# ENDEAVOUR MINING CORPORATION

## ITY CIL PROJECT

# NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT


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# EXECUTIVE SUMMARY

Mineral rights for the Ity Gold Project are 100% owned by Société Minière d'Ity (SMI), a company incorporated in Côte d'Ivoire, with the registered address at 08 BP 872 Abidjan 08 - Cocody 180 - logements Boulevard Latrille, Immeuble Palm Club, 2nd floor.

The shareholders of SMI are as follows:

- Endeavour Mining 55%.
- Société pour le Developement Minier de la Côte d'Ivoire (SODEMI) 30%.
- State of Côte d'Ivoire 10%
- Keyman Investments (Didier Drogba Group) 5%.

There are a total of nine prospect areas included in the Ity Gold Project which include four in situ gold deposits that have been or are currently in production including Mont Ity, Ity Flat, ZiaNE and Walter, plus two near mine in situ deposits including Gbeitouo and Daapleu, two rock waste dumps Teckraie and Verse Ouest and a discontinued heap leach pad Aires. The nine main prospect areas lie within an area of approximately 5 km (East) by 3 km (North).

The Project is located approximately 700 km northwest of Abidjan, which serves as Côte d'Ivoire's economic capital. The mining operations are tightly and firmly linked to the local Yacouba community who live in eight villages that surround the project site. The people work predominantly as farmers and sometimes as artisanal miners and exceptionally as fisherman.

Lycopodium Minerals Pty Ltd. was the Feasibility Study (FS) study lead consultant with a focus on study coordination, metallurgy, process plant and infrastructure design. Cube Consulting completed an updated Mineral Resource estimate. Knight Piésold Pty. Ltd. carried out site geotechnical reviews, completed a water balance study and designed the tailings storage facility, along with mine site drainage control elements. Mining Solutions completed the mine plan and Mineral Reserve, and Peter O'Bryan & Associates undertook a review of pit wall geotechnical and provided relevant inputs to mine design.

Mine environmental and social impact assessments were completed under the lead of 2D Consulting, with SNC Lavalin (Canada) engaged by 2D Consulting to compile a synthetic report from the three ESIA reports in English to comply with international and IFC standards.

The following table presents the key FS Highlights.

**Table 1 Ity CIL Project Feasibility Study Highlights**

| Financial Summary | Units | Value |
|---|---|---|
| Life of Mine Production | | |
| LOM Strip Ratio | w:o | 2.1 |
| LOM Tonnage Ore Processed | Mt | 41.0 |
| LOM Feed Grade Processed | Au g/t | 1.42 |
| Gold Processed | Moz | 1.88 |
| LOM Gold Recovery | % | 83.1% |
| LOM Gold Production | Moz | 1.56 |
| Mine Life | Years | 14.0 |
| Processing Period | Years | 13.7 |
| LOM Average Annual Gold Production | koz | 114 |
| Cash Costs per ounce, LOM average, net Ag credits | $/oz | 528 |
| AISC per ounce, LOM average | $/oz | 603 |
| | | |
| Average for Years 1 to 5 | | |
| Gold Production | Koz/pa | 165 |
| Cash Costs per ounce | $/oz | 466 |
| AISC per ounce | $/oz | 507 |
| | | |
| Average for Years 1 to 9 | | |
| Gold Production | Koz/pa | 144 |
| Cash Costs per ounce | $/oz | 475 |
| AISC per ounce | $/oz | 542 |
| | | |
| Capital Cost | | |
| Upfront Capital Cost | $M | 282 |
| Equipment Lease | $M | 25 |
| | | |
| Results of Economic Modelling | | |
| Using $1,250/oz gold price and after-tax: | | |
| Internal Rate of Return | % | 35.9% |
| Net Present Value - 0% discount rate | $M | 607.2 |
| Net Present Value - 5% discount rate | $M | 403.0 |
| Net Present Value - 10% discount rate | $M | 263.6 |
| Payback Period | Years | 2.1 |

The lty area is characterised by a series of granodioritic intrusions into a sedimentary sequence of volcano-sediments and carbonates. The volcanic are tuffaceous deposits chemistry ranges from basic to acidic. All formations have been subjected to regional metamorphism.

The deposits of lty, Zia and Flotouo are skarns developed at the contacts of the granodiorite with the carbonates. The Teckraie is the rock dump of the Flotouo open pit and sits on top of a weathered granodiorite. The Daapleu deposit is characterised by the presence of a 'rhyolitic' intrusive surrounded by a package of volcanosediments. The 'rhyolite' is locally called 'daaplite' and is leucocratic (grey to white), microgranular, schistose and rich in micas.

The drilling procedures referred to in the FS apply to the drilling campaigns 2013 to 2016. From 2013, the drilling and sampling have been under the supervision of the SMI Geology Department. All drilling prior to 2013 is considered historical and details relating to the drilling and sampling are not well documented. The drillhole types, number of holes and metres within database supplied by EDV indicate a total of 2,536 drillholes for a total of 174,173 metres.

For the purpose of public reporting of Mineral Resource for the rock dumps which include the Teckraie and Verse Ouest Mineral Resources and the Aires heap leach pad, these have not been reported inside an optimised pit shell. These deposits have been built up above the existing topography and the associated laterite is shallow and located directly below, therefore satisfying the requirement for the Mineral Resource to have reasonable prospects for future economic extraction.

The Teckraie and Verse Ouest rock dump Mineral Resources and Aires leach pad Mineral Resources have been reported above 0 g/t Au because there is unlikely to be any grade selectively during mining. The underlying laterite Mineral Resources for each of the deposits has been reported above 0.5 g/t Au given the possibility for some mining selectively. All Mineral Resources are current as at 31 May 2016. The decommissioned Aires heap leach pad and rock dumps comprise approximately 8.6Mt and 300koz of the Indicated resource, as summarised below.

**Table 2 Summary of Mineral Resources**

| Classification | '000 Tonnes | Au (g/t) | Au (koz) |
|---|---|---|---|
| Measured | 0 | 0 | 0 |
| Indicated | 48,617 | 1.49 | 2,322 |
| Measured and Indicated | 48,617 | 1.49 | 2,322 |
| Inferred | 29,090 | 1.47 | 1,302 |

The FS considered the Owner operated development of six open pits, as well as the Aires heap leach pad and Teckraie low grade stockpile. The LOM is approximately 14 years as well as three months of pre-strip.

The mining strategy for the 14 year LOM is as follows:

- Mine and stockpile low grade ore for first nine years LOM.
- Mine higher grade ore from the six pits and deliver to ROM for processing from Years 1 to 9, supplemented with higher grade ore from existing stockpiles and decommissioned pads from Years 5 to 9.

- Ore feed to ROM pad from the fourth quarter of Year 9 to 14 will come from material mined and stockpiled on low grade waste dumps during the first nine years of operation.

Total material movement of approximately 128.5 Mt comprises 87.5 Mt waste, 25.6 Mt high grade ore and 15.4 Mt of low grade ore, with total ore processed of 41.04 Mt. The Mineral Reserve is summarised below.

**Table 3 Summary of Mineral Reserves**

| Item | Ore | | | Waste | | Total Movement |
|---|---|---|---|---|---|---|
| Category | Quantity Mt | Grade Au (g/t) | Quantity Moz | Quantity Mt | Strip Ratio | Mt |
| Proven | | | | | | |
| Probable | 41.04 | 1.88 | 1.25 | 87.52 | 2.1 | 128.56 |
| **Total** | **41.04** | **1.88** | **1.25** | | | |

A metallurgical testwork program was undertaken to determine the optimum processing circuit configuration to maximise return on investment. A previous testwork program was completed in 2015, with a review by Lycopodium identifying additional infill testwork required to adequately address some key processing options and design inputs. Lycopodium selected appropriate core samples from the various ore bodies to ensure representivity across all major ore lithologies prior to commencing the infill testwork programme.

Metallurgical testwork indicated rapid leach kinetics for most of the oxide ores, with recovery of between 80-90% of total gold recovery within eight hours of leaching. Testwork also indicated significant silver levels in Ity ores, with silver to gold ratios of between 2 and 4 recorded from the variability head assays. Based on the gold and silver metallurgical recovery values obtained from the mine schedule based recoveries (83.1% and 60.2%), it is reasonable to predict that the overall gold and silver extractions for the life-of-mine will be between 78% to 89% for gold and between 55% and 69% for silver.

The proposed process plant design for the Ity CIL Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations and other Endeavour Mining Corporation Operations that are well proven in industry.

The Ity CIL plant will process a range of ore types (oxide, transition and fresh) with variable ore characteristics, gold grades and metallurgical treatment requirements. The primary ores are significantly more competent than the oxide ores.

The flowsheet will include a single stage jaw crusher, two stage SAG/ball milling Comminution circuit, gravity concentration for removal of coarse gold, pre leach thickener, CIL circuit comprising six tanks, split Anglo (AARL) elution circuit, electrowinning and gold smelting.

The Tailings storage facility (TSF) has been designed for a total capacity of 46 Mt at an average annual throughput rate of 3 Mt/y. A site selection study was conducted to evaluate several sites relative to the SNC-Lavalin selected site designed for the previous FS. A cross-valley TSF was selected and is to be located 200 m south-west of the proposed plant site, with a water / diversion dam upstream of the TSF.

Power for the CIL process plant will be provided via a connection to the national grid at Danané, approximately 58 km from the Project site. The project will construct the 90 kV single circuit lattice tower transmission line to connect to the national grid. The power station will supply the main HV switchroom inside the processing plant from which power will be distributed.

Project staff will include approximately 360 people, not including catering and cleaning staff. Approximately 10% of staff will be international expats, 43% fly in fly out from Abidjan (FIFO) Ivorian Nations in skilled, supervisory and professional roles, and 47% of staff from the local area. A 200 person camp will be constructed at the site to accommodate FIFO staff, with the remainder of employees living in nearby communities.

Environmental and social impact and mitigation studies, with a goal to be IFC complaint, were completed. The studies outline Endeavour's responsibilities to the well being of the people and the environment during the development, operation and closure of the Ity gold project. The Daapleu village and other associated camps will need to be relocated to a new site at Kouinzompleu, a resettlement operation which will be a key challenge to the new project's success.

A Resettlement Action Plan (RAP) has been developed by Endeavour. The cost of relocation and resettlement is estimated at US$2.4M, and is based on agreements between Endeavour, administrative authorities and the local community.

***Ity CIL Capital Cost Estimate***

The total estimated cost to bring the Ity CIL Project into production is approximately $306.7 million, inclusive of contingency and working capital. The capital costs are summarised below.

| Main Area | Initial Capital (US$M) |
|---|---|
| Treatment Plant | 63.3 |
| Reagents and Services | 9.2 |
| Infrastructure & Tailings | 46.2 |
| Mining | 58.6 |
| Construction Distributables | 24.5 |
| Subtotal | 201.8 |
| Management Costs | 15.8 |
| Owners Project Costs | 59.0 |
| Owners Operations Costs | 3.7 |
| Contingency | 26.4 |
| **Project Total** | **306.7** |

The results of the economic model show robust results. Applying a long term gold price of $1,250/oz on a flat line basis from the commencement of production, the after-tax NPV5% is $410.9 million, IRR is 35.9% and project payback period is 2.1 years. The life of mine average cash cost per ounce is $528, net of silver credits, and with the addition of royalties and sustaining capital, the life of mine average AISC/oz is $603. Over the first five years of production, the average AISC/oz is $507

Project gold production averages 113,656 ounces per year over the life of the project with an average of 164,989 ounces per year over the first five years. The project has low sensitivity to capital and operating costs, but is typically sensitive to gold prices. At $1,250/oz gold price the IRR is 35.9%, and the project maintains an IRR of 19.5% at $1,000/oz gold price

***Conclusions and Recommendations***

Independent studies of the mineral resources, metallurgy, mine plan, processing plant, capital costs, construction costs, environmental and social impact and relocation expenses have been carried out for the Ity CIL Project. These studies and subsequent cash flow model concluded that the project had an acceptable rate of return, and that Endeavour should proceed with project development.

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# 1.0 SUMMARY

## 1.1 Introduction

This report was prepared to provide a NI 43-101-compliant Technical Report on the Feasibility Study (FS) carried out for the Ity CIL gold project in Côte d'Ivoire. Mineral rights are 100% owned by SMI, a company incorporated in Côte d'Ivoire, with the registered address at 08 BP 872 Abidjan 08 - Cocody 180-logements Boulevard Latrille, Immeuble Palm Club, 2nd floor.

The shareholders of SMI are as follows:

- Endeavour Mining 55%.
- Société pour le Developement Minier de la Côte d'Ivoire (SODEMI) 30%.
- State of Côte d'Ivoire 10%
- Keyman Investments (Didier Drogba Group) 5%.

This Report has been prepared by Lycopodium at the request of Endeavour and has been prepared in compliance conformance with the disclosure requirements of the Canadian National Instrument 43-101standards required by (NI 43-101) and in accordance with the requirements of Form 43-101 F1(F). The Report is intended to be used to demonstrate the results of the FS.

In compliance with the requirements for preparation of a NI 43-101 Technical Report, key competent persons visited the project site to gather first-hand information. In addition, Lycopodium and other contributors have studied and constructed numerous projects in the region and are familiar with local conditions

### 1.1.1 Contributions

Contributors to the preparation of this report are summarised in Table 1.1.1. Lycopodium Minerals Pty Ltd (Lycopodium) were responsible for compilation of the report inputs and delivery of the report to Endeavour.

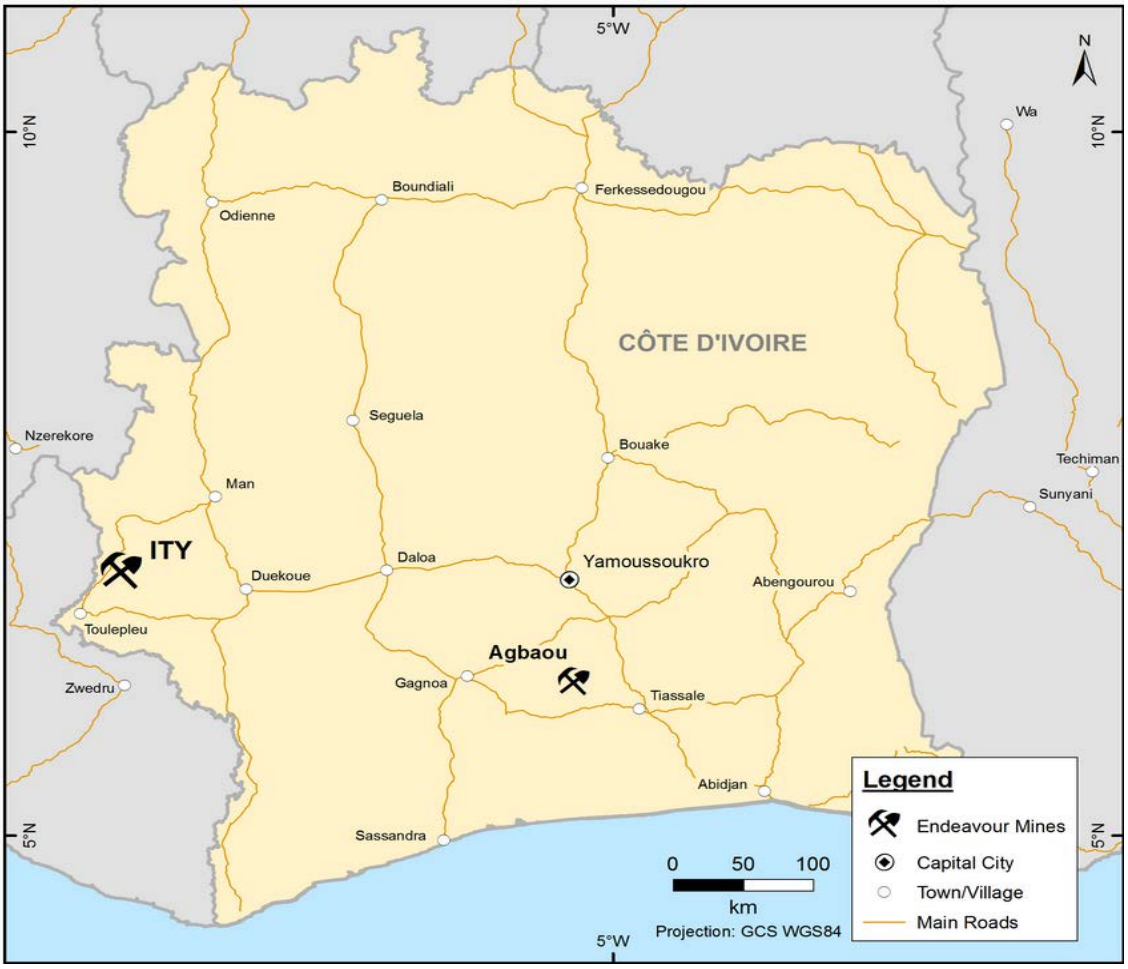
**Table 1.1.1 Report Contributors**

| Report Section | Contributors |
|---|---|
| Section 1 - Summary | Cube Consulting, Mining Solutions, Lycopodium, Knight Piésold , Endeavour Mining, Peter O' Bryan & Associates |
| Section 2 - Introduction | Lycopodium |
| Section 3 – Reliance on Other Experts | Lycopodium |
| Section 4 – Property Description and Location | Cube Consulting |
| Section 5 – Accessibility, Climate, Local Resources and Infrastructure | Lycopodium, Knight Piésold, Mining Solutions |
| Section 6 - History | Cube Consulting |
| Section 7 – Geological Setting and Mineralisation | Cube Consulting |
| Section 8 – Deposit Types | Cube Consulting |
| Section 9 - Exploration | Cube Consulting |
| Section 10 - Drilling | Cube Consulting |
| Section 11 – Sampling Preparation, analysis and Security | Cube Consulting |
| Section 12 – Data Verification | Cube Consulting |
| Section 13 – Mineral Processing and Metallurgical Testing | Lycopodium |
| Section 14 – Mineral Resource Estimates | Cube Consulting |
| Section 15 – Mineral Reserve Estimates | Mining Solutions |
| Section 16 – Mining Methods | Mining Solutions, Knight Piésold, Peter O'Bryan & Associates |
| Section 17 – Recovery Methods | Lycopodium |
| Section 18 – Project Infrastructure | Lycopodium, Knight Piésold, |
| Section 19 – Market Studies and Contracts | Endeavour Mining |
| Section 20 – Required Permits and Environmental Considerations | Endeavour Mining |
| Section 21 – Capital and Operating Cost | Lycopodium, with input from Knight Piésold , and Mining Solutions |
| Section 22 – Economic Analysis | Lycopodium |
| Section 23 – Adjacent Properties | Cube Consulting |
| Section 24 – Other Relevant Data and Information | Lycopodium, Mining Solutions, Cube Consulting, Peter O Bryan & Associates, Knight Piésold |
| Section 25 – Interpretation and Conclusions | Cube Consulting, Mining Solutions, Lycopodium |
| Section 26 – Recommendations | Cube Consulting, Mining Solutions, Lycopodium, Peter O Bryan & Associates, Knight Piésold |
| Section 27 – Selected Reference | Cube Consulting, Mining Solutions, Lycopodium, Peter O Bryan & Associates, Knight Piésold |

## 1.2 Property Access and History

Côte d'Ivoire is located along the coast of Western Africa and is bordered to the east by Ghana, to the north by Burkina Faso and Mali, and to the west by Guinea and Liberia. The Company's mining operations are close to the Liberian border as shown on Figure 1.2.1. Ity is approximately 700 km northwest of Abidjan, which serves as Côte d'Ivoire's economic capital.

**Figure 1.2.1 Ity Project Location**




The Ity Mine is located in the prefecture of ZOUAN-HOUNIEN. The project site is accessible via paved road from Abidjan, passing through the capital Yamoussoukro, Daloa and Duekoué. It is also accessible by plane from Abidjan to Man and then from Man to the site by bus operated by SMI. From Duekoué, two roads access the project from both north and south. The north access is through Man and then on to Danané and Zouan–Hounien where a 15 km unsealed road maintained by SMI leads to the village of Ouyatouo. Southern access is through Guiglo and Toulepleu.

Ity has the longest history of any gold mine in Côte d'Ivoire, and has produced more than 1.2 million ounces of gold in more than 25 years of production. Copper and gold were first discovered near the village of Ity in the 1950's during exploration by the Bureau Minière de la France d'Outre-Mer. Initial attempts to recover gold were not successful primarily due to the fineness of gold and ore rheology.

In 1983 the Société Minière d'Ity (SMI) was incorporated to develop the Flotouo deposit which poured it first gold in 1991. Substantial exploration was done in the 1990s and many of the deposits were discovered or expanded at the time. Since then ownership has changed several times until the La Mancha Group acquired a large stake in SMI in 2012. In 2014 a change in shareholders was authorised by the Government of the Côte d'Ivoire leading to the current ownership structure.

Endeavour currently operate the project using open pit mining methods and a heap leach process for gold recovery.

A scoping study for the exploitation of the mineral resource using an alternative processing route was conducted in 2013. In 2014, SNC-Lavalin Inc. (SNC) conducted a Pre-feasibility Study (PFS) on behalf of Société Minière d'Ity (SMI) to evaluate the potential of a CIL plant at a processing rate of 1.5 Mtpa.

The results of the study were positive and in late 2014 through to early 2015, SMI carried out drilling programs at the Daapleu, Zia NE and Mont Ity deposits designed to upgrade all Inferred Mineral Resource material from the 2014 mineral resource estimate to Indicated Mineral Resources, the Daapleu deposit Indicated Mineral Resource to Measured Mineral Resource, and to delineate each deposit further along strike.

The resulting 2015 mineral resource estimate update yielded a significant increase in measured and indicated mineral resources to 2.9 Moz of gold contained in five deposits, two dumps, decommissioned leach pads and a stockpile. SNC-Lavalin was mandated by SMI to update the PFS for the CIL Project using a processing rate of 2.0 Mtpa.

Coffey Mining (South Africa) Pty Ltd (Coffey) was then requested by Endeavour Mining to compile an independent Technical Report on the project, to comply with disclosure and reporting requirements set forth in the Toronto Stock Exchange Manual, National Instrument 43-101.

## 1.3 Licence Status

The SMI and La Mancha Côte d'Ivoire (LMCI) properties, which are the subject of this report, include the following two permits which contain the gold deposits:

- Permis de' Exploitation PE26 -Mining Permit.
- Permis de' Recherche PR609 (Formerly PR61) - Exploration Permit.

To comply with the conditions and changes in the new Côte d'Ivoire 2014 Mining Code, SMI abandoned PR61, and La Mancha resources requested and was granted a reassignment of the permit as PR609. Exploration permits confer on holders, within boundaries, surface and depth, the exclusive right to explore for mineral substances as well as the free disposal of the products extracted during exploration.

Mining permit PE26 comprises an area of 25 km² and includes the Ity mine, the deposits ZiaNE and Walter, the Aires decommissioned heap leach pads, dumps Verse Ouest and Teckraie, and the surrounding areas most of which are west of the Cavally River. Mining permit PE26 was first granted in 1989 as Arrêté 197/MINES/DM/(14 October 1989), and has been renewed several times, most recently as Arrêté 8-002/MIM/DGMG granted on 28 August 2014 for 10 years from 14 November 2014. The boundary coordinates for PE26 are summarised in Table 1.3.1.

**Table 1.3.1 Boundary Coordinates for Mining Permit PE26**

| Longitude (West) | Latitude (North) |
|---|---|
| 8"08'30" | 6"53'30" |
| 8"05'46" | 6"53'30" |
| 8"05'46" | 6"50'44" |
| 8"08'30" | 6"50'44" |

Exploration Permit PR609 (formerly PR61) is situated over parts of the Zouan-Hounien and Blolequin prefectures. The permit has an area of 153 km² and is orientated northeast-southeast, and is comprised of two sub portions with orebodies Gbeitouo and Daapleu and prospects Yacetouo, Morgane, Mlambopleu to the east of the Cavally River, and Floleu to the west of the Cavally River.

The original exploration permit PR61 was first issued in 1995 and was extended in two year intervals until September 12, 2014. It was replaced with PR609 in July 2015 and is valid for a period of four years. The boundary coordinates for PR609 are summarised in Table 1.3.2.

**Table 1.3.2 Boundary Coordinates for Exploration Permit PR609**

| Longitude (West) | Latitude (North) |
|---|---|
| 7"58'57" | 7"00'16" |
| 7"58'57" | 6"54'10" |
| 8"00'46" | 6"52'16" |
| 8"09'37" | 6"48'50" |
| 8"10'58" | 6"49'53" |
| 8"08'30" | 6"52'00" |
| 8"08'30" | 6"50'44" |
| 8"05'46" | 6"50'44" |
| 8"05'46" | 6"53'30" |
| 8"06'47" | 6"53'30" |

An operating permit, 'PE on demande', (being part of PR609) has also been sought by La Mancha CI in September 2015 to cover the deposits of Daapleu and Gbeitouo for a 10 year period covering an area of 13.2 km² at the coordinates listed in Table 1.3.3. This 'PE on demande' is being assessed by the Director of Mines, and requires the integration of information from this FS for final approval.

**Table 1.3.3 Boundary Coordinates for PE on Demand**

| Point | Longitude (West) | Latitude (North) |
|---|---|---|
| A | 8"05'46" | 6"53'30" |
| B | 8"04'17" | 6"53'30" |
| C | 8"04'17" | 6"50'53" |
| D | 8"05'46" | 6"50'53" |

## 1.4 Geology and Mineralisation

The Mont Ity deposits are located in the Lower Proterozoic Birimian Formation of the Toulépleu-Ity klippe. The Toulépleu-Ity klippe is a small remnant of the Birimian in the West African Craton which spans ten countries, between Côte d'Ivoire, Senegal, Niger and Ghana.

Milesi et.al, (1989) and the BRGM (1986, 1989) proposed a definition of the Birimian with a lithostratigraphic succession separated into two large groups:

- A Lower Birimian (B1) set essentially flyschoid basin fill. The whole basin is affected by three cycles of deformation including a D1 (2090-2100Ma) phase of major collision and duplication of the lower Proterozoic on the gneissic Archean basement and a break in all B1 sedimentation and intrusion of syn-kinematic granites. D2 and D3 (2090-1970Ma) deformation were responsible for the intrusion of granites mantle between 2080 and 1945Ma (D2 large sinistral offsets, related overlaps and folding: D3 dextral offsets and associated folds).

- The Upper Birimian (B2), volcanic-dominated, where fluvio-deltaic formations are intercalated in volcano-sedimentary facies.

The Ity area is characterised by a series of granodioritic intrusions into a sedimentary sequence of volcano-sediments and carbonates. The volcanic are tuffaceous deposits chemistry ranges from basic to acidic. All formations have been subjected to regional metamorphism. The main lithologies within the Ity project area include the following:

- Laterites (Facies 301) - are soils without texture, formed by supergene alteration of iron rich rocks and are generally red in colour. Four lateritic categories have been identified at Ity with all profiles containing millimetre scale fragments of magnetite and quartz.

- Saprolites - are the result of supergene alteration, hydrothermal alteration or a combination of the two. The facies vary in colour as a function of the protolith and degree of oxidation and include the following:

  - Oxidised Saprolites (Facies 201) which are essentially oxides and hydroxides of iron ($Fe^{2+}$ and $Fe^{3+}$) which are present in a wide range of colours, yellow, orange, brick-red violet, etc.

  - Reduced Saprolites (Facies 211) are coloured green to white or light grey. They are generally found deeper than that oxidised saprolites and correspond to the water table limit.

- MetaVolcanic Sediments (Facies 101) - are dark greyish-brown to grey-green and finely laminated. Minerals are primarily amphibole, chlorites, biotites, traces of magnetite, calcite as veinlets and fracture coatings and quartz veins with tourmaline.

- Carbonate (Facies 401) - typically greenish in colour and microcrystalline with chlorite-epidote alteration, disseminated pyrites and magnetite. The carbonates represent protoskarn facies.

- Granodiorite (Facies 501) - coarse-grained grey felsic intrusive rich in sericite, magnetite and large straw-like crystals of muscovite.

- Daapleu Rhyolite (Facies 601) - granitic intrusive at Daapleu prospect is locally called a rhyolite as it has a fine texture. The rhyolite is leucocratic (grey to white), microgranular, schistose and rich in sericite and contains fine-grained pyrite or magnetite disseminated and in fractures.

- Diorite (Facies 501) - coarse-grained grey-green mafic intrusive composed of amphibole, magnetite and chlorite with fine-grained disseminated pyrite and calcite veins.

- Skarn (Facies 901) - represent metamorphosed carbonate host and are associated with presence of sulphur in the form of varying occurrences of pyrite, pyrrhotite, chalcopyrite and magnetite.

The Ity Gold Project deposits are orogenic gold deposits and generally described as skarn or shear zone styles of mineralisation.

Skarn deposits within the Ity Gold Project include Mont Ity, Ity Flat, ZiaNE and Walter. The material from which the Aires Verse Ouest and Teckraie dumps have been derived were also sourced from mining of skarn deposits. The skarn corresponds to a sulphur rich skarn (pyrite-pyrrhotite-chalcopyrite, 3 - 10%) with accessory magnetite. These mineralised zones form discontinuous, sub parallel lenses at the contact with the granodiorite.

The mineralised portions of the reduced saprolites correspond to the decarbonation of marble and exoskarn .These are rocks rich in chlorite, tremolite and/or actinolite. The reduced saprolites are the transition zone between the oxidised material and the fresh rock. The oxidised saprolites form the major part of the mineralisation at Mont Ity. The mineralisation was originally a skarn which has undergone severe supergene alteration. The alteration was enhanced by the dissolution of the sulphides. The alteration was responsible for the generation of karst into which the saprolite material collapsed which provides an explanation for the geometry of the deposit.

The Daapleu and Gbeitouo deposits resemble typical shear zone deposits of the West African granite-greenstone terrane. Both deposits are associated with a major regional shear zone but developed on secondary structures. The lithologies can be any form of sediment, carbonate or igneous rock with the main feature being a shear zone between the two contrasting lithologies. Mineralisation may also be spatially related to the emplacement of intrusives.

The gold mineralisation is mesothermal in origin and occurs as free gold in quartz vein stockworks and zones of silicification, associated with arsenopyrite and to a lesser extent pyrite and antimony. Depending on the geological terrane other metals may be associated with the gold and arsenic. The gold mineralisation is found in linear zones in or near the contacts between two different rock types. This contact shows evidence of shearing. Alteration is weak to severe depending on the development of the system.

## 1.5 Exploration and Drilling

Since 2012, in order to extent the heap leach operation life and to increase the CIL resources, the group engaged in an intensive exploration program. The main objectives of this program were as follows:

- Find more oxide ore and increase heap leach mining life by delineating near mine lateritic ore, infill drilling current oxide deposits (Mont Ity, ZiaNe and Flat) , and find, test and develop other oxide targets within 5 km radius from Ity plant.

- Near mine and FS related projects such as the develop of Indicated resources for the FS deposits (ZiaNe, Daapleu, Mont Ity, Walter, Gbeitouo, Teckraie, West and East Rock dumps, Aires leach pad), assist with metallurgical test samples, develop robust geological model of the Ity area and develop other targets for CIL resources within 5 km radius from Ity plant.

- Regional Exploration projects included defining other targets to create a new pipeline for exploration projects, ground geophysical campaign (IP), soil geochemical sampling, regolith mapping, pit, trenches and auger drilling, and an aerial survey.

The prospectivity of the Ity area has been confirmed by recent exploration works. The auger drilling program coupled with gold in soil results indicate that the overall potential of the area extends well beyond the present footprint. The most significant near-mine targets identified are:

- **Bakatouo Northeast area:** Auger results clearly suggest a 1 km northeast trending anomaly located immediately to the northeast of the new Bakatouo discovery, with best values >7,000 ppb in the saprolite.

- **Vavoua area:** New 1.5 km northeast trending anomaly with best values >1,400 ppb in the saprolite.
- **Morgan / Yacetouo area:** Large and significant 3 km long and northeast trending anomaly with best values >2,450 ppb in the saprolite.
- **Le Plaque area (PR 558):** Significant 2 km long N20° trending anomaly with best values >5,000 ppb in the saprolite.
- **Daapleu Southwest area:** Large gold anomaly in saprolite that covers over 4 km², with best values >2,200 ppb.

The high grade Bakatouo discovery is located in proximity to the current Ity mining complex. The mineralisation is close to surface and occurs over 800 m along strike, while extensions remain open at depth and along strike. An additional drilling campaign started in November 2016 to tests its extensions while exploration is ongoing on other nearby targets.

Prior to 2014, drillhole data was managed on site using bespoke public domain database software. In 2014, SMI introduced AcQuire database software (an industry standard database software) as a replacement to manage all historical and modern drilling data. Since 2014 all historical data has been validated and migrated into the AcQuire database.

Cube was provided with MS Access exports by SMI for each deposit. These were imported into a single MS Access database by Cube which used as the primary data source for the geological interpretations, QAQC analysis and Mineral Resource estimation. Cube has not independently verified any of the data contained in the database except for undertaking routine validation checks and no material errors were detected in the collar, assay or survey tables during this checking process. The drillhole types and number of meters within the database used in the Mineral Resource estimate is summarised in Table 1.5.1.

**Table 1.5.1 Drillhole Database Summary - Effective Data 31 May 2016**

| Deposit | DD | RC | RC-DD | AC | RAB | TR | CS | PIT | Total |
|---|---|---|---|---|---|---|---|---|---|
| Aires | 768 | - | - | 6,455 | - | - | 39 | - | 7,262 |
| Daapleu | 29,093 | 4,242 | - | | - | - | 194 | 1,471 | 34,999 |
| Gbeitouo | 9,203 | 4,309 | 863 | - | 1,044 | - | - | 847 | 16,266 |
| Ity Combined | 46,488 | 17,735 | - | 702 | - | 28 | 17 | 4,388 | 69,358 |
| Teckraie | 1,079 | 28 | - | 3,191 | - | - | - | - | 4,298 |
| Tontouo | 1,363 | 40 | - | - | - | - | - | - | 1,403 |
| Verse Ouest | 1,181 | - | - | - | - | - | - | - | 1,181 |
| Versest | 903 | 586 | - | 900 | - | - | - | - | 2,389 |
| Walter | 6,272 | 2,253 | 613 | - | - | - | - | - | 9,137 |
| ZiaNE | 25,912 | 926 | - | 828 | - | - | 216 | - | 27,882 |
| **Total** | **122,260** | **30,118** | **1,475** | **12,076** | **1,044** | **28** | **466** | **6,706** | **174,173** |

Overall the sample collection and preparation, analytical techniques, security and QA/QC protocols implemented for the Ity Gold Project are consistent with standard industry practice and are suitable for the reporting of exploration results and for use in Mineral Resource estimation. The sampling procedures are adequate for and consistent with the style of gold mineralisation under consideration.

## 1.6 Mineral Resource

For all in situ deposits (Daapleu, Gbeitouo, Mont Ity, Ity Flat, ZiaNE and Walter), weathering interpretations based on geological logging in the 'OX_SUPERGENE' database field were used to define oxide, transitional and fresh weathering domains. Lithological interpretations based on geological logging in the 'LITHO' database field were used to create validated three dimensional solids defining the dominant lithological facies types. Robust mineralisation interpretations were based primarily on nominal gold grade cut-offs relevant to each deposit typically between 0.3 and 0.5 ppm which appears to approximate the natural grade cut-off off between mineralisation and waste rock. In addition, relevant drill log information and the lithological model were used as guidance for the interpretation.

A unique code for drill intercepts within each of the mineralised domains was used to control the compositing process whilst extracting sample and composite data combinations for statistical analysis and subsequent resource estimation. For all in situ deposits, 2 m downhole composites were extracted within the mineralised domains. For the Teckraie, Verse Ouest and Aires deposits, 2.5 m composites were extracted. Appropriate high-grade cuts were applied on an individual domain basis and are typically above the 99th percentile of the assay population. The number of cut samples is generally low and although having a material effect on reducing the coefficient of variation ('CV'), the effect of these top-cuts was not expected to be material for the gold estimate.

The primary gold estimation method implemented for all in situ deposits is Localised Uniform Conditioning (LUC). LUC is considered the most appropriate method for the estimation of local recoverable resources for these deposits as it produces accurate grade-tonnage functions which are in good accordance with volume-variance relationship principles. It limits the smearing of high grade and the over-smoothing of grade compared to other estimation methods.

The final mineral resource is a diluted recoverable resource estimate given the process attempts to estimate the recoverable tonnage and grade based on the dimensions of the selective mining unit (SMU) which is regarded as representative of what is practically achievable during actual mining. Estimation is undertaken using relatively large panels and therefore can be considered as being 'diluted' as they are estimated using all data within a broad mineralised envelope. The panel size varied between deposit depending on the data spacing however the same SMU (5 mE x 5 mN x 2.5 mRL) was used throughout. A Change of Support is applied to the large diluted panels in order to predict the likely grade tonnage distribution of SMU selectivity.

In addition to the imposition of a minimum selective mining dimension, a further change of support correction was applied to the models in the form of an information effect. The information effect is a theoretical 'penalty' adjustment to the SMU grade tonnage distribution to account for the anticipated dilution incurred when making mining selectivity decisions based on likely grade control spaced data.

A typical grade control pattern does not represent exhaustive information and some misallocation of material is inevitable when block estimates are based on a non-exhaustive dataset. In a sense, the information effect adjustment is a representation of the impact of inaccuracies inherently anticipated in the practical mining process. Unplanned mining dilution due to mining inaccuracies would still need to be accounted for as a modifying factor.

The primary gold estimation method implemented for the Teckraie and Verse Ouest rock waste dumps and Aires heap leach pad is Ordinary Kriging (OK). OK is considered an appropriate method for estimation given the low coefficients of variation for the cut composite data. No mining selectively would be anticipated during mining and therefore OK will provide a robust global estimate. As such, the global mean of the OK estimate was assigned as the final grade estimate for each deposit.

All available bulk density data was supplied by Endeavour and included a total of 3,391 determinations. The determinations that were deemed acceptable were limited to measurements undertaken on diamond drill core which had been measured using a standard water displacement method. This resulted in a total of 2,948 determinations available for inclusion in the total Ity Project. Density determinations were grouped by deposit and lithology type. Outlier samples were removed and the typically mean of each domain population was assigned to the block model based on flagged lithology. In some instances where the number of density determinations was small, the assigned density was adopted from another deposit with similar geology characteristics.

### 1.6.1 Mineral Resource Classification and Reporting

For the purpose of public reporting of the in situ Mineral Resources which include Daapleu, Gbeitouo, Mont Ity / Ity Flat, Walter and ZiaNE, these have been reported inside optimised pit shells and above a 0.5 g/t Au cut-off. Reporting within an optimised pit shell satisfies the requirement for the Mineral Resource to have reasonable prospects for eventual economic extraction. The pit optimisation assumes a US$1,500/oz Au price.

For the purpose of public reporting of Mineral Resource for the rock dumps which include the Teckraie and Verse Ouest Mineral Resources and also the Aires heap leach pad, these have not been reported inside an optimised pit shell. These deposits have been built up above the existing topography and the associated laterite is shallow and located directly below, therefore satisfying the requirement for the Mineral Resource to have reasonable prospects for eventual economic extraction.

The Teckraie and Verse Ouest rock dump Mineral Resources and Aires leach pad Mineral Resources have been reported above 0 g/t Au because there is unlikely to be any grade selectively during mining. The underlying laterite Mineral Resources for each of the deposits has been reported above 0.5 g/t Au given the possibility for some mining selectively.

All Mineral Resources are current as at 31 May 2016 and have been reported by cut-off grade and Mineral Resource category which is presented in Table 1.6.1.

**Table 1.6.1 Mineral Resources as at Effective Data 31 May 2016**

| Deposit | Cut-Off | Indicated | | | Inferred | | |
|---|---|---|---|---|---|---|---|
| | | Tonnes ('000) | Au (g/t) | Au ('000 Oz) | Tonnes ('000) | Au (g/t) | Au ('000 Oz) |
| Aires | 0 | 5,768 | 1.09 | 202 | | | |
| | 0.5 | | | | 233 | 0.78 | 6 |
| Daapleu | 0.5 | 19,882 | 1.51 | 965 | 4,329 | 1.15 | 160 |
| Gbeitouo | 0.5 | 2,865 | 1.35 | 124 | 270 | 1.48 | 13 |
| Mont Ity / Ity Flat | 0.5 | 7,491 | 2.19 | 527 | 11,084 | 1.92 | 684 |
| Teckraie | 0 | 2,819 | 1.07 | 97 | | | |
| | 0.5 | | | | 114 | 0.55 | 2 |
| Verse Ouest | 0 | | | | 5,896 | 1.03 | 195 |
| | 0.5 | | | | 2,505 | 0.43 | 35 |
| Walter | 0.5 | 2,069 | 1.21 | 81 | 657 | 1.32 | 28 |
| ZiaNE | 0.5 | 7,723 | 1.31 | 325 | 4,002 | 1.39 | 179 |
| **Total** | | **48,617** | **1.49** | **2,322** | **29,090** | **1.47** | **1,302** |

- Mineral Resources are reported inclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Rounding as required by reporting guidelines may result in small discrepancies.
- Mineral Resources are reported on a 100% basis.
- Cut-off grade of 0.5 g/t Au for situ Mineral Resources including Daapleu, Gbeitouo, Mont Ity / Ity Flat, Walter and ZiaNE have been reported inside optimised pit shells and based on 3 Mtpa CIL operating costs and a US$1,500/oz Au price.
- The Teckraie and Verse Ouest rock dump and Aires leach pad Mineral Resources have been reported above 0 g/t Au because there is unlikely to be any grade selectively during mining.
- The Laterite mineralisation below Aires, Teckraie and Verse Oust have been reported above a cut-off grade 0.5 g/t Au.

## 1.7 Mineral Reserve

The Ity mineral reserve is reported inside the pit designs. The pit design process was guided with the pit optimisation results based on the following parameters:

- US$1,250/oz gold price.
- 87.7% to 96.7% process recovery varying by the pits for the non-sulphide ore.

- 59.1% process recovery for the sulphide ore.
- $14.55/t to $18.60/t throughput cost varying by the pits and ore types (the throughput cost includes processing, G&A and additional ore mining costs),.
- $2.55/t average mining cost.
- 30, 35 and 42 degrees pit slopes for the saprolite, transition and fresh rock zones, respectively.

The Ity site layout for the pit and waste dump designs can be seen in Figure 1.11.1

The reserve statement for the Project is provided in Table 1.7.1. All the project reserves are classified in the Probable category based on the underlying resource classification of Indicated resources.

**Table 1.7.1 Mineral Reserve Estimate (Effective 29 September 2016)**

| Resource | Proven Reserve | | | Probable Reserve | | |
|---|---|---|---|---|---|---|
| | Tonnes ('000) | Au g/t | Au ('000oz) | Tonnes ('000) | Au g/t | Au ('000oz) |
| **Open Pits** | | | | | | |
| Daapleu | | | | 19,343 | 1.51 | 936 |
| Ity Flat | | | | 1,599 | 2.09 | 108 |
| Mont Ity | | | | 2,228 | 2.23 | 160 |
| Gbeitouo | | | | 2,579 | 1.35 | 112 |
| Walter | | | | 1,858 | 1.22 | 73 |
| Zia NE | | | | 4,835 | 1.24 | 192 |
| Total Open Pits | | | | 32,443 | 1.52 | 1,580 |
| **Existing Stockpiles** | | | | | | |
| Aires Leach Pad | | | | 5,766 | 1.09 | 202 |
| Teckraie Dump | | | | 2,833 | 1.07 | 97 |
| Total Stockpiles | | | | 8,599 | 1.08 | 300 |
| **Total** | | | | **41,042** | **1.42** | **1,880** |

No additional mining dilution or ore recovery factors were applied to the reported reserves since the resource estimates already incorporate dilution for the selective mining unit (SMU, 5 mE x 5 mN x 2.5 mRL) size for open pit mining.

In line with the CIM classification system, only the Indicated category resources were included in the pit optimisations and the resultant reserve estimates in the pit designs. The Inferred resources in the pit designs are classified as waste. The stripping ratio for the open pit reserves is 2.7 t:t. The overall stripping ratio, including the existing stockpiles, is 2.1 t:t.

## 1.8 Mining

### 1.8.1 Mining Inventory

The mining inventory calculated within the pit designs are summarised in Table 1.8.1. The cut-off grade for the non-sulphide ore vary from 0.42 g/t to 0.53 g/t depending on the process recoveries and throughput costs. The cut-off grade for the Daapleu sulphide ore is 0.72 g/t, relatively high due to the lower recovery. In addition to the calculated break-even cut-off grades, the analysis in the preliminary schedules showed that a cut-off grade of 0.9 g/t for the non-sulphide ore and 1.2 g/t for the sulphide ore would be appropriate to classify the high grade ore for the production schedule.

**Table 1.8.1 Mining Inventory Summary**

| | Total | | Waste | | Total Ore | | | Ore Classification | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | Low Grade (LG) | | High Grade (HG) | | Sulphide LG | | Sulphide HG | |
| | kt | kbcm | kt | kbcm | kt | kbcm | Au g/t | kt | Au g/t | kt | Au g/t | kt | Au g/t | kt | Au g/t |
| **Open Pits** | | | | | | | | | | | | | | | |
| Daapleu 1 | 18,334 | 7,913 | 9,346 | 4,285 | 8,988 | 3,629 | 1.69 | 3,405 | 0.63 | 2,768 | 1.67 | 824 | 0.94 | 1,991 | 3.83 |
| Daapleu 2 | 33,805 | 13,572 | 23,449 | 9,642 | 10,355 | 3,931 | 1.34 | 4,089 | 0.63 | 2,667 | 1.56 | 1,416 | 0.93 | 2,184 | 2.68 |
| Ity Flat | 7,226 | 3,753 | 5,627 | 2,973 | 1,599 | 780 | 2.09 | 540 | 0.69 | 1,059 | 2.80 | | | | |
| Mont Ity | 12,262 | 6,251 | 10,034 | 5,217 | 2,228 | 1,034 | 2.23 | 655 | 0.70 | 1,573 | 2.86 | | | | |
| Gbeitouo | 10,769 | 5,703 | 8,190 | 4,446 | 2,579 | 1,257 | 1.35 | 964 | 0.67 | 1,616 | 1.76 | | | | |
| Walter | 8,882 | 4,751 | 7,025 | 3,757 | 1,858 | 995 | 1.22 | 885 | 0.67 | 973 | 1.72 | | | | |
| Zia NE | 28,687 | 17,160 | 23,852 | 14,315 | 4,835 | 2,846 | 1.24 | 2,655 | 0.63 | 2,180 | 1.98 | | | | |
| **Total Open Pits** | 119,965 | 59,104 | 87,522 | 44,633 | 32,443 | 14,471 | 1.52 | 13,193 | 0.64 | 12,835 | 1.96 | 2,240 | 0.94 | 4,175 | 3.23 |
| Stockpiles | | | | | | | | | | | | | | | |
| Aires Heap Pad | 5,766 | 4,435 | | | 5,766 | 4,435 | 1.09 | | | 5,766 | 1.09 | | | | |
| Teckraie Dump | 2,833 | 2,179 | | | 2,833 | 2,179 | 1.07 | | | 2,833 | 1.07 | | | | |
| **Total Stockpiles** | 8,599 | 6,615 | | | 8,599 | 6,615 | 1.08 | | | 8,599 | 1.08 | | | | |
| **Total Inventory** | **128,564** | **65,719** | **87,522** | **44,633** | **41,042** | **21,085** | **1.42** | **13,193** | **0.64** | **21,434** | **1.61** | **2,240** | **0.94** | **4,175** | **3.23** |

### 1.8.2 Production Schedule

The graph in Figure 1.8.1 summarises the total material movement classified by waste, high grade ore and low grade ore tonnes in the production schedule. The peak production in Year 3 include 1.4 Mt of mill feed sourced from the existing stockpiles. The average open pit production rate is 15.5 Mtpa for the six years of the mine operations, then the production tails off in the last three years.

**Figure 1.8.1 Yearly Total Tonnage Mined**

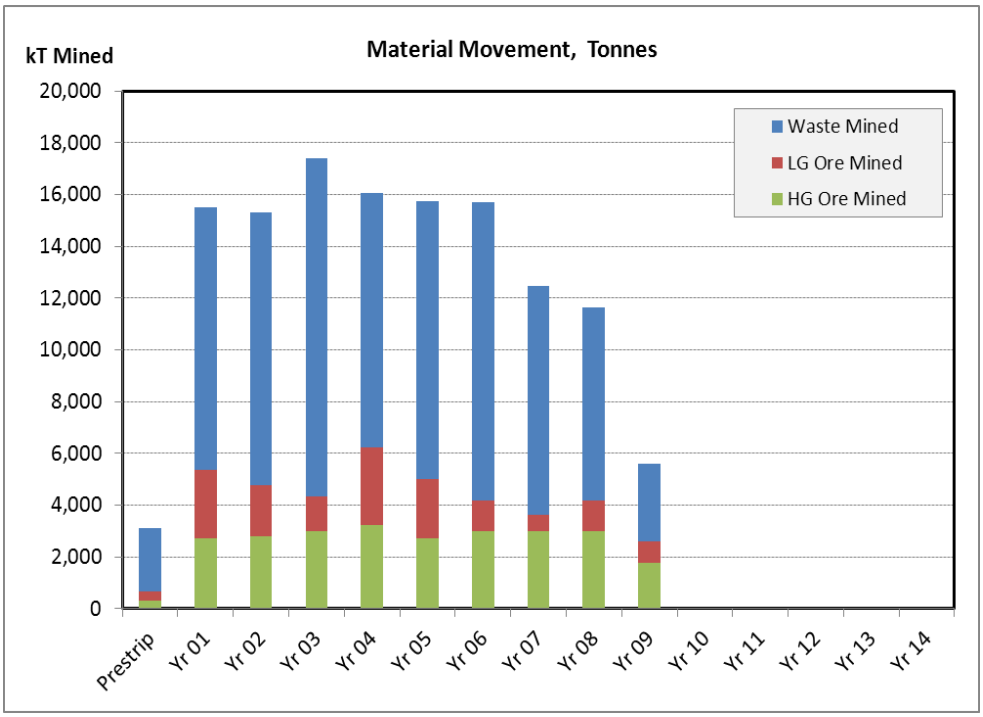
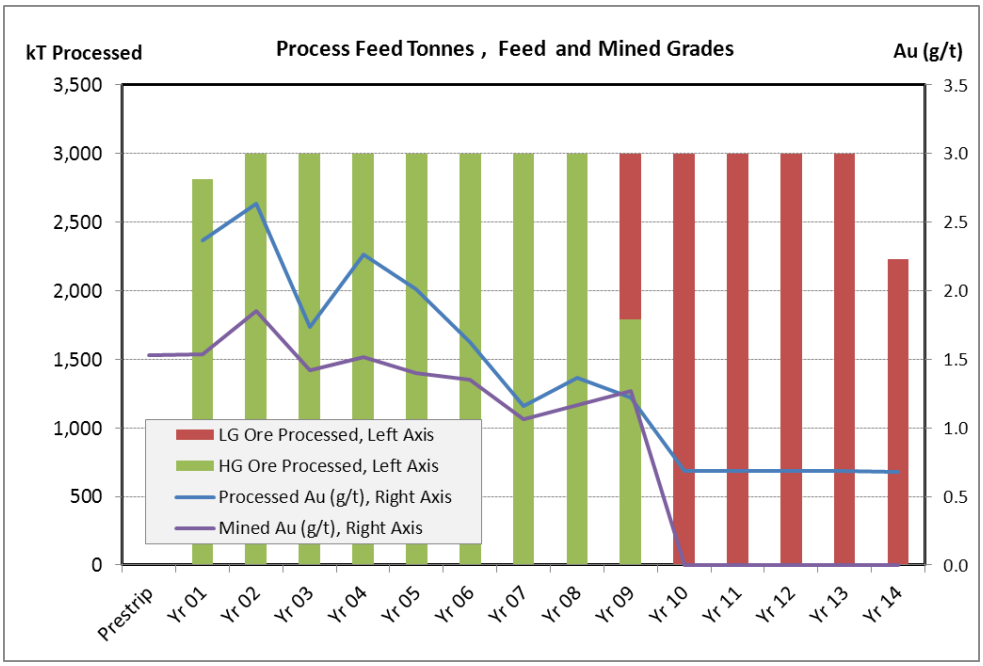



Figure 1.8.2 shows that the last five years of the process feed is from the low grade ore stockpiles accumulated during the mining of the open pits in nine years. The graph of the yearly mined and processed grades shows that the processed grade is consistently higher than the mined grade due to the stockpiling of the low grade ore. The dip in the processed grades in Years 3 and 7 are caused by supplementing the process feed from the existing ore stockpiles.

**Figure 1.8.2 Yearly Processed Tonnes, Grade and Mined Grade**




### 1.8.3 Mining Equipment and Labour Requirements

Table 1.8.2 summarises the peak mining equipment requirements to achieve the production schedule targets. The mobile equipment types were selected assuming an owner mining scenario.

**Table 1.8.2 Mining Equipment Specification and Peak Numbers**

| Mining Equipment | Model | Peak Number |
|---|---|---|
| 100 Tonne Excavator | Komatsu PC 1250SP-8R | 3 |
| Ore Rehandle Loader | Komatsu WA600-6R | 1 |
| Pit Trucks | Komatsu HD785-7 | 12 |
| SP Rehandle Trucks | Komatsu HD785-7 | 2 |
| Track Dozers | Komatsu D375A-6R | 3 |
| Wheel Dozers | Komatsu WD600-6 | 2 |
| Graders | Komatsu GD825A-2 | 2 |
| Watercarts | Komatsu HD465-7R | 4 |
| Crusher Feed FEL | Komatsu WA600-6R | 2 |
| Blasthole drills | Sandvik DI550 | 4 |

Table 1.8.3 summarises the peak mining labour requirements for the production schedule.

The sections 1.12.1 and 1.13.1 below summarises the mining operating and capital costs for the owner mining scenario.

**Table 1.8.3 Mining Labour Peak Requirements**

| Labour Category | Peak Number |
|---|---|
| **Direct Labour** | |
| Equipment Operators | 101 |
| Blasting Personnel | 8 |
| Maintenance Personnel | 33 |
| **Total** | **142** |
| **Indirect Labour** | |
| Mine Management | 2 |
| Technical Services | 13 |
| Geology | 12 |
| Production | 10 |
| Maintenance | 9 |
| **Total** | **46** |
| **Total Labour** | |
| **Total** | **188** |

## 1.9 Metallurgical

### 1.9.1 Overview

Following the unpublished testwork program completed by SNC in 2015, an infill program was completed by Lycopodium. The object of this program is to determine key design criteria for the treatment of the Ity resource.

The following provides a summary of the testwork undertaken by ALS between June and October 2016:

- Comminution tests performed on 20 samples representing all facies types across all pits and intended to produce a range of comminution data (SMC values, Bond Ball Mill Work indices, and Abrasion indices) for input into equipment sizing.
- Variability tests performed on 40 samples representing all defined facies types across all pits, under standard cyanide leaching conditions and at a fixed particle size. The intention of these tests was to determine the relative Au recovery from the various pits and facies types.
- Diagnostic tests performed on 10 variability leach residue samples which realised low (<85%) Au recovery. The purpose of these tests was to establish the reason(s) for the poor Au recovery.
- Optimisation tests performed on six composites, each representing the different pits, to establish optimum leaching conditions (grind size, pulp density, and cyanide concentration) and to establish CIL and cyanide detoxification parameters.

- Refractory test programme on one composite made up of unoxidised material from the Daapleu pit only. This was a retrospective programme aimed at determining a suitable flowsheet dedicated to the treatment of the refractive Daapleu ore. Testwork is currently ongoing.

### 1.9.2 Sample Selection

Core selection was completed in consultation with Cube Consulting and site geological personnel.

A total of 3,320 kg of drill core from six different pits was delivered to ALS, Perth, on 2 June 2016. The drill core was contained in 412 individual sample bags within 37 drums. Each sample bag typically contained a 5 m interval of ¼ drill core from the selected drill hole. From these drill core intervals, a series of samples was selected for the programme of comminution and metallurgical testwork. The sample selection was made to accommodate all facies types, different degrees of oxidation, and Au grade variability.

### 1.9.3 Comminution Testwork

The purpose of the comminution testwork programme is to gain a thorough knowledge of the ore physical characteristics to facilitate a crushing and milling circuit design appropriate for the plant throughput and feed type (Mine Plan).

**Table 1.9.1 Comminution Results**

| Sample Test | Pit | Facies | Oxidation | SMC Axb | Ball Work Index (kWh/t) | Levin test kWh/t to 75 mic | Abrasion Index (Ai) |
|---|---|---|---|---|---|---|---|
| #1 | Daapleu | 601 | U | 59.2 | 7.1 | - | 0.1915 |
| #2 | Daapleu | 811 | U | 54.7 | 9.8 | - | 0.3017 |
| #3 | Daapleu | 101 | U | 42.6 | 17.0 | - | 0.1359 |
| #4 | Daapleu | 601 | PO / U | 286.9 | 6.7[1] | - | 0.0199 |
| #5 | Daapleu | 601 | U | 66.0 | 4.9 | - | 0.1619 |
| #6 | Daapleu | 601 | U | 61.3 | 8.5 | - | 0.2874 |
| #7 | Daapleu | 201 | O | - | 18.7[1] | 3.0 | n/a |
| #8 | Daapleu | 601 | U | 98.8 | 7.3 | - | 0.0444 |
| #9 | Daapleu | 101 | U | 42.5 | 12.3 | - | 0.0774 |
| #10 | Gbeitouo | 101 | PO | - | | 10.0 | n/a |
| #11 | Ity Flat | 211 | O | - | 28.5[1] | - | n/a |
| #12 | Ity Flat | 501 | PO | 93.0 | | 10.0 | 0.2283 |
| #13 | Ity Flat | 901 | U | 52.8 | 9.5 | - | 0.0744 |
| #14 | Mont Ity | 201 | O | - | 13.8[1] | 0.5 | n/a |
| #15 | Mont Ity | 401 | U | 32.3 | 12.4 | - | 0.0306 |
| #16 | Mont Ity | 211 | O / PO / U | - | | 10.0 | n/a |
| #17 | Walter | 201 | O | - | | 1.5 | n/a |
| #18 | Zia NE | 201 | O | - | 30.7[1] | 8.9 | n/a |
| #19 | Zia NE | 201 | O | - | 14.5[1] | 0.3 | n/a |
| #20 | Zia NE | 301 | O | - | 14.7 | - | n/a |

**Note 1 – Modified ball work index**

Results indicate the following:

- There is marked variability in the ore which generally depends on oxidation state.
- Unoxidised samples show low to moderate competency and moderate abrasion index.
- Partially oxidised samples show low competence and low abrasion index.
- Oxidised samples were generally soft with very low abrasion index although the Levin test was difficult to interpret due to the presence of some soft micaceous minerals.
- This data was submitted to Orway Mineral Consultants for interpretation and this is discussed in Section 1.10 below.

### 1.9.4 Metallurgical Variability Testwork

The objective of the metallurgical variability tests was to compare the Au extraction for each sample under the same set of standard cyanidation leach conditions for each of the various facies types and across the different pits. The results of this variability programme would then be used as a basis for the subsequent Diagnostic and Optimisation testwork programmes. A series of variability leach tests was conducted on selected samples from the ¼ core delivered from Ity to ALS. Results presented in Table 1.9.2.

**Table 1.9.2 Variability Leach Test Results**

| Composite | Pit | Au Head ppm | Au Calc ppm | As calc ppm | Au Rec % | Ag Rec % | As Rec % | Cu Rec % | NaCN Kg/t | Lime Kg/t |
|---|---|---|---|---|---|---|---|---|---|---|
| #1 | Daapleu | 0.87 | 1.01 | 0.1 | 85.1 | 86.5 | 1.71 | 2.60 | 0.18 | 0.28 |
| #2 | Daapleu | 1.95 | 2.49 | 793 | 82.8 | 54.0 | 0.38 | 9.27 | 0.25 | 0.21 |
| #3 | Daapleu | 0.74 | 0.95 | 1,395 | 45.1 | 70.1 | 0.34 | 12.2 | 0.25 | 0.21 |
| #4 | Daapleu | 1.89 | 1.64 | 423 | 94.5 | 52.7 | 0.71 | 10.5 | 0.43 | 0.26 |
| #5 | Daapleu | 0.75 | 0.97 | 1,045 | 49.4 | 74.1 | 0.47 | 14.7 | 0.29 | 0.20 |
| #6 | Daapleu | 0.50 | 0.50 | 281 | 69.7 | 33.4 | 0.36 | 10.3 | 0.25 | 0.24 |
| #7 | Daapleu | 1.26 | 1.92 | 725 | 94.8 | 64.2 | 0.68 | 27.2 | 0.39 | 0.47 |
| #8 | Daapleu | 1.09 | 0.72 | 251 | 87.5 | 39.6 | 0.52 | 10.3 | 0.25 | 0.16 |
| #9 | Daapleu | 2.62 | 2.84 | 251 | 89.8 | 78.3 | 0.34 | 10.3 | 0.39 | 0.21 |
| #10 | Daapleu | 0.76 | 0.80 | 1,070 | 91.3 | 30.9 | 0.04 | 0.25 | 0.62 | 1.90 |
| #11 | Daapleu | 20.1 | 21.0 | 20,219 | 46.6 | 52.6 | 0.34 | 30.4 | 0.60 | 0.32 |
| #12 | Daapleu | 0.98 | 1.35 | 820 | 94.1 | 30.0 | 0.06 | 3.47 | 0.39 | 0.84 |
| #13 | Daapleu | 1.33 | 1.77 | 694 | 56.0 | 57.2 | 0.58 | 13.3 | 0.29 | 0.21 |
| #14 | Daapleu | 1.94 | 1.96 | 1,165 | 44.4 | 57.7 | 0.44 | 8.30 | 0.25 | 0.21 |
| #15 | Daapleu | 6.12 | 6.34 | 1,890 | 75.4 | 86.4 | 0.80 | 39.1 | 0.36 | 0.42 |
| #16 | Daapleu | 2.09 | 2.62 | 191 | 96.6 | 65.5 | 0.75 | 14.9 | 0.44 | 0.42 |
| #17 | Daapleu | 1.32 | 1.58 | 324 | 86.7 | 80.8 | 1.10 | 12.4 | 0.39 | 0.21 |
| #18 | Daapleu | 1.49 | 1.48 | 2,932 | 68.2 | 87.1 | 1.10 | 24.2 | 0.44 | 0.26 |
| #19 | Gbeitouo | 0.51 | 0.41 | 854 | 82.8 | 10.6 | 0.48 | 7.11 | 0.51 | 0.16 |
| #20 | Gbeitouo | 0.70 | 0.90 | 4,075 | 93.4 | 73.6 | 0.01 | 0.44 | 0.59 | 2.63 |
| #21 | Gbeitouo | 0.32 | 0.40 | 50 | 82.6 | 96.2 | 0.15 | 29.4 | 1.28 | 1.66 |
| #22 | Gbeitouo | 0.47 | 0.74 | 500 | 91.9 | 81.6 | 0.03 | 24.7 | 0.61 | 3.23 |
| #23 | Ity Flat | 0.97 | 1.06 | 5.1 | 87.7 | 96.6 | 1.53 | 64.5 | 2.96 | 2.37 |
| #24 | Ity Flat | 7.02 | 5.95 | 5.1 | 96.1 | 70.4 | 1.48 | 3.35 | 0.72 | 0.28 |
| #25 | Ity Flat | 7.85 | 10.3 | 5.2 | 88.9 | 83.0 | 2.95 | 18.1 | 5.11 | 1.42 |
| #26 | Ity Flat | 10.9 | 10.8 | 5.1 | 97.3 | 92.6 | 1.49 | 6.22 | 1.02 | 3.13 |
| #27 | Ity Flat | 0.10 | 0.13 | 20 | 84.9 | 78.4 | 0.38 | 21.9 | 0.47 | 0.33 |
| #28 | Ity Flat | 3.14 | 3.79 | 5.1 | 84.7 | 37.5 | 1.49 | 0.06 | 0.66 | 1.80 |
| #29 | Mont Ity | 16.3 | 18.3 | 5.1 | 98.1 | 93.9 | 1.52 | 4.02 | 0.39 | 0.26 |
| #30 | Mont Ity | 7.30 | 7.44 | 5.1 | 81.7 | 76.2 | 1.48 | 4.04 | 3.51 | 1.19 |
| #31 | Mont Ity | 4.33 | 6.14 | 40 | 98.4 | 89.9 | 0.78 | 26.0 | 1.64 | 3.37 |
| #32 | Mont Ity | 1.73 | 1.62 | 5.2 | 77.1 | 67.6 | 3.16 | 85.6 | 2.88 | 2.11 |
| #33 | Walter | 0.31 | 0.34 | 5.1 | 97.1 | 62.5 | 1.55 | 1.74 | 0.67 | 6.42 |
| #34 | Walter | 2.80 | 2.35 | 10 | 92.3 | 37.0 | 0.78 | 46.0 | 0.51 | 1.42 |
| #35 | Walter | 1.35 | 1.50 | 5.3 | 97.3 | 69.0 | 5.81 | 24.8 | 2.20 | 5.90 |
| #36 | Zia NE | 2.40 | 2.71 | 5.1 | 95.6 | 55.5 | 2.35 | 0.12 | 0.61 | 4.21 |
| #37 | Zia NE | 9.69 | 10.9 | 5.1 | 96.9 | 60.7 | 2.36 | 0.05 | 0.49 | 4.21 |
| #38 | Zia NE | 1.26 | 1.14 | 10 | 97.4 | 48.3 | 0.76 | 1.61 | 0.55 | 5.27 |
| #39 | Zia NE | 0.98 | 1.14 | 5.1 | 95.6 | 41.7 | 2.33 | 0.10 | 0.56 | 3.69 |
| #40 | Zia NE | 2.62 | 2.97 | 5.1 | 98.0 | 5.08 | 1.53 | 0.58 | 0.62 | 3.69 |

*** Leach tests conducted at 30% w/w solids pulp density due to high pulp viscosity at 40% w/w solids.**

### 1.9.5 Diagnostic Leach

To determine the cause of low extraction among some of the samples a diagnostic leach program on the residues was conducted on 10 low recovery samples including seven from Daapleu primary.

- Results for Daapleu show the bulk of losses are arsenopyrite locked.
- Ity flat shows significant free cyanidable gold suggesting liberation may be an issue.
- Mont Ity shows the gold is distributed across a number of minerals and may be difficult to extract.

### 1.9.6 Optimisation Testwork

Following completion of the comminution and variability testwork programmes, a separate optimisation testwork programme was implemented to optimise the grind size and subsequent cyanide leach conditions for the proposed CIL circuit. This optimisation programme also included oxygen uptake tests, a preg-robbing index test, standard carbon loading (kinetic) tests, cyanide detoxification tests, and slurry rheology tests.

***Head assays***

Composite head assays are shown below.

**Table 1.9.3 Composite Head Assays**

| Composite | Composite #1 Daapleu Oxide | Composite #3 Ity Flat | Composite #4 Mont Ity | Composite #5 Walter | Composite #6 Zia NE | Composite #7 Daapleu Primary |
|---|---|---|---|---|---|---|
| Au av. (ppm) | 1.36 | 5.03 | 7.87 | 0.82 | 1.18 | 1.69 |
| Ag (ppm) | 2 | 4 | 8 | <2 | <2 | 4 |
| As (ppm) | 910 | <10 | <10 | <10 | <10 | 1,620 |
| Ba (ppm) | 1,080 | 240 | 300 | 620 | 300 | 1,060 |
| C org (%) | 0.03 | 0.09 | 0.09 | 0.06 | 0.12 | <0.03 |
| Ca (%) | 0.96 | 6.10 | 11.2 | 3.54 | 1.30 | 1.99 |
| Cu (ppm) | 80 | 4200 | 1720 | 740 | 575 | 45 |
| Hg (ppm) | 0.2 | 0.1 | 0.2 | <0.1 | <0.1 | <0.1 |
| Mn (ppm) | 500 | 900 | 1,800 | 800 | 2,200 | 900 |
| Ni (ppm) | 60 | 40 | 40 | 80 | 60 | 60 |
| S (%) | 1.58 | 0.50 | 0.56 | 0.82 | 0.04 | 1.84 |

Key observations are as follows:

- Gold grade in Composite 4 is highly variable suggesting coarse gravity gold.
- Significant copper in Ity Flat and Mont Ity.

- Silver:gold ratio is lower at between 1 and 2.
- Arsenic in both Daapleu oxide and primary is significant.

***Leach Optimisation***

Preg-robbing tests showed indices were low for all samples. Leach optimisation results showed the following:

- Gravity gold recovery varied from 11% in Daapleu primary to 49% in Ity Flat with an average of 31%.
- Grind optimisation indicated there was some benefit in finer grind for four of the six composites with Ity Flat and Daapleu primary showing no significant benefit grinding from 106 to 75 micron.
- Leach results were generally rapid with most of the leaching in the first eight hours with the exception of Mont Ity.
- Optimisation testwork on pulp density showed all samples except Ity Flat and Mont Ity could be leached at 50% solids. These two samples were highly viscous and needed to be leached at 40 - 45% solids.
- Cyanide optimisation testwork showed a minimum of 350 ppm cyanide was needed to maximise recovery. This number rose to 500 ppm for Mont Ity.
- Oxygen uptake tests showed all samples except Ity Flat and Mont had a moderate level of oxygen demand. These two samples had very high demand.

***Rheology***

- Rheology testwork on the six composites showed Ity Flat, Zia NE and Mont Ity to have high apparent viscosity suggesting blending and/or dilution will be required in processing.

***Carbon adsorption testwork***

- Sequential carbon adsorption testwork on six composites indicated Fleming constants in the normal range for gold varying from a k value of 111 to 205 and an exponent (n) of 0.65 to 0.81. Fleming constants for silver vary from a k value of 74 to 135 and an exponent (n) of 0.58 to 0.66. The modelling of the adsorption circuit for the process plant has been based on k and n values for gold of 150 and 0.60, respectively, and for silver values of 100 and 0.50 have been used. The Ity Flat and Mont Ity samples show a high copper loading on the carbon. .

### 1.9.7 Cyanide Detoxification Tests

Cyanide detoxification testwork to optimise reagent addition is ongoing. However, preliminary results indicate:

- $CN_{WAD}$ concentrations in the discharge liquor following cyanide detoxification for each of the composites may be reduced to <3 ppm $CN_{WAD}$.
- The comparatively high $CN_{WAD}$ concentrations in the Mont Ity and Walter composites (591 mg/L and 345 mg/L) require greater residence times than the nominal 60 minutes required for all remaining composite.
- The target $CN_{WAD}$ value of <5 ppm in the cyanide detoxification discharge liquor is able to be achieved with the addition of 120% to 240% stoichiometric addition of SMBS.

Soluble copper is present in sufficient concentration to catalyse the cyanide detoxification reactions for all composites except the Daapleu oxide and Daapleu primary composites.

### 1.9.8 Refractory Testwork Programme

A separate refractory testwork programme was implemented on the Daapleu primary ore composite (Composite #7) only to establish any specific processing requirements for this refractory material in the light of the results from the diagnostic leach. This program will evaluate fine grinding, flotation and separate leaching, pressure oxidation and roasting at a preliminary level to establish the potential benefits.

Preliminary results are reported in Section 13 of this report but these results are not included as part of this process plant flowsheet. The refractory program will be reported separately from this study.

### 1.9.9 Metallurgical Recovery

The metallurgical results of the leach tests conducted on each of the 40 variability samples for the six different deposits and using a common set of cyanide leach criteria ($P_{80}$ 75 µm, 40% w/w solids pulp density, 500 ppm NaCN) the optimum grind, pulp density, and cyanidation conditions are summarised in Table 1.9.4.

**Table 1.9.4 Variability Tests – Gold and Silver Extractions**

| Pit | No. of Tests | Au Extraction, % | | | Ag Extraction, % | | |
|---|---|---|---|---|---|---|---|
| | | Minimum | Maximum | Average | Minimum | Maximum | Average |
| Daapleu | 18 | 44.4 | 96.6 | 75.4 | 30.0 | 87.1 | 61.2 |
| Gbeitouo | 4 | 82.6 | 93.4 | 87.7 | 10.6 | 96.2 | 65.5 |
| Ity Flat | 6 | 84.7 | 97.3 | 90.0 | 37.5 | 96.6 | 76.4 |
| Mont Ity | 4 | 77.1 | 98.4 | 88.8 | 67.6 | 93.9 | 81.9 |
| Walter | 3 | 92.3 | 97.3 | 95.6 | 37.0 | 69.0 | 56.2 |
| Zia NE | 5 | 95.6 | 98.0 | 96.7 | 5.1 | 60.7 | 42.2 |

***Testwork Results Applied to the Pit Schedule***

The approach taken to predicting the metallurgical recoveries for gold and silver is to take the average extractions obtained from the leach variability tests and then apply these to the mine schedule for the appropriate ore type to yield a weighted recovery value for each year. As silver values have not been modelled in the mine schedule, for the purposes of obtaining a predicted silver recovery it has been assumed that the silver grade is three times that of the corresponding gold value.

Table 1.9.5 presents a summary of the 3 Mtpa mine schedule, tonnage contribution from each pit, and gold and silver extractions obtained from the variability testwork for each ore source. (Gold and silver extraction values from the stockpile reserves have been assumed to be the average of the extractions across all pits).

**Table 1.9.5 Mine Schedule and Gold and Silver Extractions**

| | Daapleu Oxide | Daapleu Primary | Gbeitouo | Ity Flat | Mont Ity | Walter | Zia NE | Stock-piles |
|---|---|---|---|---|---|---|---|---|
| LoM Au, g/t | 1.05 | 2.43 | 1.35 | 2.09 | 2.23 | 1.22 | 1.24 | 1.08 |
| LoM Ag, g/t | 3.14 | 7.29 | 4.05 | 6.27 | 6.68 | 3.66 | 3.71 | 3.25 |
| % of LoM Tonnage | 31.5 | 15.5 | 6.3 | 3.9 | 5.4 | 4.5 | 11.8 | 21.0 |
| LoM Tonnes, Mt | 12.93 | 6.42 | 2.58 | 1.60 | 2.23 | 1.86 | 4.84 | 8.60 |
| Au Extraction, % | 85.2 | 60.1 | 87.7 | 90.0 | 88.8 | 95.6 | 96.7 | 86.3 |
| Ag Extraction, % | 58.4 | 65.6 | 65.5 | 76.4 | 81.9 | 56.2 | 42.2 | 63.7 |

Using this information, and assuming soluble gold and silver losses of 0.01 mg/L and 0.03 mg/L respectively and a CIL pulp density of 50% w/w solids, it is possible to predict the gold and silver recoveries for each year of the life-of-mine. These recoveries are summarised in Table 1.9.6.

**Table 1.9.6 Predicted Life of Mine Metallurgical Recoveries**

| Mine Year | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Au Extraction, % | 78.4 | 76.8 | 80.1 | 81.2 | 79.4 | 85.0 | 86.7 | 88.9 | 87.6 | 83.7 | 83.7 | 83.7 | 83.7 | 84.6 |
| Ag Extraction, % | 62.6 | 64.1 | 64.3 | 69.1 | 61.8 | 61.8 | 60.3 | 55.7 | 54.8 | 57.9 | 57.9 | 57.9 | 57.9 | 57.6 |

This yields life-of-mine average gold and silver recoveries of 83.1% and 60.2%, respectively.

***Overall Metallurgical Recovery***

Based on the gold and silver metallurgical recovery values obtained from the mine schedule based recoveries (83.1% and 60.2%), it is reasonable to predict that the overall gold and silver extractions for the life-of-mine will be between 78% to 89% for gold and between 55% and 69% for silver.

## 1.10 Process Plant

### 1.10.1 Design Philosophy

The proposed process plant design for the Ity CIL Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations that are utilised in other Endeavour Mining Corporation Operations and that are well proven in industry.

The key criteria for equipment selection are the suitability for duty and the projected mine life of the operation without unnecessarily compromising reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a compact footprint to minimise construction costs.

The Ity CIL plant will process a range of ore types (oxide, transition and fresh) with variable ore characteristics, gold grades and metallurgical treatment requirements. The primary ores are significantly more competent than the oxide ores.

Initial pit optimisation and scheduling has indicated that the largest pit is Daapleu which contains up to 50% of the mineable reserve and contains both oxide and primary ores. Testwork results have indicated that Daapleu primary ore contains significant arsenopyrite and a standard cyanide leach will only extract 60 - 65% of contained gold. By contrast the oxide ores are free-milling with leach extraction of 90 - 96%.

The pit optimisation work indicated that parallel processing of the Daapleu primary ores with free milling oxide ores gave the best return. As a result, the higher grade Daapleu material will be mined and processed for the first six years of the mine, with lower grade Daapleu material stockpiled during this time and processed from Years 9 to 14. Mill feed requirements will be supplemented successively with oxide material from Ity Flat, Mont Ity, Gbeitouo, Walter and Zia NE.

Initially higher grade ores will be processed with lower grade ores stockpiled. When high grade ores are exhausted, low grade ores and existing stockpiles and heap leach pads will be processed.

The Ity ores contain significant silver which has influenced the design of the elution circuit and goldroom.

The key project and ore specific design criteria that the plant design must meet are:

- 3 Mtpa of blended ore; 58% primary and 42% oxide (LOM).
- Crushing plant mechanical availability of 80%.
- Mechanical availability for the remainder of the plant of 91.3% supported by crushed ore storage and stand-by equipment in critical areas.
- Sufficient automated plant control to minimise the need for continuous operator intervention and allow manual override and control if and when required.

A process design criteria (1955-000-PRPDC-0001) document has been prepared incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical testwork and comminution circuit modelling. The design document is included in Appendix 6.1 and forms the basis for the design of the processing plant and required site services.

The selected milling circuit has been sized to accommodate the 85$^{th}$ percentile in terms of competence of the ores to be treated.

### 1.10.2 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing to produce a crushed product size of 100% passing ($P_{100}$) 300 mm ($P_{80}$ of 160 mm).
- Transfer conveyor feeding a coarse ore stockpile (12 hours live). Ore reclaim via apron feeders.
- Two stage SAG / Ball milling in closed circuit with hydrocyclones to produce a $P_{80}$ grind size of 75 µm and including recrushing of pebbles from the SAG mill.
- Gravity concentration and removal of coarse gold from the milling circuit recirculating load and treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to doré.
- Pre-leach thickener to increase the slurry density to the carbon in leach (CIL) circuit to minimise CIL tankage, improve slurry mixing characteristics and reduce overall reagent consumption.
- A CIL circuit incorporating six CIL tanks containing carbon for gold and silver adsorption.
- Fifteen tonne split Anglo (AARL) elution circuit, electrowinning and gold smelting to recover gold and silver from the loaded carbon to produce doré.
- Tailings treatment incorporating cyanide destruction using sulphur dioxide/air, followed by arsenic precipitation and stabilisation.
- Tailings pumping to the tailings storage facility (TSF).

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet is shown in flowsheet 000-PRPFD-0001.

### 1.10.3 Key Process Design Criteria

The key process design criteria listed in Table 1.10.1 form the basis of the detailed process design criteria and mechanical equipment list.

**Table 1.10.1 Summary of Key Process Design Criteria**

| | | LOM Blend | Source [1,2,3,4,5,6,7] |
|---|---|---|---|
| Ore Blend | | 25% Daapleu primary/ 75% mixed oxides | Mining Solutions |
| Crushing Circuit Capacity | t/y | 3,000,000 | Lycopodium |
| Plant Throughput – Nominal. | t/y | 3,000,000 | Lycopodium |
| Gold Head Grade - Range | g Au/t | 1.0-2.5 | Endeavour |
| Gold Head Grade - Design | g Au/t | 2.5 | Endeavour |
| Overall Gold Recovery | % | 59-97 | Test |
| Silver Head Grade - Design | g Ag/t | 8.5 | Endeavour |
| Design Silver Recovery | % | 63 | Test |
| Crushing Plant Utilisation | % | 80 | Lycopodium |
| Milling/CIL Plant Utilisation | % | 91.3 | Lycopodium |
| ROM Ore Top Size | mm | 800 | Assumed / OMC |
| Crushing Work Index (CWi) | kWh/t | 15 | Assume / OMC |
| Bond Ball Mill Work Index (BWi) | kWh/t | 18.7 | Testwork / OMC |
| Axb | | 40 | Assumed / OMC |
| SG | | 2.83 | Testwork / OMC |
| Abrasion Index (Ai) | | 0.30 | Testwork / OMC |
| Comminution Circuit | | SABC | OMC |
| Crush Size, $P_{80}$ | mm | 160 | OMC |
| Grind Size, $P_{80}$ | µm | 75 | Test |
| SAG Mill Pinion Power | kW | 3,370 | OMC |
| Ball Mill Pinion Power | kW | 5,052 | OMC |
| Grinding Media Consumption | kg/t | 0.97 | OMC |
| Gravity Gold Recovery | % | 31 | Test/Assumed |
| Gravity Silver Recovery | % | 20 | Assumed |
| Pre-leach Thickener Solids Loading | $t/m^2.h$ | 0.7 | Testwork/Assume |
| Leach /CIL Residence Time | hrs | 34 | Lycopodium |
| Leach Slurry Density | % w/w | 50 | Testwork |
| Number of CIL Tanks | | 6 | Lycopodium |
| Cyanide Consumption[5] | kg/t | 0.59 | Testwork |
| Quicklime Consumption[6] | kg/t | 0.32 | Testwork |
| Elution Circuit Type | | Split AARL | Lycopodium |
| Elution Circuit Capacity | t | 15 | Lycopodium |
| Frequency of Elution | strips / week | 6.6 | Lycopodium |

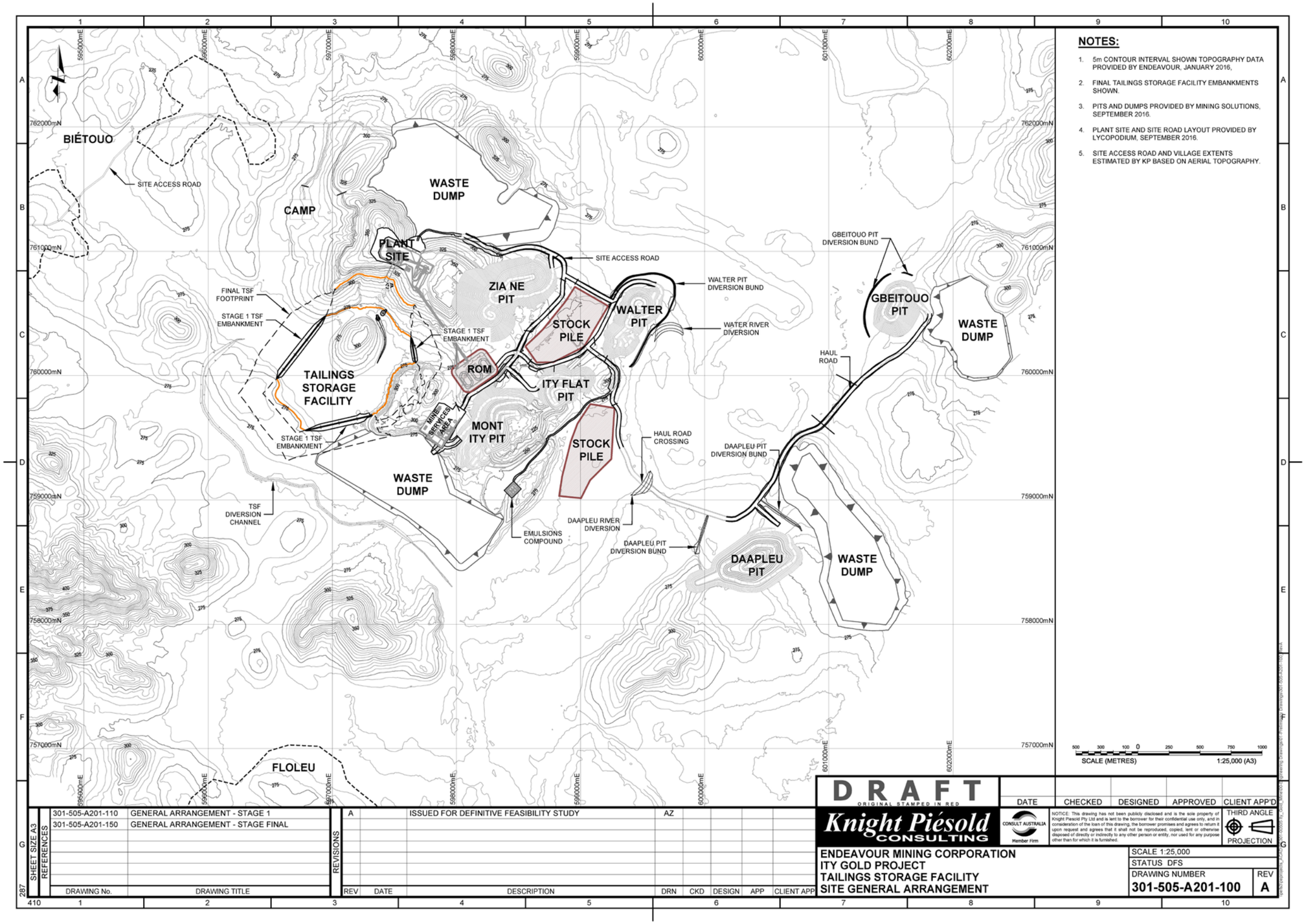
**Notes:**

1. **Mining Solutions refers to advice from the nominated mining subconsultant.**
2. **'Lycopodium' refers to Lycopodium experience or generally accepted practice.**
3. **'Testwork' refer to metallurgical testwork conducted.**
4. **'OMC' refers to advice from Orway Mineral Consultants.**
5. **Cyanide consumption makes allowance for 100 ppm residual cyanide in the CIL tail solution.**
6. **Lime consumption based on 90% CaO.**
7. **Endeavour refers to advice/agreement from Endeavour**

## 1.11 Infrastructure

The overall site development plan is shown in Figure 1.11.1 and Drawing 301-505-A201-100. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, mine services area, mine open pit and mine waste dump.

**Figure 1.11.1 Site Layout**



The main process plant has been laid out to optimise use of the saddle in which the plant is positioned and to minimise footprint. It is relatively long and narrow with overall dimensions of 320 m x 130 m. The local terrain has dictated that the plant site is approximately 800 m north-west of the primary crusher which is located just south of the proposed Zia NE pit boundary and north of the existing Ity pit.

This location was selected as it provides access for heavy vehicle movements from both the northern pits and the Daapleu pit via the river crossing. This location will ensure dust picked up by the prevailing winds from the ROM pad will not be blown over other plant or infrastructure. The accommodation camp is located west of the process plant.

The main access road approaches the site from the west and the layout provides easy access for personnel and material movements.

### 1.11.2 Buildings and Infrastructure

The process plant site will be fenced to clearly prevent animal access and deter access by unauthorised persons. Road access into the fenced area will be through a manned checkpoint. Site security is based on concentric lines of fencing / control. Security fencing will surround the accommodation camp and general site infrastructure. Monitored high security fencing will surround the process plant.

The existing road will be upgraded to an all weather, and free draining carriageway to provide access for the delivery of equipment, materials and services to the site.

The closest connection point to grid power is a connection to the national grid at Danané, approximately 58 km from the Project site. The project will construct the 90 kV single circuit lattice tower transmission line to connect to the national grid. The power station will supply the main HV switchroom inside the processing plant from which power will be distributed.

An accommodation camp will be located approximately 1 km west of the process plant and will provide accommodation for 200 salaried and security staff not originating from the local area.

Containerised accommodation units will be used as a supplement to the existing operations camp to accommodate the construction team pending early completion of elements of the permanent camp.

The permanent camp will then be used for Endeavour and EPCM contractor staff with any surplus capacity made available to senior contractor personnel if available.

During construction contractors will be required to provide their own accommodation for their construction workforce. An area will be set aside for temporary contractor camps adjacent to the construction site. Alternatively some contractors may choose to source temporary accommodation in nearby towns and villages.

A vendor package modular potable water treatment plant including filtration, ultra-violet sterilisation and chlorination will be installed. Potable water will be stored in the plant potable water tank and will be reticulated to the plant buildings, site ablutions, safety showers and other potable water outlets.

Effluent from all water fixtures in the process plant, mine services area and accommodation camp will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a vendor package sewage treatment plant system located at the process plant.

Site buildings will be 'fit for purpose' industrial type structures. The workshop and warehouse will be constructed of a concrete slab on ground with structural steel frame and metal cladding. Offices and amenity buildings will be prefabricated structures.

The following facilities will be located in a fenced area adjacent to the process plant:

- Gatehouse with turnstile and entry boom gate control.
- Main administration building.
- First aid / clinic.
- Administration junior staff mess.
- Warehouse and stores.
- Contract laboratory.
- Emergency power station.
- Fuel storage facility (by the fuel supplier).
- Mine services area.

The following buildings will be located inside the process plant high security area:

- Security access building and change room including laundry.
- Plant offices, training room, junior staff mess and ablutions.
- Plant workshop including small store, welding bay and overhead crane.
- Engineering offices and ablutions.
- Reagent storage area.
- Goldroom.

Internal communications and IT services shall be via a site wide fibre optic network.

One of the local mobile phone providers will be contracted to install facilities on site and provide a link into the local, national and international telecommunication network.

A radio network will be established with dedicated operational, security and emergency channels.

A local ground station will be installed to provide global satellite voice and data connection.

### 1.11.3 Tailings Storage and Site Water Balance

The Tailings storage facility (TSF) has been designed for a total capacity of 46 Mt at an average annual throughput rate of 3 Mt/y. A site selection study was conducted to evaluate several sites relative to the SNC-Lavalin selected site designed for the previous unpublished Draft FS. A cross-valley TSF was selected and is to be located 200 m south-west of the proposed plant site, with a water / diversion dam upstream of the TSF.

The TSF valley storage will consist of a single cell formed by three multi-zoned earthfill embankments, comprising a total footprint area (including the basin area) of approximately 77 ha for the Stage 1 TSF increasing to 160 ha for the final TSF. The TSF embankment will be constructed in stages to suit storage requirements.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the Northern and Southern embankments.

Supernatant water will be removed from the TSF via submersible pumps located within decant towers. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

## 1.12 Operating Costs

### 1.12.1 Mining Operating Costs

Table 1.12.1 presents the Life of Mine operating costs split by cost area and is displayed in US millions of dollars.

**Table 1.12.1 Mining Operating Costs**

| Mining Operating Costs | US$M |
|---|---|
| Pit Loading | 21.91 |
| Pit Hauling | 54.76 |
| Ore Rehandle Load | 12.83 |
| Ore Stockpile Haul | 7.23 |
| Ancillary | 24.65 |
| Drilling | 47.14 |
| Blasting | 42.85 |
| Personnel (indirect) | 42.12 |
| Mine Preparation and Rehabilitation | 4.07 |
| Grade Control | 18.66 |
| Support Equipment | 39.88 |
| Operating Overheads | 6.30 |
| **Subtotal – Mining** | **322.40** |

Table 1.12.2 presents the total operating costs for mining, the total material movement and unit rates for the preproduction and sustaining periods. Unit rates are exclusive of all capital costs.

**Table 1.12.2 Mining Operating Cost Unit Rates**

| Mining Operating Costs | Preproduction | Sustaining | Total |
|---|---|---|---|
| **Total Cost ($M)** | | | |
| Open Pit Mining – Ore | 3.57 | 97.13 | 100.70 |
| Open Pit Mining - Waste | 7.14 | 164.78 | 171.92 |
| Ore Rehandle | 0.10 | 49.69 | 49.79 |
| **Total Material Movement (Mt)** | | | |
| Ore Tonnes Mined | 0.96 | 31.48 | 32.44 |
| Waste Tonnes Mined | 3.38 | 84.15 | 87.52 |
| Rehandle Ore | 0.16 | 59.69 | 59.85 |
| **Unit Rates ($/t)** | | | |
| Open Pit Mining | 2.47 | 2.27 | 2.27 |
| Ore Rehandle & Crusher Feed | 0.63 | 0.83 | 0.83 |

### 1.12.2 Process Plant Operating Costs

Process plant operating costs for the Ity Gold Project have been developed with input from Endeavour.

The operating costs have been compiled from information sourced by Lycopodium and Endeavour:

- Manning levels and pay rates advised by Endeavour.
- Consumable prices from supplier budget quotations, Endeavour advice from Ity or the Lycopodium database.
- Cyanide and lime consumptions and gold extractions based on laboratory testwork results.
- Modelling by Orway Mineral Consultants (OMC) for crushing and grinding energy and consumables, using ore characteristics measured during the testwork.
- First principle estimates based on typical operating data.

The following exchange rates, as advised by Endeavour, have been used for the preparation of the operating cost estimate:

- US$1.00 = A$1.300 (Australian Dollar).
- US$1.00 = 593 XAF (CFA Franc).
- US$1.00 = 13.62 ZAR (South African Rand).
- US$1.00 = 0.891 € (Euro).

The operating cost estimates are presented at summary and detail level. The estimate is considered to have an accuracy of ±15%, is presented in US dollars (US$) and is based on prices obtained during the third quarter of 2016 (3Q16).

The detailed operating cost estimate is available.

The operating cost estimate includes all direct costs to allow production of gold bullion. All costs associated with mining are excluded from this estimate (and are included in Section 4) except mining personnel camp and catering costs and grade control laboratory costs, which are included in the general and administration cost and laboratory cost.

The treatment plant availability includes allowance for scheduled and unscheduled maintenance.

***Summary***

The battery limits for the process plant operating costs are as follows:

- Ore delivered to the ROM bin (ROM pad FEL costs are included in the Mining costs).
- Tailings discharge from the tails pipeline to the tailings storage facility (TSF).
- Gold bullion in plant goldroom safe.

Operating costs have been developed for the seven ore sources (six pits with Daapleu split into oxide and primary ore).

Process plant and administration operating costs were developed using the parameters specified in the Process Design Criteria based on 'design' annual throughputs of 3,000,000 tpa using a blended feed as described in Mining Section 4. These costs are summarised in Table 1.12.3 with the detailed estimates available separately.

**Table 1.12.3 Summary Operating Costs by Ore Type at 3 Mtpa Design Throughput**

| Cost Area | DAA Oxide | | Gbeitouo | | Ity Flat | | Mont Ity | |
|---|---|---|---|---|---|---|---|---|
| | US$/annum | US$/t | US$/annum | US$/t | US$/annum | US$/t | US$/annum | |
| Power (exc grinding) | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 6,018,257 | 2.01 |
| Power (grinding) | 3,240,750 | 1.08 | 4,677,197 | 1.56 | 4,302,635 | 1.43 | 4,190,358 | 1.40 |
| Operating Consumables | 13,035,449 | 4.35 | 16,729,300 | 5.58 | 19,194,345 | 6.40 | 21,335,548 | 7.11 |
| Maintenance | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 3,008,973 | 1.00 |
| Laboratory | 726,567 | 0.24 | 726,567 | 0.24 | 726,567 | 0.24 | 726,567 | 0.24 |
| Process & Maintenance Labour | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 3,002,248 | 1.00 |
| **Total Processing Cost** | **29,032,243** | **9.68** | **34,162,542** | **11.39** | **36,253,025** | **12.08** | **38,281,950** | **12.76** |
| Administration Labour | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 4,139,176 | 1.38 |
| General & Administration Costs | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 4,389,241 | 1.46 |
| Total General & Administration | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 8,528,418 | 2.84 |
| **Total Cost including G&A** | **37,560,661** | **12.52** | **42,690,960** | **14.23** | **44,781,443** | **14.93** | **46,810,368** | **15.60** |

| Cost Area | Walter | | Zia NE | | DAA Prim | | LOM Average * |
|---|---|---|---|---|---|---|---|
| | US$/annum | US$/t | US$/annum | US$/t | US$/annum | US$/t | US$/t |
| Power (exc grinding) | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 2.01 |
| Power (grinding) | 3,541,476 | 1.18 | 3,120,945 | 1.04 | 6,111,831 | 2.04 | 1.62 |
| Operating Consumables | 15,461,241 | 5.15 | 14,273,543 | 4.76 | 12,234,805 | 4.08 | 4.84 |
| Maintenance | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 1.00 |
| Laboratory | 726,567 | 0.24 | 726,567 | 0.24 | 726,567 | 0.24 | 0.24 |
| Process & Maintenance Labour | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 1.00 |
| **Total Processing Cost** | **31,758,761** | **10.59** | **30,150,532** | **10.05** | **31,102,681** | **10.37** | **10.72** |
| Administration Labour | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 1.38 |
| General & Administration Costs | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 1.46 |
| Total General & Administration | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 2.84 |
| **Total Cost including G&A** | **40,287,179** | **13.43** | **38,678,950** | **12.89** | **39,631,099** | **13.21** | **13.56** |

The General and Administration costs in Table 1.12.3 exclude all head office and off site costs.

Costs were split into fixed and variable components to enable then to be used to derive annual costs for changing ore blends and/or throughput with further development of the mine plan. The split of fixed and variable costs by ore type are presented in Table 1.12.4.

**Table 1.12.4 Fixed / Variable Operating Costs by Ore Type at 3 Mtpa Design Throughput**

| Cost Area | DAA Oxide | | Gbeitouo | | Ity Flat | | Mont Ity | |
|---|---|---|---|---|---|---|---|---|
| | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t |
| Power (exc grinding) | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 |
| Power (grinding) | 0 | 1.08 | 0 | 1.56 | 0 | 1.43 | 0 | 1.40 |
| Operating Consumables | 0 | 4.35 | 0 | 5.58 | 0 | 6.40 | 0 | 7.11 |
| Maintenance | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 |
| Laboratory | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 |
| Process & Maintenance Labour | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 |
| **Total Processing Cost** | **12,111,758** | **5.64** | **12,111,758** | **7.35** | **12,111,758** | **8.05** | **12,111,758** | **8.72** |
| Administration Labour | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 |
| General & Administration Costs | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 |
| Total General & Administration | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 |
| **Total Cost including G&A** | **20,640,176** | **5.64** | **20,640,176** | **7.35** | **20,640,176** | **8.05** | **20,640,176** | **8.72** |

| Cost Area | Walter | | Zia NE | | DAA Prim | | LOM | |
|---|---|---|---|---|---|---|---|---|
| | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t |
| Power (exc grinding) | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 |
| Power (grinding) | 0 | 1.18 | 0 | 1.04 | 0 | 2.04 | 0 | 1.62 |
| Operating Consumables | 0 | 5.15 | 0 | 4.76 | 0 | 4.08 | 0 | 4.84 |
| Maintenance | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 |
| Laboratory | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 |
| Process & Maintenance Labour | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 |
| **Total Processing Cost** | **12,111,758** | **6.55** | **12,111,758** | **6.01** | **12,111,758** | **6.33** | **12,111,758** | **6.68** |
| Administration Labour | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 |
| General & Administration Costs | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 |
| Total General & Administration | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 |
| **Total Cost including G&A** | **20,640,176** | **6.55** | **20,640,176** | **6.01** | **20,640,176** | **6.33** | **20,640,176** | **6.68** |

***Qualifications***

The operating cost estimates presented are exclusive of the following:

- Mining costs and other areas not specifically mentioned in the detailed estimate.
- All head office (off-shore) costs, including head office labour.
- Any impact of foreign exchange rate fluctuations.
- Any escalation from the date of the estimate.
- Any contingency allowance.
- All withholding taxes and other taxes (Import duties on consumables allowed).
- Tailings storage facility future lifts and rehabilitation.
- Government monitoring / compliance costs.
- Gold refining and bullion transport and in-transit security of gold from site.

The operating cost estimate includes the following:

- Import duties on consumable unit costs (in the consumables cost).

Costs for the preparation and assaying of 100 mine grade control samples per day and routine laboratory tests on the site water samples (in the laboratory cost).

## 1.13 Capital Costs

### 1.13.1 Summary

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this study report. Additional estimate details for items costed are available on request.

Summary mine capital costs (developed by Calsta for Mining Solutions) are included in the estimate tables below.

Knight Piésold provided quantities for the tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds with rates applied by Lycopodium to derive the capital estimate.

All costs are expressed in US dollars ($) unless otherwise stated and based on 3Q2016 pricing. The estimate is deemed to have an accuracy of ±15%.

Where costs used in the estimate were provided in other than US dollars the following exchange rates were used:

- US$1.00 = A$1.300 (Australian Dollar).
- US$1.00 = 593 XAF (CFA Franc).
- US$1.00 = 13.62 ZAR (South African Rand).
- US$1.00 = 0.891 € (Euro).

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Endeavour for scope and accuracy.

The initial project capital cost was estimated at US$306.66 million. The capital estimate is summarised in Table 1.13.1.

**Table 1.13.1 Initial Overall Capital Cost Estimate Summary (US$, 3Q2016, ±15%)**

| Main Area | Initial Capital (US$M) |
|---|---|
| Treatment Plant | 63.3 |
| Reagents and Services | 9.2 |
| Infrastructure & Tailings | 46.2 |
| Mining | 58.6 |
| Construction Distributables | 24.5 |
| Subtotal | 201.8 |
| Management Costs | 15.8 |
| Owners Project Costs | 59.0 |
| Owners Operations Costs | 3.7 |
| Contingency | 26.4 |
| **Project Total** | **306.7** |

### 1.13.2 Mining

Calsta prepared a Mining Specific Cost Model for Mining Solutions based on an Owner operated mining strategy.

Mining capital costs comprise of initial and replacement heavy mining equipment and light mining equipment purchases.

The Komatsu brand has been selected for the mining fleet except for blast hole drills which are Sandvik brand. Mining equipment pricing is as contracted for Endeavour Mining's Houndé Gold Project.

A summary of equipment type and number is included in Table 1.13.2.

**Table 1.13.2 Heavy Mining Equipment (HME) Summary**

| HME | Make | Model | Initial Quantity | Replacement Quantity | Total |
|---|---|---|---|---|---|
| 100T Excavator | Komatsu | PC1250-8R | 3 | 3 | 6 |
| Front End Loader | Komatsu | WA600 | 3 | 3 | 5 |
| Off High Trucks | Komatsu | HD785-7R | 14 | 4 | 17 |
| Track Dozers | Komatsu | D375A-6R | 3 | | 3 |
| Wheel Dozers | Komatsu | WD600-6 | 2 | | 2 |
| Graders | Komatsu | GD825A-2 | 2 | | 2 |
| Watercarts | Komatsu | HD465-7R | 4 | | 4 |
| Blast Hole Drills | Sandvik | D1550 | 0 | 6 | 6 |
| Tyre Handler | Komatsu | WA500-6 | 1 | | 1 |

Miscellaneous and light mining equipment capital costs have been sourced from a combination of quotations from suppliers and Calsta database. Heavy mining equipment pricing is as contracted for Endeavour Mining's Houndé Gold Project.

Table 1.13.3 presents the Life of Mine capital costs split by both heavy mining equipment and light mining equipment. Costs are US millions of dollars.

**Table 1.13.3 Mining Capital Costs**

| Mining Operating Costs | US$M |
|---|---|
| Heavy Mining Equipment | 39.77 |
| Light Mining Equipment | 19.62 |
| **Subtotal – Mining** | **59.39** |

Table 1.13.4 presents the capital costs by year split by heavy mining equipment and light mining equipment in millions of dollars.

**Table 1.13.4 Mining Capital Costs by Year**

| Mining Operating Costs | Yr00 | Yr01 | Yr02 | Yr03 | Yr04 | Yr05 | Yr06 | Yr07 | Yr08 | Yr09 | Yr10 | Yr11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Heavy Mining Equipment | 27.10 | 3.76 | | 1.64 | 0.86 | | 4.16 | | 0.68 | | | 1.56 |
| Light Mining Equipment | 10.67 | | | 0.56 | | 2.51 | 0.20 | | | 1.47 | 4.22 | |
| **Subtotal – Mining** | **37.77** | **3.76** | | **2.20** | **0.86** | **2.51** | **4.36** | | **0.68** | **1.47** | **4.22** | **1.56** |

Calsta has included a cost allowance of $0.18M for first fills for all lubricants and spare parts which is incorporated into the capital cost of each machine purchase. The cost for lubricant first fills is based on lubricant capacities and oil costs provided by the original equipment manufacturer. Spare part first fill costs have been calculated on the FOB price provided by the OEMs.

### 1.13.3 Process and Infrastructure

The capital cost estimate includes the following scopes of work:

- Process facility as described in Section 17.
- Infrastructure as described in Section 18.
- Tailings storage facility as described in Section 18.
- Installation costs, EPCM costs and contractor distributable costs.
- Owners costs including first fill and opening stocks of reagents and consumables as well as operating and insurance spares.
- Site earthworks, tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds.
- Mobile equipment.

Exclusions include the following:

- Project sunk costs.
- Import duties and taxes on the assumption that the Project will be exempt.
- Escalation.

The capital cost estimate was prepared in accordance with Lycopodium's standard estimating procedures and practices. The basis and methodology are summarised in Table 1.13.5.

