and promote the project to ensure positive views. Monitor political communications for any negative communications
14	Geology/Resource	Missing or Incomplete Resource Data	Possible missing elements affecting process; Zonation of mineralogical characteristics unknown; Incomplete key element data (particularly S & Fe)	MEDIUM	MEDIUM	4		Ensure data is captured in future drilling and if applicable in any grade control work.
15	Geology/Resource	Oxidised Layer	Impact of partial oxidative layer - amount and volume of oxidated material	MEDIUM	MEDIUM	4		Geological mapping and grade control to monitor the oxidised layer. Track plant performance and recovery.
16	Environmental & Rehab	EIA Delayed/Rejected	Process of obtaining EIA delayed/rejected	LOW	HIGH	4		Ensure EIA basline work is comprehensive enough. That the EIA report and EIA consultant have clearly identified the effects and applied suitable mitigation measures. Track the EIA schedule and timeline closely. Ensure open and clear communications with all parties.

No.	Risk Group	Risk Description	Description of Consequence or Impact	Probability	Consequence	Score	Estimated Cost Impact	Mitigation Measures
17	Environmental & Rehab	MRP Delayed/Rejected	Process of obtaining or acceptance of MRP is delayed or rejected	LOW	HIGH	4		As above.
18	General	Inflationary Impacts	Inflationary effects on pricing due to delays	HIGH	LOW	4	>$1million	Use of hedge instruments. Incorporate inflationary estimates into economic model.
19	Permits/Approvals	Building/Construction Permit Delays	Delays in building/construction permits issued by local government	LOW	HIGH	4		As per EIA, MRP and other government processes
20	Geology/Resource	Lower Average Grade	Resource grade lower on average than in model	MEDIUM	MEDIUM	4		Monitor through geological investigations and grade control
21	Processing/Plant	Plant Operational/Throughput Problems	Problems affecting the plant throughput - bottlenecks, breakdowns, under-performance	MEDIUM	MEDIUM	4	>$5million	Design project testwork programme to gain understanding of deposit geometallurgy and incorporate results into plant design. Assign budget to geometallurgical programme. Utilise test results in plant design.
22	External Factors	Gold Export Rule Change	Increase in current export rates for gold concentrate > 0%	LOW	MEDIUM	3		
23	Environmental & Rehab	Acid Mine Drainage	Leakage or levels above permitted	LOW	MEDIUM	3	>$1million	Containment of PAF material and control of site drainage. Incorporation of lime dosage.
24	Environmental & Rehab	Mine Closure Rehab Delayed/Rejected	Non-acceptance of mine closure rehab or delays due to rectification	LOW	MEDIUM	3		As per EIA, MRP and other government processes
25	Hydrology & Water Management	Severe Weather Events	Impact of severe weather events on the mining operations or other operations (power disruption, flooding preventing staff getting to work, etc.)	MEDIUM	LOW	3	>$100,000	Incorporate weather forecasts in routine operational planning.
26	Finance/Costs	Operating Cost Increases	Increase in some or all of operating costs	LOW	MEDIUM	3	>$1million	Maintain tight control on contract negotiations/costs; minimise unit costs and usage.
27	Mining/Operations	Production Delays	Delays in reaching full/ongoing production	LOW	MEDIUM	3		Regular and detailed project schedule to ensure no delays. Develop alternate options list should delaying events occur ahead of tme to ensure quick remedy
28	General	Major Negative Event	Major event e..g fire, loss of power supply, etc.	LOW	MEDIUM	3		Regular monitoring of all hazards, regular checks and detailed H&S training. Develop a H&S strategy to deal with any incident
29	External Factors	Royalty Rate Increase	Increase in current royalty rate >0%	LOW	MEDIUM	3		Communicate benefits of no increase and constantly monitor government opinion. Develop strategies to mitigate
30	Tailings Facility	Insufficient Waste Material	Insufficient construction material at point in time	LOW	LOW	2		Develop alternate plans and sources of material. Ensure detailed and regular short term planning to ensure no problem with waste aterial balance and supply
31	External Factors	Illegal Miners	Illegal miners stealing gold/ore or impacting operations	LOW	LOW	2	>$10,000	Employ security team to keep deposit secure; regular contact with local police; physical barriers to exclude miners (fence).
32	External Factors	Anti-Mining & Environmental Disruption	Protests or other interference from anti-mining groups or environmental groups	LOW	LOW	2	>$10,000	Regular monitoring of these groups. Good communications strategy to government and local residents. Good security and regular communications with the police.
33	Environmental & Rehab	Excessive Rehabilitation Bond	Excessive rehabilitation bond and restrictive rehab conditions	LOW	LOW	2	>$1million	Design closure plan in accordance with best practice; use of reasonable examples in bond application.
34	Contracts	Contract Conditions Not Met	Contract conditions with service provider not met on consistent basis	LOW	LOW	2	>$100,000	Close contract management - penalties and bonuses to encourage contract obeyance.
35	Contracts	Poor Contractual Terms	Poor, inconsistent or vague contract terms	LOW	LOW	2	>$100,000	Legal review of contract conditions.
36	Transport	Transport Security Issues	Security issues with concentrate transport - theft	LOW	LOW	2	>$100,000	Monitoring of vehicles, personnel and concentrate bags. Good security measures and plans
37	Transport	Transport Disruption	Disruption due to ship unavailability, road issues, truck unavailability, etc.	LOW	LOW	2	>$100,000	Develop a strategy and plan to deal with any disruptions. Ensure suitable equipment, transport, personnel and other options to meet any problems
38	Mining/Operations	Low Mine Production	Various factors impacting the mine production	LOW	LOW	2		Regular planning and operational monitoring to ensure no impacts on mine production
39	General	Labour Issues	Insufficient labour, skills level and training	LOW	LOW	2		Develop a detailed labour, training and HR policy plan. Ensure good instructors and training material available