## Table 1.13.5 Capital Cost Estimate Basis

| Description | Basis |
|---|---|
| **Site** | |
| Geographical Location | Actual site |
| Maps and Surveys | Good topographical data available |
| Geotechnical Data | Preliminary |
| **Process Definition** | |
| Process Selection | Fixed |
| Design Criteria | Fixed |
| Flowsheets / Plant Capacity | Fixed |
| P&IDs | Not Required as suitable 'go-by' costs available from Lycopodiums database |
| Mass Balances | Fixed |
| Equipment List | Fixed |
| **Process Facilities Design** | |
| Equipment Selection | Selection based on duty and budget pricing provided by vendors. |
| General Arrangement Drawings | Fixed |
| 3D model | Preliminary to a level of detail suitable for FS |
| Piping Drawings | Not required |
| Electrical Drawings | HV SLD. LV drawings not required |
| Specifications / Data Sheets | Preliminary for BQRs |
| **Infrastructure Definition** | |
| Existing Services | Not relevant other than use of exploration camp for 'pioneer' accommodation |
| Design Basis | Fixed |
| Layout | Fixed |
| Bulk Earthworks – Plant-site, Mine Services & Camp | Volume estimated from 3D Models. |
| Tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds | Bulk Quantities provided by Knight Piésold |
| Detailed Earthworks | Allowances for under pad excavation and backfill to prepare site for concrete works |
| Concrete Installation | Estimated from the layout and similar projects of comparable scale. Concrete (wet) supply rates and installation rates applied from project specific BQRs. |
| Structural Steel | Quantities estimated from the layout and similar projects of comparable scale. Supply and install rates applied from project specific BQRs. |
| Platework & Small Tanks | Quantities provided in the mechanical equipment list. Large item quantities estimated from reference projects. Smaller items compared to database. Supply and install rates applied from project specific BQRs |
| Tankage Field Erect | Quantities provided in the mechanical equipment list. Supply and install rates applied from project specific BQRs. |
| Mechanical Equipment | Quantities provided in the mechanical equipment list. Costs from responses to BQRs from reputable suppliers for all equipment packages with a value nominally >$50,000. Costs for low value items taken from the Lycopodium database. |
| Mining Fleet | Refer mining cost estimate. |
| Power Supply | Estimate provided by ECG. |
| Conveyors | Concrete & structural estimated from reference projects and layout. Mechanicals supply pricing from current database and installation rates applied from responses to BQRs. |
| Plant Piping General | Factored off mechanical costs derived by completed project actuals. |
| Overland Piping | Size and specification based on engineering selection. Quantity based on site layout. Rates based on recent market inquiries. |
| Electrical General | Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs |
| Electrical HV | Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs |
| Commodity Rates – General | Appropriate rates from responses to project specific BQRs. |
| Installation Rates – General | Appropriate rates from responses to project specific BQRs based on preliminary contracting strategy. |
| Heavy Cranes | Requirements estimated based on largest lifts and likely duration. |
| Freight General | Combination of rates per freight ton & factors. |
| Contractor Mobilisation / Demobilisation | Appropriate rates from responses to project specific BQRs, adjusted to suit the final project scope. |
| Fencing | Costed based on measured length and rate. |
| EPCM | Scope and deliverables based estimate. |
| Vendor Representatives | Estimated. |
| **Owner's Costs** | |
| Site Establishment | Requirements estimated using base rates. |
| Construction Facilities | Allowance based on projects of a similar size. |
| Opening Stocks, First Fill Reagents and Consumables | Estimated from consumption rates and costs in operating cost estimate. |
| Working Capital | Estimated from costs in operating cost estimate and likely commissioning and ramp up schedule. |
| Spares | Historical data from similar sized plants. |
| Owner's Project Team | Client estimate. |
| Project Insurances and Permits | Client estimate. |
| Sterilisation Drilling | Assumed in sunk costs. |
| Land Compensation | Client estimate. |
| Community Relations | Client estimate. |
| Plant preproduction expenses | Estimated from operating cost estimate and likely commissioning / ramp up schedule. |
| Training | Estimated from operating cost estimate manning schedule |
| Owners Team Expenses | Client estimate. |
| Duties and Taxes | Excluded on the assumption the Project will be exempt. |
| Escalation | Excluded. |

***Qualifications***

The estimate is subject to the following qualifications:

- All labour rates, materials and equipment supply costs are current at 1Q15. Contingency has been allowed based on the quality of the information, however no allowance for escalation has been included.
- Construction contractor rates include mobile equipment, vehicles, fuel, construction power and consumables for the duration of construction. Potable water and raw water supply will be provided by the Client and available at site for the use by contractors.
- Accommodation, meals and mobilisation / demobilisation / R&R flights of construction contractor personnel are incorporated in the contractor indirect labour rates on the basis of individual contractors making their own accommodation arrangements.
- Meals and accommodation for the owner and EPCM teams have been allowed in the estimate.
- Project spares have been based on other projects of similar size.
- A commissioning assistance crew is allowed for in the EPCM allowance.
- PLC programming for the process plant has been allowed for in the EPCM allowance.
- Site supply of power and raw water (for operations and construction), sewage removal and treatment, communications network for construction facilities are included in the infrastructure costs.

***Contingency***

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope. Contingencies do not include allowances for scope changes, escalation or exchange rate fluctuations.

Contingency is an integral part of an estimate and has been applied (after careful analysis) to all parts of the estimate, i.e. direct costs, indirect costs, services costs, etc.

***Owner's Costs***

The following items are included in the Owner's costs:

- Owner's project management team.
- Owner's team costs including project insurances.
- Operating and capital spares holdings.

- First fill and opening stocks of reagents and consumables.
- Cost of preproduction labour, training and operational readiness.

***Spares***

Spares have been taken as an allowance of the capital costs and benchmarked against the spares expenditure on projects of a comparable scale. A minimalist approach has been assumed, with spares stocks progressively expanded during operations.

***First Fill and Opening Stocks of Consumables***

Quantities for opening stocks and first fill consumables have been assembled from basic principles and using the Project design criteria. Unit rates are based on budget quotations solicited from suitable suppliers.

***Exclusions***

The following is excluded from the overall project capital costs:

- Duties / taxes / fees.
- Project sunk costs (allowance made in the Cash Flow Model).
- Project escalation.

## 1.14 Project Execution

The proposed approach to project implementation is to engage a suitable Engineering, Procurement and Construction Management (EPCM) contactor for design and construction management of the process plant and infrastructure, which will then be handed over to an Owner's operating team, and to engage an experienced mining contractor for the development of mine infrastructure and provision of ongoing drill and blast and mine operating services under an Owner's mine technical team.

The Owner's team will be progressively expanded to widen its skills and knowledge base to meet the needs of the Project. The team of full and part time personnel will manage both the onshore and offshore activities of the principal EPCM contractor and specialist subcontractors as well as providing specialist technical input into the Project design.

Personnel required for key onshore mine operations roles will be filled early to contribute to mine design and manage the mine contractor undertaking development of the mining infrastructure and open pit.

Endeavour's in-country management, administration and services department will manage environment and community issues and prepare the site for the coming influx of operating personnel.

A preliminary schedule shows the Project can be executed within 20 months from EPCM award. Key project milestone dates are provided in Table 1.14.1.

**Table 1.14.1 Project Schedule**

| Milestone Description | Planned Completion |
|---|---|
| EPCM Award Date | Week 0 |
| Ball Mill Contract Award | Week 8 |
| Bulk Earthworks Mobilisation | Week 10 |
| Camp Contractor Mobilisation | Week 20 |
| Plantsite Civil Contractor Mobilisation | Week 20 |
| SMP Contractor Mobilisation | Week 45 |
| E&I Contractor Mobilisation | Week 49 |
| Mills Mechanical Installation Commencement | Week 58 |
| Power On | Week 71 |
| First Gold Pour | Week 85 |

## 1.15 Environmental

A number of environmental studies and investigations have been undertaken for the project, commencing in 2013 by consultants including:

- Envitech-CI, Roche (Canada)
- 2D Consulting utilising experts from the Daloa University
- SNC Lavalin (Canada).

This work resulting in the submission of three separate ESIA submissions resulting in three environmental permits. SEIA submissions and granted permits included the following:

- The first ESIA submitted for approval in September 2013 to cover the renewal of the PE26 mining permit, and the CIL plant and TSF to be constructed on PE26. This was approved by the Agence Nationale De l'Environnement (National Environment Agency) on 25 October 2013. The first Environment Permit for the CIL project was granted in December 2013.
- A second ESIA submitted in August 2015, for Daapleu and Gbeitouo mining exploitation and was approved on 11 September, 2015, with the subsequent environmental permit granted in June 2016.
- The third ESIA was submitted in December 2015 to cover mining related infrastructure including bridge, Cavally River diversion, haulage roads and dyke protection bunds. This was approved on January 29, 2016, with the environmental permit granted in July 2016.

Following the completion of the FS, comprehensive Environmental and Social Management Plans will need to be written setting out the standards required for the CIL project, and outlining procedures and practices to be adopted during the development and operation of the project.

A preliminary Closure Plan outlining the closure and rehabilitation of the site has been included in each submitted ESIA. A more detailed Closure Plan will be developed during the initial phase of operation, following consultation with relevant stakeholders.

## 1.16 Economic Analysis

The results of the economic model show robust results. Applying a long term gold price of $1,250/oz on a flat line basis from the commencement of production, the after-tax NPV5% is $410.9 million, IRR is 35.9% and project payback period is 2.1 years. The life of mine average cash cost per ounce is $528, net of silver credits, and with the addition of royalties and sustaining capital, the life of mine average AISC/oz is $603. Over the first five years of production, the average AISC/oz is $507.

Project gold production averages 113,656 ounces per year over the life of the project with an average of 164,989 ounces per year over the first five years.

The project has low sensitivity to capital and operating costs, but is typically sensitive to gold prices. At $1,250/oz gold price the IRR is 35.9%, and the project maintains an IRR of 19.5% at $1,000/oz gold price.

The summary of the cash flow model (Condensed Annual Cash Flow) is included in Section 14.5.

Table 1.16.1 presents a summary of the production information on which the cash flow model is based and the key project financial measures.

**Table 1.16.1 Summary of Economic Analysis Results**

| Financial Summary | Units | Value |
|---|---|---|
| LOM tonnage ore processed | kt | 41,042 |
| LOM strip ratio | w:o | 2.1 |
| LOM feed grade processed | Au g/t | 1.42 |
| LOM gold recovery | % | 83.1% |
| LOM gold production | oz | 1,561,902 |
| Production period | Years | 13.7 |
| Gold, LOM average annual production | oz | 113,656 |
| Cash costs per ounce, LOM average, net Ag credits | $/oz | 528 |
| AISC per ounce, LOM average | $/oz | 603 |
| Upfront capital cost | $M | 281.2 |
| | | |
| Project Years 1 to 5: | | |
| Gold, average annual production | oz | 164,989 |
| Cash costs per ounce, net Ag credits | $/oz | 446 |
| AISC per ounce | $/oz | 507 |
| | | |
| Using $1,250/oz gold price and after-tax: | | |
| Internal rate of return | % | 35.9% |
| Net present value - 0% discount rate | $M | 607.2 |
| Net present value - 5% discount rate | $M | 403.0 |
| Net present value - 10% discount rate | $M | 263.6 |
| Payback period | Years | 2.1 |

A sensitivity analysis was performed on the after-tax cash flows by varying key variables (gold price, Capex, Opex) to +/- 20% of the base case cash flow and each sensitivity was performed independently. Figure 1.16.1 and Figure 1.16.2 show the sensitivity analysis on the after-tax $NPV_{5\%}$ and IRR economic indicators.

**Figure 1.16.1 $NPV_{5\%}$ Sensitivity**

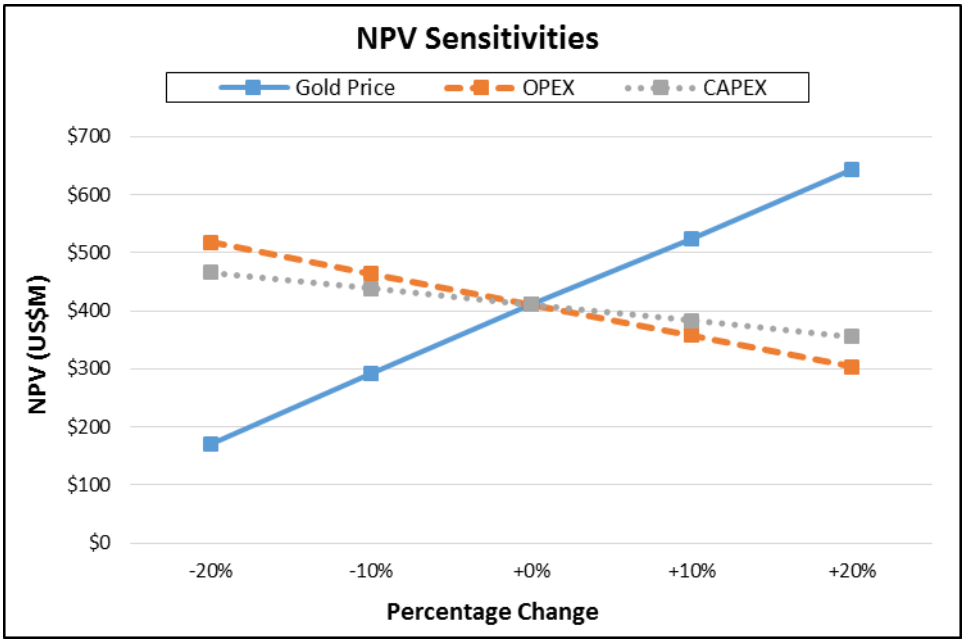
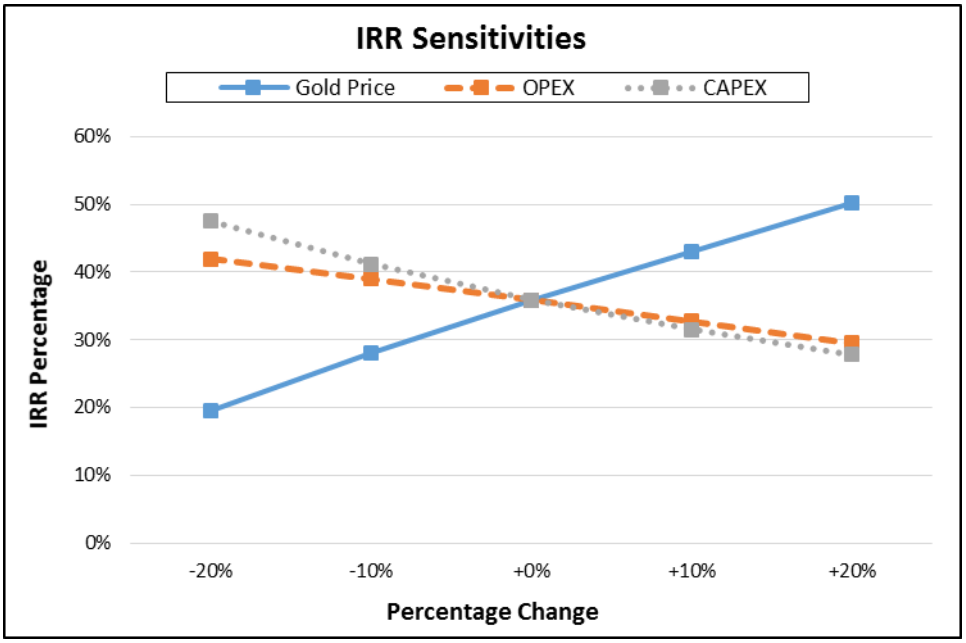



**Figure 1.16.2 IRR Sensitivity**



## 1.17 Conclusions and Recommendations

### 1.17.1 Geology

Several additional targets at different stages of exploration have been identified. The additional resources when fully defined have the potential to increase the CIL project mineral resources and mineral reserves base. While the Verse Ouest stockpile has been classified as Inferred in the FS, recommended additional drilling has been completed and Cube believes that this ore (5.88 Mt at 0.99 g/t Au for 187 koz) can be reclassified as Indicated in a future revision of the resource estimate.

Successful selective mining of the Mineral Resource will depend on the ability to discriminate between ore and waste at elevated cut-offs within the mineralised envelope. Visual selectivity is likely to be limited and diligent grade control practices including RC grade control and geological mapping of the benches will be critical to accurately defining material above the selected cut-off grades.

As additional gold recovery data is collected from the shear hosted deposits such as Daapleu and Gbeitouo, there is the opportunity to improve the understanding of the distribution of material with poor metallurgical recovery. Geo-metallurgical domaining within these deposit styles is critical and needed in order to increase the confidence in the estimation of this important modifying parameter for mine planning and processing options.

All historical and modern drilling data has been validated and migrated into a recently introduced AcQuire database. Some outstanding data issues were identified during the estimation and require further investigation and possible adjustment in the live database. In addition, current QA/QC procedures should be continued with minor improvements as recommended in the technical report in order to ensure the quality of data is sufficient for future Mineral Resource updates.

### 1.17.2 Mining

It is recommended to review and update the pit designs as necessary following further geotechnical investigations and reassessment of slope angles for all the pits.

The current production schedule and reserves assume an end of heap leach pit surface for the start of the CIL operations. This assumption needs to be reviewed for the timing of the transition between heap leach and CIL operations. The planning of heap leach and CIL operations in parallel will be required if both operations are to coexist in the transitional period.

### 1.17.3 Pit Wall Geotechnical

Geotechnical investigation boreholes are required to be drilled at Ity to increase the level of confidence and reliability of geotechnical assessment. Geotechnical investigation boreholes should be designed to provide coverage of all sectors of the proposed pits and to refine preliminary base case wall design parameters.

Data obtained from further drilling would be combined with existing data to enable more reliable interpretation of conditions and derivation of appropriate wall angles. Geotechnical data obtained from future drilling should be cross-checked with findings and assumptions presented in this assessment report.

It is recommended that Endeavour carries out further geotechnical investigation drilling prior to committing to final Daapleu, Gbeitouo, Zia NE, Mont Ity, Ity Flat and Walter open pit designs.

It is considered essential that ongoing design re-assessments, based on information obtained using observational techniques are made during mining. It may be necessary to make local design adjustments during pit development.

### 1.17.4 Metallurgy and Processing

The testwork programme completed achieved its objectives of defining a process flowsheet and engineering parameters that are sufficient for design of the plant and subsequent capital and operating cost estimates.

The comminution testwork showed significant variability in ores both in terms of alteration and on a pit by pit basis and has indicated the need to blend ores.

Gold recoveries vary significantly dependent on mineralogy with the high arsenic ores giving significantly lower extraction.

Investigations into further processing of the Daapleu primary ore have indicated flotation and finer grinding may improve gold extraction.

### 1.17.5 Infrastructure

***Plant Site***

Final site geotechnical data should be incorporated. In addition, sterilisation drilling of the plant and TSF site should be confirmed.

The plant layout should be optimised as part of the detailed design to take maximum advantage of the terrain.

***Camp Location***

The existing accommodation village on site will need to be replaced due to its age, size and proximity to CIL plant location. The selected site for the new camp requires review to ensure alternative locations are considered.

***Power***

Power supplies in Africa can pose risks for reliability, which has forced operating mines to install additional diesel generation to maintain plant availability. CIE Energies in Côte d'Ivoire considers their power supply to be reliable. An alternative 225 kV OHPL from Man, should be considered as an alternative to the currently selected 90 kV from Danane during the detailed design phase. This option is potentially more reliable, with significantly more load capability. The additional cost of this option is considered low at this point.

***Tailings Storage Facility (Knight Piésold)***

Several potential locations were identified and assessed during the FS. The final FS location was selected based on its convenient proximity to the processing facility, as well as considerable potential capital and operating cost advantages in comparison to other options.

An optimisation of the TSF in terms of location and footprint will need to be undertaken. The validation of the design of TSF should consider key issues such as geotechnical investigation, construction material sourcing, construction staging and stability.

***Water Management (Knight Piesold)***

A dewatering bore installation schedule has been developed based on the current mine plan. This includes the installation of an additional 100 bores over the LOM. The bore numbers are based on a median yield of 11 $m^3/h$ obtained from the existing Ity pit dewatering bores. The number of dewatering bores could be reduced by the installation of higher yielding bores.

A more detailed analysis of the existing geological data is recommended for the refinement and improvement of the proposed bore locations and pit inflow predictions. Potential sources of additional information include the existing block geological model, additional borehole logs and geophysical data.

Additional field investigations and testing should take place in a phased manner and successful bores could be adopted as long term dewatering bores.

The groundwater model should be updated as additional investigation and monitoring data are obtained.

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# 2.0 INTRODUCTION

## 2.1 Terms and References

Endeavour Mining Corporation (Endeavour), a Canadian reporting issuer, has requested Lycopodium Minerals Pty Ltd (Lycopodium) coordinate and prepare an independent Technical Report on their Ity CIL Project (The Project) in accordance with Canadian National Instrument 43-101 requirements. This Technical Report is based on the outcomes of the Ity CIL Project Feasibility Study (the FS) completed in October 2016.

## 2.2 Sources of Information

This report is based in part on information provided by Endeavour and other consultants including documents, data and reports compiled by Endeavour management and technical staff and previous reports by other independent experts (refer Sections 3 and 27).

## 2.3 Contributing Consultants

Lycopodium Minerals Pty Ltd is a subsidiary of Lycopodium Limited, an Australian listed public corporation, and has provided engineering and project management services to the international mining industry for over 20 years. The Perth, Western Australia, office of Lycopodium Minerals undertook the compilation of this report.

Sections of this report were authored or co-authored by Qualified Persons from Cube Consulting, Knight Piésold, Lycopodium Minerals, Mining Solutions and Peter O Bryan & Associates.

Cube Consulting is a mining and geological consulting group based in Perth, Western Australia. Cube provided the geological assessment and the mineral resource estimate.

Knight Piésold is an international firm of consulting engineers and scientists with over 90 years of experience in the mining and power sectors. The firm has over 25 offices worldwide and specialises in environmental, civil and geotechnical engineering associated with mining projects in a wide range of commodities and geographical locations. The Australian offices of KP conducted studies for tailings management, surface water management and groundwater evaluations for the project.

Calsta is a multi-disciplinary consulting company headquartered in Perth, Western Australia and specialising in mining and processing and mobile equipment. Calsta provided the definition of mining fleet and services and the mining capital and operating cost estimates for Mining Solutions.

Mining Solutions is a Perth-based mining consultancy specialising in mining assessment, pit optimisation and mine planning and scheduling. Mining Solutions provided the mineral reserve estimates, mine plans and schedules for the project.

Peter O'Bryan & Associates is a mining consultancy based in Perth, Western Australia and specialising in mining geomechanics. Peter O'Bryan & Associates provided the mining geotechnical aspects of this report.

## 2.4 Site Visits and Inspections

QP Mr David Gordon (Lycopodium) has not visited site. However, Mr Alastair Holden, Principal Process Engineer with Lycopodium and a representative of Mr David Gordon has undertaken a site visit to Ity (28 April to 5 May 2016) to supervise the metallurgical sample selection and packing for shipment. He also visited the existing plant site and facilities. In addition, Mr Stephen Monk, Principal Mechanical Engineer and design manager for the study has visited site (19 June to 21 June 2016) and assessed the proposed plant site. Lycopodium has previously completed studies and projects in Côte D'Ivoire and is familiar with the country / region.

Mr Tamer Dincer of Mining Solutions has visited the site (29 May to 31 May 2016) and viewed the core. He has also viewed the site access, the topography of the site and the local infrastructure.

Mr Peter Butler from Calsta and a representative of QP Mr Tamer Dincer has visited the site (29 May to 31 May 2016) and viewed the core and the operation. He has also viewed the site access, the topography of the site and the local infrastructure.

Mr David Morgan of Knight Piésold has undertaken a site visit (19 June to 21 June 2016) and viewed the site as well as discussing the water management and site geotechnical issues with site personnel.

Mr Mark Zammit, a representative from Cube Consulting, has undertaken a site visit (18 June to 22 June 2016) and was able to review drilling and sampling laboratory procedures as well as examine the mineralisation occurrence and associated geological features.

QP Mr Peter O'Bryan (Peter O'Bryan & Associates) has not visited site. However, Mr Scott Campbell, Geological Engineer with Peter O'Bryan and Associates and a representative of Mr Peter O'Bryan has undertaken a site visit to Ity (19 June to 21 June 2016) and assessed the mining related geotechnical issues.

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# 3.0 RELIANCE ON OTHER EXPERTS

## 3.1 General Statement

In preparing the feasibility study report on which this Technical Report is based the Qualified Persons have relied on previous work, reports and opinions from a number of sources including the following:

- SNC Lavalin Draft Study Report (628121-0000-30RA-0001 Feb 2016).
- Endeavour supplied geological data including drillhole data, QA/QC report and data, geological interpretation, topographic, overburden and weathering boundary surfaces.
- Slope angle criteria were provided by Peter O'Bryan & Associates.
- Knight Piésold water and geotechnical reports.
- Environmental and Social Impact Assessment.
- ALS metallurgical testwork report (17202).
- OMC comminution report (7753).
- Endeavour financial model.

In all the above cases the content of the contributory information and reports have been reviewed and are believed to be appropriate and factually correct for the purposes of this Technical Report.

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# 4.0 PROPERTY DESCRIPTION AND LOCATION

## 4.1 License Location

Endeavours Ity licence area is situated in the south western region of Côte d'Ivoire, Africa, as shown in Figure 4.1.1, and is approximately 35 km from the Liberian border. Administratively it is located in the prefecture of ZOUAN-HOUNIEN and Blolequin.

**Figure 4.1.1 Ity Project Location**



The Ity project site is approximately 700 km northwest of Abidjan, which serves as Côte d'Ivoire's economic capital. The project site is accessible via paved road from Abidjan, passing through the capital Yamoussoukro, Daloa and Duekoué. It is also accessible by plane from Abidjan to Man and then from Man to the site by road. From Duekoué, two roads access the project from both north and south. The north access is through Man and then on to Danané and Zouan–Hounien where a 15 km unsealed road maintained by SMI leads to the village of Ouyatouo. Southern access is through Guiglo and Toulepleu.

## 4.2 Mineral Rights

The SMI and La Mancha Côte d'Ivoire (LMCI) properties, which are the subject of this report, include the following two permits which contain the gold deposits:

- Permis de' Exploitation PE26 -Mining Permit.
- Permis de' Recherche PR609 (Formerly PR61) - Exploration Permit.

To comply with the conditions and changes in the new Côte d'Ivoire 2014 Mining Code, SMI abandoned PR61, and La Mancha resources requested and was granted a reassignment of the permit as PR609. Exploration permits confer on holders, within boundaries, surface and depth, the exclusive right to explore for mineral substances as well as the free disposal of the products extracted during exploration.

Mining permit PE26 comprises an area of 25 km² and includes the Ity mine, the deposits ZiaNE and Walter, the Aires decommissioned heap leach pads, dumps Verse Ouest and Teckraie, and the surrounding areas most of which are west of the Cavally River. Mining permit PE26 was first granted in 1989 as Arrêté 197/MINES/DM/(14 October 1989), and has been renewed several times, most recently as Arrêté 8-002/MIM/DGMG granted on 28 August 2014 for 10 years from 14 November 2014. The boundary coordinates for PE26 are summarised in Table 4.2.1.

**Table 4.2.1 Boundary Coordinates for Mining Permit PE26**

| Longitude (West) | Latitude (North) |
|---|---|
| 8"08'30" | 6"53'30" |
| 8"05'46" | 6"53'30" |
| 8"05'46" | 6"50'44" |
| 8"08'30" | 6"50'44" |

Exploration Permit PR609 (formerly PR61) is situated over parts of the Zouan-Hounien and Blolequin prefectures. The permit has an area of 153 km² and is orientated northeast-southeast, and is comprised of two sub portions with orebodies Gbeitouo and Daapleu and prospects Yacetouo, Morgane, Mlambopleu to the east of the Cavally River, and Floleu to the west of the Cavally River.

The original exploration permit PR61 was first issued in 1995 and was extended in two year intervals until September 12, 2014. It was replaced with PR609 in July 2015 and is valid for a period of four years. The boundary coordinates for PR609 are summarised in Table 4.2.2.

**Table 4.2.2 Boundary Coordinates for Exploration Permit PR609**

| Longitude (West) | Latitude (North) |
|---|---|
| 7"58'57" | 7"00'16" |
| 7"58'57" | 6"54'10" |
| 8"00'46" | 6"52'16" |
| 8"09'37" | 6"48'50" |
| 8"10'58" | 6"49'53" |
| 8"08'30" | 6"52'00" |
| 8"08'30" | 6"50'44" |
| 8"05'46" | 6"50'44" |
| 8"05'46" | 6"53'30" |
| 8"06'47" | 6"53'30" |

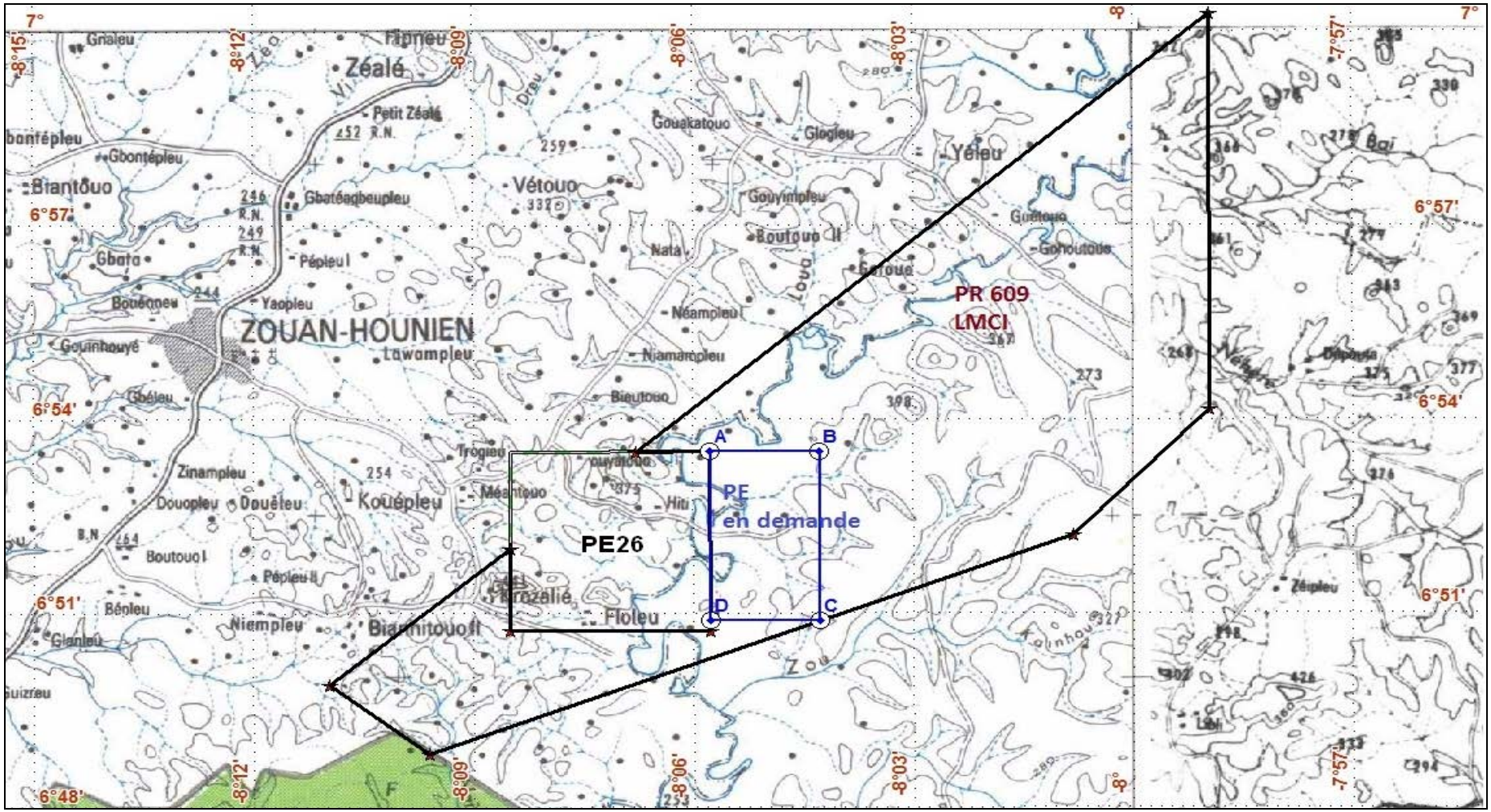
An operating permit, 'PE on demande', (being part of PR609) has also been sought by La Mancha CI in September 2015 to cover the deposits of Daapleu and Gbeitouo for a 10 year period covering an area of 13.2 km² at the coordinates listed in Table 4.2.3. This 'PE on demande' is being assessed by the Director of Mines, and requires the integration of information from this FS for final approval.

**Table 4.2.3 Boundary Coordinates for PE on Demand**

| Point | Longitude (West) | Latitude (North) |
|---|---|---|
| A | 8"05'46" | 6"53'30" |
| B | 8"04'17" | 6"53'30" |
| C | 8"04'17" | 6"50'53" |
| D | 8"05'46" | 6"50'53" |

Permit boundaries for PE26 and PR609, including the aforementioned 'PE on demande', are shown in Figure 4.2.1.

**Figure 4.2.1 Permit Areas SMI, Côte d'Ivoire**



## 4.3 Ownership

Mineral rights of PE26 are 100% owned by SMI, a company incorporated in Côte d'Ivoire, with the registered address at 08 BP 872 Abidjan 08 - Cocody 180-logements Boulevard Latrille, Immeuble Palm Club, 2nd floor.

The shareholders of SMI are as follows:

- Endeavour Mining 55%.
- Société pour le Développement Minier de la Côte d'Ivoire (SODEMI) 30%.
- State of Côte d'Ivoire 10%.
- Keyman Investments (Didier Drogba Group) 5%.

Mineral rights of PR609 are 100% owned by Endeavour Mining, registered in Côte d'Ivoire, at the same address as SMI.

## 4.4 Fiscal and Customs Systems

The main features of the fiscal and customs regime laid down in the General Taxation Code, the new Mining Code and Mining Convention are the following:

- A five year tax holiday on revenue from the date of first production. After that period the corporate tax rate is 25%.
- An exemption from VAT on the acquisition of foreign and Ivorian services and goods and sales in connection with the operations. This exemption extends to contractors.
- An exemption from export duties and taxes on product.
- An exemption from various other taxes, including, without limitation, real estate tax and employer's contribution tax on expat salaries.
- A reduction by 50% of withholding tax to 9% on interest paid on loans by the Exploitation Company.
- An exemption from import duties on imported plant and parts during construction and any expansion phase.
- An exemption from import duties on fuel, lubricants, chemical products, etc. during the operations phase.

## 4.5 Mining Conventions and Codes

In March 2014, Côte d'Ivoire adopted a new Mining Code. There are three relevant statutes namely Law No.2014-138 adopting the new mining code in Côte d'Ivoire dated 24 March 2014 (the Mining Code), Decree No.2014-397 dated 25 June 2014 which implements the Mining Code, and Ordinance No. 2014-148 dated 26 March 2014, which relates to fees, royalties and mining taxes.

The new Mining Code formalises the requirement of the holder of an exploitation permit to enter into a mining convention with the State and sets out what should be included in a convention. The new code aims to enhance mining investment in Côte d'Ivoire, particularly within the gold sector, and strengthen its contribution to local development.

The new mining code also includes new provisions relating to mining conventions for mining exploration operations. The holder of an exploration permit must now sign a mining convention with the State with 60 working days following the granting of a permit. The convention is intended to stabilise the tax and customs system, and set specific conditions for mining operations.

A mining convention was jointly signed on 19 December 2014 by the SMI General Manager, and Côte d'Ivoire state Ministers for Mining, Budget, Finance and Economics. The Convention is effective from 1 January 2015, and will valid for 12 years, with further renewals to be made on expiration in successive periods of then years, as per the conditions of the Mining Code. The Convention includes, without limitation, the following conditions and agreements:

- The state of Côte d'Ivoire guarantees to SMI, the enjoyment of full rights of its properties and the free management of mining operations.
- SMI commits to apply best principles and practices for good governance included in the Equator Principles and the ITIE (Initiative of Transparency in Extractive Industries).
- Free choice of contractors and suppliers with priority to be given to Ivorian providers to the extent the relevant services or products are available at least equivalent quality, price, skills, warranties and delivery times.
- SMI submits to declare all revenue to the Finance Ministry, and to allow Government appointed experts to enter the operation and view any document it requires.
- The State of Côte d'Ivoire will ensure that all documentation provided by SMI at any time will be treated as confidential.
- At the end of mining operations, all existing infrastructures will be returned to the State of Côte d'Ivoire.
- A Local Development Fund will be established, with SMI to contribute 0.5% of annual revenue for use by the Local Mining Development Committee.
- SMI will contribute a sum of 560,000,000 FCFA (approximately US$1.1M) for rehabilitation programs following mine closure.

- An annual contribution of FCFA 25,000,000 (US$50,000) to the Ivorian Mining Training Fund, a fund dedicated to the training of mining professionals.
- Fiscal stability for the duration of the convention.
- Legal stability for the duration of the convention with the exception of social, health, safety and environmental regulation.
- SMI is not obligated to pay any value added tax (VAT) on local and foreign acquired goods and services required by the mining operation, inclusive of SMI contractors and subcontractors.
- The CIL construction phase should not exceed two years from the start of construction, however on application this may be renewed once only for an additional two years.
- At least 80% of the workforce to be Ivorian subject to availability of required qualifications.

## 4.6 Royalties

The State of Côte d'Ivoire is entitled to production royalties as summarised in Table 4.6.1.

**Table 4.6.1 Government Royalty Rates**

| Royalty | Gold Price |
|---|---|
| 3% | Up to US$1,000 |
| 3.5% | US$1,000 – US$1,300 |
| 4% | US$1,300 – US$1,600 |
| 5% | Above US$1,600 |

The royalty is based on the gross revenue from gold produced, after deduction of transportation and refining costs.

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# 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

## 5.1 Accessibility

Yamoussoukro with a population of approximately 380,000 people is the political and administrative capital of Côte d'Ivoire. Abidjan however with a population of approximately 4.8M is the economic capital and gateway into the country, and is located approximately 700 km by road from the Ity Project site.

Abidjan International Airport, also known as Port Bouet or Félix Houphouët-Boigny Airport, receives daily international flights from other African countries, as well as flights to and from Europe and the Middle East through Dubai. Multiple airlines are located at the Abidjan Airport including Air France, Emirates and Ethiopian Airlines.

Twice weekly flights are operated by the national carrier Air Côte d'Ivoire from Abidjan to Man, the closest relatively large city near site with an approximately population of 189,000, and approximately 250 km from site. The Feasibility Study (FS) has included a conceptual level design and cost estimate for an airport to be located near the project site. This will allow charter flights to be conducted as required from Abidjan improving flexibility for operations.

The Ity Mine is located in the prefecture of ZOUAN-HOUNIEN. The project site is accessible via paved road from Abidjan, passing through the capital Yamoussoukro to Daloa and Duekoué. From Duekoué, two roads access the project from both north and south. The north access is through Man and then on to Danané and Zouan–Hounien where a 15 km unsealed road maintained by SMI leads to the village of Ouyatouo. Southern access is through Guiglo and Toulepleu.

SMI's current semi professional and professional Ivorian National Fly in Fly out component of the HL operations workforce currently commute to site via air from Abidjan to Man, with SMI operating a bus from the airport to site.

## 5.2 Climate

Côte d'Ivoire and the regions of Cavally and Tonkpi in particular, are subject to an equatorial-type transition climate. This climate is characterised by abundant rainfall averaging approximately 1,500 to 2,400 mm annually. It is characterised by high humidity (annual average of 85%), high but not excessive temperatures (constant throughout the year with an average of 25°C) and by low temperature ranges (below 5°C). The duration of sunshine varies on average between 1,500 and 2,000 h/year. The climate is punctuated by four seasons of unequal durations:

- A long dry season from December to February.
- A long rainy season from March to July.
- A small dry season from August to September.
- A small rainy season from October to November.

Average annual rainfall is approximately 1,800 mm, based on the onsite rainfall measurements from 1990 to 2014. Operations are carried out year round, however access to exploration sites on the east side of the Cavally river may be restricted during heavy rains. Key rainfall statistics are summarised in Table 5.2.1.

**Table 5.2.1 Key Ity Site Rainfall Historical Statistics**

| Parameter | Historical Averages and Maximum / Minimum Rainfall |
|---|---|
| Average annual total rainfall | 1,867 mm |
| Historical wettest year annual Rainfall (2007) | 2,312 mm |
| Historical driest year annual Rainfall (2007) | 1,202 mm |
| Maximum monthly rainfall (August) | 532 mm |
| Recorded maximum daily rainfall | 140 mm |

Temperatures in the area generally average between 25 - 30°C with key temperature data highlighted in Table 5.2.2.

**Table 5.2.2 Key Ity Site Temperatures**

| Parameter | Average and Maximum / Minimum Temperatures |
|---|---|
| Average annual air temperature | 25.0°C |
| Average monthly air temperature in hottest months (Feb to Mar) | 26.7°C |
| Average monthly air temperature in coldest months (Dec to Jan) | 23.5°C |
| Highest daily Temperature | 33.5°C |
| Lowest daily Temperature | 15.3°C |

## 5.3 Local Resources and Infrastructure

The Ity Project site is located in a remote area of Côte d'Ivoire, and therefore road is the only access to site. While considerable expenditure is occurring within Côte d'Ivoire on particularly road and rail infrastructure, road conditions vary from good between Abidjan and Yamoussoukro, with a progressive deterioration most notably close to site. A modern dual lane tollway stretches approximately 230 km between Abidjan and the capital Yamoussoukro.

While the Sitarail transnational railway operates from Abidjan to Ouagadougou in neighbouring Burkino Faso, no connection or spur line connects to areas close to the Project site.

La Société des Energies de Côte d'Ivoire (CI-ENERGIES) own the National Interconnected Power Transmission System in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government.

Power for the CIL project will be supplied via the installation of a 90 kV 58 km Overhead Power Line (OHPL) from Danané Substation to site.

An abundance of unskilled local workforce is available. Skilled workers, and general services and equipment can be found in Abidjan or in the capital city, Yamoussoukro.

## 5.4 Physiography, Topography, Elevation and Vegetation

The topography of Côte d'Ivoire is characterised by a mix of hills and flat topped plateaus in undulating plains. The elevation rises from sea levels at the coast to the highest peak of Mt Nimba at approximately 1,700 m, located at the western border with Liberia and Guinea.

The geomorphology of the Ity area is typical of rain forest areas. The landscape is often referred to as 'half orange type' due to its succession of rounded hills with convex slopes separated by a well-developed drainage system. The weathering profile is thickest on hilltops, whereas variably truncated profiles are found on the slopes. The bedrock sub outcrops near the drainage channels, which are surrounded by floodable flats. Elevations range from approximately 225 m to over 400 m.

Vegetation consists of secondary forest found on hilltops, and scrubland and small areas of cultivated farmland on the slopes. Rice farms occupy the lowlands. and other cash crops of cocoa, kola nut and coffee are also grown.

Figure 5.4.1 and Figure 5.4.2 show typical views of the immediate project site area, and general forested and hilly terrain around Ity

**Figure 5.4.1 Typical View of Immediate Project Area Landscape**



**Figure 5.4.2 Typical View of Forested Terrain**



## 5.5 Water

Two different groundwater flow systems exist within the project area. The first is a shallow or perched system encountered at the interface between laterite and saprolite soils, with the second being deep with preferential flow through fractures and karstic features. While the original groundwater level has been reported at approximately 10m, the current groundwater level on site is much lower due to dewatering practices. This is also impacted by wet season rainfall.

The Cavally River pictured in Figure 5.5.1, intersects and meanders through the project area. River levels rise as much as 7 m during the highest rainfall periods of the eight month wet season. As a consequence diversion, protection bunds and dykes are required adjacent to existing pit wall boundaries.

**Figure 5.5.1 Cavally River Adjacent Intersection on Project Site**



Site wide hydro geological studies have been undertaken by SNC Lavalin, and Knight Piésold has developed a conceptual site model (CSM) using the Ity Pit groundwater abstraction/level data and the fieldwork undertaken by SNC-Lavalin. The overall water balance is positive, with over 100 dewatering bores to be developed over the Life of Mine (LOM) to manage groundwater to allow mining operations to proceed. As a result groundwater volumes are considered plentiful to supply project and process plant requirements

## 5.6 Waste Disposal Sites

### 5.6.1 Waste Rock

The Feasibility Study level mine design indicates that with an average strip ratio of 2.7 (21 overall ratio including existing stockpiles), a total of 87.5 Mt of waste rock will be produced from six open pits during the project life. Waste rock dumps will be positioned to avoid mineralised zones as well as floodplains that intersect the project areas between pits, while at the same time minimising haul distances.

### 5.6.2 Tailings

A convenient location close to the process plant has been selected for a cross-valley Tailings Storage Facility (TSF), 200 m south-west of the proposed plant site, with a water / diversion dam upstream of the TSF. The TSF valley storage will consist of a single cell formed by three multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 77 ha for the Stage 1 TSF increasing to 160 ha for the final TSF, which is designed to store a total of 46 Mt of tailings.

## 5.7 Process Plant Site

A preferred plant site has been selected on an elevated relatively flat site, currently housing the site geology offices and core storage area, and immediately west of the decommissioned Aires heap leach pads. This site provides relatively central access to project infrastructure and pit areas. The site has been selected for its close proximity to TSF and elevated position away from low lying flood plains. The ROM pad and primary jaw crusher are approximately 700 m from the process plant, with the location selected for its centralised location close to major pits, as well as proximity to Mine services area (MSA). Conveyors connect the short distance from the ROM / crusher to the coarse ore stockpile at the CIL plant.

## 5.8 Flora and Fauna

The site contains a rich flora and fauna The predominant vegetation in the area is a moist evergreen forest, with savannahs integrated into the forest and along its northern edges, as well as pre forest savannahs. Land clearance over time for cropping and hunting purposes, as well as demographic growth have resulted in a degradation of vegetation and soils.

A total of eight special status flora species were found in this vegetation; seven of which are classified as vulnerable and one Near-Threatened on the International Union for Conservation of Nature (IUCN) Red List

Based on existing literature, about 50 species of mammals (micromammals are excluded) are likely to be found in the study area, of which five species have a special status on the IUCN Red List.

Site surveys have identified 167 bird species, of which four species are included on the IUCN Red List

None of the 12 amphibian species found during site surveys is recorded as threatened or vulnerable on the IUCN Red List, with the majority of species found being widely distributed throughout Côte d'Ivoire.

Fish survey campaigns conducted during 2014 identified 66 fish species, of which 6x of the species surveyed considered Threatened or Near Threatened according to the IUCN Red List.

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# 6.0 HISTORY

## 6.1 General History of the Property

The general history of the property up to 2008 is taken from reports by Febvey and Bottero (2013) and Danjou and Furic (2014). These extracts are given in italics.

*In the 1950's, copper and gold discovered in placers near the village of Ity attracted the attention of BUMIFOM (Bureau Miniere de Ia France d'Outre Mer). BUMIFOM was initially a copper exploration company before switching to gold exploration. Pit and drift exploration led to the discovery of gold mineralization, that lead the BRGM (France's state-owned Bureau de Recherches Geologiques et Minieres) to estimate an in-house geological resource in 1970. However, various attempts to economically extract gold from the Flotouo deposit failed because of metallurgical problems, mainly due to the fineness of the gold and the rheology of the ore.*

*In 1983, the Societe Miniere d'Ity (SMI) was incorporated in Côte d'Ivoire as a vehicle for developing the Flotouo deposit thanks to advances in technology that allowed the gold to be economically extracted (Palanque 2002). SMI was initially 60% owned by the Côte d'Ivoire government (SODEMI), and 40% owned by the BRGM subsidiary COFRAMINES.*

*Further exploration conducted by BRGM and SMI in the 1980' yielded another in-house geological resource that confirmed and increased the mineralized zones found earlier. Mine construction began in 1990. The first gold from the Flotouo deposit was poured in January 1991. COFRAMINES was later replaced by Mine Or in 1993, and then by Normandy La Source, who raised its stake in SMI to 51% by acquiring 11% from SODEMI in 1997. Mining at Zia (the northeast extension of Flotouo) began in 2000.*

*COMINOR (La Compagnie Miniere Or) was created in 1999 by the BRGM in France to receive some of the assets (including SMI and other Côte d'Ivoire exploration properties) from the Normandy Mining Group. In 2000, COM/NOR was transferred to COGEMA (Compagnie generate des matieres nucleaires, now AREVA), and in March 2002, COMINOR acquired its 51% shareholding in SMI from the Normandy Mining Group.*

*In September 2002, Côte d'Ivoire was essentially divided in half following civil unrest. The mine lay at the boundary between opposing factions. The mine site was attacked and looted on November 30, 2002, and the mining operations and the entire site were abandoned from December 2002 to June 2003. During the mine closure, COMINOR took the opportunity to completely rebuild and update the geological database in order to develop a new mine plan.*

*Production resumed in February 2004 after dewatering the open pit and refurbishing the mining equipment and offices.*

*Mine production yielded more than 0.6 Million ounces, mostly from gold-rich laterites and from the Flotouo-Zia deposit.*

In 2006 COMINOR was subsumed into the La Mancha Group via a reverse takeover of La Mancha by Compagnie Francaise de Mines et Metaux ("CFMM"), a wholly-owned subsidiary of the Areva group. This gave La Mancha an effective 45.9% interest in the Ity Gold Mine. La Mancha took over operation of the mine. La Mancha has subsequently made operational improvements to increase gold production and carried out a series of drilling programs resulting in the extension of the known deposits and discovery of additional deposits in the permit areas.

On November 12, 2012 La Mancha delisted from the Toronto and Frankfort stock exchanges.

In 2014 a change of shareholders was authorised by the government of the Côte d'Ivoire. La Mancha was able to acquire additional shares for SODEMI giving it a majority of the shares in SMI. The new set of shareholders is: Groupe La Mancha (55%), SODEMI (30%), Government of Côte d'Ivoire (10%) and private investors, Keyman Investment (5%).

In September 2015, Endeavour Mining Corporation entered into a long-term strategic partnership with La Mancha and acquired La Mancha's 55% interest in SMI. The publically reported resources at this time are summarised in Table 6.1.1.

**Table 6.1.1 Ity Historical Mineral Resource July 2015 (including Reserves)**

| La Mancha (55% attributable) | Tonnes (Mt) | Au (g/t) | Au (koz) |
|---|---|---|---|
| Measured Resource | 27.3 | 1.4 | 1,191 |
| Indicated Resource | 31.8 | 1.7 | 1,729 |
| Total Measured and Indicated | 59.1 | 1.5 | 2,920 |
| Inferred Resource | 9.7 | 1.5 | 458 |

This is a historic resource estimate which has been replaced by the current mineral resource estimation reported in this document.

## 6.2 Exploration History

The early exploration history of the Ity area is summarised in Table 6.2.1.

## Table 6.2.1 SMI Exploration Summary

| Dates | Company | Exploration | Results |
|---|---|---|---|
| 1956-1958 | BUMIFOM | Soil geochemistry for Cu and Au over the KroZyale, Ity and Pepleu areas.<br>Pits over Ity, Flotouo and Zia. | Discovery of the laterite deposit at Flotouo. |
| 1959-1962 | BRGM | Exploration for Cu and Au using grids of pits in the low areas and soil geochemistry over a 100 m x 50 m grid at Ity-Flotouo, east of the Cavally River and Zones of Neo (Morgan) Doui. | No anomaly is known to have been found and no documentation appears to have been preserved. |
| 1968-1972 | BRGM | Exploration for Cu. | Unknown. |
| 1973-1975 | BRGM | Stream sediment sampling for Au. | Several anomalies found with values 100 - 500 ppb Au. |
| 1979 | GEOTEREX | Airborne geophysics: magnetics, EM, over the Touh pleu Ity belt. | 23 Electromagnetic (EM) anomalies identified. |
| 1980 | BRGM | Ground geophysics over the main anomalies and soil geochemistry Prospecting sulphide masses and Au. | |
| 1995-96 | SMI | Stream Sediment Sampling<br>2,059 samples (2 samples/km²) analyses for Au (5ppb), 692 analyses.<br>Soils geochemistry and radiometries for 9,630 samples on a grid of 200 x 50 m. | Anomaly Floleu > 100 ppb Au (max. 936 ppb Au), trending N40°E for about 1 km on PE 26. Numerous sites of artisanals. Anomaly Est-KroZyale (PE 26): distributed around a hill of 360 m with a maximum of 445 ppb.<br>Zone Le Plaque, subdivided into numerous sub zones :Sector de Le Plaque Nord, anomaly of 1,000 x 300 m.<br>Sector Le Plaque-Falaise at centre, a large anomaly of 2,500 x 500 m.<br>Sector Gompoupleu (ex-Le Plaque Sud), anomaly of 800 x 500 m. |
| 1997 | SMI | Diamond drilling: 34 drillholes, 3,751 m at 3 sites. | |
| 1998 | SMI | Study of gold mineralisation on the sectors of Le Plaque. Purpose: locate the mineralisation among geological domains and definition of prospecting procedures for new targets. | Discovery of contact metamorphism and skarns associated with the D2 deformation event. |
| 1999 | SMI | Soils magnetics (sectors Le Plaque- Floleu, Daapleu, Gbeitouo and Bakatouo (Morgan); 24,800 stations on a grid of 100 x 12.5 m).<br>Soil geochemistry (sector de Goleu and Bakatouo (Morgan)): 514 samples, grid 100 x 50 m, 234 analyses Au (results not found in any report). | Discovery of 3 magnetic anomalies weakly to moderately magnetic<br>In the sericitic schist or meta rhyolite strongly magnetic granitic bodies were identified. |
| 2002 | SMI | Le Plaque 1<br>25 drillholes; 1,774 m drilled<br>Shallow drillholes, stropped at the contact with hard rock corresponding to the granodiorite contact.<br>Le Plaque 2<br>21 drillholes; 1,207 m drilled<br>Oxidation zone better developed than at Le Plaque 1 (60 m thick. Drillholes stopped in fresh granodiorite. | Le Plaque 1<br>Poor results. Mineralisation intersected in the 1997 campaign was not confirmed.<br>Le Plaque 2<br>Poor result in spite of indicators from artisanal workings. |
| 2007-2009 | SMI | 199 drillholes, 12,920 m drilled; 7 sites: Ity Flat, Tongou, Walter, Ity village, Meandre, Zia NE and Bakatouo.<br>400 drillhole RC-DD and 523 RAB, total drilled 42 496,72 m (PR 61 and PE 26).<br>17266 m in RC, 12,079,72 m in DD and 13,151 m in RAB. | Mineralisation discovered at Zia extension NE, Walter and Ity flat. Ity flat in production.<br>Walter: Resources estimated at : 3,690 kg gold Zia extension NE , mineralisation in laterite and saprolites in multiple lenses. Orientation is Birimian with a dip 30-40° Tontouo and Bakatouo :weak mineralisation.<br>Meandre: results negative. Gbeitouto: Resources preliminary estimate 3,910 kg gold.<br>Daapleu, Morgan and Mlambopleu: intersection of mineralisation meriting follow-up. Poor results at Yacetouo, Tiapleu and Morgan extension. |

### 6.2.1 Aires de Lixiviation (Heap Leach Pads)

The Aires de Lixiviation (Heap Leach Pads) represents the residues of the leach pads which were in operation from 1990 to 2009.

### 6.2.2 Daapleu

Historical exploration at Daapleu is presented in Table 6.2.2.

**Table 6.2.2 SMI Exploration Projects – Exploration undertaken at Daapleu after 1995**

| Dates | Type of Work | Results |
|---|---|---|
| 1995 | Geophysics | Analysis of Data from Geoterrex Surveys. |
| 1996 | Stream sediment sampling | 2,059 Samples taken. |
| 1995-1996 | Soil Geochemistry | 11,365 samples taken on a 200 x 50 m grid reduced to 100 m x 50 m. |
| 1997 | Drilling Core | 34 drillholes totalling 3,751 m from three sites, Le Plaque, Daapleu and Morgan |
| 1998 | Evaluation | Study of gold mineralisation at Le Plaque |
| 1999 | Ground Magnetics | Over areas of Le Plaque Fioleu, Daapleu, Gbeitouo and Bakatouo (Morgan) 24,8000 stations over a grid 100 m x 12.5 m |

Three large pits are exploited by the artisanal miners. Two of these are oriented east-west and a third at N50° near the village. The artisanal miners extract the mineralised oxidised saprolites and the adjacent altered daaplites (rhyolite). The gold is washed at the Cavally River and recovered as a fine grained 'gold powder'.

**Figure 6.2.1 Artisanal Workings at Daapleu**



### 6.2.3 ZiaNE

Historical exploration at ZiaNE was included as part of the overall exploration and evaluation programme of SMI. In 2008 52 drillholes for 4,234.55 m of core, were drilled. There are an additional 32 drillholes drilled in 2015 campaign with a total meterage of 19,526 m.

### 6.2.4 Verse Ouest Rock Dump

Between 1996 and 2007, the cut-off grade of the mine was 2 g/t. The material with gold grade less than 2 g/t was dumped at Verse Ouest. The low grade clays with 1 to 2 g/t Au were dumped at Verse Ouest from 2008 to 2013. No evaluation work has been done for this dump prior to the current evaluation.

### 6.2.5 Mont Ity

Refer to general history of Ity Deposits (Section 6.1). New drilling was done in 2014 and comprised 31 drillholes of which six were for metallurgical samples, five were for hydrological studies. The remainder were drilled to confirm the geological model and are incorporated into the Mineral Resource estimate.

### 6.2.6 Ity Flat

Refer to general history of Ity Deposits (Section 6.1). Two phases of drilling were done at Ity Flat in 2015. The first phase comprised of 57 RC drillholes. The aim of the phase was to delineate the mineralisation. The first phase followed a 25 m x 25 m grid. The second phase was completed to confirm the RC assay results. Eight diamond drillholes were completed in the second phase.

## 6.3 Gold Production History

Historical gold production commenced in 1990 as a heap leach operation producing an average of 150,000 tonnes per day for the first three years and ramping up to a more than 300,000 tonnes per day production by 1994. As of January 1, 2016 over 9.76 million tonnes of oxide ore at an average grade of 3.74 g/t Au has been delivered to the heap leach pad for gold recovery. The Ity mine has produced over 1,174 koz of gold since 1990. Table 6.3.1 details the annual production figures for Ity mine.

Historic gold production from the lty Mine is given in Table 6.3.1.

**Table 6.3.1 SMI Gold Project – Gold Production History– Ity Mine**

| Year | Ore Mined (t) | Grade Mined (g/t) | Gold Produced (kg) | Gold Produced (oz) |
|---|---|---|---|---|
| 1990/91 | 160,937 | 10.3 | 723 | 23,259 |
| 1991/92 | 112,597 | 8.7 | 1,248 | 40,124 |
| 1992/93 | 170,181 | 9.6 | 1,192 | 38,324 |
| 1993/94 | 320,559 | 10.3 | 1,246 | 40,060 |
| 1994/95 | 2,093 | 9.1 | 1,326 | 42,632 |
| 1995/96 | 395,695 | 6.9 | 1,276 | 41,024 |
| 1996/97 | 324,595 | 6.2 | 1,582 | 50,862 |
| 1998 | 303,355 | 6.3 | 1,464 | 47,069 |
| 1999 | 310,449 | 6.8 | 1,432 | 46,040 |
| 2000 | 330,136 | 5.7 | 1,542 | 49,576 |
| 2001 | 404,415 | 5.2 | 1,858 | 59,736 |
| 2002 | 283,151 | 5.4 | 1,600 | 51,441 |
| 2003 | 0 | - | 0 | 0 |
| 2004 | 483,577 | 3.9 | 1,263 | 40,606 |
| 2005 | 443,581 | 3.8 | 1,335 | 42,920 |
| 2006 | 387,836 | 3.6 | 1,323 | 42,535 |
| 2007 | 444,047 | 3.6 | 1,243 | 39,963 |
| 2008 | 520,051 | 4.4 | 1,694 | 54,463 |
| 2009 | 490,198 | 3.3 | 1,608 | 51,698 |
| 2010 | 367,721 | 4.0 | 1,155 | 37,134 |
| 2011 | 385,523 | 3.0 | 1,143 | 36,748 |
| 2012 | 530,114 | 4.0 | 1,693 | 54,431 |
| 2013 | 706,192 | 4.2 | 2,552 | 82,062 |
| 2014 | 863,370 | 3.31 | 2,506 | 80,578 |
| 2015 | 1,020,543 | 2.54 | 2513 | 80,807 |
| **Total** | **9,760,916** | **3.74** | **36,517** | **1,174,092** |

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# 7.0 GEOLOGICAL SETTING AND MINERALISATION

## 7.1 Regional Geology

The description below of the regional geology has been taken from the description documented in the Technical Report for the Ity Gold Mine (Body, 2015). This was confirmed by Cube during the site visit in June 2016 and from discussions with site geological personnel.

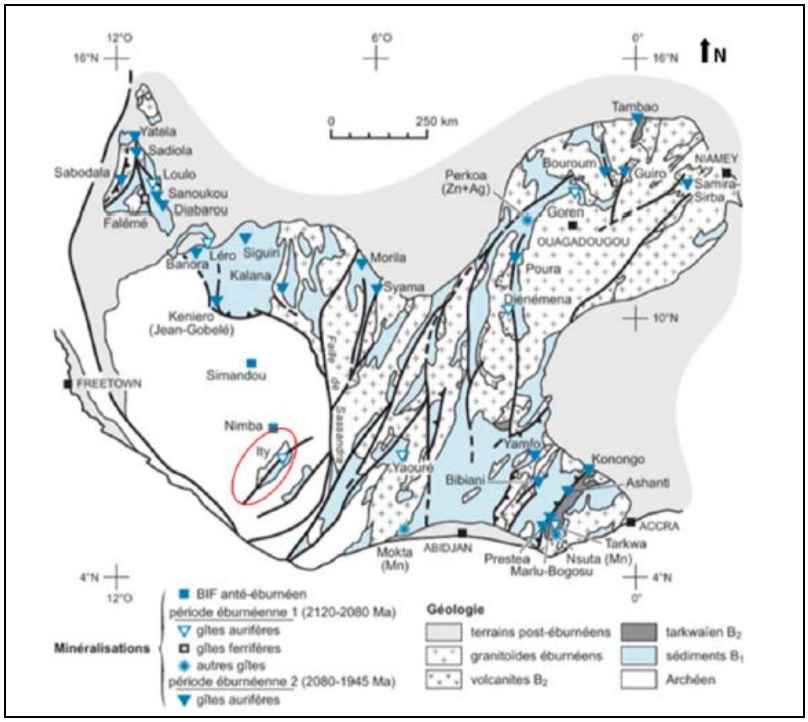
The Mont Ity deposits are located in the Lower Proterozoic Birimian Formation of the Toulépleu-Ity klippe. The Toulépleu-Ity klippe is a small remnant of the Birimian in the West African Craton which spans ten countries, between Côte d'Ivoire, Senegal, Niger and Ghana.

Milesi et.al, (1989) and the BRGM (1986, 1989) proposed a definition of the Birimian with a lithostratigraphic succession separated into two large groups:

- A Lower Birimian (B1) set essentially flyschoid basin fill. The whole basin is affected by three cycles of deformation:
  - D1 (2090-2100Ma) phase of major collision: duplication of the lower Proterozoic on the gneissic Archean basement, a break in all B1 sedimentation and intrusion of syn-kinematic granites and
  - D2 and D3 (2090-1970Ma) responsible for the intrusion of granites mantle between 2080 and 1945Ma (D2 large sinistral offsets, related overlaps and folding: D3 dextral offsets and associated folds).
- The Upper Birimian (B2), volcanic-dominated, where fluvio-deltaic formations are intercalated in volcano-sedimentary facies.

Figure 7.1.1 shows a simplified geology of the West African Craton, the location of the principal gold and base metals deposits and the Birimian window which contains the Ity deposit.

**Figure 7.1.1 Simplified Geology of the West African Craton**




According to Milesi et.al. (1989), the Eburnean metallogenic cycle, rich in gold and base metals lasted 150 Ma with:

- a first period at the time of the filling of the B1; stratiform deposits of Mn, Fe, Au, Zn-Ag are being put in place at about 2,150 Ma at the top of the stratigraphic pile. This period ends with the deposition of gold mineralisation in conglomerates

- a second, late-orogenic metallogenic period appears with the latter brittle deformation phases of D1 and D2. It is marked by mesothermal mineralisation, followed by quartz veins and paragenetic Pb-Zn-Ag-Bi deposits dated at approximately 201 Ma.

The deposits encountered in West Africa in the Birimian are diverse. Examples of deposit types are as follows:

- Gold mineralisation associated with major shear zones as in Obouasi (AngloGold Ashanti) along the Ashanti Fault Zone in Ghana.

- Gold mineralisation associated with conglomerates at the base of paleo-channels (placers) as in the deposit at Tarkwa in Ghana (Iamgold).

- Banded Iron Formation with gold anomalies at Tasiast, Mauritania (Kinross).

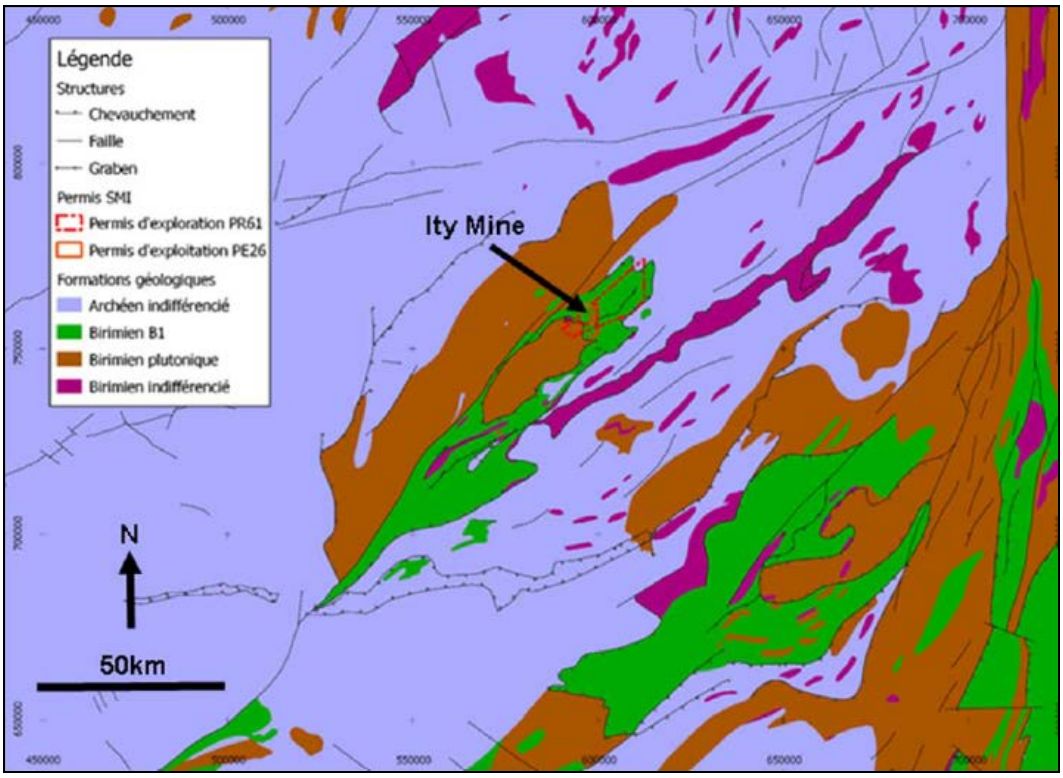
- Volcanic Massive Sulphides in the lower Birimian for example the Zinc deposit at Perkoa in Burkina Faso (Blackthorn Resources and Glencore International, project in development).

- Sedex deposits of the Nstuta manganese mine in Ghana operated by the Ghana Manganese Company Limited since 1916. Mineralisation is associated with turbidites within a volcano-clastic terrane (van Bart, 2001).

- Gold-Copper skarn at Ity. Ity is historically the only known Cu-Au skarn in the Birimian however iron skarns are known in the Kimieba-Kedougou Inlier of the Faleme District in Mali and Senegal (Shwartz & Melcher, 2004).

Figure 7.1.2 shows the location of the Toulepleu-Ity klippe and the location of the SMI permits. Two different packages of Birimian rocks are found in the klippe. On the west, plutonic and metamorphic rocks and to the east are sediments and volcano-sedimentary packages (B1). Birimian aged Formations overlie the Archean basement and were emplaced by large regional scale thrust-faulting. The Birimian B1 formations correspond to meta-sedimentary series (pelites, gresopelites and carbonates) which have been subjected to greenschist-lower amphibolite facies regional metamorphism. The Toulepleu-Ity klippe is oriented NE-SW. The emplacement of diorites and granodiorites in the meta-carbonate sediments resulted in contact metamorphism causing skarn development with masses and stockworks of sulphides and magnetite containing gold mineralisation. These phenomena are accompanied by extensive hydrothermal alteration (Palanque, 2004). Sedimentary and volcano-sedimentary formations of the Toulepleu-Ity klippe are part of the B1 terrane and are therefore affected by the three tectonic phases described previously.

**Figure 7.1.2 Simplified Geology Map of the Toulepleu-Ity klippe**



## 7.2 Local Geology

The description below of the regional geology has been taken from the description documented in the Technical Report for the Ity Gold Mine (Body, 2015). This was confirmed by Cube during the site visit in June 2016 and from discussions with site geological personnel.

The Ity area is characterised by a series of granodioritic intrusions into a sedimentary sequence of volcano-sediments and carbonates. The volcanic are tuffaceous deposits whose chemistry ranges from basic to acidic. All formations have been subjected to regional metamorphism.

The deposits of Ity, Zia and Flotouo are skarns developed at the contacts of the granodiorite with the carbonates. The Teckraie is the rock dump of the Flotouo open pit and sits on top of a weathered granodiorite. The Daapleu deposit is characterised by the presence of a 'rhyolitic' intrusive surrounded by a package of volcanosediments. The 'rhyolite' is locally called 'daaplite' and is leucocratic (grey to white), microgranular, schistose and rich in micas.

***Laterites (Facies 301)***

Laterites are soils without texture, formed by supergene alteration of iron rich rocks and are generally red in colour. Four lateritic categories have been identified at Ity with all profiles containing millimetre scale fragments of magnetite and quartz. The succession described below has been observed to be generally less than 10 m thick and includes the following:

- Lateritic clays (LF) are found at the top of the sedimentary pile and include limonites and the topsoil. Residues of organic material commonly found in this layer give the laterite a dark red-brick to brown colour.
- Lateritic gravels (LG) have a brick-red to brown clay matrix (about 30% of the volume) and contain abundant concretions of iron oxides and hydroxides (hematite, goethite, magnetite) as rounded pisoliths from millimetres to centimetres in size. Dendrites of manganese oxide (pyrolusite) several centimetres long coat fractures and pore spaces.
- Mixed Laterites (LM) are similar to the gravels in appearance but have a higher proportion of matrix to concretions.
- Mottled laterites (LT) vary in colour from red-orange to yellowish orange with patches of white kaolinite. Rare pisoliths and traces of iron oxides are present in this layer.

***Saprolites***

The saprolites in the Ity area have a plastic texture and a grainsize of less than 2 µm. The saprolites are designated silts, sandy silt and breccia and all retain texture of the protolith to a greater or lesser extent. The saprolites are the result of supergene alteration, hydrothermal alteration or a combination of the two. The facies vary in colour as a function of the protolith and degree of oxidation and include the following:

- Oxidised Saprolites (Facies 201) or Argiles Oxidees (AO), as shown in Figure 7.2.1 are essentially oxides and hydroxides of iron ($Fe^{2+}$ and $Fe^{3+}$) which are present in a wide range of colours, yellow, orange, brick-red violet, etc. The oxidised saprolites are subdivided into three types, Redox front (AOR), saprolite rich in iron oxides and manganese (AOX) and sandy saprolites (AOS).

- Reduced Saprolites (Facies 211) or Argiles Reduites (AR) as shown in Figure 7.2.2 are coloured green to white or light grey. They are generally found deeper than the oxidised saprolites and correspond to the water table limit. The minerals in this zone are principally kaolinite and hydrated minerals, tremolite and actinolite. There is a subtype which has a higher percentage of sandy material and is designated ARS in the geological logs.

In the saprolites, breccias are coded BAO and BAR depending on the surrounding lithology.

**Figure 7.2.1 Oxidised Clay (Facies 201) - ZA15-001 at 10.8 m depth**



**Figure 7.2.2 Reduced Clay (Facies 211) - S1145 at 15.10 m depth**



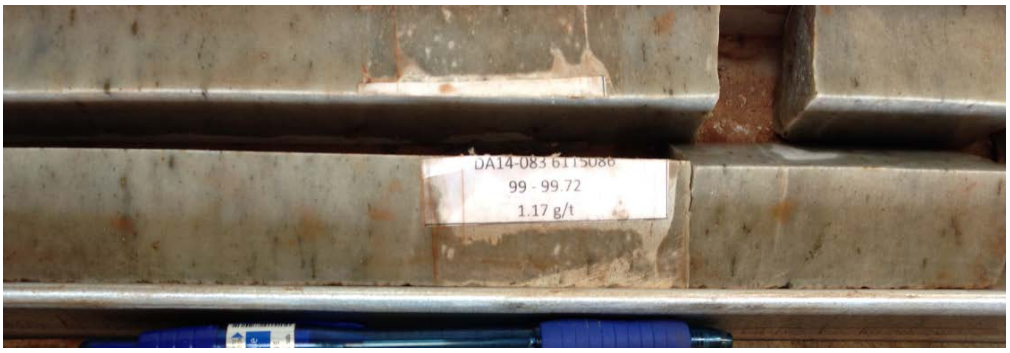
***Rock Dumps (Facies 701)***

Rock Dumps present as oxidised saprolite breccia with a brownish orange matrix with an assemblage of centimetre sized clasts of different colours (orange, green, grey, whitish, yellow, mauve and some with granular textures) giving a multi-coloured deposit. Various clasts of competent material have been identified in the dumps. These include centimetre sized fragments of quartz and gravels, lateritic zones contain pisoliths and iron oxides. In the northern area at Teckraie, the base of the rock dump is lined with fine grey material, compacted by the weight of the surrounding material. These fine particles impregnate the laterites which separate the rock dump from the altered granodiorite below. This facies is also in the leach pads.

***Daapleu Rhyolite (Facies 601)***

The granitic intrusive (Daaplite, IFMD) at Daapleu is locally called a rhyolite as it has a fine texture. The rhyolite is leucocratic (grey to white), microgranular, schistose and rich in sericite and contains fine-grained pyrite or magnetite disseminated and in fractures. Numerous pink tinged silicified zones have been identified (possibly hematite). Several zones of intercalated volcano- sediments have been intersected near the edges. The rhyolites are crossed by veins of quartz and occasional minor tourmaline. Figure 7.2.3 shows some examples of rock types within the rhyolite.

**Figure 7.2.3 Rhyolite (Daaplite or Facies 601) - DA14-083 at 99.0 m depth**



***MetaVolcanic Sediments (Facies 101)***

The volcano-sediments present are dark greyish-brown to grey-green and finely laminated. Minerals are primarily amphibole, chlorites, biotites, traces of magnetite, calcite as veinlets and fracture coatings and quartz veins with tourmaline. Disseminated and massive pyrite is found within the lamination in proximity to the daaplite contact and diminishes with depth (away from the contact). In some places the metavolcanic sediments contain abundant amounts of calcite (e.g. Drillhole DA13-016).

**Figure 7.2.4 MetaVolcanis Sediments (Facies 101) - DA14-083 at 134.5 m depth**




***Carbonate (Facies 401)***

The carbonates (CM) are typically greenish in colour and microcrystalline with chlorite-epidote alteration, disseminated pyrites and magnetite (e.g. OA13-023). The carbonates represent protoskarn facies. Carbonate Facies 401 is shown in Figure 7.2.5.

**Figure 7.2.5 Carbonate (Facies 401) - S663 at 110.50 m depth**



***Skarn (Facies 901)***

The skarns as shown in Figure 7.2.6, represent metamorphosed carbonate host and are associated with presence of sulphur in the form of varying occurrences of pyrite, pyrrhotite, chalcopyrite and magnetite.

**Figure 7.2.6 Skarn (Facies 901) - S1145 at 96.45 m depth**



***Diorite (Facies 811)***

Coarse-grained grey-green mafic intrusive (IMG) composed of amphibole, magnetite and chlorite with fine-grained disseminated pyrite and calcite veins.

***Granodiorite (Facies 501)***

Coarse-grained grey felsic intrusive (IFG) rich in sericite, magnetite and large straw-like crystals of muscovite (e.g. drillhole DA13-015) and shown in Figure 7.2.7.

**Figure 7.2.7 Granodiorite (Facies 501) - ZA15-001 at 157.0 m depth**



## 7.3 Deposit Geology

The SMI property deposits are part of a class of deposits called orogenic gold deposits. These can be described as gold-bearing quartz veins, stringers and wall rock replacement accompanied by only minor sulphides that are localised by brittle to ductile structures within variable rock types. The deposits within the Ity Gold Project are generally described as skarn or shear zone styles of mineralisation.

### 7.3.1 Skarn Deposits

Skarn deposits within the Ity Gold Project include Mont Ity, Ity Flat, ZiaNE and Walter also include the material from which the Aires and Teckraie dumps have been derived. The primary difference between Mont Ity and ZiaNE appears to be the lack of karst development at ZiaNE due to the position of the granodiorite in the footwall.

The interpreted section (Figure 7.3.1) is situated in the centre of the ZiaNE and Mont lty deposit. In this section four types of mineralisation are seen:

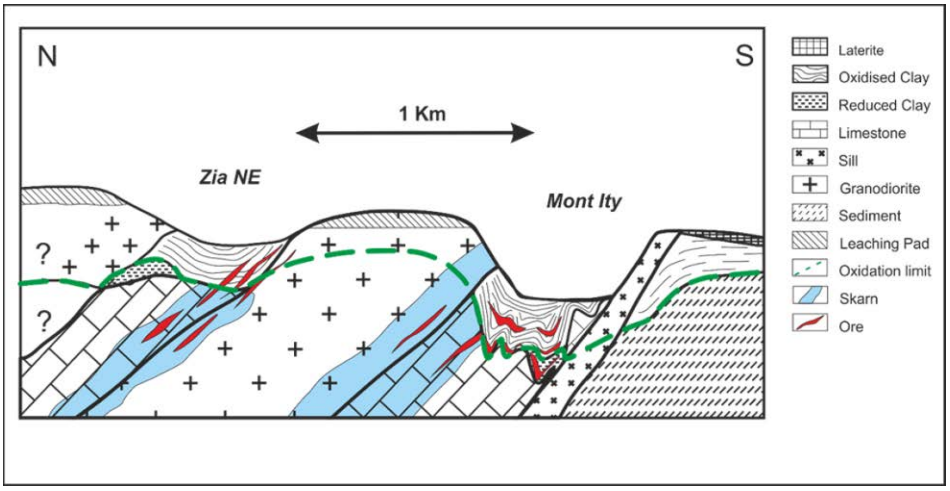
- Skarns in fresh rock.
- Reduced Saprolites.
- Oxidised saprolites.
- Laterites.

The skarn corresponds to a sulphur rich skarn (pyrite-pyrrhotite-chalcopyrite, 3-10%) with accessory magnetite. These mineralised zones form discontinuous, sub parallel lenses at the contact with the granodiorite. In the north part of the deposit economic grades and thicknesses were intercepted in the drillholes from the 2013 campaign. The sulphide mineralisation was intersected in the carbonates and the granodiorite at ZiaNE and is continuous with the mineralisation in the oxidised portions of the skarn.

The mineralised portions of the reduced saprolites correspond to the decarbonation of marble and exoskarn. These are rocks rich in chlorite, tremolite and/or actinolite. The reduced saprolites are the transition zone between the oxidised material and the fresh rock.

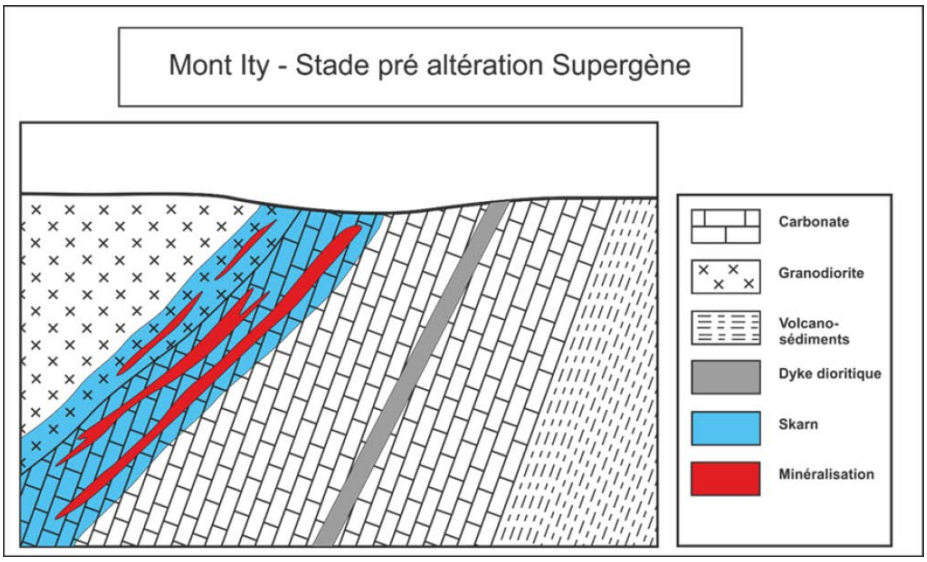
The oxidised saprolites form the major part of the mineralisation at Mont lty. The mineralisation was originally a skarn which has undergone severe supergene alteration. The alteration was enhanced by the dissolution of the sulphides. The alteration was responsible for the generation of karst into which the saprolite material collapsed which provides an explanation for the geometry of the deposit.

**Figure 7.3.1 Schematic Geological Section of the Mont lty Deposit**



The Mont Ity deposit is an unusual deposit not only because Proterozoic gold skarn deposits are rare but because the supergene alteration has completely changed the deposit geometry. Figure 7.3.2 shows the Mont Ity deposit as interpreted before the supergene alteration. The primary mineralisation is found in the endo- and exoskarn. The mineralised zones is found as discontinuous, sub parallel lenses in contact with the granodiorite and carbonates The primary Au (Cu-Mo-Ag-Zn) is associated with the intrusion of the granodiorite. The intrusion of the granodiorite is accompanied by the emplacement of a dyke of similar composition in the carbonates.

**Figure 7.3.2 Interpreted Pre-alteration Stage of the Ity Deposit**




The first stage of supergene alteration is characterised by the infiltration of meteoric water. The meteoric water reacted with the sulphide rich mineralisation, especially in the massive sulphide zones. Oxidation of the pyrite in an aqueous solution follows Equation 1.

$$FeS + H^2O + 2O^2 \longrightarrow H^2SO_4 + FeO \quad (1)$$

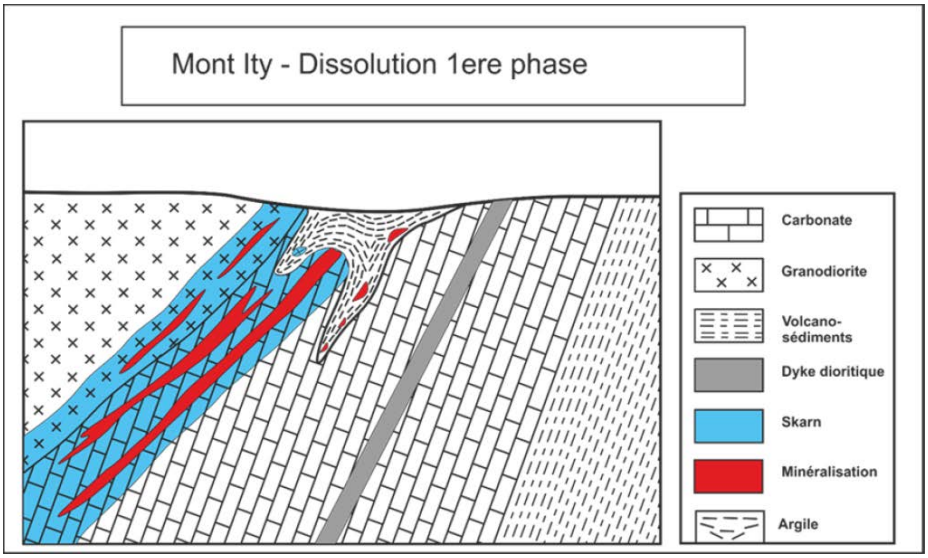
This oxidation reaction generates sulphuric acid. The acid then reacts with the carbonate following Equation 2.

$$CaCO^3 + H^2SO_4 \longrightarrow CaSO_4 + H^20 + CO^2 \quad (2)$$

The Fe from Equation 1 is found as goethite, ferrihydrite and other Fe-Mn oxides. The reaction of the in Equation 2 results in the formation of $CO^2$ and voids.

Oxidation of the sulphides results in the dissolution of the adjacent carbonates and creates cavities (Figure 7.3.3). The instability in the system causes the collapse of the mineralised zone into these karsts. The carbonates and the calcosilicates zones in the skarns were not pure calcite. This results in the formation of different decarbonised clays. The well mineralised zones in the karst are described as zones rich in iron and magnesium oxides, the products of the oxidation of the primary mineralisation. This oxidation was accompanied by a loss of volume in the mineralised zones resulting in an enrichment of the gold.

**Figure 7.3.3 Schematic Diagram of the First Phase of Alteration**



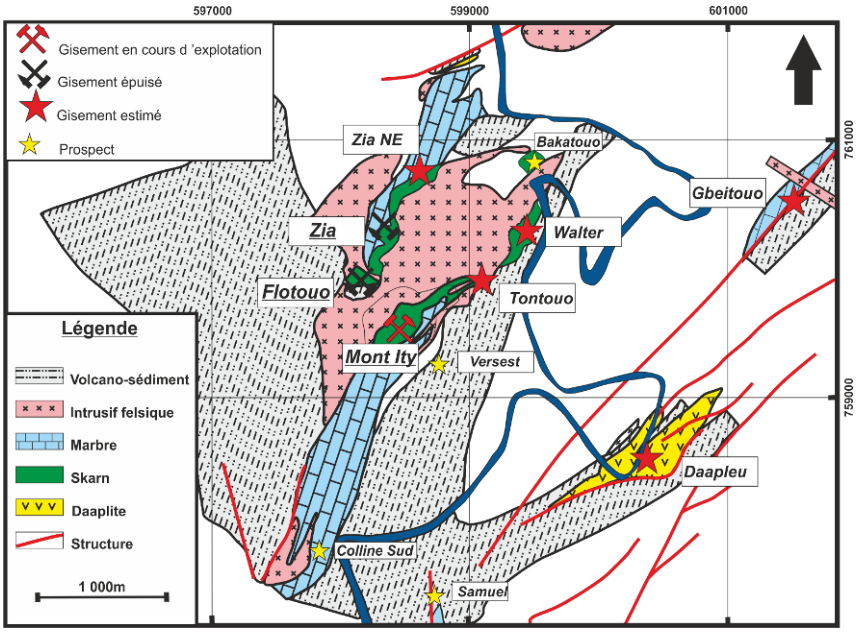

The alteration process became self-propagating as the oxidation front went deeper. The dissolution became deeper and collapse followed. Erosion and laterisation affected the upper parts of the deposit. The laterite zones rich in gold in the southwest of the pit are probably oxidised clays which collapsed as erosion removed the surface and were themselves laterised. Such a phenomenon would explain the lack of root zone in the mineralisation. This also implies that there is a lack of mobility of the gold in the laterites at Mont Ity.

### 7.3.2 Shear Zone Deposits

The Daapleu and Gbeitouo deposits resemble typical shear zone deposits of the West African granite-greenstone terrane. Both deposits are associated with a major regional shear zone but developed on secondary structures (Figure 7.3.4). The lithologies can be any form of sediment, carbonate or igneous rock with the main feature being a shear zone between the two contrasting lithologies. Mineralisation may also be spatially related to the emplacement of intrusives. The gold mineralisation is mesothermal in origin and occurs as free gold in quartz vein stockworks and zones of silicification, associated with arsenopyrite and to a lesser extent pyrite and antimony. Depending on the geological terrane other metals may be associated with the gold and arsenic.

The gold mineralisation is found in linear zones in or near the contacts between two different rock types. This contact shows evidence of shearing. Alteration is weak to severe depending on the development of the system.

**Figure 7.3.4 Schematic Geological Plan Including the Daapleu and Gbeitouo Deposits**



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**8.0 DEPOSIT TYPES** **8.1**

# 8.0 DEPOSIT TYPES

The West African Lower Proterozoic greenstone belts are often referred to as Birimian Greenstone Belts and this includes a collection of Paleoproterozoic metasedimentary and metavolcanic units and associated intrusive complexes that are the dominant hosts of gold deposits in West Africa. The Birimian Greenstone Belts have produced a number of world-class gold deposits situated within countries such as Côte d'Ivoire, Burkina Faso, Ghana, Guinea, Mali, Niger and Senegal.

Gold deposits in West Africa can broadly be classified into the following types:

- Structurally-controlled, epigenetic lode or stockwork mineralisation related to major shear zones with native gold (Poura, Burkina Faso; Kalana, Mali).
- Structurally-controlled, epigenetic lode or stockwork mineralisation related to major shear zones and characterised by the inclusion of gold in the crystal structure of the sulphides, often locked in arsenopyrite (Ashanti type: Obuasi, Ghana).
- Stratiform deposits hosted in tourmalinised turbidites (Gara Deposit Loulo, Mali).
- Disseminated sulphides hosted in volcanic or plutonic rocks (Syama, Mali; Yaouré, Ivory Coast; granitoid-hosted, Ayanfuri, Ghana).
- Paleo-placer deposits: auriferous quartz-pebble conglomerates (Tarkwa, Ghana); modern placers (eluvial, alluvial).

More specific to the Birimian, two major styles of gold mineralisation occur, which include:

- structurally controlled quartz vein style deposits, and
- chemical sediment hosted deposits where gold is associated in selvages to quartz veins.

The Ity deposits can be broadly classified as orogenic gold deposits, like most of the Birimian gold deposits with the possible exception of the Tarkwa paleoplacers.

The two different subtypes of hypogene mineralisation in the Ity area are described as:

- Au-Cu-Mo-Ag-W skarns, and
- Au-As-(Sb) shear zones.

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# 9.0 EXPLORATION

## 9.1 Exploration Overview

Since 2012, in order to extent the heap leach operation life and to increase the CIL resources, the group engaged an intensive exploration program. The main objectives of this program were as follows:

- Find more oxide ore and increase heap leach mining life by:
  - delineate the lateritic ore in the nearby- mine areas,
  - infill drilling on current oxide deposits (Mont Ity, ZiaNe and Flat), and
  - find, test and develop other oxide targets within 5 km radius from Ity plant.
- Near mine and FS related projects included:
  - develop at least to Indicated resources the FS deposits (ZiaNe, Daapleu, Mont Ity, Walter, Gbeitouo, Teckraie, West and East Rock dumps, Aires leach pad),
  - prepare representative composites for metallurgical tests for each deposit of FS,
  - define a robust geological model of the Ity area and apply it to each deposit,
  - assist the geotechnical and hydrogeological teams during their field campaigns, and
  - find, test and eventually develop other targets for CIL resources within 5 km radius from Ity plant.
- Regional Exploration projects included:
  - define other targets to create a new pipeline for exploration projects,
  - ground geophysical campaign (IP),
  - soil geochemical sampling,
  - regolith mapping,
  - pit, trenches and auger drilling, and
  - aerial survey.

Ground geophysical survey and soil geochemistry undertaken during 2013/14 are summarised in Figure 9.1.1 and Figure 9.1.2, respectively.

**Figure 9.1.1 Geophysics Performed on the PR61 - apparent resistivity map (IP survey). Gbeitouo deposit highlighted**

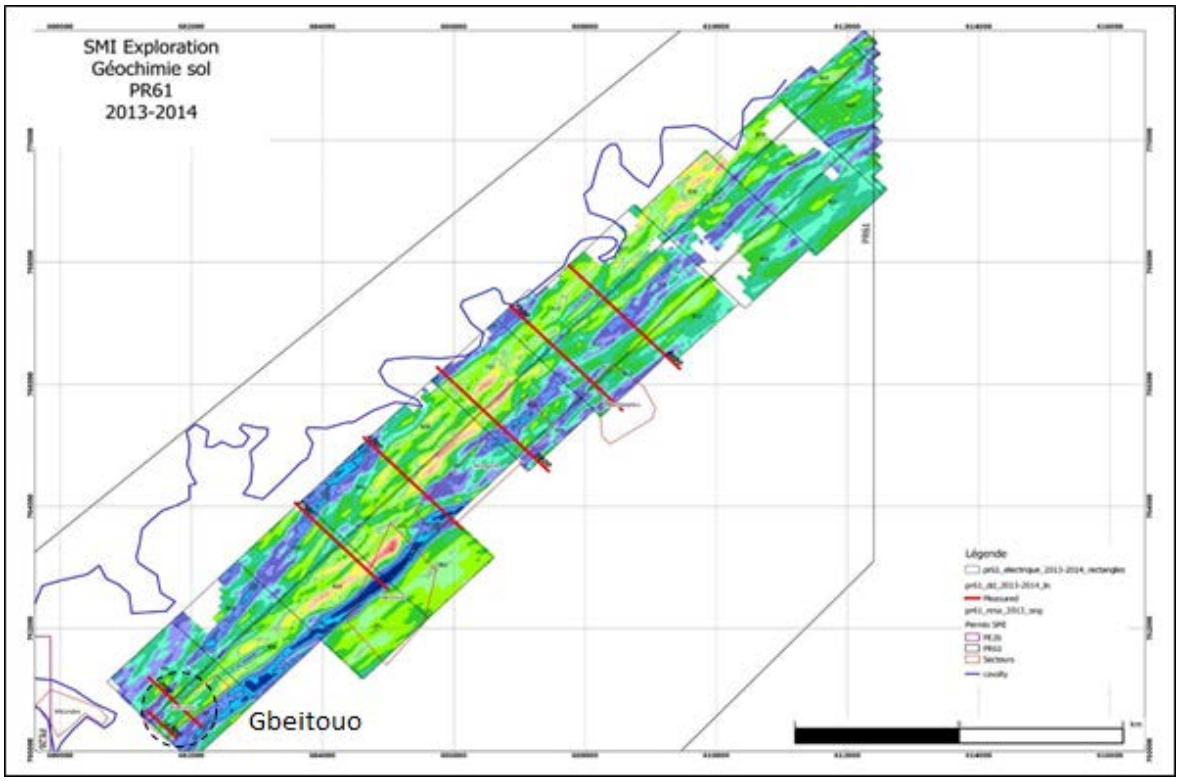
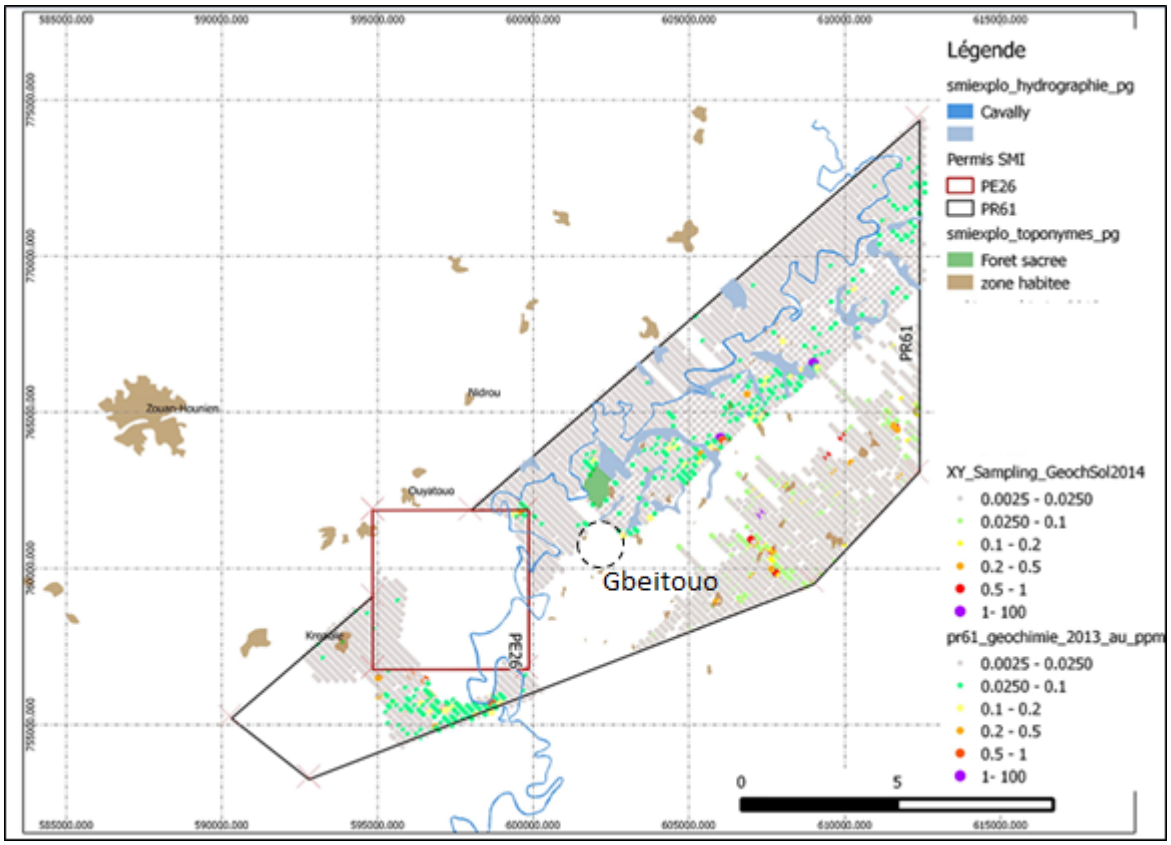


**Figure 9.1.2 Soil Geochemistry Performed on PR609 (formerly PR61). Gbeitouo deposit highlighted**




Human resources on site have been increased to supervise the recent drilling campaigns. Senior personnel with extensive experience in exploration have been employed to put in place protocols that are in agreement with international best practice standards. All protocols are available to the site geologists and are the object of continual revision and improvement.

The main exploration camp base, including a core shed and a mechanical sample preparation laboratory were built in 2013 - 2014. Additionally, a satellite exploration camp has been built on the east side of the Cavally river to facilitate the supervision of the exploration programs in Daapleu and Gbeitouo.

Drill core samples and RC samples were prepared at Bureau Veritas in Abidjan or on site at the SMI laboratory (Societe des Mines Ity). The prepared samples were analysed using a standard 50 gram gold fire assay with an Atomic Absorption finish at Bureau Veritas Laboratories (Abidjan). ICP-ES analysis were completed by ACME Laboratories Ltd. in Vancouver. Core sampling and assay data were monitored through a quality assurance/quality control program designed to follow NI 43-101 and industry best practice.

All collars have been surveyed with Differential GPS (WGS84, UTM 29N) by the surveyor team of SMI or by independent contractors (Societé Nationale de Topographie and Envitech Sarl).

## 9.2 Drilling Program

From January 2012 to October 2016, 189,279 m have been drilled in the PE26 and PR609.

FORACO is the main contractor to have undertaken the drilling. They have been working for SMI since 1992 and have gained a vast amount of experience of the geological facies types encountered in this area. However, to complete the drilling program within the allocated time, UTZM, a Russian company, was contract in 2014 for 25,000 metres.

Table 9.2.1 summarises the meterage realised by year for each target, the purpose of the drilling and the main results.

**Table 9.2.1 Drilling Program**

| | | | 2012 | 2013 | 2014 | 2015 | Sept 2016 | Objective |
|---|---|---|---|---|---|---|---|---|
| **Mont Ity et Flat** | DD | No. | 60 | 155 | 24 | 8 | | Development at 25 x 25 metres grid for Indicated resources in clay and 25 x 50 m for Inferred resources in fresh skarn at depth. The main drilling program targeted the mineralisation in clay. The proportion oxide clay, reduced clay is approximatively 70% - 30%. The drilling program allowed the extension of mineralisation at depth in the western sector. The eastern sector also intersected lower grade mineralisation hosted in volcano sediment. Additional drilling will be necessary to develop this resource to inferred level. In the south, the dissolution and enrichment process in karst disappears and no possible extension of mineralisation is possible. Drill holes for metallurgical tests have also been completed. |
| | | m | 11,458 | 21,891 | 3,045 | 492 | | |
| | RC | No. | 151 | | 9 | 57 | | |
| | | m | 3,730 | | 702 | 3,989 | | |
| **Cantine** | RC | No. | | 6 | | | | Sterilisation of areas designated to be the future mess. |
| | | m | | 901 | | | | |
| **Zia - Zia NE** | DD | No. | 75 | 38 | 69 | 32 | | Development at 25 x 25 metres grid for Indicated resources and 25 x 50 m for Inferred resources in fresh skarn at depth. |
| | | m | 10,427 | 2,226 | 7,276 | 2,779 | | |
| | RC | No. | | | | 22 | | |
| | | m | | | | 828 | | |
| **Verse Est** | RC | No. | | 16 | 14 | | | Condemnation of East waste dump. Although some visible gold is present in the area and artisanal miners are frequent, only sub economic mineralisation has been encountered in sheared volcano sediments. |
| | | m | | 1,795 | 1,21 | | | |
| | DD | No. | | 20 | 17 | | | |
| | | m | | 2,646 | 1,643 | | | |
| **Verse Ouest** | DD | No. | | 31 | 18 | | 1 | Condemnation of East waste dump. |
| | | m | | 740 | 718 | | 180 | |
| | RC | No. | | | | | 34 | |
| | | m | | | | | 1,183 | |
| **Walter** | DD | No. | | | 29 | | | Development at 25 x 25 metres grid for Indicated resources and 25 x 50 m for Inferred resources in fresh skarn at depth. |
| | | m | | | 3,386 | | | |
| | RC | No. | | | 19 | | | |
| | | m | | | 1,891 | | | |
| **Tontouo** | DD | No. | | 96 | | | 3 | Development at 25 x 25 metres grid for Indicated resources and 25 x 50 m for Inferred resources in fresh VSM at depth. |
| | | m | | 8,994 | | | 268 | |
| | AC | No. | | | 29 | 31 | | |
| | | m | | | 1,270 | 1,946 | | |
| **Aires Leach Pad** | DD | No. | | 12 | | | | Development at 25 x 25 metres grid for Indicated resources in old leach pads. |
| | | m | | 768 | | | | |
| | AC | No. | | 90 | | 70 | | |
| | | m | | 4,192 | | 2,206 | | |
| **CIL** | DD | No. | | 19 | 10 | | | Condemnation of CIL area. No mineralisation has been intercepted. The main lithology encountered is volcano sediment. |
| | | m | | 3,841 | 1,170 | | | |
| **Mt GIBEUAM** | DD | No. | | 41 | | | | Exploration holes in Colline Sud area for laterite. |
| | | m | | 1,134 | | | | |

**Table 9.2.1 Drilling Program (continued)**

| | | | 2012 | 2013 | 2014 | 2015 | Sept 2016 | Objective |
|---|---|---|---|---|---|---|---|---|
| **Teckraie** | DD | No. | | 31 | | | | Development at 25 x 25 metres grid for Indicated resources in old waste rock dump. |
| | | m | | 1,079 | | | | |
| | AC | No. | | 35 | 52 | | | |
| | | m | | 1,227 | 1,992 | | | |
| **FLOTOUO** | DD | No. | | | 8 | 1 | 4 | Exploration deep holes under Flotouo old pit and testing of Flotouo extensions. |
| | | m | | | 1,246 | 351 | 532 | |
| | AC | No. | | | | 87 | | |
| | | m | | | | 1,276 | | |
| **COLLINE SUD** | DD | No. | | | 9 | 68 | 14 | Exploration then Development at 25 x 25 metres grid for Indicated resources; 50 x 25 m for Inferred. |
| | | m | | | 1,229 | 7,179 | 1,477 | |
| | RC | No. | | | | | 25 | |
| | | m | | | | | 2,070 | |
| **BAKATOUO** | DD | No. | | | 13 | 22 | 136 | Exploration then Development at 25 x 25 and 33 x 33 metres grid for Indicated resources. |
| | | m | | | 1,834 | 3,308 | 15,966 | |
| **CAMP FORACO** | AC | No. | | 3 | | | | Exploration for mineralised laterite. No significant results were received. |
| | | m | | 300 | | | | |
| **Daapleu** | DD | No. | | 51 | 70 | 55 | | Exploration then Development at 25 x 25 metres grid for Indicated resources; 50 x 25 m for Inferred. Mineralisation is hosted in the sheared volcano sediments at the contact with rhyolite. |
| | | m | | 7,970 | 9,958 | 7,271 | | |
| | RC | No. | | 20 | | | | |
| | | m | | 1,938 | | | | |
| **Samuel** | DD | No. | | | 6 | 1 | | Test anomalous areas of gold in soil samples to generate new exploration targets. |
| | | m | | | 1,120 | 81 | | |
| | RC | No. | | 6 | | | | |
| | | m | | 725 | | | | |
| **Exploration régionale** | DD | No. | | | | 5 | | More than 10,000 samples of soil geochemistry collected over four years. Some of the anomalous gold-in soil areas were tested to generate new exploration targets. A program of 26,855 m auger drilling (2,150 holes) completed to generate new exploration targets. |
| | | m | | | | 1,238 | | |
| **Gbeitouo** | DD | No. | | 35 | 12 | | | Exploration then Development at 25 x 25 m grid for Indicated resources. Mineralisation is associated to schistose volcano sediments enriched in sulphides (pyrites and arsenopyrite) at the contact with marbles. |
| | | m | | 4,724 | 2,050 | 0 | | |
| **Total** | | | **25,615** | **67,093** | **41,952** | **32,943** | **21,676** | |

## 9.3 Sterilisation Surveys

A total of 11,050 metres (78 holes) of sterilisation drilling have been completed since 2013 on the East, West and ZiaNE Waste dumps and also Meantouo and Cantine. Additional sterilisation drilling are scheduled to be completed during 2017 and 2018 in areas which will be affected by construction of facilities associated with the CIL Plant (Gbeitouo and Daapleu waste dumps, plant CIL etc).

## 9.4 Exploration Targets

The prospectivity of the Ity area has been confirmed by recent exploration works. The auger drilling program coupled with gold in soil results indicate that the overall potential of the area extends well beyond the present footprint. Auger results indicate that very strong anomalous gold values (>1,000 ppb) occur within the saprolite at numerous locations within the area covered by the program, allowing for the definition of large and highly anomalous clusters and the generation of several new near-mine high priority exploration targets.

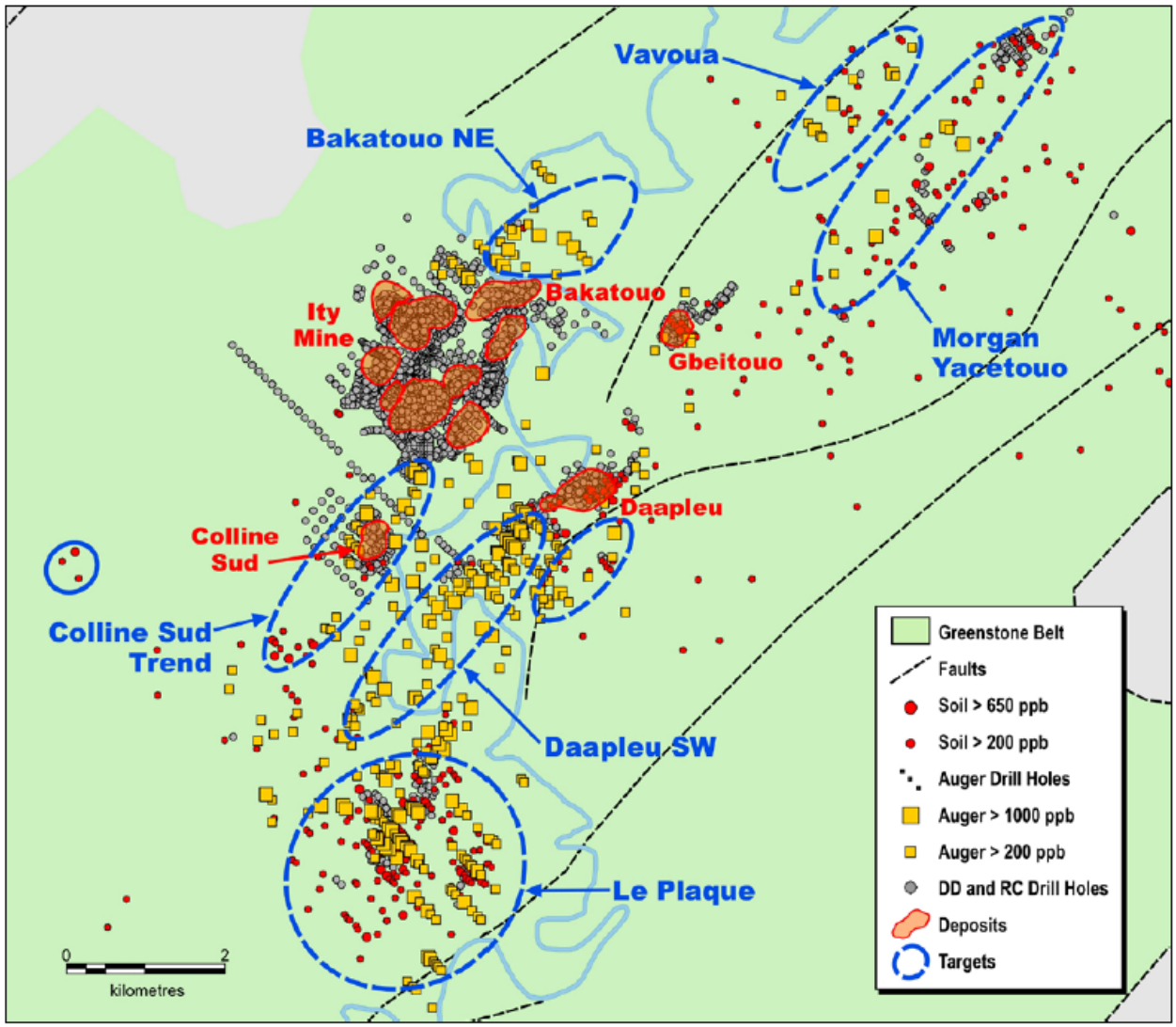
The most significant near-mine targets identified are:

- **Bakatouo Northeast area:** Auger results clearly suggest a 1 km northeast trending anomaly located immediately to the northeast of the new Bakatouo discovery, with best values >7,000 ppb in the saprolite.
- **Vavoua area**: New 1.5 km northeast trending anomaly with best values >1,400 ppb in the saprolite.
- **Morgan / Yacetouo area**: Large and significant 3 km long and northeast trending anomaly with best values >2,450 ppb in the saprolite.
- **Le Plaque area (PR 558)**: Significant 2 km long N20° trending anomaly with best values >5,000 ppb in the saprolite.
- **Daapleu Southwest area**: Large gold anomaly in saprolite that covers over 4 km², with best values >2,200 ppb.

The Auger drilling was completed on grid N135° and was designed to be orthogonal to the regional geological structural trends. A total of 9,640 samples were collected at 200 x 50 metres and 400 x 50 metres grid spacing.

The defined Auger clusters may correspond and validate previously identified gold in soil anomalies (such as in Colline Sud or Daapleu) or allow for the definition of new prospective areas (such as Bakatouo NE or Daapleu SW), where no gold in soil anomalies were previously detected due to a probable alluvial cover associated with Cavally river overflows.

**Figure 9.4.1 Mineralised Trends and Anomalous Soil and Auger Results, Ity Area**

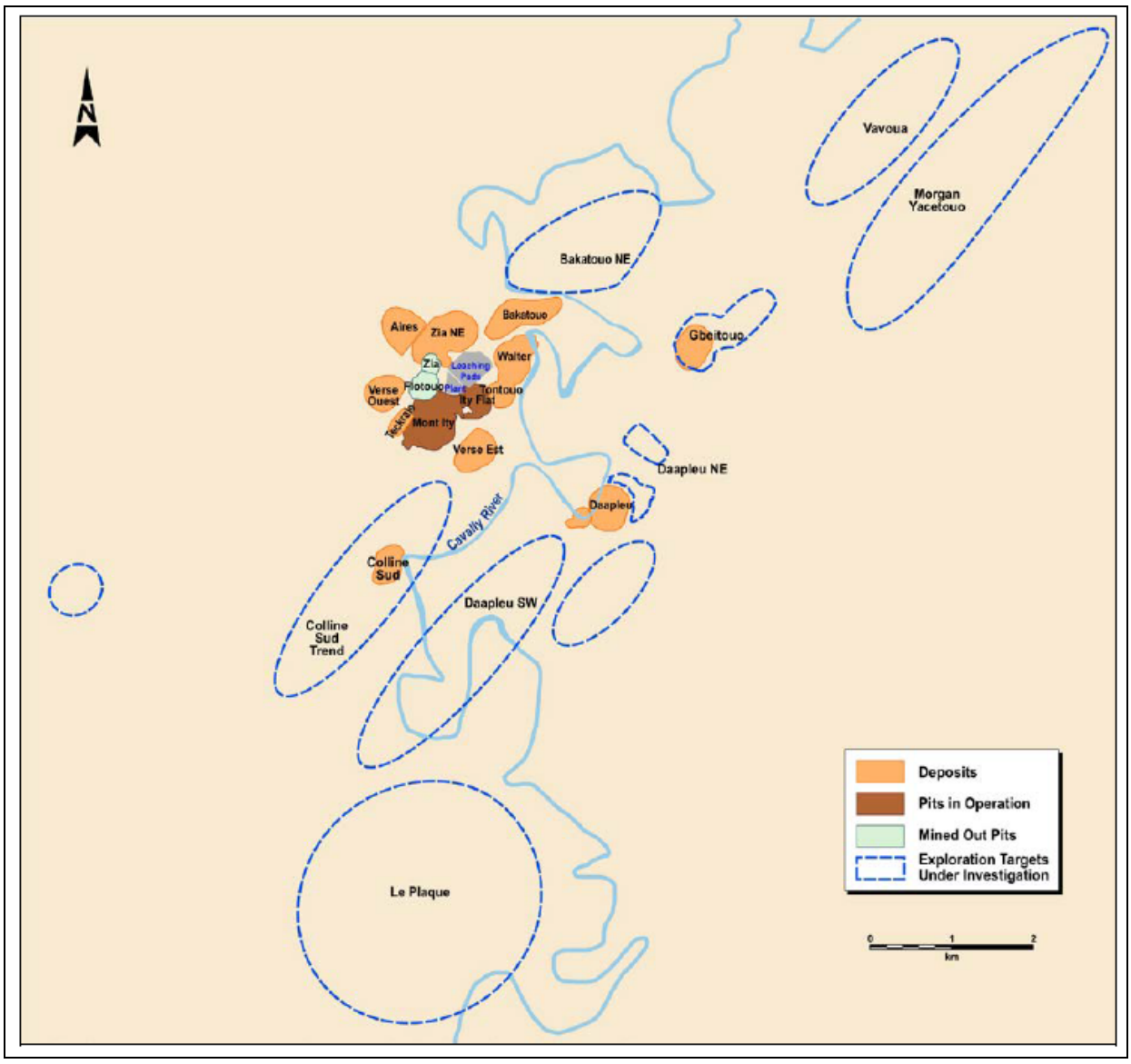



*Note: The gold-in-soil and Auger high cut-off values used compared to standard reporting values (100 ppb) were chosen to improve the readability of the above map.*

The high grade Bakatouo discovery is located in proximity to the current Ity mining complex (Figure 9.4.1 and Figure 9.4.2). The mineralisation is close to surface and occurs over 800 m along strike, while extensions remain open at depth and along strike. An additional drilling campaign started in November 2016 to tests its extensions while exploration is ongoing on other nearby targets.

Bakatouo's significant oxide mineralisation suggests potential to supplement both the existing heap leach operation and the future CIL project.

**Figure 9.4.2 Simplified Ity Area Map with Exploration Targets**



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# 10.0 DRILLING

## 10.1 Introduction

The drilling procedures referred to in this section apply to the drilling campaigns 2013 to 2016. From 2013, the drilling and sampling have been under the supervision of the SMI Geology Department. All drilling prior to 2013 is considered historical and details relating to the drilling and sampling are not well documented.

All recent and validated historical drilling data has been used in the current mineral resource estimate and is described in in greater detail in Sections 11, 12 and 14 of this report: Sample Preparation; Analysis and Security; Data Verification and; Mineral Resource Estimate, respectively.

## 10.2 Drilling Methods

Drilling was not active during the site visit undertaken by the CP and the following summary is based on discussions with SMI Geology staff and the previous description documented in the Technical Report for the Ity Gold Mine (Body, 2015).

The drilling equipment has been described as in good repair and operated by experienced contractors familiar with the equipment. Technicians on site worked confidently and are well trained. The drill sites were well ordered, clean and set up so that there was enough space to work safely. Although there are no specific procedures manuals relating to the actual drilling, the methods used by drilling contractors are described as following accepted industry practice

SMI has a comprehensive procedures document that describes the standard procedures for both diamond drilling and RC drilling (conventional and AirCore) at the drill site. The manuals suggest site layouts and give procedures and examples for the handling of samples, marking of boxes, cores, chip bags and boxes etc. The general procedures for diamond drilling are summarised in ~~Figure 10.2.2~~ Figure 10.2.1 and general procedures for RC and AirCore drilling are shown in Figure 10.2.2. ~~Figure 10.4.1.~~

The core is boxed, marked up and oriented at the drill rig. Friable core is packed in plastic to minimised loss during transport and handling. Meterage is measured, recorded and recovery calculated at the drill site. Once drilling is completed the site is cleared, sumps drained and drillholes marked with PVC pipe with all relevant drillhole information. For future reference, a 30 cm x 30 cm x 10 cm high concrete slab is built around the collar pvc, with hole identification, collar dip and azimuth, and total depth inscribed into the concrete.

Core sizes used in drill programs include PQ, HQ and NQ. ~~Triple tube?~~

RC samples are collected in bags and either spilt on site (older drilling) or split in the core shed (recent drilling). Reference samples are stored in chip trays for logging.

RC hammers were commonly 5.25" with some 5.50" in diameter, and they were of the face sampling configuration to minimise cross sample contamination.

The photographs in this report were taken by the CP during the 2016 site visit.

**Figure 10.2.1 General Procedures – Diamond Drilling**

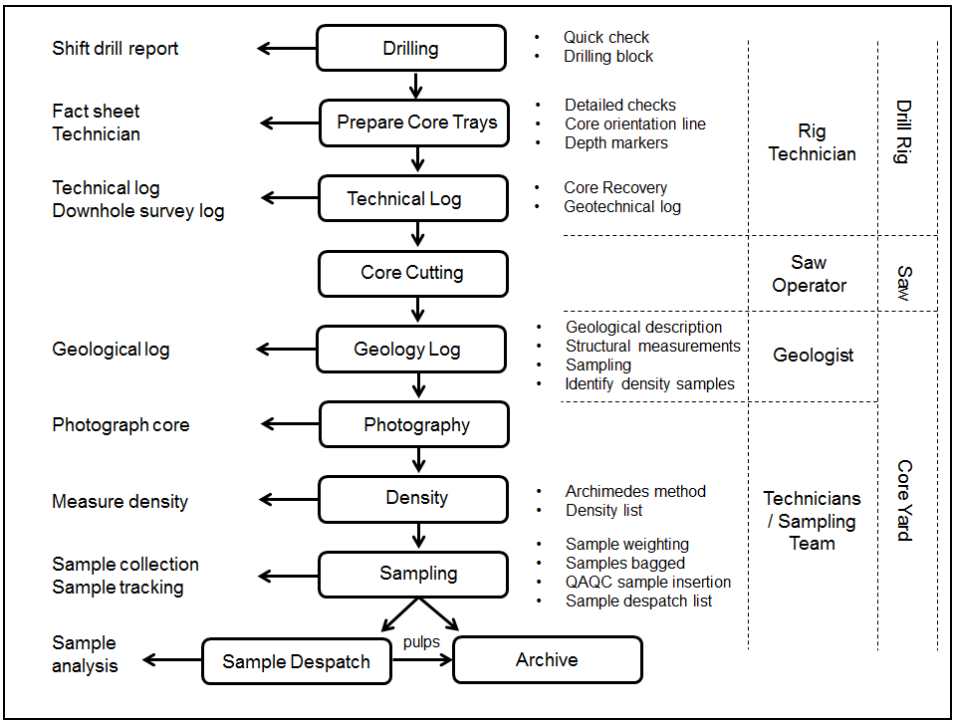
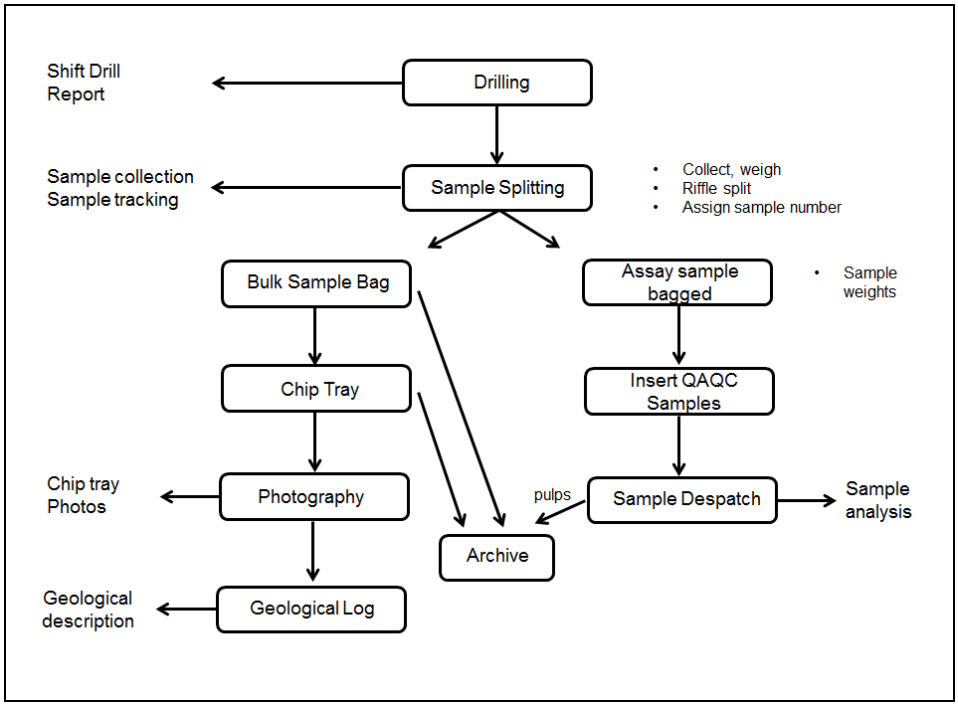



**Figure 10.2.2 General Procedures – RC and AirCore Drilling**



## 10.3 Surveying

### 10.3.1 Drillhole Collars

All drillhole collars are sited and checked using DGPS by qualified staff. Collar positions have been independently verified since the project began by plotting the collar positions on satellite images, applying the coordinate transformation and undertaking a field check to verify the position. Elevations were compared to the topographic survey and overall the collar positions appear to be accurate.

### 10.3.2 Downhole Surveying

Downhole survey deviations on diamond and RC drillholes ~~at~~ were measured using the Reflex EZ SHOT$^{TM}$ system. The first measurement is taken between 10 m and 15 m downhole to verify the inclination and orientation of the drillhole. Subsequent measurements are taken at 25 m intervals for the first 100 m, then at 50 m intervals until the end of the drillhole. The last measurement is taken at the bottom of the hole. Orientation of the core was undertaken in fresh rock using the Reflex ACTII system with measurements approximately every 30 m.

A full procedures manual for downhole surveying including a quality control / verification procedure has been prepared by SMI.

Cube did not observe any surveying as there were no active drilling during the site visit. The database was interrogated and where some minor anomalies were found the data was either corrected or removed.

## 10.4 In Fill Drilling

### 10.4.1 Aires

Drilling was carried out in three phases, an initial diamond drilling phase to understand the layering and depth and a second phase to sample the deposit. The slope of the dump was not drilled in the second phase because access was not possible for the drill rig. A third phase of drilling in 2015 sampled the slopes of the leach pad.

***Diamond Core Drilling***

Diamond core drilling at the Aires was completed as the first phase of drilling with nine drillholes (767.5 m) to define the different levels and facies found in the residues. One deep drillhole was drilled to identify the material beneath the Aires.

***AirCore Drilling***

The second and third phase of drilling at the Aires used the AirCore method. A 40 m x 40 m grid was drilled totalling 89 drillholes for 4,249 m. An additional 58 drillholes were completed on the slopes during 2015. Close spaced drilling included a cross at 5 m spacing. This was intended to assess the short scale variability of the deposits.

### 10.4.2 Daapleu

Drilling at Daapleu was undertaken between July and December 2013 and again in 2014 and 2015. Diamond core was the primary sampling method. RC drilling was used to twin drillholes completed for the BRGM 1998 drill campaign; to test a soil anomaly in Daapleu South (two drillholes) and to test a structure in the southwest of Daapleu (four drillholes).

***Diamond Core Drilling***

A total of 79 drillholes were completed as part of the 2014 / 2015 drilling campaign to test the extent of mineralisation at Daapleu. The drilling grid was on a nominal 25 m x 50 m spacing with drillholes oriented towards N135° and with a dip of minus 50 - 60° from horizontal. At Daapleu four drillholes were drilled vertically.

**Figure 10.4.1 Drillhole Collar DA14-139 at the Daapleu Deposit**



***RC Drilling***

A total of 1,946 m in 19 drillholes was completed with 18 dipping drillholes and one vertical. Four of the drillholes were oriented towards N110° with the remainder oriented at N135°.

### 10.4.3 ZiaNE

The combined 2008 and 2014 drilling has totalled 375 drillholes for 19,526 m completed. Historical assays for ZiaNE were determined to be unreliable and these drillholes were re-assayed in 2014. During the 2015 exploration campaign an additional 32 diamond drillholes were completed.

### 10.4.4 Verse Ouest

A total of 32 diamond drillholes were completed at the Verse Ouest dump. In 2013, 14 drillholes were drilled with the remainder completed in 2014. The total drill metres at Verse Ouest is 1,180 m.

### 10.4.5 Mont lty

Drilling at Mont lty dates back to 1996, when BRGM drilled nine diamond drillholes to test for gold occurrences. Drilling for the current Mont lty mineral resources was completed in 2014 with the details covered in the 2014 mineral resource report (Bose and Falleau, 2014) and in the end of drilling campaign report (Dosso, 2014).

### 10.4.6 lty Flat

BRGM started drilling lty Flat in the early 1990's. In 1994, BRGM drilled a further 11 diamond drillholes totalling 1,133 m to test for gold mineralisation.

## 10.5 Drilling Summary

The drillhole types, number of holes and metres within database supplied by EDV is summarised in Table 10.5.1 and Table 10.5.2.

**Table 10.5.1 Drillhole Database Summary - Number of Holes**

| Deposit | DD | RC | RC-DD | AC | RAB | TR | CS | PIT | Total |
|---|---|---|---|---|---|---|---|---|---|
| Aires | 10 | - | - | 159 | - | - | 3 | - | 172 |
| Daapleu | 212 | 49 | - | - | - | - | 20 | 131 | 412 |
| Gbeitouo | 68 | 56 | 7 | - | 41 | - | - | 64 | 236 |
| lty Combined | 341 | 347 | - | 9 | - | 1 | 1 | 428 | 1,127 |
| Teckraie | 31 | 2 | - | 85 | - | - | - | - | 118 |
| Tontouo | 41 | 1 | - | - | - | - | - | - | 42 |
| Verse Ouest | 32 | - | - | - | - | - | - | - | 32 |
| Versest | 9 | 6 | - | 9 | - | - | - | - | 24 |
| Walter | 70 | 25 | 6 | - | - | - | - | - | 101 |
| ZiaNE | 237 | 10 | - | 22 | - | - | 3 | - | 272 |
| **Total** | **1,051** | **496** | **13** | **284** | **41** | **1** | **27** | **623** | **2,536** |

**Table 10.5.2 Drillhole Database Summary - Number of Metres**

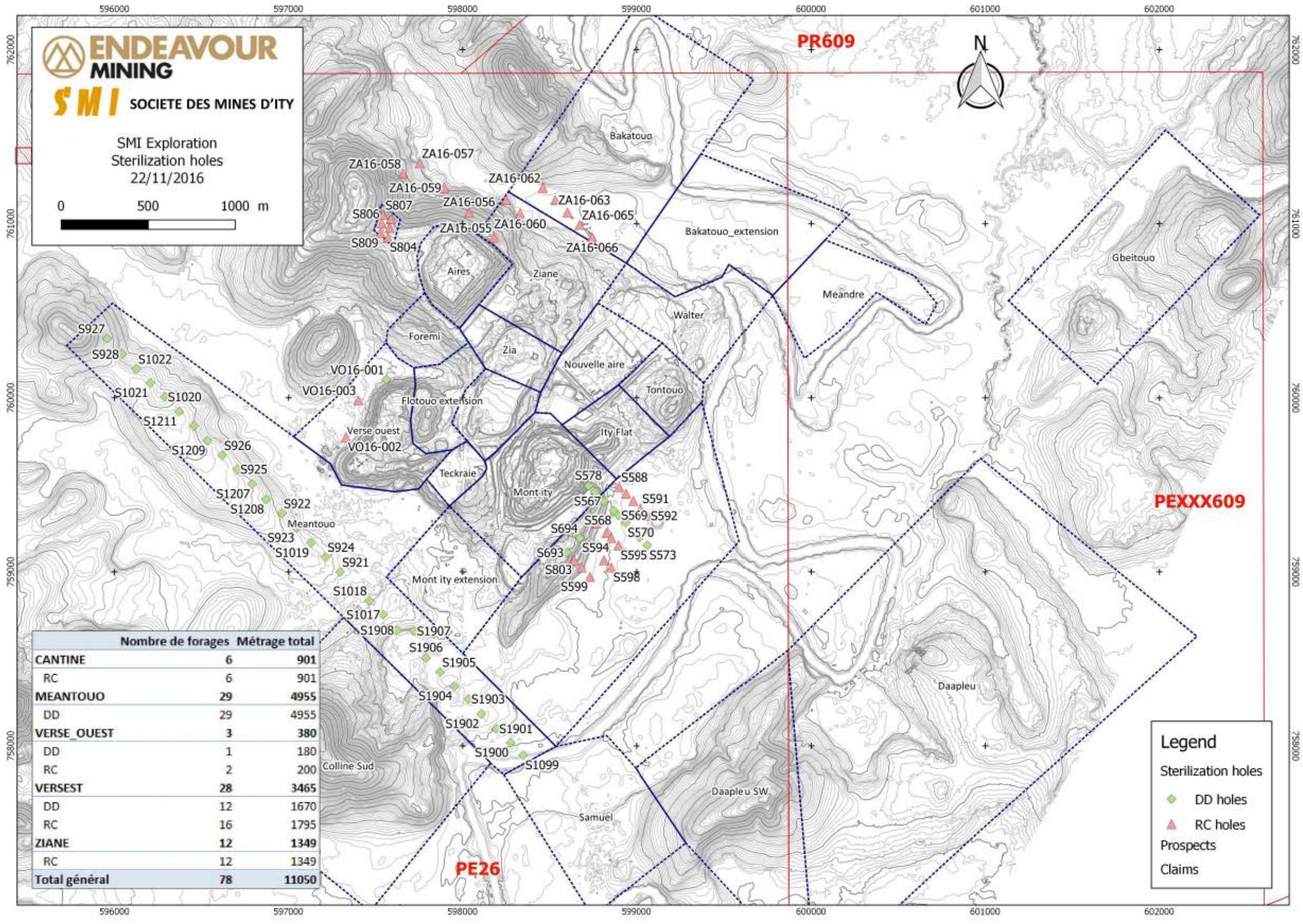
| Deposit | DD | RC | RC-DD | AC | RAB | TR | CS | PIT | Total |
|---|---|---|---|---|---|---|---|---|---|
| Aires | 768 | - | - | 6,455 | - | - | 39 | - | 7,262 |
| Daapleu | 29,093 | 4,242 | - | | - | - | 194 | 1,471 | 34,999 |
| Gbeitouo | 9,203 | 4,309 | 863 | - | 1,044 | - | - | 847 | 16,266 |
| Ity Combined | 46,488 | 17,735 | - | 702 | - | 28 | 17 | 4,388 | 69,358 |
| Teckraie | 1,079 | 28 | - | 3,191 | - | - | - | - | 4,298 |
| Tontouo | 1,363 | 40 | - | - | - | - | - | - | 1,403 |
| Verse Ouest | 1,181 | - | - | - | - | - | - | - | 1,181 |
| Versest | 903 | 586 | - | 900 | - | - | - | - | 2,389 |
| Walter | 6,272 | 2,253 | 613 | - | - | - | - | - | 9,137 |
| ZiaNE | 25,912 | 926 | - | 828 | - | - | 216 | - | 27,882 |
| **Total** | **122,260** | **30,118** | **1,475** | **12,076** | **1,044** | **28** | **466** | **6,706** | **174,173** |

## 10.6 Sterilisation Drilling

Since 2013, 11,050 metres (78 holes) of sterilisation drilling were completed on the following areas:

- East waste dump.
- West waste dum.
- ZiaNE waste dump.
- Meantouo.
- Cantine.

Sterilisation drilling still needs to be completed in areas which will be affected by construction of facilities associated with the CIL Plant (Gbeitouo and Daapleu waste dumps, plant CIL, etc.). These drill programs are scheduled for 2017 and 2018.




| | Nombre de forages | Métrage total |
|---|---|---|
| **CANTINE** | **6** | **901** |
| RC | 6 | 901 |
| **MEANTOUO** | **29** | **4955** |
| DD | 29 | 4955 |
| **VERSE_OUEST** | **3** | **380** |
| DD | 1 | 180 |
| RC | 2 | 200 |
| **VERSEST** | **28** | **3465** |
| DD | 12 | 1670 |
| RC | 16 | 1795 |
| **ZIANE** | **12** | **1349** |
| RC | 12 | 1349 |
| **Total général** | **78** | **11050** |

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# 11.0 SAMPLING PREPARATION, ANALYSIS AND SECURITY

## 11.1 Sampling

From 2013, the sample preparation, analysis and security have been under the supervision of the SMI Exploration Department. The processes described below apply to all project areas for drilling campaigns from 2013 onwards and for any resampling of historical drilling during this period. SMI has procedure manuals which document all sampling protocols in detail. In addition, regular monitoring of sampling procedures have been undertaken to ensure compliance and allow improvements where appropriate. Information pertaining to historical drilling and sampling prior to 2013 is limited.

Sample collection by SMI, or by third parties contracted by SMI, were subject to quality control procedures that ensured that industry best practice was utilised for the handling, sampling, transport, analysis, storage and documentation of sample material and analytical results.

### 11.1.1 Sampling

***RC Sampling***

RC samples were collected as 1 m sample lengths in bags directly from the cyclone discharge at the drill rig. The sample was then riffle split into a numbered sample bag with the corresponding sample tag to produce a sample of approximately 2 - 4 kg in size. The riffle splitters, plates, tubs and working areas were cleaned with compressed air after each sample was processed. The remaining reject sample was returned to the bulk plastic bag and either remained at the drill site or transported to the core yard facility. A 2 - 4 kg in size representative sample was riffle split at the drill site and stored in large plastic drums in addition to a small portion of drill chips collected from the reject and catalogued in chips trays for future reference.

Approximately 30 assay samples were placed in large polyweave bags and transported to either the on-site SMI preparation laboratory or Bureau Veritas in Abidjan, Côte d'Ivoire. The Bureau Veritas laboratory is not currently ISO17025 accredited but are in the process to achieve this. The laboratory does however work under the accreditation of the global Bureau Veritas group of laboratories including Australia and Canada and are covered by the groups ISO9001, ISO14001, ISO18001 and IFIA certificates.

***Diamond Drilling Sampling***

Core from diamond drilling ('DD') was placed in either steel or timber core boxes. Core trays were marked with the drillhole number and downhole distance for the start and finish of the core in the tray. Core orientation was undertaken in competent fresh rock and completed at the drill site. The core was geotechnically logged and photographing at the drill site before being transported to the SMI sample facility on site where the core was laid out for geological logging and other data collection.

Specific intervals for sampling based on lithology were identified during the logging process and the core was cut in half along its longitudinal axis with a purpose-built diamond-blade core saw. After the core was cut, the right hand side of the cut core (looking down hole) was placed in sample bags and sealed with the remaining core left in the core box. The samples were placed in large polyweave bags and transported to either the on-site SMI preparation laboratory or Bureau Veritas (Abidjan).

### 11.1.2 Sample Submission

Sample submission forms detailing the sample number sequences and instructions accompanied every sample dispatch to both the on-site SMI preparation laboratory and the Bureau Veritas laboratory in Abidjan, Ivory Coast. On arrival at the SMI preparation laboratory, the samples were checked against the submission to ensure all samples were received. On arrival at the Bureau Veritas laboratory, the samples were again checked against the submission to ensure all samples were received. On completion of assaying and once results have been delivered by Bureau Veritas, SMI geological staff confirmed that all the submitted samples were analysed by the methods requested by the Client.

### 11.1.3 Sample Preparation and Analysis

***Sample Preparation***

Sample preparation is undertaken at both the onsite SMI laboratory and the Bureau Veritas laboratory in Abidjan. The SMI laboratory and QA/QC protocols were set up using Intertek guidelines and supervision. Both the SMI and Bureau Veritas laboratory protocols are essentially the same and include:

- dried in oven at 105 to 110°C
- crushed to a nominal 2 mm diameter using a jaw crusher and sub-sampled with a riffle splitter
- approximately 0.5 kg is pulverised with a LM2 such that 85 to 90% passing 75 µm
- systematic checks are undertaken to verify particle size.

**Figure 11.1.1 SMI Sample Preparation Facility**



***Sample Assaying***

When the sample pulps are received at the Bureau Veritas laboratory (Abidjan) they are logged into the laboratory information management system ('LIMS'). Fire Assay with an Atomic Absorption finish is the primary technique for gold analysis with three alternate assay methods also used depending on the gold grade. The assay methods include the following:

- Fire Assay (BV code FA450 or FA001) - Nominal 50 g charge analysed. Silver used as secondary collector and Au is determined with AAS finish. Samples are analysed at the Abidjan facility.

- ICP-ES (BV code AQ300) - digest for 36 elements is done for samples with greater than 0.5 g/t Au by ACME laboratories in Vancouver Canada. It must be noted that the Aqua Regia method is a partial digest and major elements are not used for whole rock analysis.

- Fire Assay with gravimetric finish (BV FA550 or FAGRA01) - is carried out on all samples with grade greater than 30 g/t Au.

**Figure 11.1.2 AAS Gold Determination - Bureau Veritas Laboratory (Abidjan)**




***ICP Analysis***

From a total of 179,732 assays contained within the database, 22,729 samples were analysed using ICP for multiple element analysis. The main sample type submitted for ICP analysis is from DD with a total of 17,019 samples. In addition, 2,325 samples were submitted from RC, 3,303 samples from aircore drilling ('AC') and 82 others unknown. A total of 34 elements were analysed being Ag, Al, As, B, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, V, W, Y and Zn.

The sample preparation was carried out by the site SMI laboratory or Bureau Veritas (Abidjan). Where the fire assay grade was greater than 0.5 g/t Au, Bureau Veritas was instructed by SMI to send the pulps to AMCE laboratories in Vancouver, Canada for ICP analysis. The AMCE laboratory in Vancouver is ISO17025 and ISO9001 accredited.

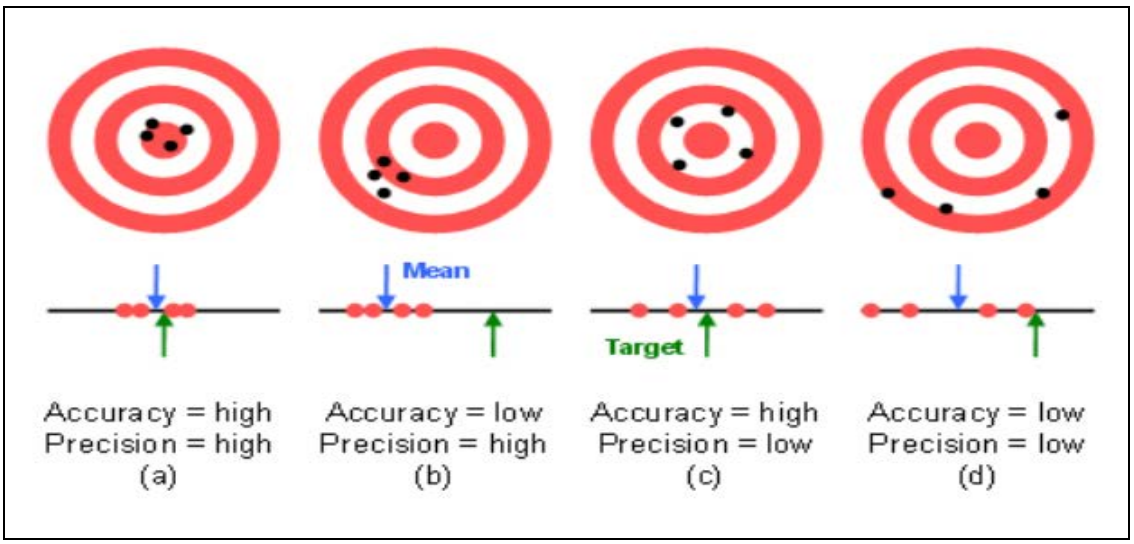
### 11.1.4 Quality Assurance and Quality Control Programs

Cube has reviewed and independently assessed all available QA/QC sample data collected by Endeavour Mining Corporation from RC and DD drilling at the Ity Gold Project for the period December 2011 to June 2016.

For the QA/QC review, all gold assay values are reported in ppm units and all assay values reported as being below the lower analytical detection limit are set to half the detection limit for the analysis. All control samples were assessed on the basis of accuracy and precision. The precision of sample results is the measure of how closely the results can be repeated. The accuracy of sample results relates to how similar the results are to the known value.

It is possible to have good accuracy without good precision or good precision without good accuracy as shown in Figure 11.1.3. Precision is measured by the use of duplicate and replicate assays, whereas accuracy is measured through the use of reference materials.

**Figure 11.1.3 Accuracy and Precision Concept**




The assay laboratory and analytical techniques utilised by Ity Gold Mine are summarised in Table 11.1.1. Bureau Veritas laboratory is the primary assay laboratory for sample preparation and Au analysis. ACME (Vancouver) is utilised as a secondary laboratory for multi element analysis. ALS is utilised as a secondary laboratory for umpire results. A SMI onsite laboratory was utilised as the primary laboratory before 2012 for both sample preparation and Au analysis. Since 2014, a second SMI sample preparation facility has been constructed on site for the processing of exploration drilling. ALS and SGS were utilised before 2012 for historical information and SGS has been used during 2016 for mine grade control samples.

**Table 11.1.1 Analytical Techniques**

| Laboratory | Laboratory Method | Description | Detection Limit (ppm) | Analyte | Country | Assay Aliquot Size |
|---|---|---|---|---|---|---|
| Bureau Veritas | Au_FA001 | Lead collection Fire Assay with AAS finish | 0.01 | Au | Cote d'Ivoire | 40 g |
| | Au_BL002S | Bulk Leach extractable gold ppb | 0.01 | Au | | 1 kg |
| | Au_FA450 | Lead collection Fire Assay with AAS finish | 0.01 | Au | | 50 g |
| | Au_FA550 | Lead collection Fire Assay / Gravimetric | HG | Au | | 50 g |
| | Au_FAGRA01 | Lead collection Fire Assay / Gravimetric | HG | Au | | 50 g |
| ALS | Au_AA26 | Atomic absorption spectroscopy | 0.01 | Au | Cote d'Ivoire | 50 g |
| | Au_GRA22 | Fire Assay Fusion / Gravimetric | 0.05 | Au | | 50 g |
| | Au-FA | Generic method - unknown | 0.01 | Au | | Unknown |
| SMI | Au_TOT | Unknown – onsite laboratory | 0.01 | Au | Ity Site | Unknown |
| | Au_CY | Unknown | 0.01 | Au | | Unknown |
| | Au_Unk | Unknown | 0.01 | Au | | Unknown |
| SGS | Au_FAA505 | Fire Assay with AAS finish | 0.01 | Au | Mali | 50 g |

***Certified Reference Material***

The performance of the Certified Reference Material (CRM) sample data was assessed by plotting the replicate assay values of the CRMs against time on the control charts. Good quality analysis of the CRMs will be characterised by a random distribution of data points around the certified mean value, with 95% of the data points lying within two standard deviations of the mean (Abzalov, 2008). If more than 5% of the CRM's submitted are outside three standard deviations of the certified mean value, then corrective action should be taken. In addition, no trends or significant bias should be observed in the control charts.

Any obvious assay 'outliers' that are likely to be the result of sample mishandling or transcription errors were removed from the dataset prior to analysis, to avoid any skewing of the dataset. Refer to Data Issues for these errors.

In addition, the CRM data set was assessed by Cube using two statistical tests to demonstrate that the analytical accuracy and precision of the assays were comparable to the certified value of the CRM, and considered acceptable within the 95% confidence limit (Abzalov, 2008).

Accuracy Test – involved the comparison of the arithmetic mean of the replicate analysis of the CRM (m) against its certified mean ($\mu$), and if the following condition is satisfied then the analytical results are considered acceptable with regard to accuracy:

$$|\mathrm{m} - \mu| \leq 2\sigma_L$$

here $\sigma_L$ is the standard deviation of the replicate analyses of the CRM.

Precision Test (Chi Square) – involves the comparison of the estimated standard deviation of the replicate assays against the CRM deviation, and if the following condition is satisfied then the analytical precision is considered acceptable:

$$\left(\frac{S_w}{\sigma_c}\right)^2 \leq \frac{X^2_{(n-1)0.95}}{n-1}$$

Where:

- $S_w$ is the standard deviation of the replicate analyses of the CRM
- $\sigma_C$ is the certified value of the CRM standard deviation
- $X^2_{(n-1)0.95}$ is the critical value of the 0.95 quartile of the $X^2$ distribution at $(n-1)$ degrees of freedom
- $n$ is the number of replicate assays of the CRM.

All CRMs with less than five replicate assays were considered as having insufficient data to assess the performance of the CRM and were excluded from the analysis. In general, a bias of greater than 5% is considered not acceptable.

Cube has reviewed the QA/QC protocols and the supplied control assays for all sample data for the assay period October 2013 – May 2016. A total of 170,686 samples containing 14,559 Certified Reference Materials ('CRMs') at an 8.5% insertion rate, and 2,079 field duplicate checks (1.5% insertion rate) for a combined total insertion rate of 10% were submitted to Bureau Veritas.

Most of the CRMs were sourced from Geostats, with an additional two sourced from Rocklabs due to supply shortages. All of the CRMs analysed were inserted at the SMI preparation laboratory and analysed using Fire Assay. Two poorly performing CRMs G912-7 and G998-1, were identified both of which passed on accuracy, only to fail on precision. This suggests issues within the laboratory, however it is difficult to confirm without adequate umpire sample batches to review. G912-7 has only 19 samples for review and ceased being inserted into the sample stream in early 2014. G998-1 has 481 samples with several failed results scattered across all grade ranges and a strong positive bias. The issue with G998-1 has been identified by site SMI staff and discussed with the laboratory. It was concluded the expected value for the CRM was considered too high as a plausible check on FA analysis.

A selection of CRM control charts including G998-1 are shown below in Figure 11.1.4 to Figure 11.1.6

**Figure 11.1.4 Summary Results for CRM G397-3**

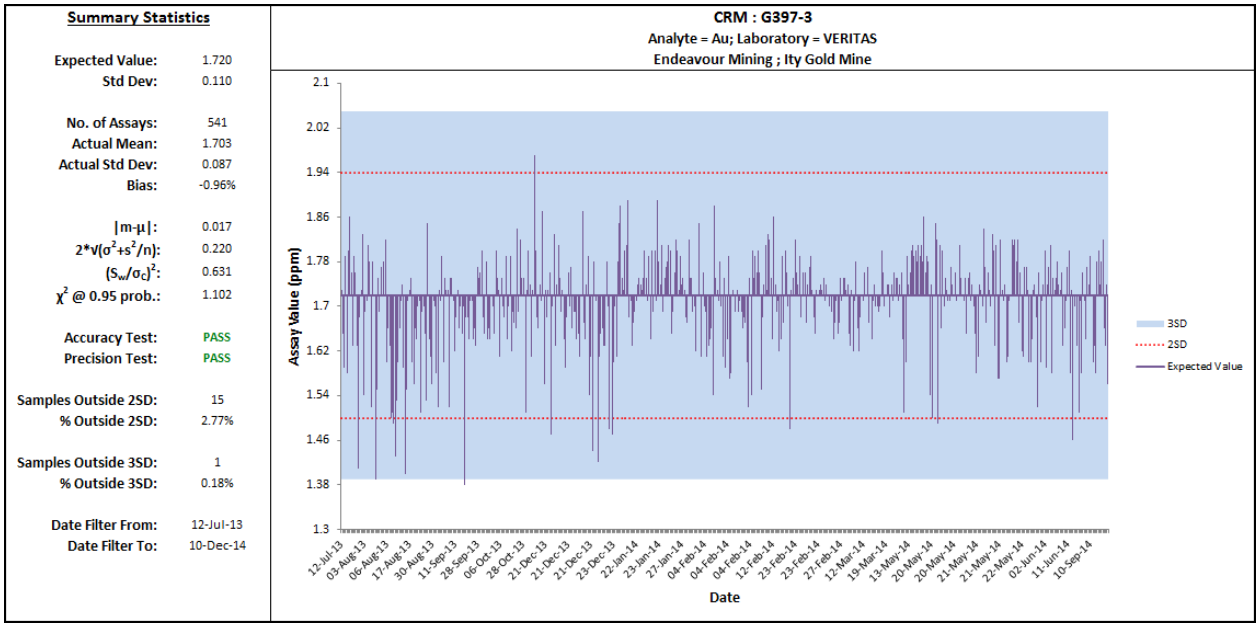


**Figure 11.1.5 Summary Results for CRM G909-5**

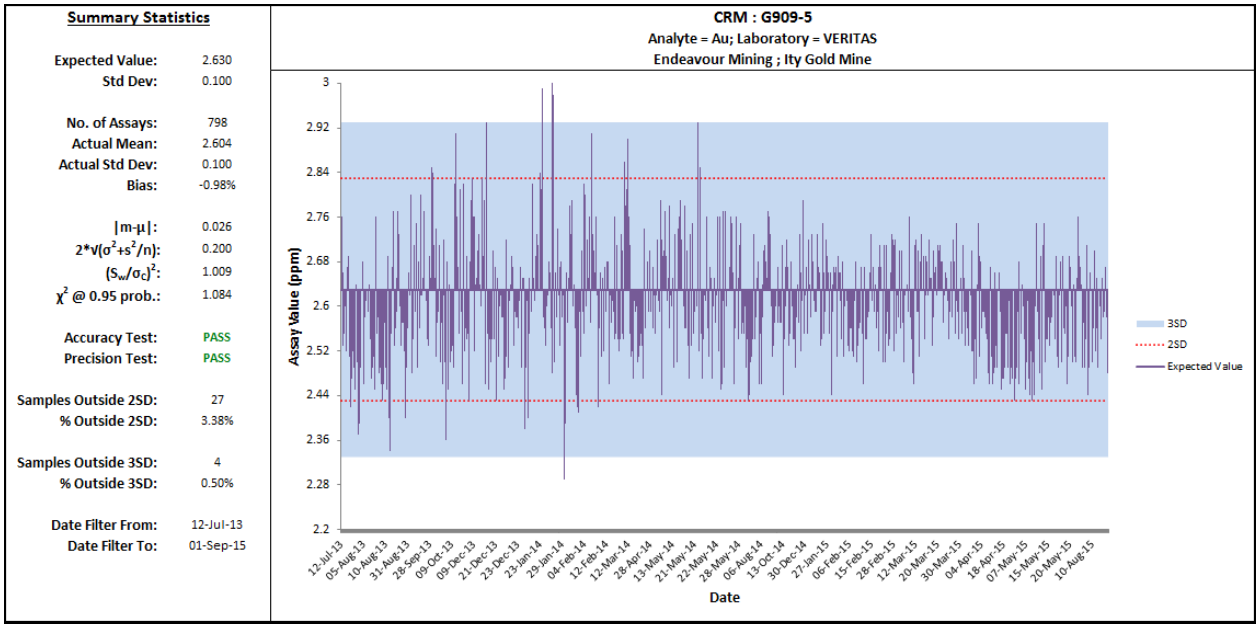
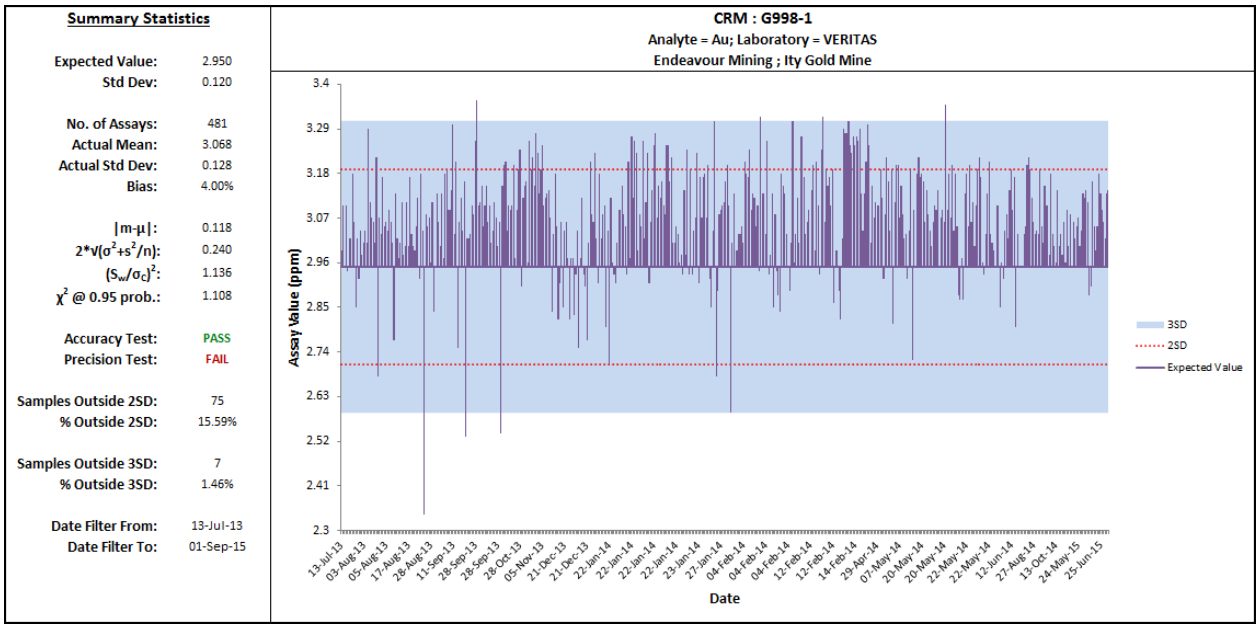



**Figure 11.1.6 Summary Results for CRM G998-1**



### 11.1.5 Blanks

Three different blanks (Coarse Blank, Fine Blanks and False Blank) were also inserted into the sample stream during the period 2012 to 2016, all of which were analysed by the Bureau Veritas laboratory. The majority of these were Coarse and Fine Blanks which passed both accuracy and precision tests. There were also a limited number of False Blanks inserted to check the laboratory does actually assay blank samples. These failed on the precision test which is expected given the samples are not a true blank with an unknown expected value greater than 0.01 Au ppm. The Coarse and Fine Blank types were locally sourced and collected close to Abidjan. They are a mixture of local barren material such as beach sand for the Fine Blank and granite from SISAG quarry in Attinguie for the Coarse Blank, which have been previously tested and confirmed as barren material.

The Bureau Veritas laboratory overall has demonstrated analytical accuracy that is considered acceptable at the 95% confidence limit.

**Table 11.1.2 CRM Performance Summary for Au Fire Assay Method**

| Laboratory | CRM | Certified Value (ppm) | No. Assays | Accuracy Test | Precision Test | % Passing 3SD | % Bias | Period in Use |
|---|---|---|---|---|---|---|---|---|
| Bureau Veritas | G303-2 | 4.15 | 16 | PASS | PASS | 100 | 0 | 2013-2014 |
| | G305-2 | 0.32 | 23 | PASS | PASS | 100 | -1 | 2015-2016 |
| | G306-3 | 8.66 | 17 | PASS | PASS | 100 | 0 | 2015-2016 |
| | G310-10 | 48.53 | 731 | PASS | PASS | 100 | 2 | 2014-2015 |
| | G311-9 | 10.01 | 28 | PASS | PASS | 100 | -1 | 2015 |
| | G313-10 | 46.27 | 11 | PASS | PASS | 100 | 5 | 2015 |
| | **G397-1** | **0.76** | **1** | **FAIL** | **FAIL** | **0** | | **2014** |
| | G397-3 | 1.72 | 541 | PASS | PASS | 100 | -1 | 2013-2014 |
| | G901-7 | 1.52 | 811 | PASS | PASS | 99 | 0 | 2013-2015 |
| | G901-8 | 47.24 | 125 | PASS | PASS | 98 | 2 | 2013 |
| | G908-8 | 9.65 | 574 | PASS | PASS | 99 | 2 | 2013-2014 |
| | G909-5 | 2.63 | 798 | PASS | PASS | 100 | -1 | 2013-2015 |
| | G910-5 | 5.23 | 24 | PASS | PASS | 100 | 2 | 2013-2014 |
| | G912-1 | 7.29 | 496 | PASS | PASS | 100 | 3 | 2013-2014 |
| | G912-4 | 1.91 | 326 | PASS | PASS | 100 | -2 | 2014-2015 |
| | **G912-7** | **0.42** | **19** | **PASS** | **FAIL** | **90** | **6** | **2013-2014** |
| | G996-4 | 0.51 | 25 | PASS | PASS | 100 | -4 | 2013-2014 |
| | G997-3 | 1.41 | 21 | PASS | PASS | 100 | 0 | 2015-2016 |
| | **G998-1** | **2.95** | **481** | **PASS** | **FAIL** | **99** | **4** | **2014-2015** |
| | G998-6 | 0.8 | 22 | PASS | PASS | 100 | 1 | 2015 |
| | GMO-07 | 0 | 1 | | | | | 2014 |
| | SK62 | 4.07 | 74 | PASS | PASS | 100 | 1 | 2013-2014 |
| | SN60 | 8.59 | 75 | PASS | PASS | 99 | 1 | 2013-2014 |
| | ST502 | 51.69 | 9 | PASS | PASS | 100 | 0 | 2013 |
| | **Total** | | **5,249** | | | | | |

**Table 11.1.3 Blank Performance Summary for Au Fire Assay Method**

| Laboratory | CRM | Certified Value (ppm) | No. Assays | Accuracy Test | Precision Test | % Passing 3SD | % Bias | Period in Use |
|---|---|---|---|---|---|---|---|---|
| Bureau Veritas | Fine BLANK | 0.01 | 6,546 | PASS | PASS | 99 | 0 | 2012-2015 |
| | Coarse BLANK | 0.01 | 2,723 | PASS | PASS | 100 | 0 | 2013-2015 |
| | False BLANK | 0.01 | 10 | PASS | FAIL | 80 | 0 | 2013-2015 |

***Duplicate Samples***

The measurement of the relative precision error between paired duplicate sample data is based on the average coefficient of variation ('ACV'). Approximate guidelines for assessing analytical quality allows for a maximum ACV of around 40% for field duplicates in gold deposits with very coarse grained nuggetty gold and 30% for a coarse to medium grained gold deposit.

The field duplicate assays for data collected during 2012 to 2016 for the project to date are within the acceptable ACV limits for Au, with an average ACV of 30% for RC and 36% for DD samples, after applying a 0.10 ppm Au mineralised threshold to the data. The field duplicate assays carried out at ALS in 2012 were outside of the acceptable ACV limits for Diamond but acceptable for RC. These duplicate fails are identified in the data issues section of this report.

Across all duplicate types, there are 37 fails where there is a >150% difference between the original and duplicate results. On closer examination and after applying the 0.10 ppm Au mineralised threshold, many of these failed assay results are significant. This poor correlation could indicate issues with the field sampling procedures or relate specifically to mineral distribution as a consequence of the poor correlation between assays from the original half core sample and the quarter core duplicate sample.

Coarse and Pulp duplicates have passed just within the acceptable limit.

The duplicate performance is summarised in Table 11.1.4 and relative mean paired difference (RMPD), scatter and Q-Q plots for a selection of duplicates in displayed in Figure 11.1.7 to Figure 11.1.12.

## Table 11.1.4 Duplicate Performance Summary

| Description | Laboratory | Number Samples | Absolov guidelines pass/fail | Number Samples >0.1 ppm Au | Absolute Relative Average Difference % | For Filtered Data >0.1ppm Au | | | | Period In Use |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | ACV% | Assays within 10% | Assays within 20% | Assays within 50% | |
| Field | ALS Vs ALS_RC | 246 | Pass | 100 | 4.4 | 35.4 | 34 | 53 | 78 | 2012 |
| Field | ALS Vs ALS_DD | 208 | Fail | 124 | -1.8 | 48.9 | 17 | 31 | 62 | 2012 |
| Field | ALS Vs Unk_DD | 178 | Fail | 145 | -11.7 | 56.1 | 16 | 34 | 65 | 2012-? |
| Field | SMI Vs ALS_DD | 99 | Fail | 99 | -60.9 | 59 | 6 | 17 | 50 | 2012 |
| Field | SMI Vs ALS_RC;DD | 15 | Fail | 14 | -115.5 | 87.6 | 7 | 7 | 7 | 2012 |
| Field | Veritas_Vs_Unk _DD | 15 | Pass | 11 | -11.2 | 41.1 | 18 | 36 | 73 | 2012-? |
| Field | Veritas_Vs_Unk _RC;AC | 213 | Pass | 194 | -1.5 | 28.7 | 35 | 61 | 86 | 2013-? |
| Field | Veritas_Vs_Veritas_RC:AC | 1625 | Pass | 1078 | -0.5 | 31.2 | 39 | 60 | 85 | 2011-2016 |
| Field | Veritas_Vs_Veritas_DD | 3870 | Pass | 2293 | -0.9 | 36 | 25 | 45 | 77 | 2012-2015 |
| Pulp | Veritas_Vs_Veritas_DD;RC | 546 | Pass | 356 | 1.2 | 19.4 | 55 | 78 | 94 | 2014-2015 |
| Coarse | Veritas_Vs_Veritas_DD;RC;AC | 747 | Pass | 485 | -2.7 | 22.2 | 40.2 | 70 | 93 | 2014-2015 |
| Umpire | Veritas_Vs_Unk_DD;RC | 142 | Fail | 142 | 6 | 76.6 | 11 | 19 | 49 | 2012-? |
| Umpire | SMI Vs ALS_RC;DD | 15 | Fail | 14 | -115.5 | 87.6 | 7 | 7 | 7 | 2012 |
| Umpire | SMI Vs ALS_DD | 99 | Fail | 99 | -60.9 | 59 | 6 | 17 | 50 | 2012 |

## Figure 11.1.7 RMPD Plot ALS– Field RC

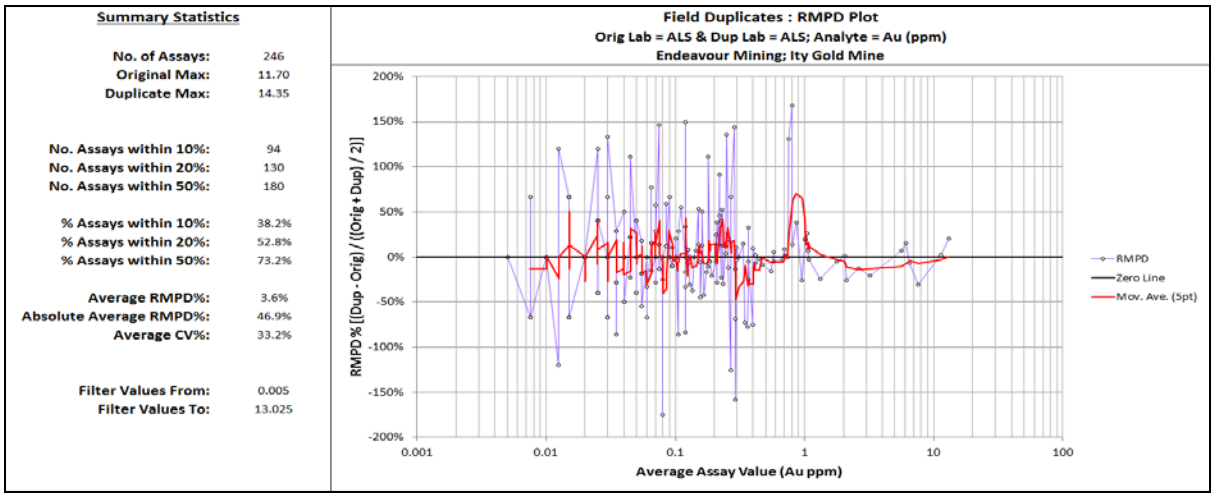


**Figure 11.1.8 Scatter Plot ALS– Field RC**

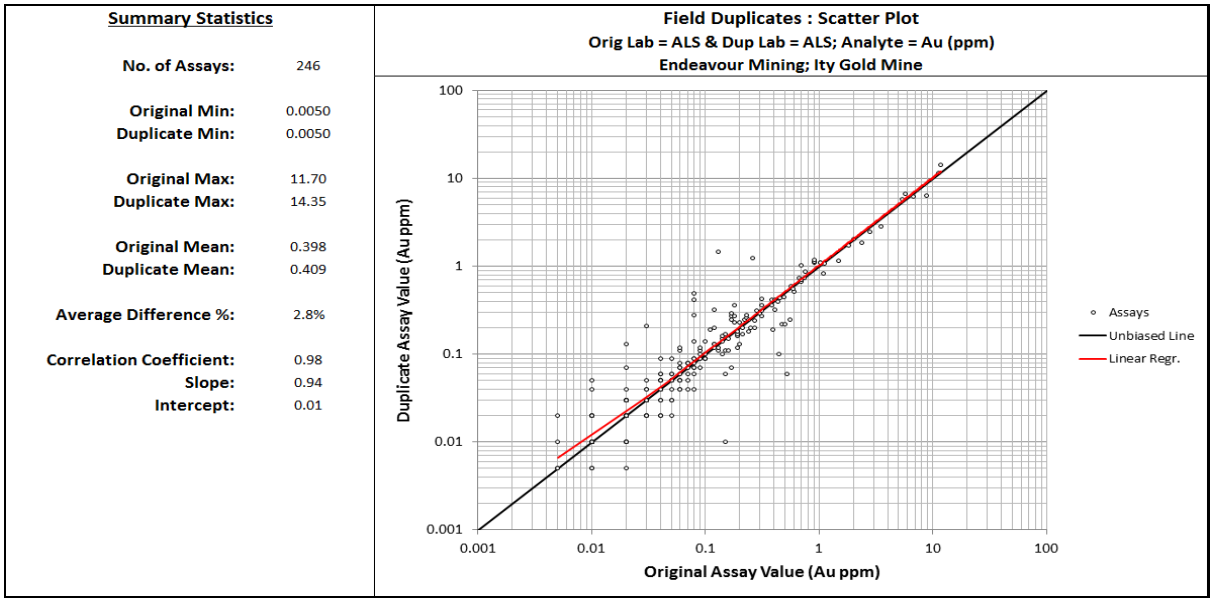



**Figure 11.1.9 Q-Q Plot ALS – Field RC**

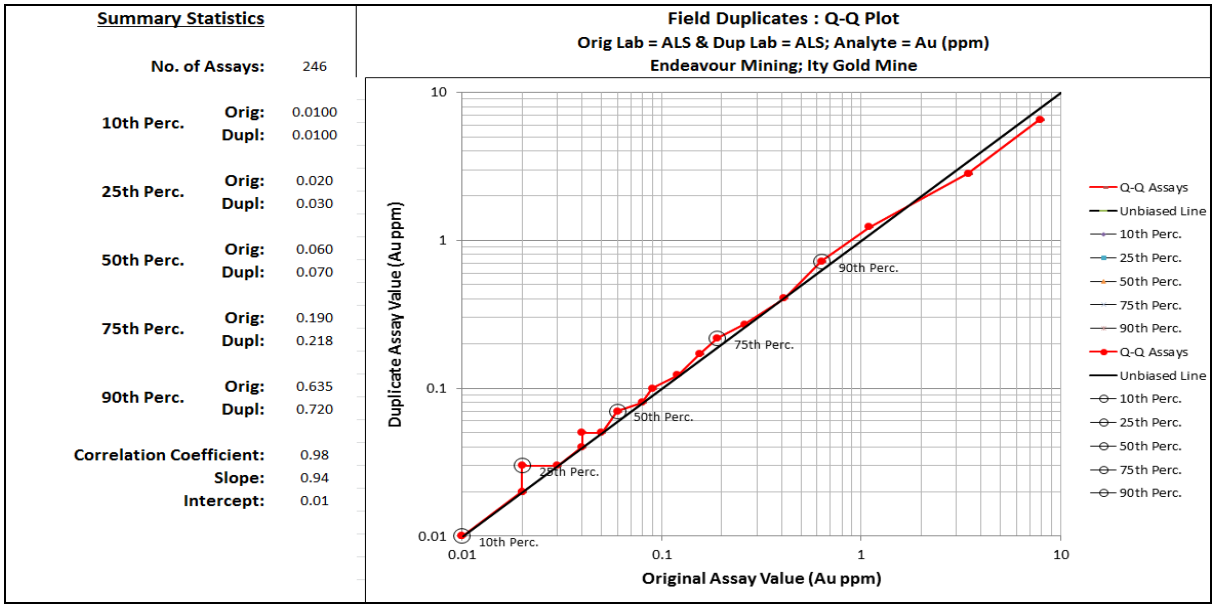


**Figure 11.1.10 RMPD Plot Bureau Veritas – Field DD**

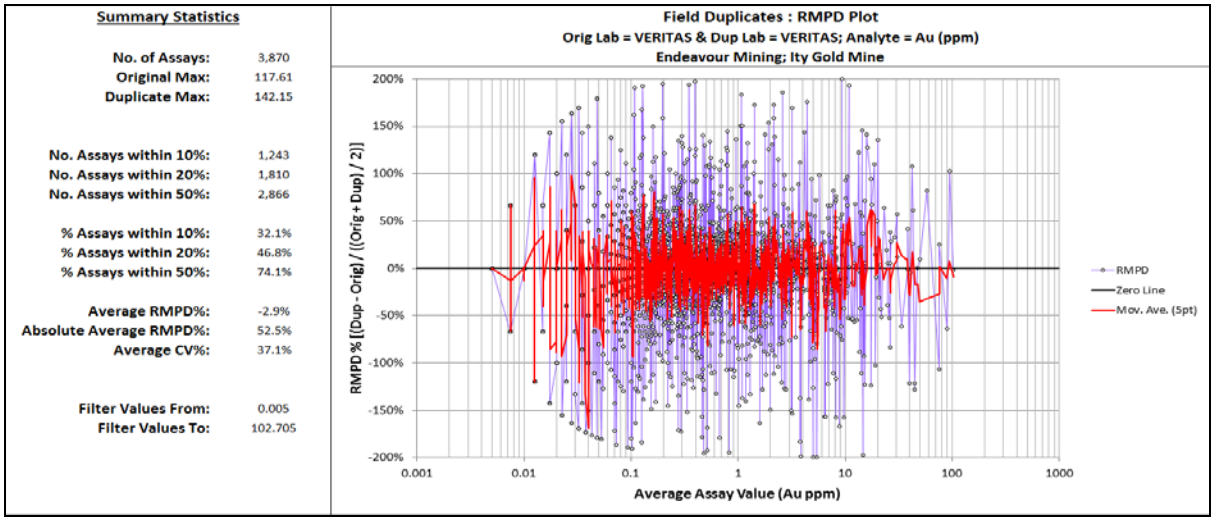



**Figure 11.1.11 Scatter Plot Bureau Veritas – Field DD**

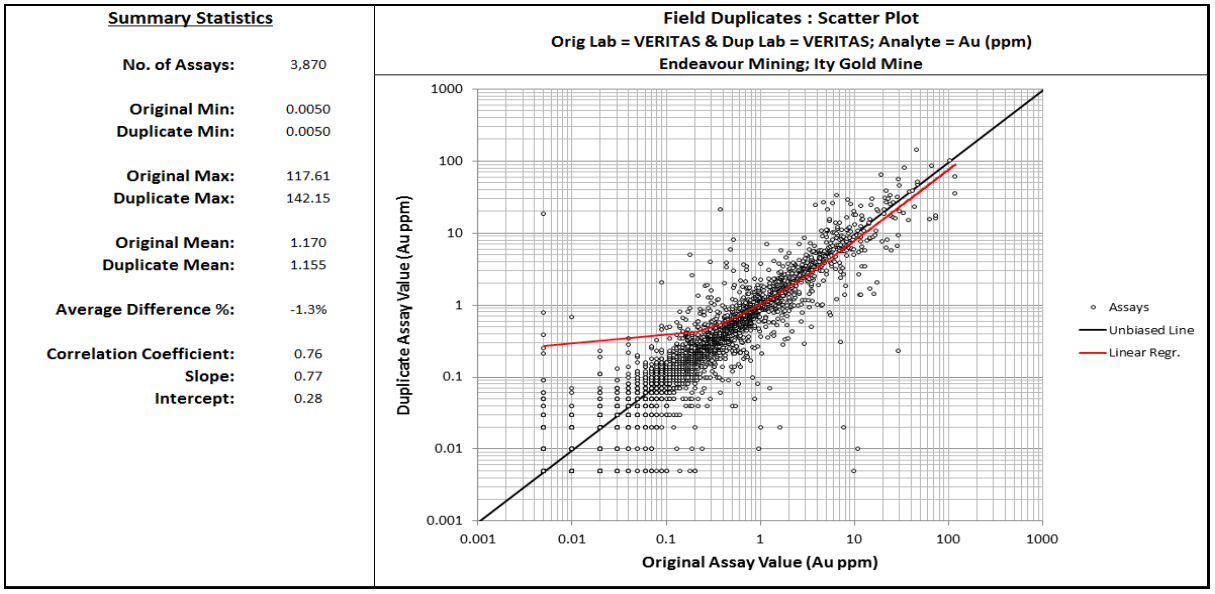
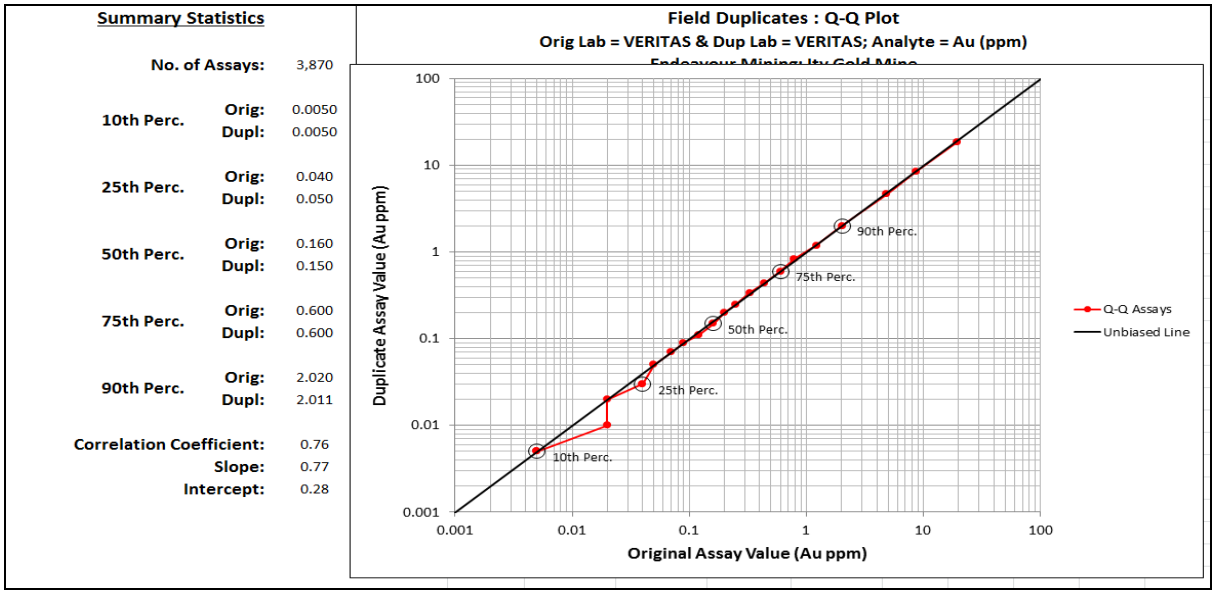


**Figure 11.1.12 Q-Q Plot Bureau Veritas – Field DD**




***Summary and Recommendations***

The following issues were highlighted during the review:

- No umpire duplicate sampling was part of the data supplied to Cube, however some has been undertaken for the recent drilling (2016).
- Laboratory ID and Labjob IDs and laboratory methods are important factors when reviewing duplicate results, however the dataset contains conflicting Laboratory IDs and missing Labjob IDs and laboratory methods in part due to limitations with the database extraction, which makes assessing the reliability of the data difficult.
- Using multiple CRMs containing similar expected assay values is unnecessary.
- Data issues which need to be resolved have been identified in Appendix 3.1 – these include examples of Misclassified Standard IDs; Possible contamination; Possible swapped samples and high and low failed results all of which require further investigation and advised adjustment within the live database.

The following general recommendations are summarised below:

- Coarse reject samples for the mineralised zones in each batch should be retained for a period of time and be the primary source of Field duplicates for DD. The coarse rejects can be used as duplicate samples to perform additional checks on the precision and reproducibility of results from the primary laboratory. These samples are of superior quality repeatability than ¼ coring as a field duplicate.

- Coarse reject and pulp duplicates should be focused on mineralised zones after the original results have been received and reviewed. Precision and accuracy of non-mineralised samples outside of mineralised areas are of limited value.

- It is recommended that umpire duplicate assaying (either coarse rejects or pulps) be carried out at an alternate laboratory on a routine basis. Representative pulps should be chosen from mineralised zones and submitted to a secondary umpire laboratory on a regular basis to determine whether there is any assay bias at the primary laboratory. Recommended 3-5% of DD samples.

- A review of Laboratory ID; Laboratory Method; Laboratory Job number data is required between records in Summary despatch; QA/QC and Assay tables to ensure consistent information is recorded.

- A review of all duplicate types is required to clearly define each category and allow for clean individual export per category.

- Reduce down to one of each of the similarly valued CRMs identified.

### 11.1.6 Bulk Density

SMI has historically used various methodologies to measure bulk density depending on the type of material. Three general methods have been used and are described as hydrostatic, filled volume and volume displacement.

Hydrostatic method - uses whole core and consists of taking measurement of the mass of the core in air and suspended in water (Figure 11.1.4). Porous samples are covered with plastic wrap (food grade) or paraffin wax before immersion in water. Densities are normally measured on dry samples however where samples are not dried, this is documented and a separate calculation is used.

- The density is calculated as the ratio:
  - (dry weight - suspended weight) / dry weight.
- Filled volume method - used for unconsolidated material from DD core or RC drilling. In this method a container of known mass is filled, lightly compacted and weighed. The density is calculated as the ratio:
  - mass / volume.
- Volume displacement method - uses a graduated cylinder filled to a set volume. A piece or pieces of rock are immersed in the water and the change in volume recorded. This method has a large error due to the lack of accuracy of the graduated cylinder used for measurement. This method has only been used where the other two methods are not feasible. The density is calculated as the ratio:
  - mass of the rock / change in volume.

**Figure 11.1.13 Hydrostatic Density Determination Equipment**



All available bulk density data was supplied to Cube by SMI. This is a total of 3,391 determinations. Only determinations measured using the hydrostatic method were used as part of the Mineral Resource estimation. This is a total of 2,948 determinations for the entire Ity Gold Project. Density statistics are summarised for individual deposits.

### 11.1.7 Sample Security

All DD core and RC samples are transported at the end of every shift to SMI's secure sample facility adjacent to the Ity Exploration office. The sample facility has 24 hour security by G4S, an independent security firm whom monitor activities in the lab as well as the broader exploration site facilities. Once the DD core is sampled, both RC and DD samples have the QA/QC samples inserted into the sample stream by SMI personnel prior to dispatch to the laboratory. For assay samples that are prepared and analysed off site, the samples are placed in sealed and numbered polyweave or plastic bags for transport. Where the preparation has been completed in the on-site SMI laboratory, the pulps sample weights are recorded prior to bagging and transportation. The samples are delivered by batch to the laboratory by SMI personnel. Upon receipt, the laboratory personnel signed off on the secure receipt of all the samples.

All aspects of the RC and DD sample collection were conducted by personnel under the supervision of SMI's experienced geological staff.

### 11.1.8 Authors Statement

The author was not able to witness RC drill sampling during the site visit as drilling was not in progress. The author was however able to discuss the procedures with site personnel and was able to:

- View DD core sampling.
- Sample preparation at the on-site SMI laboratory.

- Density determination at the on-site SMI laboratory.
- Sample preparation and analysis at the Bureau Veritas laboratory (Abidjan).
- Sample storage and security at the on-site SMI sample facility.

Assay methods are appropriate for the deposit types encountered on the property. ICP should be extended to include a percentage of samples with gold grades less than 0.5 g/t Au, particularly at Daapleu, in order to fully understand the relationship between gold and arsenic.

The on-site SMI preparation laboratory is an open air facility with partially closed sections for the sample splitting and milling stages. The SMI staff appears to be well trained and able to undertake the work in a proficient manner. Audits of the preparation laboratory are undertaken by staff from the Exploration Department. This is encouraged to continue to maintain an acceptable standard.

Bureau Veritas is a global independent provider of assaying and analytical testing services for the mining and mineral exploration industry with consistent quality standards implemented across all regions. Whilst the laboratory in Abidjan is not formally accredited, the laboratory aims to operate to ISO 17025 standards. The AMCE laboratories in Vancouver Canada is formally accredited to ISO 9001 and 17025 standards.

Overall the sample collection and preparation, analytical techniques, security and QA/QC protocols implemented for the Ity Gold Project are consistent with standard industry practice and are suitable for the reporting of exploration results and for use in Mineral Resource estimation. The sampling procedures are adequate for and consistent with the style of gold mineralisation under consideration.

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# 12.0 DATA VERIFICATION

In accordance with National Instrument 43-101 guidelines, the Author visited the Ity Project between18 and 22 June 2016.

The site visit involved data verification, inspections and reviews of the following:

- Geology and exploration history of the permit area.
- Current drilling and sampling procedures.
- Ground checking of project areas.
- Geochemistry and drilling data.
- Diamond core inspection for each deposit.
- QA/QC procedures and control data.
- Data and database management systems.
- SMI laboratory on site at Ity.
- Bureau Veritas analytical laboratory in Abidjan.
- Sample handling and storage facility at Ity Exploration.

## 12.1 Project Drill Hole Database

Prior to 2014, drillhole data was managed on site using bespoke public domain database software. In 2014, SMI introduced AcQuire database software (an industry standard database software) as a replacement to manage all historical and modern drilling data. Since 2014 all historical data has been validated and migrated into the AcQuire database.

Cube was provided with MS Access exports by SMI for each deposit. These were imported into a single MS Access database by Cube which used as the primary data source for the geological interpretations, QAQC analysis and Mineral Resource estimation. Cube has not independently verified any of the data contained in the database except for undertaking the following routine validation checks:

- Check for erroneous hole collar outliers - easting, northing, elevation.
- Discrepancies in maximum hole depths between collar and assay, survey and geology records.
- Survey table drillhole azimuths were checked to be within the expected 0 to 360 degrees range, and that no excessive deviation exists between successive downhole readings.

- Survey table was checked for any positive or near zero drillhole inclinations.
- Assay table was checked for overlaps of assay sample intervals.
- Assay table was checked for negative assays, missing assays or assays outside of expected ranges.
- Visual inspection of the drillholes in Surpac 3D workspace to identify spatial inconsistencies of drillhole traces (unlikely hole deviations).
- Visual inspection in Surpac 3D workspace of drillhole collar position in relation to topography.

No material errors were detected in the collar, assay or survey tables during this checking process.

## 12.2 QA/QC Analysis

A program of quality control (QC) and quality assurance (QA) has been undertaken by SMI to check for contamination, accuracy and precision within the drill sampling and assaying process. The types of check samples that have been introduced into the sample stream include blank samples ('blanks'), certified reference materials ('standards'), and field duplicate samples.

A review of the QA/QC samples is discussed in detail in Section 11 Sample Preparation, Analyses and Security. Overall the QA/QC samples have performed satisfactorily and indicate the sample data is suitable for mineral resource estimation.

## 12.3 Historical Data Validation

From 2013, SMI has undertaken the task of collecting all available historical exploration data and converting it to electronic format. This included substantial validation of the data to confirm consistency and accuracy. Accuracy and completeness of record keeping, logging, surveys and assays have been assessed in the validation procedures.

A flag in the database is used to identify all data that has been validated to date. Where there are multiple data sets of the same type the data sets are ranked on accuracy and a preferred data is recorded for use in geological modelling and mineral resource estimation. Assay procedures have been evaluated for accuracy. Assays have been validated for individual campaigns, laboratories and methods. Validation has included re-assaying of many of the older drillholes. Where data was found to be accurate, the historical data was flagged for use. In the case of ZiaNE, re-assaying of historical drillholes showed the assays to be inaccurate and the entire drillhole data set was re-assayed.

Independent audits and reviews have been undertaken by previous CP's for the Ity project including Coffey and Arethuse since 2013. Validation exercises are extensively documented in SMI reports as well as previous mineral resource estimation reports by Arethuse and Coffey.

## 12.4 Twinned Hole Comparison

The following twinned hole analysis was undertaken by Coffey and documented in Technical Report for the Ity Gold Mine (Body, 2015).

### 12.4.1 Aires

Four diamond drill drillholes were drilled by SMI to verify the results obtained from the initial RC drilling. This drilling enabled a comparison to be made between the two different types of drilling undertaken and the variability of mineralisation over a short distance. A comparison of the original drillhole compared to the twin drillhole is presented in Figure 12.4.1.

The drillhole twinning although limited to four holes confirmed that the mineralisation is fairly well defined by the RC drilling. From the limited data the average grade of the dump decreased in two of the twin diamond holes and increased in mean grade in two of the diamond holes compared to the original RC drilling. The RC average grade for all 196 samples within the four mineralised zones was 1.16 g/t Au compared to 0.94 g/t Au for the diamond drilling, a difference of 23%. The difference in grade is attributed to the high nugget nature of gold mineralisation and one outlier of 52.7 g/t Au from 0 m to 1 m in S831. This was not replicated in the twin hole S1372. Capping the outlier to 15.6 g/t Au reduces the average grade is reduced to 0.96 g/t Au, being a 2% difference between the original drillhole and the twinned drillhole. A scatter plot of AirCore drilling compared to diamond drilling is shown in Figure 12.4.2 which indicates a reasonable correlation between the two drilling methods.

**Figure 12.4.1 Twin Hole Drilling Comparison - Aires**

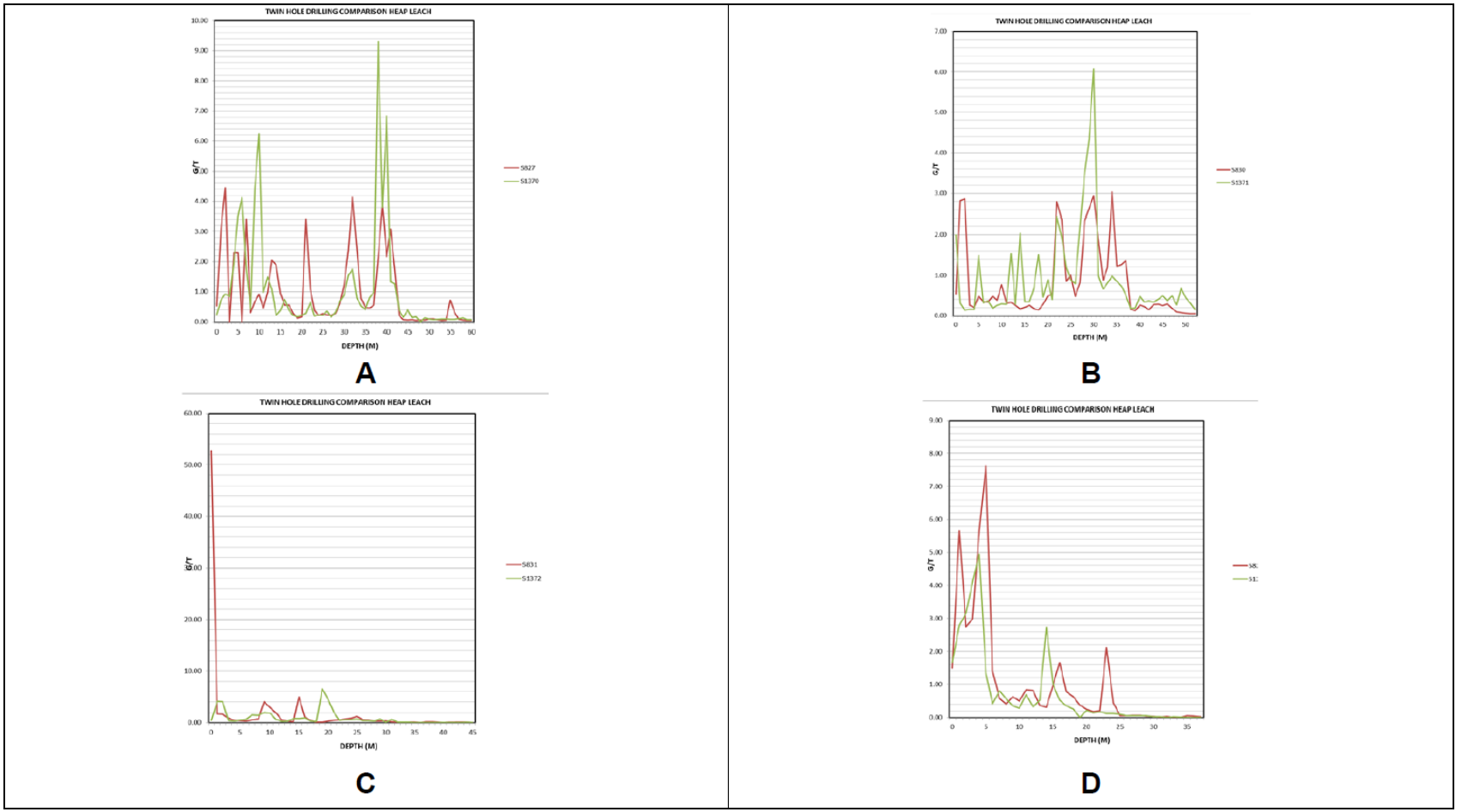
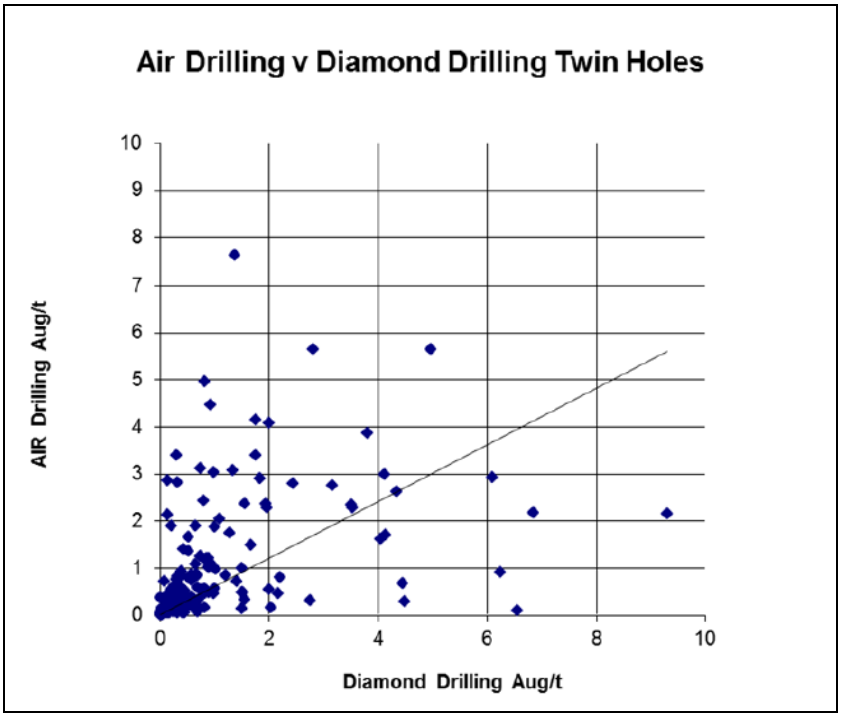


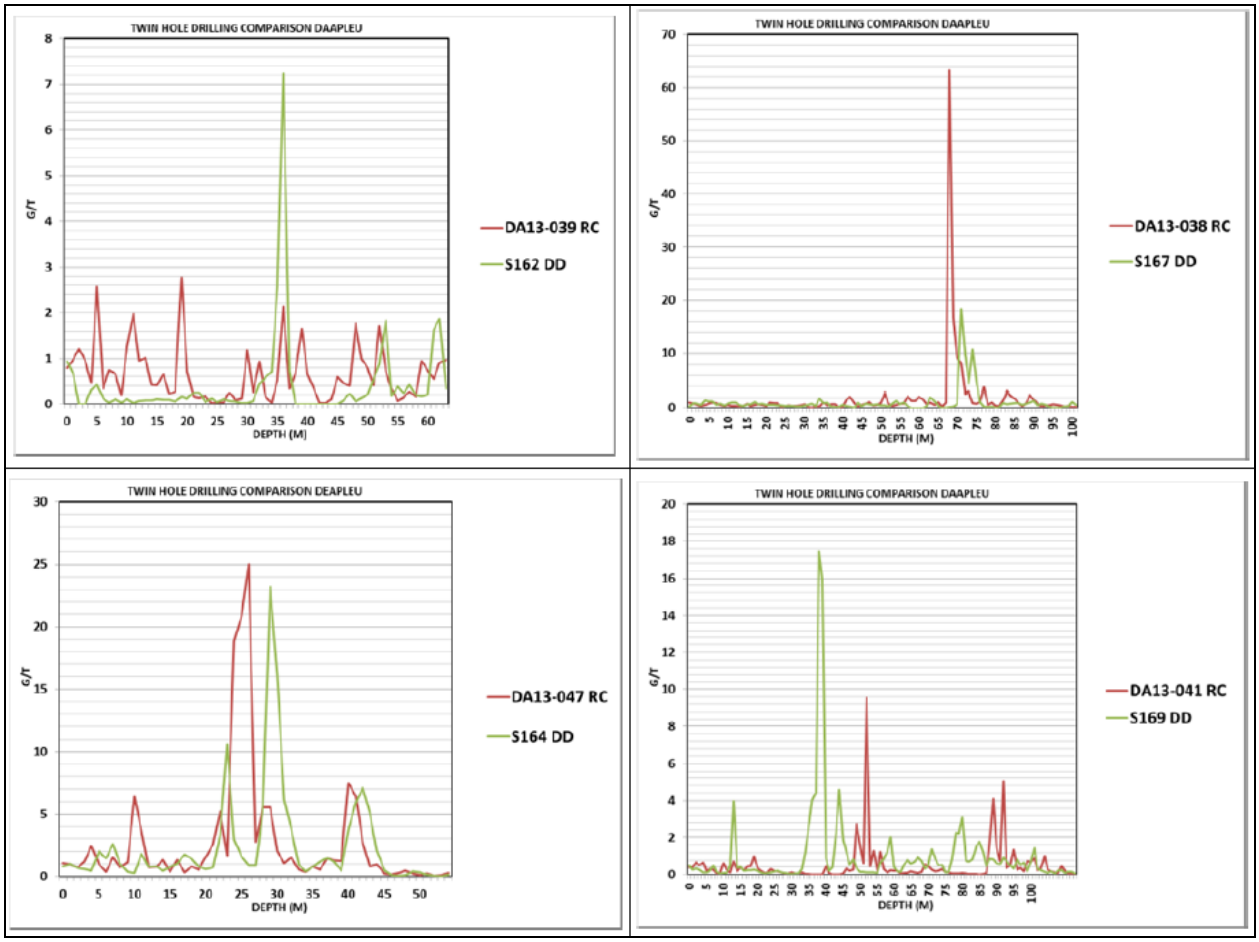
**Figure 12.4.2 Air Core vs Diamond Drilling Comparison - Aires**




### 12.4.2 Daapleu

RC drillholes were drilled by SMI in 2013 to verify the results obtained from the initial diamond drilling undertaken in 1997. This drilling enabled a comparison to be made between the two different types of drilling undertaken and the variability of mineralisation over a short distance. A comparison of the original drillholes and the twin drillholes is presented in Figure 12.4.3.

**Figure 12.4.3 Twin Hole Drilling Comparison - Daapleu**




From the data there is a moderate to good correlation between the original and current drilling for three of the four twin drillholes with the fourth (DA13-039 and S162) displaying a moderate to poor correlation. The average grade of the deposit was lower in three of the four original drillholes compared to the current RC drilling. One sample of 64.43 g/t Au in hole DA13-038 was not replicated in the original hole S167, although high grades in excess of 10 g/t Au were obtained in close proximity to the outlier. The difference in grade between the individual twin samples is attributed to the high nugget nature of the gold mineralisation, the drillholes not being exact twins and the mineralisation being slightly off set between drillholes, e.g. DA13-047 and S164.

### 12.4.3 ZiaNE

No twin drillhole comparison is available for ZiaNE. Data used for the estimation is from the 2014 and 2015 drilling campaigns. Assay data from the 2008 drilling campaign has been assessed and deemed unreliable. It has been replaced with the results of the re-assay undertaken in 2014, of the relevant core.

### 12.4.4 Verse Ouest

No drillholes were twinned at Verse Ouest.

### 12.4.5 Mont Ity

Seven diamond drillholes were drilled to confirm the results obtained from the RC samples. Overall, the drillholes compare favourably. Due to the nuggety nature of gold and the separation between drillholes of up to 10 m, some variation is expected between holes but the width of the mineralised zone and major trends in the mineralisation are expected to be similar (Figure 12.4.4).

### 12.4.6 Ity Flat

The 2015 drilling campaign included seven diamond drillholes drilled to confirm RC assay results. The results show the high nuggety nature of the gold at Ity Flat. The results are not good but acceptable for mineral resource studies (Figure 12.4.5). Further follow-up is needed.

**Figure 12.4.4 Twin Hole Drilling Comparison - Mont Ity**

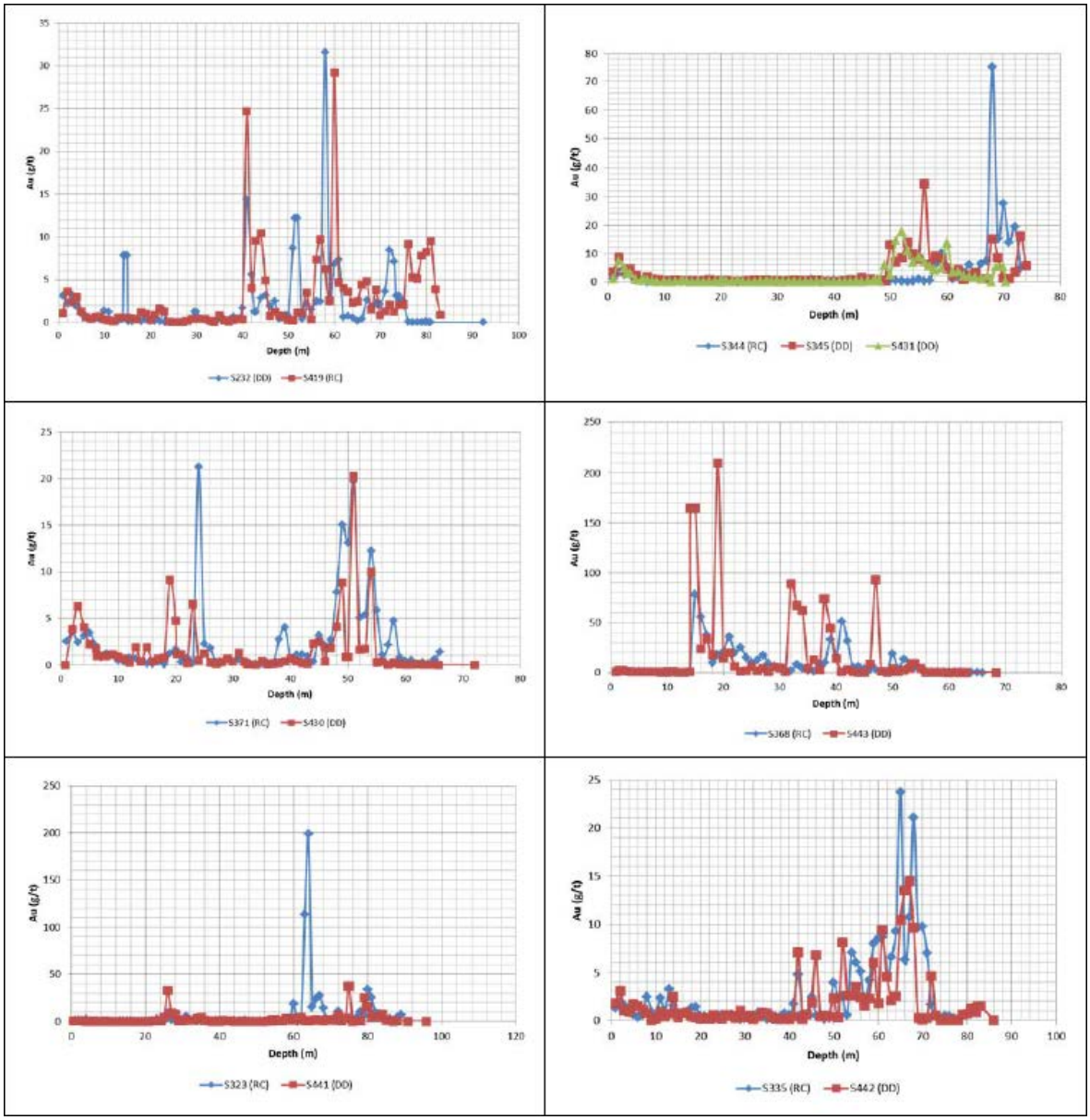
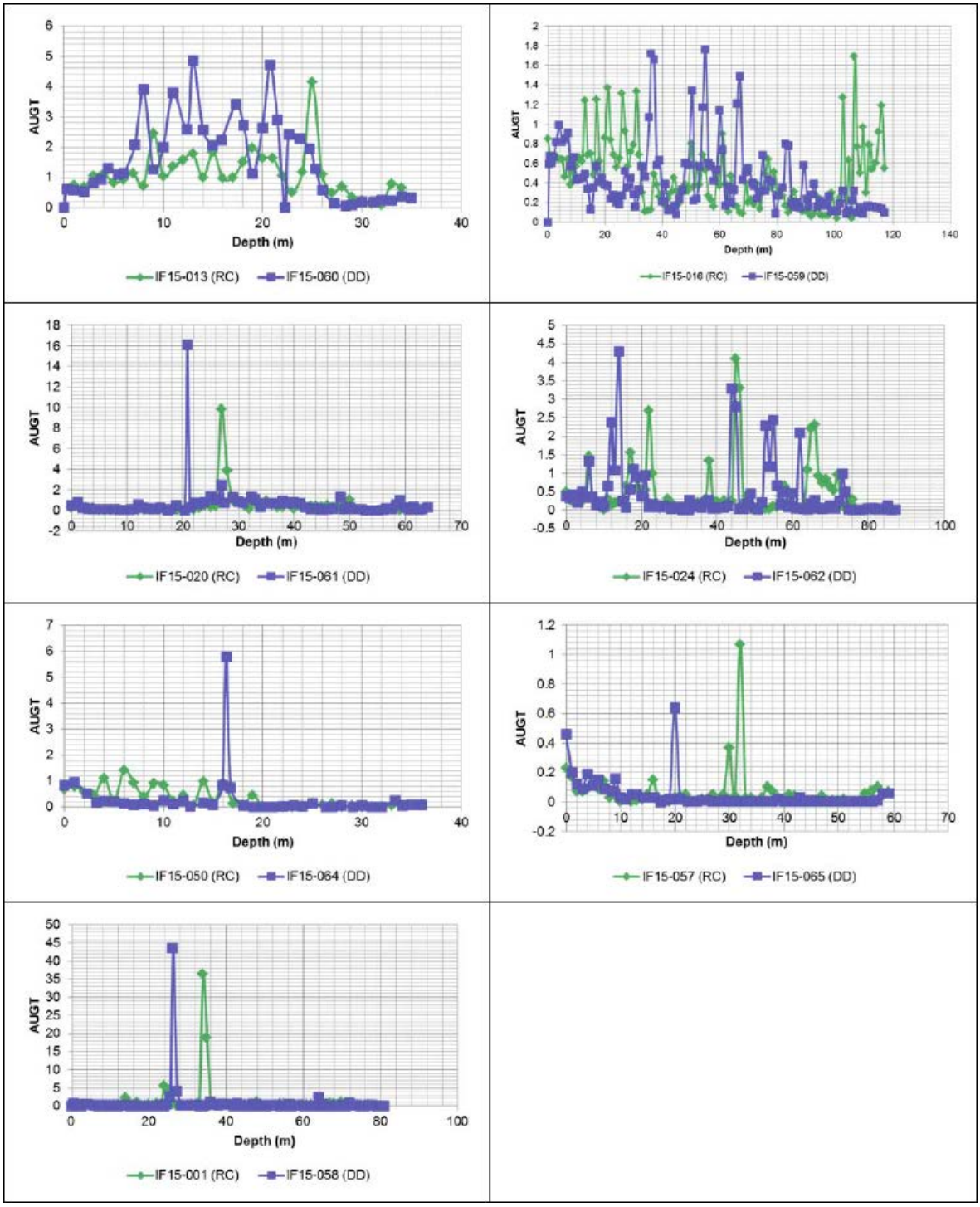


**Figure 12.4.5 Twin Hole Drilling Comparison - Ity Flat**




A program of quality control (QC) and quality assurance (QA) has been undertaken by SMI to check for contamination, accuracy.

## 12.5 Independent Verification Samples

The exercise of taking independent samples was not undertaken by the author. The Ity Project has a long history of gold production with active mining at the project area since the early 1990's. Given the history of successful mining and the twinning program completed previously to validate the historical drilling, the author is satisfied with the validity of the drillhole data.

## 12.6 Authors Statement

The author has visited site and independently reviewed and assessed all of the available quality control sample data relating to the RC and diamond drilling completed by SMI at the Ity Project.

Overall, the sample control data has performed well and indicates the sample assay data to be of a high standard In addition, the twinned drillholes shows that the mineralised zones are predictable and that the differences between the twinned drillholes represent the inherit variability on the deposits.

The author considers the drillhole logging and sampling data for the deposits referred to in this technical report are appropriate for the reporting of exploration results and use in Mineral Resource estimation.

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# 13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction

Following completion of a draft study in 2015 it was determined that further testwork was required to address some key process data across all potential ore sources. The following provides a summary of the testwork undertaken by ALS between June and October 2016:

- Comminution tests performed on 20 samples representing all facies types across all pits and intended to produce a range of comminution data (SMC values, Bond Ball Mill Work indices, and Abrasion indices) for input into equipment sizing.
- Variability tests performed on 40 samples representing all defined facies types across all pits, under standard cyanide leaching conditions and at a fixed particle size. The intention of these tests was to determine the relative Au recovery from the various pits and facies types.
- Diagnostic tests performed on 10 variability leach residue samples which realised low (<85%) Au recovery. The purpose of these tests was to establish the reason(s) for the poor Au recovery.
- Optimisation tests performed on six composites, each representing the different pits, to establish optimum leaching conditions (grind size, pulp density, and cyanide concentration) and to establish CIL and cyanide detoxification parameters.
- Refractory test programme on one composite made up of unoxidised material from the Daapleu pit only. This was a retrospective programme aimed at determining a suitable flowsheet dedicated to the treatment of the refractive Daapleu ore. Testwork is currently ongoing.
- Details of testwork methodology and individual test results are reported in the ALS testwork which is available on request.

## 13.2 Sample Selection Methodology

A total of 3,320 kg of drill core from six different pits was delivered to ALS, Perth, on 2 June 2016. The drill core was contained in 412 individual sample bags within 37 drums. Each sample bag typically contained a 5 m interval of ¼ drill core from the selected drill hole. From these drill core intervals, a series of samples was selected for the programme of comminution and metallurgical testwork. The sample selection was made to accommodate all facies types, different degrees of oxidation, and Au grade variability.

Having arrived at ALS, the sample bags were removed from their respective drums and placed on the ground for inspection by process engineers from Lycopodium and OMC, and geologists from Cube Consulting (Cube). During the inspection, the Cube geologists confirmed that there had been some misallocation of facies type to a number of the samples due to the different modelling techniques used by Coffey and Cube – Coffey had based their facies allocation on an indicator estimation method, while Cube have adopted the more traditional, and precise, sectional interpretation of facies type with hard wire frame boundaries. The anomalous facies classifications were subsequently rectified in line with the revised Cube interpretation.

## 13.3 Comminution Testwork

The purpose of the comminution testwork programme is to gain a thorough knowledge of the ore physical characteristics to facilitate a crushing and milling circuit design appropriate for the plant throughput and feed type (Mine Plan). Based on the proposed comminution circuit, the equipment was sized and costed to provide input to a capital cost estimate while simultaneously yielding critical operating data (power, grinding media consumption, etc.) for input to the operating cost estimate.

Each suite of 20 comminution tests comprised the following:

- SMC tests (@ -31.5+26.5 mm size fraction).
- Bond Abrasion Index (Ai) determination.
- Bond Ball Mill Work Index (BWi) determination (@ 106 µm closing screen to yield $P_{80}$ of 75 µm).
- Modified Bond Ball Mill Work Index determination on all 'soft' material (pre-screen and remove all -106 µm material).
- Levin Test on all samples unsuitable for SMC and BWi tests.

The work flowsheet used by ALS is available in the ALS report.

### 13.3.1 Sample Selection

Comminution testwork sample selection was based on the relative proportions of each individual facies within the various pits and the overall contribution to the life of mine (LoM) ore tonnage. (Derivation of the facies and their overall tonnage contribution in the various mine pits is available on request.)

This facilitated the development of a sample matrix, with a total of 20 comminution samples being selected for the testwork programme. The sample matrix is shown in Table 13.3.1.

**Table 13.3.1 Comminution Sample Matrix**

| Mine Pit | Mass Contribution to Mine Plan (%) | Facies | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | 101 | 201 | 211 | 301 | 401 | 501 | 601 | 811 | 901 |
| | | U / PO | O / PO | U | O | U | PO | U / PO | U | U |
| Daapleu | 49.5 | 2 | 1 | | | | | 5 | 1 | |
| Gbeitouo | 5.3 | 1 | | | | | | | | |
| Ity Flat | 7.5 | | | 1 | | | 1 | | | 1 |
| Mont Ity | 11.1 | | 1 | 1 | | | 1 | | | |
| Walter | 5.5 | | 1 | | | | | | | |
| Zia NE | 21.1 | | 2 | | 1 | | | | | |
| **Total** | **100** | **3** | **5** | **2** | **1** | | **2** | **5** | **1** | **1** |

**Notes: O, PO, and U refer to oxidised, partially oxidised, and unoxidised ore.**

By way of an example, facies 601 contributes 58% of the total mass of the Daapleu deposit, while Daapleu material itself contributes nearly 50% to the overall mine plan tonnage. Thus facies type 601 is responsible for more than a quarter of the total mine tonnage and thus makes up five of the 20 total comminution samples.

Having established the number of samples and the mass of sample required for each test, appropriate samples were then selected from the available material to accommodate the range of facies, oxidation state, and Au grade, while attempting to achieve spatial coverage across each pit.

### 13.3.2 Test Sample Sizing

Of the 20 samples, eight were considered too fine to perform a standard Bond Ball Mill Work Index test, viz sample numbers 7, 10, 11, 14, 16, 17, 18, and 19. (As-received samples are generally considered too fine when the $F_{80}$ comprises >20% of the total sample mass relative to the required $P_{80}$). As an alternative to the standard Bond Ball Mill Work Index test, therefore, modified Bond and Levin tests were performed on these eight samples.

### 13.3.3 SMC Tests

A series of SMC tests were performed on 11 of the 20 comminution samples. The remaining nine samples were not sufficiently competent to perform the SMC tests – these were subjected to Modified Bond Work Index and Levin tests. A summary of the parameters derived from the SMC tests is provided in Table 13.3.2.

**Table 13.3.2 SMC Test Results**

| Test Sample | DWi kWh/m³ | DWi % | Mia kWh/t | Mih kWh/t | Mic kWh/t | A | b | S.G. | ta | SCSE kWh/t |
|---|---|---|---|---|---|---|---|---|---|---|
| # 1 | 4.53 | 24.0 | 14.4 | 9.8 | 5.1 | 65.8 | 0.90 | 2.67 | 0.57 | 8.29 |
| # 2 | 4.98 | 29.0 | 15.2 | 10.6 | 5.5 | 64.4 | 0.85 | 2.74 | 0.52 | 8.65 |
| # 3 | 6.67 | 52.0 | 18.5 | 13.7 | 7.1 | 58.3 | 0.73 | 2.83 | 0.39 | 9.86 |
| # 4 | 0.92 | 1.0 | 4.1 | 2.0 | 1.0 | 79.7 | 3.60 | 2.64 | 2.81 | 5.18 |
| # 5 | 4.08 | 19.0 | 13.1 | 8.8 | 4.6 | 65.3 | 1.01 | 2.69 | 0.63 | 7.96 |
| # 6 | 4.39 | 23.0 | 13.9 | 9.4 | 4.9 | 63.2 | 0.97 | 2.70 | 0.59 | 8.21 |
| # 8 | 2.75 | 9.0 | 9.5 | 5.9 | 3.0 | 67.2 | 1.47 | 2.71 | 0.94 | 6.88 |
| # 9 | 6.45 | 49.0 | 18.5 | 13.6 | 7.0 | 61.6 | 0.69 | 2.76 | 0.40 | 9.71 |
| # 12 | 3.19 | 12.0 | 9.8 | 6.3 | 3.2 | 54.7 | 1.70 | 2.96 | 0.81 | 7.22 |
| # 13 | 5.82 | 40.0 | 15.3 | 11.0 | 5.7 | 61.4 | 0.86 | 3.07 | 0.45 | 9.35 |
| # 15 | 8.64 | 76.0 | 23.2 | 18.1 | 9.3 | 66.0 | 0.49 | 2.77 | 0.30 | 11.10 |

### 13.3.4 Standard Bond Ball Mill Work Index Tests

A series of standard Bond ball mill work index (BWi) tests were performed on 10 of the 20 comminution samples with a closing screen size of 106 µm (to achieve the target $P_{80}$ of 75 µm). Of the 20 comminution samples, four contained an excessive fines fraction (>30% of target grind size) and these were subjected to Levin tests only. A further six of the 20 samples also contained a high proportion of fines and were subjected to a modified BWi test. A summary of results is provided in the Table 13.3.3.

**Table 13.3.3 Standard BWi Test Results**

| Test Sample | Pit | Facies | Oxidation | $F_{80}$ (µm) | $P_{80}$ (µm) | GRP (g/rev) | Aperture Pi (µm) | BBWi (kWh/t) |
|---|---|---|---|---|---|---|---|---|
| #1 | Daapleu | 601 | U | 2,625 | 85 | 3.303 | 106 | 7.07 |
| #2 | Daapleu | 811 | U | 2,310 | 79 | 2.148 | 106 | 9.80 |
| #3 | Daapleu | 101 | U | 2,330 | 78 | 1.078 | 106 | 17.0 |
| #5 | Daapleu | 601 | U | 2,256 | 80 | 5.050 | 106 | 4.91 |
| #6 | Daapleu | 601 | U | 2,644 | 81 | 2.533 | 106 | 8.53 |
| #8 | Daapleu | 601 | U | 2,029 | 91 | 3.495 | 106 | 7.30 |
| #9 | Daapleu | 101 | U | 2,577 | 81 | 1.619 | 106 | 12.3 |
| #13 | Ity Flat | 901 | U | 2,334 | 84 | 2.309 | 106 | 9.53 |
| #15 | Mont Ity | 401 | U | 2,367 | 76 | 1.554 | 106 | 12.4 |
| #20 | Zia NE | 301 | O | 2,494 | 77 | 1.269 | 106 | 14.7 |

The highest BWi recorded from the Standard BWi tests was 17 kWh/t (Daapleu pit, Facies 101, unoxidised). Although the Daaplite (facies type 601, unoxidised) demonstrated reasonable competency and was able to undergo the SMC testwork programme, it generated the four lowest Bond Ball Mill Work indices with an average 6.95 kWh/t. In contrast, unoxidised facies type 101 from the Daapleu deposit recorded the highest BWi of 17.0 kWh/t with an average 14.65 kWh/t (Facies 101 contributes approximately 15% of the tonnage of the overall Ity mine reserve).

### 13.3.5 Modified Bond Ball Mill Work Index Tests

The modified BWi test differs from the standard BWi test in that the samples were pre-screened at 106 µm to remove excessive fines prior to the test. A summary of results from the six modified BWi tests is provided in Table 13.3.4.

**Table 13.3.4 Modified BWi Test Results**

| Test Sample | Pit | Facies | Oxidation | $F_{80}$ (µm) | $P_{80}$ (µm) | GRP (g/rev) | Aperture Pi (µm) | BWi (kWh/t) |
|---|---|---|---|---|---|---|---|---|
| #4 | Daapleu | 601 | PO / U | 1,742 | 90 | 3.894 | 106 | 6.7 |
| #7 | Daapleu | 201 | O | 1,139 | 95 | 1.288 | 106 | 18.7 |
| #11 | Ity Flat | 211 | O | 227 | 86 | 1.537 | 106 | 28.5 |
| #14 | Mont Ity | 201 | O | 787 | 92 | 2.014 | 106 | 13.8 |
| #18 | Zia NE | 201 | O | 484 | 90 | 0.899 | 106 | 30.7 |
| #19 | Zia NE | 201 | O | 529 | 90 | 2.135 | 106 | 14.5 |

Despite high BWi results being reported for samples #11 and #18 (28.5 and 30.7 kWh/t respectively), these were treated with scepticism. When considered in conjunction with the Levin test data for these particular samples it is evident that there is little size reduction with increasing power input. This would typically suggest the presence of platelets (micaceous material) and/or fibrous material which would tend to skew the particle size distribution and thus lead to an over-estimation of the BWi. Subsequent inspection of the Levin test samples indeed indicated the presence of a significant fraction of flaky material. The high BWi values reported for samples #11 and #18 are, therefore, considered invalid.

### 13.3.6 Levin Tests

Of the 20 comminution test samples, four contained an excessive fines fraction (>30% of the target grind size) and were subjected to a Levin test only. A further four Levin tests were performed on samples which were also subjected to a modified BBWi test. A summary of the results from the Levin tests is provided in Table 13.3.5.

As Levin tests were carried out on material considered too fine for a standard Bond Ball Mill Work Index tests, the relatively low specific energy input values obtained from the Levin tests (10.0 kWh/t or less) were not unexpected.

**Table 13.3.5 Levin Test Results**

| Test Sample | Pit | Facies | Oxidation | Particle Size Distribution Determination, $P_{80}$ µm | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Feed | Energy Input, kWh/t | | | | | | |
| | | | | | 0.25 | 0.5 | 1.0 | 1.5 | 2.0 | 3.0 | 5.0 | $P_{80}$ 75 µm |
| #7 | Daapleu | 201 | O | 172 | | | 92 | 85 | 79 | 73 | | 3.0 |
| #10 | Gbeitouo | 101 | PO | 593 | | | 150 | | 96 | | | 10.0 |
| #12 | Ity Flat | 501 | PO | 1293 | | | 452 | | | | 97 | 10.0 |
| #14 | Mont Ity | 401 | U | 190 | 105 | 62 | 58 | | | | | 0.46 |
| #16 | Mont Ity | 211 | O / PO / U | 1163 | | | 479 | | | | 101 | 10.0 |
| #17 | Walter | 201 | O | 199 | | | 82 | 76 | | | | 1.5 |
| #18 | Zia NE | 201 | O | 264 | | | 131 | | 124 | 123 | | 8.9 |
| #19 | Zia NE | 201 | O | 127 | 81 | 48 | 46 | | | | | 0.26 |

### 13.3.7 Bond Abrasion Index

A series of standard Bond Abrasion Index tests was performed on the 20 comminution samples. A summary of the Bond Abrasion Index results is provided in Table 13.3.6.

**Table 13.3.6 Bond Abrasion Index Results**

| Sample Test | Pit | Facies | Oxidation | Paddle Mass before test (g) | Paddle Mass after test (g) | Abrasion Index (Ai) | Comments |
|---|---|---|---|---|---|---|---|
| #1 | Daapleu | 601 | U | 81.191 | 80.999 | 0.1915 | Moderately Abrasive |
| #2 | Daapleu | 811 | U | 81.160 | 80.858 | 0.3017 | Abrasive |
| #3 | Daapleu | 101 | U | 82.107 | 81.972 | 0.1359 | Moderately Abrasive |
| #4 | Daapleu | 601 | PO / U | 84.024 | 84.004 | 0.0199 | Moderately Abrasive |
| #5 | Daapleu | 601 | U | 82.533 | 82.371 | 0.1619 | Moderately Abrasive |
| #6 | Daapleu | 601 | U | 82.149 | 81.861 | 0.2874 | Abrasive |
| #7 | Daapleu | 201 | O | - | - | n/a | Non Abrasive |
| #8 | Daapleu | 601 | U | 81.229 | 81.185 | 0.0444 | Moderately Abrasive |
| #9 | Daapleu | 101 | U | 76.201 | 76.123 | 0.0774 | Moderately Abrasive |
| #10 | Gbeitouo | 101 | PO | - | - | n/a | Non Abrasive |
| #11 | Ity Flat | 211 | O | - | - | n/a | Non Abrasive |
| #12 | Ity Flat | 501 | PO | 81.021 | 80.793 | 0.2283 | Abrasive |
| #13 | Ity Flat | 901 | U | 84.006 | 83.932 | 0.0744 | Moderately Abrasive |
| #14 | Mont Ity | 201 | O | - | - | n/a | Non Abrasive |
| #15 | Mont Ity | 401 | U | 81.748 | 81.717 | 0.0306 | Moderately Abrasive |
| #16 | Mont Ity | 211 | O / PO / U | - | - | n/a | Non Abrasive |
| #17 | Walter | 201 | O | - | - | n/a | Non Abrasive |
| #18 | Zia NE | 201 | O | - | - | n/a | Non Abrasive |
| #19 | Zia NE | 201 | O | - | - | n/a | Non Abrasive |
| #20 | Zia NE | 301 | O | - | - | n/a | Non Abrasive |

The criteria used for determining the relative abrasiveness of the material is provided in Table 13.3.7.

**Table 13.3.7 Bond Abrasion Index – Relative Abrasiveness**

| Material Property | Bond Ai |
|---|---|
| Non Abrasive | <0.01 |
| Moderately Abrasive | 0.01 - 0.20 |
| Abrasive | 0.20 - 0.40 |
| Highly Abrasive | 0.40 - 0.60 |
| Extremely Abrasive | >0.60 |

None of the material tested in the Comminution testwork programme was classified as 'Highly Abrasive' or 'Extremely Abrasive'.

### 13.3.8 Comminution Testwork – Results Interpretation

Results indicate the following:

- There is marked variability in the ore which generally depends on oxidation state.
- Unoxidised samples show low to moderate competency and moderate abrasion index.
- Partially oxidised samples show low competence and low abrasion index.
- Oxidised samples were generally soft with very low abrasion index although the Levin test was difficult to interpret due to the presence of some soft micaceous minerals.

This data was submitted to Orway Mineral Consultants for interpretation and this is discussed in Section 17.

## 13.4 Metallurgical Variability Testwork

The objective of the metallurgical variability tests was to compare the gold and silver extractions for each sample under the same set of standard cyanidation leach conditions for each of the various facies types and across the different pits. The results of this variability programme would then be used as a basis for the subsequent Diagnostic and Optimisation testwork programmes.

### 13.4.1 Sample Selection

A series of variability leach tests was conducted on selected samples from the ¼ core delivered from Ity to ALS. Each test comprised a standard cyanidation leach requiring approximately 5 kg of sample. The sample selection was based on the relative proportions of each individual facies within the various pits and the overall contribution to the life of mine (LoM) ore tonnage.

This facilitated the development of a sample matrix, with a total of 40 variability samples being selected for the testwork programme. The sample matrix is shown in Table 13.4.1.

**Table 13.4.1 Variability Sample Matrix**

| Mine Pit | Mass Contribution to Mine Plan (%) | Facies | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | 101 | 201 | 211 | 301 | 401 | 501 | 601 | 811 | 901 |
| | | U / PO | O / PO | U | O | U | PO | U / PO | U | U |
| Daapleu | 49.5 | 3 | 2 | | | | | 12 | 1 | |
| Gbeitouo | 5.3 | 2 | 1 | 1 | | | | | | |
| Ity Flat | 7.5 | | 2 | 1 | | | 1 | | | 2 |
| Mont Ity | 11.1 | | 1 | 1 | | 2 | | | | |
| Walter | 5.5 | | 2 | 1 | | | | | | |
| Zia NE | 21.1 | | 3 | | 2 | | | | | |
| **Total** | **100** | **5** | **11** | **4** | **2** | **2** | **1** | **12** | **1** | **2** |

### 13.4.2 Variability Testwork Programme

Each suite of 40 variability tests comprised the following:

- Sample head assay determination.
- Grind to $P_{80}$ of 75 µm.
- Standard 36 hour bottle roll cyanide leach test at 40% w/w solids pulp density.

The criteria used for the variability leach conditions are based on typical industry values.

### 13.4.3 Variability Testwork Results

A summary of the head assay results is provided in Table 13.4.2. There was generally good agreement between the triplicate gold assays for each of the variability samples suggesting a lack of coarse gold particles.

Results of the leach variability tests are presented in Table 13.4.3. Gold extraction varied significantly across the samples, with 12 of the 40 variability leach tests returning gold extractions >95% while nine of the tests yielded gold extractions <80%, of which eight were conducted on unoxidised Daapleu ore samples (Facies Type 601 and 101). The majority of these low recovery Daapleu samples formed the basis of the subsequent diagnostic test programme to establish the reason for the poor gold recovery.

**Table 13.4.2 Head Assay Results**

| Variability Sample | Pit | Facies | Oxidation | Au (ppm)1 | Au (ppm)2 | Au (ppm)3 | Au (ppm)Av | Ag (ppm) | As (ppm) | Cu (ppm) |
|---|---|---|---|---|---|---|---|---|---|---|
| #1 | Daapleu | 811 | U | 0.84 | 0.86 | 0.91 | 0.87 | 6 | 110 | 36 |
| #2 | Daapleu | 601 | U | 2.14 | 1.79 | 1.93 | 1.95 | 20 | 630 | 22 |
| #3 | Daapleu | 601 | U | 0.89 | 0.62 | 0.71 | 0.74 | 4 | 1,390 | 12 |
| #4 | Daapleu | 101 | U | 1.92 | 1.28 | 2.47 | 1.89 | <2 | 410 | 66 |
| #5 | Daapleu | 601 | U | 0.75 | 0.93 | 0.57 | 0.75 | 8 | 1,080 | 16 |
| #6 | Daapleu | 601 | U | 0.49 | 0.54 | 0.48 | 0.50 | <2 | 250 | 4 |
| #7 | Daapleu | 601 | O | 1.35 | 1.82 | 0.60 | 1.26 | <2 | 810 | 22 |
| #8 | Daapleu | 601 | U | 2.06 | 0.59 | 0.61 | 1.09 | <2 | 220 | 16 |
| #9 | Daapleu | 101 | U | 2.95 | 2.39 | 2.51 | 2.62 | 4 | 250 | 36 |
| #10 | Daapleu | 201 | O | 0.87 | 0.77 | 0.63 | 0.76 | <2 | 1,190 | 102 |
| #11 | Daapleu | 601 | U | 20.7 | 19.7 | 19.8 | 20.1 | 36 | 23,000 | 82 |
| #12 | Daapleu | 201 | O | 1.06 | 1.00 | 0.89 | 0.98 | <2 | 860 | 18 |
| #13 | Daapleu | 601 | U | 1.28 | 1.43 | 1.29 | 1.33 | 4 | 730 | 14 |
| #14 | Daapleu | 601 | U | 1.48 | 2.13 | 2.21 | 1.94 | <2 | 1,210 | 8 |
| #15 | Daapleu | 601 | O / PO | 6.35 | 5.98 | 6.03 | 6.12 | 14 | 2,090 | 38 |
| #16 | Daapleu | 601 | U | 2.10 | 1.78 | 2.40 | 2.09 | <2 | 190 | 28 |
| #17 | Daapleu | 601 | U | 1.86 | 1.20 | 0.91 | 1.32 | 18 | 320 | 38 |
| #18 | Daapleu | 101 | U | 1.50 | 1.42 | 1.56 | 1.49 | 6 | 2,520 | 42 |
| #19 | Gbeitouo | 101 | U | 0.62 | 0.40 | 0.50 | 0.51 | <2 | 890 | 76 |
| #20 | Gbeitouo | 201 | O | 0.64 | 0.71 | 0.76 | 0.70 | <2 | 3,710 | 100 |
| #21 | Gbeitouo | 211 | O / PO | 0.29 | 0.33 | 0.34 | 0.32 | 6 | 60 | 128 |
| #22 | Gbeitouo | 101 | PO | 0.58 | 0.47 | 0.37 | 0.47 | <2 | 520 | 68 |
| #23 | Ity Flat | 501 | PO | 0.92 | 0.99 | 1.01 | 0.97 | 4 | <10 | 1,770 |
| #24 | Ity Flat | 901 | U | 7.38 | 7.53 | 6.14 | 7.02 | 2 | <10 | 3,660 |
| #25 | Ity Flat | 211 | U | 7.55 | 8.47 | 7.54 | 7.85 | 8 | <10 | 11,700 |
| #26 | Ity Flat | 201 | O | 10.9 | 11.0 | 10.7 | 10.9 | 10 | <10 | 1,720 |
| #27 | Ity Flat | 901 | PO | 0.10 | 0.10 | 0.09 | 0.10 | <2 | 20 | 80 |
| #28 | Ity Flat | 201 | O | 3.02 | 3.18 | 3.22 | 3.14 | <2 | <10 | 726 |
| #29 | Mont Ity | 401 | U | 16.7 | 15.9 | 16.3 | 16.3 | 8 | <10 | 246 |
| #30 | Mont Ity | 401 | U | 7.48 | 7.47 | 6.95 | 7.30 | 12 | <10 | 4,720 |
| #31 | Mont Ity | 201 | O | 3.48 | 5.11 | 4.40 | 4.33 | 14 | 30 | 2,005 |
| #32 | Mont Ity | 211 | O | 1.62 | 1.56 | 2.02 | 1.73 | <2 | <10 | 1,210 |
| #33 | Walter | 201 | O | 0.31 | 0.30 | 0.32 | 0.31 | <2 | <10 | 670 |
| #34 | Walter | 211 | U | 2.54 | 2.53 | 3.34 | 2.80 | <2 | 10 | 138 |
| #35 | Walter | 201 | O / PO | 1.29 | 1.46 | 1.30 | 1.35 | <2 | <10 | 2,775 |
| #36 | Zia NE | 201 | O | 2.42 | 2.35 | 2.43 | 2.40 | <2 | 10 | 2,065 |
| #37 | Zia NE | 201 | O | 9.75 | 11.3 | 8.02 | 9.69 | <2 | <10 | 912 |
| #38 | Zia NE | 301 | O | 1.24 | 1.21 | 1.33 | 1.26 | 2 | 20 | 170 |
| #39 | Zia NE | 201 | O | 1.04 | 0.92 | 0.99 | 0.98 | <2 | <10 | 426 |
| #40 | Zia NE | 301 | O | 2.44 | 2.62 | 2.81 | 2.62 | 2 | 20 | 526 |

**Table 13.4.3 Variability Leach Test Results**

| ALS Test No. | Variability Sample | Pit | Facies | Oxidation | Head Assay (Au ppm) | Calc. Head (Au ppm) | Head Assay (As ppm) | Calc. Head (As ppm) | Au | | Ag | | As | | Cu | | Consumption (g/t) | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | Tail (ppm) | Rec % | Tail (ppm) | Rec % | Tail (ppm) | Rec % | Tail (ppm) | Rec % | NaCN | Lime |
| CT1486 | #1 | Daapleu | 811 | U | 0.87 | 1.01 | 110 | 0.1 | 0.15 | 85.1 | 1.0 | 86.5 | 110 | 1.71 | 358 | 2.60 | 0.18 | 0.28 |
| CT1487 | #2 | Daapleu | 601 | U | 1.95 | 2.49 | 630 | 793 | 0.43 | 82.8 | 12.0 | 54.0 | 790 | 0.38 | 42 | 9.27 | 0.25 | 0.21 |
| CT1488 | #3 | Daapleu | 601 | U | 0.74 | 0.95 | 1,390 | 1,395 | 0.52 | 45.1 | 1.0 | 70.1 | 1,390 | 0.34 | 16 | 12.2 | 0.25 | 0.21 |
| CT1489 | #4 | Daapleu | 101 | U | 1.89 | 1.64 | 410 | 423 | 0.09 | 94.5 | 1.0 | 52.7 | 420 | 0.71 | 72 | 10.5 | 0.43 | 0.26 |
| CT1490 | #5 | Daapleu | 601 | U | 0.75 | 0.97 | 1,080 | 1,045 | 0.49 | 49.4 | 1.0 | 74.1 | 1,040 | 0.47 | 20 | 14.7 | 0.29 | 0.20 |
| CT1491 | #6 | Daapleu | 601 | U | 0.50 | 0.50 | 250 | 281 | 0.15 | 69.7 | 1.0 | 33.4 | 280 | 0.36 | 8.0 | 10.3 | 0.25 | 0.24 |
| CT1492 | #7 | Daapleu | 601 | O | 1.26 | 1.92 | 810 | 725 | 0.10 | 94.8 | 1.0 | 64.2 | 720 | 0.68 | 16 | 27.2 | 0.39 | 0.47 |
| CT1493 | #8 | Daapleu | 601 | U | 1.09 | 0.72 | 220 | 251 | 0.09 | 87.5 | 1.0 | 39.6 | 250 | 0.52 | 8.0 | 10.3 | 0.25 | 0.16 |
| CT1494 | #9 | Daapleu | 101 | U | 2.62 | 2.84 | 250 | 251 | 0.29 | 89.8 | 1.0 | 78.3 | 250 | 0.34 | 36 | 10.3 | 0.39 | 0.21 |
| CT1495 | #10 | Daapleu | 201 | O | 0.76 | 0.80 | 1,190 | 1,070 | 0.07 | 91.3 | 1.0 | 30.9 | 1,070 | 0.04 | 96 | 0.25 | 0.62 | 1.90 |
| CT1496 | #11 | Daapleu | 601 | U | 20.1 | 21.0 | 23,000 | 20,219 | 11.2 | 46.6 | 18.0 | 52.6 | 20,150 | 0.34 | 55 | 30.4 | 0.60 | 0.32 |
| CT1497 | #12 | Daapleu | 201 | O | 0.98 | 1.35 | 860 | 820 | 0.08 | 94.1 | 1.0 | 30.0 | 820 | 0.06 | 15 | 3.47 | 0.39 | 0.84 |
| CT1498 | #13 | Daapleu | 601 | U | 1.33 | 1.77 | 730 | 694 | 0.78 | 56.0 | 4.0 | 57.2 | 690 | 0.58 | 25 | 13.3 | 0.29 | 0.21 |
| CT1499 | #14 | Daapleu | 601 | U | 1.94 | 1.96 | 1,210 | 1,165 | 1.09 | 44.4 | 1.0 | 57.7 | 1,160 | 0.44 | 10 | 8.30 | 0.25 | 0.21 |
| CT1500 | #15 | Daapleu | 601 | O / PO | 6.12 | 6.34 | 2,090 | 1,890 | 1.56 | 75.4 | 2.0 | 86.4 | 1,875 | 0.80 | 25 | 39.1 | 0.36 | 0.42 |
| CT1501 | #16 | Daapleu | 601 | U | 2.09 | 2.62 | 190 | 191 | 0.09 | 96.6 | 1.0 | 65.5 | 190 | 0.75 | 30 | 14.9 | 0.44 | 0.42 |
| CT1502 | #17 | Daapleu | 601 | U | 1.32 | 1.58 | 320 | 324 | 0.21 | 86.7 | 4.0 | 80.8 | 320 | 1.10 | 45 | 12.4 | 0.39 | 0.21 |
| CT1503 | #18 | Daapleu | 101 | U | 1.49 | 1.48 | 2,520 | 2,932 | 0.47 | 68.2 | 1.0 | 87.1 | 2,900 | 1.10 | 35 | 24.2 | 0.44 | 0.26 |
| CT1504 | #19 | Gbeitouo | 101 | U | 0.51 | 0.41 | 890 | 854 | 0.07 | 82.8 | 1.0 | 10.6 | 850 | 0.48 | 75 | 7.11 | 0.51 | 0.16 |
| CT1505 | #20 | Gbeitouo | 201 | O | 0.70 | 0.90 | 3,710 | 4,075 | 0.06 | 93.4 | 1.0 | 73.6 | 4,075 | 0.01 | 100 | 0.44 | 0.59 | 2.63 |
| CT1506 | #21 | Gbeitouo | 211 | O / PO | 0.32 | 0.40 | 60 | 50 | 0.07 | 82.6 | 0.3 | 96.2 | 50 | 0.15 | 106 | 29.4 | 1.28 | 1.66 |
| CT1507 | #22 | Gbeitouo | 101 | PO | 0.47 | 0.74 | 520 | 500 | 0.06 | 91.9 | 0.2 | 81.6 | 500 | 0.03 | 66 | 24.7 | 0.61 | 3.23 |
| CT1508 | #23 | Ity Flat | 501 | PO | 0.97 | 1.06 | <10 | 5.1 | 0.13 | 87.7 | 0.2 | 96.6 | 5.0 | 1.53 | 716 | 64.5 | 2.96 | 2.37 |
| CT1509 | #24 | Ity Flat | 901 | U | 7.02 | 5.95 | <10 | 5.1 | 0.23 | 96.1 | 0.9 | 70.4 | 5.0 | 1.48 | 3,920 | 3.35 | 0.72 | 0.28 |
| CT1510 | #25 | Ity Flat | 211 | U | 7.85 | 10.3 | <10 | 5.2 | 1.14 | 88.9 | 1.8 | 83.0 | 5.0 | 2.95 | 11,600 | 18.1 | 5.11 | 1.42 |
| CT1511 | #26 | Ity Flat | 201 | O | 10.9 | 10.8 | <10 | 5.1 | 0.29 | 97.3 | 1.2 | 92.6 | 5.0 | 1.49 | 1,720 | 6.22 | 1.02 | 3.13 |
| CT1512 | #27 | Ity Flat | 901 | PO | 0.10 | 0.13 | 20 | 20 | 0.02 | 84.9 | 0.2 | 78.4 | 20 | 0.38 | 100 | 21.9 | 0.47 | 0.33 |
| CT1513 | #28 | Ity Flat | 201 | O | 3.14 | 3.79 | <10 | 5.1 | 0.58 | 84.7 | 0.3 | 37.5 | 5.0 | 1.49 | 706 | 0.06 | 0.66 | 1.80 |
| CT1514 | #29 | Mont Ity | 401 | U | 16.3 | 18.3 | <10 | 5.1 | 0.35 | 98.1 | 0.6 | 93.9 | 5.0 | 1.52 | 232 | 4.02 | 0.39 | 0.26 |
| CT1515 | #30 | Mont Ity | 401 | U | 7.30 | 7.44 | <10 | 5.1 | 1.36 | 81.7 | 3.3 | 76.2 | 5.0 | 1.48 | 4,400 | 4.04 | 3.51 | 1.19 |
| CT1516 | #31 | Mont Ity | 201 | O | 4.33 | 6.14 | 30 | 40 | 0.10 | 98.4 | 2.0 | 89.9 | 40 | 0.78 | 1,646 | 26.0 | 1.64 | 3.37 |
| CT1517 | #32 | Mont Ity | 211 | O | 1.73 | 1.62 | <10 | 5.2 | 0.37 | 77.1 | 1.0 | 67.6 | 5.0 | 3.16 | 220 | 85.6 | 2.88 | 2.11 |
| CT1518 | #33 | Walter | 201 | O | 0.31 | 0.34 | <10 | 5.1 | 0.01 | 97.1 | 1.0 | 62.5 | 5.0 | 1.55 | 626 | 1.74 | 0.67 | 6.42 |
| CT1519 | #34 | Walter | 211 | U | 2.80 | 2.35 | 10 | 10 | 0.18 | 92.3 | 1.0 | 37.0 | 10 | 0.78 | 90 | 46.0 | 0.51 | 1.42 |
| CT1520 | #35 | Walter | 201 | O / PO | 1.35 | 1.50 | <10 | 5.3 | 0.04 | 97.3 | 1.0 | 69.0 | 5.0 | 5.81 | 2,340 | 24.8 | 2.20 | 5.90 |
| CT1521* | #36 | Zia NE | 201 | O | 2.40 | 2.71 | 10 | 5.1 | 0.12 | 95.6 | 1.0 | 55.5 | 5.0 | 2.35 | 2,176 | 0.12 | 0.61 | 4.21 |
| CT1522* | #37 | Zia NE | 201 | O | 9.69 | 10.9 | <10 | 5.1 | 0.34 | 96.9 | 1.0 | 60.7 | 5.0 | 2.36 | 918 | 0.05 | 0.49 | 4.21 |
| CT1523 | #38 | Zia NE | 301 | O | 1.26 | 1.14 | 20 | 10 | 0.03 | 97.4 | 2.0 | 48.3 | 10 | 0.76 | 162 | 1.61 | 0.55 | 5.27 |
| CT1524* | #39 | Zia NE | 201 | O | 0.98 | 1.14 | <10 | 5.1 | 0.05 | 95.6 | 1.0 | 41.7 | 5.0 | 2.33 | 448 | 0.10 | 0.56 | 3.69 |
| CT1525 | #40 | Zia NE | 301 | O | 2.62 | 2.97 | 20 | 5.1 | 0.06 | 98.0 | 4.0 | 5.08 | 5.0 | 1.53 | 530 | 0.58 | 0.62 | 3.69 |

* Leach tests conducted at 30% w/w solids pulp density due to high pulp viscosity at 40% w/w solids.

As a general trend, the low gold extractions correspond to high feed arsenic assays, which may indicate that the gold is locked within arsenopyrite as a solid solution. Additional processing techniques other than conventional milling and cyanidation leaching may, therefore, be required to improve gold recovery from high arsenic ore types.

## 13.5 Diagnostic Testwork Programme

A diagnostic testwork programme was implemented on selected samples which had earlier demonstrated poor gold recovery from the variability tests. The objective of the diagnostic work was to determine the reason(s) for the low gold recovery by performing a series of sequential gravity / leach tests on selected residue samples from the variability testwork campaign.

### 13.5.1 Sample Selection

The residue samples for diagnostic testing were selected on the basis of low gold recovery (typically less than 85%) and with a head assay >0.5 g/t Au. A summary of the samples used for the diagnostic testwork is provided in Table 13.5.1.

**Table 13.5.1 Diagnostic Testwork Samples**

| ALS Variability Test No. | Variability Sample | Pit | Facies | Oxidation | Head Assay (Au ppm) | Tails Assay (Au ppm) | Au Recovery (%) |
|---|---|---|---|---|---|---|---|
| CT1487 | #2 | Daapleu | 601 | U | 2.49 | 0.43 | 82.8 |
| CT1490 | #5 | Daapleu | 601 | U | 0.97 | 0.49 | 49.4 |
| CT1496 | #11 | Daapleu | 601 | U | 21.0 | 11.2 | 46.6 |
| CT1498 | #13 | Daapleu | 601 | U | 1.77 | 0.78 | 56.0 |
| CT1499 | #14 | Daapleu | 601 | U | 1.96 | 1.09 | 44.4 |
| CT1500 | #15 | Daapleu | 601 | O / PO | 6.34 | 1.56 | 75.4 |
| CT1503 | #18 | Daapleu | 101 | U | 1.48 | 0.47 | 68.2 |
| CT1513 | #28 | Ity Flat | 201 | O | 3.79 | 0.58 | 84.7 |
| CT1515 | #30 | Mont Ity | 401 | U | 7.44 | 1.36 | 81.7 |
| CT1517 | #32 | Mont Ity | 211 | O | 1.62 | 0.37 | 77.1 |

### 13.5.2 Diagnostic Testwork Programme

The diagnostic testwork programme was a multi-stage sequential diagnostic gold leach analysis aimed at determining the presence of various minerals which may be preventing high gold recovery from a conventional milling and cyanidation circuit. The tests were performed sequentially as follows:

- Gravity separation using a Knelson concentrator followed by mercury amalgamation of the gravity concentrate to determine the gravity gold content.
- Cyanidation of the combined Knelson concentrator and amalgamation tails to determine the free (cyanidable) gold content.

- Dilute hydrochloric acid (3M HCl) digest of the direct cyanidation leach residue, followed by cyanidation, to determine the gold locked in carbonate minerals.
- Nitric acid digest of the HCl / cyanidation residue, followed by cyanidation, to determine the gold locked in arsenopyrite and other arsenical minerals.
- Concentrated aqua regia digest on the cyanidation residue to determine the gold locked within all remaining sulphides (predominantly pyrite).
- Fire assay of the residue from the aqua regia digest to determine the gold encapsulated in silicates.

### 13.5.3 Diagnostic Testwork Results

A summary of the diagnostic test programme results conducted on the 10 variability leach residue samples is provided in Table 13.5.2. This shows the fraction (%) of the residual gold recovered at each stage of the diagnostic testing.

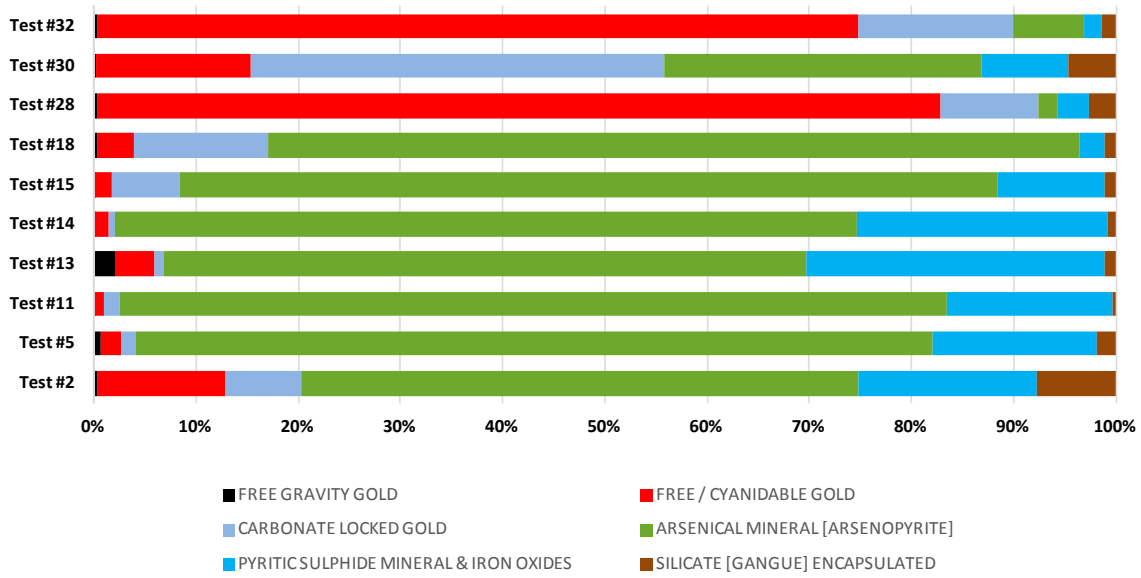
**Table 13.5.2 Diagnostic Testwork Results**

| ALS Variability Test No | Variability Sample | Free Gravity Gold | Free Cyanidable Gold | Carbonate Locked Gold | Arsenical Mineral | Pyritic Sulphide Mineral and Iron Oxides | Silicate Locked |
|---|---|---|---|---|---|---|---|
| CT1487 | #2 | 0.34 | 12.5 | 7.54 | 54.52 | 17.45 | 7.66 |
| CT1490 | #5 | 0.62 | 2.03 | 1.40 | 77.96 | 16.21 | 1.78 |
| CT1496 | #11 | 0.03 | 0.92 | 1.53 | 81.01 | 16.18 | 0.32 |
| CT1498 | #13 | 2.03 | 3.86 | 0.91 | 62.87 | 29.18 | 1.15 |
| CT1499 | #14 | 0.14 | 1.37 | 0.62 | 72.48 | 24.61 | 0.78 |
| CT1500 | #15 | 0.09 | 1.74 | 6.54 | 80.04 | 10.55 | 1.04 |
| CT1503 | #18 | 0.36 | 3.55 | 13.16 | 79.41 | 2.49 | 1.03 |
| CT1513 | #28 | 0.43 | 82.47 | 9.51 | 1.87 | 3.11 | 2.60 |
| CT1515 | #30 | 0.15 | 15.26 | 40.43 | 31.01 | 8.47 | 4.68 |
| CT1517 | #32 | 0.44 | 74.36 | 15.13 | 6.99 | 1.75 | 1.32 |

A graphical representation of the diagnostic test results, showing the gold distribution among the various mineral, types is presented in Figure 13.5.1. These results indicate that the majority (between 54% and 81%) of the unleached gold from the Daapleu variability residue samples (#2, #5, #11, #13, #14, #15, and #18) is locked within arsenopyrite, confirming the refractory nature of the Daapleu primary ore and the need to adopt a different processing route for this particular ore type if gold recoveries are to be improved.

Confirmation of the refractory nature of the Daapleu primary ore led to the development of a separate refractory testwork programme dedicated to a composite derived from Daapleu core samples identified as having unoxidised mineralogy (Composite #7).

**Figure 13.5.1 Diagnostic Gold Analysis – Overall Distribution**




## 13.6 Optimisation Testwork Programme

Following completion of the comminution and variability testwork programmes, a separate optimisation testwork programme was implemented to optimise the grind size and subsequent cyanide leach conditions for the proposed CIL circuit. This optimisation programme also included oxygen uptake tests, a preg-robbing index test, standard carbon loading (kinetic) tests, cyanide detoxification tests, and slurry rheology tests.

### 13.6.1 Sample Selection

The optimisation testwork programme was implemented on six different composites representing each of the different pits. Two separate composites (Composite #1 and Composite #7) were generated from Daapleu as being representative of the oxidised and unoxidised (primary) ore respectively. (Note that there was insufficient Gbeitouo core sample to compile the required minimum mass of ~50 kg, and thus the Gbeitouo composite, Composite #2, was excluded from the optimisation testwork programme. As ore from the Gbeitouo deposit only contributes approximately 6% of the total Ity ore reserve, failure to conduct optimisation testwork on Gbeitouo ore was not considered to be a significant omission).

The samples for each composite were selected to provide a final composite that was representative of the different facies and gold grades within each pit. Having selected the appropriate core samples, the core was crushed and blended to yield six different composites upon which to perform the optimisation testwork.

### 13.6.2 Optimisation Testwork Programme

The Optimisation Testwork Programme was carried out on six separate composites. The following summarises the optimisation testwork programme performed on each composite:

- Composite preparation.
- Gravity gold determination – this was performed upfront, i.e. all subsequent cyanidation tests were performed on the combined gravity gold and amalgamation tails to avoid mass balance discrepancies.
- Grind optimisation – cyanidation leach tests performed at three grind sizes ($P_{80}$, 75 µm, 90 µm and 106 µm) to establish optimum grind.
- Leach optimisation:
  - Select optimum grind size.
  - 48 hour cyanidation test (kinetic sampling at 1, 2, 4, 6, 8, 24, 36, 48 hours).
  - Three different pulp densities (30, 35, 40% w/w solids for the Zia NE composite; 40, 45, 50% w/w solids for the remaining five composites).
  - Three different free CN concentrations (150, 250, 350 ppm).
- Preg-robbing index tests.
- Oxygen uptake rate tests.
- Pulp viscosity tests.
- Triple contact carbon adsorption tests (at optimised leach conditions) to determine k and n values.
- Carbon equilibrium loading tests.
- Cyanide detoxification tests (ambient temperature, one hour, pH 8.5, oxygen, SMBS) to achieve target $CN_{WAD}$ <10 ppm.
- Generation of tailings for settling and geochem testing (by others).

### 13.6.3 Head Analysis

The results of the head analyses for each of the six composites is shown in Table 13.6.1. The original gold head analysis for Composite #4 yielded three variable results (6.12 g/t, 7.55 g/t, to 13.10 g/t) resulting in poor correlation with the calculated head from the leach tests. Consequently, the assays were repeated and again returned variable gold head values (9.47 g/t and 3.10 g/t), indicative of a nuggety gold effect in this sample. (As the calculated head from the leach tests is a much larger sub-sample (1 kg) than the 50 g sub-sample used in fire assay, as well as the fact that the calculated head is so consistent (~3.3 g/t), the calculated head is likely to be a more accurate reflection of the 'true' gold assay for Composite #4).

**Table 13.6.1 Composite Head Assays**

| Composite | Composite #1 | Composite #3 | Composite #4 | Composite #5 | Composite #6 | Composite #7 |
|---|---|---|---|---|---|---|
| Au (ppm) | 1.26 | 5.22 | 6.12 | 0.79 | 1.11 | 1.42 |
| Au (ppm) | 1.48 | 4.77 | 7.55 | 0.81 | 1.32 | 2.43 |
| Au (ppm) | 1.34 | 5.10 | 13.10 | 0.85 | 1.11 | 1.22 |
| Au (ppm) | | | 9.47 | | | |
| Au (ppm) | | | 3.10 | | | |
| Au av. (ppm) | 1.36 | 5.03 | 7.87 | 0.82 | 1.18 | 1.69 |
| Ag (ppm) | 2 | 4 | 8 | <2 | <2 | 4 |
| Al (%) | 6 | 8 | 5 | 7 | 11 | 6 |
| As (ppm) | 910 | <10 | <10 | <10 | <10 | 1,620 |
| Ba (ppm) | 1,080 | 240 | 300 | 620 | 300 | 1,060 |
| Be (ppm) | <20 | <20 | <20 | <20 | <20 | <20 |
| Bi (ppm) | <25 | <25 | <25 | <25 | <25 | <25 |
| C (%) | 0.18 | 1.65 | 3.96 | 0.06 | 0.15 | 0.51 |
| C org (%) | 0.03 | 0.09 | 0.09 | 0.06 | 0.12 | <0.03 |
| Ca (%) | 0.96 | 6.10 | 11.2 | 3.54 | 1.30 | 1.99 |
| Cd (ppm) | <20 | <20 | <20 | <20 | <20 | <20 |
| Co (ppm) | <20 | 20 | 20 | 40 | 40 | <20 |
| Cr (ppm) | 125 | 100 | 50 | 125 | 125 | 125 |
| Cu (ppm) | 80 | 4200 | 1720 | 740 | 575 | 45 |
| Fe (%) | 2.54 | 5.90 | 8.76 | 6.80 | 18.7 | 3.02 |
| Hg (ppm) | 0.2 | 0.1 | 0.2 | <0.1 | <0.1 | <0.1 |
| K (%) | 3.65 | 0.25 | 0.05 | 0.8 | 0.15 | 3.98 |
| Li (ppm) | <20 | <20 | <20 | <20 | <20 | <20 |
| Mg (%) | 1.00 | 5.32 | 6.12 | 4.72 | 0.24 | 1.44 |
| Mn (ppm) | 500 | 900 | 1,800 | 800 | 2,200 | 900 |
| Mo (ppm) | <20 | 60 | <20 | <20 | 20 | <20 |
| Na (ppm) | 2,250 | 700 | 500 | 2,850 | 950 | 3,050 |
| Ni (ppm) | 60 | 40 | 40 | 80 | 60 | 60 |
| P (ppm) | 500 | 250 | 500 | 500 | 1250 | 500 |
| Pb (ppm) | 60 | 20 | <20 | <20 | <20 | <20 |
| S (%) | 1.58 | 0.50 | 0.56 | 0.82 | 0.04 | 1.84 |
| S-2 (%) | 1.30 | 0.44 | 0.46 | 0.82 | <0.02 | 1.48 |
| Sb (ppm) | 7.2 | 0.5 | 1.0 | 0.6 | 1.1 | 7.6 |
| SiO2 (%) | 66.0 | 39.2 | 28.0 | 52.8 | 37.4 | 53.2 |
| Sr (ppm) | 105 | 80 | 80 | 95 | 50 | 140 |
| Te (ppm) | <0.2 | 4.8 | 6.4 | 1.4 | 4.0 | <0.2 |
| Ti (ppm) | 1,800 | 3,600 | 2,400 | 4,400 | 6,000 | 1,800 |
| V (ppm) | 50 | 125 | 75 | 115 | 265 | 45 |
| Y (ppm) | <100 | <100 | <100 | <100 | <100 | <100 |
| Zn (ppm) | 240 | 625 | 315 | 150 | 125 | 95 |

### 13.6.4 Preg-Robbing Tests

Results of the preg-robbing tests performed on each of the six composites are summarised in Table 13.6.2.

**Table 13.6.2 Preg-Robbing Test Summary**

| Composite | Au Concentration (mg/L) | | | |
|---|---|---|---|---|
| | Au Spike | AuCN | AuPR | PRI |
| Composite #1 [Daapleu] | 3.44 | 0.55 | 3.93 | 0.06 |
| Composite #3 [Ity Flat] | 3.44 | 1.47 | 4.90 | 0.01 |
| Composite #4 [Mont Ity | 3.44 | 1.23 | 4.89 | -0.22 |
| Composite #5 [Walter] | 3.44 | 0.37 | 3.88 | -0.08 |
| Composite #6 [Zia NE] | 3.44 | 0.50 | 4.13 | -0.20 |
| Composite #7 [Daapleu Primary] | 3.44 | 0.45 | 4.03 | -0.15 |

A preg-robbing index (PRI) between zero and 1.0 indicate minimal preg-robbing activity, between 1.0 and 2.5 moderate activity, and a PRI greater than 2.5 indicates the ore is highly preg-robbing. The results of the preg-robbing tests performed on each of the composites indicate that preg-robbing is unlikely to be a factor.

### 13.6.5 Grind Optimisation Tests

Cyanide leach tests were performed on each of the six composites at three different grind sizes ($P_{80}$ 106 µm, 90 µm, 75 µm) under standard leach conditions (40% w/w solids, 500 ppm initial NaCN, pH 10.5, oxygen sparging) for 48 hours to establish the effect of different grind sizes on gold recovery. Each composite was first subjected to a gravity gold determination using a 3'' Knelson concentrator. The gravity concentrate was amalgamated with mercury and the concentrator and amalgam tails recombined for feed to the subsequent cyanide leach tests. The results are summarised in Table 13.6.3 and Figures 13.6.1, 13.6.2 and 13.6.3 below.

**Table 13.6.3 Grind Optimisation Summary**

| ALS Test No | Composite | Grind Size $P_{80}$ (µm) | Gravity Au (%) | Calc. Head (Au ppm) | Leach Residue (Au ppm) | Au Extraction, % | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | | 4 hours | 8 hours | 24 hours | 48 hours |
| CT1593 | Composite #1 [Daapleu] | 106 | 29.1 | 1.51 | 0.28 | 77.47 | 78.84 | 80.18 | 81.46 |
| CT1594 | | 90 | 27.7 | 1.48 | 0.31 | 74.49 | 76.40 | 76.40 | 79.05 |
| CT1595 | | 75 | 24.1 | 1.27 | 0.22 | 81.04 | 81.58 | 82.65 | 82.65 |
| CT1596 | Composite #3 [Ity Flat] | 106 | 42.4 | 5.50 | 0.15 | 85.16 | 88.48 | 96.08 | 97.27 |
| CT1597 | | 90 | 49.0 | 6.12 | 0.22 | 87.01 | 92.24 | 95.77 | 96.41 |
| CT1598 | | 75 | 47.2 | 5.62 | 0.18 | 86.98 | 93.70 | 95.86 | 96.79 |
| CT1599 | Composite #4 [Mont Ity] | 106 | 28.2 | 3.46 | 0.21 | 68.71 | 78.62 | 83.30 | 93.94 |
| CT1600 | | 90 | 36.2 | 3.22 | 0.19 | 70.95 | 79.03 | 83.86 | 94.10 |
| CT1601 | | 75 | 32.8 | 3.09 | 0.15 | 67.80 | 78.47 | 82.84 | 95.15 |
| CT1602 | Composite #5 [Walter] | 106 | 16.8 | 1.10 | 0.08 | 86.48 | 90.90 | 92.74 | 92.74 |
| CT1603 | | 90 | 19.2 | 1.12 | 0.06 | 89.75 | 91.01 | 94.63 | 94.63 |
| CT1604 | | 75 | 18.8 | 1.06 | 0.05 | 90.85 | 90.85 | 95.30 | 95.30 |
| CT1605 | Composite #6 [Zia NE] | 106 | 21.2 | 1.22 | 0.05 | 92.03 | 93.74 | 94.30 | 95.91 |
| CT1606 | | 90 | 18.7 | 1.18 | 0.05 | 92.20 | 95.20 | 95.20 | 95.76 |
| CT1607 | | 75 | 16.4 | 1.10 | 0.03 | 94.14 | 96.07 | 96.07 | 97.26 |
| CT1608 | Composite #7 [Daapleu Primary] | 106 | 11.4 | 1.57 | 0.55 | 59.82 | 62.06 | 65.06 | 65.06 |
| CT1609 | | 90 | 20.9 | 1.77 | 0.58 | 63.84 | 64.24 | 66.14 | 67.26 |
| CT1610 | | 75 | 16.7 | 1.65 | 0.57 | 62.52 | 63.78 | 64.60 | 65.40 |

**Figure 13.6.1 Grind Optimisation Results**

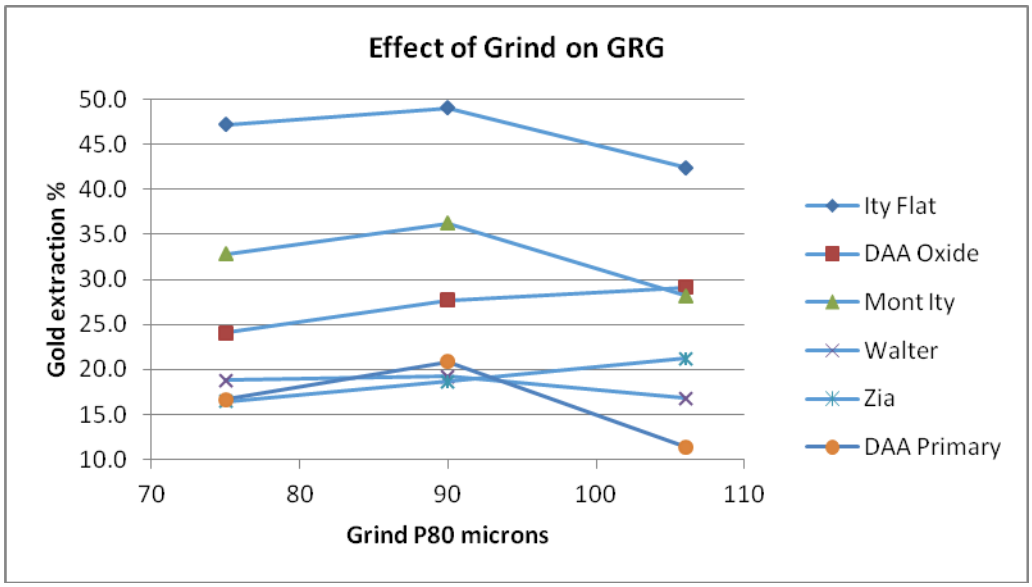


**Figure 13.6.2 Grind Optimisation Results**

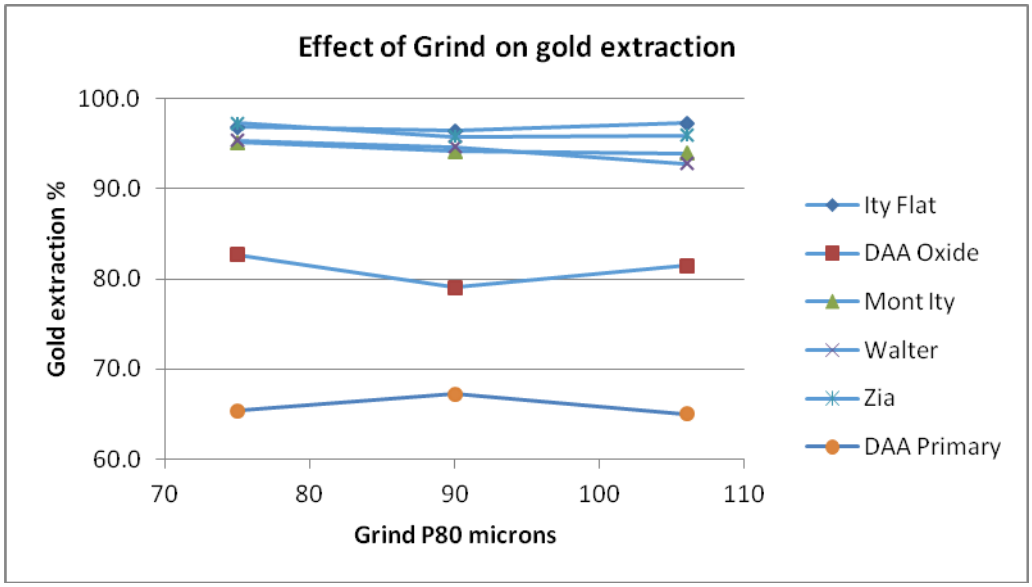
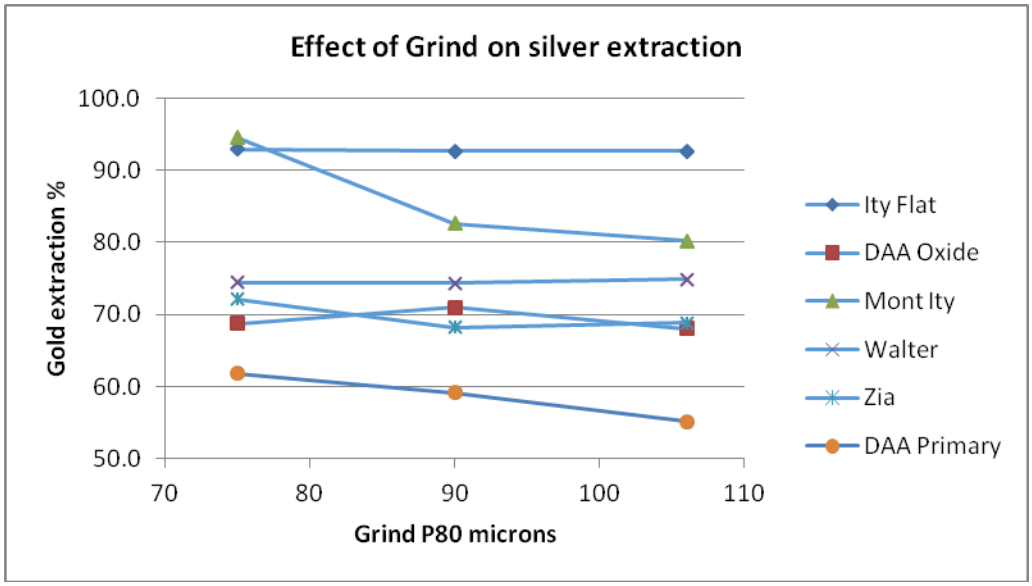



**Figure 13.6.3 Grind Optimisation Results**




The results indicate a significant mass of gravity recoverable gold (approximately 24 to 49%) for the Daapleu oxide and primary ore mix (Composite #1), Ity Flat (Composite #3), and Mont Ity (Composite #4). Gravity recoverable gold from Walter (Composite #5), Zia NE (Composite #6) and Daapleu primary ore (Composite #7) is generally less than c.20%.

All ore composites, with the exception Mont Ity (Composite #4) indicated that there was little additional gold extraction after 24 hours of leaching time. The Mont Ity ore composite achieved an approximate 10 to 12% improvement in gold recovery by doubling the leach residence time to 48 hours.

These results typically indicate a 1 to 2% increase in gold recovery at the end of the 48 hour cyanide leach using a $P_{80}$ grind size of 75 µm compared to the coarser 90 and 106 µm size fractions. Consequently 75 µm was adopted for all subsequent cyanide and pulp density optimisation tests. Similarly, the consistently high proportion of gravity recoverable gold for all composites meant that the Knelson concentrator was included ahead of cyanidation for the subsequent cyanide and pulp density optimisation testwork.

### 13.6.6 Pulp Density Optimisation Tests

A suite of gravity gold tests followed by cyanidation were performed on each of the six composites at three different pulp densities (30%, 35%, and 40% solids for Composite #6, Zia NE, and 40%, 45%, and 50% solids for the other five composites), each at the optimum grind size ($P_{80}$ 75 µm), same starting NaCN concentration (500 ppm), and for a 48 hour leach duration. The results are summarised in Table 13.6.4.

**Table 13.6.4 Pulp Density Optimisation Summary**

| Composite | Pulp Density (% solids) | Head Assay (Au ppm) | Calc. Head (Au ppm) | Gravity Au (%) | Overall Au Recovery (%) | Gravity Ag (%) | Overall Ag Recovery (%) |
|---|---|---|---|---|---|---|---|
| Composite #1 [Daapleu Oxide] | 40 | 1.36 | 1.29 | 18.9 | 79.9 | 1.4 | 67.3 |
| | 45 | 1.36 | 1.50 | 16.2 | 80.0 | 1.3 | 71.2 |
| | 50 | 1.36 | 1.36 | 18.0 | 80.9 | 1.4 | 67.4 |
| Composite #3 [Ity Flat] | 40 | 5.03 | 6.03 | 44.7 | 97.3 | 2.8 | 92.6 |
| | 45 | 5.03 | 6.06 | 44.5 | 97.5 | 2.6 | 86.2 |
| | 50 | 5.03 | 5.99 | 45.0 | 97.5 | 2.4 | 87.4 |
| Composite #4 [Mont Ity] | 40 | 8.92 | 3.57 | 25.1 | 89.4 | 2.3 | 70.3 |
| | 45 | 8.92 | 3.69 | 24.2 | 94.3 | 2.2 | 76.5 |
| | 50 | 8.92 | 3.31 | 27.1 | 92.1 | 2.4 | 60.5 |
| Composite #5 [Walter] | 40 | 0.82 | 1.04 | 17.2 | 95.2 | 2.0 | 87.1 |
| | 45 | 0.82 | 0.99 | 18.0 | 96.0 | 2.3 | 85.1 |
| | 50 | 0.82 | 1.09 | 16.5 | 96.3 | 2.2 | 85.8 |
| Composite #6 [Zia NE] | 30 | 1.18 | 1.09 | 16.5 | 96.3 | 1.3 | 70.4 |
| | 35 | 1.18 | 1.09 | 16.5 | 96.3 | 1.4 | 70.0 |
| | 40 | 1.18 | 1.08 | 16.7 | 96.3 | 1.4 | 70.0 |
| Composite #7 [Daapleu Primary] | 40 | 1.69 | 1.70 | 14.8 | 63.5 | 3.1 | 69.6 |
| | 45 | 1.69 | 1.74 | 14.5 | 63.5 | 2.9 | 65.0 |
| | 50 | 1.69 | 1.67 | 15.1 | 64.0 | 2.8 | 61.4 |

The results indicate a consistently high recovery of gold to the gravity concentrate for the Ity Flat material (~45%) with reasonably high gold recovery to a gravity concentrate reported for Mont Ity (~25%). Each of the four remaining composites demonstrated similar gold recoveries to a gravity concentrate of between approximately 15% and 19%. Conversely, silver recovery to the gravity concentrate was consistently low for all composites (between 1.3% and 3.1%).

With the exception of Composite #4 (Mont Ity), the results indicate little discernible effect on gold recovery at the three different pulp densities used for each composite. The Mont Ity results show a 94.3% gold recovery at 45% solids, while at 50% solids the gold recovery falls to 92.1% (this may be due to pulp viscosity – refer to Section 5.6.9). Consequently, for all composites other than Mont Ity, the highest of the three pulp densities was adopted as the optimum condition for subsequent testwork.

### 13.6.7 Cyanide Optimisation Tests

A suite of gravity gold recovery followed by cyanidation tests was performed on each of the six composites at three different starting NaCN concentrations (150 ppm, 250 ppm, and 350 ppm), each at the optimum grind size ($P_{80}$ 75 µm), pulp density (40% w/w solids), and for a 48 hour leach duration. The results are summarised in Table 13.6.5. (The table also shows the results for the pulp density optimisation tests performed at 40% w/w solids and 500 ppm NaCN for comparison).

**Table 13.6.5 Cyanide Optimisation Summary**

| ALS Test No | Composite | Initial NaCN (ppm) | Calc. Head (Au ppm) | Gravity Au (%) | Overall Au Recovery (%) | Gravity Ag (%) | Overall Ag Recovery (%) |
|---|---|---|---|---|---|---|---|
| CT1636 | | 150 | 1.40 | 17.4 | 80.0 | 1.3 | 62.0 |
| CT1637 | Composite #1 | 250 | 1.40 | 17.5 | 82.1 | 1.4 | 69.1 |
| CT1638 | [Daapleu] | 350 | 1.28 | 19.1 | 85.9 | 1.5 | 74.7 |
| CT1633 | | 500 | 1.29 | 18.9 | 79.9 | 1.4 | 67.3 |
| CT1642 | | 150 | 5.74 | 47.0 | 76.5 | 3.0 | 21.0 |
| CT1643 | Composite #3 | 250 | 5.48 | 49.2 | 95.6 | 2.8 | 78.1 |
| CT1644 | [Ity Flat] | 350 | 6.21 | 43.4 | 97.1 | 2.7 | 85.5 |
| CT1639 | | 500 | 6.03 | 44.7 | 97.3 | 2.8 | 92.6 |
| CT1648 | | 150 | 3.44 | 26.0 | 53.8 | 2.7 | 5.6 |
| CT1649 | Composite #4 | 250 | 3.32 | 27.0 | 71.4 | 2.4 | 12.7 |
| CT1650 | [Mont Ity] | 350 | 3.28 | 27.3 | 75.6 | 3.0 | 33.8 |
| CT1645 | | 500 | 3.57 | 25.1 | 89.4 | 2.3 | 70.3 |
| CT1654 | | 150 | 1.12 | 16.0 | 87.5 | 2.5 | 34.9 |
| CT1655 | Composite #5 | 250 | 1.05 | 17.0 | 94.3 | 2.1 | 86.1 |
| CT1656 | [Walter] | 350 | 1.05 | 17.0 | 96.2 | 1.8 | 88.0 |
| CT1651 | | 500 | 1.04 | 17.2 | 95.2 | 2.0 | 87.1 |
| CT1660 | | 150 | 1.08 | 16.7 | 95.4 | 1.3 | 71.3 |
| CT1661 | Composite #6 | 250 | 1.08 | 16.6 | 96.3 | 1.3 | 70.7 |
| CT1662 | [Zia NE] | 350 | 1.10 | 16.4 | 95.4 | 1.1 | 61.8 |
| CT1657 | | 500 | 1.09 | 16.5 | 96.3 | 1.3 | 70.4 |
| CT1666 | | 150 | 1.68 | 15.0 | 60.2 | 2.9 | 59.1 |
| CT1667 | Composite #7 | 250 | 1.71 | 14.8 | 64.9 | 2.9 | 59.4 |
| CT1668 | [Daapleu Primary] | 350 | 1.72 | 14.7 | 62.8 | 2.0 | 39.6 |
| CT1663 | | 500 | 1.70 | 14.8 | 63.5 | 3.1 | 69.6 |

The cyanide optimisation tests consistently showed that the lowest gold recoveries were achieved at the lowest (150 ppm) cyanide concentration. Only the Mont Ity composite (Composite #4) demonstrated a higher gold recovery when starting with a 500 ppm NaCN concentration instead of 350 ppm. Consequently, for all composites other than Mont Ity, the starting NaCN concentration of 350 ppm was adopted as the optimum conditions for the subsequent testwork.

***Optimised Pulp Density and Cyanide Conditions***

Based on the results from the pulp density and cyanide optimisation tests (all conducted at the optimised $P_{80}$ of 75 µm) a set of optimised leach conditions was developed for each composite for the subsequent bulk cyanidation testwork. This set of criteria is summarised in Table 13.6.6.

**Table 13.6.6 Optimised Leach Conditions for Each Composite**

| Composite | Grind Size $P_{80}$ (µm) | Pulp Density (% w/w solids) | NaCN Conc (ppm) |
|---|---|---|---|
| Composite #1 [Daapleu Oxide] | 75 | 50 | 350 |
| Composite #3 [Ity Flat] | 75 | 50 | 350 |
| Composite #4 [Mont Ity] | 75 | 45 | 500 |
| Composite #5 [Walter] | 75 | 50 | 350 |
| Composite #6 [Zia NE] | 75 | 40 | 350 |
| Composite #7 [Daapleu Primary] | 75 | 50 | 350 |

Results of the bulk leach tests are presented in Figures 13.6.4 to 13.6.9 below.

**Figure 13.6.4 Rate of Metal Extraction – Daapleu Oxide**

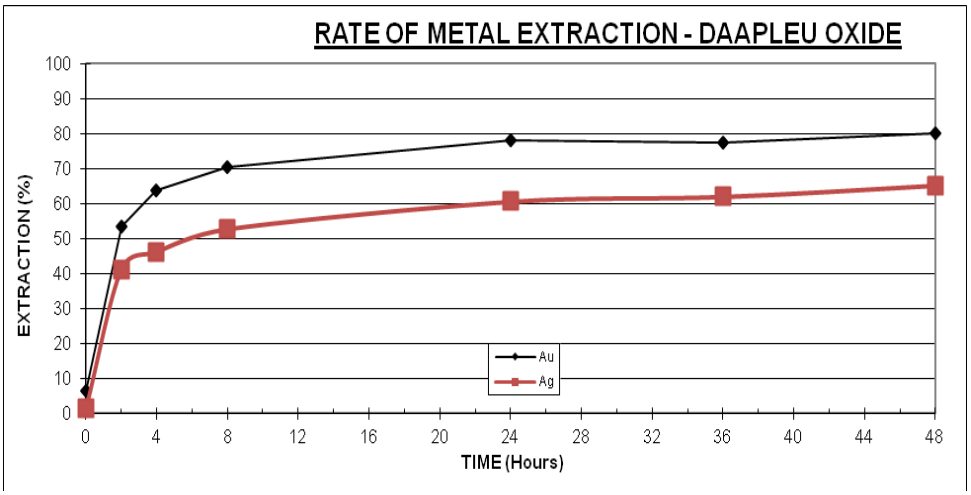


**Figure 13.6.5 Rate of Metal Extraction – Ity Flat**

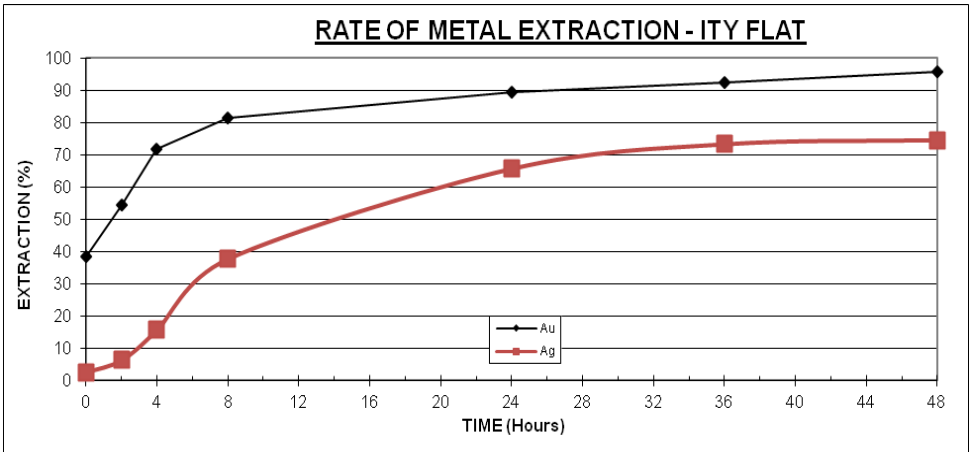



**Figure 13.6.6 Rate of Metal Extraction – Mont Ity**

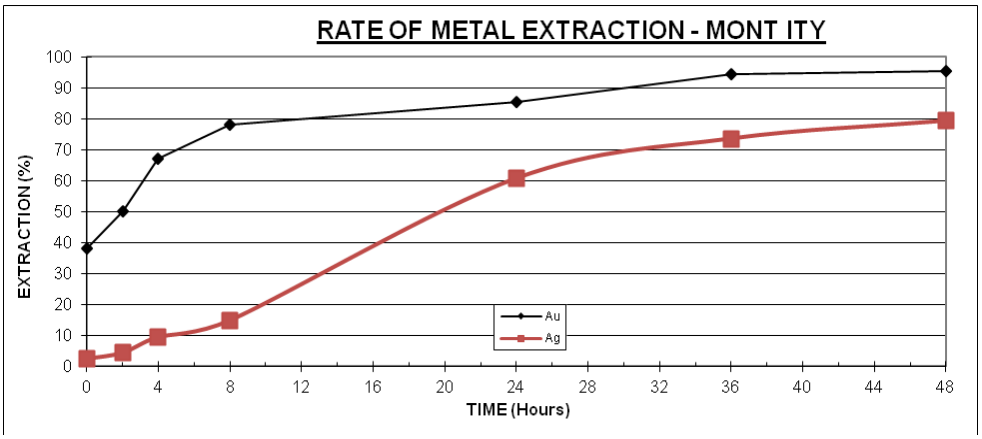



**Figure 13.6.7 Rate of Metal Extraction – Walter**

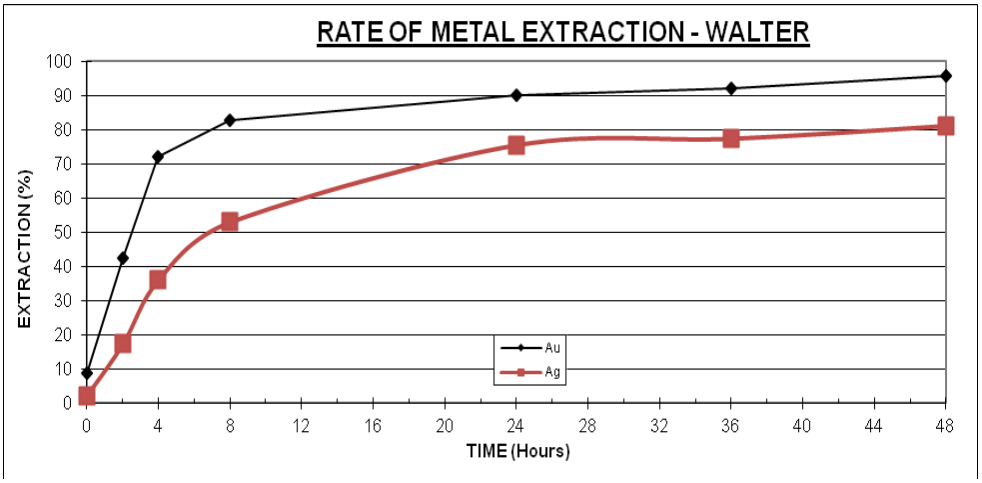


**Figure 13.6.8 Rate of Metal Extraction – Zia NE**

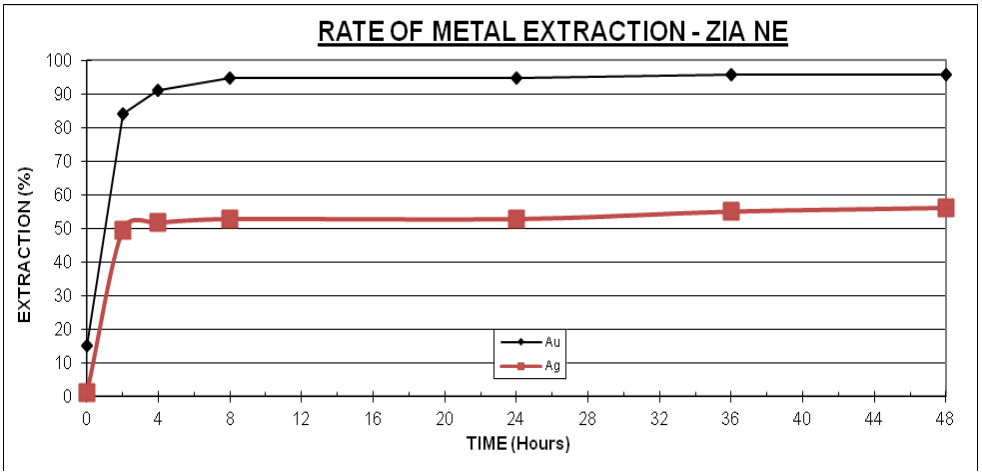
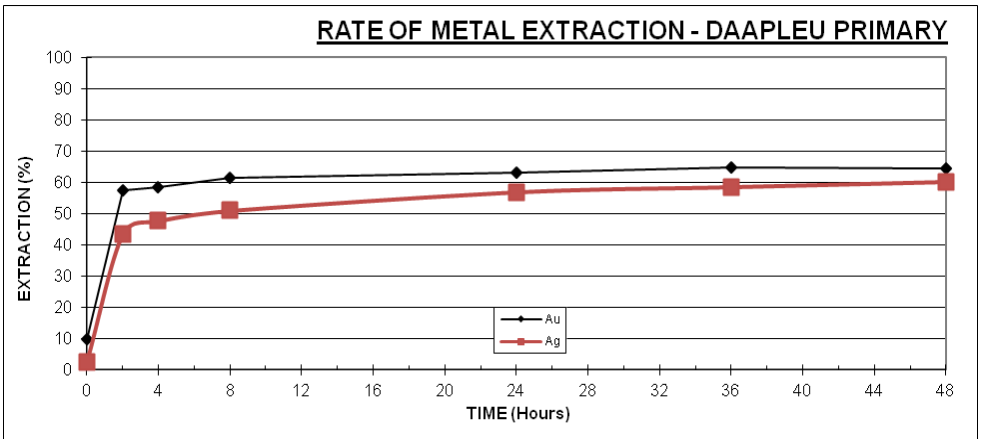



**Figure 13.6.9 Rate of Metal Extraction – Daapleu Primary**




### 13.6.8 Oxygen Uptake Rate Tests

A series of oxygen uptake rate tests was performed on each of the composites to provide an indication of the oxygen requirements for the leaching circuit. The tests were each performed under the optimised conditions of grind, pulp density, and starting cyanide concentration shown in Table 13.6.6.

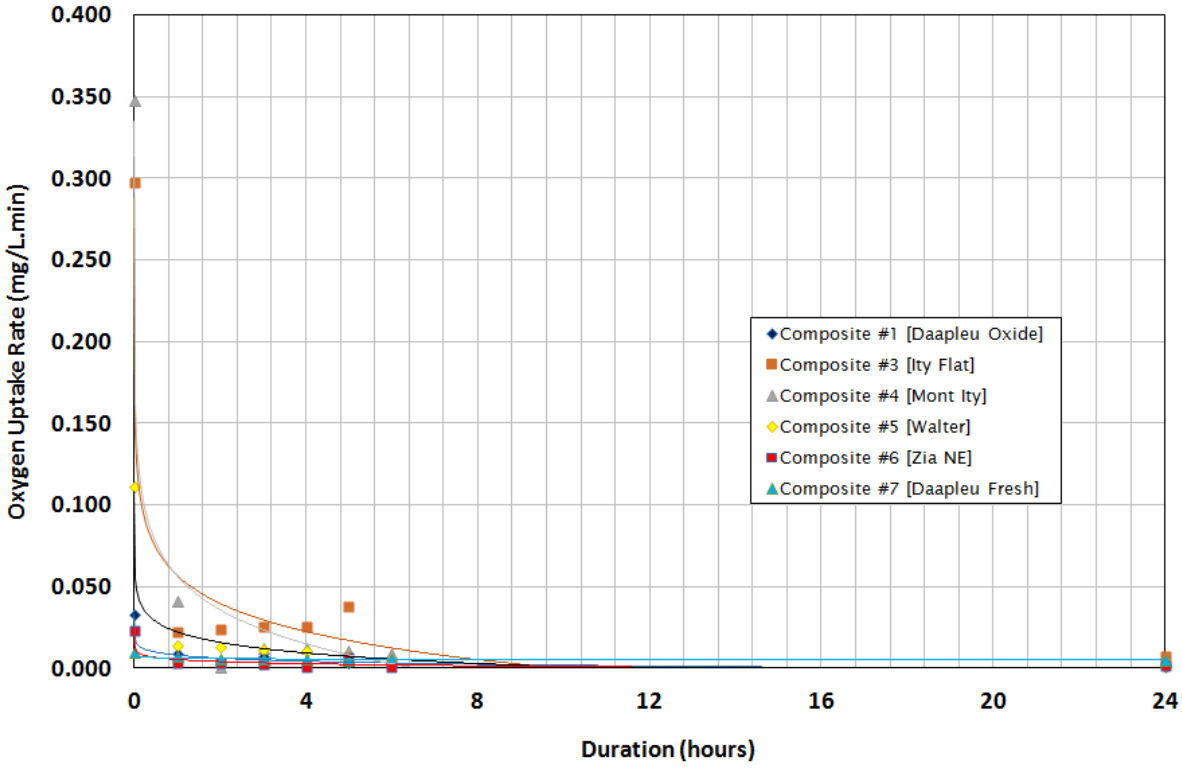
The standard procedure used for the oxygen uptake rate tests was adopted. The series of oxygen decay measurements was repeated after 1, 2, 3, 4, 5, 6, and 24 hours.

A summary of the results is provided in Table 13.6.7.

**Table 13.6.7 Oxygen Uptake Results Summary**

| Composite | Oxygen Uptake Rate (mg/L.min) at Elapsed Time (hours) | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 24 |
| Composite #1 [Daapleu Oxide] | 0.0332 | 0.0090 | 0.0038 | 0.0071 | 0.0035 | 0.0034 | 0.0022 | 0.0009 |
| Composite #3 [Ity Flat] | 0.2979 | 0.0225 | 0.0237 | 0.0258 | 0.0252 | 0.0378 | 0.0039 | 0.0074 |
| Composite #4 [Mont Ity] | 0.3475 | 0.0416 | 0.0011 | 0.0120 | 0.0126 | 0.0105 | 0.0087 | 0.0055 |
| Composite #5 [Walter] | 0.1113 | 0.0143 | 0.0136 | 0.0113 | 0.0107 | 0.0076 | 0.0036 | 0.0018 |
| Composite #6 [Zia NE] | 0.0234 | 0.0030 | 0.0032 | 0.0026 | 0.0008 | 0.0066 | 0.0009 | 0.0016 |
| Composite #7 [Daapleu Primary] | 0.0100 | 0.0070 | 0.0055 | 0.0058 | 0.0054 | 0.0053 | 0.0061 | 0.0047 |

**Figure 13.6.10 Oxygen Uptake Results Summary**




The data indicates that ore from Mont Ity and Ity Flat, in particular, has a high oxygen consumption within the first hour of leaching, followed by ore from Walter. Oxygen demand for Ity Flat remains highest for the duration of the test and does not tend to taper as rapidly with increasing time. The Daapleu primary ore (Composite #7) has a consistently low oxygen demand.

Although the oxygen uptake demand is typically low (i.e. consistently less than 0.15 mg/L.min for all values recorded except the initial readings for the Mont Ity and Ity Flat composites), the addition of oxygen instead of air to the CIL tanks is proposed for the preliminary Ity CIL plant design as it offers a number of significant benefits:

- Reduced cyanide consumption.
- Improved leach kinetics that will assist in improving overall gold recovery where leach residence time may be marginal.
- Assist in the oxidation of any reactive sulphides that may be present.
- Reduced potential for cyanide stripping and HCN gas generation within the CIL circuit.

Based on the results of the oxygen uptake rate tests and the likely consumption of oxygen within the cyanide detoxification circuit (refer to Section 13.6.11) it is proposed to install 2 x 10 t/day oxygen plant modules to satisfy anticipated demand.

### 13.6.9 Rheology Tests

A series of rheology tests were performed at three different pulp densities for each of the six composites. A summary of the results is presented in Table 13.6.8.

**Table 13.6.8 Rheology Results Summary**

| Composite | Solids % w/w | Bohlin Visco 88 Viscosity (cP) @ Shear Rate (sec-1) | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | 4.2 | 7.4 | 13.1 | 21.9 | 38.9 | 67.4 | 119.2 | 209.5 |
| Composite #1 [Daapleu Oxide] | 60 | 338 | 297 | 240 | 179 | 141 | 116 | 124 | 137 |
| | 50 | 0 | 0 | 0 | 0 | 0 | 30 | 45 | 67 |
| | 40 | 0 | 0 | 0 | 0 | 0 | 19 | 34 | 55 |
| Composite #3 [Ity Flat] | 50 | 11,761 | 7,760 | 4,665 | 2,757 | 1,668 | 1,033 | 661 | 400 |
| | 40 | 3,309 | 2,039 | 1,188 | 811 | 480 | 289 | 172 | 145 |
| | 30 | 752 | 489 | 300 | 194 | 121 | 79 | 68 | 76 |
| Composite #4 [Mont Ity] | 60 | 23,976 | 19,567 | 12,718 | 7,834 | 4,670 | 2,977 | 1,886 | 1,120 |
| | 50 | 7,484 | 4,843 | 2,832 | 1,730 | 1,044 | 650 | 400 | 245 |
| | 40 | 2,332 | 1,360 | 828 | 546 | 339 | 207 | 129 | 112 |
| Composite #5 [Walter] | 60 | 8,317 | 6,535 | 5,816 | 3,607 | 2,300 | 1,387 | 875 | 569 |
| | 50 | 1,956 | 1,211 | 720 | 452 | 278 | 175 | 130 | 133 |
| | 40 | 0 | 0 | 0 | 65 | 60 | 51 | 57 | 69 |
| Composite #6 [Zia NE] | 50 | 14,165 | 8,911 | 5,624 | 3,559 | 2,203 | 1,351 | 816 | 478 |
| | 40 | 4,137 | 2,443 | 1,440 | 904 | 536 | 324 | 196 | 148 |
| | 30 | 1,091 | 659 | 408 | 258 | 161 | 102 | 80 | 79 |
| Composite #7 [Daapleu Primary] | 60 | 14,165 | 8,911 | 5,624 | 3,559 | 2,203 | 1,351 | 816 | 478 |
| | 50 | 0 | 0 | 0 | 0 | 0 | 23 | 43 | 68 |
| | 40 | 0 | 0 | 0 | 0 | 0 | 14 | 32 | 51 |

The results of the viscosity tests indicate the following:

- The Daapleu oxide composite demonstrated relatively low viscosity characteristics for the pulp densities tested – even at 60% w/w solids pulp density the results indicate that adsorption kinetics would not be an issue in a CIL circuit design.

- The Ity Flat (Composite #3) rheology testwork returned high viscosity values and indicated that centrifugal pumps would not be selected for pumping slurries containing greater than c.45% w/w solids. Moreover, operation of a CIL circuit would only be appropriate at a pulp density of less than c.35% w/w solids.

- Of the three pulp densities tested for the Mont Ity (Composite #4) ore, only 40% w/w solids would be suitable for a CIL circuit design and the use of centrifugal slurry pumps – pulp densities of 50% w/w and 60% w/w solids returned viscosity data that would preclude the adoption of a conventional CIL circuit and of the use of centrifugal slurry pumps at these densities.

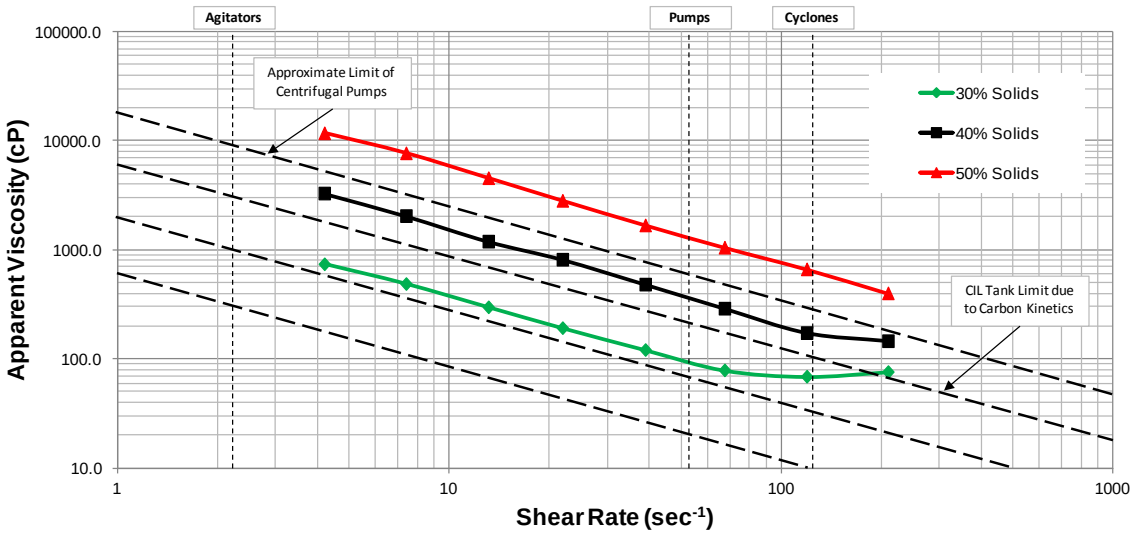
- The Walter (Composite #5) ore rheology testwork indicated that a conventional CIL circuit would not operate satisfactorily at a pulp density greater than approximately 50% w/w solids.

- The selection of 30%, 40%, and 50% w/w solids pulp densities for the rheology testwork for Zia NE (Composite #6) based on earlier leach optimisation work, was vindicated by the viscosity results. The ore showed similar rheological characteristics to those of Ity Flat (Composite #3), indicating a maximum pulp density in CIL of approximately 35% w/w solids.

- Like Daapleu Oxide (Composite #1) ore, Daapleu Primary (Composite #7) ore demonstrated low viscosity characteristics and indicated that a CIL circuit could be successfully operated at a pulp density of up to 60% w/w solids.

These results indicate that blending the oxide ores with Daapleu will be necessary to allow processing at higher slurry densities. Ity Flat rheology results are shown below. Full results of the rheology testwork programme are available.

**Figure 13.6.11 Rheology Results – Composite #3 (Ity Flat)**




## 13.6.10 Carbon Adsorption Tests

A programme of carbon adsorption testwork, comprising sequential triple contact adsorption and equilibrium carbon loading testing was conducted on each of the composites following gravity separation and cyanidation leaching under optimised conditions.

### *Bulk Cyanidation Testwork*

Bulk gravity separation and cyanidation testwork was conducted on each of the six composites under the optimised conditions provided in Table 13.6.6. A summary of the bulk cyanidation testwork results is provided in Table 13.6.9.

**Table 13.6.9 Bulk Cyanidation Leach Results at Optimised Conditions**

| Composite | Grind Size $P_{80}$ (µm) | % Solids (w/w) | Initial NaCN (ppm) | Calculated Head (g/t) | | Gravity Recoverable (%) | | Overall Recovery (%) | | Reagent Consumption | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | NaCN (kg/t) | Lime (kg/t) |
| | | | | Au | Ag | Au | Ag | Au | Ag | | |
| Composite #1 [Daapleu Oxide] | 75 | 50 | 350 | 1.45 | 4.31 | 6.2 | 1.2 | 80.0 | 65.2 | 0.17 | 0.71 |
| Composite #3 [Ity Flat] | 75 | 50 | 250 | 5.29 | 4.71 | 38.4 | 2.4 | 95.8 | 74.5 | 1.26 | 2.27 |
| Composite #4 [Mont Ity] | 75 | 45 | 500 | 3.50 | 7.24 | 38.2 | 2.3 | 95.4 | 79.3 | 1.93 | 1.27 |
| Composite #5 [Walter] | 75 | 50 | 350 | 0.95 | 1.06 | 8.8 | 2.2 | 95.8 | 81.2 | 1.66 | 2.53 |
| Composite #6 [Zia NE] | 75 | 40 | 350 | 1.22 | 2.73 | 15.2 | 1.0 | 95.9 | 56.0 | 0.15 | 3.93 |
| Composite #7 [Daapleu Primary] | 75 | 50 | 350 | 1.77 | 6.03 | 9.7 | 2.5 | 64.3 | 60.2 | 0.10 | 0.33 |

### *Sequential Triple Contact Carbon Adsorption Tests*

A triple contact sequential carbon loading test was conducted on each of the six composites to determine the Fleming k and n values for the CIL system. These constants allow simulation of the CIL circuit and calculation of the expected carbon and solution profiles.

The results of the Fleming constants arising from the triple contact sequential carbon loading kinetic tests are summarised in Table 13.6.10.

**Table 13.6.10 Summary of Fleming Constants**

| Composite | Fleming Rate Constant, k (h-1) | | Fleming Equilibrium Constant, n | | Cumulative Carbon Loading (g Au/t C) | |
|---|---|---|---|---|---|---|
| | Au | Ag | Au | Ag | g Au/t C | g Ag/t C |
| Composite #1 [Daapleu Oxide] | 163 | 116 | 0.753 | 0.638 | 1,373 | 2,449 |
| Composite #3 [Ity Flat] | 114 | 74 | 0.807 | 0.632 | 3,386 | 2,238 |
| Composite #4 [Mont Ity] | 111 | 79 | 0.808 | 0.648 | 1,751 | 3,180 |
| Composite #5 [Walter] | 144 | 87 | 0.716 | 0.656 | 920 | 616 |
| Composite #6 [Zia NE] | 137 | 94 | 0.696 | 0.666 | 717 | 833 |
| Composite #7 [Daapleu Primary] | 205 | 135 | 0.650 | 0.576 | 1,265 | 3,185 |

The carbon loading kinetics are within the range observed in CIL operations. The modelling of the adsorption circuit for the process plant has been based on k and n values for gold of 150 and 0.60, respectively, and for silver values of 100 and 0.50 have been used.

### *Equilibrium Carbon Loading Tests*

A series of carbon equilibrium loading tests were performed on each of the six composites using filtrate (pregnant solution) from the slurry derived from each of the optimised bulk cyanidation tests. The tests involved contacting varying volumes of pregnant solution with a known mass of fresh, pre-attritioned activated carbon (Haycarb YAO 6 x 12) to achieve a desired gold loading range. A summary of the gold and silver equilibrium carbon loading for each composite is provided in Table 13.6.11.

**Table 13.6.11 Summary of Equilibrium Carbon Loading Tests**

| Composite | Equilibrium Carbon Loading (g/t) | | | | | |
|---|---|---|---|---|---|---|
| | Equilibrium Solution 1.00 mg/L | | Equilibrium Solution 0.50 mg/L | | Equilibrium Solution 0.20 mg/L | |
| | Au | Ag | Au | Ag | Au | Ag |
| Composite #1 [Daapleu Oxide] | 3,444 | 3,001 | 2,672 | 2,213 | 1,910 | 1,480 |
| Composite #3 [Ity Flat] | 5,463 | 1,440 | 3,670 | 903 | 2,169 | 488 |
| Composite #4 [Mont Ity] | 5,143 | 2,602 | 3,700 | 1,680 | 2,394 | 942 |
| Composite #5 [Walter] | 2,618 | 2,548 | 2,053 | 1,568 | 1,488 | 826 |
| Composite #6 [Zia NE] | 2,123 | 2,632 | 1,763 | 1,835 | 1,378 | 1,138 |
| Composite #7 [Daapleu Primary] | 2,987 | 3,655 | 2,409 | 2,712 | 1,812 | 1,827 |

Loaded carbon values on operating Au plants range from 300 to 20,000 g/t of carbon, but typically have the range of 1,500 to 4,000 g/t. It is evident from the equilibrium Au loading tests that the results are within industry norms.

### 13.6.11 Cyanide Detoxification Tests

A programme of cyanide detoxification testwork using the $SO_2$ / Air oxidation process has been adopted to determine the reagents and conditions required to achieve the target effluent. In this process a mixture of sulphur dioxide ($SO_2$) and air is used to oxidise free cyanide and weakly complexed cyanide to cyanate (OCN) at pH 8 to 10, using copper as a catalyst. Copper sulphate is added if the tailings do not contain the required amount of copper. Sodium metabisulphite ($Na_2S_2O_5$ or SMBS) solution is added as the source of sulphur dioxide which is liberated according to the following reactions:

$$Na_2S_2O_5 + H_2O \rightarrow Na_2SO_3 + H_2SO_3$$

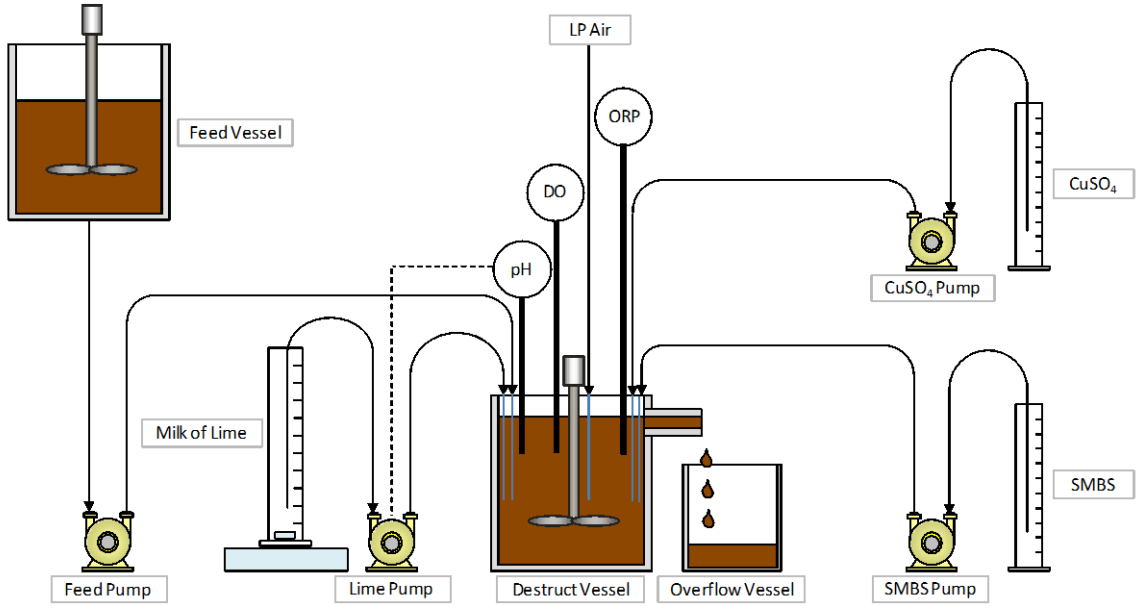
$$Na_2SO_3 + CN^- + O_2 \rightarrow Na_2SO_4 + CNO^-$$

$$H_2SO_3 + CN^- + O_2 \rightarrow H_2SO_4 + CNO^-$$

Sulphuric acid is generated during the process and is neutralised with lime to control the pH between 8.0 and 9.0. All cyano-complexes, with the exception of iron, are destroyed and the metals precipitate as hydroxides, while iron is precipitated as an insoluble ferrocyanide salt. Compressed air or oxygen may be sparged into the tanks to provide the oxygen requirements for the reaction.

A series of cyanide detoxification tests were performed on each of the six composites using slurry from the carbon adsorption tests, previously obtained from the optimised bulk cyanidation leaches. Each sample was analysed for Cu, Fe, Ni, Zn, $CN_{WAD}$, and $CN_{FREE}$ and the remainder of the sample was used as feed to the cyanide detoxification mini plant. A schematic of the cyanide detoxification mini plant is illustrated in Figure 13.6.12.

**Figure 13.6.12 Schematic of Cyanide Detoxification Mini Plant**




A residence time of nominally 60 minutes was selected. A summary of the cyanide detoxification testwork is provided in Table 13.6.12.

**Table 13.6.12 Summary of Cyanide Detoxification Testwork**

| Composite | Test Conditions | | | | | | Solution Assays | |
|---|---|---|---|---|---|---|---|---|
| | Retention Time (minutes) | pH | DO (ppm) | Reagents | | | Detox Feed $CN_{WAD}$ (mg/L) | Treated Effluent $CN_{WAD}$ (mg/L) |
| | | | | $SO_2$ (g/g $CN_{WAD}$) | $CuSO_4.5H_2O$ (mg/L) | Lime (g/g $SO_2$) | | |
| Composite #1 [Daapleu Oxide] | 56 | 8.53 | 4.79 | 4.67 | 76 | 1.91 | 154 | 2.47 |
| Composite #3 [Ity Flat] | 57 | 8.51 | 3.52 | 6.20 | 0 | 0.57 | 282 | 1.55 |
| Composite #4 [Mont Ity] | 101 | 8.60 | 2.69 | 6.00 | 0 | 1.27 | 591 | 4.59 |
| Composite #5 [Walter] | 83 | 8.53 | 2.95 | 6.36 | 0 | 0.83 | 345 | 1.63 |
| Composite #6 [Zia NE] | 57 | 8.52 | 4.45 | 6.00 | 0 | 0.88 | 158 | 2.53 |
| Composite #7 [Daapleu Primary] | 57 | 8.72 | 5.29 | 4.92 | 61 | 0.97 | 120 | 0.55 |

The cyanide detoxification feed and discharge solution assays are summarised in Table 13.6.13.

**Table 13.6.13 Summary of Cyanide Detoxification Solution Assays**

| Composite | | Solution Assay | | | | | |
|---|---|---|---|---|---|---|---|
| | | Cu (mg/L) | Fe (mg/L) | Ni (mg/L) | Zn (mg/L) | $CN_{WAD}$ (mg/L) | $CN_{TOT}$ (mg/L) |
| Composite #1 | Feed | 12.7 | 0.60 | 1.80 | 20.5 | 154 | 156 |
| [Daapleu Oxide] | Discharge | 0.20 | 1.35 | 0.70 | 0.02 | 2.47 | 6.19 |
| Composite #3 | Feed | 266 | 0.13 | 0.43 | 0.05 | 423 | 282 |
| [Ity Flat] | Discharge | 3.92 | <0.10 | <0.05 | 0.04 | 1.55 | 1.83 |
| Composite #4 | Feed | 538 | 0.20 | 0.65 | 1.90 | 591 | 592 |
| [Mont Ity] | Discharge | 17.6 | <0.10 | <0.05 | 0.02 | 4.59 | 4.62 |
| Composite #5 | Feed | 204 | 0.30 | 1.25 | 3.50 | 345 | 346 |
| [Walter] | Discharge | TBA | TBA | TBA | TBA | 1.63 | TBA |
| Composite #6 | Feed | 6.4 | <0.10 | 0.30 | 17.1 | 158 | 158 |
| [Zia NE] | Discharge | 0.12 | 0.65 | 0.50 | <0.02 | 2.53 | 4.32 |
| Composite #7 | Feed | 6.7 | 1.60 | 0.80 | 1.18 | 120 | 124 |
| [Daapleu Primary] | Discharge | 0.13 | 1.00 | 0.15 | <0.02 | 0.47 | 3.22 |

Although the optimised conditions for each composite have not yet been established, the cyanide detoxification work performed to date indicates the following:

- $CN_{WAD}$ concentrations in the discharge liquor following cyanide detoxification for each of the composites may be reduced to <3 ppm $CN_{WAD}$.
- The comparatively high $CN_{WAD}$ concentrations in the Mont Ity and Walter composites (591 mg/L and 345 mg/L) require greater residence times than the nominal 60 minutes required for all remaining composite.
- The target $CN_{WAD}$ value of <5 ppm in the cyanide detoxification discharge liquor is able to be achieved with the addition of 120% to 240% stoichiometric addition of SMBS.
- Soluble copper is present in sufficient concentration to catalyse the cyanide detoxification reactions for all composites except the Daapleu oxide and Daapleu primary composites.

Optimisation of conditions for the cyanide detoxification testwork is currently in progress.

### 13.6.12 Arsenic Precipitation Tests

Analysis of the cyanide detoxification effluent has indicated that the solution from Daapleu oxide and primary composites contains residual arsenic. A programme of arsenic precipitation testwork using ferrous sulphate is currently underway.

## 13.7 Design Criteria Development

The programme of comminution and metallurgical testwork on each of the different pits within the Ity prospect has highlighted a large variation in ore physical characteristics and their metallurgical response. In developing a robust set of plant design criteria, therefore, it is necessary to consider the relative contributions of each material type and their proposed sequence of exploitation throughout the life of mine. The preliminary mine schedule indicates that ore will be mined from several pits simultaneously and thus the process plant will typically receive a blended feed and no one single ore or facies type.

### 13.7.1 Selected Treatment Route

The metallurgical treatment route selected has been based on the results of the current testwork programme with consideration of results from previous testwork programmes and may be summarised as follows:

- Single stage primary crushing.
- SAG and ball mill milling with a recycle pebble crushing facility (SABC circuit).
- Gravity concentration.
- CIL feed thickening.
- Carbon-in-leach (CIL).
- AARL stripping circuit (including Cu removal).
- Air / $SO_2$ cyanide detoxification circuit (including As precipitation).

### 13.7.2 Key Process Design Parameters

The key process design parameters derived from testwork are as follows:

***Milling Circuit Design***

| | |
|---|---|
| SMC A x b | 40.0 |
| Bond Ball Mill Work Index (BWi) | 18.7 kWh/t |
| Levin Test | 10.0 kWh/t |
| Drop Weight Index (DWi) | 8.64 kWh/m³ |
| Abrasion Index (Ai) | 0.30 |
| Crushing Circuit Product Size, $P_{80}$ | 160 mm |

| | |
|---|---|
| Milling Circuit Product Size, $P_{80}$ | 75 µm |
| Gravity Gold Recovery | 31% |
| Gravity Silver Recovery | 20% |
| ***CIL Design*** | |
| Residence Time | 34 hours |
| Pulp Density | 50% w/w solids |
| Cyanide Addition | 0.52 kg/t of ore |
| Lime Addition | 0.88 kg/t of ore |
| pH | 10.5 |
| Carbon loading kinetic parameters, k and n values | 150, 0.60 (gold) |
| | 100, 0.50 (silver) |

### 13.7.3 Metallurgical Recoveries

The range of metallurgical results for the leach tests conducted on each of the 40 variability samples for the six different deposits using a common set of cyanide leach criteria ($P_{80}$ 75 µm, 40% w/w solids pulp density, 500 ppm NaCN) are summarised in Table 13.7.1. Note that the Daapleu variability samples have been divided into 'oxide' and 'primary' based on the nominal arsenic head assay – all samples which assayed less than 1,000 ppm arsenic are classified as 'oxide' and all those assaying greater than 1,000 ppm arsenic as 'primary'. Development of the preliminary mine schedule has differentiated between ‘oxide’ and ‘primary’ Daapleu ore on this basis.

**Table 13.7.1 Variability Tests – Gold and Silver Extractions**

| Pit | No. of Tests | Au Extraction, % | | | Ag Extraction, % | | |
|---|---|---|---|---|---|---|---|
| | | Minimum | Maximum | Average | Minimum | Maximum | Average |
| Daapleu Oxide | 11 | 56.0 | 96.6 | 85.2 | 30.0 | 86.5 | 58.4 |
| Daapleu Primary | 7 | 44.4 | 91.3 | 60.1 | 30.9 | 87.1 | 65.6 |
| Gbeitouo | 4 | 82.6 | 93.4 | 87.7 | 10.6 | 96.2 | 65.5 |
| Ity Flat | 6 | 84.7 | 97.3 | 90.0 | 37.5 | 96.6 | 76.4 |
| Mont Ity | 4 | 77.1 | 98.4 | 88.8 | 67.6 | 93.9 | 81.9 |
| Walter | 3 | 92.3 | 97.3 | 95.6 | 37.0 | 69.0 | 56.2 |
| Zia NE | 5 | 95.6 | 98.0 | 96.7 | 5.1 | 60.7 | 42.2 |

***Testwork Results Applied to the Mine Schedule***

The approach taken to predict the metallurgical recoveries for gold and silver is to take the average extractions obtained from the leach variability tests and then apply these to the mine schedule for the appropriate ore type to yield a weighted recovery value for each year. As silver values have not been modelled in the mine schedule, for the purposes of obtaining a predicted silver recovery it has been assumed that the silver grade will typically be three times that of the corresponding gold value (silver : gold ratios of between 2 and 4 were recorded from the variability head assays).

Table 13.7.2 presents a summary of the 3 Mtpa mine schedule, tonnage contribution from each pit, and gold and silver extractions obtained from the variability testwork for each ore source.

**Table 13.7.2 Mine Schedule and Gold and Silver Extractions**

| Pit | LoM Au (g/t) | LoM Ag (g/t) | % of LoM Tonnage | LoM Tonnes (Mt) | Overall Au Extraction (%) | Ag Extraction (%) |
|---|---|---|---|---|---|---|
| Daapleu Oxide | 1.05 | 3.14 | 31.5 | 12.93 | 85.2 | 58.4 |
| Daapleu Primary | 2.43 | 7.29 | 15.5 | 6.42 | 60.1 | 65.6 |
| Gbeitouo | 1.35 | 4.05 | 6.3 | 2.58 | 87.7 | 65.5 |
| Ity Flat | 2.09 | 6.27 | 3.9 | 1.60 | 90.0 | 76.4 |
| Mont Ity | 2.23 | 6.68 | 5.4 | 2.23 | 88.8 | 81.9 |
| Walter | 1.22 | 3.66 | 4.5 | 1.86 | 95.6 | 56.2 |
| Zia NE | 1.24 | 3.71 | 11.8 | 4.84 | 96.7 | 42.2 |
| Stockpiles | 1.08 | 3.25 | 21.0 | 8.60 | 86.3 | 63.7 |

Gold and silver extraction values from the stockpile reserves have been assumed to be the average of the extractions across all pits. These extraction values may be a slightly understated as the stockpile reserves will be predominantly oxide material.

Using this information, and assuming soluble gold and silver losses of 0.01 mg/L and 0.03 mg/L respectively and a CIL pulp density of 50% w/w solids, it is possible to predict the gold and silver recoveries for each year of the life-of-mine. These recoveries are summarised in Table 13.7.3.

**Table 13.7.3 Predicted Life of Mine Metallurgical Recoveries**

| Mine Year | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Au Extraction, % | 78.4 | 76.8 | 80.1 | 81.2 | 79.4 | 85.0 | 86.7 | 88.9 | 87.6 | 83.7 | 83.7 | 83.7 | 83.7 | 84.6 |
| Ag Extraction, % | 62.6 | 64.1 | 64.3 | 69.1 | 61.8 | 61.8 | 60.3 | 55.7 | 54.8 | 57.9 | 57.9 | 57.9 | 57.9 | 57.6 |

This yields life-of-mine average gold and silver recoveries of 83.1% and 60.2%, respectively.

### 13.7.4 Overall Metallurgical Recovery

The gold and silver recovery is a statement of the anticipated metallurgical extraction for the Ity CIL circuit and does not include the additional benefit that a gravity circuit may offer prior to CIL (i.e. the gold and silver recovery data used for this prediction were taken from the variability testwork programme and did not include a gravity circuit prior to cyanidation). Consequently, it is not a statement of the anticipated plant performance. Consideration must be given to soluble losses, potential for short circuiting in the leach circuit and other associated plant problems that may impact the overall plant gold and silver recovery.

Based on the gold and silver metallurgical recovery values obtained from the mine schedule based recoveries (83.1% and 60.2%), it is reasonable to predict that the overall gold and silver extractions for the life-of-mine will be between 78% and 89% for gold and between 55% and 69% for silver.

### 13.7.5 Reagent Consumption

***CIL Circuit***

A summary of the average lime and cyanide consumption data for the 40 variability leach tests conducted for each of the six different Ity pits is provided in Table 13.7.4. All variability leach tests were performed using Perth tap water.

**Table 13.7.4 Variability Leach Tests – Average Lime and Cyanide Consumption**

| Pit | NaCN Consumption kg/t | Lime[1] Consumption kg/t |
|---|---|---|
| Daapleu | 0.36 | 0.39 |
| Gbeitouo | 0.75 | 1.92 |
| Ity Flat | 1.82 | 1.56 |
| Mont Ity | 2.10 | 1.73 |
| Walter | 1.13 | 4.58 |
| Zia NE | 0.56 | 4.21 |

**Note: 1. Testwork conducted using commercial lime with 60% available CaO.**

Having performed a series of cyanide leach tests at optimum leach conditions as part of the optimisation testwork programme (refer to Table 13.6.6), a revised set of lime and cyanide consumption data has been generated, as summarised in Table 13.7.5. All leach tests performed under optimised leach conditions for each composite used Perth tap water.

**Table 13.7.5 CIL Lime and Cyanide Consumption**

| Composite | NaCN Consumption kg/t | Lime[1] Consumption kg/t |
|---|---|---|
| Composite #1 (Daapleu Oxide) | 0.17 | 0.71 |
| Composite #2 (Gbeitouo)[2] | 0.75 | 1.92 |
| Composite #3 (Ity Flat) | 1.26 | 2.27 |
| Composite #4 (Mont Ity) | 1.93 | 1.27 |
| Composite #5 (Walter) | 1.66 | 2.53 |
| Composite #6 (Zia NE) | 0.15 | 3.93 |
| Composite #7 (Daapleu Primary) | 0.10 | 0.33 |

**Note: 1. Testwork conducted using commercial lime with 60% available CaO.**

**Note: 2. Values for Gbeitouo are based on the Variability leach results (due to lack of sample to form Composite #2).**

***Anticipated Plant Reagent Requirements***

The anticipated plant lime addition rate to the CIL circuit differs from the testwork results as the testwork was conducted with commercial lime having a CaO availability of approximately 60%. The lime supplied is likely to have an available CaO content in excess of 90% and the required plant lime addition has been re-calculated accordingly. Based on this re-calculated value and the relative ore contribution from each pit to the overall Ity reserve, the expected life-of-mine sodium cyanide and lime consumptions for the selected treatment route may be calculated.

A summary of the life-of-mine sodium cyanide and lime consumptions is provided in Table 13.7.6.

**Table 13.7.6 CIL Lime and Cyanide Consumption**

| Composite | Tonnage % of Ity Reserve | NaCN Consumption kg/t | Lime[1] Consumption kg/t |
|---|---|---|---|
| Composite #1 (Daapleu Oxide) | 5.1 | 0.17 | 0.48 |
| Composite #2 (Gbeitouo)[2] | 9.5 | 0.75 | 1.28 |
| Composite #3 (Ity Flat) | 6.6 | 1.26 | 1.51 |
| Composite #4 (Mont Ity) | 9.8 | 1.93 | 0.84 |
| Composite #5 (Walter) | 5.8 | 1.66 | 1.69 |
| Composite #6 (Zia NE) | 13.2 | 0.15 | 2.62 |
| Composite #7 (Daapleu Primary) | 50.0 | 0.10 | 0.22 |
| **Life-of-Mine** | **100** | **0.52** | **0.88** |

**Note: 1. Lime consumption based on 90% available CaO.**

**Note: 2. Values for Gbeitouo are based on the Variability leach results (due to lack of sample to form Composite #2).**

## 13.8 Refractory Testwork Programme

A separate Refractory testwork programme was implemented on the Daapleu primary ore composite (Composite #7) only to establish any specific processing requirements for this refractory material. Specifically the programme involved performing the following:

- Whole-of-ore cyanidation leach tests at $P_{80}$ grind sizes of 75 µm, 45 µm, 25 µm, and 15 µm to establish the effect of grind size on gold and silver recovery.
- Batch flotation tests at $P_{80}$ grind sizes of 125 µm, 106 µm, and 75 µm to establish optimum grind size for maximum sulphur recovery.
- Bulk flotation at the optimum grind size to generate a flotation concentrate for subsequent testing.
- Bulk flotation concentrate cyanidation leach tests (4 off) at four different concentrate regrind $P_{80}$ values of 75 µm (as-received), 45 µm, 25 µm, and 15 µm.
- Bulk flotation concentrate roast followed by a cyanidation leach test (1 off) on the roast product.
- Bulk flotation concentrate pressure oxidation (POX) leach test at $P_{80}$ values of 45 µm, 25 µm, and 15 µm, followed by cyanidation leaching of the residue from each POX leach test.
- Bulk flotation concentrate bacterial oxidation (BIOX) amenability test followed by cyanidation of the BIOX product.
- Cyanidation leach of the bulk flotation tails.

### 13.8.1 Whole-of-Ore Leaching at Different Grind Sizes

Separate samples of the Daapleu primary ore composite (Composite #7) were subjected to ultra-fine grinding in a vertically stirred bead mill to achieve target particle size $P_{80}$ values of 75 µm, 45 µm, 25 µm, and 15 µm. The sizing was confirmed using a Malvern laser sizer.

Overall gold and silver extraction results, together with reagent consumptions, for each of the whole-of-ore leach tests performed at the four different grind sizes are shown in Table 13.8.1.

**Table 13.8.1 Effect of Grind Size on Whole-of-Ore Cyanidation Leach Extraction**

| Grind Size $P_{80}$ (µm) | Au Extraction | | | Ag Extraction | | | Reagent Consumption | |
|---|---|---|---|---|---|---|---|---|
| | Calculated Head (g/t) | Leach Residue (g/t) | Overall Extraction (%) | Calculated Head (g/t) | Leach Residue (g/t) | Overall Extraction (%) | NaCN (kg/t) | Lime (kg/t) |
| 75 | 1.61 | 0.59 | 63.3 | 4.64 | 0.90 | 80.6 | 1.13 | 0.20 |
| 45 | 1.64 | 0.49 | 70.1 | 5.43 | 0.90 | 83.4 | 1.58 | 0.55 |
| 25 | 1.60 | 0.48 | 69.9 | 5.34 | 0.60 | 88.8 | 1.60 | 0.65 |
| 15 | 1.76 | 0.44 | 75.0 | 5.04 | 0.30 | 94.1 | 2.76 | 0.85 |

Graphical presentations of the whole-of-ore cyanidation leach gold and silver extractions at the four different grind sizes are presented in Figure 13.8.1 and Figure 13.8.2, respectively.

**Figure 13.8.1 Whole-of-Ore Cyanidation Leach – Grind Size vs. Au Extraction**

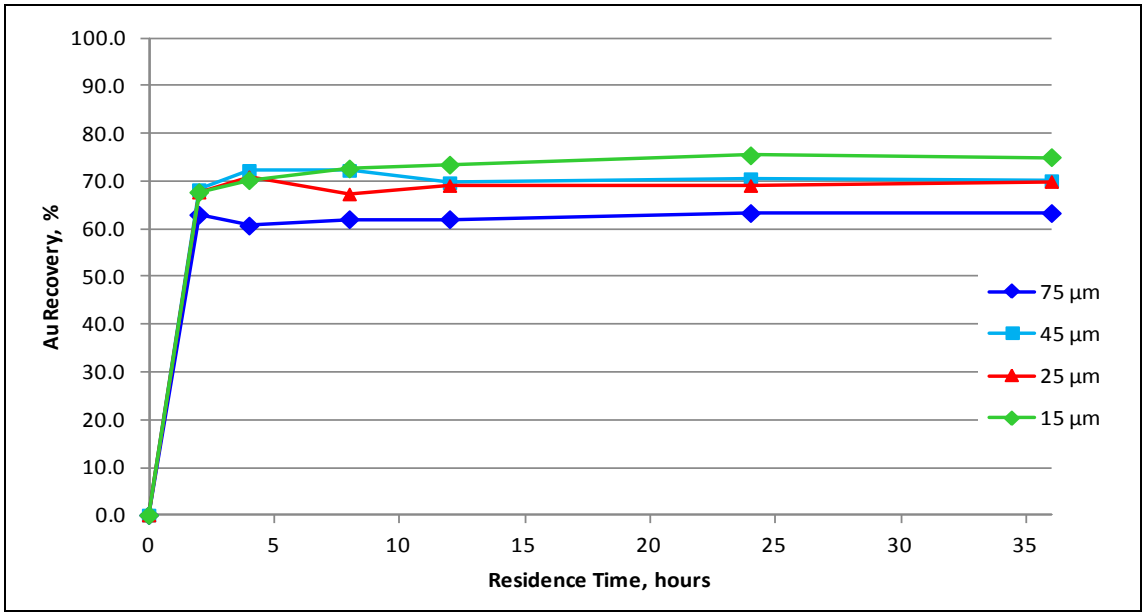
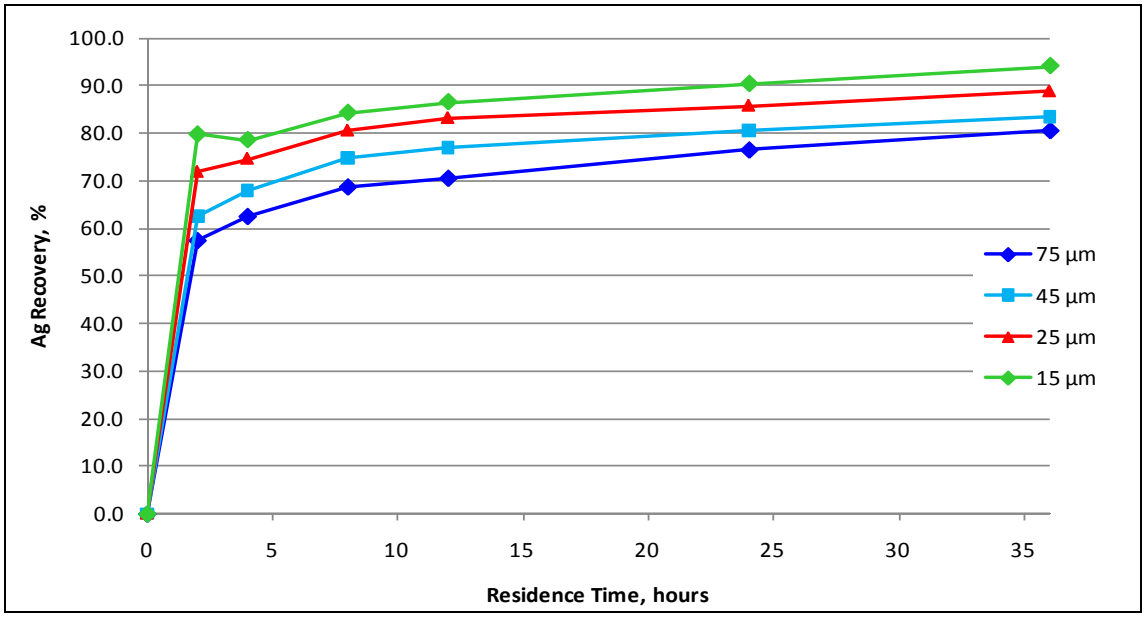


**Figure 13.8.2 Whole-of-Ore Cyanidation Leach – Grind Size vs. Ag Extraction**




The testwork results indicate that the gold and silver extractions increase with increasing fineness of grind – gold extraction increases from 63.3% at a $P_{80}$ of 75 µm to 75.0% at a $P_{80}$ of 15 µm, while silver shows a similar trend, increasing from 80.6% to 94.1% overall extraction. It may be concluded that there is benefit in grinding to a $P_{80}$ of 45 µm but there is little improvement in gold extraction unless much finer grinding is performed (to a $P_{80}$ of 15 µm).

### 13.8.2 Whole-of-Ore and Flotation Concentrate UFG Comparison

Following the improvement in gold and silver recovery with decreasing particle size, four additional tests were performed on the refractory Composite #7 as follows:

- Whole-of-ore direct cyanidation at a grind $P_{80}$ of 10 µm.
- Cyanidation of a bulk flotation tails, at a nominal $P_{80}$ of 75 µm.
- Cyanidation of the equivalent bulk flotation concentrate, ground to a $P_{80}$ of 10 µm.
- Cyanidation of the equivalent bulk flotation concentrate, ground to a $P_{80}$ of 25 µm.

This enabled a direct comparison to be made of the potential benefit of installing a flotation circuit and concentrate fine grinding prior to cyanidation.

The results of the 10 µm whole-of-ore leach, 10 µm flotation concentrate leach, 25 µm flotation concentrate leach, and 75 µm flotation tails leach are summarised in Table 13.8.2.

**Table 13.8.2 Effect of Grind Size on Whole-of-Ore Cyanidation Leach Extraction**

| Test | Grind Size $P_{80}$ (µm) | Gold | | | Silver | | | Reagent Consumption | |
|---|---|---|---|---|---|---|---|---|---|
| | | Calc'd Head (g/t) | Leach Residue (g/t) | Overall Extr'n (%) | Calc'd Head (g/t) | Leach Residue (g/t) | Overall Extr'n (%) | NaCN (kg/t) | Lime (kg/t) |
| Whole-of-Ore | 10 | 1.73 | 0.35 | 79.8 | 6.23 | 0.60 | 90.4 | 4.20 | 1.85 |
| Flotation Con | 10 | 25.0 | 5.23 | 79.1 | 84.0 | 12.0 | 85.7 | 36.8 | 2.50 |
| Flotation Con | 25 | 25.2 | 7.70 | 69.5 | 59.2 | 4.80 | 91.9 | 28.6 | 1.80 |
| Flotation Tails | 75 | 0.26 | 0.05 | 80.8 | 2.56 | 0.60 | 76.6 | 3.48 | 0.75 |

From these tests the following may be concluded:

- The whole-of-ore leach test conducted at a $P_{80}$ of 10 µm yields gold and silver recoveries of 79.8% and 90.4%, respectively.
- Cyanidation of a re-ground (10 µm) bulk flotation concentrate sample achieved gold and silver extractions of 79.1% and 85.7%, respectively. Cyanidation of a re-ground (25 µm) bulk flotation concentrate sample achieved gold and silver extractions of 69.5% and 91.9%, respectively.
- Cyanidation of a sample of bulk flotation tailings achieved gold and silver extractions of 80.8% and 76.6%, respectively.
- Based on the mass pull to flotation concentrate and on the relative metal extractions for both the flotation 10 µm re-ground concentrate and flotation tailings cyanidation tests, an overall gold extraction of 79.3% may be calculated, with 85.3% overall silver extraction. By comparison, overall gold and silver extractions for the 25 µm re-ground concentrate and flotation tailings cyanidation tests of 70.5% and 90.5%, respectively, were achieved.

### 13.8.3 Flotation Grind Optimisation

A series of flotation tests were performed on Composite #7 to establish the effect of grind size on flotation recovery of gold and sulphur. The tests were performed at three different grind sizes ($P_{80}$ of 125 µm, 106 µm, and 75 µm) aimed at maximising the recovery of gold-bearing sulphides.

A summary of the various flotation recoveries and mass pulls to concentrate at the three grind sizes is presented in Table 13.8.3.

**Table 13.8.3 Effect of Grind Size on Flotation Recovery**

| Grind Size $P_{80}$ (µm) | Mass Pull (%) | Gold | | | | Sulphur | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | Head Assay (ppm) | Calc. Head (ppm) | Flotation Con (% dist) | Flot Con Assay (g/t) | Head Assay (ppm) | Calc. Head (ppm) | Flotation Con (% dist) | Flot Con Assay (%) |
| 125 | 8.39 | 1.69 | 1.67 | 84.1 | 16.8 | 1.84 | 1.79 | 96.9 | 20.6 |
| 106 | 6.63 | 1.69 | 1.77 | 86.8 | 23.1 | 1.84 | 1.79 | 97.9 | 26.5 |
| 75 | 6.51 | 1.69 | 1.87 | 88.5 | 25.4 | 1.84 | 1.74 | 99.5 | 26.6 |

The results indicate the following:

- Sulphur recovery to the flotation concentrate for each test was consistently high and increased with increasing fineness of grind.
- Gold recovery followed a similar trend – a final gold recovery to flotation concentrate of 88.5% was achieved at a $P_{80}$ of 75 µm compared to 84.1% at 125 µm.
- The mass pull to flotation concentrate decreased with increasing fineness of grind.

Based on this information, flotation at a $P_{80}$ of 75 µm achieved the lowest mass pull (only 6.5% of the total feed reported to the flotation concentrate) and the highest gold recovery (88.5%). Consequently a $P_{80}$ of 75 µm was selected as the grind size for the bulk flotation testwork to generate a concentrate for subsequent UFG, roast, POX, and BIOX amenability testing.

### 13.8.4 Bulk Flotation

Bulk rougher flotation tests were performed on a single 51 kg sample of Composite #7 to provide flotation concentrate upon which to conduct amenability testwork. The bulk flotation test was performed at a $P_{80}$ of 75 µm at a 35% w/w solids pulp density in a 120 litre Agitair laboratory flotation cell for 32 minutes. Collector (PAX) and frother (MIBC) additions were 80 g/t and 5 g/t, respectively. No copper sulphate activator was used. A summary of the results is provided in Table 13.8.4.

**Table 13.8.4 Bulk Flotation Results Summary**

| Grind Size $P_{80}$ (µm) | Mass Pull (%) | Gold | | Silver | | Arsenic | | Sulphur | | Copper | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Head Assay (g/t) | Dist'n. (%) | Head Assay (g/t) | Dist'n. (%) | Head Assay (%) | Dist'n. (%) | Head Assay (%) | Dist'n. (%) | Head Assay (g/t) | Dist'n. (%) |
| Ro Con 1 | 2.83 | 38.4 | 61.7 | 98 | 56.6 | 5.20 | 72.1 | 47.2 | 74.7 | 680 | 37.3 |
| Ro Con 2 | 1.31 | 32.6 | 24.4 | 72 | 19.3 | 3.08 | 19.9 | 26.8 | 19.7 | 640 | 16.3 |
| Ro Con 3 | 0.54 | 10.3 | 3.18 | 36 | 3.99 | 1.04 | 2.77 | 5.44 | 1.66 | 360 | 3.79 |
| Ro Con 4 | 0.47 | 5.87 | 1.58 | 28 | 2.71 | 0.48 | 1.12 | 2.62 | 0.70 | 240 | 2.20 |
| Tail | 94.8 | 0.17 | 9.16 | 0.9 | 17.4 | 0.01 | 4.18 | 0.06 | 3.19 | 22 | 40.4 |

The low mass pull to concentrate (5.16% w/w) yielded approximately 2.6 kg of concentrate with which to perform subsequent pressure oxidation (POX) leaching tests and a roast amenability test. Insufficient concentrate sample was generated to perform BIOX amenability testwork for this current refractory testwork programme.

### 13.8.5 Flotation Concentrate Pressure Oxidation Leaching

A pressure oxidation (POX) leach test was performed on a sample of flotation concentrate in a Parr laboratory autoclave at 220°C for 60 minutes, and with an oxygen overpressure of 800 kPa. The POX leach test was preceded by pre-acidulation. Preliminary results of the gold extractions from the subsequent cyanidation of the POX leach product indicate a stage extraction of 93%. The POX amenability testwork results will be reported in an addendum to this report.

### 13.8.6 Flotation Concentrate Roasting

A sample of flotation concentrate was subjected to a 2-stage roast amenability test followed by cyanidation of the calcine product. The first stage roast was performed at 500°C for two hours followed by the second stage conducted at 650°C for a further two hours. Preliminary results of the gold extractions from the subsequent cyanidation of the calcine product indicate a stage extraction of 90%. The calcine amenability testwork results will be reported in an addendum to this report.

### 13.8.7 Refractory Program – Interim Conclusions

At the time of writing the results of the refractory program had not been completed. However the following interim conclusions can be drawn:

- Whole of ore grinding to finer sizes and subsequent leaching improves gold and silver extraction with gold extraction improving from 63 to 75 % and silver improving from 80 to 94% by grinding to 15 microns.
- Flotation of the whole ore composite at three different grind sizes indicated grinding to finer sizes improves gold recovery to concentrate from 84 to 88.5 % by grinding to 75 microns. There was no consistent change in silver recovery.
- Bulk Flotation of the whole ore composite at a grind of 75 microns recovered 90.8% of the gold and 82.6% of the silver to 5.2% of the mass.
- Regrinding of the flotation concentrate followed by separate leaching of the flotation concentrate and tail showed that overall gold and silver extraction could be improved. Combined gold extraction improved from 63% to 79.3% by regrinding the concentrate to 10 microns. Similarly, silver extraction improved from 80.6% to 85.3%.
- Pressure oxidation of the flotation concentrate indicated a gold extraction of 94% giving a combined concentrate/tail extraction of 91.8%.
- Roasting of the flotation concentrate indicated a gold extraction of 90% giving a combined concentrate / tail extraction of 88.6%.
- At this point the flotation / regrind / leach option offers an opportunity for improved extraction at modest cost and will be pursued further in the next stage of development.

## 13.9 Tailings Sample Preparation

Cyanidation tailings consultant testwork was performed on a 'Master Composite' prepared at ALS. A 20 kg sub-sample of master composite was prepared according to the following recipe:

- Composite #1 (Daapleu Oxide) 5.1 kg
- Composite #3 (Ity Flat) 1.8 kg

- Composite #4 (Mont Ity 1.0 kg
- Composite #5 (Walter) 1.2 kg
- Composite #6 (Zia NE) 3.3 kg
- Composite #7 (Daapleu Primary) 7.6 kg

One kilogram sub-samples of each selected composite were ground to the target grind size $P_{80}$ of 75 µm using a laboratory rod mill at 50% w/w solids, and then filtered in a laboratory pressure filter. The solids filter cakes were then combined according to the master composite recipe and subjected to a bulk cyanidation leach under the same conditions used for the variability leach tests.

Sub-samples of the cyanidation tailings slurries were then dispatched to the Knight Piésold laboratory in East Perth, for geochemical testwork (30 litres), and the Outotec Laboratory in West Perth, for thickening and filtration testwork (15 litres).

The testwork results from Knight Piésold are reported in Section 17. The Outotec thickening and filtration test report is still outstanding and will be provided in an addendum to the metallurgical testwork report.

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# 14.0 MINERAL RESOURCE ESTIMATES

## 14.1 Introduction

The Ity CIL Project consists of a total of nine prospect areas which include four in situ gold deposits Mont Ity, Ity Flat, ZiaNE and Walter that have been or are currently in production plus two near mine in situ deposits Gbeitouo and Daapleu, two rock waste dumps Teckraie and Verse Ouest and a discontinued heap leach pad Aires. The nine main prospects lie within an area of approximately 5 km (East) by 3 km (North).

The Indicated and Inferred Mineral Resource estimated by Cube have incorporated all RC, DC and AC drilling completed at the Ity CIL Project.

Mineral Resources were prepared by Mark Zammit, Principal Consultant Geologist, a full-time employee of Cube Consulting. Cube is independent of Endeavour.

## 14.2 Previous Resource Estimate

The previous Mineral Resource estimate for the Ity Gold Mine reported on 30 September 2015 contained 1,190,600 oz. Au in the Measured category, 1,729,000 oz. Au in the Indicated category and 458,300 oz. Au, in the Inferred category. This Mineral Resource was reported inside an optimised pit shell using the following parameters:

- Overall pit slope of approximately 30 to 40 degrees.
- Commodity price of USD $1,500/oz Au.
- Process recovery between 73% and 96%.
- Process cost between USD $12.1/t and USD $16.5/t.
- Refinery, selling and royalty costs of 4% of sell prices.

**Table 14.2.1 Summary of September 2015 Mineral Resource Estimate**

| Deposit | Cut-off Grade (g/t Au) | Measured | | | Indicated | | | Inferred | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | kt | g/t | Au koz | Kt | g/t | Au oz | kt | g/t | Au koz |
| Mont Ity | 0.5 | | | | 5,100 | 2.35 | 385.6 | 140 | 2.75 | 12.4 |
| Daapleu | 0.5 | 21,188 | 1.45 | 984.7 | 9,604 | 1.46 | 452 | 1,553 | 1.21 | 60.5 |
| ZiNE | 0.5 | | | | 6,741 | 1.56 | 337.9 | 3,838 | 1.78 | 219.8 |
| Walter | 0.5 | | | | 2,613 | 2.24 | 188.5 | 200 | 1.42 | 9.1 |
| Gbeitouo | 0.8 | | | | 1,954 | 2.30 | 144.7 | 57 | 1.29 | 2.3 |
| Aires | 0.0 | 6,134 | 1.04 | 205.9 | | | | | | |
| Verse Ouest | 0.0 | | | | 3,844 | 1.22 | 150.8 | 3,591 | 1.25 | 144.3 |
| Teckraie | 0.0 | | | | 1,945 | 1.11 | 69.5 | 304 | 1.01 | 9.9 |
| **Total** | | **27,322** | **1.36** | **1,190.6** | **31,801** | **1.69** | **1,729** | **9,682** | **1.47** | **458.3** |

## 14.3 Data Supplied

Data supplied to Cube by Endeavour included the following:

- Drill hole data in the form of Microsoft Excel and Access files.
- Historical interpretation files.
- Previous block models.
- Topographical survey (carrières_fin_mai_2016.dtm) representing the mining surface as at 31 May 2016.

## 14.4 Geological Interpretation and Modelling

### 14.4.1 Lithology Domains

For all situ gold deposits, lithological interpretations were undertaken by Cube and provided to Endeavour. The interpretations are based on geological logging codes stored in the 'LITHO' field in the 'Geology' database table. The interpretation polylines were 'snapped' to drillholes traces in most instances and used to create validated three dimensional solids (3DMs). The lithology 3DMs was used for direct assignment of lithology facies into the block model. Often the 3DMs were constructed with overlapping boundaries in some places to ensure no 'gaps' of unassigned facies occurred in the block model.

At the southern limit of the current Mont Ity / Ity Flat pit, a DTM surface was used to define the base of the low grade stockpile. A topography DTM surface was created by combining the current surveyed topography and the collar position for the historical drilling to re-create a pre-mining depletion topographical surface. The two DTM surfaces were then used for the assignment of the low grade stockpile (Facies 331).

The lithological facies types and corresponding block model assignment is summarised in Table 14.4.1. Cross sections through each of the in situ deposits and plan images of the rock dumps and heap leach pad are shown in Figure 14.4.1 to Figure 14.4.7.

**Table 14.4.1 Lithological Facies and Block Model Assignment**

| Deposit | Lithology | Block Model Assignment (facies) |
|---|---|---|
| All | Soil | 311 |
| All | Laterite | 301 |
| All | Reduced Clay | 211 |
| All | Oxidised Clay | 201 |
| All | Volcano Sediments | 101 |
| Mont Ity / Ity Flat | SE Low Grade Rock Dump | 331 |
| All | Carbonate | 401 |
| All | Granodiorite | 501 |
| All | Mafic Intrusive | 811 |
| All | Daaplite | 601 |
| Teckraie and Verse Ouest | Rock Dump | 701 |
| Aires | Heap Leach | 701 |
| All | Mafic Intrusive | 811 |
| All | Skarn | 901 |

**Figure 14.4.1 Daapleu Lithology Interpretation – Section 1250 Looking Northeast**

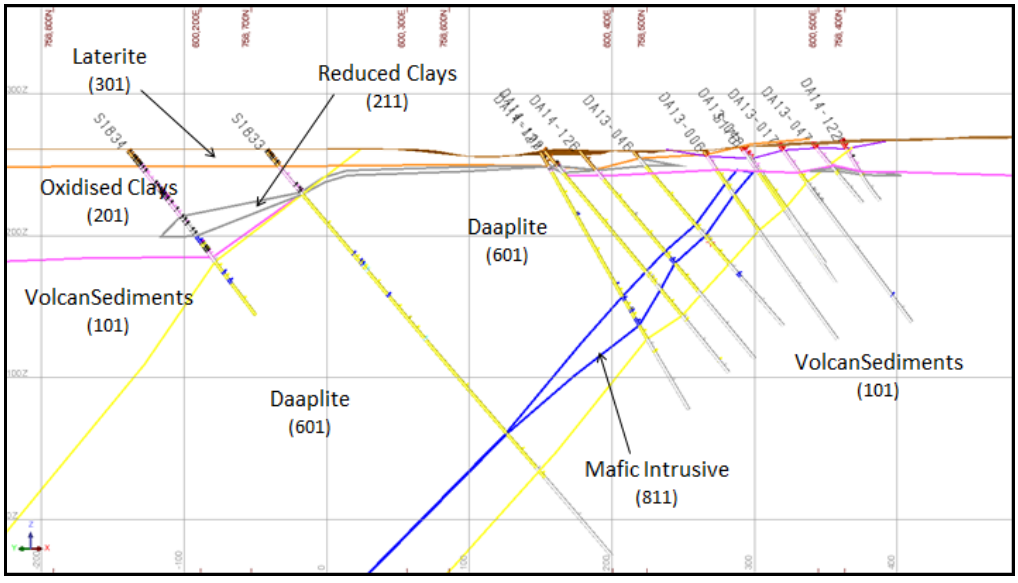


**Figure 14.4.2 Gbeitouo Lithology Interpretation – Section 280 Looking Northeast**

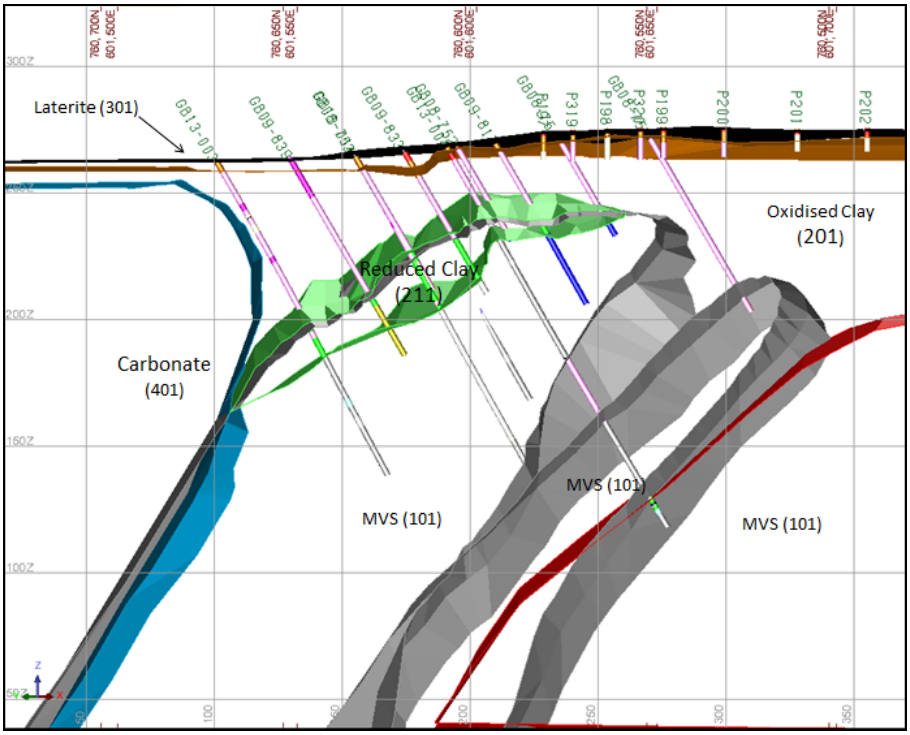



**Figure 14.4.3 Mont Ity / Ity Flat Lithology Interpretation – Section 450 Looking Northeast**

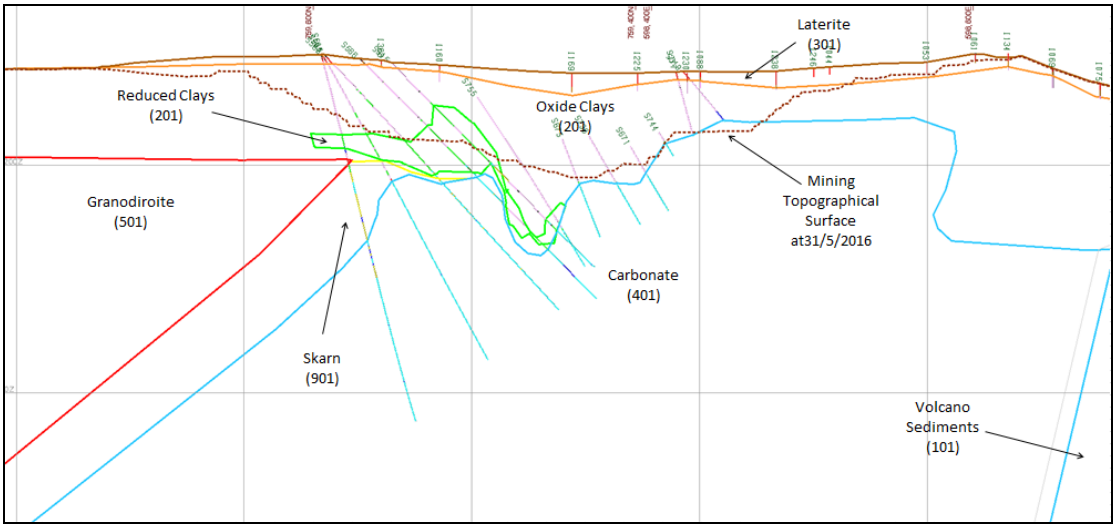


**Figure 14.4.4 Walter Lithology Interpretation – Section 100 Looking Northeast**

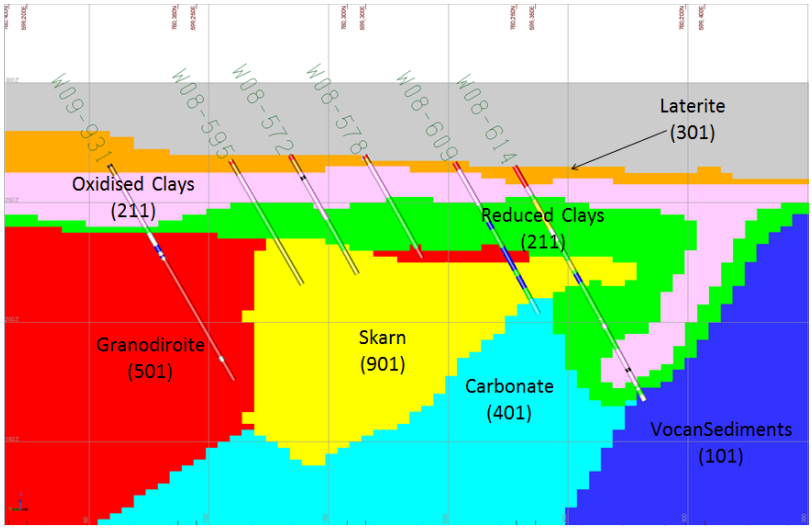



**Figure 14.4.5 ZiaNE Lithology Interpretation – Section -200 Looking Northeast**

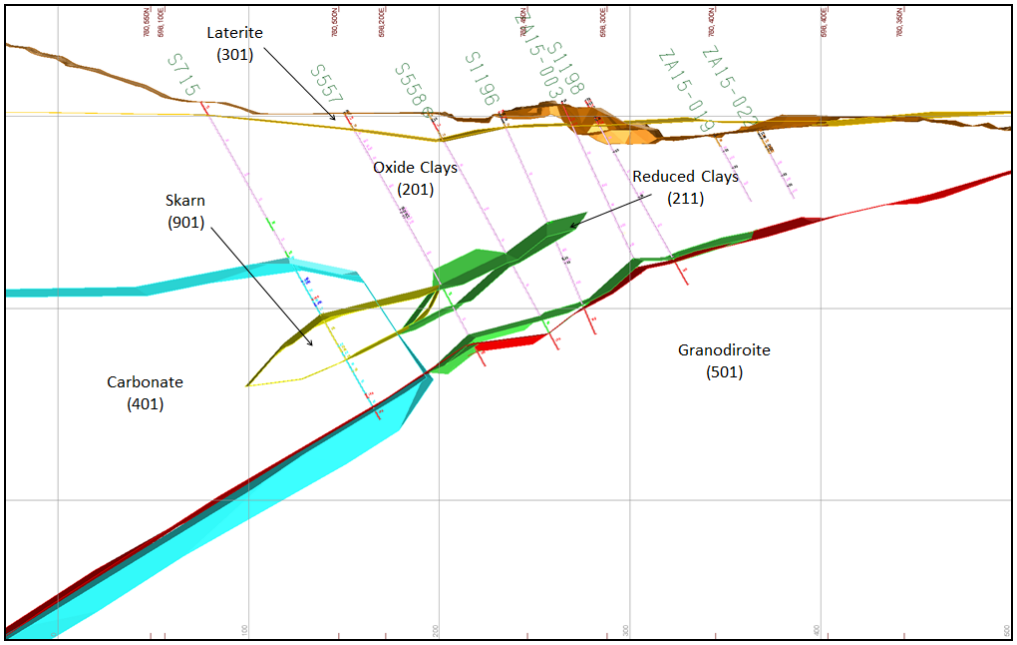


**Figure 14.4.6 Teckraie and Verse Ouest Facies Interpretation – Plan View**

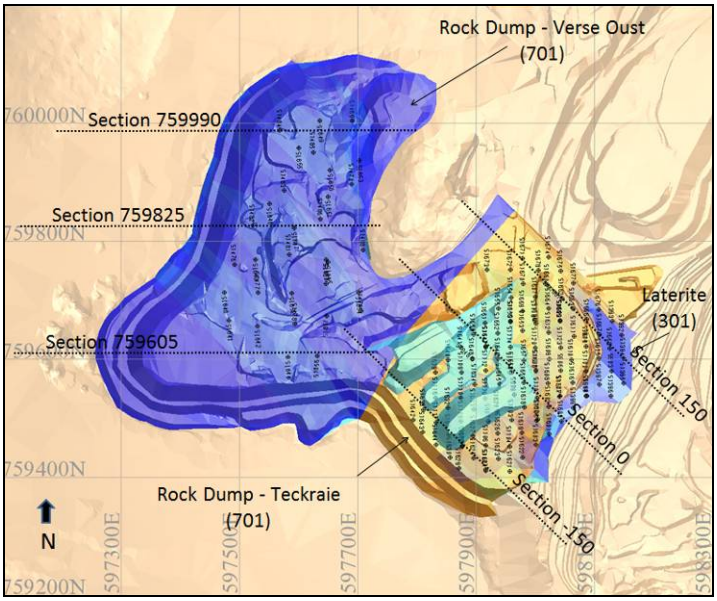
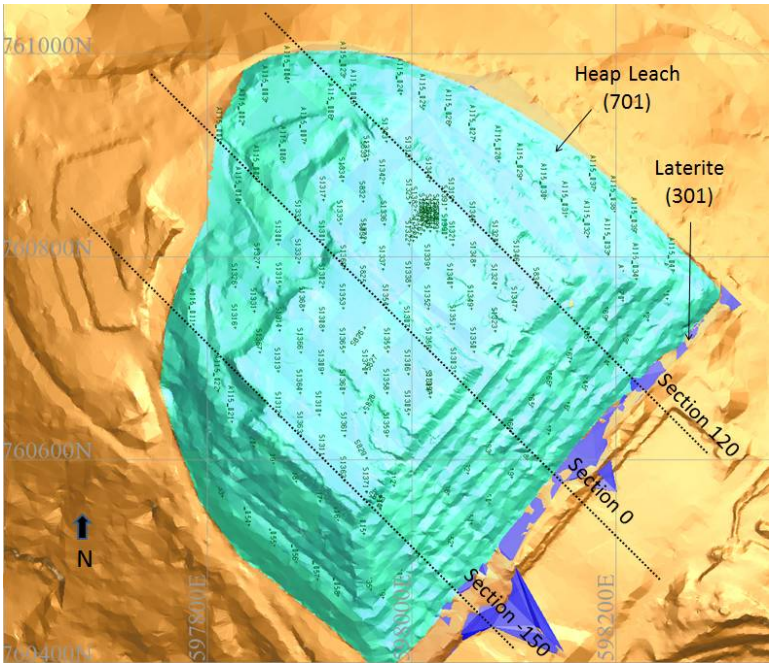



**Figure 14.4.7 Aires Facies Interpretation – Plan View**



### 14.4.2 Weathering Domains

For all in situ gold deposits, three weathering domains were interpreted by Cube defining Oxide, Transitional and Fresh rock. The interpretation was based on geological logging codes stored in the 'GL_OX_SUPERGENE' field in the 'Geology' database table. Two 3DM wireframes were created for each deposit to define the Oxide and Fresh weathering domains. The Transitional material was defined as the remaining volume of material outside the Oxide and Fresh domains. The wireframes were 'snapped' to drillhole traces for all drillholes in the estimation area. The block model assignment for weathering is summarised in Table 14.4.2.

**Table 14.4.2 In Situ Deposit Weathering Interpretation and Block Model Assignment**

| Weathering | Block Model Assignment (ox_state) |
|---|---|
| Air | 0 |
| Oxide | 3 |
| Transition | 2 |
| Fresh | 1 |

For the waste rock dumps and heap leach pad, two weathering domains were based on the Facies interpretation and defined as Oxide and Laterite rock. Oxide material includes both the heap leach, rock dump and all other material below the interpreted laterite domain. The block model assignment for weathering is summarised in Table 14.4.3.

**Table 14.4.3 Rock Dump and Leach Pad Weathering Interpretation and Block Model Assignment**

| Weathering | Block Model Assignment (ox_state) |
|---|---|
| Oxide | 3 |
| Laterite | 4 |

### 14.4.3 Mineralisation Domains

The following is a description of the approach to mineralisation domaining for each of the deposits.

Trend surfaces have also been created for each of the in situ deposits to define the orientation of mineralisation for later use in controlling the local rotation of the variogram models and search neighbourhoods during the kriging process. In addition, wireframe solids defining the outer limit of drill coverage for each deposit was also modelled to limit the extent of the background estimate.

***Daapleu***

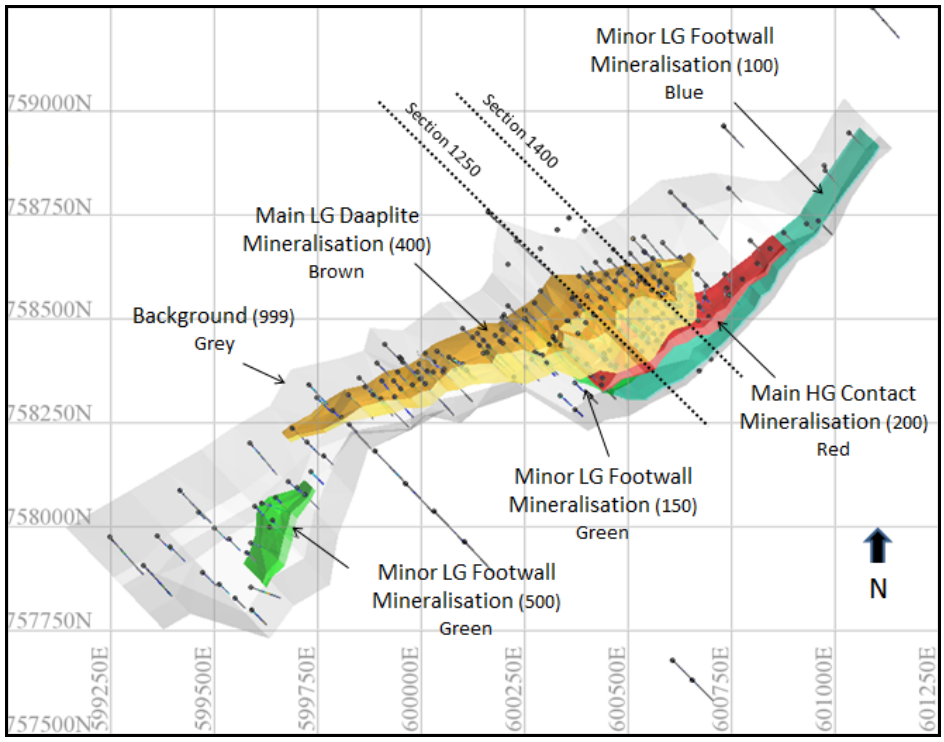
The mineralisation at Daapleu was interpreted as sectional polygons based primarily on a nominal 0.35 ppm Au grade cut-off which appears to approximate the natural grade cut-off off between mineralisation and waste rock. In addition, relevant drill log information and the lithological model were used as guidance for the interpretation. Six mineralised domains have been modelled and are grouped into four main lithological types. Domain 400 is the largest mineralised domain by volume and is identified by typically lower grade stockwork style of mineralisation hosted within the Daaplite. Domain 200 is characterised by significantly higher grade mineralisation occurring within a sheared envelope at the contact between the Daaplite and Volcano Sediments. Domains 100, 150, and 500 represent lower grade domains located within the footwall Volcano Sediments and are likely to represent structurally controlled mineralisation occurring sub-parallel to the dominant Domain 400. Mineralisation contained with the laterite horizon has also been modelled separately.

A summary of the relevant wireframe models is listed in Table 14.4.4 and a plan 3D image of the wireframe model is shown in Figure 14.4.8.

**Table 14.4.4 Daapleu Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Laterite Mineralisation | daapleu_mz_au_20160618 | 301 | 301 |
| Main LG Daapleu Mineralisation | daapleu_mz_au_20160618 | 400 | 400 |
| Main HG Contact Mineralisation | daapleu_mz_au_20160618 | 200 | 200 |
| Minor LG Footwall Mineralisation | daapleu_mz_au_20160618 | 100 | 100 |
| Minor LG Footwall Mineralisation | daapleu_mz_au_20160618 | 150 | 150 |
| Minor LG Footwall Mineralisation | daapleu_mz_au_20160618 | 500 | 500 |
| Background | daapleu_drill_limit_20160614 | 1 | 999 |

**Figure 14.4.8 Daapleu Mineralisation Interpretation – Plan View**




***Gbeitouo***

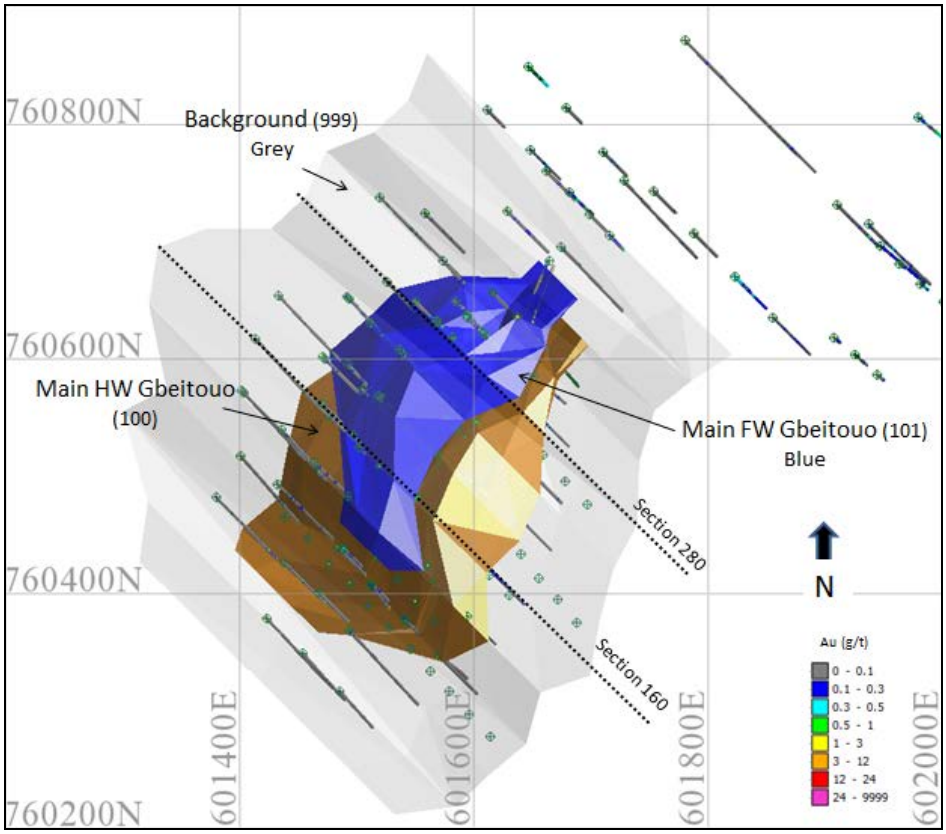
The mineralisation at Gbeitouo was interpreted as sectional based primarily on a nominal 0.3 ppm Au grade cut-off. In addition, relevant drill log information and the lithological model were used as guidance for the interpretation. Two main mineralisation domains have been modelled and these interpretations contain a higher grade core that shows an approximate cut-off grade of 0.7 ppm Au, but has not been deterministically wireframed, as the boundary is relatively diffuse.

A summary of the relevant wireframe models is listed in Table 14.4.5 and a plan 3D image of the wireframe model is shown in Figure 14.4.9.

**Table 14.4.5 Gbeitouo Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Main FW Gbeitouo | mz_gbeit_july_2016 | 100 | 100 |
| Main HW Gbeitouo | mz_gbeit_july_2016 | 101 | 101 |
| Background | mz_drill_limit_july_2016 | 999 | 999 |

**Figure 14.4.9 Gbeitouo Mineralisation Interpretation – Plan View**




***Mont Ity / Ity Flat***

The mineralisation at Mont Ity / Ity Flat was interpreted as sectional polygons based primarily on a nominal 0.3 ppm Au grade cut-off. The interpretation was also strongly guided by the lithological model. Trend surfaces that define the orientation of the mineralised domains were generated for later use in controlling the local rotation of the variogram models and search neighbourhoods during the kriging process.

A summary of the relevant wireframe models is listed in Table 14.4.6. A plan 3D image of the wireframe model is shown in Figure 14.4.10 and a cross-sectional view displayed in Figure 14.4.11.

**Table 14.4.6 Mont Ity / Ity Flat Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Laterite | Ity_mz_au_201607 | 301 | 301 |
| Main Mont Ity/Ity Flat | Ity_mz_au_201607 | 400 | 400 |
| Minor Mont Ity HW | Ity_mz_au_201607 | 100 | 100 |
| Minor Ity Flat HW | Ity_mz_au_201607 | 200 | 200 |
| Minor Ity Flat FW | Ity_mz_au_201607 | 500 | 500 |
| Minor Ity Flat FW | Ity_mz_au_201607 | 600 | 600 |
| HW Background | Ity_mz_au_201607 | 999 | 888 |
| | mz_ity_waste_fw_hw_888_999 | 888 | |
| FW Background | Ity_mz_au_201607 | 999 | 999 |
| | mz_ity_waste_fw_hw_888_999 | 999 | |

**Figure 14.4.10 Mont Ity / Ity Flat Mineralisation Interpretation – Plan View**

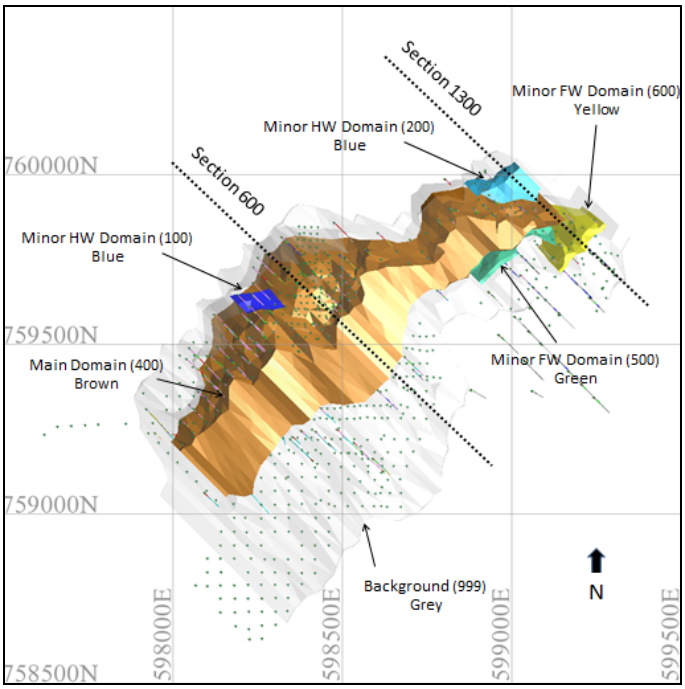
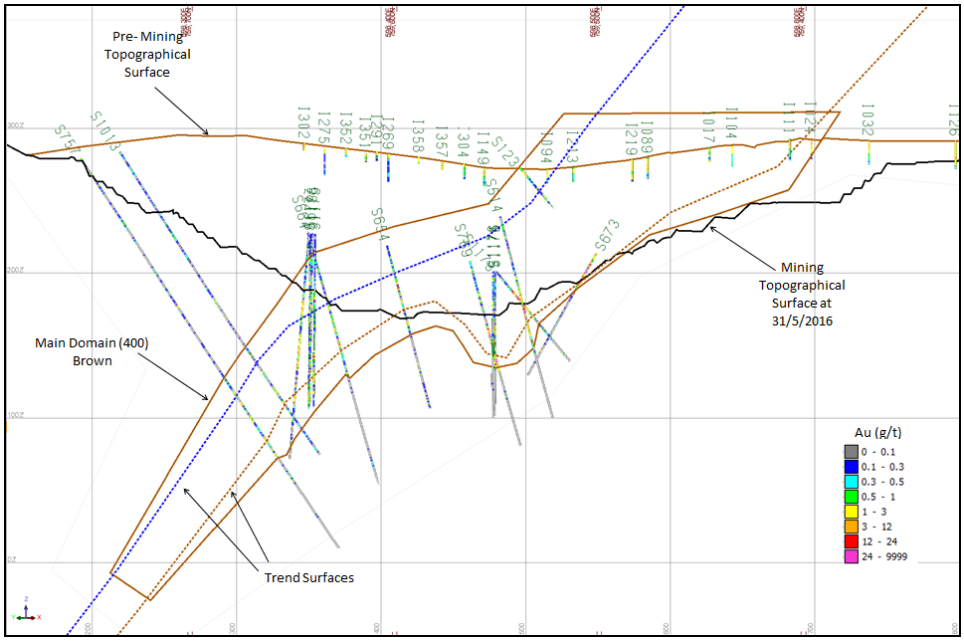


**Figure 14.4.11 Mont Ity / Ity Flat Mineralisation Interpretation – Section 600 Looking Northeast**




***Walter***

A robust mineralisation interpretation at Walter was based primarily on a nominal 0.3 ppm Au grade cut-off which appears to approximate the natural grade cut-off between mineralisation and waste rock. In addition, relevant drill log information and the lithological model were used as guidance for the interpretation. A total of 20 mineralisation domains have been modelled and grouped according to the main lithology type.

A summary of the relevant wireframe models is listed in Table 14.4.7. Plan and cross-sectional images of the wireframe model are shown in Figure 14.4.12 and Figure 14.4.13.

**Table 14.4.7 Walter Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Minor lodes on contact or within 101 facies | mz_walter_july_2016 | 100 - 104 | 100 - 104 |
| | | 600 | 600 |
| Major and Minor lodes within 201 and 211 facies | mz_walter_july_2016 | 200 - 211 | 200 - 211 |
| Laterite lode 301 facies | mz_walter_july_2016 | 300 | 300 |
| Minor lodes within 501 facies | mz_walter_july_2016 | 500 | 500 |
| | | 800 | 800 |
| Main lode with 901 facies | mz_walter_july_2016 | 900 | 900 |

**Figure 14.4.12 Walter Mineralisation Interpretation – Plan View**

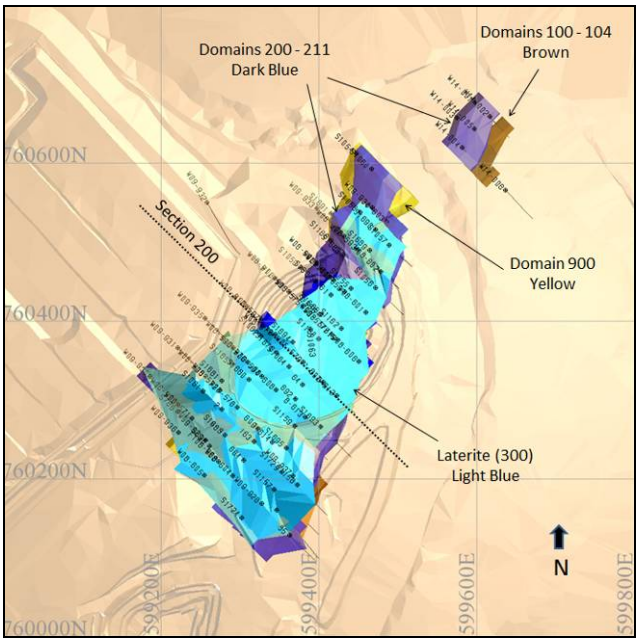
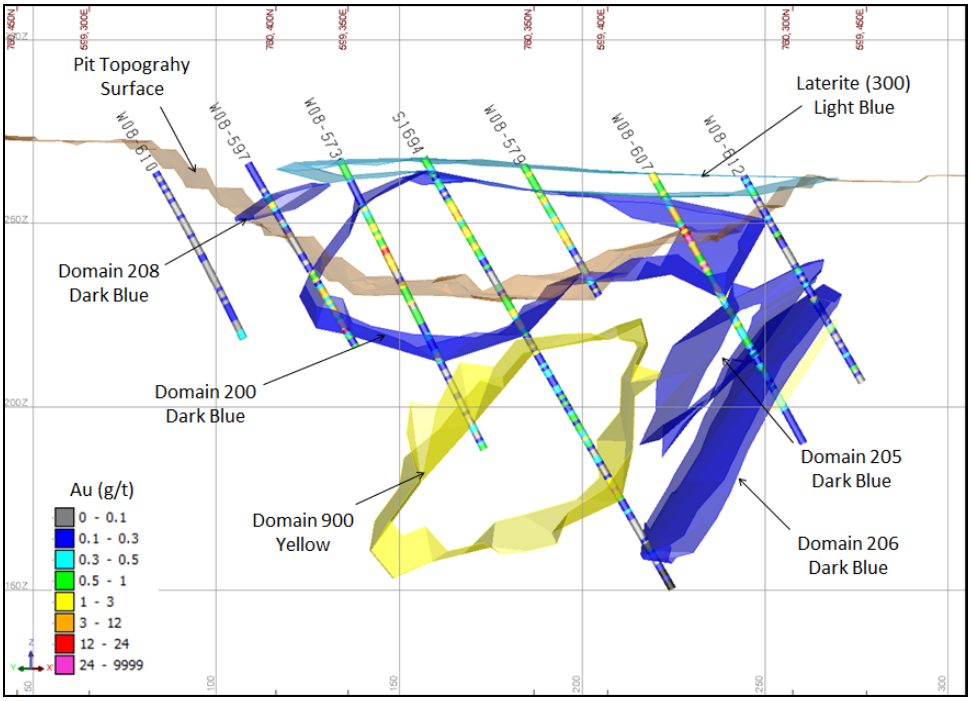



**Figure 14.4.13 Walter Mineralisation Interpretation – Section -200 Looking Northeast**



***ZiaNE***

The mineralisation at ZiaNE was interpreted based primarily on a nominal 0.3 ppm Au grade cut-off resulting in one main and three minor mineralisation domains. In addition, mineralisation within the laterite has been estimated plus a halo around all drilling to capture minor background mineralisation.

A summary of the relevant wireframe models is listed in Table 14.4.8. A plan 3D image of the wireframe model is shown in

Figure 14.4.14 and a sectional view at section position -175 is displayed in Figure 14.4.15.

**Table 14.4.8 ZiaNE Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Minor HW ZiaNE | ziane_mz_au_20160713 | 100 | 100 |
| Main ZiaNE | ziane_mz_au_20160713 | 200 | 200 |
| Laterite | ziane_mz_au_20160713 | 301 | 301 |
| Minor FW ZiaNE | ziane_mz_au_20160713 | 400 | 400 |
| Minor FW ZiaNE | ziane_mz_au_20160713 | 500 | 500 |
| Background | ziane_drill_limit_20160713 | 1 | 999 |

**Figure 14.4.14 ZiaNE Mineralisation Interpretation – Plan View**

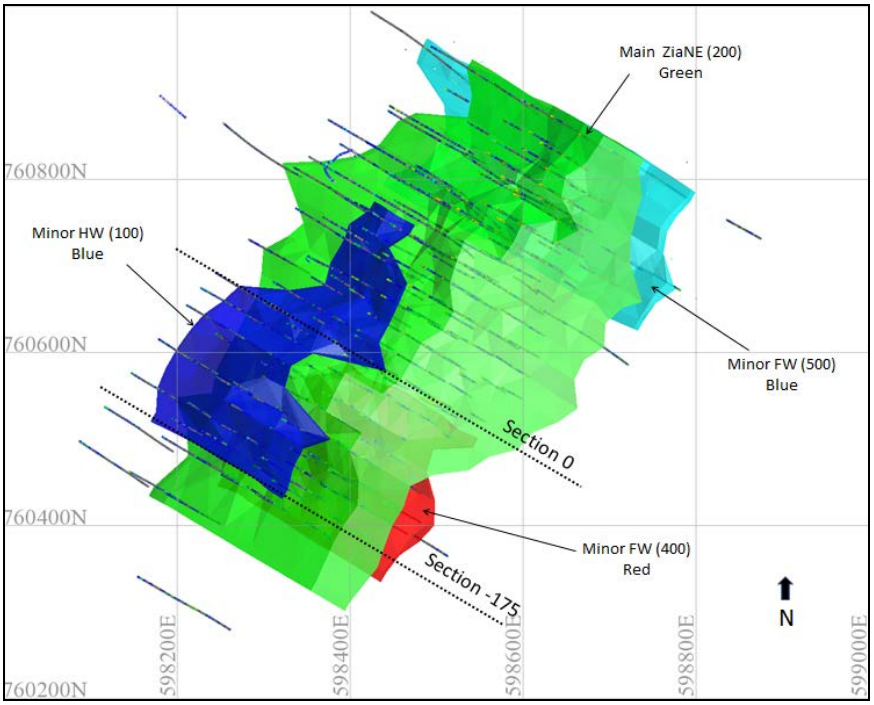
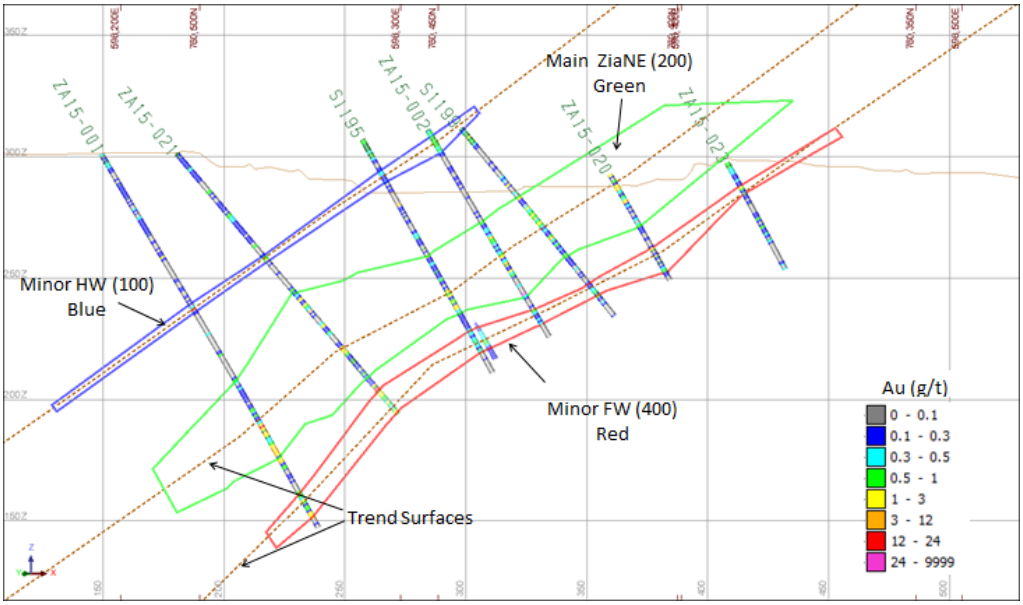


**Figure 14.4.15 ZiaNE Mineralisation Interpretation – Section -175 Looking Northeast**




***Aires***

The mineralisation domaining at Aires is based on the Facies modelling. The Facies domain 3DM's were used for direct assignment of mineralisation domaining into the block model. The domain interpretation, 3DM files and block model assignment is summarised in Table 14.4.9.

**Table 14.4.9 Aires Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Laterite (301) | geo_laterite301_20160715 | 301 | 301 |
| Heap Leach (701) | geo_heapleach303_20160718 | 303 | 303 |

***Teckraie and Verse Ouest***

The mineralisation domaining at Teckraie and Verse Ouest is based on the Facies modelling described above. The rock dumps are not physically separate and Cube interpreted a nominal line of separation that best honours the drilling data and previous description of the two rock dumps. There is also a small proportion of unknown material that has been deposited on top of the Teckraie rock dump. The origin of this material is not known and it appears to postdate the drilling and therefore has not been sampled. As such, this material has been modelled separately to the main Teckraie rock dump. The domain interpretation, 3DM files and block model assignment is displayed in Figure 14.4.16 and summarised in Table 14.4.10.

**Figure 14.4.16 Teckraie and Verse Ouest Mineralisation Interpretation – Plan View**

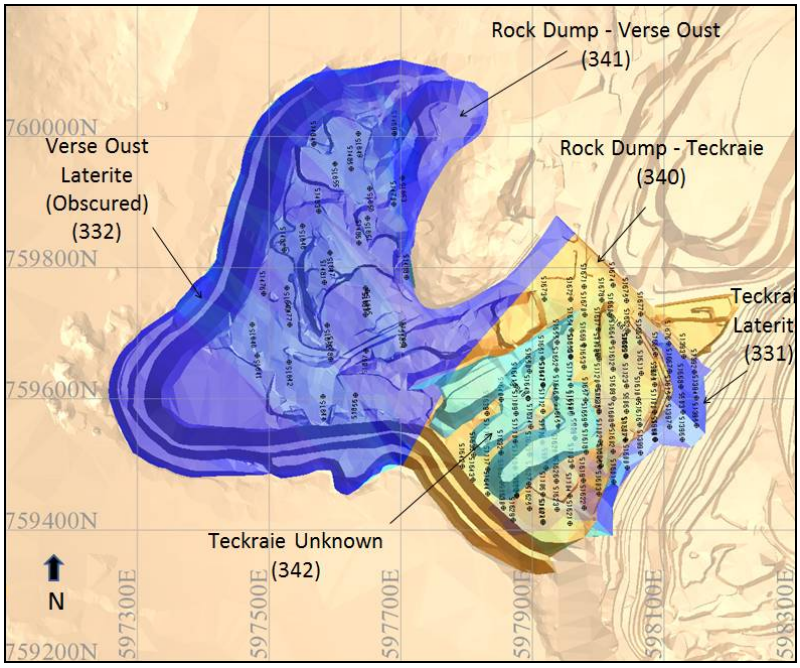



**Table 14.4.10 Teckraie and Verse Ouest Mineralisation Solids and Assignment**

| Domain Description | 3DM | Object No. | Block Model Assignment (minzone_au) |
|---|---|---|---|
| Teckraie Laterite (331) | geo_laterite301_20160715 | 331 | 331 |
| Verse Ouest Laterite (332) | geo_laterite301_20160715 | 332 | 332 |
| Teckraie Dump (340) | geo_dumps340_20160725 | 340 | 340 |
| Teckraie Unknown (342) | geo_dumps340_20160725 | 342 | 342 |
| Verse Ouest Dump (341) | geo_dumps340_20160725 | 341 | 341 |

### 14.4.4 Bulk Density

All available bulk density data was supplied to Cube by Endeavour and includes a total of 3,391 determinations. The determinations that were deemed acceptable were limited to measurements undertaken on diamond drill core which had been measured using a standard water displacement method. This resulted in a total of 2,948 determinations available for inclusion in the total Ity Project resource estimation.

***Daapleu***

Density determinations specific to the Daapleu deposit were grouped by lithology type and also weathering where applicable. A filter was applied to remove determinations less than 1.2 or greater than 4.0 as these were determined by Cube as erroneous and Table 14.4.11 below summarises the basic statistics.

**Table 14.4.11 Daapleu Bulk Density Data Summary**

| Lithology | All | Volcano Sediments | Oxide Clay | Reduced Clay | Laterite | Soil | Daaplite (Fresh) | Daaplite (Trans) | Mafic Intrusive |
|---|---|---|---|---|---|---|---|---|---|
| Facies | | 101 | 201 | 211 | 301 | 311 | 601 | 601 | 811 |
| No. | 1,143 | 434 | 104 | 48 | 28 | 27 | 424 | 41 | 37 |
| Min. | 1.217 | 1.27 | 1.22 | 1.24 | 1.22 | 1.34 | 1.45 | 1.30 | 1.69 |
| Max. | 3.498 | 3.40 | 3.04 | 2.87 | 2.08 | 1.90 | 3.50 | 2.73 | 2.89 |
| Mean | 2.527 | 2.82 | 1.64 | 1.76 | 1.61 | 1.58 | 2.67 | 2.15 | 2.71 |
| Median | 2.699 | 2.84 | 1.57 | 1.66 | 1.57 | 1.57 | 2.68 | 2.06 | 2.75 |
| CoV | 0.187 | 0.05 | 0.18 | 0.23 | 0.11 | 0.09 | 0.05 | 0.23 | 0.08 |

Table 14.4.12 summarises the bulk densities values assigned to the block model based on 'lithology' and also 'weathering' for the Daaplite and Volcano Sediments. The bulk densities values were selected based on the population statistics summarised above.

**Table 14.4.12 Daapleu Assigned Block Model In Situ Bulk Density**

| Lithology | Facies | Assigned Density |
|---|---|---|
| Volcano Sediments (Transition) | 101 | 1.94 |
| Volcano Sediments (Fresh) | 101 | 2.82 |
| Oxidised Clay | 201 | 1.64 |
| Reduced Clay | 211 | 1.76 |
| Laterite | 301 | 1.61 |
| Soil | 311 | 1.58 |
| Daaplite (Transition) | 601 | 2.15 |
| Daaplite (Fresh) | 601 | 2.67 |
| Mafic Intrusive | 811 | 2.71 |

***Gbeitouo***

The Gbeitouo deposit density determinations were grouped by lithology type and Table 14.4.13 summarises the basic statistics.

**Table 14.4.13 Gbeitouo Bulk Density Data Summary**

| Lithology | All | Laterite | Oxide Clay | Reduced Clay | Volcano Sediments | Carbonate |
|---|---|---|---|---|---|---|
| Facies | - | 301 | 201 | 211 | 101 | 401 |
| No. | 334 | 6 | 97 | 23 | 189 | 19 |
| Min. | 1.04 | 1.05 | 1.14 | 1.20 | 1.04 | 2.68 |
| Max. | 2.96 | 2.70 | 2.95 | 2.85 | 2.96 | 2.85 |
| Mean | 2.25 | 2.01 | 1.53 | 1.63 | 2.65 | 2.77 |
| Median | 2.59 | 1.91 | 1.43 | 1.57 | 2.78 | 2.75 |
| CoV | 0.29 | 0.31 | 0.26 | 0.23 | 0.13 | 0.02 |

Table 14.4.14 summarises the bulk density values assigned to the block model based on 'lithology'. The bulk density values were selected based on the population statistics summarised above but also from density determinations for surrounding deposits within the Ity Project areas with similar lithologies. For example, given there are only a small number of density determinations for the lateritic material at Gbeitouo, the density assigned has been adopted from Daapleu which has a greater sample population.

**Table 14.4.14 Gbeitouo Assigned Block Model In Situ Bulk Density**

| Lithology | Laterite | Oxide Clay | Reduced Clay | Volcano Sediments | Carbonate |
|---|---|---|---|---|---|
| Facies | 301 | 201 | 211 | 101 | 401 |
| Assigned Density | 1.61 | 1.43 | 1.57 | 2.75 | 2.75 |

***Mont Ity / Ity Flat***

The Mont Ity / Ity Flat density determinations were grouped by lithology type. A cut filter was applied in some instances to remove determinations considered by Cube as erroneous and Table 14.4.15 summarises the basic statistics.

**Table 14.4.15 Mont Ity / Ity Flat In Situ Density Data Summary**

| Lithology | Volcano Sediments | Oxidised Clay | Reduced Clay | Laterite | Carbonate | Granodiorite | Mafic Intrusive | Skarn |
|---|---|---|---|---|---|---|---|---|
| Facies | 101 | 201 | 211 | 301 | 401 | 501 | 811 | 901 |
| Cut | - | 1.50 | 2.50 | - | 3.50 | - | - | 3.10 |
| No. | 59 | 212 | 108 | 44 | 242 | 27 | 32 | 75 |
| Min. | 2.43 | 1.02 | 1.12 | 1.09 | 1.07 | 1.19 | 1.10 | 1.91 |
| Max. | 3.06 | 1.95 | 2.48 | 2.76 | 3.31 | 3.99 | 3.15 | 3.07 |
| Mean | 2.77 | 1.50 | 1.69 | 1.63 | 2.77 | 2.57 | 2.74 | 2.80 |
| Median | 2.74 | 1.52 | 1.64 | 1.62 | 2.79 | 2.57 | 2.81 | 2.85 |
| CoV | 0.04 | 0.14 | 0.17 | 0.20 | 0.10 | 0.19 | 0.16 | 0.09 |

Table 14.4.16 summarises the bulk density values assigned to the block model based on 'lithology'. The bulk density values were selected based on the cut population statistics summarised above.

**Table 14.4.16 Mont Ity / Ity Flat Assigned Block Model In Situ Bulk Density**

| Lithology | Volcano Sediments | Oxidised Clay | Reduced Clay | Laterite | Carbonate | Granodiorite | Mafic Intrusive | Skarn |
|---|---|---|---|---|---|---|---|---|
| Facies | 101 | 201 | 211 | 301 | 401 | 501 | 811 | 901 |
| Assigned Density | 2.75 | 1.50 | 1.69 | 1.62 | 2.77 | 2.57 | 2.74 | 2.80 |

***Walter***

The Walter deposit density determinations were grouped by lithology type. A cut filter was applied in some instances to remove determinations considered by Cube as erroneous and Table 14.4.17 summarises the basic statistics.

**Table 14.4.17 Walter In Situ Density Data Summary**

| Lithology | Laterite | Oxide Clay | Reduced Clay | Carbonate | Volcano Sediment | Skarn |
|---|---|---|---|---|---|---|
| Facies | 301 | 201 | 211 | 401 | 101 | 901 |
| Cut | - | 1.8 | 2.0 | - | - | - |
| No. | 5 | 23 | 20 | 7 | 25 | 23 |
| Min. | 1.62 | 1.10 | 1.36 | 2.51 | 1.88 | 1.29 |
| Max. | 1.86 | 1.80 | 1.95 | 2.94 | 3.10 | 3.25 |
| Mean | 1.77 | 1.51 | 1.69 | 2.71 | 2.79 | 2.72 |
| Median | 1.64 | 1.51 | 1.67 | 2.61 | 2.81 | 2.83 |
| CoV | 0.07 | 0.14 | 0.11 | 0.06 | 0.09 | 0.16 |

Table 14.4.18 summarises the bulk density values assigned to the block model based on 'lithology'. The bulk density values were selected based on the population statistics summarised above but also from density determinations for surrounding deposits within the Ity Project areas with similar lithologies. For example, given there are only a small number of density determinations for the lateritic and granodiorite material at Walter, the assigned densities have been adopted from Mont Ity which has a greater sample population.

**Table 14.4.18 Walter Assigned Block Model In Situ Bulk Density**

| Lithology | Laterite | Oxide Clay | Reduced Clay | Carbonate | Granodiorite | Volcano Sediment | Skarn |
|---|---|---|---|---|---|---|---|
| Facies | 301 | 201 | 211 | 401 | 501 | 101 | 901 |
| Assigned Density | 1.62 | 1.51 | 1.69 | 2.72 | 2.57 | 2.79 | 2.72 |

***ZiaNE***

The ZiaNE deposit density determinations were grouped by lithology type. A cut filter was applied in some instances to remove determinations considered by Cube as erroneous and Table 14.4.19 summarises the basic statistics.

**Table 14.4.19 ZiaNE Cut Bulk Density Data Summary**

| Lithology | Laterite | Oxide Clay | Reduced Clay | Carbonate | Granodiorite | Mafic Intrusive | Skarn |
|---|---|---|---|---|---|---|---|
| Facies | 301 | 201 | 211 | 401 | 501 | 811 | 901 |
| Cut | 2.0 | 2.1 | 2.5 | 3.6 | 3.5 | - | 3.2 |
| No. | 24 | 212 | 108 | 242 | 26 | 32 | 77 |
| Min. | 1.227 | 1.02 | 1.12 | 1.07 | 1.19 | 1.10 | 1.91 |
| Max. | 1.95 | 1.95 | 2.48 | 3.31 | 3.02 | 3.15 | 3.19 |
| Mean | 1.69 | 1.50 | 1.69 | 2.77 | 2.52 | 2.74 | 2.81 |
| Median | 1.72 | 1.52 | 1.64 | 2.79 | 2.54 | 2.81 | 2.85 |
| CoV | 0.12 | 0.14 | 0.17 | 0.10 | 0.16 | 0.16 | 0.09 |

Table 14.4.20 summarises the bulk density values assigned to the block model based on 'lithology'. The bulk density values were selected based on the population statistics summarised above.

**Table 14.4.20 ZiaNE Assigned Block Model In Situ Bulk Density**

| Lithology | Laterite | Oxide Clay | Reduced Clay | Carbonate | Granodiorite | Mafic Intrusive | Skarn |
|---|---|---|---|---|---|---|---|
| Facies | 301 | 201 | 211 | 401 | 501 | 811 | 901 |
| Assigned Density | 1.69 | 1.50 | 1.69 | 2.77 | 2.52 | 2.74 | 2.81 |

***Aires***

The Aires deposit density determinations were grouped by lithology type and Table 14.4.21 summarises the basic density determination statistics.

**Table 14.4.21 Aires Bulk Density Data Summary**

| Lithology | Heap Leach |
|---|---|
| Facies | 303 |
| No. | 101 |
| Min. | 1.00 |
| Max. | 1.96 |
| Mean | 1.35 |
| Median | 1.32 |
| CoV | 0.13 |

Table 14.4.22 summarises the bulk density values assigned to the block model based on 'lithology'. The bulk density values were selected based on the population statistics summarised above but also from density determinations for surrounding deposits within the Ity Project areas with similar lithologies. Given there are no density determinations for the laterite and oxidised clay material at Aires, the densities assigned has been adopted from the Mont Ity and Ity Flat deposits.

**Table 14.4.22 Assigned Aires Block Model In Situ Bulk Density**

| Lithology | Heap Leach | Laterite | Oxide Clay |
|---|---|---|---|
| Facies | 701 | 301 | 201 |
| Assigned Density | 1.30 | 1.60 | 1.50 |

***Teckraie and Verse Ouest***

The density determinations for Teckraie and Verse Ouest were grouped by deposit and facies and Table 14.4.23 summarises the basic density determination statistics.

**Table 14.4.23 Teckraie and Verse Ouest Bulk Density Data Summary**

| Mineralisation Domain | Laterite | Teckraie | Verse Ouest |
|---|---|---|---|
| Facies | 301 | 701 | 701 |
| No. | 54 | 25 | 15 |
| Min. | 1.12 | 1.08 | 1.58 |
| Max. | 2.83 | 1.90 | 3.11 |
| Mean | 1.66 | 1.40 | 1.83 |
| Median | 1.65 | 1.34 | 1.72 |
| CoV | 0.19 | 0.16 | 0.20 |

The small population of data contained within Verse Ouest appears to be erroneously high given the material represents broken rock and oxide clay. Therefore Cube decided to ignore these measurements and applied the rounded median value for Teckraie to all rock dump material. Table 14.4.24 below summarises the bulk density values assigned to the block model based on 'lithology'. Given there are no density determinations for the oxidised clay material at Teckraie and Verse Ouest, the density assigned has been adopted from the Mont Ity and Ity Flat deposits.

**Table 14.4.24 Teckraie and Verse Ouest Assigned Block Model In Situ Bulk Density**

| Lithology | Heap Leach | Laterite | Oxide Clay |
|---|---|---|---|
| Facies | 701 | 301 | 201 |
| Assigned Density | 1.30 | 1.60 | 1.50 |

## 14.5 Compositing

### 14.5.1 Coding and Compositing of Data

In the resource database specific to each deposit, a field called 'resource' was added to the 'collar' table. All drillholes to be included in the resource estimation were flagged as 'july2016'. These holes were restricted to drill types DD, RC and AC. Drillhole types named PIT and TR were excluded from the compositing process due to the lack of historical documentation, the perceived poor quality of sampling and the risk of sample contamination. In some instances a small number of drill holes were not included to avoid any potential bias due to partial sampling of these holes.

A unique code for drill intercepts within each of the mineralised domains was added to the database table 'zonecode'. The process of coding the database was carried out automatically using the domain wireframes given that they were snapped to drill traces. In addition, the intersections were manually checked to ensure the appropriate downhole intervals were coded and assigned the correct unique code for each of the domain wireframes. This coded interval was used to control the compositing process whilst extracting sample and composite data combinations for statistical analysis and subsequent resource estimation.

### 14.5.2 Composite Selection

Five main criteria were considered when determining the most appropriate compositing method:

- Sample length statistics.
- Additivity of variables.
- Homogeneity and variability of composited zones.
- Suitability of the composites for the chosen estimation method.
- Expected parent block and smallest mining unit ('SMU') size.

The principal objective of compositing is to create samples of equal support which is a mandatory requirement for the estimation process. A further important objective is to reduce the variability inherent in raw samples or small composites relative to estimation block dimensions and to assist in reducing the nugget effect and improving the quality of variography.

After an examination of the above criteria, Cube decided that 2 m downhole composites were appropriate for compositing the gold assays within the in situ deposit mineralised domains. The compositing routine was undertaken in Surpac using the 'Best Fit' algorithm. The 'Best Fit' routine allocates the same proportion of the residue composite length (e.g. if last composite of a domain is <50% of the composite length) to each of the drill hole composites, resulting in slight differences in composite lengths between drill holes. The 'Best Fit' routine ensures the assay sample composite length is as close as possible to the nominated 2 m composite length and reduces the potential issues associated with residual composites located at domain boundaries.

A similar compositing routine was used for the rock dump and leach pad deposits, however Cube decided that 2.5 m downhole composites was more appropriate for compositing the gold assays. Given the relatively narrow width of the laterite domains below the leach pad and rock dumps, Cube selected 1 m downhole composites for these laterite estimation domains.

Gold composites were extracted from the 'Au_Cube' field within the assay table specific to each deposit and the 'zonecode' interval table was used to ensure only mineralised coded sample intervals were composited.

The minor background elements comprising arsenic, copper and sulphur were included in the compositing process. The multi-element assaying at the Ity Project has generally been confined to zones where gold assays were greater than 0.5 ppm. As a result, there are risks associated with the multi-element data given it is not systematically represented in the total assay data set. It is not possible to draw any definitive conclusions from the multi-element data and it is to be used as a guide only.

## 14.6 Statistical Analysis

A statistical and visual analysis of the extracted 2 m downhole composites was undertaken for each of the mineralised domains for all deposits. The statistical analysis undertaken consisted of:

- Basic Statistics – tabulated and graphed for each domain
- Weathering Analysis – investigate the requirement for sub-domaining based on weathering and
- High Cut Sensitivity Analysis – investigation of outliers to determine whether high grade assay limits were to be applied to the composite data.

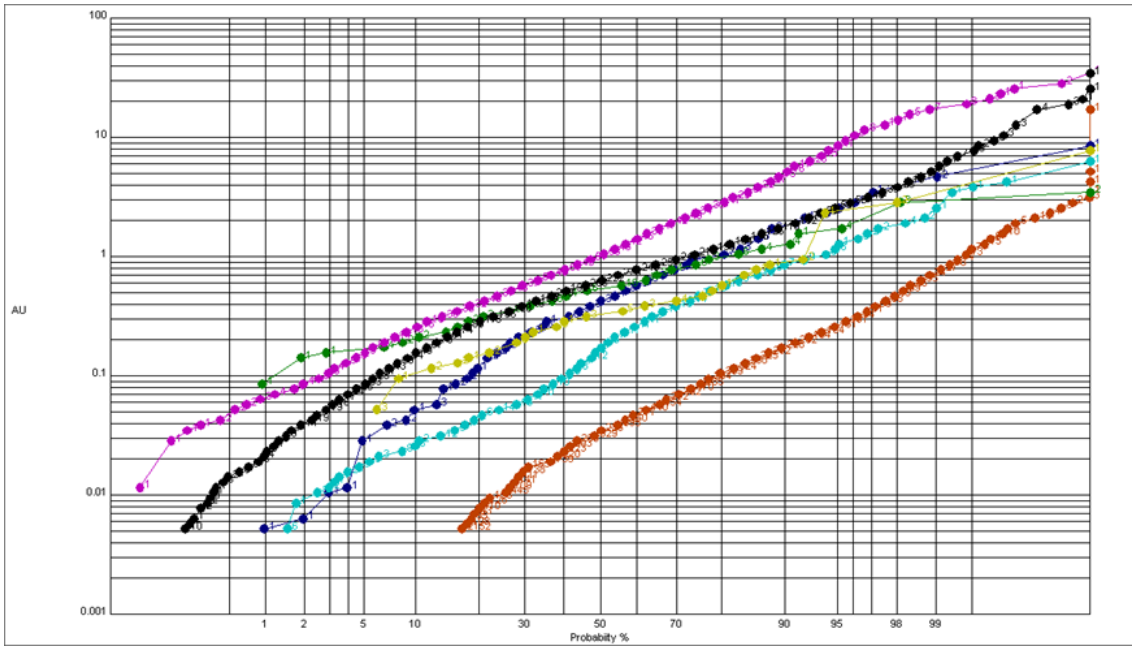
***Basic Statistics***

The basic descriptive statistics for gold grade in the mineralised and background domains for each of the deposits is summarised in Table 14.6.1 to Table 14.6.7 and selected log probability plots are shown in Figure 14.6.1 to Figure 14.6.8.

**Table 14.6.1 Daapleu Summary Gold Statistics for 2 m Composites by Domain**

| Domain | 100 | 150 | 200 | 301 | 400 | 500 | 999 |
|---|---|---|---|---|---|---|---|
| Number | 102 | 106 | 1,428 | 405 | 4,966 | 50 | 6434 |
| Minimum | 0.01 | 0.09 | 0.01 | 0.01 | 0.01 | 0.05 | 0.01 |
| Maximum | 8.20 | 3.60 | 35.55 | 6.42 | 33.96 | 7.76 | 6.79 |
| Mean | 0.81 | 0.75 | 2.30 | 0.38 | 0.97 | 0.64 | 0.09 |
| Median | 0.44 | 0.57 | 1.08 | 0.18 | 0.66 | 0.33 | 0.04 |
| Std. Dev. | 1.15 | 0.66 | 3.52 | 0.59 | 1.38 | 1.19 | 0.18 |
| Variance | 1.31 | 0.43 | 12.39 | 0.35 | 1.90 | 1.42 | 0.03 |
| Coeff. Var. | 1.41 | 0.88 | 1.53 | 1.56 | 1.43 | 1.87 | 2.13 |
| **Percentile** | | | | | | | |
| 10th | 0.05 | 0.21 | 0.27 | 0.03 | 0.17 | 0.12 | 0.01 |
| 20th | 0.12 | 0.33 | 0.43 | 0.05 | 0.30 | 0.16 | 0.01 |
| 30th | 0.24 | 0.39 | 0.62 | 0.07 | 0.41 | 0.22 | 0.02 |
| 40th | 0.32 | 0.46 | 0.80 | 0.10 | 0.54 | 0.27 | 0.03 |
| 50th | 0.44 | 0.57 | 1.08 | 0.18 | 0.66 | 0.33 | 0.04 |
| 60th | 0.60 | 0.63 | 1.49 | 0.28 | 0.82 | 0.39 | 0.06 |
| 70th | 0.82 | 0.82 | 2.08 | 0.41 | 1.00 | 0.45 | 0.08 |
| 80th | 1.09 | 1.03 | 2.96 | 0.59 | 1.28 | 0.60 | 0.12 |
| 90th | 1.85 | 1.28 | 5.22 | 0.90 | 1.90 | 0.91 | 0.19 |
| 95th | 2.60 | 1.78 | 8.96 | 1.24 | 2.76 | 2.58 | 0.28 |
| 97.5th | 3.99 | 2.93 | 13.02 | 1.83 | 3.65 | 2.87 | 0.43 |
| 99th | 4.60 | 3.34 | 18.17 | 2.65 | 5.71 | 5.33 | 0.72 |

**Figure 14.6.1 Daapleu Log - Probability Plot of 2 m Composites Gold by Domain**




**(Red=999, Dark Blue=100, Green=150, Purple=200, Light Blue=301, Black=400, Light Green=500**

**Table 14.6.2 Gbeitouo Summary Gold Statistics for 2 m Composites by Domain**

| Domain | 100 | 101 | 999 |
|---|---|---|---|
| Number | 850 | 735 | 3,566 |
| Minimum | 0.01 | 0.01 | 0.01 |
| Maximum | 28.56 | 16.58 | 2.21 |
| Mean | 1.36 | 0.57 | 0.05 |
| Median | 0.62 | 0.36 | 0.02 |
| Std. Dev. | 2.12 | 0.87 | 0.11 |
| Variance | 4.47 | 0.76 | 0.01 |
| Coeff. Var. | 1.56 | 1.53 | 2.08 |
| **Percentile** | | | |
| 10th | 0.10 | 0.08 | 0.01 |
| 20th | 0.20 | 0.14 | 0.01 |
| 30th | 0.31 | 0.20 | 0.01 |
| 40th | 0.45 | 0.28 | 0.02 |
| 50th | 0.62 | 0.36 | 0.02 |
| 60th | 0.91 | 0.47 | 0.03 |
| 70th | 1.46 | 0.58 | 0.05 |
| 80th | 2.13 | 0.79 | 0.07 |
| 90th | 3.28 | 1.20 | 0.13 |
| 95th | 4.59 | 1.73 | 0.19 |
| 97.5th | 6.13 | 2.50 | 0.26 |
| 99th | 9.40 | 3.49 | 0.46 |

**Figure 14.6.2 Gbeitouo Log - Probability Plot of 2 m Gold Composites – Domain 100**

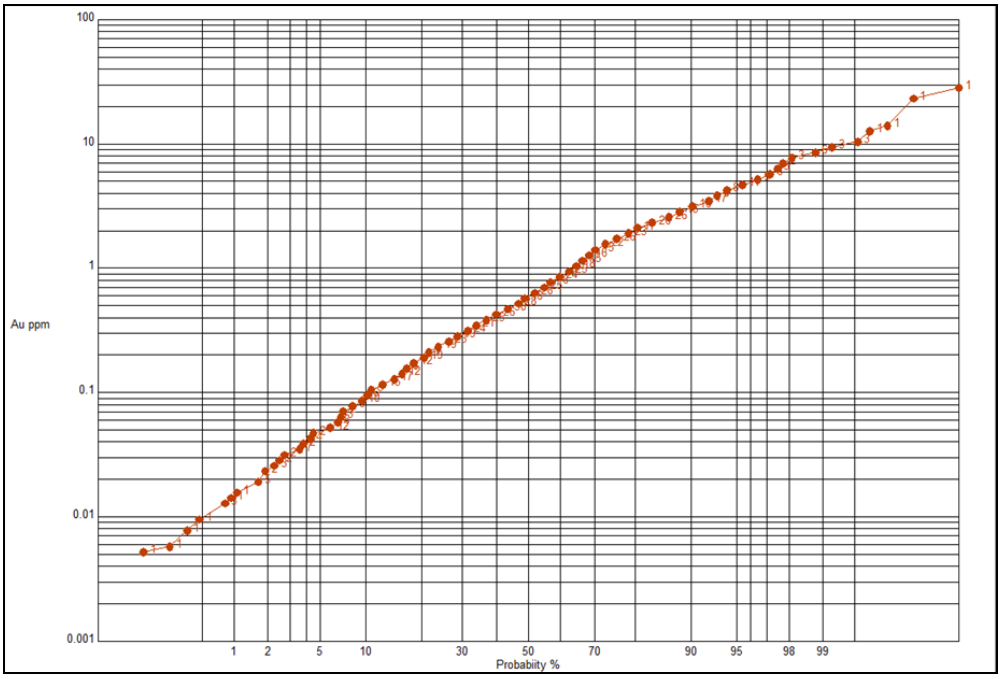


**Table 14.6.3 Mont Ity / Ity Flat Summary Gold Statistics for 2 m Composites by Domain**

| Domain | 100 | 200 | 301 | 400 | 500 | 600 | 888 | 999 |
|---|---|---|---|---|---|---|---|---|
| Number | 51 | 455 | 2,399 | 10,610 | 349 | 290 | 4,880 | 8,062 |
| Minimum | 0.05 | 0.03 | 0.01 | 0.01 | 0.01 | 0.02 | 0.01 | 0.01 |
| Maximum | 1.84 | 230.93 | 109.11 | 360.57 | 14.88 | 14.08 | 39.12 | 68.66 |
| Mean | 0.48 | 1.28 | 1.27 | 1.93 | 0.69 | 1.07 | 0.23 | 0.16 |
| Median | 0.38 | 0.43 | 0.65 | 0.46 | 0.33 | 0.48 | 0.13 | 0.04 |
| Std. Dev. | 0.34 | 10.98 | 2.78 | 6.69 | 1.30 | 1.77 | 0.77 | 1.07 |
| Variance | 0.12 | 120.51 | 7.70 | 44.77 | 1.68 | 3.12 | 0.59 | 1.14 |
| Coeff. Var. | 0.71 | 8.61 | 2.19 | 3.47 | 1.88 | 1.66 | 3.36 | 6.74 |
| **Percentile** | | | | | | | | |
| 10th | 0.01 | 0.09 | 0.11 | 0.04 | 0.03 | 0.05 | 0.02 | 0.01 |
| 20th | 0.07 | 0.12 | 0.16 | 0.06 | 0.05 | 0.06 | 0.03 | 0.01 |
| 30th | 0.16 | 0.18 | 0.23 | 0.10 | 0.09 | 0.10 | 0.04 | 0.01 |
| 40th | 0.24 | 0.25 | 0.33 | 0.17 | 0.15 | 0.17 | 0.06 | 0.01 |
| 50th | 0.26 | 0.30 | 0.42 | 0.24 | 0.20 | 0.28 | 0.08 | 0.02 |
| 60th | 0.32 | 0.35 | 0.52 | 0.34 | 0.26 | 0.38 | 0.11 | 0.03 |
| 70th | 0.38 | 0.43 | 0.65 | 0.46 | 0.33 | 0.48 | 0.13 | 0.04 |
| 80th | 0.46 | 0.52 | 0.84 | 0.64 | 0.44 | 0.61 | 0.16 | 0.06 |
| 90th | 0.50 | 0.64 | 1.14 | 0.97 | 0.57 | 0.89 | 0.20 | 0.09 |
| 95th | 0.66 | 0.84 | 1.67 | 1.71 | 0.78 | 1.32 | 0.26 | 0.15 |
| 97.5th | 0.87 | 1.35 | 2.76 | 4.20 | 1.28 | 2.39 | 0.39 | 0.25 |
| 99th | 1.05 | 2.29 | 4.34 | 8.19 | 2.58 | 3.70 | 0.58 | 0.43 |

**Figure 14.6.3 Mont Ity / Ity Flat Log - Probability Plot of 2 m Composites Gold by Domain**

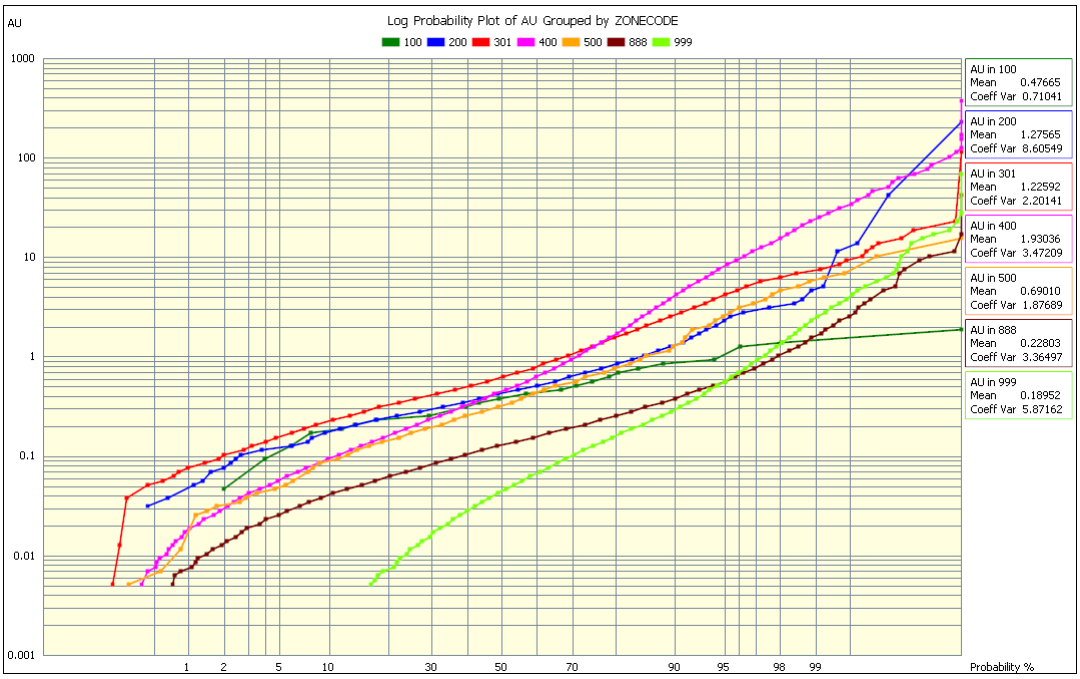


**Table 14.6.4 Walter Summary Gold Statistics for 2 m Composites by Domain**

| Domain | 900 | 800 | 600 | 500 | 300 | 211 | 210 |
|---|---|---|---|---|---|---|---|
| Number | 736 | 6 | 5 | 52 | 187 | 14 | 8 |
| Minimum | 0.02 | 0.42 | 0.35 | 0.25 | 0.21 | 0.30 | 0.47 |
| Maximum | 15.52 | 26.84 | 1.01 | 7.19 | 5.54 | 10.34 | 1.94 |
| Mean | 0.60 | 8.26 | 0.57 | 1.04 | 0.89 | 3.52 | 0.92 |
| Median | 0.27 | 3.69 | 0.45 | 0.57 | 0.69 | 1.75 | 0.79 |
| Std Dev | 1.18 | 10.14 | 0.26 | 1.40 | 0.67 | 3.50 | 0.49 |
| Variance | 1.40 | 102.80 | 0.07 | 1.95 | 0.45 | 12.22 | 0.24 |
| Coeff Var | 1.96 | 1.23 | 0.46 | 1.34 | 0.75 | 0.99 | 0.54 |
| **Percentiles** | | | | | | | |
| 10 | 0.07 | 0.42 | 0.35 | 0.29 | 0.36 | 0.36 | 0.47 |
| 20 | 0.12 | 0.48 | 0.35 | 0.34 | 0.44 | 0.46 | 0.50 |
| 30 | 0.17 | 0.67 | 0.38 | 0.42 | 0.53 | 0.81 | 0.53 |
| 40 | 0.21 | 1.92 | 0.42 | 0.48 | 0.60 | 1.21 | 0.60 |
| 50 | 0.27 | 3.69 | 0.45 | 0.57 | 0.69 | 1.75 | 0.79 |
| 60 | 0.36 | 4.55 | 0.48 | 0.63 | 0.83 | 2.81 | 0.87 |
| 70 | 0.49 | 6.64 | 0.54 | 0.78 | 0.99 | 4.42 | 0.90 |
| 80 | 0.71 | 11.20 | 0.59 | 1.14 | 1.18 | 5.36 | 1.06 |
| 90 | 1.18 | 18.37 | 0.80 | 1.92 | 1.55 | 8.90 | 1.42 |
| 95 | 1.93 | 22.61 | 0.91 | 3.55 | 1.90 | 9.45 | 1.68 |
| 97.5 | 4.02 | 24.72 | 0.96 | 5.80 | 2.15 | 9.89 | 1.81 |
| 99 | 6.32 | 25.99 | 0.99 | 6.53 | 3.37 | 10.16 | 1.89 |

**Table 14.6.4 Walter Summary Gold Statistics for 2 m Composites by Domain (Continued)**

| Domain | 208 | 207 | 206 | 205 | 204 | 203 | 202 |
|---|---|---|---|---|---|---|---|
| Number | 14 | 4 | 119 | 91 | 7 | 78 | 32 |
| Minimum | 0.36 | 0.29 | 0.05 | 0.09 | 0.49 | 0.07 | 0.09 |
| Maximum | 1.57 | 3.46 | 17.71 | 5.96 | 4.65 | 12.94 | 3.11 |
| Mean | 0.64 | 1.33 | 1.70 | 1.13 | 1.82 | 1.71 | 0.71 |
| Median | 0.44 | 0.38 | 0.72 | 0.61 | 1.11 | 0.87 | 0.44 |
| Std Dev | 0.37 | 1.48 | 2.68 | 1.20 | 1.45 | 2.22 | 0.63 |
| Variance | 0.14 | 2.19 | 7.20 | 1.44 | 2.12 | 4.94 | 0.40 |
| Coeff Var | 0.58 | 1.12 | 1.58 | 1.07 | 0.80 | 1.30 | 0.88 |
| **Percentiles** | | | | | | | |
| 10 | 0.36 | 0.29 | 0.21 | 0.30 | 0.49 | 0.17 | 0.26 |
| 20 | 0.38 | 0.29 | 0.34 | 0.42 | 0.58 | 0.29 | 0.29 |
| 30 | 0.39 | 0.31 | 0.47 | 0.49 | 0.73 | 0.44 | 0.34 |
| 40 | 0.41 | 0.34 | 0.56 | 0.54 | 0.87 | 0.63 | 0.38 |
| 50 | 0.44 | 0.38 | 0.72 | 0.61 | 1.11 | 0.87 | 0.44 |
| 60 | 0.54 | 0.69 | 0.87 | 0.81 | 1.51 | 1.29 | 0.58 |
| 70 | 0.58 | 1.01 | 1.25 | 0.99 | 2.22 | 1.55 | 0.81 |
| 80 | 0.81 | 1.63 | 2.18 | 1.51 | 2.36 | 2.23 | 1.05 |
| 90 | 1.09 | 2.55 | 3.98 | 3.06 | 3.06 | 4.94 | 1.17 |
| 95 | 1.35 | 3.01 | 8.17 | 3.75 | 3.86 | 6.21 | 1.73 |
| 97.5 | 1.46 | 3.24 | 10.51 | 4.77 | 4.26 | 7.28 | 2.41 |
| 99 | 1.52 | 3.37 | 10.98 | 5.27 | 4.49 | 8.81 | 2.83 |

**Table 14.6.4 Walter Summary Gold Statistics for 2 m Composites by Domain (Continued)**

| Domain | 201 | 200 | 104 | 103 | 102 | 101 | 100 |
|---|---|---|---|---|---|---|---|
| Number | 58 | 874 | 19 | 3 | 23 | 19 | 8 |
| Minimum | 0.07 | 0.05 | 0.26 | 0.29 | 0.13 | 0.10 | 0.68 |
| Maximum | 2.79 | 44.02 | 3.12 | 2.24 | 5.91 | 5.14 | 2.04 |
| Mean | 0.68 | 1.56 | 0.94 | 1.36 | 0.92 | 0.95 | 1.37 |
| Median | 0.53 | 0.67 | 0.66 | 0.92 | 0.55 | 0.59 | 1.49 |
| Std Dev | 0.54 | 3.62 | 0.75 | 0.99 | 1.21 | 1.12 | 0.48 |
| Variance | 0.29 | 13.10 | 0.57 | 0.97 | 1.45 | 1.25 | 0.23 |
| Coeff Var | 0.79 | 2.33 | 0.80 | 0.73 | 1.31 | 1.17 | 0.35 |
| **Percentiles** | | | | | | | |
| 10 | 0.23 | 0.23 | 0.31 | 0.29 | 0.17 | 0.22 | 0.68 |
| 20 | 0.31 | 0.34 | 0.42 | 0.29 | 0.22 | 0.34 | 0.73 |
| 30 | 0.37 | 0.42 | 0.46 | 0.29 | 0.25 | 0.43 | 0.90 |
| 40 | 0.44 | 0.52 | 0.50 | 0.54 | 0.33 | 0.48 | 1.18 |
| 50 | 0.53 | 0.67 | 0.66 | 0.92 | 0.55 | 0.59 | 1.49 |
| 60 | 0.60 | 0.87 | 0.76 | 1.29 | 0.74 | 0.65 | 1.52 |
| 70 | 0.74 | 1.12 | 0.94 | 1.61 | 0.99 | 0.89 | 1.53 |
| 80 | 0.83 | 1.60 | 1.31 | 1.82 | 1.24 | 1.16 | 1.63 |
| 90 | 1.35 | 3.06 | 1.76 | 2.03 | 1.52 | 1.59 | 1.84 |
| 95 | 1.80 | 5.14 | 2.30 | 2.13 | 1.88 | 2.05 | 1.94 |
| 97.5 | 2.22 | 9.03 | 2.71 | 2.19 | 3.63 | 3.59 | 1.99 |
| 99 | 2.46 | 15.88 | 2.96 | 2.22 | 5.00 | 4.52 | 2.02 |

**Figure 14.6.4 Walter Log - Probability Plot of 2 m Gold Composites – Domain 200**

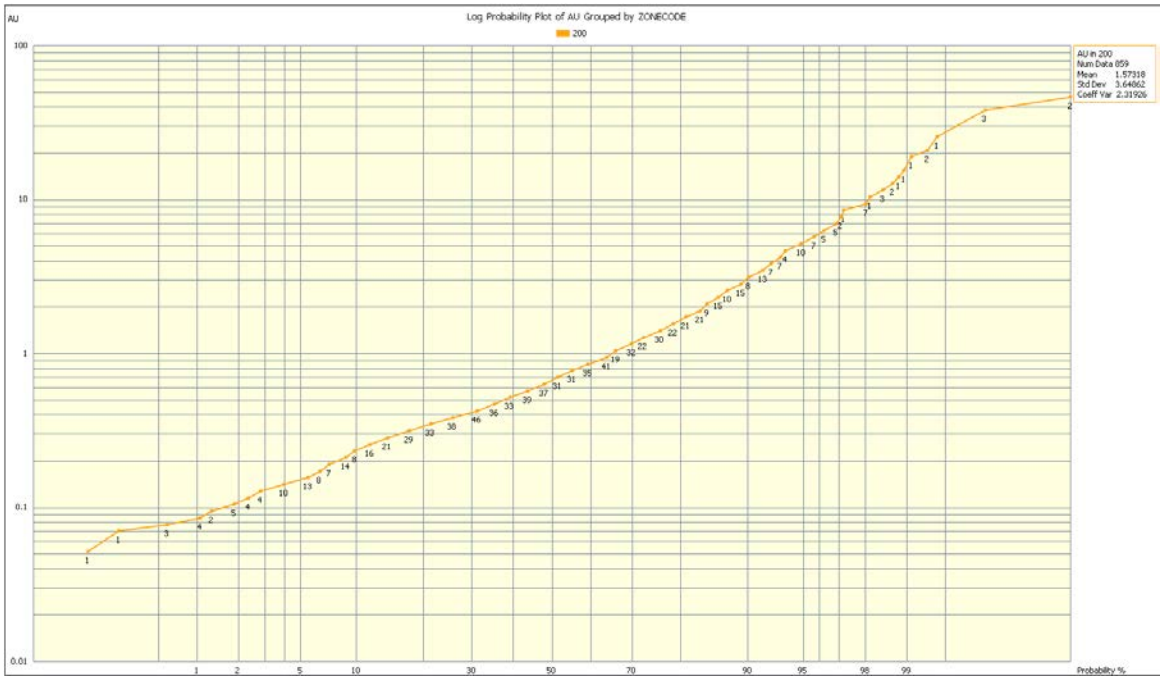


**Table 14.6.5 ZiaNE Summary Gold Statistics for 2 m Composites by Domain**

| Domain | 100 | 200 | 301 | 400 | 500 | 999 |
|---|---|---|---|---|---|---|
| Number | 350 | 4,337 | 1,231 | 413 | 531 | 6,068 |
| Minimum | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 |
| Maximum | 41.95 | 253.28 | 39.32 | 11.72 | 53.91 | 29.60 |
| Mean | 0.84 | 1.09 | 1.11 | 0.48 | 1.05 | 0.17 |
| Median | 0.29 | 0.39 | 0.71 | 0.28 | 0.36 | 0.09 |
| Std Dev | 2.66 | 4.96 | 2.09 | 0.90 | 3.18 | 0.74 |
| Variance | 7.07 | 24.63 | 4.35 | 0.80 | 10.11 | 0.54 |
| Coeff Var | 3.17 | 4.57 | 1.89 | 1.87 | 3.02 | 4.22 |
| **Percentiles** | | | | | | |
| 10 | 0.08 | 0.09 | 0.20 | 0.06 | 0.09 | 0.02 |
| 20 | 0.12 | 0.15 | 0.34 | 0.12 | 0.16 | 0.04 |
| 30 | 0.18 | 0.21 | 0.44 | 0.18 | 0.23 | 0.05 |
| 40 | 0.23 | 0.29 | 0.56 | 0.23 | 0.29 | 0.07 |
| 50 | 0.29 | 0.39 | 0.71 | 0.28 | 0.36 | 0.09 |
| 60 | 0.37 | 0.50 | 0.87 | 0.36 | 0.46 | 0.12 |
| 70 | 0.51 | 0.69 | 1.08 | 0.46 | 0.64 | 0.15 |
| 80 | 0.73 | 1.04 | 1.39 | 0.58 | 0.98 | 0.19 |
| 90 | 1.56 | 2.02 | 1.97 | 0.81 | 1.91 | 0.28 |
| 95 | 3.08 | 3.41 | 2.85 | 1.24 | 3.28 | 0.46 |
| 97.5 | 5.15 | 5.64 | 4.19 | 2.43 | 6.96 | 0.73 |
| 99 | 10.39 | 11.83 | 6.33 | 4.03 | 13.23 | 1.27 |

**Figure 14.6.5 ZiaNE Log - Probability Plot of 2 m Gold Composites – Domain 200**

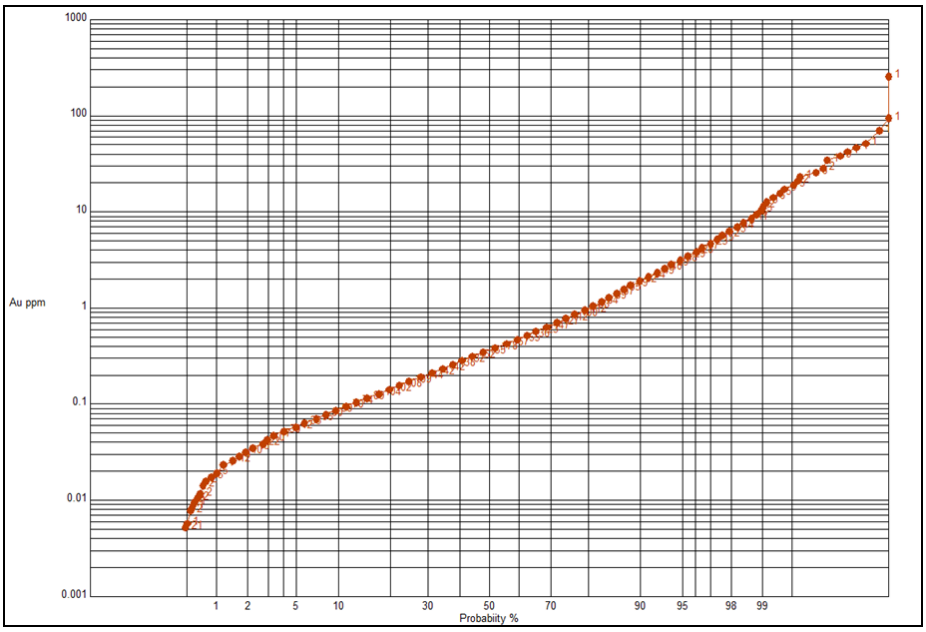


**Table 14.6.6 Teckraie and Verse Ouest Summary Gold Statistics for Downhole Composites by Domain**

| Domain | 331 | 332 | 340 | 341 |
|---|---|---|---|---|
| Number | 133 | 287 | 766 | 313 |
| Minimum | 0.01 | 0.01 | 0.08 | 0.15 |
| Maximum | 48.12 | 66.46 | 14.13 | 19.49 |
| Mean | 0.97 | 0.63 | 1.12 | 1.21 |
| Median | 0.43 | 0.16 | 0.93 | 0.97 |
| Std Dev | 4.15 | 3.97 | 0.92 | 1.40 |
| Variance | 17.25 | 15.73 | 0.85 | 1.96 |
| Coeff Var | 4.31 | 6.30 | 0.82 | 1.15 |
| **Percentiles** | | | | |
| 2.5 | 0.04 | 0.02 | 0.27 | 0.19 |
| 5 | 0.08 | 0.03 | 0.36 | 0.32 |
| 10 | 0.15 | 0.04 | 0.43 | 0.42 |
| 20 | 0.21 | 0.06 | 0.57 | 0.55 |
| 30 | 0.27 | 0.08 | 0.69 | 0.68 |
| 40 | 0.37 | 0.11 | 0.80 | 0.82 |
| 50 | 0.43 | 0.16 | 0.93 | 0.97 |
| 60 | 0.53 | 0.22 | 1.08 | 1.07 |
| 70 | 0.69 | 0.31 | 1.27 | 1.22 |
| 80 | 0.93 | 0.54 | 1.46 | 1.51 |
| 90 | 1.35 | 0.94 | 1.90 | 2.03 |
| 95 | 1.71 | 1.39 | 2.33 | 2.81 |
| 97.5 | 2.01 | 2.81 | 2.84 | 3.36 |
| 98 | 2.25 | 3.33 | 3.49 | 3.82 |
| 99 | 2.86 | 4.90 | 4.29 | 4.17 |

**Figure 14.6.6 Teckraie and Verse Ouest Log - Probability Plot of 2.5 m Gold Composites – Domain 340 and 341**

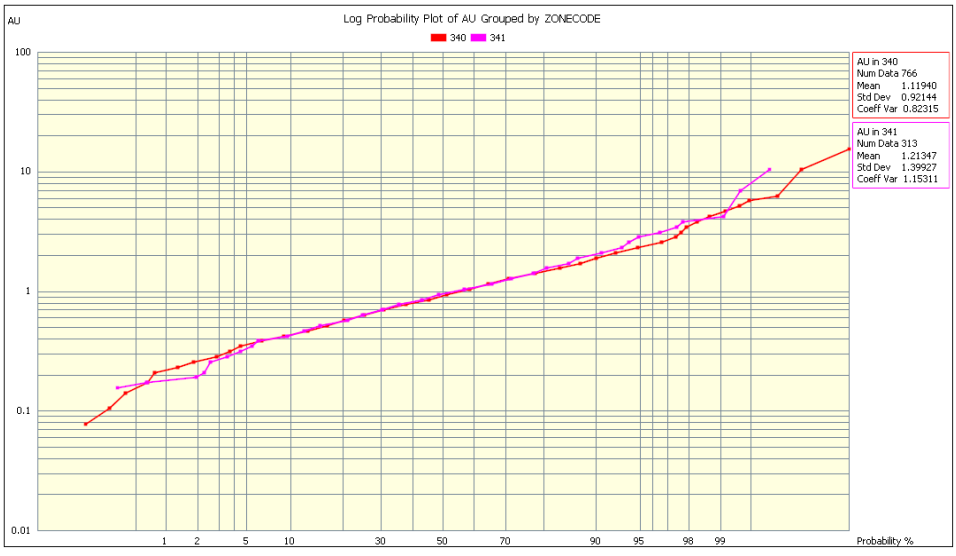


**Figure 14.6.7 Teckraie and Verse Ouest Log - Probability Plot of 1 m Gold Composites – Domain 331 and 332**

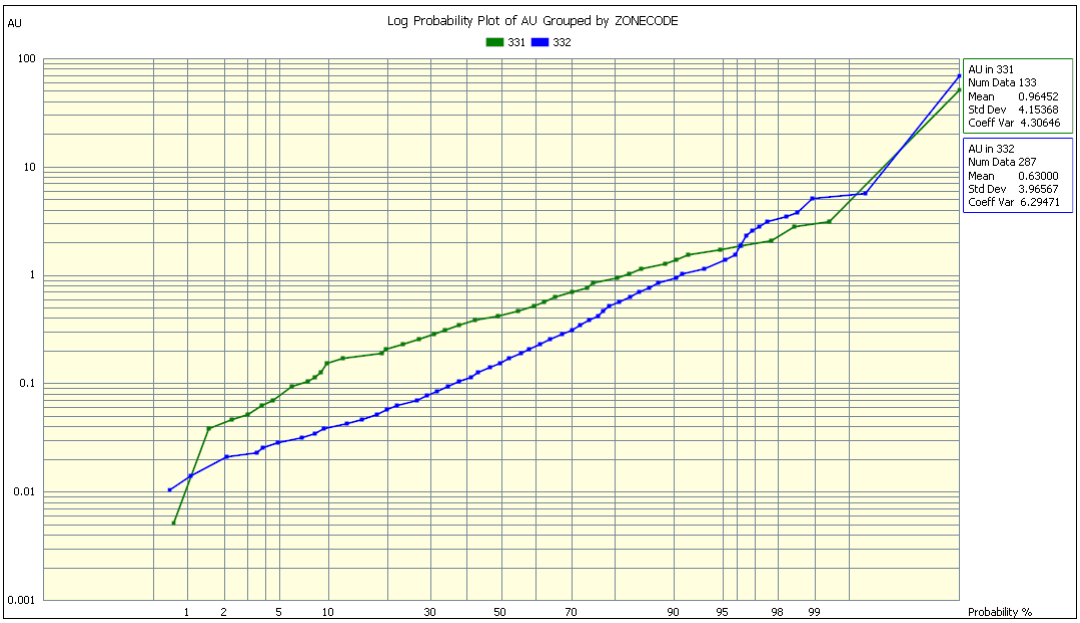
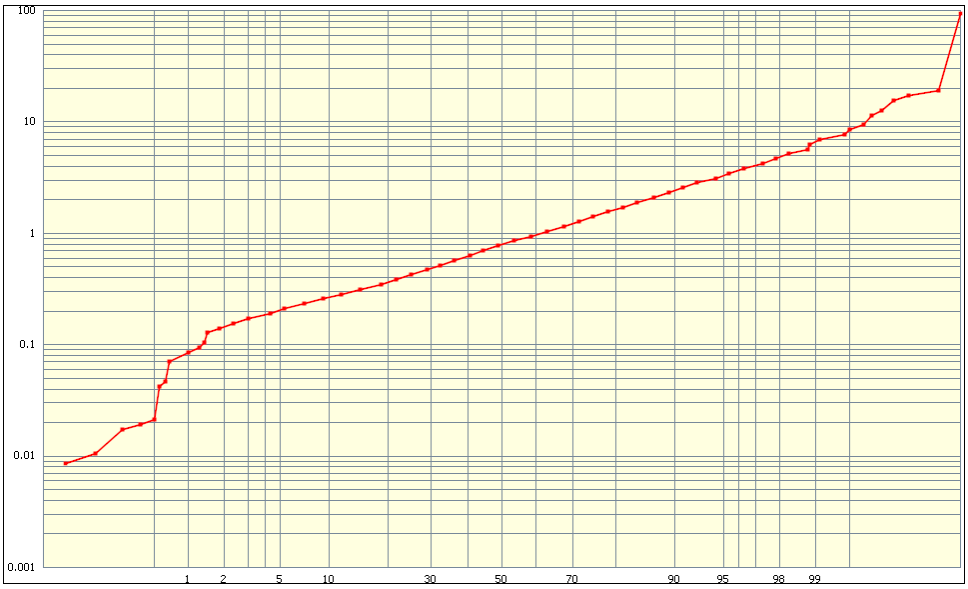



**Table 14.6.7 Aires Summary Gold Statistics for 2.5 m Composites by Domain**

| Domain | 301 | 303 |
|---|---|---|
| Number | 478 | 1,597 |
| Minimum | 0.01 | 0.01 |
| Maximum | 5.42 | 86.03 |
| Mean | 0.30 | 1.22 |
| Median | 0.18 | 0.79 |
| Std Dev | 0.44 | 2.51 |
| Variance | 0.19 | 6.31 |
| Coeff Var | 1.47 | 2.06 |
| **Percentiles** | | |
| 10 | 0.05 | 0.27 |
| 20 | 0.08 | 0.36 |
| 30 | 0.10 | 0.48 |
| 40 | 0.14 | 0.62 |
| 50 | 0.18 | 0.79 |
| 60 | 0.24 | 0.98 |
| 70 | 0.30 | 1.22 |
| 80 | 0.38 | 1.64 |
| 90 | 0.56 | 2.42 |
| 95 | 0.98 | 3.36 |
| 97.5 | 1.30 | 4.51 |
| 99 | 2.31 | 6.60 |

**Figure 14.6.8 Aires Log - Probability Plot of 2.5 m Gold Composites – Domain 303**




***Weathering Domain Analysis***

A comparison of the sample composite data for the oxide, transition and fresh weathered material was completed for each in situ deposit to determine if the composites required sub-domaining based on the interpreted weathering profile. Log probability plots and boundary analysis typically showed the gold population distributions for fresh, transition, and oxide as relatively similar for all weathering material types. Two examples of log probability plots for the Daapleu Domains 200 (Figure 14.6.9) and 400 (Figure 14.6.10) display the composite data split by weathering.

The graphs show the gold population distributions for fresh (green line) transition (red line) and oxide (blue line) are relatively similar for all weathering material types. Also, given the high complexity of the interpreted weathering surfaces and the likelihood of grade variation across the weathering interfaces, it is likely that grade changes are gradational and therefore the mineralised domains were not split by weathering for the purposes of grade interpolation for this resource estimate.

**Figure 14.6.9 Daapleu Probability Plot of Combined Domains 200 Gold Composites Split by Weathering**

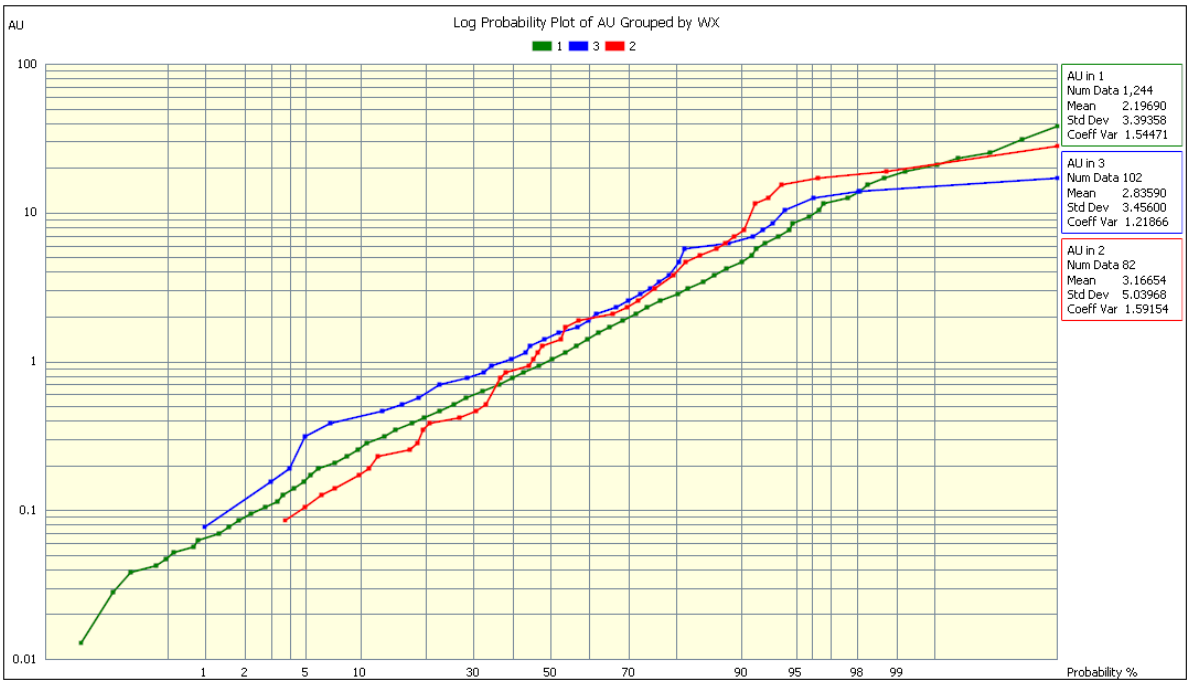
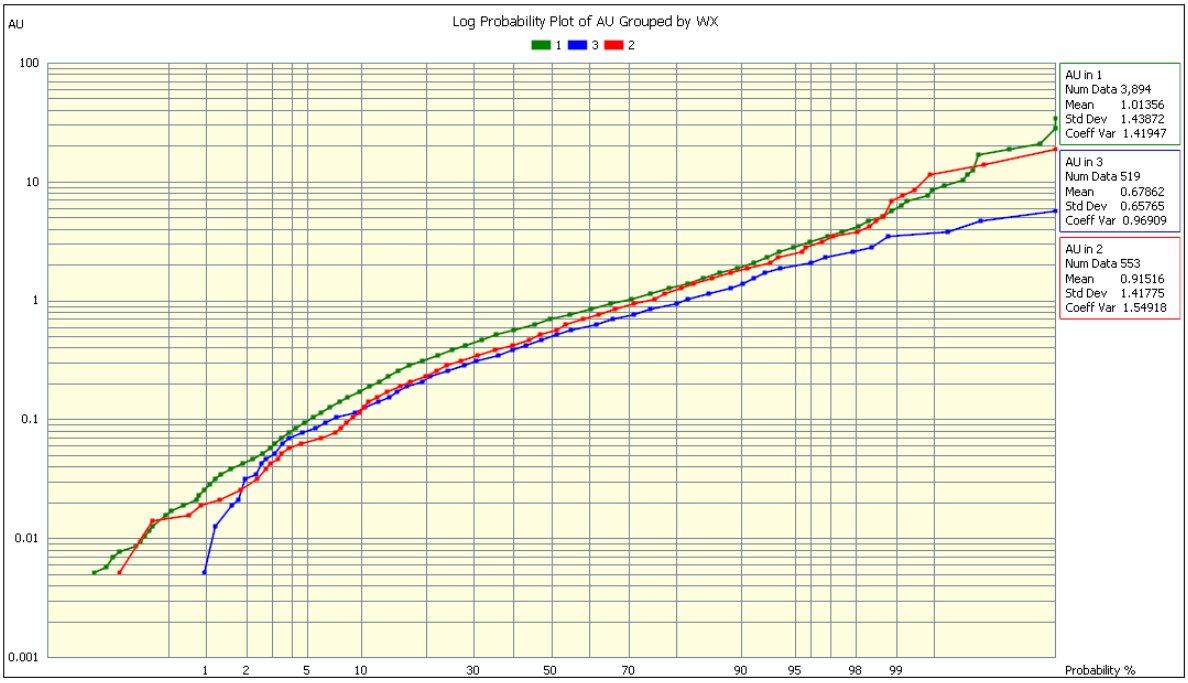



**Figure 14.6.10 Daapleu Probability Plot of Combined Domains 400 Gold Composites Split by Weathering**



***Evaluation of Grade Outliers and Top Cuts***

Several factors were reviewed in assessing the impact of applying gold composite top cuts, including:

- The shape of the gold grade distribution, especially at the upper tail.
- The spatial distribution of high grade outliers relative to nearby samples.
- Impact of the top cut on the average grade of the composite samples.
- Amount of metal removed after top cutting.

The appropriate high-grade cuts were applied on an individual domain basis and were typically all above the 99th percentile of the assay population. The top-cuts chosen by Cube for each deposit are summarised in Table 14.6.8 to Table 14.6.14. The number of cut samples is low and has had a material effect on reducing the coefficient of variation ('CV'), however the effect of these top-cuts is not expected to be material for the gold estimate.

**Table 14.6.8 Daapleu Summary Gold Statistics for 2 m Composites by Domain - Cut Gold**

| Domain | 100 | 150 | 200 | 301 | 400 | 500 | 999 |
|---|---|---|---|---|---|---|---|
| Number | 102 | 106 | 1,428 | 405 | 4,966 | 50 | 6434 |
| Minimum | 0.01 | 0.09 | 0.01 | 0.01 | 0.01 | 0.05 | 0.01 |
| Maximum | 8.20 | 3.60 | 35.55 | 6.42 | 33.96 | 7.76 | 6.79 |
| Mean | 0.81 | 0.75 | 2.30 | 0.38 | 0.97 | 0.64 | 0.09 |
| Median | 0.44 | 0.57 | 1.08 | 0.18 | 0.66 | 0.33 | 0.04 |
| Std Dev | 1.15 | 0.66 | 3.52 | 0.59 | 1.38 | 1.19 | 0.18 |
| Variance | 1.31 | 0.43 | 12.39 | 0.35 | 1.90 | 1.42 | 0.03 |
| Coeff Var | 1.41 | 0.88 | 1.53 | 1.56 | 1.43 | 1.87 | 2.13 |
| Top Cut | 6 | 2.5 | 25 | 5 | 15 | 2 | 3 |
| Number Cut | 1 | 5 | 4 | 1 | 10 | 4 | 7 |
| Percentile Cut | 99 | 97 | 99 | 99 | 99 | 93 | 99 |
| Cut Mean | 0.79 | 0.72 | 2.29 | 0.37 | 0.96 | 0.48 | 0.08 |
| Cut Median | 0.44 | 0.56 | 1.08 | 0.18 | 0.66 | 0.33 | 0.04 |
| Cut Std Dev | 1.02 | 0.55 | 3.43 | 0.56 | 1.21 | 0.50 | 0.16 |
| Cut Variance | 1.04 | 0.30 | 11.75 | 0.31 | 1.47 | 0.25 | 0.03 |
| Cut Coeff Var | 1.29 | 0.76 | 1.50 | 1.49 | 1.27 | 1.04 | 1.90 |

**Table 14.6.9 Gbeitouo Summary Gold Statistics for 2 m Composites by Domain - Cut Gold**

| Domain | 100 | 101 | 999 |
|---|---|---|---|
| Number | 850 | 735 | 3,566 |
| Minimum | 0.01 | 0.01 | 0.01 |
| Maximum | 28.56 | 16.58 | 2.21 |
| Mean | 1.36 | 0.57 | 0.05 |
| Median | 0.62 | 0.36 | 0.02 |
| Std Dev | 2.12 | 0.87 | 0.11 |
| Variance | 4.47 | 0.76 | 0.01 |
| Coeff Var | 1.56 | 1.53 | 2.08 |
| Top Cut | 15 | 8 | 1 |
| Number Cut | 2 | 1 | 9 |
| Percentile Cut | 99.88 | 99.99 | 99.78 |
| Cut Mean | 1.33 | 0.56 | 0.05 |
| Cut Median | 0.62 | 0.36 | 0.02 |
| Cut Std Dev | 1.86 | 0.70 | 0.09 |
| Cut Variance | 3.44 | 0.49 | 0.01 |
| Cut Coeff Var | 1.39 | 1.25 | 1.76 |

**Table 14.6.10 Mont Ity / Ity Flat Summary Gold Statistics for 2 m Composites by Domain - Cut Gold**

| Domain | 100 | 200 | 301 | 400 | 500 | 600 | 888 | 999 |
|---|---|---|---|---|---|---|---|---|
| Number | 51 | 455 | 2,399 | 10,610 | 349 | 290 | 4,880 | 8,062 |
| Minimum | 0.05 | 0.03 | 0.01 | 0.01 | 0.01 | 0.02 | 0.01 | 0.01 |
| Maximum | 1.84 | 230.93 | 109.11 | 360.57 | 14.88 | 14.08 | 39.12 | 68.66 |
| Mean | 0.48 | 1.28 | 1.27 | 1.93 | 0.69 | 1.07 | 0.23 | 0.16 |
| Median | 0.38 | 0.43 | 0.65 | 0.46 | 0.33 | 0.48 | 0.13 | 0.04 |
| Std Dev | 0.34 | 10.98 | 2.78 | 6.69 | 1.30 | 1.77 | 0.77 | 1.07 |
| Variance | 0.12 | 120.51 | 7.70 | 44.77 | 1.68 | 3.12 | 0.59 | 1.14 |
| Coeff Var | 0.71 | 8.61 | 2.19 | 3.47 | 1.88 | 1.66 | 3.36 | 6.74 |
| Top Cut | - | 5.00 | 12.00 | 70.00 | 8.00 | 10.00 | 10.00 | 10.00 |
| Number Cut | 0 | 3 | 8 | 11 | 1 | 3 | 4 | 11 |
| Percentile Cut | 100 | 99 | 99 | 99 | 99 | 99 | 99 | 99 |
| Cut Mean | 0.48 | 0.68 | 1.21 | 1.86 | 0.66 | 1.04 | 0.22 | 0.14 |
| Cut Median | 0.38 | 0.43 | 0.65 | 0.46 | 0.33 | 0.48 | 0.13 | 0.04 |
| Cut Std Dev | 0.34 | 0.77 | 1.56 | 5.10 | 1.07 | 1.61 | 0.50 | 0.52 |
| Cut Variance | 0.12 | 0.60 | 2.44 | 26.02 | 1.14 | 2.60 | 0.25 | 0.27 |
| Cut Coeff Var | 0.71 | 1.14 | 1.29 | 2.74 | 1.61 | 1.55 | 2.26 | 3.67 |

**Table 14.6.11 Walter Summary Gold Statistics for 2 m Composites by Domain - Cut Gold**

| Domain | 900 | 800 | 600 | 500 | 300 | 211 | 210 |
|---|---|---|---|---|---|---|---|
| Number | 736 | 6 | 5 | 52 | 187 | 14 | 8 |
| Minimum | 0.02 | 0.42 | 0.35 | 0.25 | 0.21 | 0.30 | 0.47 |
| Maximum | 15.52 | 26.84 | 1.01 | 7.19 | 5.54 | 10.34 | 1.94 |
| Mean | 0.60 | 8.26 | 0.57 | 1.04 | 0.89 | 3.52 | 0.92 |
| Median | 0.27 | 3.69 | 0.45 | 0.57 | 0.69 | 1.75 | 0.79 |
| Std Dev | 1.18 | 10.14 | 0.26 | 1.40 | 0.67 | 3.50 | 0.49 |
| Variance | 1.40 | 102.80 | 0.07 | 1.95 | 0.45 | 12.22 | 0.24 |
| Coeff Var | 1.96 | 1.23 | 0.46 | 1.34 | 0.75 | 0.99 | 0.54 |
| Top Cut | 8.25 | 999 | 999 | 999 | 4 | 7.5 | 1.3 |
| Number Cut | 2 | - | - | - | 2 | 3 | 1 |
| Percentile Cut | 99.6 | - | - | - | 99 | 79 | 92 |
| Cut Mean | 0.59 | 8.26 | 0.57 | 1.04 | 0.89 | 3.12 | 0.69 |
| Cut Median | 0.27 | 3.69 | 0.38 | 0.57 | 0.70 | 1.24 | 0.50 |
| Cut Std Dev | 1.05 | 10.14 | 0.26 | 1.40 | 0.58 | 2.78 | 0.34 |
| Cut Variance | 1.10 | 102.80 | 0.07 | 1.95 | 0.34 | 7.75 | 0.11 |
| Cut Coeff Var | 1.78 | 1.23 | 0.46 | 1.34 | 0.65 | 0.89 | 0.49 |

**Table 14.6.11 Walter Summary Gold Statistics for 2 m Composites by Domain - Cut Gold (continued)**

| Domain | 209 | 208 | 207 | 206 | 205 | 204 | 203 |
|---|---|---|---|---|---|---|---|
| Number | 10 | 14 | 4 | 119 | 91 | 7 | 78 |
| Minimum | 0.15 | 0.36 | 0.29 | 0.05 | 0.09 | 0.49 | 0.07 |
| Maximum | 1.50 | 1.57 | 3.46 | 12.00 | 5.96 | 2.40 | 9.00 |
| Mean | 0.67 | 0.64 | 1.33 | 1.66 | 1.13 | 1.50 | 1.66 |
| Median | 0.52 | 0.41 | 0.29 | 0.72 | 0.61 | 0.81 | 0.81 |
| Std Dev | 0.46 | 0.37 | 1.48 | 2.43 | 1.20 | 0.85 | 2.00 |
| Variance | 0.21 | 0.14 | 2.19 | 5.92 | 1.44 | 0.72 | 3.99 |
| Coeff Var | 0.68 | 0.58 | 1.12 | 1.47 | 1.06 | 0.56 | 1.21 |
| Top Cut | 999 | 999 | 999 | 12 | 999 | 2.4 | 9 |
| Number Cut | - | - | - | 1 | - | 1 | 1 |
| Percentile Cut | - | - | - | 99 | - | 99 | 99 |
| Cut Mean | 0.67 | 0.64 | 1.33 | 1.66 | 1.13 | 1.50 | 1.66 |
| Cut Median | 0.52 | 0.41 | 0.29 | 0.72 | 0.61 | 0.81 | 0.81 |
| Cut Std Dev | 0.46 | 0.37 | 1.48 | 2.43 | 1.20 | 0.85 | 2.00 |
| Cut Variance | 0.21 | 0.14 | 2.19 | 5.92 | 1.44 | 0.72 | 3.99 |
| Cut Coeff Var | 0.68 | 0.58 | 1.12 | 1.47 | 1.06 | 0.56 | 1.21 |

**Table 14.6.11 Walter Summary Gold Statistics for 2m Composites by Domain - Cut Gold (continued)**

| Domain | 202 | 201 | 200 | 104 | 103 | 102 | 101 |
|---|---|---|---|---|---|---|---|
| Number | 32 | 58 | 859 | 19 | 3 | 23 | 19 |
| Minimum | 0.09 | 0.07 | 0.05 | 0.26 | 0.29 | 0.13 | 0.10 |
| Maximum | 2.00 | 2.79 | 30.00 | 2.50 | 2.24 | 2.75 | 1.89 |
| Mean | 0.67 | 0.68 | 1.52 | 0.91 | 1.36 | 0.79 | 0.78 |
| Median | 0.44 | 0.52 | 0.67 | 0.66 | 0.15 | 0.55 | 0.59 |
| Std Dev | 0.49 | 0.54 | 3.09 | 0.66 | 0.99 | 0.67 | 0.54 |
| Variance | 0.24 | 0.29 | 9.56 | 0.44 | 0.97 | 0.45 | 0.29 |
| Coeff Var | 0.73 | 0.79 | 2.04 | 0.72 | 0.73 | 0.86 | 0.69 |
| Top Cut | 2 | 999 | 30 | 2.5 | 999 | 2.75 | 1.89 |
| Number Cut | 1 | - | 5 | 1 | - | 1 | 1 |
| Percentile Cut | 96 | - | 99 | 95 | - | 96 | 95 |
| Cut Mean | 0.67 | 0.68 | 1.52 | 0.91 | 1.36 | 0.79 | 0.78 |
| Cut Median | 0.44 | 0.52 | 0.67 | 0.66 | 0.15 | 0.55 | 0.59 |
| Cut Std Dev | 0.49 | 0.54 | 3.09 | 0.66 | 0.99 | 0.67 | 0.54 |
| Cut Variance | 0.24 | 0.29 | 9.56 | 0.44 | 0.97 | 0.45 | 0.29 |
| Cut Coeff Var | 0.73 | 0.79 | 2.04 | 0.72 | 0.73 | 0.86 | 0.69 |

**Table 14.6.12 ZiaNE Summary Gold Statistics for 2 m Composites by Domain - Cut Gold**

| Domain | 100 | 200 | 301 | 400 | 500 | 999 |
|---|---|---|---|---|---|---|
| Number | 350 | 4,337 | 1,231 | 413 | 531 | 6,068 |
| Minimum | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 |
| Maximum | 41.95 | 253.28 | 39.32 | 11.72 | 53.91 | 29.60 |
| Mean | 0.84 | 1.09 | 1.11 | 0.48 | 1.05 | 0.17 |
| Median | 0.29 | 0.39 | 0.71 | 0.28 | 0.36 | 0.09 |
| Std Dev | 2.66 | 4.96 | 2.09 | 0.90 | 3.18 | 0.74 |
| Variance | 7.07 | 24.63 | 4.35 | 0.80 | 10.11 | 0.54 |
| Coeff Var | 3.17 | 4.57 | 1.89 | 1.87 | 3.02 | 4.22 |
| Top Cut | 10 | 30 | 15 | 8 | 20 | 8 |
| Number Cut | 5 | 10 | 6 | 2 | 3 | 6 |
| Percentile Cut | 98.5 | 99 | 99 | 99 | 99 | 99 |
| Cut Mean | 0.73 | 0.99 | 1.05 | 0.47 | 0.96 | 0.16 |
| Cut Median | 0.29 | 0.39 | 0.71 | 0.28 | 0.36 | 0.09 |
| Cut Std Dev | 1.46 | 2.47 | 1.44 | 0.79 | 2.16 | 0.39 |
| Cut Variance | 2.13 | 6.09 | 2.08 | 0.62 | 4.66 | 0.15 |
| Cut Coeff Var | 1.99 | 2.49 | 1.37 | 1.68 | 2.24 | 2.42 |

**Table 14.6.13 Teckraie and Verse Ouest Summary Gold Statistics for 2.5 m Composites by Domain - Cut Gold**

| Domain | 331 | 332 | 340 | 341 |
|---|---|---|---|---|
| Number | 133 | 287 | 766 | 313 |
| Minimum | 0.01 | 0.01 | 0.08 | 0.15 |
| Maximum | 48.12 | 66.46 | 14.13 | 19.49 |
| Mean | 0.97 | 0.63 | 1.12 | 1.21 |
| Median | 0.43 | 0.16 | 0.93 | 0.97 |
| Std Dev | 4.15 | 3.97 | 0.92 | 1.40 |
| Variance | 17.25 | 15.73 | 0.85 | 1.96 |
| Coeff Var | 4.31 | 6.30 | 0.82 | 1.15 |
| Top Cut | 5 | 5 | 5 | 5 |
| Number Cut | 1 | 3 | 6 | 3 |
| Percentile Cut | 99 | 99 | 99 | 99 |
| Cut Mean | 0.64 | 0.41 | 1.10 | 1.14 |
| Cut Median | 0.43 | 0.16 | 0.93 | 0.97 |
| Cut Std Dev | 0.65 | 0.76 | 0.73 | 0.80 |
| Cut Variance | 0.43 | 0.57 | 0.54 | 0.65 |
| Cut Coeff Var | 1.02 | 1.83 | 0.67 | 0.70 |

**Table 14.6.14 Aires Summary Gold Statistics for 2.5 m Composites by Domain - Cut Gold**

| Domain | 301 | 303 |
|---|---|---|
| Number | 478 | 1,597 |
| Minimum | 0.01 | 0.01 |
| Maximum | 5.42 | 86.03 |
| Mean | 0.30 | 1.22 |
| Median | 0.18 | 0.79 |
| Std Dev | 0.44 | 2.51 |
| Variance | 0.19 | 6.31 |
| Coeff Var | 1.47 | 2.06 |
| Top Cut | 5 | 12 |
| Number Cut | 1 | 5 |
| Percentile Cut | 99 | 99 |
| Cut Mean | 0.30 | 1.16 |
| Cut Median | 0.18 | 0.79 |
| Cut Std Dev | 0.43 | 1.27 |
| Cut Variance | 0.18 | 1.62 |
| Cut Coeff Var | 1.45 | 1.10 |

## 14.7 Variography

Variography has been modelled to analyse the spatial continuity within the mineralised zones and to determine the appropriate estimation inputs to the interpolation process. Variography and evaluation of suitable estimation parameters based on the final variogram models was undertaken using Isatis software.

The experimental semi-variograms were generated on the top-cut composite data. Declustering of the data was undertaken to clean up the areas of clustered sampling and transformed to a Gaussian distribution to reduce the influence of extreme values of variance within the lag spaces. The variogram modelling process followed by Cube involved the following steps:

- Calculate and model the downhole variogram to characterise the nugget effect.
- Calculate and model the variograms for each of the domains in the direction of maximum continuity and the orthogonal direction.
- Vary the lag distances in the major, semi-major and minor directions to optimise variograms for each these directions.

The final variogram models have been back transformed from Gaussian space into the raw values for all insitu deposit domains and the variogram model parameters are summarised in Table 14.7.1 to Table 14.7.5. An example of the Gaussian and back-transformed variogram models for the Daapleu Domain 200 is provided in Figure 14.7.1 and Figure 14.7.2, respectively. Variogram models for the 2.5 m composites for the rock dump and heap leach pad are summarised in Table 14.7.6 and Table 14.7.7.

**Figure 14.7.1 Daapleu Deposit Domain 200 – Cut Au Gaussian Transformed Variogram**

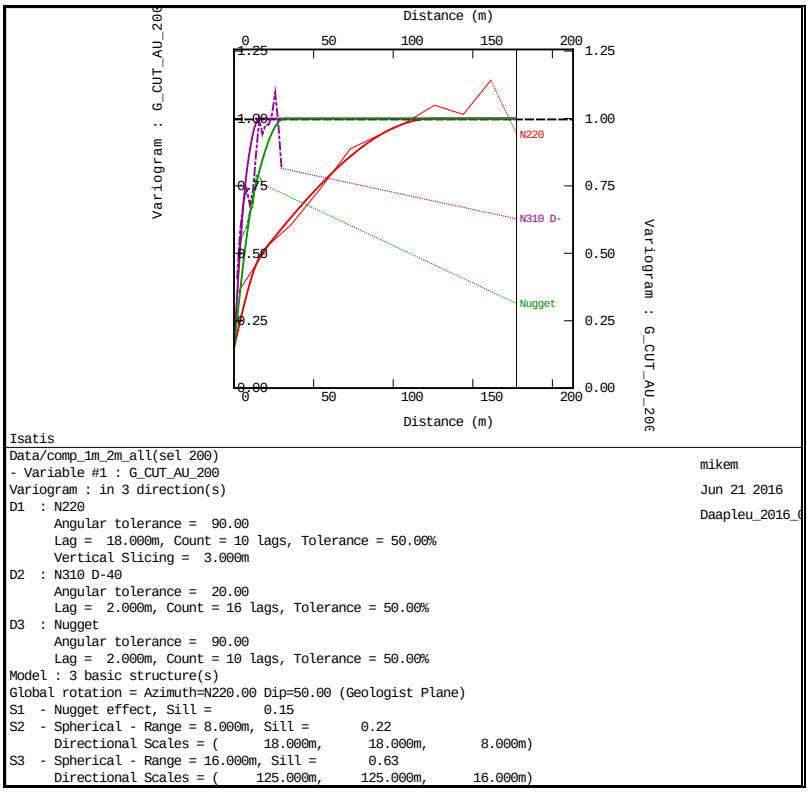



**Figure 14.7.2 Daapleu Deposit Domain 200 – Cut Au Back Transformed Variogram**

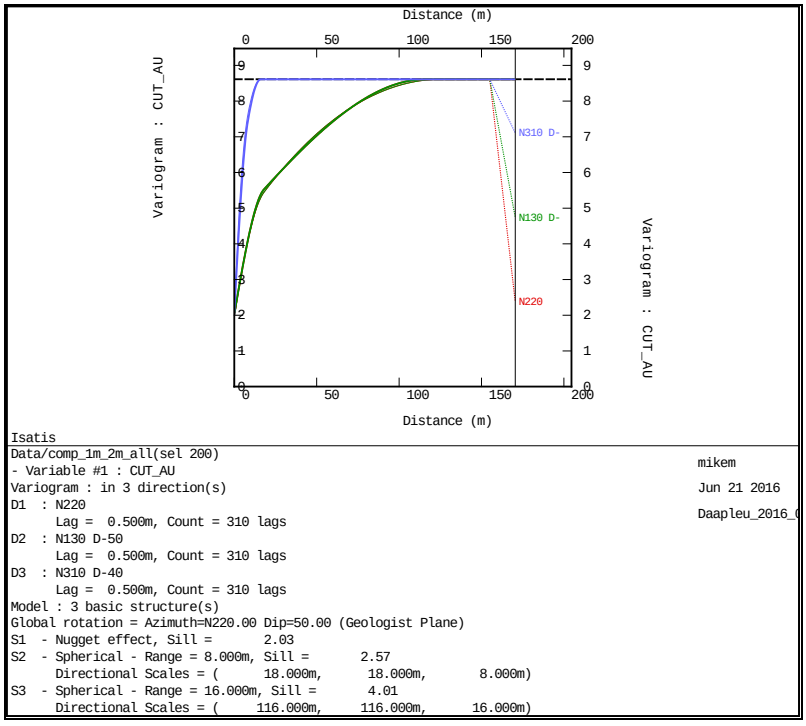


**Table 14.7.1 Daapleu Relative Variogram Models - Cut Gold**

| Domain (Variable) | Attribute | Nugget | Spherical 1 | | | | Spherical 2 | | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | sill | major | semi | minor | sill | major | semi | minor | A | +X | -Z |
| 100 | Cut Au | 0.155 | 0.553 | 22 | 22 | 8 | 0.292 | 150 | 150 | 15 | 220 | 50 | 0 |
| 150 | Cut Au | 0.136 | 0.535 | 22 | 22 | 8 | 0.329 | 150 | 150 | 15 | 220 | 50 | 0 |
| 200 | Cut Au | 0.236 | 0.298 | 18 | 18 | 8 | 0.466 | 116 | 116 | 16 | 220 | 50 | 0 |
| 301 | Cut Au | 0.162 | 0.181 | 15 | 15 | 6 | 0.657 | 99 | 99 | 12 | 90 | 0 | 0 |
| 400 | Cut Au | 0.198 | 0.574 | 21 | 21 | 8 | 0.228 | 146 | 146 | 15 | 240 | 55 | 0 |
| 500 | Cut Au | 0.198 | 0.513 | 22 | 22 | 8 | 0.289 | 148 | 148 | 15 | 220 | 50 | 0 |
| 999 | Cut Au | 0.290 | 0.444 | 30 | 30 | 12 | 0.266 | 550 | 550 | 50 | 240 | 55 | 0 |

**Table 14.7.2 Gbeitouo Relative Variogram Models - Cut Gold**

| Domain | Nugget | Spherical 1 | | | | Spherical 2 | | | | Spherical 3 | | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | sill | major | semi | minor | sill | major | semi | minor | sill | major | semi | minor | A | +X | -Z |
| 100 | 0.345 | 0.317 | 3 | 3 | 3 | 0.193 | 28 | 28 | 10 | 0.145 | 70 | 70 | 20 | 210 | 50 | -180 |
| 101 | 0.318 | 0.350 | 8 | 8 | 5 | 0.213 | 25 | 25 | 15 | 0.118 | 50 | 50 | 20 | 215 | 50 | -180 |
| 999 | 0.339 | 0.453 | 25 | 25 | 20 | 0.208 | 110 | 110 | 30 | 0.000 | 0 | 0 | 0 | 210 | 50 | 180 |

**Table 14.7.3 Mont Ity / Ity Flat Relative Variogram Models - Cut Gold**

| Domain | Nugget | Spherical Model 1st Structure | | | | Spherical Model 2nd Structure | | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Sill | Maj. | Semi | Minor | Sill | Maj. | Semi | Minor | A | +X | -Z |
| 100 | 0.0256 | 0.064 | 14 | 14 | 10 | 0.0203 | 90 | 90 | 18 | 270 | 50 | 0 |
| 200 | 0.148 | 0.273 | 14 | 14 | 10 | 0.066 | 90 | 90 | 18 | 255 | 30 | 0 |
| 301 | 0.362 | 0.595 | 50 | 50 | 8 | 0.561 | 285 | 285 | 15 | 0 | 0 | 0 |
| 400 | 9 | 11.2 | 12 | 12 | 8 | 1.9 | 88 | 88 | 18 | 220 | 40 | 0 |
| 500 | 0.36 | 0.61 | 13 | 13 | 9 | 0.145 | 90 | 90 | 18 | 220 | 40 | 0 |
| 600 | 0.58 | 0.8 | 6 | 6 | 7 | 1.08 | 24 | 24 | 14 | 215 | 65 | 0 |
| 888 | 0.125 | 0.075 | 33 | 33 | 7 | 0.02 | 260 | 260 | 14 | 220 | 40 | 0 |
| 999 | 0.15 | 0.105 | 35 | 35 | 14 | 0.029 | 250 | 250 | 38 | 220 | 40 | 0 |

**Table 14.7.4 Walter Relative Variogram Models - Cut Gold**

| Domain | Nugget | Spherical 1 | | | | Spherical 2 | | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | sill | major | semi | minor | sill | major | semi | minor | A | +X | -Z |
| 101 | 0.62 | 0.35 | 44.5 | 50.3 | 10.3 | 0.03 | 83.7 | 14.4 | 10.3 | 210 | 51 | -90 |
| 102 | 0.46 | 0.52 | 63.8 | 63.8 | 7.9 | 0.02 | 7.4 | 7.4 | 3.8 | 211 | 56 | -90 |
| 104 | 0.62 | 0.04 | 8.0 | 8.0 | 10.0 | 0.34 | 50.0 | 50.0 | 10.0 | 210 | 51 | -90 |
| 201 | 0.51 | 0.33 | 40.0 | 10.0 | 15.0 | 0.16 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 202 | 0.51 | 0.32 | 40.0 | 10.0 | 15.0 | 0.17 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 203 | 0.50 | 0.35 | 40.0 | 10.0 | 15.0 | 0.15 | 60.0 | 40.0 | 15.0 | 215 | 32 | 180 |
| 205 | 0.47 | 0.40 | 40.0 | 10.0 | 15.0 | 0.14 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 206 | 0.51 | 0.37 | 40.0 | 10.0 | 15.0 | 0.12 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 208 | 0.47 | 0.37 | 40.0 | 10.0 | 15.0 | 0.16 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 210 | 0.51 | 0.31 | 40.0 | 10.0 | 15.0 | 0.19 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 211 | 0.47 | 0.37 | 40.0 | 10.0 | 15.0 | 0.17 | 60.0 | 40.0 | 15.0 | 225 | 25 | -180 |
| 300 | 0.47 | 0.26 | 34.1 | 22.6 | 8.0 | 0.26 | 137.6 | 35.4 | 11.1 | 30 | 11 | 5 |
| 500 | 0.57 | 0.43 | 22.7 | 22.7 | 22.7 | 0.00 | 0.0 | 0.0 | 0.0 | 0 | 0 | 0 |
| 900-HG | 0.54 | 0.20 | 10.0 | 10.0 | 10.0 | 0.25 | 100.0 | 55.0 | 10.0 | 225 | 30 | -180 |
| 900-HG | 0.57 | 0.19 | 10.0 | 10.0 | 6.0 | 0.25 | 110.0 | 110.0 | 25.0 | 225 | 30 | -180 |
| 900 | 0.57 | 0.25 | 30.0 | 85.0 | 8.0 | 0.18 | 185.0 | 85.0 | 30.0 | 220 | 20 | -180 |

**Table 14.7.5 ZiaNE Relative Variogram Models - Cut Gold**

| Domain | Nugget | Spherical 1 | | | | Spherical 2 | | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | sill | major | semi | minor | sill | major | semi | minor | A | +X | -Z |
| 100 | 0.42 | 0.50 | 10 | 10 | 5 | 0.08 | 50 | 50 | 20 | 215 | 35 | -180 |
| 200 | 0.49 | 0.42 | 10 | 10 | 8 | 0.09 | 65 | 65 | 20 | 215 | 32 | -180 |
| 301 | 0.23 | 0.62 | 60 | 45 | 10 | 0.14 | 90 | 60 | 20 | 35 | 0 | 0 |
| 400 | 0.37 | 0.35 | 3 | 3 | 3 | 0.28 | 60 | 60 | 20 | 215 | 30 | -180 |
| 500 | 0.49 | 0.34 | 15 | 15 | 10 | 0.17 | 75 | 75 | 20 | 215 | 32 | -180 |

**Table 14.7.6 Teckraie and Verse Ouest Relative Variogram Models**

| Domain (Variable) | Nugget | Spherical 1 | | | | Spherical 2 | | | | Surpac Range Ratios | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | sill | maj. | semi | minor | sill | maj. | semi | minor | Spherical 1 | | Spherical 2 | |
| 331 (Au) | 0.390 | 0.113 | 10 | 10 | 10 | 0.497 | 68 | 68 | 68 | 1 | 1 | 1 | 1 |
| 340 (Au) | 0.166 | 0.351 | 12 | 12 | 7 | 0.482 | 78 | 78 | 18 | 1 | 1.71 | 1 | 4.33 |
| 340 (Cu) | 0.132 | 0.360 | 12 | 12 | 7 | 0.508 | 165 | 165 | 18 | 1 | 1.71 | 1 | 9.17 |
| 340 (S) | 0.291 | 0.424 | 18 | 18 | 18 | 0.285 | 75 | 75 | 75 | 1 | 1 | 1 | 1 |
| 332 (Au) | 0.199 | 0.467 | 10 | 10 | 10 | 0.334 | 75 | 75 | 75 | 1 | 1 | 1 | 1 |
| 341 (Au) | 0.253 | 0.397 | 30 | 30 | 10 | 0.351 | 245 | 245 | 20 | 1 | 3 | 1 | 12.25 |
| 341 (Cu) | 0.241 | 0.143 | 30 | 30 | 8 | 0.617 | 113 | 113 | 14 | 1 | 3.75 | 1 | 8.07 |
| 341 (S) | 0.198 | 0.330 | 10 | 10 | 10 | 0.473 | 52 | 52 | 52 | 1 | 1 | 1 | 1 |

**Table 14.7.7 Aires Relative Variogram Models**

| Domain (Variable) | Nugget | Spherical 1 | | | | Spherical 2 | | | | Surpac Range Ratios | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | sill | maj. | semi | minor | sill | maj. | semi | minor | Spherical 1 | | Spherical 2 | |
| 301 (Au) | 0.182 | 0.316 | 15 | 15 | 10 | 0.502 | 108 | 108 | 17 | 1 | 1.5 | 1 | 6.4 |
| 303 (Au) | 0.295 | 0.430 | 30 | 30 | 10 | 0.275 | 190 | 190 | 14 | 1 | 3 | 1 | 13.6 |
| 303 (Cu) | 0.221 | 0.359 | 30 | 30 | 10 | 0.420 | 200 | 200 | 16 | 1 | 3 | 1 | 12.5 |
| 303 (S) | 0.156 | 0.335 | 7 | 7 | 7 | 0.508 | 37 | 37 | 37 | 1 | 1 | 1 | 1 |

## 14.8 Block Model Set Up

Regular block models were set up in Surpac using UTM-WGS84 grid coordinates for storage of the of the mineral resource estimate for each individual deposit.

Following discussions with site personnel, a decision was reached to use a 5 mE x 5 mN x 2.5 mRL SMU size block for the estimation of the in situ deposits, which is deemed to be broadly reflective of the anticipated mining selectivity for mining of the open pit operations. In some instances a rotated block model was considered most appropriate due to the overall trend of mineralisation.

The Surpac block model extents were large enough to allow for the pit optimisation to be completed but constrained enough to limit the amount of electronic storage required. The block model name for each of the deposits are summarised in Table 14.8.1 and final block model parameters are summarised in Table 14.8.2 to Table 14.8.4.

**Table 14.8.1 Block Model File Names**

| Deposit | Block Model File ID |
|---|---|
| Daapleu | bm_smu_daapleu_jul16.mdl |
| Gbeitouo | bm_smu_gbeit_jul16.mdl |
| Mont Ity/Ity Flat | bm_smu_ity_jul16.mdl |
| Walter | bm_smu_Walter_jul16.mdl |
| ZiaNE | bm_smu_ziane_jul16.mdl |
| Teckraie and Verse Ouest | bm_final_ity_dumps_jul16.mdl |
| Aires | bm_final_ity_aires_de_lixiviation_jul16.mdl |

**Table 14.8.2 Final Block Model Definition Summary**

| Type | Daapleu | Gbeitouo | Mont Ity / Ity Flat | Walter | ZiaNE | Teckraie & Verse Ouest | Aires |
|---|---|---|---|---|---|---|---|
| Origin Y (m) | 757,250 | 760,300 | 759,400 | 760,200 | 760,400 | 759,300 | 760,350 |
| Origin X (m) | 598,900 | 601,100 | 597,500 | 598,800 | 597,700 | 597,200 | 597,500 |
| Origin Z (m) | -100 | 50 | -75 | 100 | -100 | 200 | 260 |
| Extent Northing (m) | 1,950 | 800 | 1,650 | 1,000 | 1,000 | 820 | 700 |
| Extent Easting (m) | 2,500 | 600 | 1,125 | 650 | 1,000 | 1,040 | 860 |
| Extent RL (m) | -450 | 250 | 400 | 200 | 500 | 120 | 140 |
| Panel Block Y (m) | 20 | 25 | 25 | 25 | 20 | 20 | 20 |
| Panel Block X (m) | 20 | 25 | 25 | 25 | 20 | 20 | 20 |
| Panel Block Z (m) | 5 | 5 | 5 | 5 | 5 | 2.5 | 2.5 |
| Sub-Cell Y (m) | 5 | 5 | 5 | 5 | 5 | 2.5 | 2.5 |
| Sub-Cell X (m) | 5 | 5 | 5 | 5 | 5 | 2.5 | 2.5 |
| Sub-Cell Z (m) | 2.5 | 2.5 | 2.5 | 2.5 | 2.5 | 2.5 | 2.5 |
| Discretisation Y | 6 | 10 | 6 | 5 | 6 | 4 | 4 |
| Discretisation X | 6 | 10 | 6 | 5 | 6 | 4 | 4 |
| Discretisation Z | 3 | 3 | 3 | 2 | 3 | 2 | 2 |
| Rotation | None | 030 | 045 | 045 | 030 | None | None |

The block model attributes and descriptions are summarised in Table 14.8.3.

**Table 14.8.3 Block Model Attributes – In Situ Deposits**

| Attribute Name | Background | Description |
|---|---|---|
| as_ok_ppm | -1 | Arsenic OK estimate - As (ppm) |
| au_final_ppm | 0.001 | Gold final estimate for reporting - Au (ppm) |
| au_luc_ppm | 0.001 | Gold LUC estimate - Au (ppm) |
| au_ok_avd | -1 | Gold OK average distance to samples |
| au_ok_ns | -1 | Gold OK number of samples used |
| au_ok_ppm | -1 | Gold OK panel estimate - Au (ppm) |
| au_ok_sor | -1 | Gold OK slope of regression |
| cu_ok_ppm | -1 | Copper OK estimate - Cu (ppm) |
| density | -1 | Dry Bulk Density |
| depletion | 1 | Depletion state: 0=Air; 0=Mined; 1=In situ |
| facies | -1 | Rock Facies Code |
| min_prop | 0 | Proportion inside interpretation domain |
| minzone_au | -1 | Gold Estimation/Mineralisation Domain Code |
| ox_state | 3 | Weathering state: 1=Fresh; 2=Transitional; 3=Oxide; 0=Air |
| res_class | -1 | Classification: 0=Air; 1=Measured; 2=Indicated; 3=Inferred |
| s_ok_pct | -1 | Sulphur OK estimate - S (wt%) |

**Table 14.8.4 Block Model Attributes – Rock Dumps and Heap Leach Pad**

| Attribute Name | Background | Description |
|---|---|---|
| au_final_ppm | 0.001 | Gold final estimate for reporting - Au (ppm) |
| au_id_ppm | 0.001 | Gold IDW2 estimate - Au (ppm) |
| au_ok_ppm | -1 | Gold OK estimate - Au (ppm) |
| au_nn_ppm | -1 | Gold Nearest Neighbour estimate - Au (ppm) |
| cu_final_ppm | -1 | Copper final estimate - Cu (ppm) |
| s_final_pct | -1 | Sulphur final estimate - S (wt%) |
| estpass_au | 0.001 | Estimation pass for gold |
| estpass_cu | 0.001 | Estimation pass for copper |
| estpass_s | 0.001 | Estimation pass for sulphur |
| ox_state | 3 | 1=Fresh; 2=Transitional; 3=Oxide; 4=Laterite; 0=Air |
| depletion | 1 | 0=Air; 1=Mined; 2=In situ; 3=Dump |
| facies | -1 | Rock Facies Code |
| density | -1 | Dry Bulk Density |
| depletion | 1 | Depletion state: 0=Air; 0=Mined; 1=In situ |
| facies | -1 | Rock Facies Code |
| res_class | -1 | Classification: 0=Air; 1=Measured; 2=Indicated; 3=Inferred |
| minzone_au | -1 | Gold Estimation/Mineralisation Domain Code |

## 14.9 Block Model Grade Estimation

### 14.9.1 In Situ Deposits

The primary gold estimation method implemented by Cube for the in situ deposits including Daapleu, Gbeitouo, Mont Ity / Ity Flat, Walter and ZiaNE is Localised Uniform Conditioning (LUC) and was completed using Isatis software.

The demonstrated complexity of geology / mineralisation at Ity, high gold variability and positively skewed gold distribution strongly argues for estimation into smaller SMU-sized blocks to satisfactorily represent such a gold distribution and its small scale rapid changes and also to model for mining selectivity. LUC is an industry standard non-linear method used for the express purpose of estimating the distribution of gold in small (SMU) blocks prior to the acquisition of very tight grade control data. Therefore LUC is considered the most appropriate method for the estimation of local recoverable resource for the types of deposits found at Ity as it produces accurate grade-tonnage functions which are in good accordance with volume-variance relationship principles. It limits the smearing of high grade and the over-smoothing of grade compared to other estimation methods.

The final mineral resource is a diluted recoverable resource estimate given the process attempts to estimate the recoverable tonnage and grade based on the dimensions of the selective mining unit (SMU) which is regarded as representative of what is practically achievable during actual mining. Estimation is undertaken using relatively large panels and therefore can be considered as being 'diluted' as they are estimated using all data within a broad mineralised envelope. A Change of Support is applied to the large diluted panels in order to predict the likely grade tonnage distribution of SMU selectivity. In addition to the imposition of a minimum selective mining dimension, a further Change of Support correction was applied to the models in the form of an Information Effect. The Information Effect is a theoretical ‘penalty’ adjustment to the SMU grade tonnage distribution to account for the anticipated dilution incurred when making mining selectivity decisions based on likely grade control spaced data. A typical grade control pattern does not represent exhaustive information and some misallocation of material is inevitable when block estimates are based on a non-exhaustive dataset. In a sense, the Information Effect adjustment is a representation of the impact of inaccuracies inherently anticipated in the practical mining process. Unplanned mining dilution due to mining inaccuracies would still need to be accounted for as a modifying factor.

The LUC method consists of the following steps:

- Undertake an Ordinary Kriging (OK) estimate of gold grade into panels.
- Undertake a CoS process, using the Discrete Gaussian Model (DGM). The DGM makes use of a Gaussian model fitted to the sample data, in conjunction with the gold variogram model, to predict the grade distribution for smaller, SMU-sized blocks defined as 5 mX x 5 mY x 2.5 mZ.
- Implement Uniform Conditioning (UC), which uses the results of CoS and the panel OK to produce an array variable of volume proportion and grade above a chosen set of grade thresholds. This represents the grade distribution of SMUs inside each panel. The UC results are conditioned to the original panel OK grade estimate.
- The UC is post-processed using the Abzalov (2006) approach to produce a single grade per SMU (5 mX x 5 mY x 2.5 mZ) block. The mean grade of the SMUs in any given panel is equal to the OK grade for that panel, and so the total metal is preserved.

A number of criteria including data spacing, geometry of mineralised domains and volume fill were the primary considerations taken into account when selecting an appropriate estimation block size. Cube reviewed the 'physical' data spacing relative to the mineralised zones to be estimated and decided that the estimation panel size summarised in Table 14.8.2 as appropriate.

Note that for domain 900 within the Walter deposit, Cube identified the presence of two populations and used a probabilistic approach to define internal high and low grade sub-domains. The probabilistic approach involved:

- Defining an indicator variable based on >0.4ppm Au threshold (i.e. 1 if >0.4 ppm Au or 0 if <0.4 ppm Au).
- Calculate and model the experimental indicator variogram.

- Estimate the indicator variable using Ordinary Kriging.
- A cell was coded as:
  - 900-LG if estimated indicator value was <0.5
  - 900-HG if the indicator value was >0.5.
- A post processing erosion and dilation morphological step was implemented. A cell was coded as 900-LG if a LG cell is located on the major cell boundaries (i.e. morphological cross). The subsequent dilation step coded a cell as 900-HG (i.e. ≥ 0.5) if a single mineralised cell is located on the major cell boundary. The erosion and dilation step removes the presence of single cells coded as HG within a large volume of LG or vice versa; and
- The flagged block model was used to code the composited drillhole file.

The primary gold estimation method implemented by Cube for the Teckraie and Verse Ouest deposits was Ordinary Kriging (OK) and this was completed using Surpac software. OK is considered an appropriate method for estimation given the low coefficients of variation for the cut composite data. No mining selectively would be anticipated when mining historic rock dumps such as Teckraie and Verse Ouest and therefore OK will provide a robust global estimate. As such, the global mean of the OK estimate was assigned to the block model attribute 'au_final_ppm'.

Two alternate estimation methods for gold were also used and these included Inverse Distance Squared (IDW2) and Nearest Neighbour (NN).

The minor background elements including arsenic, sulphur and copper were estimated by Ordinary Kriging or Inverse Distance. As mentioned in Section 14.5, it is important to note that the estimate for the minor elements is based on a much smaller and biased data set compared to gold and it is not possible to draw any definitive conclusions from the estimate and should be used as a guide only.

***Search Neighbourhood Analysis***

The spatial relationship of the data was characterised using variography on the selected representative domains. The search strategies were aimed at producing a robust block estimate whilst at the same time minimising estimation error and conditional biases. Fundamental to the search strategy is the determination of appropriate minimum and maximum numbers of composites for estimation.

Kriging Neighbourhood Analysis (KNA) was undertaken to assist with the decision on the minimum and maximum number of samples to use for the LUC estimation. The kriging Slope of Regression (SoR) and Weight of the Mean were used in conjunction with the number of negative weights to reach a decision. The gold grade variograms listed in Table 14.7.1 to Table 14.7.7 were used for KNA. The search neighbourhood shape was chosen interactively, in 3D, with due consideration being given to the pattern of sample selection observed. This was done with the continuity of orientation of mineralisation in mind and with a view to not selecting too many samples from any single drillhole.

Dynamic local rotation of the variogram model and search neighbourhood was implemented using the set of surfaces constructed in Surpac and exported to Isatis. Following some intermediate processing steps, these surfaces were converted to local rotation parameters for use in the Isatis kriging algorithm. Typically a single or in some instances multiple surfaces were created for each estimation domain. An example of the trend surfaces used in the Daapleu estimation model is displayed in Figure 14.9.1.

The CoS and UC routines were run based on the results of the OK panel estimate. An Information Effect Correction, based on a grade control grid of 10 mX x 10 mY x 2.5 mRL was implemented.

The search neighbourhood parameters used for LUC panel estimation of gold are displayed in Table 14.9.1 to Table 14.9.6.

**Figure 14.9.1 Daapleu Mineralisation Interpretation with Trend Surfaces – Section 1250 Looking Northeast**

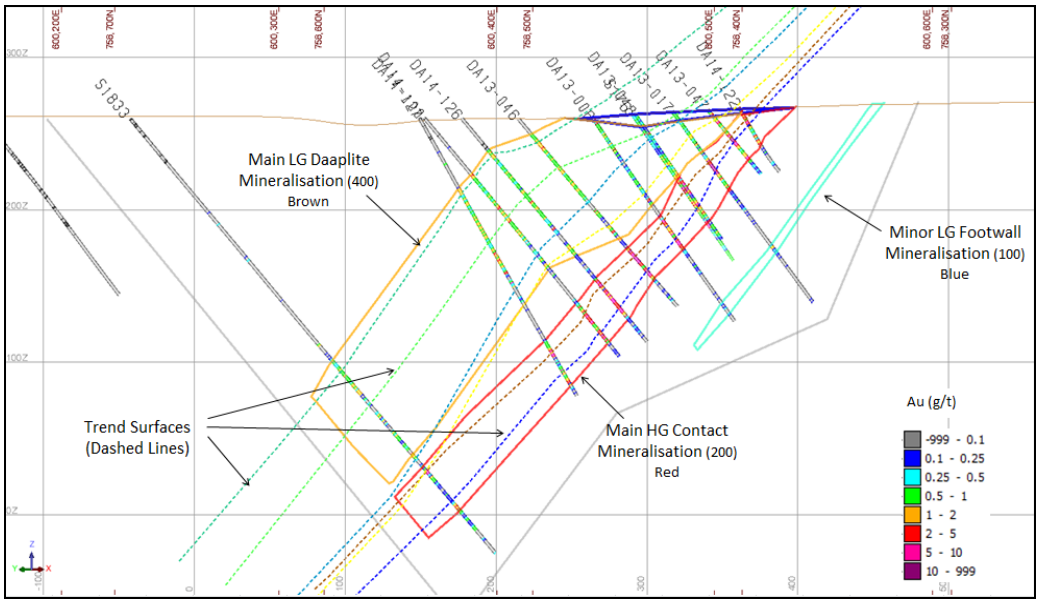


**Table 14.9.1 Daapleu Search Neighbourhood Parameters – LUC Panel Gold Grade Estimates**

| Domain | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | u (major) | v (semi) | w (minor) | E | N | RL | A | +X | -Z |
| 100 | LUC | 8 | 26 | 260 | 260 | 65 | 6 | 6 | 3 | Dynamic Local Rotations | | |
| 150 | LUC | 8 | 26 | 120 | 120 | 30 | 6 | 6 | 3 | | | |
| 200 | LUC | 8 | 24 | 100 | 100 | 25 | 6 | 6 | 3 | | | |
| 301 | LUC | 8 | 18 | 570 | 570 | 190 | 6 | 6 | 3 | | | |
| 400 | LUC | 8 | 16 | 220 | 220 | 55 | 6 | 6 | 3 | | | |
| 500 | LUC | 8 | 26 | 140 | 140 | 35 | 6 | 6 | 3 | | | |
| 999 | LUC | 8 | 20 | 400 | 400 | 100 | 6 | 6 | 3 | | | |

**Table 14.9.2 Gbeitouo Search Neighbourhood Parameters – LUC Panel Gold Grade Estimates**

| Domain | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | u (major) | v (semi) | w (minor) | E | N | RL | A | +X | -Z |
| 100 | LUC | 10 | 28 | 120 | 120 | 30 | 10 | 10 | 3 | Dynamic Local Rotations | | |
| 101 | LUC | 10 | 26 | 120 | 120 | 30 | 10 | 10 | 3 | | | |
| 999 | LUC | 10 | 26 | 300 | 300 | 75 | 10 | 10 | 3 | | | |

**Table 14.9.3 Mont Ity / Ity Flat Search Neighbourhood Parameters – LUC Panel Gold Grade Estimates**

| Domain | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | u (major) | v (semi) | w (minor) | E | N | RL | A | +X | -Z |
| 100 | LUC | 8 | 26 | 125 | 125 | 25 | 6 | 6 | 3 | Dynamic Local Rotations | | |
| 200 | LUC | 8 | 26 | 125 | 125 | 25 | 6 | 6 | 3 | | | |
| 301 | LUC | 6 | 20 | 300 | 300 | 30 | 6 | 6 | 3 | | | |
| 400 | LUC | 8 | 26 | 150 | 150 | 35 | 6 | 6 | 3 | | | |
| 500 | LUC | 8 | 26 | 125 | 125 | 25 | 6 | 6 | 3 | | | |
| 600 | LUC | 8 | 28 | 150 | 150 | 50 | 6 | 6 | 3 | | | |
| 888 | LUC | 8 | 30 | 600 | 600 | 150 | 6 | 6 | 3 | | | |
| 999 | LUC | 8 | 30 | 600 | 600 | 150 | 6 | 6 | 3 | | | |

**Table 14.9.4 Walter First Pass Search Neighbourhood Parameters – LUC Panel Gold Grade Estimates**

| Domain | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | u (major) | v (semi) | w (minor) | E | N | RL | A | +X | -Z |
| 101 | LUC | 8 | 15 | 100 | 100 | 10 | 5 | 5 | 3 | 210 | 51 | -90 |
| 102 | LUC | 8 | 15 | 100 | 100 | 10 | 5 | 5 | 2 | 210 | 51 | -90 |
| 104 | LUC | 8 | 15 | 100 | 100 | 10 | 5 | 5 | 2 | 210 | 51 | -90 |
| 200 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 201 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 202 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 203 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 205 | LUC | 12 | 32 | 75 | 45 | 15 | 5 | 5 | 2 | 225 | 25 | -180 |
| 206 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 208 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 210 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 211 | LUC | 12 | 32 | 50 | 30 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 300 | LUC | 10 | 28 | 75 | 20 | 5 | 5 | 5 | 2 | 45 | 0 | 0 |
| 900-lg | LUC | 8 | 24 | 35 | 15 | 5 | 5 | 5 | 2 | 225 | 30 | -180 |
| 900-hg | LUC | 8 | 24 | 35 | 15 | 5 | 5 | 5 | 2 | 225 | 30 | -180 |

**Table 14.9.5 Walter Second Pass Search Neighbourhood Parameters – LUC Panel Gold Grade Estimates**

| Domain | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | u (major) | v (semi) | w (minor) | E | N | RL | A | +X | -Z |
| 101 | LUC | 4 | 20 | 200 | 200 | 20 | 5 | 5 | 3 | 210 | 51 | -90 |
| 102 | LUC | 4 | 20 | 200 | 200 | 20 | 5 | 5 | 2 | 210 | 51 | -90 |
| 104 | LUC | 4 | 20 | 200 | 200 | 20 | 5 | 5 | 2 | 210 | 51 | -90 |
| 200 | LUC | 4 | 32 | 200 | 120 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 201 | LUC | 4 | 32 | 200 | 120 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 202 | LUC | 4 | 32 | 200 | 120 | 20 | 5 | 5 | 2 | 225 | 25 | -180 |
| 203 | LUC | 4 | 32 | 200 | 120 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 205 | LUC | 4 | 32 | 200 | 120 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 206 | LUC | 4 | 32 | 200 | 120 | 20 | 5 | 5 | 2 | 225 | 25 | -180 |
| 208 | LUC | 4 | 32 | 200 | 120 | 20 | 5 | 5 | 2 | 225 | 25 | -180 |
| 210 | LUC | 4 | 32 | 200 | 120 | 10 | 5 | 5 | 2 | 225 | 25 | -180 |
| 211 | LUC | 4 | 32 | 200 | 120 | 20 | 5 | 5 | 2 | 225 | 25 | -180 |
| 300 | LUC | 10 | 32 | 175 | 80 | 10 | 5 | 5 | 2 | 45 | 0 | 0 |
| 900-lg | LUC | 2 | 24 | 100 | 150 | 20 | 5 | 5 | 2 | 225 | 30 | -180 |
| 900-hg | LUC | 2 | 24 | 100 | 150 | 20 | 5 | 5 | 2 | 225 | 30 | -180 |

**Table 14.9.6 ZiaNE Search Neighbourhood Parameters – LUC Panel Gold Grade Estimates**

| Domain | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Isatis Rotation (Geol. Plane) | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | u (major) | v (semi) | w (minor) | E | N | RL | A | +X | -Z |
| 100 | LUC | 12 | 28 | 120 | 120 | 30 | 6 | 6 | 3 | Dynamic Local Rotations | | |
| 200 | LUC | 12 | 28 | 150 | 150 | 45 | 6 | 6 | 3 | | | |
| 301 | LUC | 8 | 22 | 140 | 140 | 40 | 6 | 6 | 3 | | | |
| 400 | LUC | 12 | 28 | 120 | 120 | 30 | 6 | 6 | 3 | | | |
| 500 | LUC | 12 | 28 | 120 | 120 | 30 | 6 | 6 | 3 | | | |
| 999 | LUC | 12 | 22 | 250 | 250 | 75 | 6 | 6 | 3 | | | |

***Discretisation***

Discretisation is the process applied to a parent block within the model during estimation to reduce the impact of the volume-variance affect by splitting the block into smaller portions. A discretisation size as summarised in Table 14.8.2 was used in the estimation which effectively splits the parent block into a number of nodes or estimation points within the original parent block.

***Un-estimated Domains and Blocks***

For the gold estimates for each in situ deposit all blocks were estimated in a single search and there was no need to implement a multiple search pass strategy or assignment of grade to address the issue of un-estimated blocks.

In some instance the minor elements were not estimated within a single search pass strategy given there is a lot less assay data available compared to that used for the gold estimate. Un-estimated blocks were assigned the mean grade for each domain.

### 14.9.2 Rock Dump and Heap Leach Deposits

The primary gold estimation method implemented by Cube for the rock dumps Teckraie and Verse Ouest and heap leach pad Aires was Ordinary Kriging (OK) and this was completed using Surpac software. OK is considered an appropriate method for estimation given the low coefficients of variation for the cut composite data and will provide a robust global estimate. As such, the global mean of the OK estimate was assigned to the block model attribute 'au_final_ppm'.

Two alternate estimation methods for gold were also used and these included Inverse Distance Squared (IDW2) and Nearest Neighbour (NN).

***Search Neighbourhood Analysis***

The spatial relationship of the data was characterised using variography on the selected representative domains. The search strategies were aimed at producing a robust block estimate whilst at the same time minimising estimation error and conditional biases. Fundamental to the search strategy is the determination of appropriate minimum and maximum numbers of composites for estimation.

Kriging Neighbourhood Analysis (KNA) was undertaken to assist with the decision on the minimum and maximum number of samples to use for the estimation. The search neighbourhood parameters used for estimation are displayed in Table 14.9.7 to Table 14.9.9.

**Table 14.9.7 Teckraie and Verse Ouest Search Neighbourhood Parameters – Grade Estimates for Rock Dumps**

| Domain Variable | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Search Orientation | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | major | semi | minor | E | N | RL | Bearing | Plunge | Dip |
| 340 (Au) | OK | 8 | 20 | 78 | 78 | 18 | 4 | 4 | 2 | 0 | 0 | 0 |
| 340 (Au) | IDW2 | 8 | 20 | 78 | 78 | 18 | 4 | 4 | 2 | 0 | 0 | 0 |
| 340 (Au) | NN | 8 | 20 | 78 | 78 | 18 | 4 | 4 | 2 | 0 | 0 | 0 |
| 340 (Cu) | OK | 8 | 20 | 165 | 165 | 18 | 4 | 4 | 2 | 0 | 0 | 0 |
| 340 (S) | OK | 8 | 20 | 75 | 75 | 75 | 4 | 4 | 2 | 0 | 0 | 0 |
| 341 (Au) | OK | 8 | 20 | 245 | 245 | 20 | 4 | 4 | 2 | 0 | 0 | 0 |
| 341 (Au) | IDW2 | 8 | 20 | 245 | 245 | 20 | 4 | 4 | 2 | 0 | 0 | 0 |
| 341 (Au) | NN | 8 | 20 | 245 | 245 | 20 | 4 | 4 | 2 | 0 | 0 | 0 |
| 341 (Cu) | OK | 8 | 20 | 113 | 113 | 14 | 4 | 4 | 2 | 0 | 0 | 0 |
| 341 (S) | OK | 8 | 20 | 52 | 52 | 52 | 4 | 4 | 2 | 0 | 0 | 0 |

**Table 14.9.8 Teckraie and Verse Ouest Search Neighbourhood Parameters – Grade Estimates for Laterite**

| Domain Variable | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Search Orientation | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | major | semi | minor | E | N | RL | Bearing | Plunge | Dip |
| 331 (Au) | OK | 10 | 26 | 68 | 68 | 68 | 4 | 4 | 2 | 0 | 0 | 0 |
| 331 (Au) | IDW2 | 10 | 26 | 68 | 68 | 68 | 4 | 4 | 2 | 0 | 0 | 0 |
| 331 (Au) | NN | 1 | 1 | 68 | 68 | 68 | 1 | 1 | 1 | 0 | 0 | 0 |
| 332 (Au) | OK | 10 | 26 | 75 | 75 | 75 | 4 | 4 | 2 | 0 | 0 | 0 |
| 332 (Au) | IDW2 | 10 | 26 | 75 | 75 | 75 | 4 | 4 | 2 | 0 | 0 | 0 |
| 332 (Au) | NN | 1 | 1 | 75 | 75 | 75 | 1 | 1 | 1 | 0 | 0 | 0 |

**Table 14.9.9 Aires Search Neighbourhood Parameters – Grade Estimates**

| Domain Variable | Method | Min Samp | Max Samp | Search Radii (m) | | | Block Discretisation | | | Search Orientation | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | major | semi | minor | E | N | RL | Bearing | Plunge | Dip |
| 301(Au) | OK | 10 | 26 | 100 | 100 | 15.75 | 4 | 4 | 2 | 0 | 0 | 0 |
| 303(Au) | OK | 8 | 20 | 150 | 150 | 12.5 | 4 | 4 | 2 | 0 | 0 | 0 |
| 303(Cu) | OK | 8 | 20 | 150 | 150 | 12.5 | 4 | 4 | 2 | | | |
| 303 (S) | OK | 8 | 20 | 40 | 40 | 40 | 4 | 4 | 2 | | | |

***Discretisation***

Discretisation is the process applied to a parent block within the model during estimation to reduce the impact of the volume-variance affect by splitting the block into smaller portions. A discretisation size as summarised in Table 14.8.2 was used in the estimation which effectively splits the parent block into a number of nodes or estimation points within the original parent block.

***Un-estimated Domains and Blocks***

For the Teckraie and Verse Ouest gold estimates, blocks not estimated in the first pass were assigned the mean grade of the estimated blocks for each mineralisation domain. A summary of the proportion of un-estimated blocks and the assigned grade is summarised in Table 14.9.10.

**Table 14.9.10 Summary of Teckraie and Verse Ouest Un-estimated Blocks and Grade Assignment**

| Domain | 331 | 332 | 340 | 341 |
|---|---|---|---|---|
| Un-estimated Proportion | 3% | 18% | 23% | 1% |
| Assigned Grade | 0.55 | 0.43 | 1.07 | 1.03 |

For the unknown material that lies on top of the Teckraie rock dump (defined as mineralisation Domain 342) this has been assigned a gold grade of 0.001.

For the Aires gold estimate all blocks were estimated in a single search and there was no need to implement a multiple search pass strategy or assignment of grade to address the issue of un-estimated blocks.

## 14.10 Model Validation

Block model validation was undertaken by the following methods:

- Visual inspection of the final block estimates in relation to drill data on a section-by-section basis.
- Global statistical comparisons of sample and block grades.

- Global comparison of different estimation methods.
- Semi-local comparison of composite and block grades using Swath Plots.
- Comparison of LUC and LIK grade-tonnage estimates at a range of grade cut-offs.

***Visual Validation***

Visual validation of the final LUC grade estimate shows a good correspondence between the model and informing data as shown in Figure 14.10.1 to Figure 14.10.5.

**Figure 14.10.1 Daapleu Block Model Gold Grade Estimate with Drilling – Section 1250**

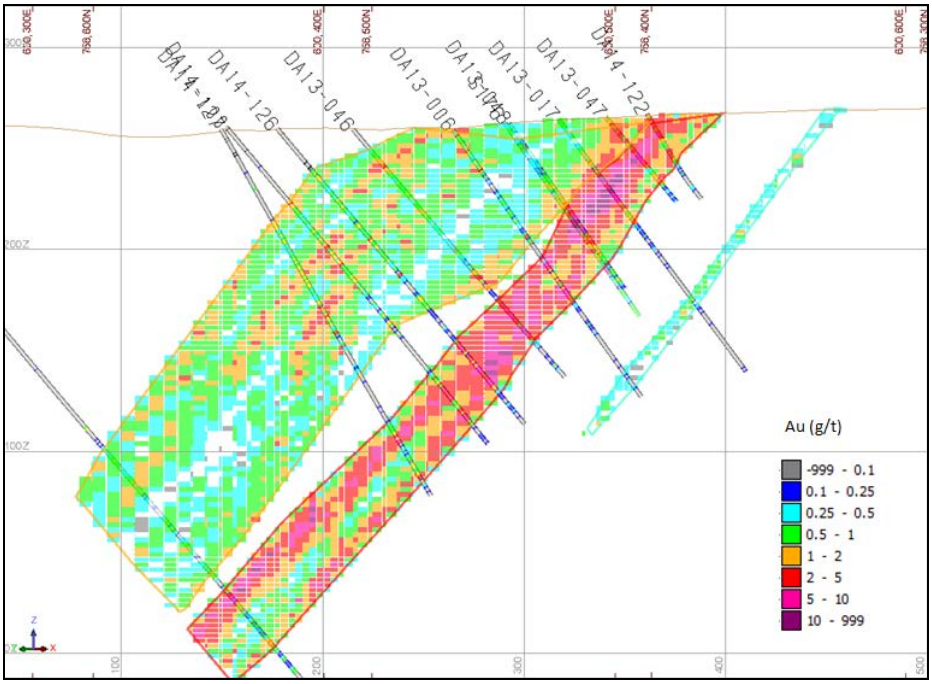


**Figure 14.10.2 Gbeitouo Block Model Gold Grade Estimate with Drilling – Section 160**

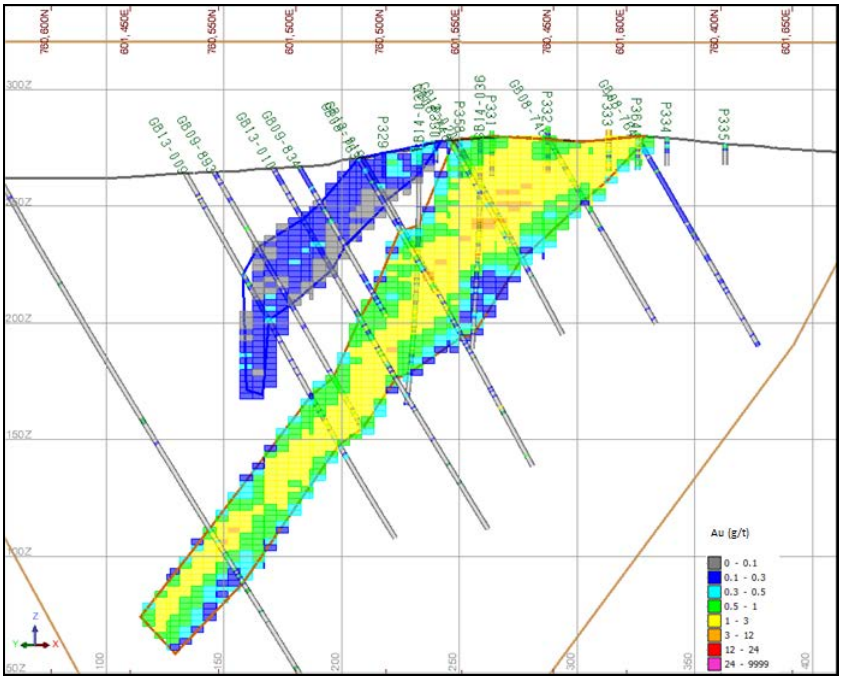



**Figure 14.10.3 Mont Ity / Ity Flat Block Model Gold Grade Estimate with Drilling – Section 600**

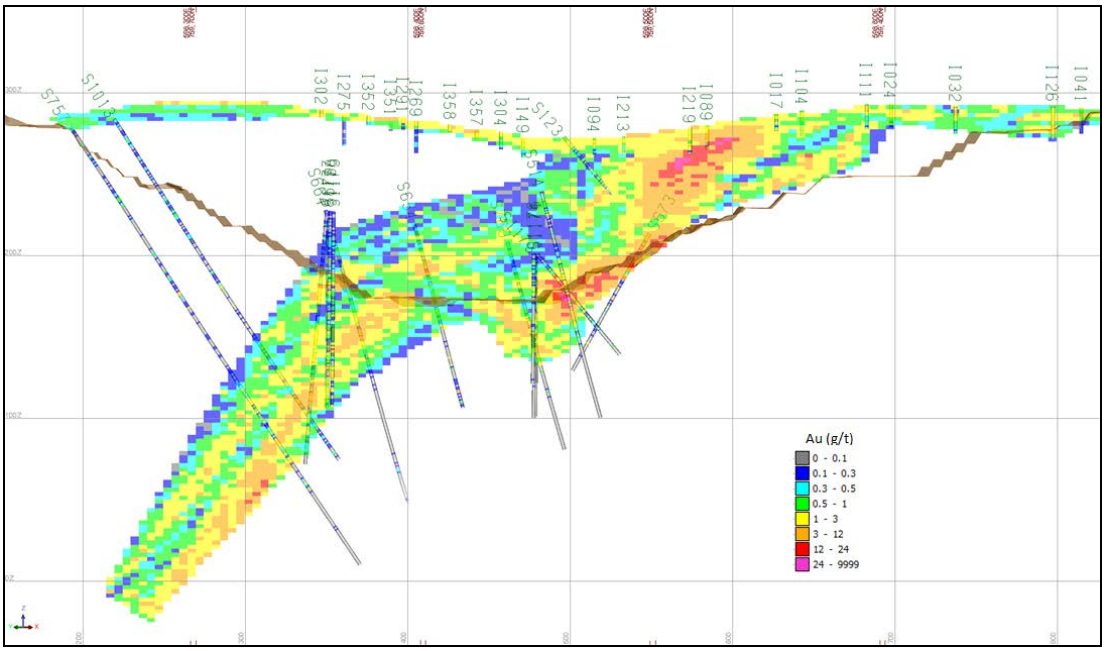


**Figure 14.10.4 Walter Block Model Gold Grade Estimate with Drilling – Section -100**

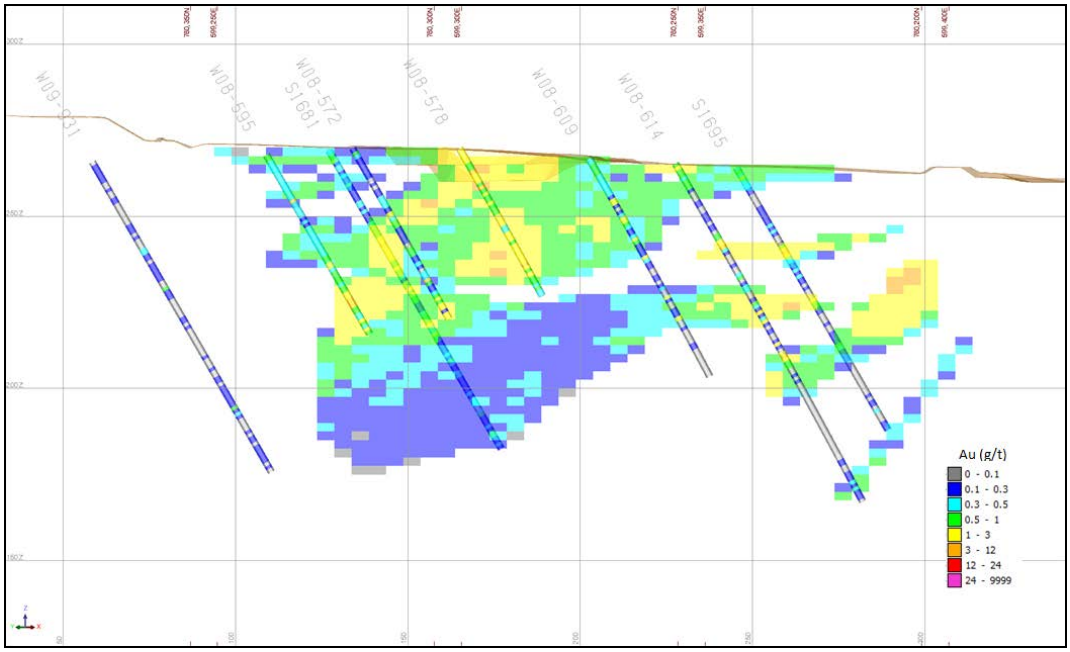
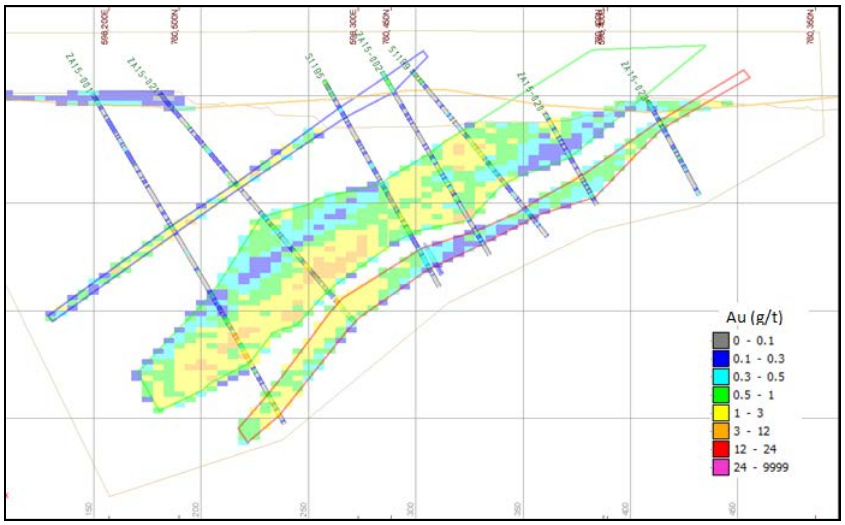



**Figure 14.10.5 ZiaNE Block Model Gold Grade Estimate with Drilling – Section -175**



***Model Validation - Global Comparisons***

The match between sample and estimated grades is relatively good, considering the difficulties associated with trying to obtain an unbiased global mean for the samples, which require declustering. Table 14.10.1 to Table 14.10.7 summarise the statistical comparison between the block estimate and informing sample data for all domains which indicated there are no out-of-range values in the estimates.

**Table 14.10.1 Daapleu Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | N | Min | Max | Mean | Variance |
|---|---|---|---|---|---|---|
| 100 | Cut Samples (undecl.) | 102 | 0.005 | 6.000 | 0.792 | 1.030 |
| | Cut Samples (cell decl.) | 102 | 0.005 | 6.000 | 0.787 | 0.807 |
| | Cut Samples (krig decl.) | 102 | 0.005 | 6.000 | 0.811 | 0.894 |
| | LUC Au | 20,864 | 0.047 | 3.990 | 0.764 | 0.303 |
| | LIK Au | 20,864 | 0.027 | 5.517 | 0.809 | 0.367 |
| 150 | Cut Samples (undecl.) | 106 | 0.087 | 2.500 | 0.721 | 0.299 |
| | Cut Samples (cell decl.) | 106 | 0.087 | 2.500 | 0.644 | 0.245 |
| | Cut Samples (krig decl.) | 106 | 0.087 | 2.500 | 0.705 | 0.311 |
| | LUC Au | 8,225 | 0.144 | 2.126 | 0.676 | 0.108 |
| | LIK Au | 8,225 | 0.207 | 2.459 | 0.728 | 0.133 |
| 200 | Cut Samples (undecl.) | 1,428 | 0.012 | 25.000 | 2.286 | 11.745 |
| | Cut Samples (cell decl.) | 1,428 | 0.012 | 25.000 | 1.799 | 8.618 |
| | Cut Samples (krig decl.) | 1,428 | 0.012 | 25.000 | 1.825 | 8.169 |
| | LUC Au | 48,414 | 0.064 | 21.418 | 2.165 | 5.961 |
| | LIK Au | 48,414 | 0.043 | 28.212 | 2.211 | 4.892 |
| 301 | Cut Samples (undecl.) | 405 | 0.005 | 5.000 | 0.374 | 0.309 |
| | Cut Samples (cell decl.) | 405 | 0.005 | 5.000 | 0.278 | 0.205 |
| | Cut Samples (krig decl.) | 405 | 0.005 | 5.000 | 0.154 | 0.079 |
| | LUC Au | 91,712 | 0.018 | 4.258 | 0.138 | 0.044 |
| | LIK Au | 91,712 | 0.026 | 8.162 | 0.150 | 0.079 |
| 400 | Cut Samples (undecl.) | 4,966 | 0.005 | 15.000 | 0.956 | 1.470 |
| | Cut Samples (cell decl.) | 4,966 | 0.005 | 15.000 | 0.999 | 2.029 |
| | Cut Samples (krig decl.) | 4,966 | 0.005 | 15.000 | 0.931 | 1.496 |
| | LUC Au | 184,176 | 0.025 | 13.431 | 0.896 | 0.698 |
| | LIK Au | 184,176 | 0.025 | 32.126 | 0.909 | 0.996 |
| 500 | Cut Samples (undecl.) | 50 | 0.052 | 2.000 | 0.482 | 0.246 |
| | Cut Samples (cell decl.) | 50 | 0.052 | 2.000 | 0.494 | 0.253 |
| | Cut Samples (krig decl.) | 50 | 0.052 | 2.000 | 0.485 | 0.232 |
| | LUC Au | 9,109 | 0.090 | 1.705 | 0.497 | 0.089 |
| | LIK Au | 9,109 | 0.240 | 3.035 | 0.653 | 0.133 |

**Table 14.10.2 Gbeitouo Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | N | Min | Max | Mean | Variance |
|---|---|---|---|---|---|---|
| 100 | Cut Samples (undecl.) | 847 | 0.005 | 15.000 | 1.336 | 3.445 |
| | Cut Samples (cell decl.) | 847 | 0.005 | 15.000 | 1.151 | 3.208 |
| | Cut Samples (krig decl.) | 847 | 0.005 | 15.000 | 1.152 | 2.711 |
| | LUC Au | 31,071 | 0.095 | 7.722 | 1.311 | 0.728 |
| 101 | Cut Samples (undecl.) | 733 | 0.005 | 8.000 | 0.558 | 0.488 |
| | Cut Samples (cell decl.) | 733 | 0.005 | 8.000 | 0.568 | 0.476 |
| | Cut Samples (krig decl.) | 733 | 0.005 | 8.000 | 0.545 | 0.463 |
| | LUC Au | 17,370 | 0.031 | 3.810 | 0.562 | 0.175 |

**Table 14.10.3 Mont Ity / Ity Flat Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | N | Min | Max | Mean | Variance |
|---|---|---|---|---|---|---|
| 100 | Cut Samples (undecl.) | 51 | 0.05 | 1.84 | 0.48 | 0.11 |
| | Cut Samples (cell decl.) | 51 | 0.05 | 1.84 | 0.46 | 0.11 |
| | Cut Samples (krig decl.) | 51 | 0.05 | 1.84 | 0.51 | 0.13 |
| | LUC Au | 3,001 | 0.15 | 1.36 | 0.52 | 0.04 |
| 200 | Cut Samples (undecl.) | 455 | 0.03 | 5.00 | 0.68 | 0.60 |
| | Cut Samples (cell decl.) | 455 | 0.03 | 5.00 | 0.59 | 0.49 |
| | Cut Samples (krig decl.) | 455 | 0.03 | 5.00 | 0.65 | 0.62 |
| | LUC Au | 10,249 | 0.07 | 3.60 | 0.68 | 0.20 |
| 301 | Cut Samples (undecl.) | 2,399 | 0.01 | 12.00 | 1.21 | 2.44 |
| | Cut Samples (cell decl.) | 2,399 | 0.01 | 12.00 | 0.92 | 1.53 |
| | Cut Samples (krig decl.) | 2,397 | 0.01 | 12.00 | 1.05 | 1.71 |
| | LUC Au | 130,701 | 0.04 | 10.69 | 1.14 | 1.16 |
| 400 | Cut Samples (undecl.) | 10,610 | 0.01 | 70.00 | 1.86 | 26.02 |
| | Cut Samples (cell decl.) | 10,610 | 0.01 | 70.00 | 1.58 | 22.08 |
| | Cut Samples (krig decl.) | 10,609 | 0.01 | 70.00 | 1.52 | 19.76 |
| | LUC Au | 350,787 | 0.04 | 51.91 | 1.68 | 7.82 |
| 500 | Cut Samples (undecl.) | 349 | 0.01 | 8.00 | 0.66 | 1.13 |
| | Cut Samples (cell decl.) | 349 | 0.01 | 8.00 | 0.66 | 1.12 |
| | Cut Samples (krig decl.) | 349 | 0.01 | 8.00 | 0.68 | 1.16 |
| | LUC Au | 19,100 | 0.06 | 4.93 | 0.67 | 0.33 |
| 600 | Cut Samples (undecl.) | 290 | 0.02 | 10.00 | 1.04 | 2.59 |
| | Cut Samples (cell decl.) | 290 | 0.02 | 10.00 | 1.15 | 2.48 |
| | Cut Samples (krig decl.) | 290 | 0.02 | 10.00 | 1.18 | 2.83 |
| | LUC Au | 20,261 | 0.05 | 6.70 | 1.17 | 0.74 |

**Table 14.10.4 Walter Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | N | Min | Max | Mean | SD |
|---|---|---|---|---|---|---|
| 101 | Cut Samples (undecl.) | 19 | 0.10 | 1.89 | 0.78 | 0.53 |
| | Cut Samples (Decl.) | 19 | 0.10 | 1.89 | 0.79 | 0.52 |
| | LUC | 483 | 0.45 | 1.17 | 0.82 | 0.16 |
| 102 | Cut Samples (undecl.) | 23 | 0.13 | 2.75 | 0.79 | 0.66 |
| | Cut Samples (Decl.) | 23 | 0.13 | 2.75 | 0.80 | 0.68 |
| | LUC | 566 | 0.33 | 1.45 | 0.79 | 0.29 |
| 104 | Cut Samples (undecl.) | 19 | 0.26 | 2.50 | 0.91 | 0.64 |
| | Cut Samples (Decl.) | 19 | 0.26 | 2.50 | 0.80 | 0.61 |
| | LUC | 414 | 0.58 | 1.42 | 0.88 | 0.17 |
| 200 | Cut Samples (undecl.) | 859 | 0.05 | 30.00 | 1.52 | 3.09 |
| | Cut Samples (Decl.) | 859 | 0.05 | 30.00 | 1.40 | 2.84 |
| | LUC | 13,396 | 0.23 | 9.50 | 1.22 | 0.96 |
| 201 | Cut Samples (undecl.) | 58 | 0.07 | 2.79 | 0.68 | 0.53 |
| | Cut Samples (Decl.) | 58 | 0.07 | 2.79 | 0.68 | 0.54 |
| | LUC | 1,617 | 0.34 | 1.23 | 0.68 | 0.15 |
| 202 | Cut Samples (undecl.) | 32 | 0.09 | 2.00 | 0.67 | 0.48 |
| | Cut Samples (Decl.) | 32 | 0.09 | 2.00 | 0.73 | 0.54 |
| | LUC | 627 | 0.38 | 1.28 | 0.66 | 0.17 |
| 203 | Cut Samples (undecl.) | 78 | 0.07 | 9.00 | 1.66 | 1.99 |
| | Cut Samples (Decl.) | 78 | 0.07 | 9.00 | 1.66 | 1.96 |
| | LUC | 1,508 | 0.51 | 4.38 | 1.69 | 0.66 |
| 205 | Cut Samples (undecl.) | 91 | 0.09 | 5.96 | 1.13 | 1.19 |
| | Cut Samples (Decl.) | 91 | 0.09 | 5.96 | 1.15 | 1.12 |
| | LUC | 2,504 | 0.29 | 3.38 | 1.05 | 0.50 |
| 206 | Cut Samples (undecl.) | 119 | 0.05 | 12.00 | 1.66 | 2.42 |
| | Cut Samples (Decl.) | 119 | 0.05 | 12.00 | 1.56 | 2.30 |
| | LUC | 2,962 | 0.31 | 6.23 | 1.66 | 0.89 |
| 208 | Cut Samples (undecl.) | 14 | 0.36 | 1.57 | 0.64 | 0.35 |
| | Cut Samples (Decl.) | 14 | 0.36 | 1.57 | 0.63 | 0.33 |
| | LUC | 186 | 0.43 | 0.85 | 0.60 | 0.09 |
| 210 | Cut Samples (undecl.) | 13 | 0.32 | 1.30 | 0.69 | 0.32 |
| | Cut Samples (Decl.) | 13 | 0.32 | 1.30 | 0.65 | 0.27 |
| | LUC | 269 | 0.47 | 0.92 | 0.67 | 0.11 |
| 211 | Cut Samples (undecl.) | 14 | 0.30 | 7.50 | 3.12 | 2.68 |
| | Cut Samples (Decl.) | 14 | 0.30 | 7.50 | 3.16 | 2.76 |
| | LUC | 450 | 1.42 | 5.36 | 3.20 | 0.75 |
| 300 | Cut Samples (undecl.) | 198 | 0.21 | 4.00 | 0.89 | 0.58 |
| | Cut Samples (Decl.) | 198 | 0.21 | 4.00 | 0.78 | 0.51 |
| | LUC | 2,777 | 0.41 | 1.57 | 0.81 | 0.24 |
| 900 hg | Cut Samples (undecl.) | 263 | 0.02 | 10.00 | 1.08 | 1.55 |
| | Cut Samples (Decl.) | 263 | 0.02 | 10.00 | 1.15 | 1.65 |
| | LUC | 9,057 | 0.26 | 3.97 | 1.08 | 0.50 |
| 900 lg | Cut Samples (undecl.) | 481 | 0.01 | 4.80 | 0.34 | 0.53 |
| | Cut Samples (Decl.) | 481 | 0.01 | 4.80 | 0.40 | 0.62 |
| | LUC | 18,231 | 0.07 | 1.53 | 0.30 | 0.13 |

**Table 14.10.5 ZiaNE Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | N | Min | Max | Mean | Variance |
|---|---|---|---|---|---|---|
| 100 | Cut Samples (undecl.) | 349 | 0.008 | 10.000 | 0.735 | 2.126 |
| | Cut Samples (cell decl.) | 349 | 0.008 | 10.000 | 0.870 | 2.978 |
| | Cut Samples (krig decl.) | 349 | 0.008 | 10.000 | 0.785 | 2.452 |
| | LUC Au | 10,192 | 0.083 | 4.922 | 0.693 | 0.300 |
| 200 | Cut Samples (undecl.) | 4,324 | 0.005 | 30.000 | 0.992 | 60.099 |
| | Cut Samples (cell decl.) | 4,324 | 0.005 | 30.000 | 0.977 | 5.861 |
| | Cut Samples (krig decl.) | 4,324 | 0.005 | 30.000 | 1.086 | 7.903 |
| | LUC Au | 119,786 | 0.047 | 18.647 | 1.031 | 1.355 |
| 301 | Cut Samples (undecl.) | 1,219 | 0.005 | 15.000 | 1.046 | 1.986 |
| | Cut Samples (cell decl.) | 1,219 | 0.005 | 15.000 | 1.050 | 1.982 |
| | Cut Samples (krig decl.) | 1,165 | 0.005 | 15.000 | 0.801 | 1.329 |
| | LUC Au | 32,286 | 0.018 | 12.425 | 0.853 | 0.708 |
| 400 | Cut Samples (undecl.) | 413 | 0.005 | 8.000 | 0.470 | 0.619 |
| | Cut Samples (cell decl.) | 413 | 0.005 | 8.000 | 0.489 | 0.593 |
| | Cut Samples (krig decl.) | 413 | 0.005 | 8.000 | 0.568 | 0.925 |
| | LUC Au | 11,221 | 0.086 | 4.254 | 0.490 | 0.137 |
| 500 | Cut Samples (undecl.) | 531 | 0.005 | 20.000 | 0.963 | 4.656 |
| | Cut Samples (cell decl.) | 531 | 0.005 | 20.000 | 0.990 | 4.847 |
| | Cut Samples (krig decl.) | 531 | 0.005 | 20.000 | 0.972 | 4.584 |
| | LUC Au | 29,879 | 0.045 | 10.793 | 0.943 | 0.982 |

**Table 14.10.6 Teckraie and Verse Ouest Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | Mean |
|---|---|---|
| 331 | Cut Samples (Decl.) | 0.64 |
| | Au (OK) | 0.55 |
| | Au (IDW2) | 0.56 |
| | AU (NN) | 0.57 |
| 332 | Cut Samples (Decl.) | 0.43 |
| | Au (OK) | 0.43 |
| | Au (IDW2) | 0.41 |
| | AU (NN) | 0.44 |
| 340 | Cut Samples (Decl.) | 1.12 |
| | Au (OK) | 1.07 |
| | Au (IDW2) | 1.07 |
| | AU (NN) | 1.08 |
| 341 | Cut Samples (Decl.) | 1.16 |
| | Au (OK) | 1.03 |
| | Au (IDW2) | 1.03 |
| | AU (NN) | 0.99 |

**Table 14.10.7 Aires Global Statistical Comparison between Samples and Block Estimates**

| Domain | Variable | Mean |
|---|---|---|
| 301 | Cut Samples (Decl.) | 0.29 |
| | Au (OK) | 0.29 |
| | Au (IDW2) | 0.28 |
| | AU (NN) | 0.29 |
| 303 | Cut Samples (Decl.) | 1.08 |
| | Au (OK) | 1.09 |
| | Au (IDW2) | 1.10 |
| | AU (NN) | 1.09 |

***Model Validation - Local Comparisons***

Swath plots (grade trend profiles) showing the estimated tonnes, estimated grade, number of composites and mean cut composite grade were created for the all domains. Selected plots representative of the largest domains by tonnage are presented below in Figure 14.10.6 to Figure 14.10.10. The limitations of this comparison should be kept in mind when drawing conclusions; however there is generally good correlation between the block estimate and declustered composite mean for all domains. The greatest differences occur in poorly sampled areas and where the composites display high degrees of local variation, however Cube considers the estimates to be a fair and accurate reflection of the informing data.

**Figure 14.10.6 Daapleu Block Model Validation by Easting – Domain 200**

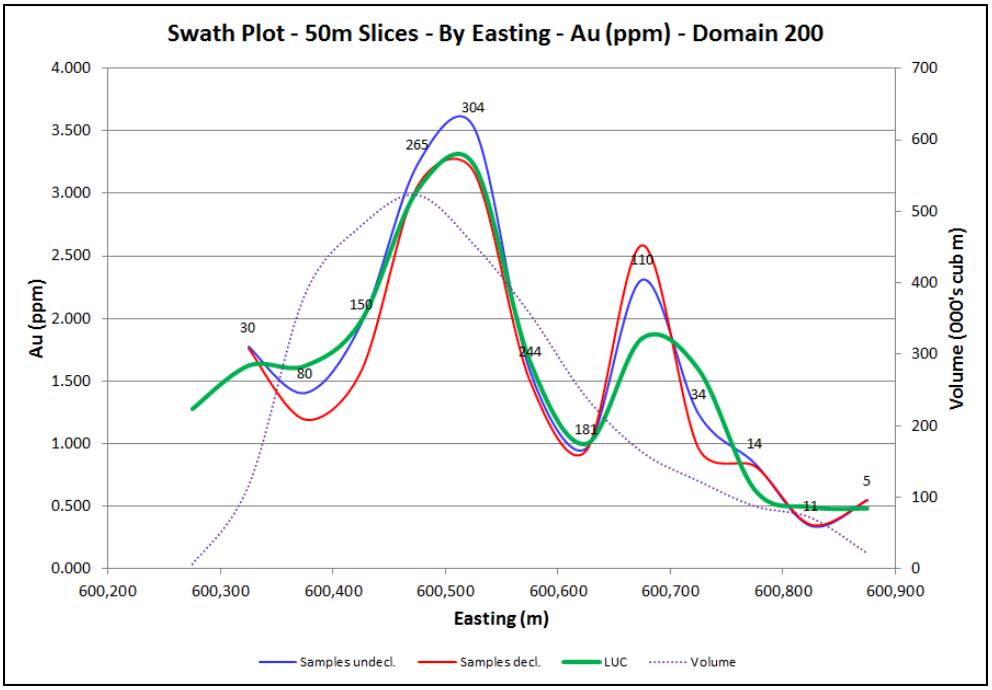


**Figure 14.10.7 Gbeitouo Block Model Validation by Easting – Domain 100**

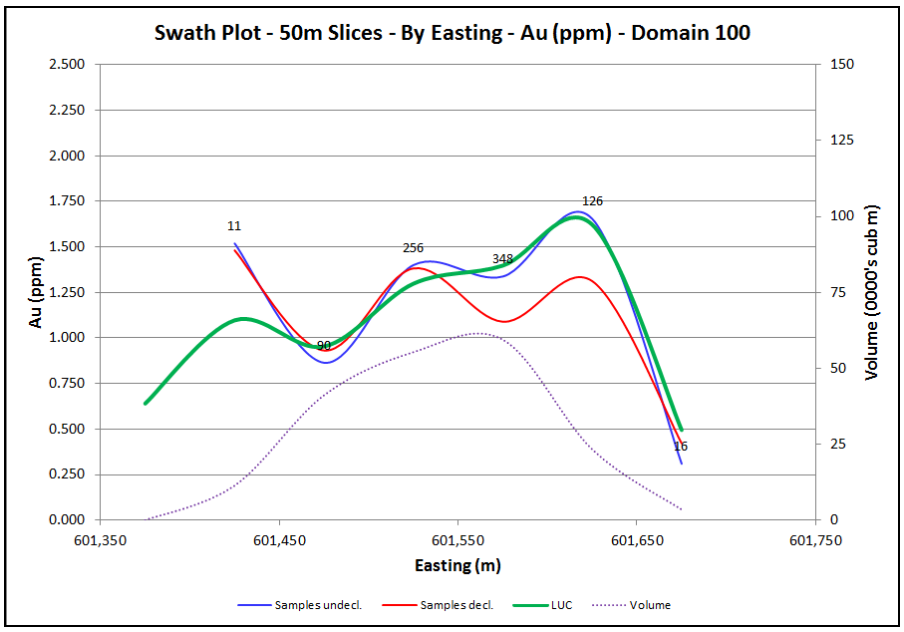



**Figure 14.10.8 Mont Ity / Ity Flat Block Model Validation by Easting – Domain 400**

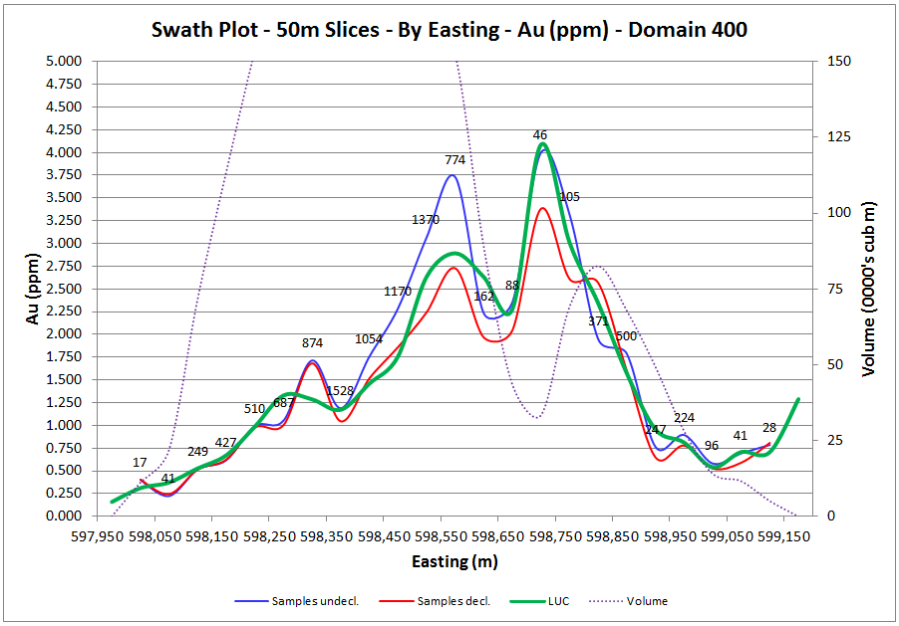


**Figure 14.10.9 Walter Block Model Validation by Northing – Domain 900**

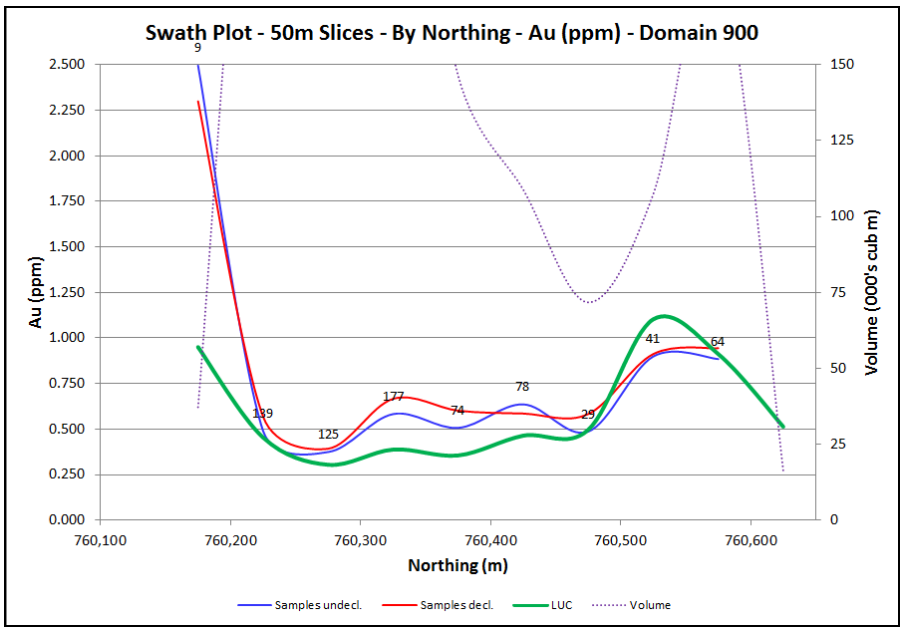
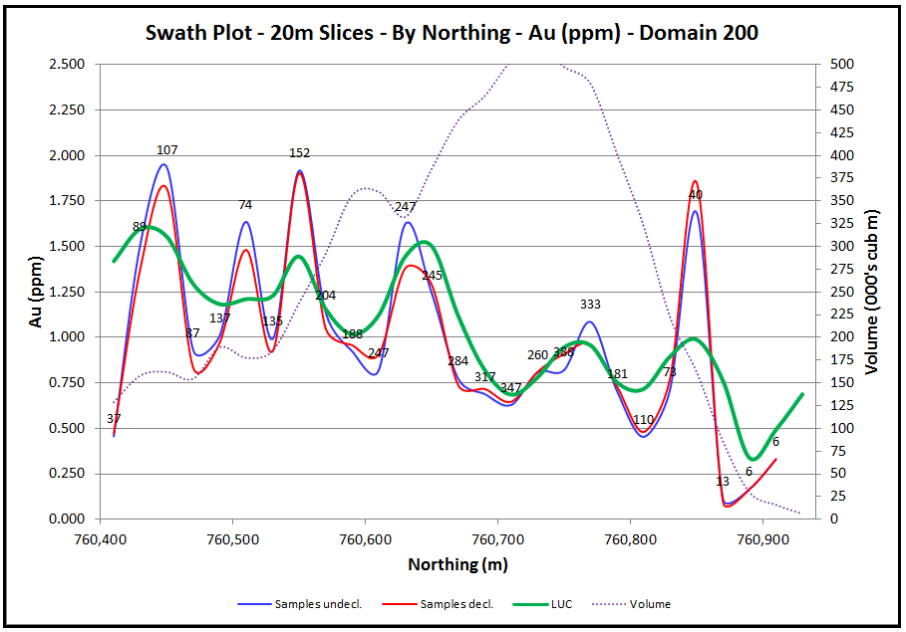



**Figure 14.10.10 ZiaNE Block Model Validation by Northing – Domain 200**



***Mining Depletion***

No mining depletion has been undertaken for either the Daapleu or Gbeitouo deposits.

Depletion by open pit mining at Mont Ity, Ity Flat and ZiaNE was active at the time of this Mineral Resource estimate update and has also been active in the past at the Walter deposit.

A detailed topographical survey (carrières_fin_mai_2016.dtm) of the mine workings was supplied to Cube by the mine survey department and represents the mining surface as at 31 May 2016. This surface was used to assign a depletion flag into the block model as summarised in Table 14.10.8.

**Table 14.10.8 Mining Depletion Block Model Assignment**

| Attribute Name | Assignment | Description |
|---|---|---|
| depletion | 0 | All material mined or originally air above the 31 May 2016 survey DTM |
| | 1 | All in situ material below the 31 May 2016 survey |

## 14.11 Mineral Resource Classification

The Ity CIL Project consists of a total of nine prospect areas which include four in situ gold deposits Mont Ity, Ity Flat, ZiaNE and Walter that have been or are currently in production plus two near mine in situ deposits Gbeitouo and Daapleu, two rock waste dumps Teckraie and Verse Ouest and a discontinued heap leach pad Aires. The nine main prospects lie within an area of approximately 5 km (East) by 3 km (North).

The in situ, rock dump and heap leach pad deposits have been sufficiently drilled to allow classification as an Indicated and Inferred Mineral Resources in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014). As with any non-rigidly defined classification there will always be some blocks within categories that depart from defined criteria. The final outcome must reflect a practical combination of both geological knowledge and estimation quality parameters that may be more numerical in nature. This approach to classification aims to avoid creating a complex numerically based ‘mosaic’ distribution of classified blocks.

A range of criteria were considered when assessing the Mineral Resource classification for each deposit and are detailed below.

***Geological Continuity and Mineralised Volume Models***

The continuity and volume of the in situ mineralised domains has been established from drilling to a confidence level where the grade and quantity can be reasonably assumed. The approach to defining the mineralised volume was firstly based on a geological model, modified by applying a geological cut-off grade for the mineralised wireframe. In general, the interpreted mineralised volumes have not been extrapolated to unreasonable distances beyond data limits.

For the rock dumps and heap leach pad, the approach to defining the mineralised volume for the stockpile material and laterite was based on drillhole logging. No cut-off grade has been used for domain modelling.

***Drill Spacing and Drill Data Quality***

The supplied drilling database represents an appropriate record of the drilling and sampling undertaken at the project. In general, drilling, surveying, sampling, analytical methods and controls are considered appropriate for the style of mineralisation under consideration and are considered sufficient to form the basis of a Mineral Resource estimate suitable for classification under the NI43-101 guidelines.

Cube has undertaken an independent QAQC analysis which has confirmed the integrity of the assay data, and endorses the drilling database for the purposes of Mineral Resource estimation.

The drill spacing for the insitu deposits is variable, generally ranging from nominal 25 m to 100 m spacing drillholes on 25 m to 100 m spaced sections. Indicated Mineral Resources are typically defined by a drill spacing of 25 m x 25 m but can include some areas up to 50 m x 50 m with an average distance to the nearest sample of less than 50 m. Inferred Mineral Resources include the remaining interpreted and estimated mineralisation. It is reasonable to expect that further resource definition drilling within the Inferred areas could result in significant material departures of grade and tonnage both positive and negative from the current Inferred Mineral Resource estimate, which is reflected in the classification.

The drill spacing for the Aires heap leach pad is dominated by regularly spaced 30 m x 30 m drillholes with a gap in drilling around the leach pad slope of approximately 70 m. This overall drill spacing is considered sufficient for classifying the Leach Pad domain as Indicated. The underlying Laterite domain has been classified as Inferred due to uncertainty regarding the grade distribution.

For the rock dumps, the drill spacing defining the majority of Teckraie is regularly spaced 30 m x 30 m drillholes. This drill spacing is considered sufficient for classifying the Teckraie rock dump as Indicated. The drill spacing defining the majority of Verse Ouest is less regular with an average drillhole spacing of approximately 55 m x 55 m. This drill spacing is considered sufficient for classifying the Verse Ouest rock dump as Inferred. The underlying Laterite domains have been classified as Inferred due to uncertainty regarding the grade distribution.

***Modelling Technique***

The 3D, and the associated search and interpolation parameters used in the LUC methodology for the in situ deposits is considered appropriate for estimation of the Indicated and Inferred Mineral Resources. The LUC methodology produces accurate grade-tonnage functions which are in good accordance with volume-variance relationship principles. It limits the smearing of high grade and the over-smoothing of grade compared to other methods.

The OK modelling method for the rock dumps and heap leach pad is considered appropriate for estimation of the Indicated and Inferred Mineral Resource and has produce robust and unbiased global estimates.

Appropriate risk adjustments in the form of high grade assay cuts have been applied in some cases to limit the influence of statistical outliers and rigorous model validation has been undertaken.

***Local Estimation Bias***

The comparison of the LUC estimation model with the OK and NN block estimation models has established that the relative difference in contained metal between the LUC model and models using other estimation methods is negligible.

***Conclusions***

Cube has assessed the relevant criteria and interpreted robust and simplistic classification boundaries to capture the level of confidence in the quality, quantity and distribution of the data. The Mineral Resources have been classified as a combination of Indicated and Inferred Mineral Resources. No material has been classified as Measured Mineral Resources due primarily to the lack of close spaced drilling. Figure 14.11.1 to Figure 14.11.7 display the overall block model classification for each of the deposits.

**Figure 14.11.1 Daapleu Mineral Resource Classification – Oblique View Looking South-East**

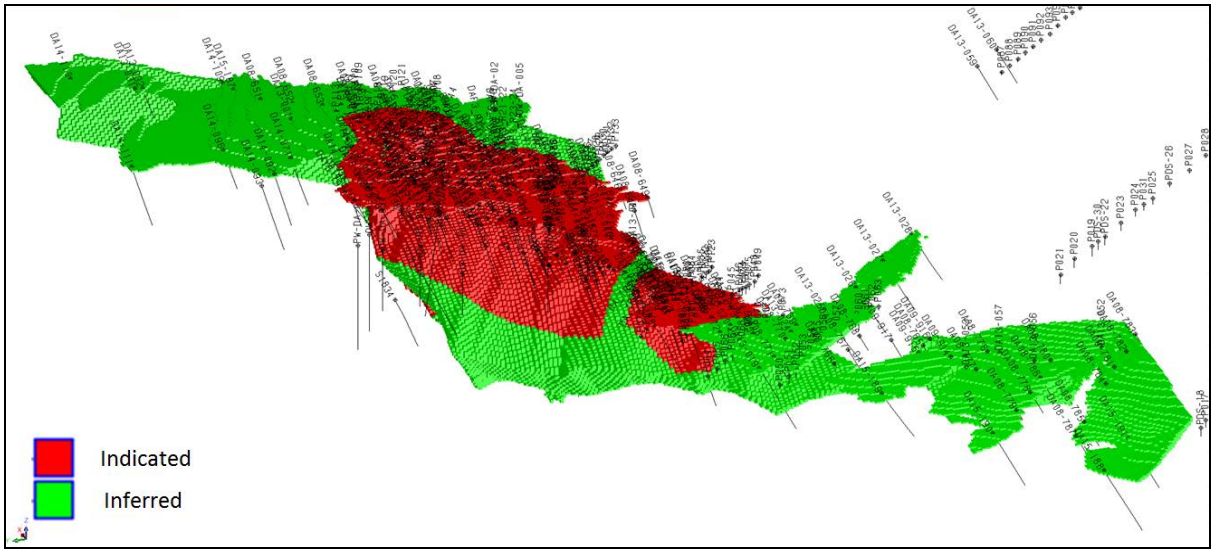


**Figure 14.11.2 Gbeitouo Mineral Resource Classification – Oblique View Looking East**

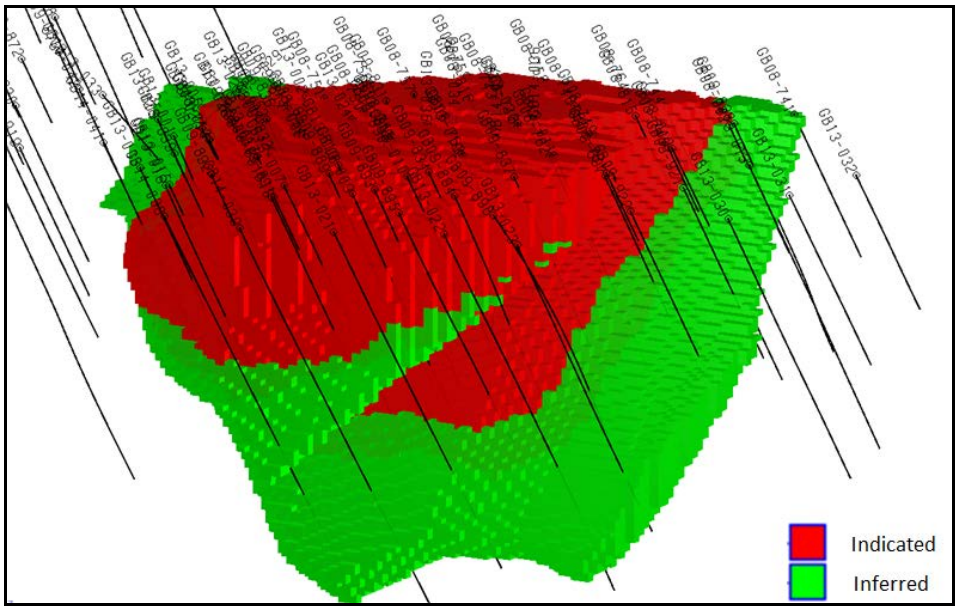



**Figure 14.11.3 Mont Ity / Ity Flat Mineral Resource Classification – Oblique View Looking North-East**

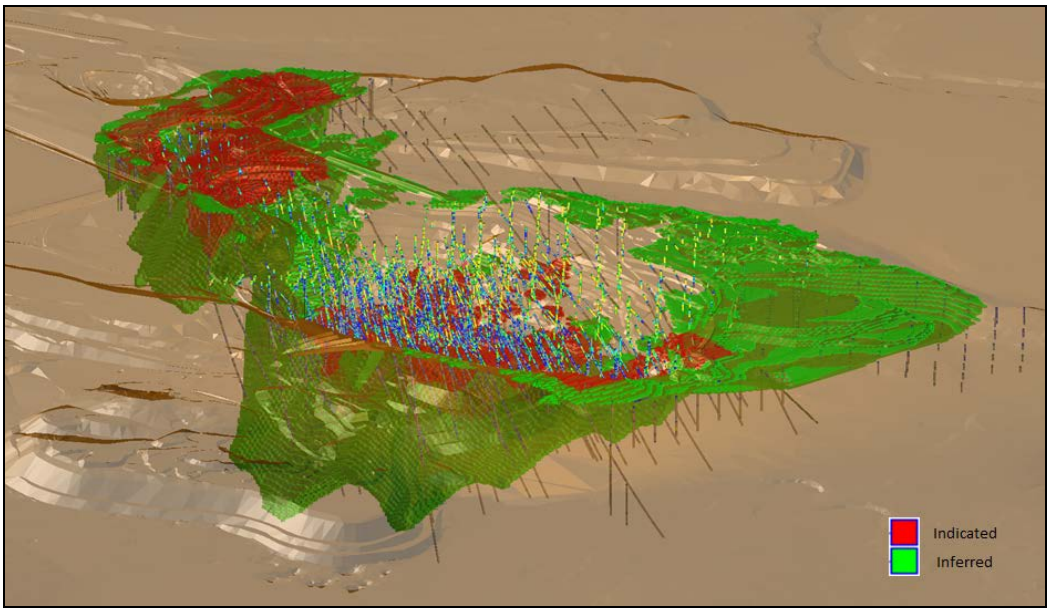


**Figure 14.11.4 Walter Mineral Resource Classification – Oblique View Looking East**

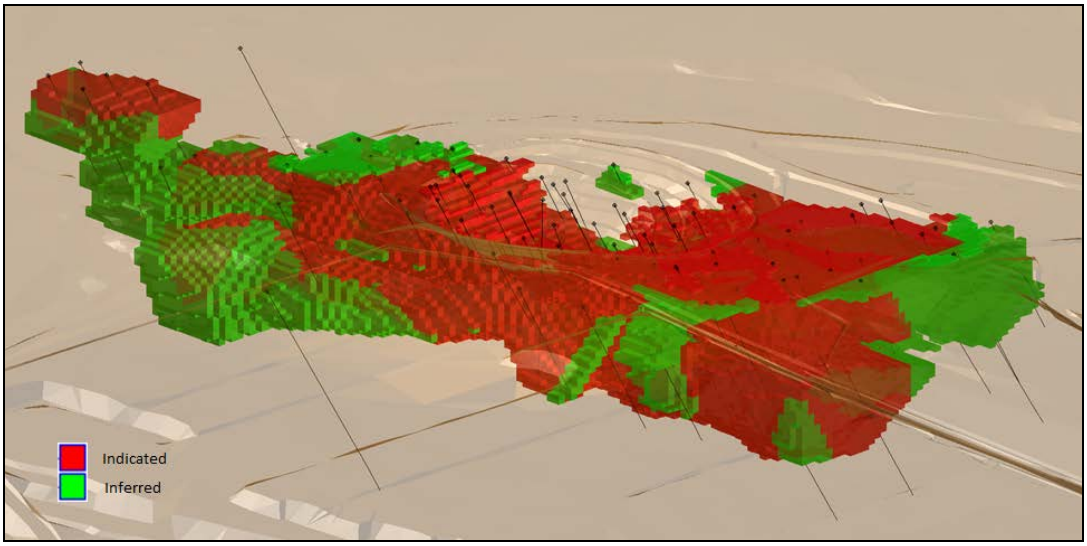



**Figure 14.11.5 ZiaNE Mineral Resource Classification – Plan View**

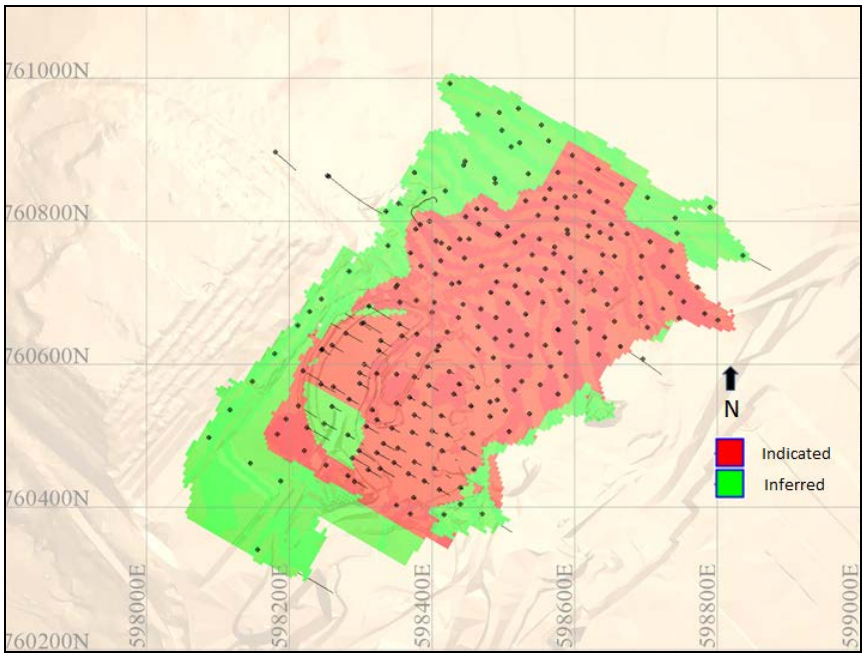


**Figure 14.11.6 Teckraie and Verse Ouest Mineral Resource Classification – Plan View**

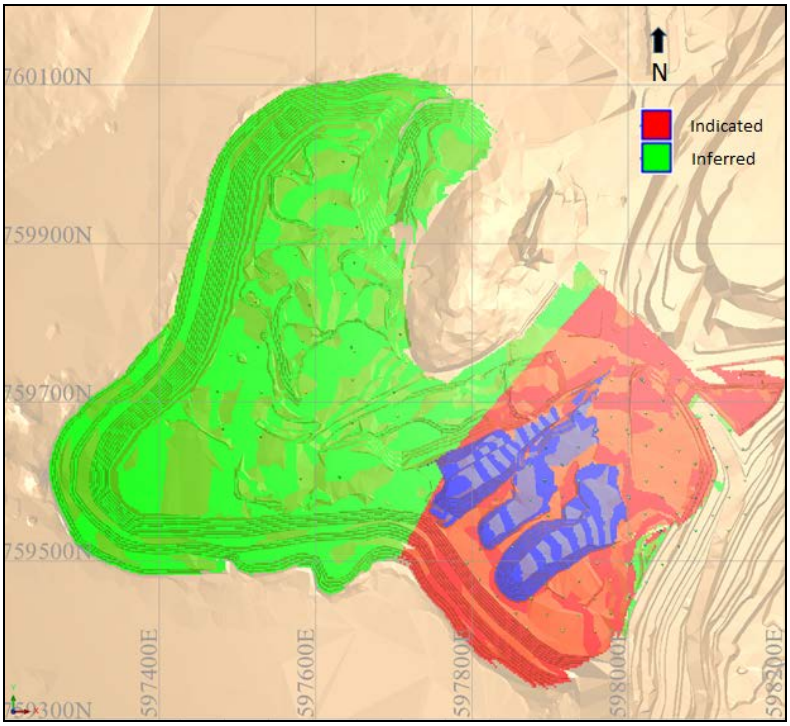
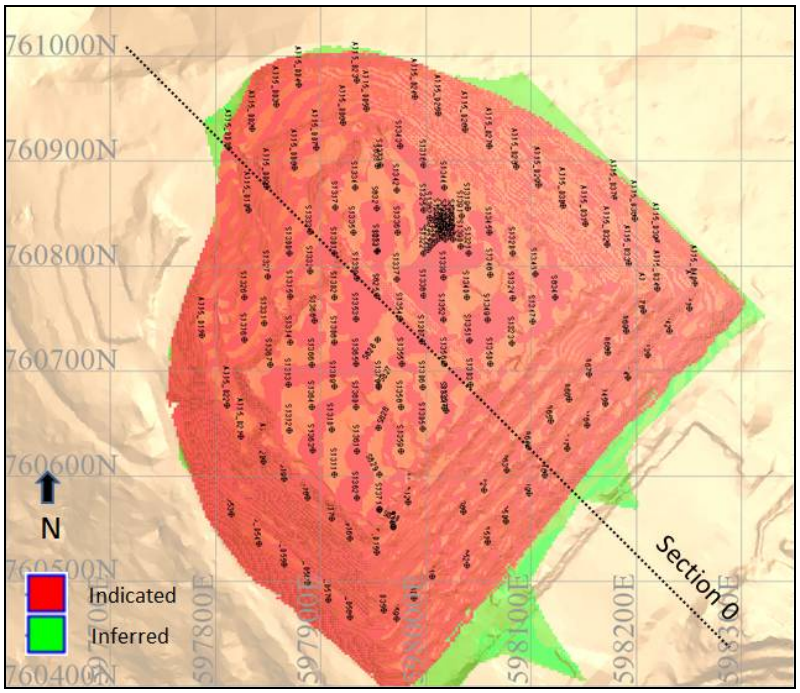



**Figure 14.11.7 Aires Mineral Resource Classification – Plan View**



## 14.12 Mineral Resource Statement

For the purpose of public reporting of the in situ Mineral Resources which include Daapleu, Gbeitouo, Mont Ity / Ity Flat, Walter and ZiaNE, these have been reported inside optimised pit shells and above a 0.5 g/t Au cut-off. Reporting within an optimised pit shell satisfies the requirement for the Mineral Resource to have reasonable prospects for future economic extraction. The pit optimisation assumes a 3 Mtpa operating costs and a US$1,500/oz Au price and Table 14.12.1 and Table 14.12.2 summarise the main economic parameters and the pit wall angles respectively used in the optimisation. No dilution or mining recoveries were applied to the optimisations as these are accounted for in the Local Uniform Conditioning (LUC) recoverable resource methodology used in the formulation of the resource models as described in Section 14.9.1.

**Table 14.12.1 Pit Optimisation Parameters for Mineral Resources Reporting Shells**

| Deposit | Recovery | Mining Cost $'000 | Throughput Cost $'000 | Total Cost $'000 | Revenue $'000 | Operating Surplus $'000 | Unit Cost $/oz |
|---|---|---|---|---|---|---|---|
| Daapleu | 74.8% | 231,605 | 422,350 | 653,955 | 1,269,807 | 615,852 | 743 |
| Gbeitouo | 87.7% | 31,907 | 55,962 | 87,869 | 178,814 | 90,945 | 708 |
| Mont Ity/Ity Flat | 91.6% | 216,259 | 311,354 | 527,613 | 1,291,645 | 764,032 | 589 |
| Walter | 95.6% | 26,403 | 53,481 | 79,884 | 155,915 | 76,031 | 739 |
| ZiaNE | 96.7% | 119,427 | 177,234 | 296,661 | 564,827 | 268,166 | 757 |
| **Total** | **85.3%** | **625,601** | **1,020,381** | **1,645,982** | **3,461,008** | **1,815,026** | **686** |

**Table 14.12.2 Pit Optimisation Slope Angles**

| Rock Type | Wall Angle |
|---|---|
| Soil, overburden and laterite | 28º |
| Saprolite, reduced clay, highly weathered rock | 33º |
| Transitional rock | 38º |
| Fresh rock | 45º |

For the purpose of public reporting of Mineral Resource for the rock dumps which include the Teckraie and Verse Ouest Mineral Resources and the Aires heap leach pad, these have not been reported inside an optimised pit shell. These deposits have been built up above the existing topography and the associated laterite is shallow and located directly below, therefore satisfying the requirement for the Mineral Resource to have reasonable prospects for future economic extraction. The Teckraie and Verse Ouest rock dump Mineral Resources and Aires leach pad Mineral Resources have been reported above 0 g/t Au because there is unlikely to be any grade selectively during mining. The underlying laterite Mineral Resources for each of the deposits has been reported above 0.5 g/t Au given the possibility for some mining selectively.

All Mineral Resources are current as at 31 May 2016 and have been reported by cut-off grade and Mineral Resource category which is presented in Table 14.12.3. Mineral Resources are reported inclusive of Mineral Reserves.

**Table 14.12.3 Mineral Resources as at Effective Data 31 May 2016**

| Deposit | Type | Cut-Off | Indicated | | | Inferred | | |
|---|---|---|---|---|---|---|---|---|
| | | | Tonnes ('000t) | Au (g/t) | Au ('000 Oz) | Tonnes ('000) | Au (g/t) | Au ('000 Oz) |
| Daapleu | In Situ | 0.5 | 19,882 | 1.51 | 965 | 4,329 | 1.15 | 160 |
| Mont Ity / Ity Flat | | 0.5 | 7,491 | 2.19 | 527 | 11,084 | 1.92 | 684 |
| Gbeitouo | | 0.5 | 2,865 | 1.35 | 124 | 270 | 1.48 | 13 |
| Walter | | 0.5 | 2,069 | 1.21 | 81 | 657 | 1.32 | 28 |
| ZiaNE | | 0.5 | 7,723 | 1.31 | 325 | 4,002 | 1.39 | 179 |
| Aires | Leach Pad | 0 | 5.768 | 1.09 | 202 | | | |
| | | 0.5 | | | | 233 | 0.78 | 6 |
| Teckraie | Rock Dump | 0 | 2,819 | 1.07 | 97 | | | |
| | | 0.5 | | | | 114 | 0.55 | 2 |
| Verse Ouest | | 0 | | | | 5,896 | 1.03 | 195 |
| | | 0.5 | | | | 2,505 | 0.43 | 35 |
| **Total** | | | **48,617** | **1.49** | **2,322** | **29,090** | **1.47** | **1,302** |

Mineral Resources are reported inclusive of Mineral Reserves.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Rounding as required by reporting guidelines may result in small discrepancies.

Mineral Resources are reported on a 100% basis.

Cut-off grade of 0.5 g/t Au for situ Mineral Resources including Daapleu, Gbeitouo, Mont Ity / Ity Flat, Walter and ZiaNE have been reported inside optimised pit shells and based on 3 Mtpa CIL operating costs and a US$1,500/oz Au price.

The Teckraie and Verse Ouest rock dump and Aires leach pad Mineral Resources have been reported above 0 g/t Au because there is unlikely to be any grade selectively during mining.

The Laterite mineralisation below Aires, Teckraie and Verse Oust have been reported above a cut-off grade 0.5 g/t Au.

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# 15.0 MINERAL RESERVE ESTIMATES

## 15.1 Mining and Mineral Reserve Estimation Approach

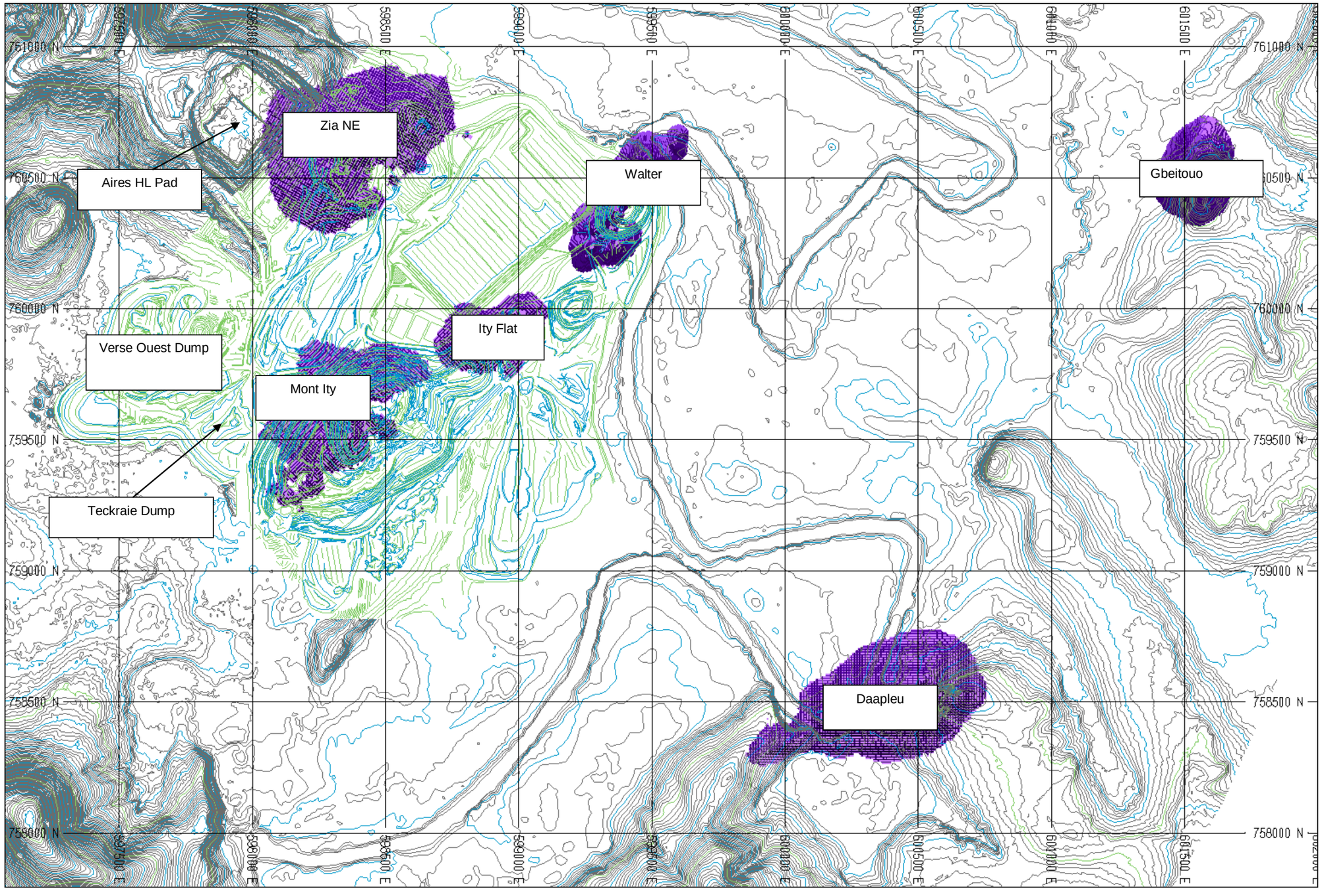
The mining study includes six gold deposits, named Ity Flat, Mont ity, Zia NE (north-east), Walter, Gbeitouo and Daapleu (Figure 15.1.1). The first three deposits are currently mined to supply ore to the heap leach plant and the Walter pit was mined recently. The CIL mine plan excludes the heap leach reserves for these pits as estimated by Endeavour. It is assumed that the CIL operations would start at the beginning of 2019 after two years of heap leach processing at 1,050 ktpa. The mined ore not suitable for heap leaching but stockpiled for CIL operation was also not taken into account in the CIL reserve estimate.

Decommissioned Aires leach pad and Teckraie dump contain Indicated and Verse Ouest dump contains Inferred resources that were included in the preliminary production schedules. Verse Ouest dump was later excluded from the CIL reserves and final production schedule. Although the grades are relatively low for these stockpiles (slightly above 1.0 g/t), they provide relatively low cost ounces for the CIL project.

The tasks completed for the planning of the Ity open pits can be summarised as follows:

- Site visit between 29 May and 1 June 2016 to identify any issues that may affect the development of the open pits, selection of the mining equipment and operating cost estimates.
- Data transfer and preparation of mining resource models and estimation of preliminary operating costs for pit optimisation and reserve estimation purposes.
- Pit limit optimisations to determine the ultimate pit limits and pit stages.
- Preliminary production schedules for plant throughput rate and cut-of grade optimisation purposes.
- Mine design, including pit designs, waste dump designs, stockpile areas and haul roads.
- Calculation of open pit mining inventory and reserves statement.
- Preparation of detailed production schedules for mining, milling and stockpiling operations based on multiple pits and ore types.
- Mining equipment and manpower estimates based on the production schedule.
- Mine operating and capital cost estimates from first principles.
- Reconciliation of the study results and preparation of final study report.

**Figure 15.1.1 Ity Project Site Plan View – Location of Deposits and Pit Shells**



The major inputs to the mining study were provided by the following sources:

- All the resource models and topographic surface used in the study were provided by Cube Consulting (Cube) by late July 2016.
- Endeavour provided the end of heap leach pit design surfaces as the starting surface for the CIL study in August 2016.
- General and administration costs, processing costs and process recoveries were supplied by Lycopodium Engineering Pty Ltd (Lycopodium, August 2016).
- Geotechnical appraisal and pit slope recommendations were provided by Peter O'Bryan & Associates (POA, August / September 2016).
- Preliminary mining cost estimates for pit optimisation purposes were completed by Mining Solutions Consultancy Pty Ltd (MSC, August 2016).
- Mining equipment selection, mine capital and operating cost estimates were completed by Calsta Pty Ltd (Calsta, September 2016) in consultation for Mining Solutions.

## 15.2 Key Assumptions / Basis of Estimates

### 15.2.1 Block Models and Surfaces

The resource models received from Cube included the gold grades, rock type (facies), oxidation state, confidence category and density values. The topographic surfaces received from Cube and the end of heap leach design surfaces received from Endeavour were applied as the starting surface in the pit optimisations and volume / tonnage calculations.

The resource models provided were already classified into Indicated and Inferred categories complying with the CIM Definition Standards for Mineral Resources and Mineral Reserves . Also in line with the CIM Definition Standards, only the Indicated category resources were included in the pit optimisations and the resultant reserve estimates in the pit designs.

### 15.2.2 Dilution and Ore Loss

No additional mining dilution or ore recovery factors were applied to the reported reserves since the resource estimates already incorporate dilution for the selective mining unit (SMU, 5 m x 5 m x 2.5 m) size for open pit mining.

### 15.2.3 Gold Price, Royalty and Discounting

A gold price of US$1,250/oz was used in the pit optimisations and the calculation of the economic cut-off grades for mineral reserves reporting. As shown in Table 15.2.1 the net gold price is $1,200.60/oz after deducting refining charges and royalty from the payable gold price.

The indicative operating NPVs (Net Present Value) in pit optimisations were calculated using a discount rate of 10%. The selection of the optimal pit limits were not found to be sensitive to the change in the discount rate due to the relatively short mine life except the Daapleu deposit.

**Table 15.2.1 Gold Price and Deductions**

| | | |
|---|---|---|
| Gold Price | $/oz | 1,250 |
| Valorem Tax | % | 3.50 |
| Payable Gold | % | 99.95 |
| Refining and Transport | $/oz | 5.00 |
| Net Price | $/oz | 1,200.63 |
| Pit Optimisation Net Price | $/g | 38.60 |

### 15.2.4 Mining Cost

The preliminary mining cost estimates used in the pit optimisations were based on similar size owner mining operations employing 90 t trucks and 120 t class hydraulic excavators. Ground conditions were also considered in the preliminary estimates depending on the density values. The oxide and transitional material with low densities (<2.0 t/bcm) were classified as free dig and the transitional and fresh rocks with high densities (>2.5 t/bcm) were classified for drilling and blasting.

The resultant average mining cost reported in the pit optimisations is $2.55/t. This preliminary estimate is approximately 20% higher than the final mine operating cost estimate in Section 21.1. A rehandling cost of $1.30/t was estimated for the stockpile reclaim.

### 15.2.5 Throughput Rates and Operating Costs

As the base case for the feasibility study, a throughput rate of 3.0 Mtpa was used in the production schedules and cost estimates. The sensitivity of the project for 2.0 Mtpa, 3.5 Mtpa and 4.0 Mtpa throughput rate was also investigated in the pit optimisations.

The throughput costs used in the pit optimisations for 3.0 Mtpa base case have been listed in Table 15.2.2. The processing and general and administration costs in the first section of the table are the preliminary estimates supplied by Lycopodium. The high arsenic ore in the table is to represent the sulphide mineralisation with lower CIL recoveries. The mining costs per tonne of ore in the second section of the table, including pit supervision, grade control, ore rehandle and haulage costs, are the preliminary estimates carried out by MSC.

Note that the mine operating cost estimates have been revised as the detailed estimates carried out in the finalisation stage of the study. Please refer to the operating cost estimate in Section 21.1 of this report for the details of final estimates.

**Table 15.2.2 Throughput Cost Preliminary Estimates – 3.0 Mtpa**

| Throughput Costs | | Daapleu | Gbeitouo | Ity Flat | Mont Ity | Walter | Zia NE |
|---|---|---|---|---|---|---|---|
| **Processing & G&A** | | | | | | | |
| Ore (Non-Sulphide) | $/t ore | 10.87 | 12.42 | 15.54 | 15.76 | 14.48 | 13.60 |
| High Arsenic Ore (Sulphide) | $/t ore | 12.82 | | | | | |
| **Mining** | | | | | | | |
| Mine Supervision | $/t ore | 1.07 | 1.07 | 1.07 | 1.07 | 1.07 | 1.07 |
| Extra Ore Cost | $/t ore | 0.25 | 0.25 | 0.25 | 0.25 | 0.25 | 0.25 |
| Grade Control | $/t ore | 0.65 | 0.60 | 0.65 | 0.65 | 0.65 | 0.65 |
| Crusher Feed | $/t ore | 0.75 | 0.65 | 0.75 | 0.75 | 0.75 | 0.75 |
| Ore haulage and stockpile rehandling | $/t ore | 0.96 | 1.27 | 0.13 | 0.13 | 0.32 | 0.13 |
| Total Mine Throughput | $/t Ore | 3.68 | 3.83 | 2.84 | 2.84 | 3.04 | 2.84 |
| **Total Throughput Cost** | | | | | | | |
| Ore (Non-Sulphide) | $/t ore | 14.55 | 16.25 | 18.38 | 18.60 | 17.52 | 16.44 |
| High Arsenic Ore (Sulphide) | $/t ore | 16.50 | | | | | |

### 15.2.6 Process Recoveries

Based on the metallurgical test work results, the process recoveries recommended by Lycopodium for pit optimisation purposes are listed in Table 15.2.3.

**Table 15.2.3 Process Recoveries**

| Process Recovery | | Daapleu | Gbeitouo | Ity Flat | Mont Ity | Walter | Zia NE |
|---|---|---|---|---|---|---|---|
| Ore (Non-Sulphide) | % | 90.3 | 87.7 | 90.0 | 92.7 | 96.0 | 96.7 |
| High Arsenic Ore (Sulphide) | % | 59.1 | | | | | |

### 15.2.7 Pit Slope Angles

The pit slope angles used in the pit optimisations were based on POA preliminary recommendations (August 2016) as follows:

- 28 degrees in soil, overburden and laterite.
- 33 degrees in saprolite, reduced clay material, highly weathered rock.
- 38 degrees in transitional rock.
- 45 degrees in fresh rock.

These slope angles were flattened by 3 degrees in the pit optimisations to take into account the effect of the ramps on the pit wall slopes. The summary of the geotechnical investigation results and the details of pit wall design parameters used in the pit designs are provided in Section 16.3 below.

### 15.2.8 Capital and Other Costs

No allowances were made for capital costs or taxation in the calculation of the project cash flows or NPVs in the pit optimisations. The pit optimisation studies and evaluations were carried out on the basis of the operating costs and values. The impact of capital cost, taxation and project financing requirements are to be assessed by Endeavour in the financial analysis stage of the study. As an input to the project financial model, the mine capital and operating cost estimates are provided in Sections 21.1.1 and 21.2.2 of this report.

## 15.3 Pit Optimisation Results

Pit optimisations were carried out using Whittle Four-X pit optimisation software. For given block model, cost, recovery and slope data, Whittle Four-X software calculates a series of incremental pit shells in which each shell is an optimum for a slightly higher price factor. These pit shells are analysed in Whittle through preliminary production and cost schedules to generate indicative operating NPVs. The optimal pit shells for the Ity deposits have been selected on the basis of the maximum indicative operating NPV criterion.

Table 15.3.1 summarises the optimal pit physical and financial quantities for the 2.0 Mtpa, 3.0 Mtpa, 3.5 Mtpa and 4.0 Mtpa throughput rates. The indicative NPVs are operating values discounted to the start of processing for each pit for comparison purposes.

The results in Table 15.3.1 show that the higher throughput rates add significant value to the project compared to the 2.0 Mtpa option. The resource tonnage is 20% and the operating value is 13% higher for the 3.0 Mtpa option. The increase from 3.0 Mtpa to 4.0 Mtpa case is relatively subtle with 10% increase in the resource tonnage and 8% increase in the operating value.

## Table 15.3.1 Pit Optimisation Results Summary

| Deposit | Pit No | Resource kt | Resource g/t | Waste kt | Total kt | W/O Ratio kt | Mining Cost $'000 | Th'put Cost $'000 | Total Cost $'000 | Revenue $'000 | Operating Surplus $'000 | Unit Cost $/oz | Indicative Op. NPV $'000 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **2 Mtpa Option** | | | | | | | | | | | | | |
| Daapleu | 21 | 14,559 | 1.70 | 26,745 | 41,305 | 1.84 | 112,383 | 271,077 | 383,460 | 700,919 | 317,459 | 657 | 224,729 |
| Gbeitouo | 25 | 2,085 | 1.51 | 4,812 | 6,897 | 2.31 | 16,262 | 40,958 | 57,220 | 106,486 | 49,266 | 645 | 44,782 |
| Mont Ity+Flat | 22 | 3,473 | 2.41 | 10,366 | 13,839 | 2.98 | 36,330 | 76,172 | 112,502 | 295,995 | 183,493 | 456 | 162,652 |
| Walter | 24 | 1,461 | 1.36 | 3,388 | 4,849 | 2.32 | 10,905 | 30,559 | 41,464 | 73,208 | 31,744 | 680 | 29,609 |
| Zia NE | 23 | 3,926 | 1.39 | 17,228 | 21,154 | 4.39 | 45,399 | 77,970 | 123,369 | 204,366 | 80,997 | 725 | 69,468 |
| **Total** | | **25,505** | **1.72** | **62,539** | **88,044** | **2.45** | **221,279** | **496,736** | **718,015** | **1,380,974** | **662,959** | **624** | **531,239** |
| | Years | 12.8 | | | | | | | | | -102,776 | | -101,341 |
| **3 Mtpa Option** | | | | | | | | | | | | | |
| Daapleu | 25 | 18,391 | 1.52 | 30,948 | 49,339 | 1.68 | 135,700 | 279,606 | 415,306 | 793,795 | 378,489 | 628 | 281,221 |
| Gbeitouo | 26 | 2,388 | 1.40 | 4,948 | 7,336 | 2.07 | 17,335 | 38,803 | 56,138 | 113,183 | 57,045 | 595 | 52,878 |
| Mont Ity+Flat | 27 | 4,028 | 2.18 | 11,040 | 15,068 | 2.74 | 39,441 | 74,553 | 113,994 | 310,745 | 196,751 | 440 | 177,646 |
| Walter | 27 | 1,780 | 1.25 | 4,106 | 5,886 | 2.31 | 13,442 | 31,179 | 44,621 | 82,124 | 37,503 | 652 | 35,441 |
| Zia NE | 27 | 4,770 | 1.25 | 17,227 | 21,997 | 3.61 | 47,495 | 78,411 | 125,906 | 221,853 | 95,947 | 681 | 85,394 |
| **Total** | | **31,356** | **1.54** | **68,269** | **99,626** | **2.18** | **253,413** | **502,552** | **755,965** | **1,521,700** | **765,735** | **596** | **632,580** |
| | Years | 10.5 | | | | | | | | | | | |
| **3.5 Mtpa Option** | | | | | | | | | | | | | |
| Daapleu | 26 | 19,085 | 1.49 | 31,533 | 50,618 | 1.65 | 139,325 | 269,911 | 409,236 | 808,263 | 399,027 | 608 | 304,086 |
| Gbeitouo | 28 | 2,567 | 1.35 | 5,276 | 7,843 | 2.06 | 18,698 | 39,042 | 57,740 | 117,533 | 59,793 | 590 | 55,757 |
| Mont Ity+Flat | 29 | 4,244 | 2.11 | 11,479 | 15,724 | 2.70 | 41,171 | 74,090 | 115,261 | 316,451 | 201,190 | 437 | 182,363 |
| Walter | 29 | 1,863 | 1.22 | 4,129 | 5,992 | 2.22 | 13,708 | 30,666 | 44,374 | 83,834 | 39,460 | 636 | 37,509 |
| Zia NE | 29 | 4,970 | 1.22 | 17,249 | 22,219 | 3.47 | 48,059 | 76,493 | 124,552 | 225,689 | 101,137 | 663 | 90,842 |
| **Total** | | **32,729** | **1.50** | **69,666** | **102,396** | **2.13** | **260,961** | **490,202** | **751,163** | **1,551,770** | **800,607** | **581** | **670,557** |
| | Years | 9.4 | | | | | | | | | 34,872 | | 37,977 |
| **4 Mtpa Option** | | | | | | | | | | | | | |
| Daapleu | 28 | 20,311 | 1.44 | 31,913 | 52,223 | 1.57 | 143,737 | 271,647 | 415,384 | 830,402 | 415,018 | 601 | 320,364 |
| Gbeitouo | 29 | 2,755 | 1.32 | 5,891 | 8,646 | 2.14 | 20,880 | 39,785 | 60,665 | 122,693 | 62,028 | 594 | 58,087 |
| Mont Ity+Flat | 30 | 4,352 | 2.08 | 11,800 | 16,152 | 2.71 | 42,287 | 72,657 | 114,944 | 319,419 | 204,475 | 432 | 185,775 |
| Walter | 30 | 1,994 | 1.19 | 4,724 | 6,717 | 2.37 | 15,418 | 31,278 | 46,696 | 87,758 | 41,062 | 639 | 39,157 |
| Zia NE | 30 | 5,317 | 1.17 | 17,394 | 22,710 | 3.27 | 49,230 | 77,728 | 126,958 | 232,110 | 105,152 | 657 | 94,884 |
| **Total** | | **34,727** | **1.45** | **71,722** | **106,449** | **2.07** | **271,552** | **493,095** | **764,647** | **1,592,382** | **827,735** | **577** | **698,266** |
| | Years | 8.7 | | | | | | | | | 62,000 | | 65,687 |

| **Mtpa** | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 2.0 | | 25,505 | 1.72 | 62,539 | 88,044 | 2.45 | 221,279 | 496,736 | 718,015 | 1,380,974 | 662,959 | 624 | 531,239 |
| 3.0 | | 31,356 | 1.54 | 68,269 | 99,626 | 2.18 | 253,413 | 502,552 | 755,965 | 1,521,700 | 765,735 | 596 | 632,580 |
| 3.5 | | 32,729 | 1.50 | 69,666 | 102,396 | 2.13 | 260,961 | 490,202 | 751,163 | 1,551,770 | 800,607 | 581 | 670,557 |
| 4.0 | | 34,727 | 1.45 | 71,722 | 106,449 | 2.07 | 271,552 | 493,095 | 764,647 | 1,592,382 | 827,735 | 577 | 698,266 |

## 15.4 Mine Design

### 15.4.1 Design Criteria

The pit design contours were generated at 5 m vertical intervals in line with the blasting bench height. The criteria used in the pit designs can be summarised as follows:

- Ramp gradient: 1 in 9
- Dual lane ramp width: 22.0 m
- Pit slope design: Based on the slope designs specified in Section 4.4
- Minimum mining width: 22.0 m (Ramp width)
- Dump slope design: 18 degrees overall slopes

The pit wall slope design parameters used in the pit designs are reported in Section 16.x below. Note that the base of complete oxidation (BOCO) and top of fresh (TOFR) surfaces which the slopes designs are based on are generally highly variable. As it was not practical to follow these variable surfaces, the approximate average surface elevations were used in the pit designs. The resultant designs are for feasibility study evaluation purposes and needs to be reviewed for the variable local geological conditions before the implementation.

The dual ramp width of 22 m is standard in the designs for the full depth of the pits. If the local geological and geotechnical conditions allow, the last 30 m depth of the pits can be redesigned 15 m wide ramps for single lane operation to improve the open pit economics.

### 15.4.2 Daapleu Stage 1 and 2 Pit Designs

It is planned to develop the Daapleu deposit in two stages as shown in Figures 15.4.1 and 15.4.2. Stage 1 pit is 125 m deep targeting the relatively shallow and high grade ore at the footwall side. The south and east walls of the Stage 1 pit are the final walls. Stage 2 cutback pushes the north and west walls of the pit back to develop ultimate pit limits to 210 m depth.

The access to Stage 2 cutback will be initially through the temporary ramp located at the northern wall. The temporary ramp will be partially mined down as the lower level access established from the ramp developed in Stage 1. This portion of the Stage 1 ramp will be developed further in Stage 2 to access the lower mining levels. The Daapleu pit exit is located at the east relatively close to the proposed waste dump.

Cavally River needs to be diverted and a bridge needs to be constructed to be able to develop the Daapleu and Gbeitouo pits.

**Figure 15.4.1 Daapleu Stage 1 Pit Design**

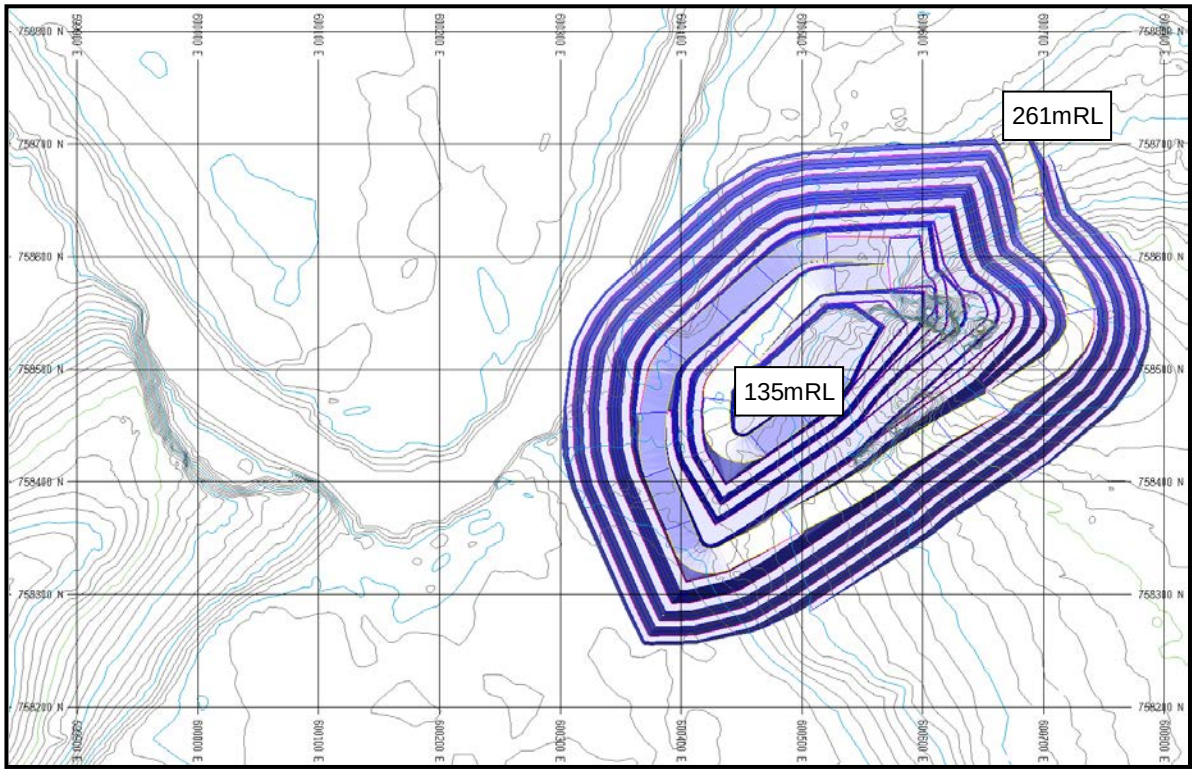
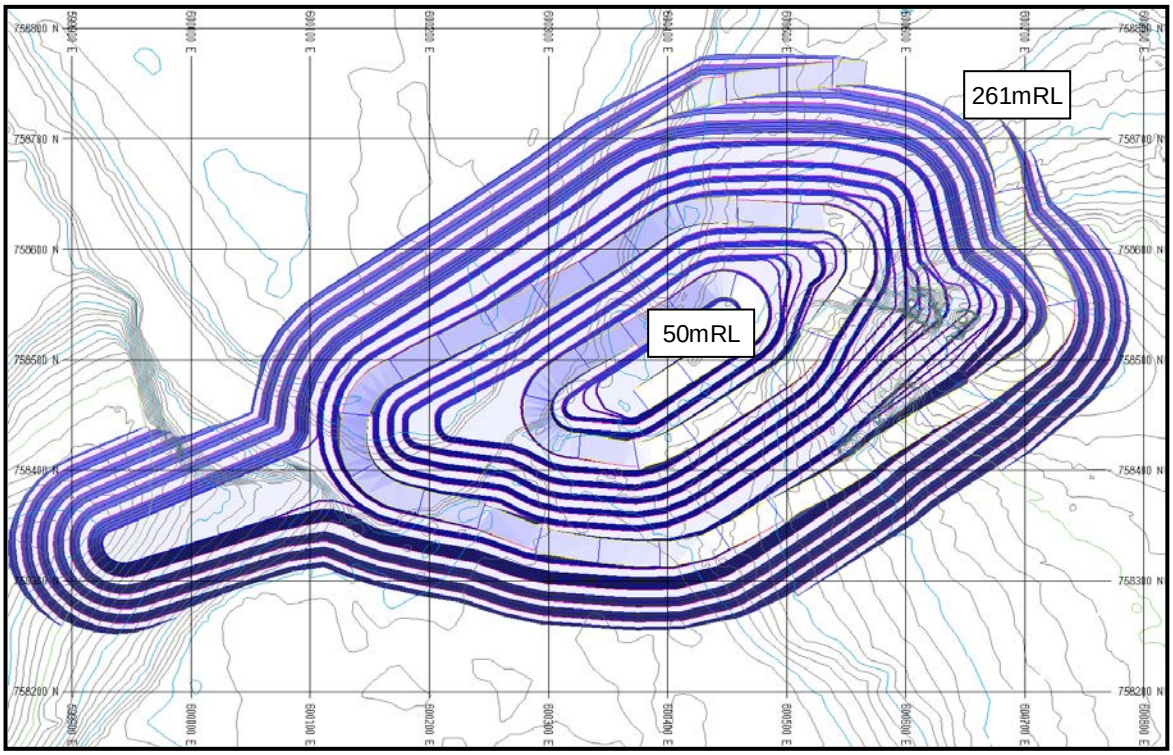


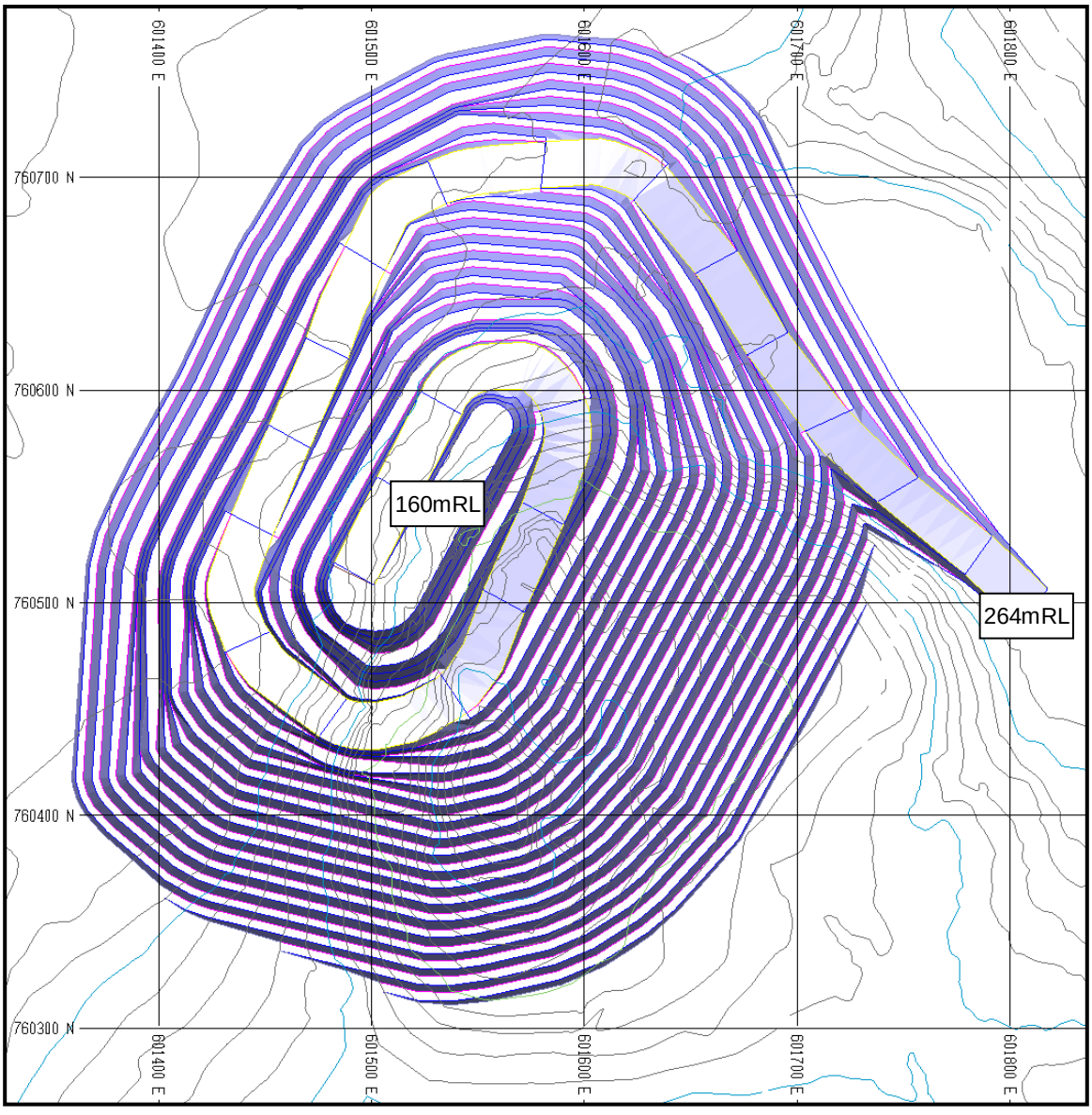

**Figure 15.4.2 Daapleu Stage 2 Cutback Design**



### 15.4.3 Gbeitouo Pit Design

The Gbeitouo pit design seen in Figure 15.4.3 is 330 m wide and 450 m long at the surface, developing ore down to a depth of 100 m. The ramp exit is located at the south east close to the proposed Gbeitouo waste dump.

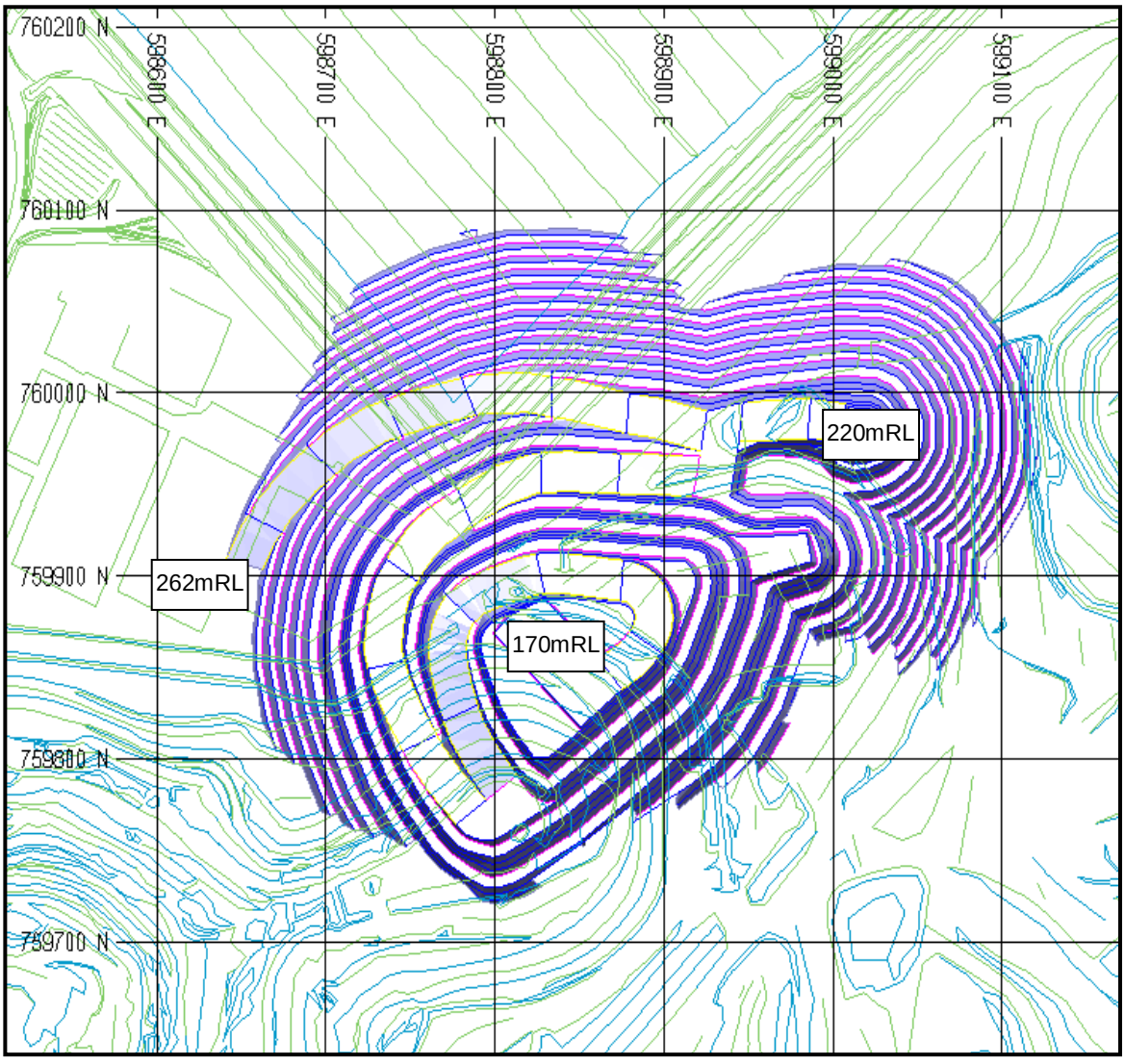
**Figure 15.4.3 Gbeitouo Pit Designs**



### 15.4.4 Ity Flat Pit Design

The Ity Flat pit cutback design shown in Figure 15.4.4 is approximately 90 m deep below the pit exit elevation. The access is through the switch-back ramp system located on the north and west walls. The south-west corner of the currently operating heap leach pad needs to be mined to develop the proposed design. The ramp exit is located at the west in the direction of the proposed Ity Flat and Mont Ity waste dump.

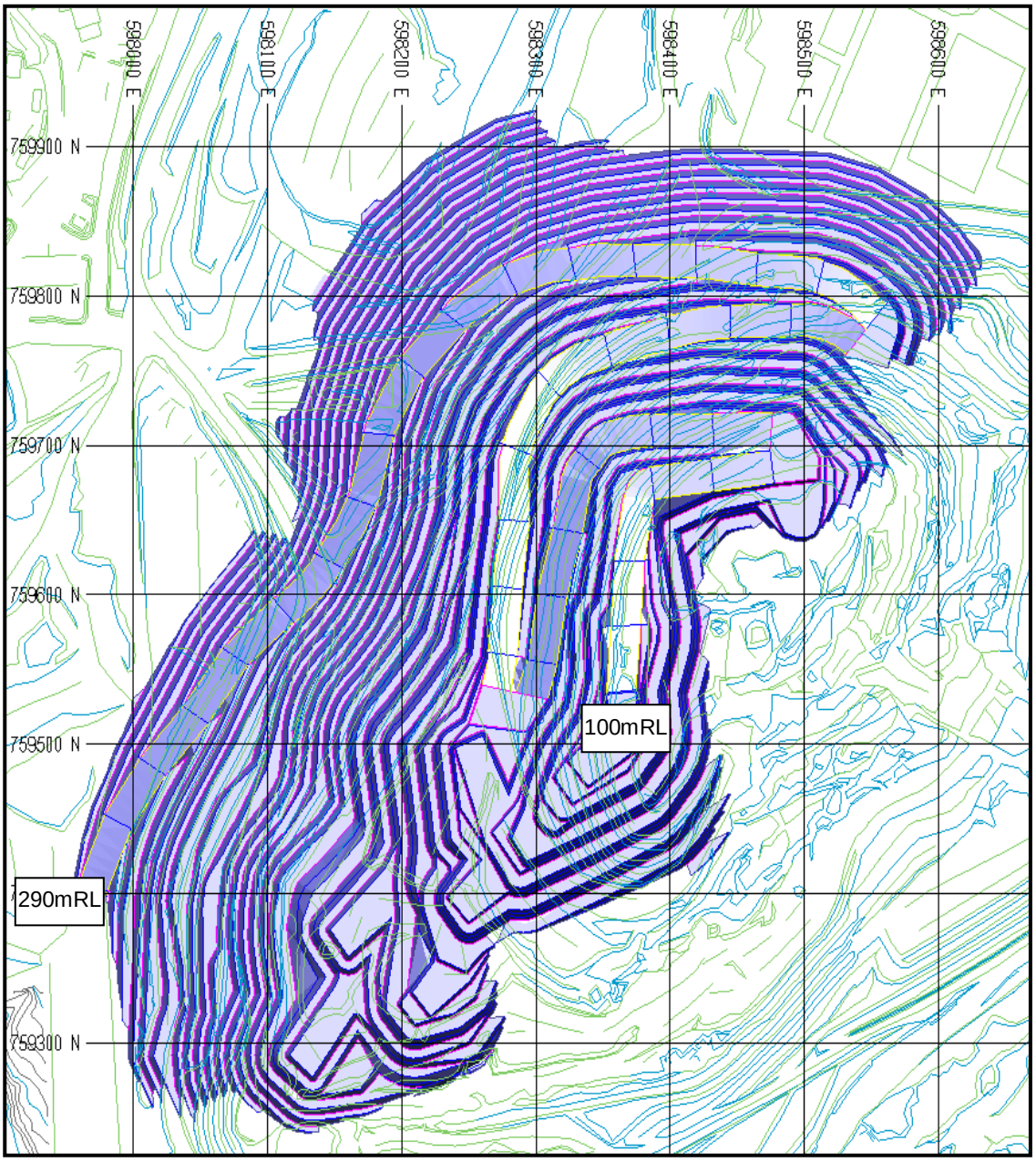
**Figure 15.4.4 Ity Flat Pit Cutback Design**



### 15.4.5 Mont Ity Pit Design

The Mont Ity pit cutback design shown in Figure 15.4.5 pushes the north and west walls of the current pit to develop ore down to a depth of 190 m. The cutback width is minimal (22 m) initially at the upper levels of the southern part of the western wall. The access to the design is through the switch-back ramp system developed on the west and north pit walls. The ramp exit is located close to the proposed waste dump at the west.

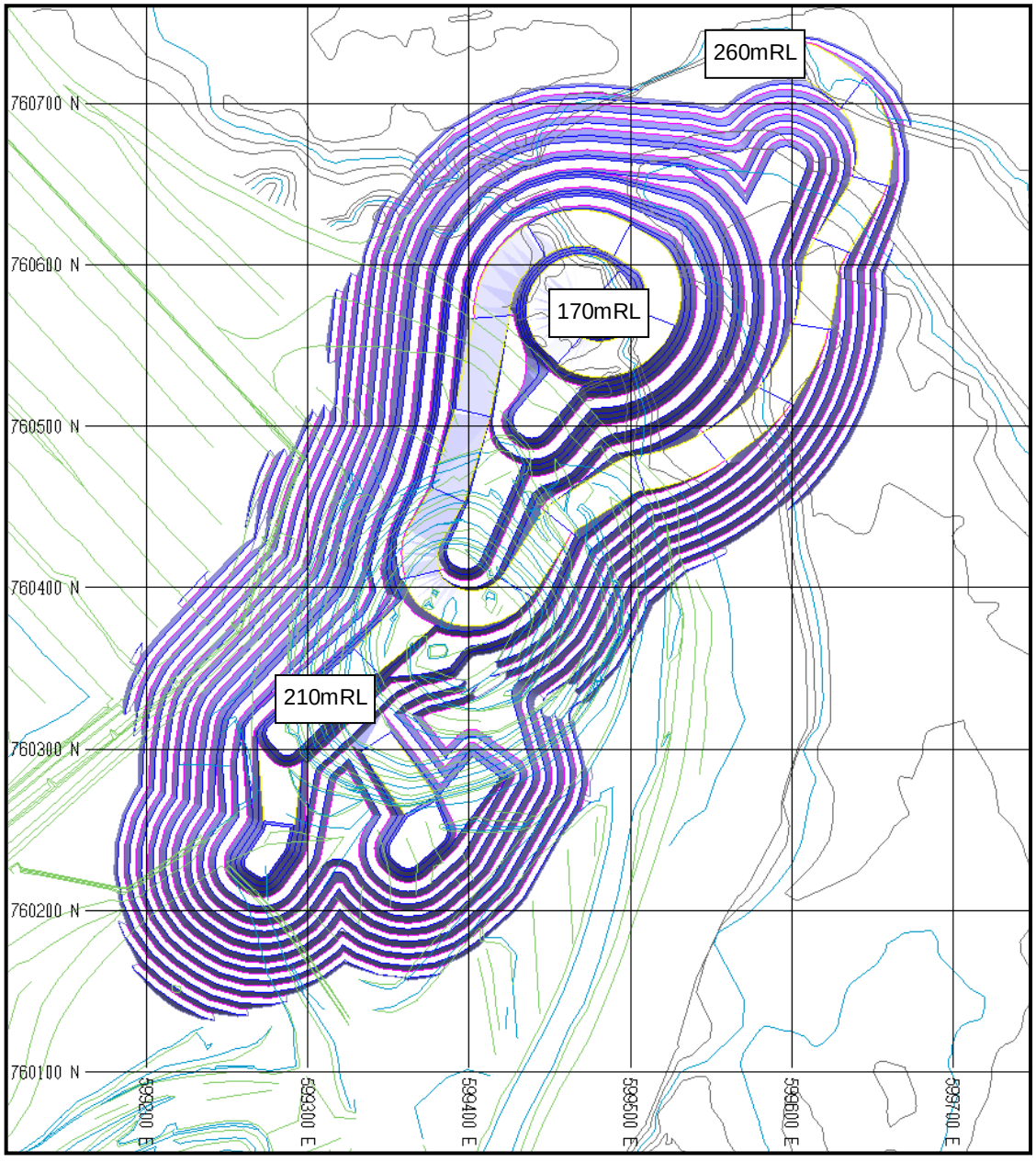
**Figure 15.4.5 Mont Ity Pit Cutback Design**



### 15.4.6 Walter Pit Design

The Walter pit design in Figure 15.4.6 is 90 m deep below the pit exit elevation. The design is approximately 300 m wide and 750 m long at the surface, mining the southeast corner of the currently operating heap leach pad. The ramp exit is to the north in the direction of the proposed Zia NE and Walter waste dump. Cavally river needs to be diverted to be able to mine the northeast of the proposed Walter pit.

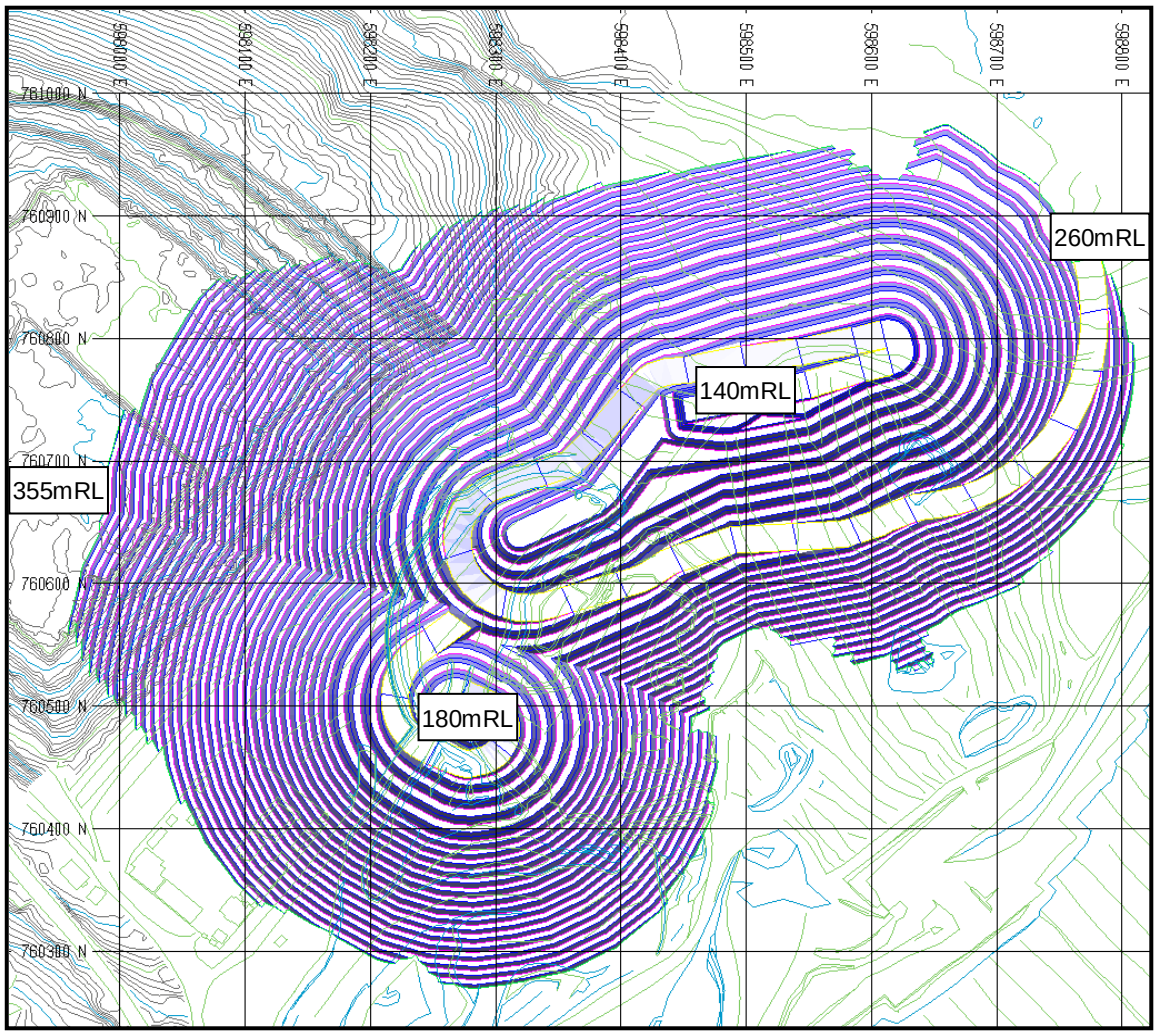
**Figure 15.4.6 Walter Pit Design**



### 15.4.7 Zia NE Pit Design

The Zia NE (North East) pit design n shown in Figure 15.4.7 is 120 m deep below the pit exit location and 215 m deep below the decommissioned Aires heap leach pad. The design mines the south east face of the heap leach pad relatively extensively. The ramp exit is located at the eastern end of the design close to the proposed waste dump for Zia NE and Walter pits.

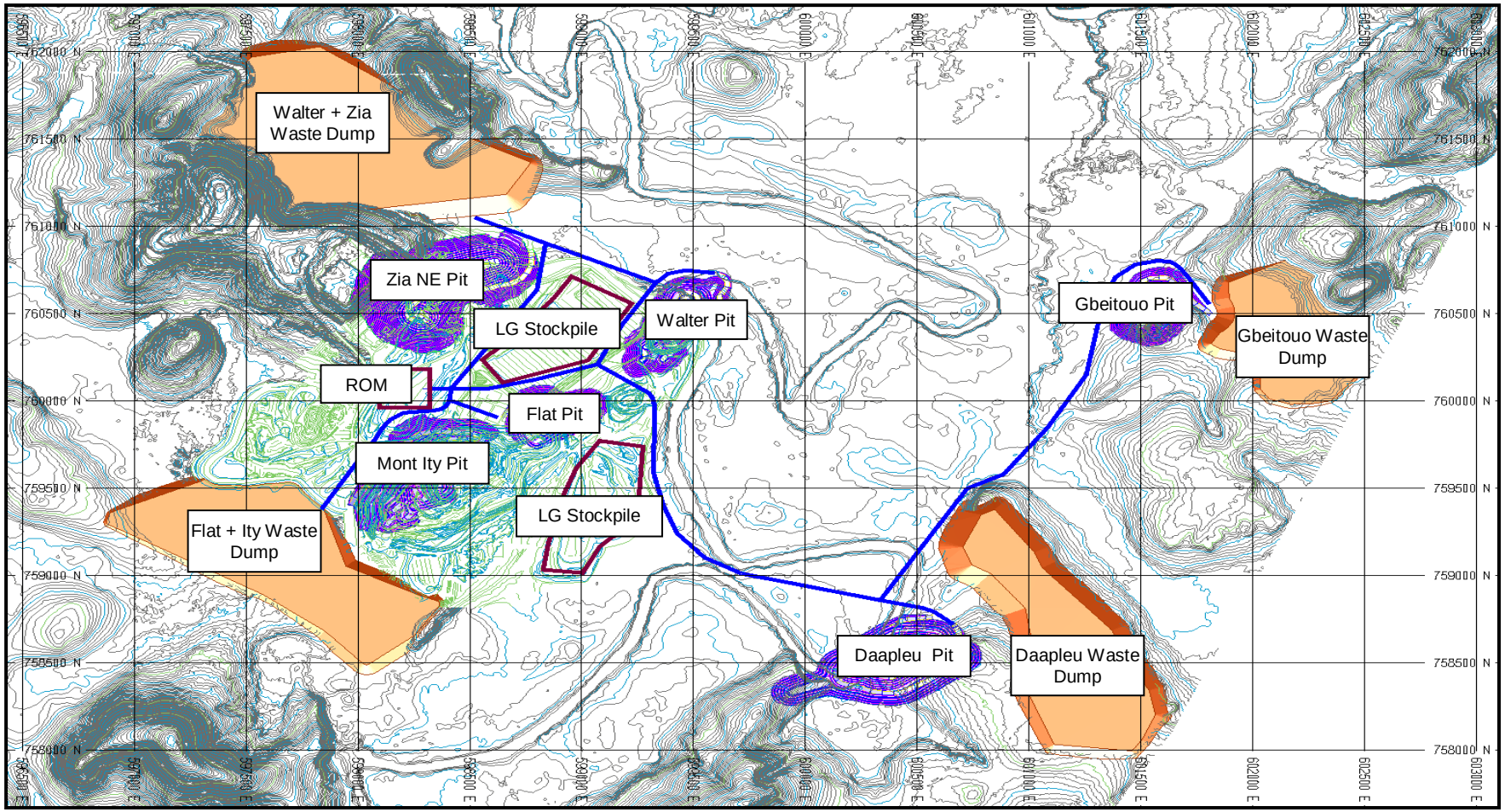
**Figure 15.4.7 Zia NE Pit Design**




### 15.4.8 Waste Dumps and Site Layout

The plan view in Figure 15.4.8 shows the overall site layout including the pit designs, waste dump designs, potential stockpile areas and preliminary mine roads.

**Figure 15.4.8 Waste Dumps and Site Layout**



## 15.5 Mining Quantities and Mineral Reserve Estimate

### 15.5.1 Cut off Grades

The cut-off grades listed in Table 15.5.1 were used in the calculation of the open pit quantities for production schedule and ore reserves. The calculation of the break-even cut-off grades are based on the net revenue (gold price x process recovery) and the throughput (ore related) costs.

In addition to the calculated break-even cut-off grades, the analysis in the preliminary schedules showed that a cut-off grade of 0.9 g/t for the non-sulphide ore and 1.2 g/t for the sulphide ore would be appropriate to classify the high grade ore. Processing the high grade ore selectively and stockpiling the low grade ore for processing at the end of mine life improved the cash flow significantly as discussed in the Section 4.6 above.

**Table 15.5.1 Cut-off Grades for Ore Classification**

| Ore Type | Break-even Cut-off Au g/t | High Grade Cut-off Au g/t |
|---|---|---|
| Daapleu Ore | 0.42 | 0.90 |
| Daapleu Sulphide Ore | 0.72 | 1.20 |
| Gbeitouo Ore | 0.48 | 0.90 |
| Ity Flat Ore | 0.53 | 0.90 |
| Mont Ity Ore | 0.52 | 0.90 |
| Walter Ore | 0.47 | 0.90 |
| Zia NE Ore | 0.44 | 0.90 |

### 15.5.2 Mining Inventory Summary

The mining quantities calculated within the pit designs are summarised in Table 15.5.2. The pit designs are significantly larger, containing 20% more tonnage overall, compared to the optimal pit shells. The overall stripping ratio for the pit designs is 2.7 t/t, which is significantly higher than the 2.2 t/t ratio for the optimal pit shells. These significant changes are due to the inclusion of the ramps and mining width constraints in the pit designs. The cost of the ramps is significant due to the development of six different open pits with separate ramp systems.

These results indicate that the pit designs can be further improved with the iteration of overall slope angles in the pit optimisations. Although the pit designs is likely to be smaller as a result of the iterations, another factor for the change is the 20% lower mining cost in the final cost estimates compared to the preliminary costs used in pit optimisations. The significantly lower costs in the final estimates would result in larger optimal pit limits, partly compensating for the likely decrease in pit size due to the slope iterations.

**Table 15.5.2 Mining Inventory Summary**

| | Total | | Waste | | Total Ore | | | Ore Classification | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | Low Grade (LG) | | High Grade (HG) | | Sulphide LG | | Sulphide HG | |
| | kt | kbcm | kt | kbcm | kt | kbcm | Au g/t | kt | Au g/t | kt | Au g/t | kt | Au g/t | kt | Au g/t |
| **Open Pits** | | | | | | | | | | | | | | | |
| Daapleu 1 | 18,334 | 7,913 | 9,346 | 4,285 | 8,988 | 3,629 | 1.69 | 3,405 | 0.63 | 2,768 | 1.67 | 824 | 0.94 | 1,991 | 3.83 |
| Daapleu 2 | 33,805 | 13,572 | 23,449 | 9,642 | 10,355 | 3,931 | 1.34 | 4,089 | 0.63 | 2,667 | 1.56 | 1,416 | 0.93 | 2,184 | 2.68 |
| Ity Flat | 7,226 | 3,753 | 5,627 | 2,973 | 1,599 | 780 | 2.09 | 540 | 0.69 | 1,059 | 2.80 | | | | |
| Mont Ity | 12,262 | 6,251 | 10,034 | 5,217 | 2,228 | 1,034 | 2.23 | 655 | 0.70 | 1,573 | 2.86 | | | | |
| Gbeitouo | 10,769 | 5,703 | 8,190 | 4,446 | 2,579 | 1,257 | 1.35 | 964 | 0.67 | 1,616 | 1.76 | | | | |
| Walter | 8,882 | 4,751 | 7,025 | 3,757 | 1,858 | 995 | 1.22 | 885 | 0.67 | 973 | 1.72 | | | | |
| Zia NE | 28,687 | 17,160 | 23,852 | 14,315 | 4,835 | 2,846 | 1.24 | 2,655 | 0.63 | 2,180 | 1.98 | | | | |
| **Total Open Pits** | 119,965 | 59,104 | 87,522 | 44,633 | 32,443 | 14,471 | 1.52 | 13,193 | 0.64 | 12,835 | 1.96 | 2,240 | 0.94 | 4,175 | 3.23 |
| Stockpiles | | | | | | | | | | | | | | | |
| Aires Heap Pad | 5,766 | 4,435 | | | 5,766 | 4,435 | 1.09 | | | 5,766 | 1.09 | | | | |
| Teckraie Dump | 2,833 | 2,179 | | | 2,833 | 2,179 | 1.07 | | | 2,833 | 1.07 | | | | |
| **Total Stockpiles** | 8,599 | 6,615 | | | 8,599 | 6,615 | 1.08 | | | 8,599 | 1.08 | | | | |
| **Total Inventory** | **128,564** | **65,719** | **87,522** | **44,633** | **41,042** | **21,085** | **1.42** | **13,193** | **0.64** | **21,434** | **1.61** | **2,240** | **0.94** | **4,175** | **3.23** |

### 15.5.3 Mineral Reserve Estimate

The reserve statement for the Project is provided in Table 15.5.3. All the project reserves are classified in the Probable category based on the underlying resource classification of Indicated resources.

**Table 15.5.3 Mineral Reserve Estimate Effective Date 29 September 2016**

| Resource | Proven Reserve | | | Probable Reserve | | |
|---|---|---|---|---|---|---|
| | Tonnes ('000) | Au g/t | Au ('000oz) | Tonnes ('000) | Au g/t | Au ('000oz) |
| **Open Pits** | | | | | | |
| Daapleu | | | | 19,343 | 1.51 | 936 |
| Ity Flat | | | | 1,599 | 2.09 | 108 |
| Mont Ity | | | | 2,228 | 2.23 | 160 |
| Gbeitouo | | | | 2,579 | 1.35 | 112 |
| Walter | | | | 1,858 | 1.22 | 73 |
| Zia NE | | | | 4,835 | 1.24 | 192 |
| Total Open Pits | | | | 32,443 | 1.52 | 1,580 |
| **Existing Stockpiles** | | | | | | |
| Aires Leach Pad | | | | 5,766 | 1.09 | 202 |
| Teckraie Dump | | | | 2,833 | 1.07 | 97 |
| Total Stockpiles | | | | 8,599 | 1.08 | 300 |
| **Total** | | | | **41,042** | **1.42** | **1,880** |

Notes:

- The mineral reserve estimation approach is provided in Section 15.1.
- The basis of the estimate and key assumptions are provided in Section 15.2.
- The reserves are calculated within the pit designs reported in Section 15.4.
- The cut-off grades for the reserves are provided in Table 15.5.1. The mineral resource grade estimates include dilution for the selected SMU size.
- The mineral resource estimates in Section 14 of this report are inclusive of (not in addition to) mineral reserves reported here.
- The Inferred resources in the pit designs are classified as waste. The stripping ratio for the open pit reserves is 2.7 t:t. The overall stripping ratio, including the existing stockpiles, is 2.1 t:t.

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# 16.0 MINING METHODS

## 16.1 Mining Method

The current Ity heap leach mine is a conventional owner mining open pit operation employing articulated trucks (40 t) and hydraulic back hoe excavators (80 t). Currently limited drilling and blasting activities are required at the site as the material mined is mostly oxidised (clays and laterites). This will change in the CIL operation as more fresh material will be mined from the deeper open pits, especially from the deepest Daapleu pit with limited oxidised zone. The mining equipment will also change as the 90 t capacity dump trucks and 120 t class backhoe excavators are selected for CIL operations to sustain the required production rates.

The selected mining method for the Project is conventional open pit excavator-truck operation with the production unit operations (drilling, blasting, loading, hauling and dumping) carried out by owner mining personnel and equipment. Ore and waste production rates will be monitored and material reconciliation will be carried out continuously for the pit areas in production. Mining plans will be updated and production schedule revisions will be issued regularly as the mining progresses.

The pit areas will be cleared of shrubs and trees and these will be dumped to the specified locations. The topsoil from the open pit and waste dump areas will be stockpiled to the designated areas to use later for rehabilitation. The area preparation cost estimates in the feasibility study cost model include both the clearing and topsoil removal operations.

The production drilling and blasting operations will be carried out at 5 m benches. A half bench height (flitch) of 2.5 m will be mined in ore and waste to achieve a high degree of selectivity in loading and hauling operations.

The average blasting powder factor was assumed to be 0.60 kg/bcm (ANFO equivalent) for the transitional and fresh rock with a density above 2.0 t/bcm. The highly weathered zone (clays and laterites) and transitional zone with a density below 2.0 t/bcm will amenable to free digging as per the current mining practice at the site. It is assumed that emulsion will be used in both wet and dry blasting for efficiency.

It is expected that Endeavour will contract the supply of explosives and blasting accessories directly to an approved explosives supplier. The explosives contractor will provide, in addition to the supply of primary explosives and blasting accessories, mixing equipment and technical blasting advice when needed.

Grade control drilling will be carried out by a drilling contractor and the samples will be tested in the onsite laboratory. Sampling will commence with grade control drilling ahead of the mining front, aimed at assisting the short to medium term mine planning process. The grade control will be based on 138 mm diameter RC drilling and sampling practice. A grade control pattern of 5 m x 10 m is suggested for 20 m deep holes and 2.0 m vertical sampling intervals. The holes will be angled 60 degrees from the hanging wall side of the ore zones to provide a good intersection with the mineralised structures.

Environmental monitoring of air and water quality will be conducted on a regular basis. Potentially environmentally hazardous artificial materials, such as chemicals and rubbish will be disposed following appropriate standard procedures. If present, any rocks mined with acid drainage potential will be dumped inside the waste dumps and covered with the neutral oxide material also mined from the open pits.

Dust from the pits, dumps, stockpile operations and road surfaces will be controlled continuously by water suppression.

The final faces of the waste dumps will be dozed to the specified slope angles and contoured for rehabilitation. The rehabilitation of the waste dumps will be carried out progressively as the mining of the open pits is completed.

## 16.2 Hydrogeology

This section outlines the development of a mine dewatering plan for the six open pits (Daapleu, Mont Ity, Ity Flat, Walter, Zia NE and Gbeitouo) at the Project. The dewatering plan is based on a numerical groundwater flow model developed by Knight Piésold (Ref 1) that has been used to make groundwater inflow predictions.

### 16.2.1 Previous Investigations

Groundwater investigations were undertaken by SNC-Lavalin as part of a broader detailed feasibility study (FS) include a site wide hydrogeological field investigation report (Ref 2) and a groundwater inflow report for the Daapleu, Zia-NE and Walter pits (Ref 3).

The site wide hydrogeological investigation (Ref 3) describes the site geology and the hydrogeological field tests undertaken. Most importantly five constant rate pumping tests (two tests in the Daapleu, Zia-NE and one at the Walter pit) were undertaken that provided useful data. Falling head test and packer tests were also undertaken. Pit inflow predictions and the associated dewatering bore recommendations contained within the groundwater inflow report (Ref 3) are summarised in Table 16.2.1.

**Table 16.2.1 Ity Pits Groundwater Inflow Predictions Bores Summary (SNC-Lavalin, Ref 3)**

| Pit | Pit Life (Years) | Predicted Inflows ($m^3/h$) | Recommended Number of Bores |
|---|---|---|---|
| Daapleu | 8 | • Rainy season: 78 to 1,621<br>• Dry season: 13 to 427<br>• Years 1 to 8 | • Years 1 to 2: 8 to16 shallow bores<br>• Years 3 to 8: Flat pit area, 1 to 3 bores<br>• Years 4 to 9: Major Pit Footwall, 9 to 14 bores |
| Walter | 1 (dry season only) | • 386 (Dry season) | • 12 shallow bores |
| Zia-NE | 4 | • Rainy season: 23 to 162<br>• Dry season: 11 to 94<br>• Years 3 and 4 | • Year 4: Up to 6 bores |

The SNC-Lavalin (Ref 3) inflow predictions are summarised as follows:

- The predicted inflows into Daapleu are very high and up to 30 dewatering bores were predicted to be required.
- The predicted Walter pit inflows are also high (although over only one season) with 12 bores estimated to be required.
- The Zia-NE was predicted to be not affected by the Cavally River and as a result the predicted inflows are less but are still significant with up to six dewatering bores predicted to be required.

The SNC-Lavalin report (Ref 3) predicted pit inflows were generated using Darcy's law in which the radius of influence was held constant. This was considered to be an over simplistic approach for this level of study. A numerical groundwater model was therefore developed, as summarised below.

### 16.2.2 Groundwater Flow Model

Knight Piésold (Ref 1) developed a conceptual site model (CSM) using the Ity Pit groundwater abstraction / level data and the fieldwork undertaken by SNC-Lavalin (Refs 2 and 3). The CSM was then used as a basis for the development of a three dimensional numerical groundwater flow model. The model was calibrated against historical dewatering rates and groundwater levels in the existing Ity Pit for the period 2013 to 2016. After calibration, pit shells for the Daapleu, Mont Ity, Ity Flat, Walter, Zia NE and Gbeitouo pits (six pits) were entered into the model which was then run in predictive mode to estimate groundwater inflow rates to each pit with time.

The groundwater model was developed using the MODFLOW code (Ref 4) and its latest update MODFLOW-NWT (Ref 5). The latest version of the graphical interface Visual MODFLOW (Waterloo Hydrogeologic) was used as a pre/post processor to prepare the input data and evaluate the outputs. The model was initially calibrated against the field data by trial and error and then by the inverse modelling code PEST (Ref 6). MODFLOW-NWT (Ref 5) and the most recent version of MODFLOW – SURFACT (Waterloo Hydrogeologic) were used to run the predictive scenarios.

Model calibration was performed by selecting appropriate aquifer hydraulic properties and stresses to generate model outputs resembling as closely as possible equivalent field measured values. The steady state calibration gave a Normalised RMS error of 9.5 per cent which is within the commonly accepted NRMS error of 10 per cent (Ref 7). The simulated hydraulic conductivity values are generally consistent with the range and distribution of values produced from field testing. By using the groundwater level variations from 2013 to 2016 the transient model calibration indicated specific storage (Ss) and specific yield (Sy) values of 1E-7 and 2.8 per cent, respectively. These values are consistent with the range of values produced during field pumping tests.

### 16.2.3 Groundwater Model Predictions

The predicted monthly average groundwater inflow rates to each pit are summarised in Table 16.2.2. Inflows range from 20 m³/h in the Gbeitouo Pit to 224 m³/h in the Mont Ity Pit whilst the absolute maximum inflow rates range from 116 m³/h in the Ity Flat Pit to 287 m³/h in the Daapleu Pit (considerably lower than the SNC-Lavalin predictions).

The following assumptions were made:

- For the dewatering plan the target groundwater level was set at five metres below the minimum pit elevation.
- Since the Cavally River intersects the Daapleu and Walter pit shells it was assumed the river will be diverted.
- The predicted inflows are from groundwater flow only and do not include direct rainfall or rainfall runoff which are likely to be significant in the wet season.

**Table 16.2.2 Monthly Average Pit Inflow Rates (m³/h)**

| Month | Daapleu | Gbeitouo | Ity Flat | Mont Ity | Walter | Zia NE |
|---|---|---|---|---|---|---|
| 1 | 175.8 | 90.5 | 80.3 | 141.5 | 106.2 | 45.8 |
| 2 | 196.6 | 105.5 | 47.9 | 206.7 | 116.8 | 92.6 |
| 3 | 170.4 | 81.7 | 58.0 | 119.2 | 113.0 | 82.3 |
| 4 | 153.1 | 86.8 | 19.7 | 148.8 | 111.9 | 69.1 |
| 5 | 148.4 | 32.6 | 40.2 | 159.2 | 80.6 | 88.2 |
| 6 | 171.0 | 20.1 | 53.5 | 201.5 | 82.2 | 88.0 |
| 7 | 174.0 | 36.1 | 64.1 | 185.8 | 130.1 | 88.4 |
| 8 | 201.9 | 28.1 | 53.6 | 182.2 | 158.8 | 85.6 |
| 9 | 210.8 | 29.4 | 81.8 | 176.4 | 165.4 | 83.9 |
| 10 | 197.2 | 59.9 | 84.0 | 224.3 | 171.2 | 90.4 |
| 11 | 165.6 | 42.3 | 90.8 | 195.3 | 162.7 | 94.0 |
| 12 | 164.2 | 53.2 | 40.7 | 195.9 | 107.7 | 86.4 |
| Monthly min | 148.4 | 20.1 | 19.7 | 119.2 | 80.6 | 45.8 |
| Monthly max | 210.8 | 105.5 | 90.8 | 224.3 | 171.2 | 94.0 |
| Absolute max | 287.3 | 127.3 | 116.1 | 286.8 | 185.8 | 236.8 |

### 16.2.4 Dewatering Plan

The estimated number of dewatering bores required for each pit is summarised in Table 16.2.3. The bore numbers are based on a median yield of 11 m³/h obtained from the existing Ity Pit dewatering bores. The number of dewatering bores could be reduced by the installation of higher yielding bores.

**Table 16.2.3 Estimated Number of Dewatering Bores**

| Pit | Max Dewatering Rate ($m^3/h$) | Assumed Median Bore Yield ($m^3/h$) | Number of Dewatering Bores |
|---|---|---|---|
| Daapleu | 287.3 | 11 | 25 |
| Gbeitouo | 127.3 | 11 | 12 |
| Ity Flat | 116.1 | 11 | 11 |
| Mont Ity | 286.5 | 11 | 26 (13)* |
| Walter | 185.8 | 11 | 17 |
| Zia NE | 236.8 | 11 | 22 |
| **Total Bores** | | | **113** |
| **Total New Bores*** | | | **100** |

***The southern part of the Ity Pit is already equipped with dewatering bores. Thus the required number of bores is reduced by 13.**

A bore installation schedule based on the mining plan is provided in Table 16.2.4. A three month lead-in time should be incorporated to allow the required dewatering level targets to be attained ahead of mining.

**Table 16.2.4 Ity Pits Dewatering Bore Installation Schedule**

| | Number of Dewatering Bores | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| **Pit Name** | **Year 0** | **Year 1** | **Year 2** | **Year 3** | **Year 4** | **Year 5** | **Year 6** | **Year 7** | **Total** |
| Daapleu | 17 | 8 | - | - | - | - | - | - | 25 |
| Gbeitouo | - | - | - | 9 | 3 | - | - | - | 12 |
| Ity Flat | 8 | 3 | - | - | - | - | - | - | 11 |
| Mont Ity | - | 6 | 7 | - | - | - | - | - | 13 |
| Walter | - | - | - | - | 17 | - | - | - | 17 |
| Zia NE | - | - | - | - | 3 | - | 14 | 5 | 22 |
| **Total** | **25** | **17** | **7** | **9** | **23** | | **14** | **5** | **100** |

### 16.2.5 Conclusions

The following conclusions are made regarding the dewatering of the Ity open pits:

- The predicted monthly average groundwater inflows range from 20 $m^3/h$ in the Gbeitouo Pit to 224 $m^3/h$ in the Mont Ity Pit whilst the absolute maximum inflow rates range from 116 $m^3/h$ in the Ity Flat Pit to 287 $m^3/h$ in the Daapleu Pit.

- An additional 25 bores are required by Year 0 and a further 17 bores are required in Year 1. For efficient dewatering of the pits the installation of 100 new bores, over the mining period of eight years is predicted although this number may reduce if higher yielding bores can be installed.

- The pit inflows predicted by the Knight Piésold (Ref 3) are significantly less than those previously predicted by SNC-Lavalin (Ref 2) probably due to the fixed head adopted in the SNC-Lavalin (Ref 2) model which is considered to be unrealistic.

- Conversely Knight Piésold (Ref 3) predict the need for a larger number of bores than SNC-Lavalin (Ref 2) due to there now being six instead of three pits and also due to what is considered to be an unrealistically high yield per bore assumed by SNC-Lavalin (Ref 2).

- A more detailed analysis of the existing geological data is recommended for the refinement and improvement of the proposed bore locations and pit inflow predictions. Potential sources of additional information include the existing block geological model, additional borehole logs and geophysical data.

- Additional field investigations and testing should take place in a phased manner and successful bores could be adopted as long term dewatering bores.

- The groundwater model should be updated as additional investigation and monitoring data are obtained.

***REFERENCES***

1. Knight Piésold (2016) Technical Memo. Ity Gold Project – Groundwater Model For Pit Dewatering. KP Ref PE16-00842.

2. SNC Lavalin (August 31 2015a) Feasibility Study Site Wide Hydrogeological field Investigation Factual report. FS Appendix AB. Ref 628121-0000-4WER-001.

3. SNC Lavalin (November 30 2015b) Feasibility Study Pit Dewatering Assessment Report. FS Appendix Q. Ref 628121-0000-4WER-0002.

4. Harbaugh, A.W., 2005, MODFLOW-2005, the U.S. Geological Survey modular ground-water model -- the Ground-Water Flow Process: U.S. Geological Survey Techniques and Methods 6-A16.

5. Niswonger, R.G., Panday, Sorab, and Ibaraki, Motomu, 2011, MODFLOW-NWT, A Newton formulation for MODFLOW-2005: U.S. Geological Survey Techniques and Methods 6–A37, 44 p.

6. Doherty, J., 2015. Calibration and Uncertainty Analysis for Complex Environmental Models. Watermark Numerical Computing, Brisbane, Australia. ISBN: 978-0-9943786-0-6.

7. Barnett et al (2012).Australian Groundwater Modelling Guidelines, Waterlines Report, National Water Commission, Canberra.

## 16.3 Geotechnical Evaluation

Slope design parameters were recommended by Peter O'Bryan & Associates, independent geotechnical consultants engaged by Endeavour.

Ground conditions influencing wall stability in proposed open pit mining at Endeavour Mining Corporation's (Endeavour) Daapleu, Gbeitouo, Zia NE, Mont Ity, Ity Flat and Walter deposits have been investigated using:

- current geological interpretations
- inspection of the Ity site and review of existing open pit excavations
- inspection of selected geotechnical investigation cores
- review of geotechnical data and geotechnical assessments contained in a report compiled by SNC Lavalin (SNC) in 2015
- geotechnical and structural geological data obtained by others from geotechnical investigation cores
- laboratory measurement of physical properties of representative samples of deposit rocks
- experience in geotechnical assessment and review in similar geological and geotechnical settings.

Assessment and analysis of future open pit wall stability has used:

- current interpretations of geological and geotechnical conditions
- structural geological assessment
- results of laboratory testing of physical properties of rocks in which future pit walls will be developed
- kinematic stability analysis
- limit equilibrium analysis
- experience-based assessment of expected pit wall conditions.

The Geotechnical Design Domains are provided in Figure 16.4.1 to Figure 16.3.4, with associated wall design criteria provided in Table 16.3.1 to Table 16.3.6.

### 16.3.1 Recommended Open Pit Preliminary Base Case Wall Design Parameters

The Geotechnical Design Domains are provided in Figure 16.4.1 to Figure 16.3.4, with associated wall design criteria provided in Table 16.3.1 to Table 16.3.6.

As noted, the recommended wall design parameters are predicated on achievement of essentially drained / depressurised wall rock conditions. It is recommended that Endeavour independently confirms that achievement of adequate depressurisation at Ity is practicable.

Achievement of stable pit walls based on the above parameters will depend highly on consistent successful application of appropriate drill and blast excavation methods in wall development. It is essential that production blasting does not unduly disturb or pre-condition wall rocks and that appropriate perimeter blasting methods are applied in all wall development blasts.

Use of quantitative slope displacement monitoring is recommended for all Ity pits. The wall design parameters are recommended with an expectation that initial mining will allow the use of observational techniques to refine slope parameters for final walls. That is, assessment of staged / interim slopes will permit confirmation and/ or amendment of the design parameters.

***Daapleu***

A summary of Daapleu base case wall design parameters is shown in Table 16.5.1.

**Table 16.3.1 Daapleu Base Case Wall Design Parameters**

| From (mbs) | To (mbs) | Material Type | Face Height (m) | Face Angle (°) | Berm Width (m) | IRA (°) |
|---|---|---|---|---|---|---|
| Natural surface | 5 | Alluvials/ laterites | 5 | 50 | 3 | 34.8 |
| 5 | 25 | Weathered material / rocks | 10 | 50 | 5 | 36.7 |
| 25 | 45 | Transitional rocks | 20 | 60 | 7 | 47.2 |
| 45 | 205 | Fresh rocks* | 20 | 75 | 7 | 58.3 |

***All Walls***

*Batter Face Height*

5 m Natural surface to 5 metres below surface (mbs) (allowance for alluvials and laterites)

10 m 5 mbs to 25 mbs (allowance for weathered material / rocks)

20 m 25 mbs to 205 mbs (planned base of mining)

*Batter Face Angle*

50° Natural surface to 25 mbs (alluvials, laterites and weathered material / rocks)

60° 25 mbs to 45 mbs (transitional rocks)

75° 45 mbs to 205 mbs (fresh rocks*)

*Berm Width*

3.0 m At 5 mbs

5.0 m At 15 mbs and 25 mbs

7.0 m ≤ 20 m vertical intervals between 25 mbs and 205 mbs

*Bench Stack*

≤ 80 m 4 × 20 m high batters (where wall not segmented by a ramp pass)
Stack Berm width ≥ 14 m

*Inter Ramp Angle (IRA)*

5 m at 50° with 3.0 m berm = 34.8°

10 m at 50° with 5.0 m berm = 36.7°

20 m at 60° with 7.0 m berm = 47.2°

20 m at 75° with 7.0 m berm = 58.3°

*Overall Slope Angles (OSA)*

49.9° Natural surface to 205 mbs (2 × stack berms and no ramp pass)

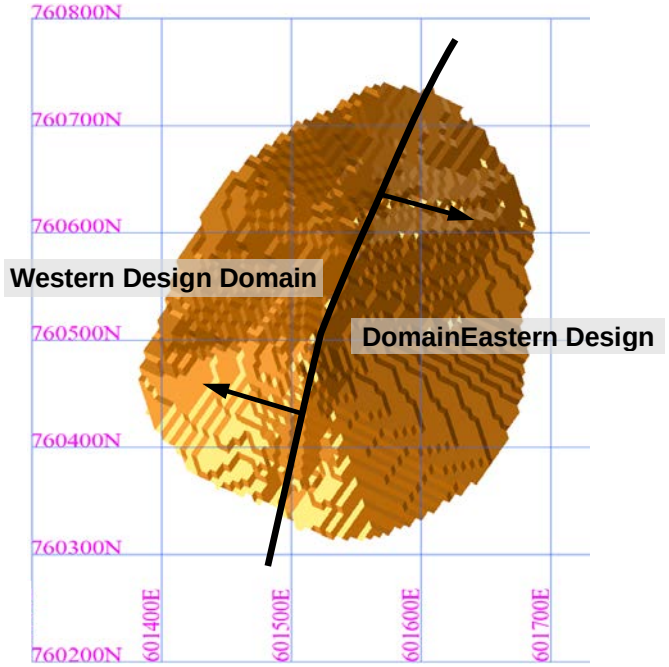
***Gbeitouo***

A summary of Gbeitouo base case wall design parameters is shown in Table 16.3.2.

**Table 16.3.2 Gbeitouo Base Case Wall Design Parameters**

| Domain | From | To | Material Type | Face Height (m) | Face Angle (°) | Berm Width (m) | IRA (°) |
|---|---|---|---|---|---|---|---|
| All | Natural surface | Base of soil / overburden | Soil, overburden & laterite | 4 | 50 | 4 | 28.5 |
| | Base of soil / overburden | BOCO / Base of reduced clays | Saprolite, reduced clay and highly weathered rock | 5 | 50 | 3.5 | 33.0 |
| | BOCO/ base of reduced clays | TOFR | Transitional material | 10 | 55 | 5.5 | 38.7 |
| Western | TOFR | Base of model | Fresh rock | 15 | 70 | 6 | 52.6 |
| Eastern | TOFR | Base of model | Fresh rock | 15 | 55 | 6 | 42.3 |

**Figure 16.3.1 Gbeitouo Geotechnical Domains / Design Sectors**




*Batter Face Height*

| | |
|---|---|
| 4 m | Natural surface to base of soil, overburden and laterite material (indicated to be up to 9 mbs within some pit sectors) |
| 5 m | Saprolite, reduced clay and highly weathered rock |
| 10 m | Transitional material (material between base of complete oxidation - BOCO and top of fresh rock - TOFR) |
| 15 m | Fresh rock |

*Batter Face Angle*

| | |
|---|---|
| 50° | Natural surface to BOCO and within reduced clays |
| 55° | Transitional material |
| 70° | Western Domain Fresh rock |
| 55° | Eastern Domain Fresh rock (accounts for inferred rock structure and deep weathering within eastern domain) |

*Berm Width*

| | |
|---|---|
| 4.0 m | Below 4 m high batter faces |
| 3.5 m | Below 5 m high batter faces |
| 5.5 m | Below 10 m high batter faces |
| 6.0 m | Below 15 m high batter faces for Western and Eastern Domain |

*IRA*

4 m at 50° with 4.0 m berm = 28.5°

5 m at 50° with 3.5 m berm = 33.0°

10 m at 55° with 5.5 m berm = 38.7°

15 m at 70° with 6.0 m berm = 52.6°

15 m at 55° with 6.0 m berm = 42.3°

***Zia NE***

A summary of Zie NE base case wall design parameters is shown in Table 16.3.3.

**Table 16.3.3 Zia NE Base Case Wall Design Parameters**

| From (mbs) | To (mbs) | Material Type | Face Height (m) | Face Angle (°) | Berm Width (m) | IRA (°) |
|---|---|---|---|---|---|---|
| Natural surface | Base of soil, overburden& heap leach material | Soil, laterite & heap leach | 4 | 50 | 4 | 28.5 |
| Base of soil, overburden & heap leach material | BOCO | Saprolite, reduced clay & highly weathered rocks | 5 | 50 | 3.5 | 33.0 |
| BOCO | TOFR | Transitional | 10 | 55 | 5.5 | 38.7 |
| TOFR | Base of model | Fresh rocks | 15 | 70 | 6 | 52.6 |

**All Walls**

*Batter Face Height*

4 m Natural surface to base of soil, overburden, laterite and historic heap leach material

5 m Saprolite, reduced clay and highly weathered rock

10 m Transitional material (material between BOCO and TOFR)

15 m Fresh rock

*Batter Face Angle*

50° Natural surface to BOCO and within reduced clays

55° Transitional material

70° Fresh rock

*Berm Width*

4.0 m Below 4 m high batter faces

3.5 m Below 5 m high batter faces

5.5 m Below 10 m high batter faces

6.0 m Below 15 m high batter faces

*IRA*

4 m at 50° with 4.0 m berm = 28.5°

5 m at 50° with 3.5 m berm = 33.0°

10 m at 55° with 5.5 m berm = 38.7°

15 m at 70° with 6.0 m berm = 52.6°

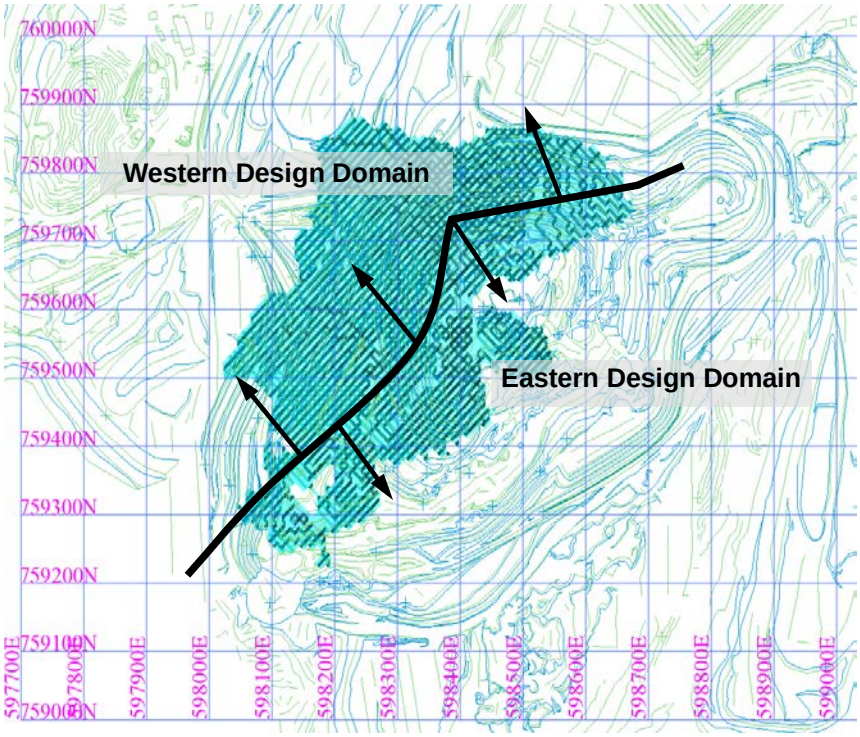
***Mont Ity***

A summary of Mont Ity base case wall design parameters is shown in Table 16.3.4.

**Table 16.3.4 Mont Ity Base Case Wall Design Parameters**

| Domain | From | To | Material Type | Face Height (m) | Face Angle (°) | Berm Width (m) | IRA (°) |
|---|---|---|---|---|---|---|---|
| All | Natural surface | Base of soil / overburden | Soil, overburden & laterite | 4 | 50 | 4 | 28.5 |
| | Base of soil / overburden | BOCO / Base of reduced clays | Saprolite, reduced clay and highly weathered rock | 5 | 50 | 3.5 | 33.0 |
| | BOCO / base of reduced clays | TOFR | Transitional material | 10 | 55 | 5.5 | 38.7 |
| Western | TOFR | Base of model | Fresh rock | 15 | 70 | 6 | 52.6 |
| Eastern | TOFR | Base of model | Fresh rock | 15 | 65 | 8 | 45.0 |

**Figure 16.3.2 Mont Ity Geotechnical Domains / Design Sectors**



*Batter Face Height*

| | |
|---|---|
| 4 m | Natural surface to base of soil, overburden and laterite material (indicated to generally be less than 8 m depth) |
| 5 m | Saprolite, reduced clay and highly weathered rock |
| 10 m | Transitional material (material between BOCO and TOFR) |
| 15 m | Fresh rock |

*Batter Face Angle*

| | |
|---|---|
| 50° | Natural surface to interpreted BOCO surface |
| 55° | Transitional material |
| 70° | Western Domain Fresh rock |
| 65° | Eastern Domain Fresh rock (Eastern Domain design IRA approximates deposit stratigraphy dip) |

*Berm Width*

| | |
|---|---|
| 4.0 m | Below 4 m high batter faces |
| 3.5 m | Below 5 m high batter faces |
| 5.5 m | Below 10 m high batter faces |
| 6.0 m | Below 15 m high batter faces Western Domain fresh rock |
| 8.0 m | Below 15 m high batter faces Eastern Domain fresh rock |

*IRA*

4 m at 50° with 4.0 m berm = 28.5°

5 m at 50° with 3.5 m berm = 33.0°

10 m at 55° with 5.5 m berm = 38.7°

15 m at 70° with 6.0 m berm = 52.6°

15 m at 65° with 8.0 m berm = 45.0°

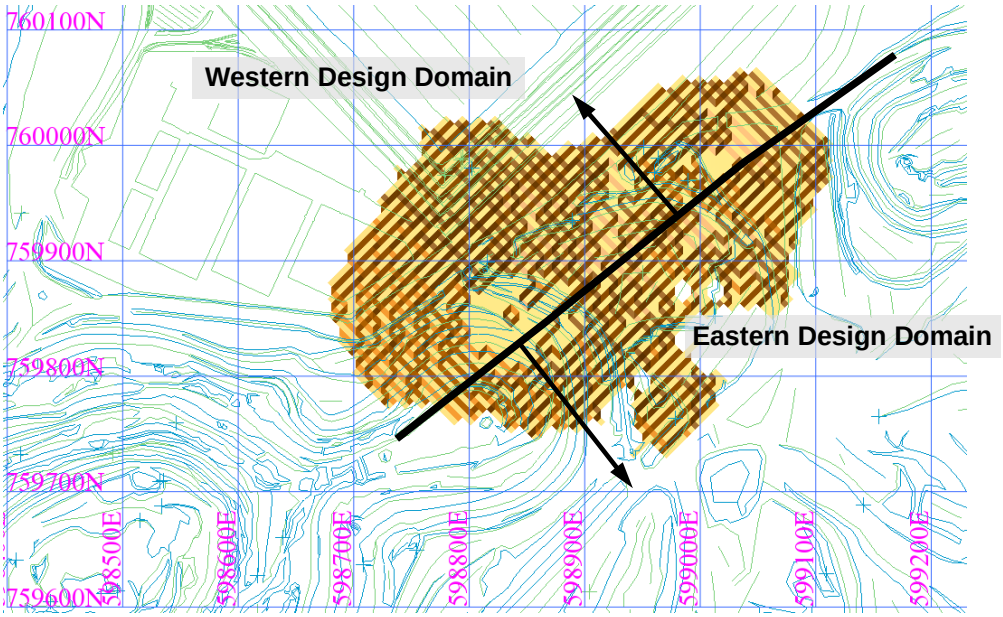
***Ity Flat***

A summary of Ity Flat base case wall design parameters is shown in Table 16.3.5.

**Table 16.3.5 Ity Flat Base Case Wall Design Parameters**

| Domain | From | To | Material Type | Face Height (m) | Face Angle (°) | Berm Width (m) | IRA (°) |
|---|---|---|---|---|---|---|---|
| All | Natural surface | Base of soil, laterite & heap leach material | Soil, overburden, laterite and heap leach | 4 | 50 | 4 | 28.5 |
| | Base of soil, laterite & heap leach material | BOCO | Saprolite, reduced clay & highly weathered rock | 5 | 50 | 3.5 | 33.0 |
| | BOCO | TOFR | Transitional | 10 | 55 | 5.5 | 38.7 |
| Western | TOFR | Base of model | Fresh rocks | 15 | 70 | 6 | 52.6 |
| Eastern | TOFR | Base of model | Fresh rocks | 15 | 55 | 6 | 42.3 |

**Figure 16.3.3 Ity Flat Geotechnical Domains / Design Sectors**




*Batter Face Height*

| | |
|---|---|
| 4 m | Natural surface to base of soil, overburden, laterite and historic heap leach material (indicated to be up to 20 m depth) |
| 5 m | Saprolite, reduced clay and highly weathered rock |
| 10 m | Transitional material (material between BOCO and TOFR) |
| 15 m | Fresh rock |

*Batter Face Angle*

| | |
|---|---|
| 50° | Natural surface to BOCO and within reduced clays |
| 55° | Transitional material |
| 70° | Western Domain Fresh rock |
| 55° | Eastern Domain Fresh rock (Eastern Domain design IRA approximates deposit stratigraphy dip) |

*Berm Width*

| | |
|---|---|
| 4.0 m | Below 4 m high batter faces |
| 3.5 m | Below 5 m high batter faces |
| 5.5 m | Below 10 m high batter faces |
| 6.0 m | Below 15 m high batter faces for Western and Eastern Domain fresh rocks |

*IRA*

4 m at 50° with 4.0 m berm = 28.5°

5 m at 50° with 3.5 m berm = 33.0°

10 m at 55° with 5.5 m berm = 38.7°

15 m at 70° with 6.0 m berm = 52.6°

15 m at 55° with 6.0 m berm = 42.3°

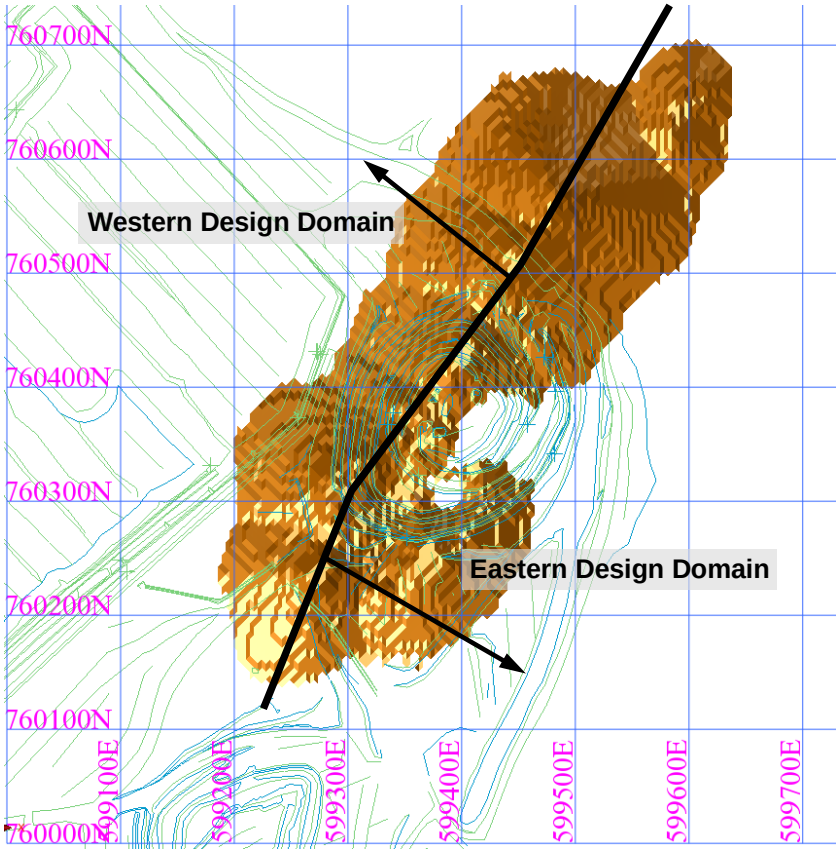
***Walter***

A summary of Walter base case wall design parameters is shown in Table 16.3.6.

**Table 16.3.6 Walter Base Case Wall Design Parameters**

| Domain | From | To | Material Type | Face Height (m) | Face Angle (°) | Berm Width (m) | IRA (°) |
|---|---|---|---|---|---|---|---|
| All | Natural surface | Base of soil, laterite & heap leach material | Soil, overburden, laterite and heap leach | 4 | 50 | 4 | 28.5 |
| | Base of soil, laterite & heap leach material | BOCO | Saprolite, reduced clay & highly weathered rock | 5 | 50 | 3.5 | 33.0 |
| | BOCO | TOFR | Transitional | 10 | 55 | 5.5 | 38.7 |
| Western | TOFR | Base of model | Fresh rocks | 15 | 70 | 6 | 52.6 |
| Eastern | TOFR | Base of model | Fresh rocks | 15 | 55 | 6 | 42.3 |

**Figure 16.3.4 Walter Geotechnical Domains / Design Sectors**



*Batter Face Height*

| | |
|---|---|
| 4 m | Natural surface to base of soil, overburden, laterite and historic heap leach material |
| 5 m | Saprolite, reduced clay and highly weathered rock |
| 10 m | Transitional material (material between BOCO and TOFR) |
| 15 m | Fresh rock |

*Batter Face Angle*

| | |
|---|---|
| 50° | Natural surface to BOCO and within reduced clays |
| 55° | Transitional material |
| 70° | Western Domain Fresh rock |
| 55° | Eastern Domain Fresh rock (Eastern Domain design IRA approximates deposit stratigraphy dip) |

*Berm Width*

| | |
|---|---|
| 4.0 m | Below 4 m high batter faces |
| 3.5 m | Below 5 m high batter faces |
| 5.5 m | Below 10 m high batter faces |
| 6.0 m | Below 15 m high batter faces for Western and Eastern Domain fresh rocks |

*IRA*

4 m at 50° with 4.0 m berm = 28.5°

5 m at 50° with 3.5 m berm = 33.0°

10 m at 55° with 5.5 m berm = 38.7°

15 m at 70° with 6.0 m berm = 52.6°

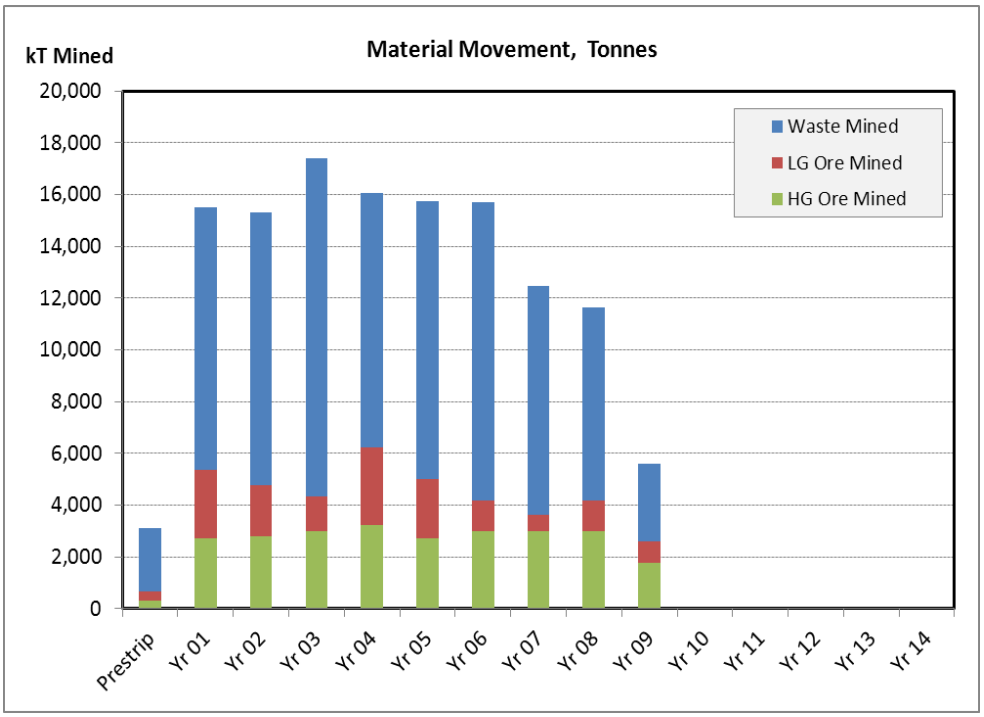
15 m at 55° with 6.0 m berm = 42.3°

## 16.4 Production Schedule

The details provided in the production schedule were considered sufficient to highlight any potential operational difficulties throughout the mine life as well as to provide a basis for the long-term cash flow analysis.

Annual summary of the production schedule (v2c) is provided in Table 16.4.1. The mined tonnages in the schedule (Figure 16.6.1) include the quantities reclaimed from the existing stockpiles to supplement the production from the open pits. The peak production in Year 3 include 1.4 Mt of mill feed sourced from the existing stockpiles. The average open pit production rate is 15.5 Mtpa for the six years of the mine operations. The pit production tails off in the last three Years of mine life as the Zia NE pit developed last in the production schedule.

**Figure 16.4.1 Yearly Total Tonnage Mined**




The chart for the yearly process feed tonnages in Figure 16.4.2 shows that the total feed is 2.8 Mt in the first year due to the ramping up of the plant production at the start-up. The last five years of the process feed is from the low grade stockpiles after the mining of the open pits are completed in nine years.

**Figure 16.4.2 Yearly Process Feed Tonnes**

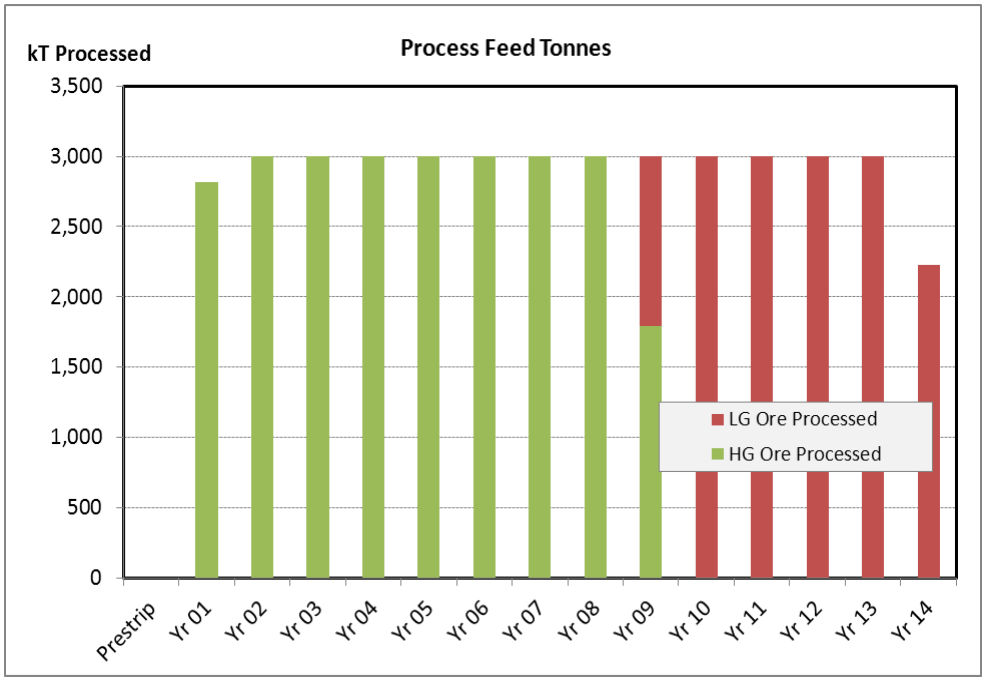


**Table 16.4.1 Production Schedule Annual Summary**

| Production Schedule v.2c - 3 Mtpa Annual Summary | | Total | Yr 0 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Yr 10 | Yr 11 | Yr 12 | Yr 13 | Yr 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Physicals** | | | | | | | | | | | | | | | | | |
| Waste Mined | kBCM | **44,633** | 1,514 | 4,880 | 5,825 | 5,309 | 4,080 | 5,680 | 5,867 | 5,765 | 4,256 | 1,457 | | | | | |
| Ore Mined | kBCM | **21,085** | 419 | 2,328 | 1,896 | 2,273 | 2,587 | 2,334 | 2,127 | 2,661 | 2,875 | 1,585 | | | | | |
| **Total Mined** | **kBCM** | **65,719** | **1,933** | **7,208** | **7,721** | **7,582** | **6,668** | **8,014** | **7,994** | **8,426** | **7,131** | **3,042** | | | | | |
| Waste Mined | kT | **87,522** | 2,435 | 10,113 | 10,558 | 13,053 | 9,822 | 10,744 | 11,536 | 8,810 | 7,462 | 2,989 | | | | | |
| Ore Mined (Including Heap Leach and Dumps) | kT | **41,042** | 680 | 5,388 | 4,774 | 4,340 | 6,243 | 4,996 | 4,190 | 3,644 | 4,165 | 2,622 | | | | | |
| Au Grade | g/t | **1.42** | 1.53 | 1.54 | 1.85 | 1.42 | 1.51 | 1.40 | 1.35 | 1.06 | 1.16 | 1.27 | | | | | |
| **Total Mined** | **kT** | **128,564** | **3,116** | **15,501** | **15,332** | **17,394** | **16,064** | **15,740** | **15,727** | **12,454** | **11,626** | **5,610** | | | | | |
| Ore Stockpiled | kT | **16,363** | 680 | 2,887 | 2,100 | 1,371 | 3,261 | 2,296 | 1,216 | 669 | 1,192 | 690 | | | | | |
| Ore Reclaimed | kT | **16,363** | | 314 | 326 | 31 | 18 | 300 | 26 | 26 | 28 | 1,068 | 3,000 | 3,000 | 3,000 | 3,000 | 2,227 |
| Stockpile Balance | kT | **14,605** | 680 | 3,253 | 5,027 | 6,367 | 9,610 | 11,606 | 12,797 | 13,440 | 14,605 | 14,227 | 11,227 | 8,227 | 5,227 | 2,227 | 0 |
| Process Plant Feed | kT | **41,042** | | 2,815 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 2,227 |
| Au Grade | Au | **1.42** | | 2.37 | 2.64 | 1.74 | 2.26 | 2.01 | 1.62 | 1.16 | 1.37 | 1.22 | 0.69 | 0.69 | 0.69 | 0.69 | 0.68 |
| **Ounces** | | | | | | | | | | | | | | | | | |
| Mines | koz | **1,880** | | 214.5 | 254.3 | 167.5 | 218.4 | 194.2 | 156.6 | 111.8 | 131.8 | 117.7 | 66.2 | 66.2 | 66.2 | 66.2 | 48.4 |
| Recovered | koz | **1,562** | | 162.5 | 192.5 | 134.4 | 185.2 | 150.3 | 133.3 | 103.4 | 123.9 | 109.1 | 56.3 | 56.3 | 56.3 | 56.3 | 41.9 |
| Recovery | % | **83.1%** | | 75.8% | 75.7% | 80.2% | 84.8% | 77.4% | 85.1% | 92.5% | 94.0% | 92.7% | 85.1% | 85.1% | 85.1% | 85.1% | 86.6% |

The graph of the yearly mined and processed grades in Figure 16.4.3 shows that the processed grade is consistently higher than the mined grade due to the stockpiling of the low grade ore. The dip in the processed grades in Years 3 and 7 are caused by supplementing the process feed from the existing ore stockpiles (Aires heap pad). Figure 16.4.4 shows the decreasing trend of the contained and recovered gold ounces over the years in the production schedule.

**Figure 16.4.3 Yearly Mined and Processed Ore Grades**

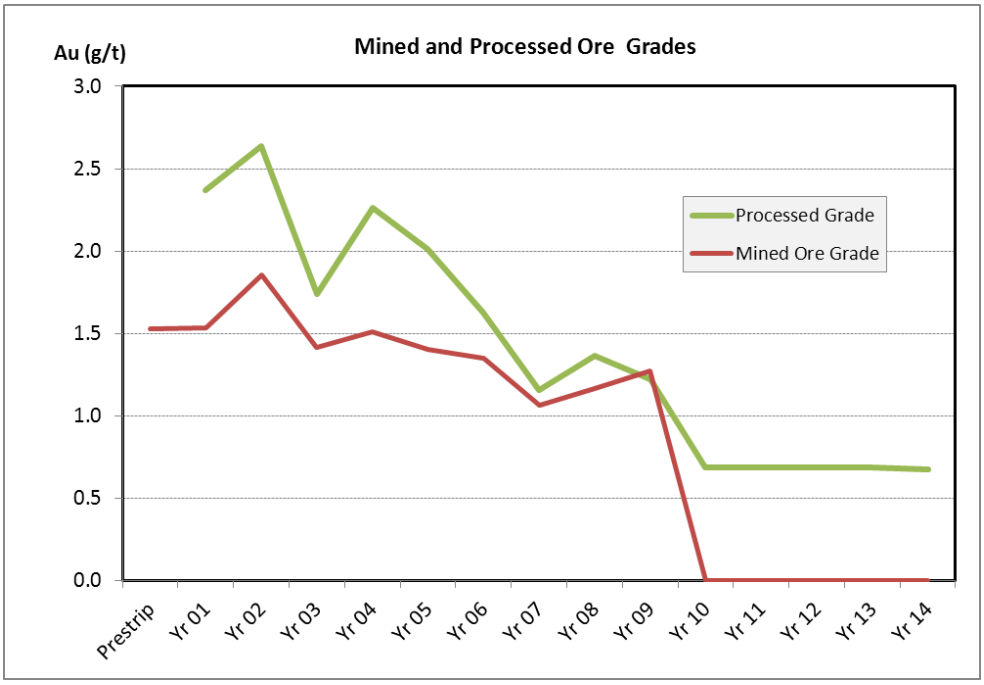
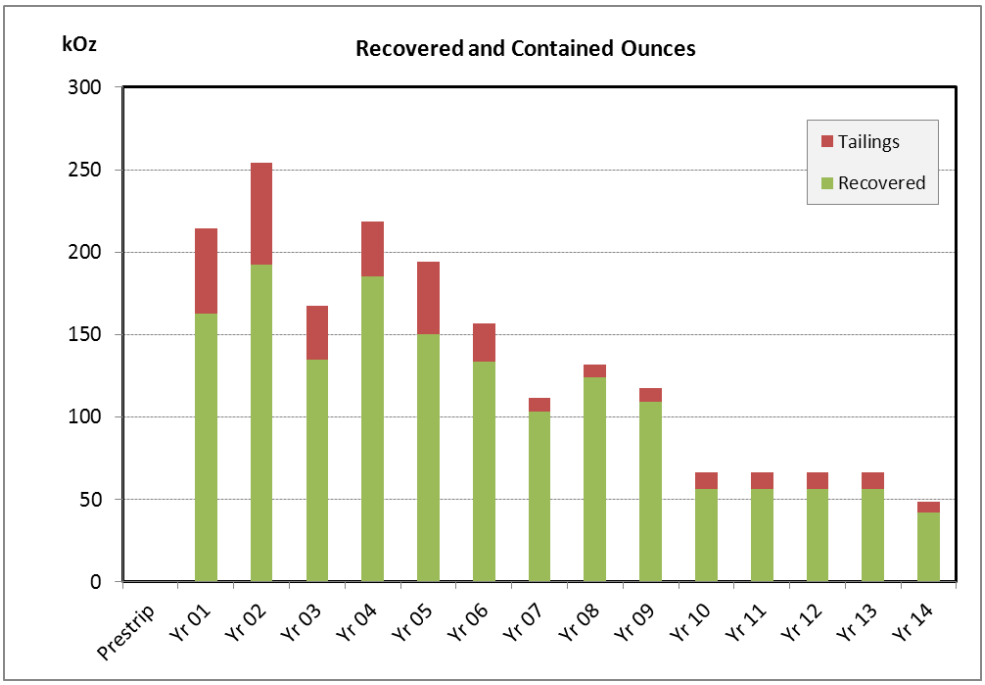



**Figure 16.4.4 Yearly Recovered and Contained Ounces**



## 16.5 Fleet Size and Personnel Numbers

### 16.5.1 Mining Fleet

The criteria for the selection of mine equipment fleet types was to ensure a high level of ore / waste selectivity whilst providing mining flexibility when considering:

- the gold mineralisation style
- operation of multiple open pits
- required material movements
- ore reclamation from stockpiles.

For the purpose of the project study, specific equipment OEM types were assigned based on the above criteria. These provided the basis for purchase prices, equipment specifications and life cycle operating costs. Mobile equipment types were selected assuming an Owner Mining scenario and are summarised in Table 16.5.1.

**Table 16.5.1 Mining Equipment Selection**

| Activity | Equipment Specification | Equipment Model |
|---|---|---|
| Excavating | 100 tonne Hydraulic Backhoe Excavator | Komatsu PC1250SP-8R |
| Pit material Haulage | Rigid chassis dump truck – 90 tonne payload | Komatsu HD785-7 |
| Blasthole Drilling | Down The Hole Rig, 102 – 165 mm drill range | Sandvik DI550 |
| Track Dozers | 70 tonne operating weight track type tractor | Komatsu D375A-6R |
| Wheel Dozers | 50 tonne operating weight wheel dozer | Komatsu WD600-6 |
| Watercarts | Rigid chassis dump truck fitted with 50 kL capacity water body | Komatsu HD465-7R |
| Graders | Motor Grader – 16 foot blade | Komatsu GD825A-2 |
| Ore SP Rehandle FEL | 50 tonne operating weight front end wheel loader | Komatsu WA600-6R |
| Crusher Feed FEL | 50 tonne operating weight front end wheel loader | Komatsu WA600-6R |
| Ore SP Haulage | Rigid chassis dump truck – 90 tonne payload | Komatsu HD785-7 |

Equipment numbers were calculated on the following basis:

- Loading unit productivity rates in line with industry standards, based on equipment specifications, adjusted to site operating factors, including weather and skill levels.
- Truck productivity rates by pit and bench based on the FS determined haulage profiles, assigned truck speeds for various segments at target payloads. Overall truck productivity rates include an allowance for load and dump static times.
- Ancillary equipment (dozers, graders, watercarts) based on allocated proportions of the loading units.

- Blasthole drilling based on allocated proportions of fresh rock material using assumed drill productivity factors.

- Y00 and Y01 include productivity ramp factors by equipment to allow for equipment commissioning and operator training. This results in included operating hours for these periods.

Table 16.5.2 provides the life of mine equipment numbers by year.

**Table 16.5.2 Mining Equipment Schedule**

| Equipment | Yr00 | Yr01 | Yr02 | Yr03 | Yr04 | Yr05 | Yr06 | Yr07 | Yr08 | Yr09 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 100 Tonne Excavator | 2 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 3 | 2 | | | | | |
| Ore Rehandle Loader | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Pit Trucks | 5 | 11 | 11 | 11 | 13 | 12 | 10 | 6 | 7 | 5 | | | | | |
| SP Rehandle Trucks | | 2 | 1 | 2 | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| Track Dozers | 2 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 3 | 2 | | | | | |
| Wheel Dozers | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Graders | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Watercarts | 2 | 4 | 4 | 4 | 4 | 4 | 4 | 3 | 4 | 3 | 2 | 2 | 2 | 2 | 2 |
| Crusher Feed FEL | | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| Blasthole drills | | 3 | 2 | 4 | 4 | 3 | 2 | 0 | 0 | 2 | | | | | |
| Tyre Handler | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |

All equipment will be purchased new, with procurement of additional as required by the schedule, including replacement units where assigned economic life is achieved.

**Table 16.5.3 Mobile Equipment Purchase Schedule**

| | Yr00 | Yr01 | Yr02 | Yr03 | Yr04 | Yr05 | Yr06 | Yr07 | Yr08 | Yr09 | Yr10 | Yr11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Excavators | 3 | | | | | | 3 | | | | | |
| Front End Loaders | 3 | | | 1 | | | | | | | | 2 |
| Trucks | 13 | 2 | | 2 | | | | | | | | |
| Track Dozers | 3 | | | | | | | | | | | |
| Wheel Dozers | 2 | | | | | | | | | | | |
| Graders | 2 | | | | | | | | | | | |
| Watercarts | 4 | | | | | | | | | | | |
| Blasthole drills | 2 | 2 | | | | | 1 | | 1 | | | |
| Tyre Handler | 1 | | | | | | | | | | | |

### 16.5.2 Personnel and Manning

***Direct Labour***

Direct Mining labour personnel have been assigned on a 12 hour shift basis on a 10 days on, five days off roster. This roster relates to a 15-day work cycle relating to a three panel roster on two shifts and a two panel roster on one shift. Table 16.5.4 provides key labour operating assumptions.

**Table 16.5.4 Personnel Basis**

| Activity | Comment |
|---|---|
| Open pit drill, load, haul | Blasthole drilling as well as Load and Haul commences on a two-shift basis then drops to one shift in the latter part of the pit mining schedule. |
| Ore SP rehandle | Ore rehandle from external stockpiles commences on single shift then moves to a two shift arrangement as the schedule quantity increases. |
| ROM ore rehandle | Ore rehandle from the ore ROM pad is carried out on a two-shift arrangement throughout the mine schedule. |
| Blasting | Scheduled on a dayshift only basis throughout the mining schedule. |
| Maintenance | Maintenance labour for standard repair and maintenance tasks is based on equipment SMU hours per operating period with allocated factors applied for each machine class. The requirement for double or single shift is aligned with open pit mining. The maintenance rates include an allowance for OEM planned maintenance labour as an addition. |

The average annualised direct labour numbers are provided in Table 16.5.5. Maximum numbers (159) occurs in Year 1. Labour numbers include an allowance for pre-production operator training.

**Table 16.5.5 Direct Labour Schedule**

| | Yr00 | Yr01 | Yr02 | Yr03 | Yr04 | Yr05 | Yr06 | Yr07 | Yr08 | Yr09 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Equipment Operators | 54 | 94 | 90 | 101 | 99 | 98 | 94 | 70 | 74 | 52 | 23 | 23 | 23 | 23 | 17 |
| Blasting Personnel | | 7 | 8 | 8 | 8 | 8 | 8 | 2 | 2 | 6 | | | | | |
| Maintenance Personnel | 24 | 47 | 44 | 33 | 33 | 32 | 32 | 24 | 24 | 15 | 9 | 9 | 9 | 9 | 7 |
| **Total** | **77** | **148** | **142** | **142** | **140** | **137** | **133** | **96** | **100** | **73** | **32** | **32** | **32** | **32** | **24** |

***Mine Management and Technical Services (Indirect Labour)***

The cost estimate for Mine Management and Technical Services (Indirect) personnel numbers have been developed by considering the functions performed by each office department. Indirect labour numbers are reasonably steady throughout the open pit mining period after which there is a decrease from year nine when mining activities consist of stockpile rehandle and crusher feed only.

Table 16.5.6 provides a summary of the indirect personnel on an annual basis.

**Table 16.5.6 Indirect Labour Schedule**

| | Yr00 | Yr01 | Yr02 | Yr03 | Yr04 | Yr05 | Yr06 | Yr07 | Yr08 | Yr09 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mine Management | 3 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Technical Services | 9 | 13 | 13 | 13 | 13 | 13 | 13 | 13 | 12 | 8 | 8 | 8 | 8 | 8 | 6 |
| Geology | 8 | 12 | 12 | 12 | 12 | 12 | 12 | 12 | 11 | 6 | 5 | 5 | 5 | 5 | 4 |
| Production | 17 | 13 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 7 | 5 | 5 | 5 | 5 | 4 |
| Maintenance | 11 | 11 | 9 | 9 | 9 | 9 | 9 | 9 | 8 | 5 | 2 | 2 | 2 | 2 | 2 |
| **Total** | **47** | **51** | **46** | **46** | **46** | **46** | **46** | **46** | **42** | **28** | **21** | **21** | **21** | **21** | **16** |

***Summary of Mining Personnel by Category***

Table 16.5.7 provides a summary of the total mining personnel on an annual basis reported by principal labour category.

**Table 16.5.7 Total Mining Labour**

| | Yr00 | Yr01 | Yr02 | Yr03 | Yr04 | Yr05 | Yr06 | Yr07 | Yr08 | Yr09 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Foreign Expatriate | 9 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 10 | 7 | 4 | 4 | 4 | 4 | 3 |
| Ivorian FIFO | 35 | 34 | 29 | 29 | 29 | 29 | 29 | 29 | 26 | 17 | 13 | 13 | 13 | 13 | 10 |
| Local Area | 81 | 155 | 149 | 149 | 147 | 144 | 140 | 103 | 106 | 77 | 36 | 36 | 36 | 36 | 27 |
| **Total** | **124** | **200** | **187** | **187** | **186** | **183** | **179** | **141** | **141** | **101** | **53** | **53** | **53** | **53** | **40** |

## 16.6 Mining Capital and Operating Costs

Details of mining capital and operating costs are provided in Section 21.1 Mining Cost Estimates.

## 16.7 Technical Risks and Opportunities

Table 16.7.1 and Table 16.7.2 summarise potential risks and opportunities related to mine design, scheduling, equipment selection and cost estimation.

**Table 16.7.1 Mining Related Risks**

| Risks | |
|---|---|
| **Risk** | **Control** |
| **Development** | |
| Over-estimation of Resources and Reserves. | Resource and Reserve estimates comply with CIM classification system. Inferred resources have been excluded from the reserve estimate. The mineral reserves estimate is the result of the feasibility study. |
| Under-estimation of operating costs (mining, processing and administration) for reserve estimate. | The cost estimates are from first principles and based on experience in other projects and operations. Sensitivity analysis in the financial model would show the impact on the pit economically. |
| Over-estimation of metallurgical performance for reserve estimate. | The process recovery estimates are based on analysis of the data from metallurgical test work. The samples spatially cover the mining areas reasonably well. |
| Feasibility pit designs are not fully achievable. | The pit designs currently have dual lane ramp system for the full pit depth. The ramps can be designed single lane for the last ~30m depth of the pits to mine more ore or to compensate for any changes due to the adjustment in slope angles. The pit designs need to be reviewed regularly for the changes in the geotechnical and geological conditions. |
| **Operational** | |
| Production schedule is not achievable. | Monitoring of the production and regular reconciliations can identify the potential issues earlier to rectify the production schedule. The vertical advance rates are particularly low for the largest main Daapleu open pit. Any production shortfall during operations can be sourced from the existing heap and dump stockpiles if necessary. |
| Grade control practice underperforms. | This might be an issue at the start of the operations. It is expected that the grade control system will settle after mining several benches and reconciling the production for each deposit. |
| Environmental and local public relation issues. | The noise, dust, water quality and traffic issues needs to be managed well. The structure for this is already in place as the Ity heap leach mine has been operating since 1991. |

| Risks | |
|---|---|
| **Risk** | **Control** |
| **Development** | |
| Poor haul road design, construction and / or poor ongoing road maintenance leading to operational inefficiencies.<br><br>As an example, poor of road management would lead to:<br>Reduced truck speeds and therefore increased truck cycle times and requirements and therefore operating costs<br>Reduced tyre life<br>Increase operating costs | The FS assumes that the mining surface roads will be built to a high standard, including design of a quality drainage system to manage the high rain in the area. Selected ancillary equipment resourcing and operating hours in the FS allows for ongoing road maintenance (e.g. graders, wheel dozers, watercarts). |
| Requirement to drill and blast a proportion of Transitional material, resulting in increased drill and blast resources and costs. | The FS mining cost estimate includes a secondary blasting allowance and rock breakers to account for this risk. |
| Increased Presplit wall control requirement. Post FS drill and blast optimisation work related to presplit wall control could lead to an increased pre-split requirement. | Unknown for FS. Risk and its treatment is reliant on post FS drill and blast study. |
| Increased crusher FEL requirement due to higher than assumed crusher unavailability due to clearing blockages or other downstream constraints in the processing circuit. | FS modelling has in built latent FEL crusher feed capacity. |
| Open pit mining delays resulting from higher water inflow rates. This could result from either storm events or higher than assumed ground water inflow rates to the base of the pits. | The risk of higher water inflow rates would be controlled in the first instance by construction of a quality surface drainage system. Regarding in pit pump capacity, the FS has used conservative instantaneous allowances for in pit pump rates. In the event the rates are higher than allowed for, additional pumps could be mobilised quickly and with relative ease to manage dewatering demand spikes. |
| Lower than planned productivity rates from equipment. | The FS allocated equipment productivity rates assume a level of reasonable local operator skill. This risk is managed in the FS through allocation of specialist trainers, together with an upfront productivity ramp-up factors and the fact that such skills are currently available onsite. |
| **Equipment Specific** | |
| Delivery times. Poor management in this area could result in delayed equipment delivery and commissioning timing, hence delayed mine commencement. | This risk would be managed by engagement of competent OEMs who would be working under a detailed mining Execution Plan to be defined by Endeavour as a post FS activity. |

**Table 16.7.2 Mining Related Opportunities**

| OPPORTUNITIES | |
|---|---|
| **Opportunity** | **Control** |
| **Resource and Design** | |
| Verse Ouest dump resource upgrade from Inferred to Indicated resources. | Post FS study.<br>The recent resource upgrade for the Verse Ouest dump resources has not been included in the feasibility study yet. The Indicated resources of 5.9Mt at 0.99g/t Au grade will add two years to the mine life and improve the cash flow relatively significantly. The grade is relatively low for the stockpile resources but the cost per ounce is also reasonably low. |
| Bakatouo deposit resource estimate. | Post FS study.<br>The evaluation and planning of the new deposit needs to be carried out to include in the Ity production schedule. |
| Iterative pit optimisation and pit design process. | Post FS study.<br>The pit designs in the feasibility study is significantly larger than the optimal pit shells due to the inclusion of the ramps and mining width. The final operating cost estimate is significantly lower than the initial estimate used in pit optimisations. The economics of the Project can potentially be improved through revised pit optimisations and pit redesign process.<br>The feasibility pit designs also need to be reviewed for local variations in geotechnical and geological conditions. |
| **Operational** | |
| Increased available working days per year. The FS assumes 28 days lost per annum due to weather. | Operationally there is potential to improve on this assumption with quality road construction and ongoing best in class road maintenance. This would have a flow on affect in areas such as increased tyre life and reduced operating costs. |
| Reduced blasting requirements following a post FS specialised drill and blast assessment | Unknown for FS. Reliant on a post FS study. |
| Reduced blasthole drills numbers as a result of schedule smoothing.<br>Allocation of fresh rock only blasting results in some blasthole drill number variances. There is potential to eliminate one drill through operational smoothing/optimisation | Reliant on both the post FS drill and blast review, together with post FS open pit inventory and schedule optimisation. |

| OPPORTUNITIES | |
|---|---|
| **Opportunity** | **Control** |
| **Resource and Design** | |
| Converse to the Presplit risk, further work in this area could result in elimination of any requirement for Pre-split wall control. Should this be the case, drill and blast resourcing and therefore costs would decrease. | Unknown for FS. Reliant on a post FS drill and blast study. Overall, the risk/opportunity for this area is seen as being neutral. |
| Reduced Expat and training allocation costs by realising higher actual local labour skill levels during project execution/ongoing operations. | Unknown for FS. Dependent on the actual skill levels of those employed. |
| **Equipment Specific** | |
| Reduced loading and haulage engine hours as a result of optimising tray and bucket capacities. | Post FS activity. |
| Further to the above, improved truck productivity by increasing payloads to plus 10% within the OEM recommendations through operational optimisation | Post FS activity. |
| Reduction in mobile equipment unit capital prices as a result of post FS negotiations and solid firm pricing requests. | Post FS activity. |
| Equipment specification refinement further to the RFBP to ensure optimal equipment configuration for required duties and operating efficiencies. | Post FS activity |

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# 17.0 RECOVERY METHODS

## 17.1 Process Selection

### 17.1.1 Design Philosophy

The proposed process plant design for the Ity CIL Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations and other Endeavour Mining Corporation operations that are well proven in industry.

The key criteria for equipment selection are the suitability for duty and the projected mine life of the operation without unnecessarily compromising reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a compact footprint to minimise construction costs.

The Ity CIL plant will process a range of ore types (oxide, transition and fresh) with variable ore characteristics, gold grades and metallurgical treatment requirements. The primary ores are significantly more competent than the oxide ores.

Initial pit optimisation and scheduling has indicated that the largest pit is Daapleu which contains up to 50% of the mineable reserve and contains both oxide and primary ores. Testwork results have indicated that Daapleu primary ore contains significant arsenopyrite and a standard cyanide leach will only extract 60 - 65% of contained gold. By contrast the oxide ores are free-milling with leach extraction of 90 - 96%.

The pit optimisation work indicated that parallel processing of the Daapleu primary ores with free milling oxide ores gave the best return. As a result, the higher grade Daapleu material will be mined and processed for the first six years of the mine, with lower grade Daapleu material stockpiled during this time and processed from Years 9 to 14. Mill feed requirements will be supplemented successively with oxide material from Ity Flat, Mont Ity, Gbeitouo, Walter and Zia NE.

The Ity Project ores contain significant silver which has influenced the design capacity of the elution circuit and goldroom.

The key project and ore specific design criteria that the plant design will meet are as follows:

- 3 Mtpa of blended ore; 58% primary and 42% oxide (LOM).
- Crushing plant mechanical availability of 80%.
- Mechanical availability for the remainder of the plant of 91.3% supported by crushed ore storage and stand-by equipment in critical areas.
- Sufficient automated plant control to minimise the need for continuous operator intervention and allow manual override and control if and when required.

A process design criteria document has been prepared incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical testwork and comminution circuit modelling. The design document forms the basis for the design of the processing plant and required site services.

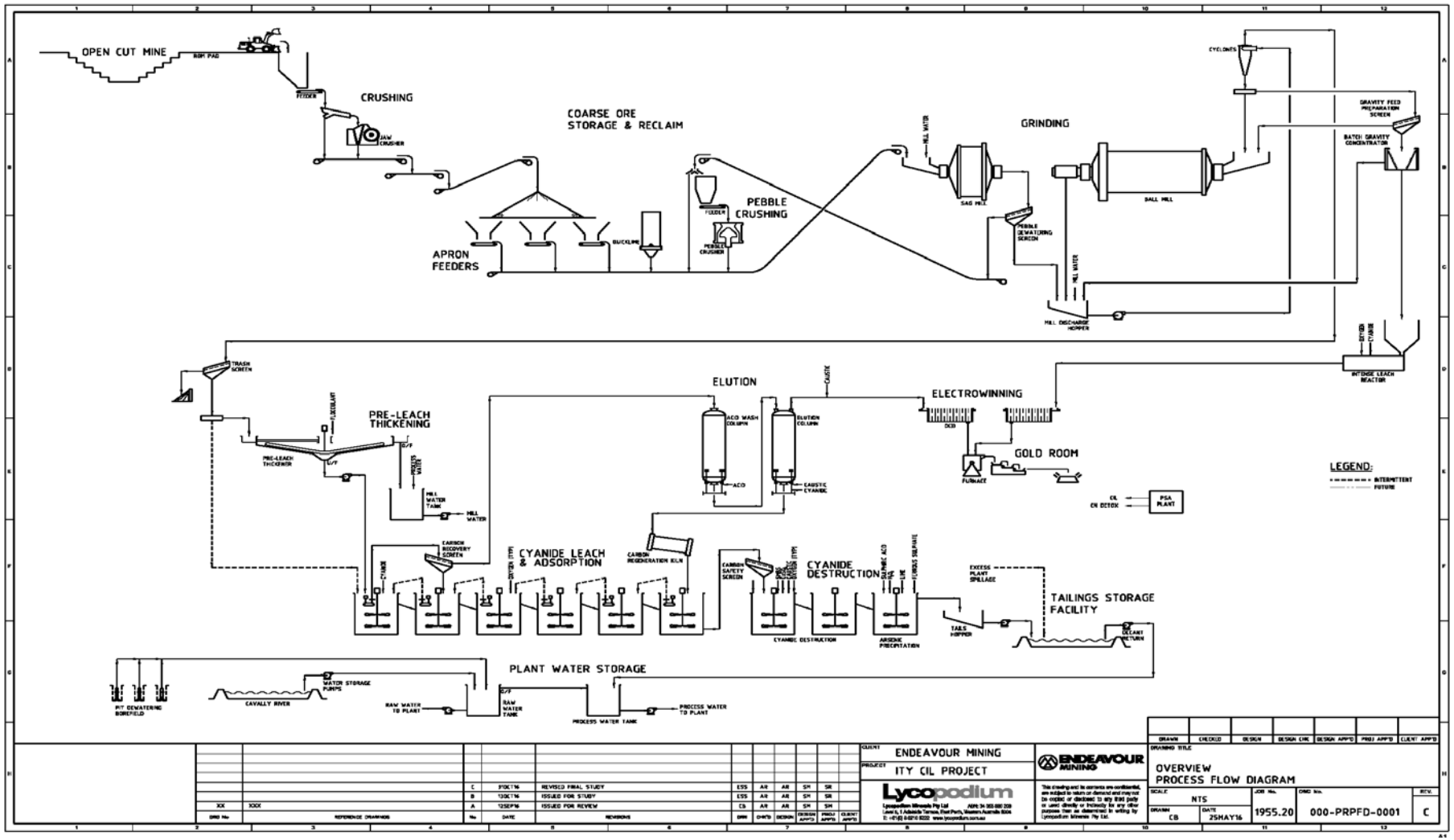
### 17.1.2 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Single stage primary crushing to produce a crushed product size of 100% passing ($P_{100}$) 300 mm ($P_{80}$ of 160 mm).
- Transfer conveyor feeding a coarse ore stockpile (12 hours live). Ore reclaim via apron feeders.
- Two stage SAG / Ball milling in closed circuit with hydrocyclones to produce a $P_{80}$ grind size of 75 µm and including re-crushing of pebbles from the SAG mill.
- Gravity concentration and removal of coarse gold from the milling circuit recirculating load and treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to doré.
- Pre-leach thickener to increase the slurry density to the carbon in leach (CIL) circuit to minimise CIL tankage, improve slurry mixing characteristics and reduce overall reagent consumption.
- A CIL circuit incorporating six CIL tanks containing carbon for gold and silver adsorption.
- Fifteen tonne capacity split Anglo (AARL) elution circuit, electrowinning and gold smelting to recover gold and silver from the loaded carbon to produce doré.
- Tailings treatment incorporating cyanide destruction using sodium metabisulphite / air, followed by arsenic precipitation and stabilisation.
- Tailings pumping to the tailings storage facility (TSF).

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet is shown in Figure 17.1.1.

**Figure 17.1.1 Process Flowsheet**



### 17.1.3 Plant Design Basis

The key issues considered in the process and equipment selection are outlined in this section.

***Process Plant***

The plant design has been based on a nominal capacity of 3.0 Mtpa of blended ore. The feed will consist primarily of ore from the Daapleu pit (47% of total, 80% of mill feed in the first two years, falling to 16% in Year 6, nil in Years 7 and 8, and increasing to approximately 63% in Years 9 to 14 with lower grade stockpiled Daapleu material). This will be supplemented progressively with mostly oxide ores from Ity Flat, Mont Ity, Gbeitouo, Walter, Zia NE and existing ore stockpiled from current HL operations.

***ROM Pad***

The ROM pad will be an important part of the mine operation with ore being sourced from up to three pits at one time. It will be used to provide a buffer between the pits and the plant. The ROM stockpile will allow blending of feed stocks and ensure a consistent feed type and rate to the plant. A mobile rock breaker will be used to break any oversize rocks on the ROM pad.

***Comminution Circuit Selection***

The comminution circuit design has been based on an evaluation of 2016 testwork with historical comminution testwork reviewed, but not used in detail, because of the lack of data on the origins of the samples tested. The findings are summarised in the comminution design report compiled by Orway Mineral Consultants (OMC).

The conclusions from the review were as follows:

- The fresh Daaplite is the most significant ore type contributing 30% of the mill feed over the life of the mine with the Daapleu meta-volcanic sediments contributing a further 14% and these ores are thus the main circuit design drivers. Review of the data on these ore types indicated some variability between recent and previous testing so only the recent testwork was used to size the circuit.

- The ore comminution properties indicated a significant proportion of softer oxides with some specific harder materials. As there was significant variability in the ore coming from a number of sources, a flexible circuit was needed which could accommodate the softer ores while also making provision for the harder components in Daapleu and Mont Ity.

- In addition, the fine grind of 75 microns selected from the testwork favoured a two-stage grinding circuit.

For variable ores of this type three options can be considered:

- Multi stage crushing (2 or 3 stages) followed by single stage ball milling. While this circuit is suitable and is expected to be the most energy efficient and easily operable for the high competency fresh ores it will be difficult to operate when treating oxide ores with materials handling problems in the crushing and screening circuits.

- Primary crushing, single stage SAG milling and recycle crushing (1C SS SAC). This circuit is well suited to homogeneous ore feed or ores in which the competency trends with grindability. When the rock becomes more competent the BWi goes up as well or vice versa. However, variable competency ores will lead to a variable load in the SAG mill and as such insufficient media may be available to grind to fine grind sizes. Given the variability it was determined that constantly achieving a product size of 75 microns may be problematic and as such Single Stage SAG milling was not viewed as the preferred circuit.

- Primary crushing, two-stage SAG / Ball milling and recycle crushing (SABC). This circuit has the advantage of accommodating coarse feed in the SAG mill with the final grind being achieved and recirculating load being controlled via the ball mill. More competent ores will discharge scats which will be re-crushed in the pebble crushing circuit. This circuit effectively separates the size reduction of coarse ore in the SAG mill from the need to produce a fine final product size in the ball mill. This circuit will still be prone to over grinding at times, however, this is an acceptable outcome for a grinding circuit feeding a leach circuit.

- The circuit option selected was SABC due to the extreme variability of the ore and the relatively fine product size of 75 microns

**Selected Comminution Circuit**

- A summary of the equipment selection and expected consumables is provided in Table 17.1.1 and Table 17.1.2.

**Table 17.1.1 Summary of Selected Milling Circuit**

| Parameters | Condition | Units | Design Blend |
|---|---|---|---|
| Design Blend | | % | 100 |
| CWi | | kWh/t | 15.0 |
| BWi | | kWh/t | 13.1 |
| Axb | | - | 49.3 |
| Milling Feed $F_{80}$ | | mm | 160 |
| Milling Product $P_{80}$ | | µm | 75 |
| Mill Specific Energy | @ $P_{80}$ 75 µm | kWh/t | 18.9 |
| **SAG Mill** | | | |
| Mill Diameter | | m | 7.32 |
| Mill EGL | | m | 3.79 |
| L:D Ratio | | | 0.52 |
| Ball charge | Duty | % | 10 |
| | Max | % | 15 |
| Total load | Duty | % | 25 |
| | Max | % | 35 |
| Mill speed | | % | 75 |
| Drive type | | | Variable Speed |
| Mill Pinion Power | Nominal | kW | 2,774 |
| | Max | kW | 3,370 |
| Installed power | | kW | 4,000 |
| Discharge Arrangement | | | Grate |
| Open area | | % | 8 |
| Grinding Media | | mm | 125 |
| **Ball Mill** | | | |
| Mill Diameter | | m | 5.80 |
| Mill EGL | | m | 9.20 |
| L:D Ratio | | | 1.59 |
| Ball charge | Duty | % | 27 |
| | Max | % | 35 |
| Mill speed | | %Nc | 75 |
| Drive type | | | Fixed Speed |
| Mill Pinion Power | Nominal | kW | 4,259 |
| | Max | kW | 5,052 |
| Installed power | | kW | 5,600 |
| Discharge Arrangement | | | Overflow |
| Grinding Media | | mm | 50 |

**Table 17.1.2 Comminution Consumables by Ore Type (@ Design Throughputs)**

| Equipment / Parameter | Unit | Design | Average | Oxide Ave | Part Oxide Ave | Fresh ore Ave |
|---|---|---|---|---|---|---|
| Abrasion Index | g | 0.26 | 0.14 | 0.10 | 0.12 | 0.15 |
| Primary Crusher | | | | | | |
| Liner Consumption – Fixed Jaw | hours/unit | 1,540 | 2,500 | 3,100 | 2,700 | 2,410 |
| Liner Consumption – Moving Jaw | hours/unit | 2,290 | 3,690 | 4,600 | 4,020 | 3,580 |
| Power Consumption | kWh/t milled | 0.17 | 0.12 | 0.08 | 0.12 | 0.17 |
| SAG Mill | | | | | | |
| Media Consumption | kg/t milled | 0.292 | 0.132 | 0.049 | 0.054 | 0.207 |
| Liner Consumption | kg/t milled | 0.057 | 0.027 | 0.010 | 0.011 | 0.042 |
| Power Consumption | kWh/t milled | 8.0 | 4.6 | 1.9 | 2.0 | 7.0 |
| Ball Mill | | | | | | |
| Media Consumption | kg/t milled | 0.673 | 0.364 | 0.237 | 0.306 | 0.419 |
| Liner Consumption | kg/t milled | 0.088 | 0.049 | 0.032 | 0.041 | 0.056 |
| Power Consumption | kWh/t milled | 12.31 | 8.37 | 6.21 | 7.45 | 9.39 |
| Pebble Crusher | | | | | | |
| Liner Consumption | hours/unit | 1,170 | 1,990 | 2,660 | 2,000 | 1,990 |
| Power Consumption | kWh/t milled | 0.16 | 0.11 | 0.05 | 0.11 | 0.16 |

***Crushing and Ore Storage***

The crushing circuit has been sized on the basis of achieving 80% utilisation at a feed rate 15% above that of the milling throughput. Excess crushed ore will be stockpiled and fed to the process plant using a FEL via the crushed ore reclaim system during periods of crusher maintenance. A grizzly with an aperture of 800 mm will be installed on the ROM bin to minimise oversize material entering the bin and causing down-stream blockages.

A primary apron feeder will withdraw ore from the ROM bin and can accommodate both softer ores which can compact with a vibrating feeder and harder more competent rocks. A vibrating grizzly will allow crusher product sized ore to bypass the jaw crusher, reducing the load and wear on the jaw crusher. A jaw crusher has been selected as being able to process soft and hard ores. As the abrasion index of the ore is low to moderate no significant wear issues are expected.

Crushed ore will feed a 12 hour capacity live stockpile. The stockpile is relatively small in size to accommodate the softer ores in the schedule and prevent hang-up in the feeder chutes. Excess ore from the crusher will be stockpiled and can be fed via two reclaim chutes.

***Milling and Classification***

The milling circuit has been selected to reduce crushed product to the nominal circuit $P_{80}$ size of 75 µm. The SAG mill will be equipped with a variable speed drive and will be capable of operating between 60% and 80% of critical speed. The ball mill will be fixed speed. The variability in ore competence can therefore be addressed by varying the speed of the SAG mill. The softer ores will require a higher ball charge and slower speed. The harder ores will require a higher mill speed.

Cyclones have been selected for the classification duty to ensure that there are sufficient duty cyclones to allow maintenance on the units without compromising operability.

Selection of a pre-leach thickener (see below) to increase leach feed density allows operation of the milling and classification circuit to be optimised by operating cyclones at feed densities that increase classification efficiency, thereby reducing circulating load and overall circuit power consumption.

***Gravity Concentration***

Gravity testwork indicated that 11 - 49% of the gold in feed can be recovered by gravity gold methods. A gravity recovery circuit with intensive cyanidation of the concentrates allows the feed grade to the CIL circuit to be smoothed and thus allows the CIL operation to be optimised by removing spikes of high grade from the feed stream. A separate electrowinning circuit will be provided for the intensive cyanidation solution to assist with metallurgical accounting and eliminate any potential impact on the operation of the carbon elution and electrowinning circuit.

***Pre-leach Thickening***

As well as allowing the grinding circuit to operate at optimum classification densities, inclusion of a pre-leach thickener ahead of the CIL circuit achieves higher and more constant leach feed density. The higher leach feed density reduces the overall tankage volume required to achieve the target residence time. Additionally a higher CIL density improves slurry mixing characteristics and reduces reagent costs. Pre-leach thickening also recovers process water for re-use in the plant and allows cyanide-free water to be added in the milling circuit.

Thickening testwork showed high settling rates for most of the oxide and primary ores however the viscosity of some of the oxide ore (Ity Flat and Zia NE) at the target grind size increased significantly as the thickened slurry solids concentration increased. As the feed to the plant will be a blend of Daapleu with other ore types, a thickener underflow density maximum of 50% w/w solids was selected for the plant.

***Leach and Adsorption Circuit***

Metallurgical testwork indicated that:

- the Ity CIL ores do not show any indication of 'preg-robbing' characteristics
- the initial leach kinetics for most of the oxide ores are rapid, with 80 - 90% of total gold extraction being achieved in the first four hours of leaching
- the leach kinetics of the Daapleu primary ore are also fast although with significantly lower extraction
- some ores show somewhat slower leach kinetics requiring extended leach times
- silver levels in Ity ores are significant with silver:gold ratio between 2 and 4 and overall silver extraction of 60%.

On this basis, a circuit configuration comprising six CIL tanks has been adopted to achieve acceptable stage efficiency and target solution tails grades. The tanks will be identical in size with cyanide added to the CIL tanks as required. The CIL circuit has been designed for a residence time of 34 hours at 375 t/h solids at 50% w/w density. This is based on the slower leaching behaviour of some specific ores. Oxygen will be added to all tanks to oxygenate the slurry and oxidise any cyanicides and to maintain an adequate dissolved oxygen level for leaching in the CIL tanks.

If more viscous ore types are encountered the CIL circuit will have a residence time of 25 hours at 40% w/w density.

***Elution and Goldroom***

The average daily movement of carbon has been calculated based on the design feed grade and maximum CIL gold and silver extraction (assuming no gravity gold recovery). On this basis, a fifteen tonne capacity split Anglo elution circuit has been selected requiring just under seven strips per week. The Anglo circuit has been selected as it offers the flexibility to run more than one elution in a day.

The key driver for the capacity of the elution circuit is the high silver level with an overall loaded carbon grade of 5,200 g/t of gold and silver. This flows on to the goldroom which requires three large electrowinning cells to recover the high precious metals load.

Some of the ore types (Ity Flat and Mont Ity) contain significant soluble copper. Some of this copper is likely to adsorb onto carbon and dilute the doré. To reduce this effect, the CIL circuit will need to operate at elevated cyanide levels to minimise the level of copper adsorption. In addition, a cold cyanide wash step has been included in the flowsheet after acid wash and prior to commencement of elution. This step produces a cyanide and copper rich stream which will be directed to cyanide destruction.

***Raw Water***

Water for the Project will be sourced from pit dewatering within the Project boundary. The Project area is close to the Cavally River and dewatering bores will be installed at all pit locations. It is expected that sufficient water for process plant operations will be produced from dewatering and the excess will be treated prior to release back into the river.

TSF decant return water will be pumped to the process water tank. A separate mill water tank will receive pre-leach thickener overflow and will be topped up with process water as required.

Raw water will be sourced from pit dewatering bores and will be used as the feed stream for the potable water treatment plant.

### 17.1.4 Key Process Design Criteria

The key process design criteria listed in Table 17.1.3 form the basis of the detailed process design criteria and mechanical equipment list.

**Table 17.1.3 Summary of Key Process Design Criteria**

| | | LOM Blend | Source [1,2,3,4,5,6,7] |
|---|---|---|---|
| Ore Blend | | 25% Daapleu primary/ 75% mixed oxides | Mining Solutions |
| Crushing Circuit Capacity | t/y | 3,000,000 | Lycopodium |
| Plant Throughput - Nominal | t/y | 3,000,000 | Lycopodium |
| Gold Head Grade - Range | g Au/t | 1.0-2.5 | Endeavour |
| Gold Head Grade - Design | g Au/t | 2.5 | Endeavour |
| Overall Gold Recovery | % | 59-97 | Test |
| Silver Head Grade – Design [8] | g Ag/t | 8.5 | Endeavour |
| Design Silver Recovery | % | 63 | Test |
| Crushing Plant Utilisation | % | 80 | Lycopodium |
| Milling / CIL Plant Utilisation | % | 91.3 | Lycopodium |
| ROM Ore Top Size | mm | 800 | Assumed / OMC |
| Crushing Work Index (CWi) | kWh/t | 15 | Assumed / OMC |
| Bond Ball Mill Work Index (BWi) | kWh/t | 13.1 | Testwork / OMC |
| Axb | | 40 | OMC |
| SG | | 2.83 | Testwork / OMC |
| Abrasion Index (Ai) | | 0.30 | Testwork / OMC |
| Comminution Circuit | | SABC | OMC |
| Crush Size, $P_{80}$ | mm | 160 | OMC |
| Grind Size, $P_{80}$ | µm | 75 | Test |
| SAG Mill Pinion Power | kW | 3,370 | OMC |
| Ball Mill Pinion Power | kW | 5,052 | OMC |
| Grinding Media Consumption | kg/t | 0.49 | OMC |
| Gravity Gold Recovery | % | 31 | Test / Assumed |
| Gravity Silver Recovery | % | 20 | Assumed |
| Pre-leach Thickener Solids Loading | $t/m^2.h$ | 0.7 | Testwork / Assumed |
| Leach / CIL Residence Time | hrs | 34 | Lycopodium |
| Leach Slurry Density | % w/w | 50 | Testwork |
| Number of CIL Tanks | | 6 | Lycopodium |
| Cyanide Consumption[5] | kg/t | 0.59 | Testwork |
| Quicklime Consumption[6] | kg/t | 0.74 | Testwork |
| Elution Circuit Type | | Split AARL | Lycopodium |
| Elution Circuit Capacity | t | 15 | Lycopodium |
| Frequency of Elution | strips / week | 6.6 | Lycopodium |

Notes:

1. Mining Solutions refers to advice from the nominated mining subconsultant.
2. 'Lycopodium' refers to Lycopodium experience or generally accepted practice.
3. 'Testwork' refer to metallurgical testwork conducted.
4. 'OMC' refers to advice from Orway Mineral Consultants.
5. Cyanide consumption makes allowance for 100 ppm residual cyanide in the CIL tail solution.
6. Lime consumption based on 90% CaO.
7. Endeavour refers to advice/agreement from Endeavour.
8. Silver grade for design only. Financial assessment based on reserve grade.

## 17.2 Process and Plant Description

The following section provides a description of the process plant, reagents and services area. A summary of the major equipment sizes is provided in Table 17.2.1. Detailed flowsheets, general arrangement drawings, mechanical equipment list, electrical load list are available on request.

**Table 17.2.1 Summary of Major Equipment**

| | Units | |
|---|---|---|
| ROM Bin | **t** | 252 |
| Primary Crusher (w x l) | mm | 1,400 x 1,070 |
| Coarse Ore Stockpile | t | 4,500 |
| SAG Mill (dia x EGL) | m | 7.32 x 3.79 |
| Ball mill (dia x EGL) | m | 5.80 x 9.20 |
| Pebble Crusher Capacity | tph | 150 |
| Gravity Concentrator Capacity | tph | 400 |
| Intensive Cyanidation Reactor Capacity | t/d | 2.8 |
| Hydrocyclones - (No. x dia) | mm | 15 x 380 |
| Pre-Leach Thickener diameter | m | 30 |
| CIL Tanks - (No. x dia) | m | 6 x 2,900 |
| Intertank Screens - (No. x dia) | m | 6 x 2.4 |
| Cyanide Destruction Tanks - (No. x dia) | m | 2 x 550 |
| Arsenic Precipitation Tanks - (No. x dia) | m | 1 x 550 |
| Elution Circuit Type | | Split AARL |
| Elution Circuit Capacity | t | 15 |
| Elution Heater Capacity | kW | 5,000 |
| Pregnant Solution Capacity | $m^3$ | 210 |
| Electrowinning cells (No. x size) | | 3 x 1m sq x 22 cathode |
| Smelting Furnace Capacity | Litre | 49 |
| Cyanide Storage | $m^3$ | 145 |
| Lime Storage | t | 180 |
| Oxygen Plant Capacity | tpd | 20 |
| Raw Water Storage | $m^3$ | 2,200 |
| Process Water Storage | $m^3$ | 1,200 |
| Potable Water Plant Capacity | $m^3/h$ | 13 |

### 17.2.2 Run-of-Mine (ROM) Pad

Haul trucks operating directly from the pit will deliver run-of-mine (ROM) ore to the ROM pad where it will be dumped in blending 'finger' stockpiles arranged by ore grade and lithology. A front end loader (FEL) will be used to reclaim and tram ore from the various stockpiles to the ROM bin. Where possible, ore will be direct tipped into the ROM Bin. Ore will be blended under the guidance of mine geologists and process personnel to maintain a relatively constant feed grade and ore hardness to the process plant.

### 17.2.3 Crushing Circuit

ROM ore will generally be loaded into the crushing circuit feed bin (ROM bin) by FEL from the ROM pad stockpiles. The system is flexible enough for direct tipping of the mined material if required. A grizzly will be fitted to the ROM bin to protect the downstream equipment from oversize material. A mobile rock breaker will be utilised to break oversize rocks.

ROM ore will be drawn from the ROM bin at a controlled rate by an apron feeder which will discharge over a vibrating grizzly. The grizzly oversize will report to the jaw crusher. The grizzly undersize and jaw crusher discharge will gravitate to the primary crushing product conveyor.

The primary crushing product conveyor will discharge onto a 750 m long transfer conveyor which will run up to a 150 m long stockpile feed conveyor. A suspended tramp metal magnet will be located at the transfer point between the primary crushing product conveyor and the overland conveyor and will remove any coarse metal to protect the overland conveyor. A weightometer on the overland conveyor will measure crusher product.

Coarse spillage in the crusher area will be cleaned up by FEL and transported to the ROM pad for drying or fed directly to the primary crusher. Water sprays will be installed for dust suppression as required. A 10 tonne hoist will be provided over the jaw crusher to facilitate regular maintenance.

The crushing circuit will be controlled locally. The FEL driver will ensure feed is maintained to the crushing circuit and will communicate with the crushing operator using a two way radio to supply information on crusher feed operation. The speed of the primary feeder will be controlled to a target set-point and measured using the crushing weightometer.

### 17.2.4 Ore Storage and Reclaim

The stockpile feed conveyor will discharge onto a 5,000 tonne live capacity stockpile. This stockpile will provide 12 hours feed to the milling circuit to allow for regular maintenance on the crusher. Water sprays will be installed for dust suppression as required.

Primary ore will be reclaimed from the stockpile via a single variable speed apron feeder. In addition, two more feeders will be located in line with the SAG mill feed conveyor but outside the live stockpile and will be used to reclaim ore from the dead area of the stockpile in the event of an extended crusher shutdown. Each of the feeders will be rated to reclaim at the full mill feed rate, but may be run together to ensure a regular feed to the SAG mill and to compensate for hang-up in the stockpile feed chute. The feeders will be located in a concrete tunnel above the SAG mill feed conveyor. A forced air fan will ventilate the reclaim tunnel and a pulsed jet dust scrubber will reduce airborne dust generation at the loading points of the apron feeders in the reclaim tunnel.

Quicklime, used for pH control in the leach circuit, will be added directly onto the mill feed conveyor. Quicklime will be stored in a lime silo and will be metered onto the belt using a variable speed rotary feeder. The speed of the rotary feeder will be varied according to the mill feed tonnage. The silo will be loaded pneumatically from bulk lime trucks. An emergency back-up system using lime in bulk bags will provide an alternative in the event that bulk lime deliveries are delayed due to weather conditions. The silo will be fitted with a dust collector.

A conveyor weightometer will be installed on the SAG mill feed conveyor to provide accurate total instantaneous and accumulated mass flow feed into the plant. SAG mill grinding media will be loaded onto the SAG mill feed conveyor from the SAG grinding media storage bunker using a FEL.

The reclaim circuit will be controlled from the main control room. Area shift operators will also monitor the circuit.

### 17.2.5 Grinding and Classification Circuit

The grinding circuit will consist of an SABC circuit with hydrocyclones.

***SAG Mill***

A 7.32 m dia x 3.79 m EGL SAG mill complete with a 4,000 kW variable speed drive will operate at up to a maximum 15% volumetric ball loading. Variable speed control of the mill, accomplished through a VVVF system, will provide flexibility for processing of the various ore types. A speed range of 60% to 80% critical speed will be available.

SAG mill grinding media, typically 100 - 125 mm diameter balls, will be stored adjacent to the bulk ball mill grinding media bunker. SAG mill grinding media will be removed from the bulk storage bunker using a FEL and transferred to the SAG mill feed conveyor.

SAG mill liners and grates will be handled by a mill liner handler capable of managing the new liners.

***Pebble Crushing***

The SAG mill product will discharge to a vibrating single deck heavy duty pebble dewatering screen fitted with a nominal 15 mm aperture rubber screen deck. Undersize from the screen will feed the mill discharge hopper. The oversize pebbles (nominally +15 mm) will be transferred via a transfer conveyor to a pebble crusher feed bin ahead of a single 250 kW pebble crusher.

A single stage of tramp metal removal via a self cleaning belt magnet across the pebble transfer conveyor will be used to remove mill balls and any other magnetic steel debris discharged from the SAG mill. Tramp metal ejected will be deposited in a bunker area with concrete walls on three sides which will provide access for metal removal and protection to personnel in the area. Metal detection will provide a final level of protection against metal entering the pebble crusher. Upon detection of metal, the flop gate at the head of the transfer conveyor will be activated and the ore stream will bypass the pebble crusher surge bin for a predetermined period and will be deposited directly onto the SAG mill feed conveyor.

The pebble crusher surge bin will hold approximately 25 minutes capacity to improve steady state operation of the pebble crusher. The nominal design allows for use of a single pebble crusher to meet the expected pebble load as the competence of the ore is moderate.

A variable speed pebble crusher vibrating feeder will transfer the pebbles at a controlled rate from the surge bin into the pebble crusher. Pebbles will be crushed from a nominal top size of 75 mm to a $P_{80}$ of approximately 12 mm. The pebble crusher will operate at a closed side setting of 11 mm, depending on the ore competency, moisture content and crusher power draw. Product from the pebble crusher will be transferred to the SAG mill feed conveyor. An overhead hoist will be provided to facilitate crusher maintenance

***Ball Mill***

A 5.80 m dia x 9.2 m EGL overflow ball mill complete with a 5,600 kW fixed speed motor will operate at up to 35% volumetric ball loading. Product size from the grinding circuit will be 80% passing 75 µm on 100% primary ore. The ball mill will be fixed speed operating at 75% critical speed.

Ball mill grinding media will be 65 mm diameter balls and will be stored adjacent to the bulk SAG mill grinding media bunker. Ball mill grinding media will be removed from the bulk storage bunker and transferred to a kibble. Grinding media will be loaded into the ball mill via the ball mill ball charging hoist. The ball charging hoist will be used to manoeuvre the ball charging kibble onto the ball mill feed box from where the grinding media will enter the ball mill.

### 17.2.6 Classification and Trash Screening

The discharge from the SAG and ball mills will be combined in a common mill discharge hopper and diluted with process water prior to classification. The combined mill discharge slurry will be pumped using duty / stand-by pumps, to the classifying cyclones. The cyclone cluster will comprise 15 x 380 mm diameter hydrocyclones operating at 100 kPa. The number of operating cyclones will depend on the ore type being treated.

Cyclone overflow, at 35% w/w solids, will flow by gravity to the trash screen. Trash screen oversize will gravitate to a trash collection bin. The trash screen has been located to allow the underflow to gravitate to the 30 m diameter pre-leach thickener feed box or to bypass the thickener and feed the CIL circuit directly. The underflow from the cyclone cluster will be split with part feeding the gravity circuit and the remainder feeding a distribution box. This box will allow part of the cyclone underflow to be returned to the SAG mill if required to balance power draw with the main flow returning to ball mill feed.

### 17.2.7 Gravity

Feed for the gravity circuit will be taken from the hydrocyclone underflow distributor. The gravity circuit feed stream will gravitate to a single vibrating 'degritting' screen to remove coarse (+2 mm) material and fragments of broken mill balls. This oversize will return to ball mill feed. The screen undersize stream will gravitate to a 48 inch centrifugal concentrator. The tails slurry from the centrifugal concentrator will gravitate to the mill discharge hopper. The concentrator will be operated on a semi-batch basis with periodic discharge of the coarse, high SG material (gravity concentrate) to the concentrate storage hopper as part of the intensive cyanidation reactor.

The intensive cyanidation reactor (ICR) will process the concentrate once per day in a rotating drum leach vessel. Cyanide and caustic together with oxygen will be introduced into the slurry and the drum will be rotated for up to 20 hours to leach out gold and silver. At the end of this time the pregnant liquor will be separated from the solids and pumped to the dedicated pregnant liquor tank. Reactor tails will be pumped back to the mill discharge hopper for additional milling to recover any remaining entrained gold and silver.

A dedicated pregnant liquor pump will feed the gravity electrowinning cell in the goldroom with gold and silver recovered onto stainless steel cathodes and barren liquor returned to the pregnant liquor tank. The cathodes from the gravity electrowinning cell will be treated separately to assist in metallurgical accounting. Spent electrolyte will be recycled to the head of the CIL circuit.

### 17.2.8 Mill Area

A davit crane will be provided over the cyclones for maintenance. A seven axis liner handler will be provided for mill liner change outs.

The mill floor slab will be sloped towards a large drive in collection sump at the discharge end of the mills where submersible pumps will return spillage to the mill discharge hopper or tails tank.

The slab below the SAG mill feed chute will incorporate a drive-in sump to allow a front end loader to remove the coarse spillage which can occur in that area when the feed spout is removed. A sump pump will also be provided in this area. Scats from the ball mill will be collected in a drive in scats bunker to facilitate bulk scats removal via a front end loader for disposal. The scats bunker will drain into the mill area drive in collection sump.

### 17.2.9 Trash Screening and Pre-leach Thickening

Trash screen underflow from the grinding circuit will be thickened in a high rate thickener to the nominal leach feed density of 50% w/w solids. Thickener underflow will be pumped to the CIL circuit and thickener overflow will gravitate to the mill water tank.

### 17.2.10 Leach and Carbon Adsorption Circuit

Pre-leach thickener underflow will be pumped to the CIL circuit. The circuit will consist of six tanks interconnected with launders and slurry will flow by gravity through the tank train.

Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing and a mechanically swept woven wire intertank screen to retain the carbon. All tanks will be fitted with bypass facilities to allow any tank to be removed from service for agitator or screen maintenance.

Quicklime added directly to the mill feed conveyor will ensure that the slurry pH is suitable for cyanidation. Sodium cyanide solution will be metered into the CIL feed box and tanks from a ring main system. Oxygen gas from the pressure swing adsorption (PSA) plant will be distributed to all tanks and sparged down the agitator shafts to oxidise any cyanicides and to provide oxygen for the leach.

Fresh / regenerated carbon will be returned to the circuit at CIL Tank 6 and will be advanced counter current to the slurry flow by pumping slurry with a recessed impeller slurry pump and carbon from CIL Tank 6 to CIL Tank 5 and so on. The intertank screen in CIL Tank 5 will retain the carbon whilst allowing the slurry to flow by gravity back to Tank 6. This counter-current process will be repeated until the carbon eventually reaches CIL Tank 1. A recessed impeller pump will be used to transfer slurry to the loaded carbon recovery screen mounted above the acid wash column in the elution circuit. The carbon will be washed and dewatered on the recovery screen prior to reporting to the acid wash column. The associated slurry and wash water will return to CIL Tank 1.

Slurry from the last CIL tank (CIL tails) will gravitate to the carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. Screen underflow will gravitate to the detoxification circuit. Screen oversize (recovered carbon) will be collected in the fine carbon bin for potential return to the circuit.

Barren carbon returning to the adsorption circuit from the carbon regeneration kiln will be screened on the carbon sizing screen to remove fine carbon and quench water. The sized and regenerated carbon will report directly to CIL Tank 6.

### 17.2.11 Stripping Plant and Goldroom Operations

The following operations will be carried out in the stripping and goldroom areas:

- Acid washing of carbon.
- Cold cyanide washing of carbon as required.
- Stripping of gold from loaded carbon using the split AARL method.
- Electrowinning of gold from pregnant solution.
- Smelting of electrowinning products.

The split AARL stripping circuit will be automated and will contain a separate acid wash and an elution column.

The total carbon movement around the elution circuit on a daily basis will be approximately 15 tonnes, with a solution flow rate in the elution circuit of 2 BV/h (64 $m^3$/h).

***Acid Wash***

Loaded carbon will be received into the 15 tonne capacity acid wash column. During acid washing, a dilute solution of hydrochloric acid will be pumped into the bottom of the column to remove contaminants, predominantly carbonates, from the carbon. After the soak period of 30 minutes has elapsed, the loaded carbon will be rinsed with water. Dilute acid and rinse water will be pumped to the tailings hopper for disposal. Washed carbon from the acid wash column will be pressure transferred from the acid wash column to the elution column and the water will be drained out.

***Cold Cyanide Wash***

A cold cyanide solution (2% caustic, 2% NaCN) will be made up in the cold cyanide wash tank sufficient for two bed volumes of the elution column. The cyanide solution will be circulated through the column and back to the tank for 60 minutes to remove any soluble copper. The solution will then be pushed through the column and into the tank with fresh water. This copper cyanide rich solution will then be pumped to the cyanide destruction tanks for treatment.

***Pre-soak and Elution***

The split AARL elution process will be used to recover gold adsorbed onto carbon recovered from the CIL circuit. Initially lean eluate from the lean eluate tank will be heated to approximately 95°C and pumped into the base of the elution column using the strip solution pump. Sodium hydroxide (NaOH) and sodium cyanide (NaCN) will be pumped from the respective storage tanks and injected into the suction line of the strip solution pump. The loaded carbon will be pre-soaked in the cyanide / caustic solution for 30 minutes to elute gold.

The pregnant eluate will then be rinsed from the carbon by up to ten bed volumes of solution heated to approximately 135°C. The first five bed volumes of the elution will be drawn from the lean eluate tank and directed to the pregnant solution tank for recovery of gold by electrowinning. The last five bed volumes of the elution will be drawn from the treated water tank and will be directed to the lean eluate tank for re-use during the next elution cycle.

Strip solutions will be heated indirectly by diesel fired oil heaters and a heat input heat exchanger. Heat recovered from solution exiting the elution column will be used to pre-heat solution prior to the heat input circuit.

Solution samplers will be provided to collect pregnant and stripped eluant for assay.

***Electrowinning and Goldroom***

Once the elution cycle is completed, recovery of gold and silver by electrowinning will proceed. Direct current will be passed through stainless steel anodes and stainless steel mesh cathodes within the electrowinning cells and electrolytic action will cause the gold and silver in solution to plate out on the cathodes. Three electrowinning cells arranged in parallel will be required. Electrowinning will take approximately 8 to 12 hours. An overhead crane (2 t capacity) will be provided to assist with handling of cathodes and anodes. The cathodes will be washed with high pressure spray water and the gold sludge will be recovered in a vacuum pan filter. The gold sludge filter cake will be dried in ovens and direct smelted with fluxes in an electric induction furnace to produce doré bars.

Fume extraction systems will be provided to remove fumes and dust from the electrowinning cells, calcine ovens, barring furnace and flux mixing area. In addition to this, fresh air fans will be provided to ensure there is adequate ventilation inside the goldroom.

***Site Security***

The goldroom design is based on full security surveillance by a security guard and a second level of surveillance by remote control CCTV cameras with remote viewing and recording facilities. Additional security methods and practices will be used; however, description of these is intentionally excluded from this report and is generally implemented very late in the construction period.

Additional cameras will be located at key locations to maintain surveillance particularly in regard to gravity gold processing and to assist with operational monitoring.

***Carbon Regeneration***

After completion of the elution process, the barren carbon will be transferred to a horizontal diesel-fired regeneration kiln circuit. The carbon will be hydraulically transferred to a dewatering screen prior to entering the feed hopper of the regeneration kiln. In the feed hopper any residual and interstitial water will be drained from the carbon before it enters the kiln.

The carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow effective regeneration to occur. Regenerated carbon from the kiln will discharge to a quench tank. The quenched carbon will be pumped, using a recessed impeller pump, to the carbon sizing screen. Carbon sizing screen oversize will enter the CIL tanks and screen undersize will join the CIL tailings flow which will pass over the carbon safety screens.

### 17.2.12 Tailings Treatment

***Cyanide Destruction***

The Project is committed to meeting or exceeding the ICMC requirements. An $SO_2$/ oxygen cyanide destruction circuit will be utilised to meet this requirement. The $SO_2$ / oxygen destruction circuit will reduce the weak acid dissociable cyanide ($CN_{WAD}$) in the slurry discharged from the CIL circuit to less than 50 mg/L prior to pumping to the TSF.

The cyanide destruction circuit will consist of two tanks providing 1.6 hours residence time. The tanks will be interconnected with launders to allow the circuit to be run in parallel or series.

Underflow from the CIL circuit carbon safety screen and the cold cyanide copper wash from elution will gravitate to the cyanide destruction circuit. Copper sulphate and sodium metabisulphite (SMBS) solutions will be added to provide the required copper and sulphur dioxide for the cyanide destruction process. Oxygen from the PSA plant will be sparged down the shafts of the cyanide destruction agitators to provide oxygen to the slurry. Provision will be made for caustic solution to be added to maintain a slurry pH 8.0 to 9.0.

***Arsenic Precipitation***

Some ores contain soluble arsenic which will be treated and stabilised before discharge to the TSF. The arsenic precipitation and stabilisation reaction involves the use of ferrous sulphate. The reaction is rapid and a single tank of 45 minutes residence time will be provided. Discharge from the cyanide destruction circuit will gravitate to the arsenic precipitation circuit. Ferrous sulphate and sulphuric acid will be added in the presence of oxygen. The arsenic in solution will be oxidised to ferric arsenate.

### 17.2.13 Tails Disposal

Arsenic precipitation tails and other miscellaneous waste streams from the process plant will be combined in the tails hopper and pumped to the TSF.

Tailings will be deposited into the TSF using a peripheral discharge method. Excess water will be recovered using a decant method. Cyclic spigot deposition at various locations will be used to allow consolidation and drying of deposited material into beaches to direct supernatant water to a pond around the decant tower.

### 17.2.14 Decant Return

Supernatant water will be recovered from the TSF and returned as process water for the plant.

### 17.2.15 Reagents

***Quicklime***

Quicklime will be delivered to site in bulk tankers and pneumatically transferred into the silo. Quicklime will be metered via a rotary valve directly onto the mill feed conveyor for circuit pH control. An emergency lime addition system for bulk bags will include an enclosed bag breaker and hopper with a rotary valve. A dust collector will minimise dust emissions during loading of quicklime into the storage silo.

***Cyanide***

Cyanide will be delivered to site in one tonne boxes containing bulka bags of cyanide briquettes. The briquettes will be added to the cyanide mixing tank using an electric hoist and enclosed bag breaker and dissolved in process water to achieve the required solution strength.

The cyanide solution will be transferred to the storage tank for use in the process. Cyanide will be reticulated to the CIL circuit via a ring main and dosed to the CIL tanks as required. A dedicated pump will provide cyanide solution to the elution circuit and intensive leach reactor as required.

***Caustic Soda***

Caustic soda (sodium hydroxide) will be delivered to site in bulk bags of 'pearl' pellets. Caustic bags will be added to the mixing tank by electric hoist via a bag breaker and screw feeder on the receiving hopper and dissolved in raw water to the required solution strength.

The caustic solution will be pumped to the elution circuit and intensive leach reactor as required, with separate dedicated pumps for the cyanide destruction and arsenic precipitation facilities.

***Hydrochloric Acid***

Concentrated hydrochloric acid (32% w/w) will be delivered to site in 1,000 L isotainers. The concentrated hydrochloric acid will be pumped into the acid mixing / storage tank where it will be diluted with the correct quantity of raw water to achieve the required acid wash concentration.

The dilute acid solution will be pumped to the elution circuit as required.

***Activated Carbon***

Activated carbon will be delivered to site in 500 kg bulk bags. Carbon will be added to the carbon quench tank as required for carbon make-up to the CIL inventory. Carbon will be added directly to the last adsorption tank, as required, for carbon make-up or via the regeneration kiln to allow fines removal on the sizing screen.

***Grinding Media***

Grinding balls will be delivered to site in bulk. Balls will be charged to the SAG and ball mill as required to achieve the target power draw.

Grinding balls will be charged to the SAG mill via the SAG mill feed conveyor. Grinding balls will be charged to the ball mill by loading the balls using a fork lift with a hydraulic drum tipper attachment into ball charging kibbles and lifting the kibble via a hoist into the ball mill feed chute.

***Flocculant***

Flocculant will be delivered to site in 750 kg bulk bags. Flocculant will be added to the flocculant plant storage hopper using an electric hoist and bag breaker. The vendor supplied package flocculant mixing plant will automatically mix batches of flocculant with filtered water and transfer the mixed flocculant to a separate storage tank after each mixing cycle is complete.

The flocculant solution will be pumped to the pre-leach thickener and intensive leach reactor as required.

***Sodium Metabisulphite***

SMBS powder will be delivered in bulk bags and will be mixed with filtered water and transferred to a storage tank. SMBS solution will be metered to the cyanide destruction circuit by dosing pumps as required to meet ICMC requirements.

***Copper Sulphate***

Copper sulphate will be delivered in 25 kg bags and will be mixed with filtered water. Copper sulphate solution will be metered to the cyanide destruction circuit by dosing pumps as required to meet ICMC requirements.

***Ferrous Sulphate***

Ferrous sulphate will be delivered in 25 kg bags and will be mixed with filtered water. Ferrous sulphate solution will be metered to the arsenic precipitation circuit by dosing pumps as required.

***Sulphuric Acid***

Concentrated sulphuric acid (98% w/w) will be delivered to site in 1,000 L isotainers. The concentrated sulphuric acid will be pumped into the arsenic precipitation tank as required.

***Fluxes***

Sodium borate (borax), silica flour, sodium nitrate (nitre) and sodium carbonate (soda ash) will be used as fluxes for gold smelting. The fluxes will be delivered in 25 kg bags and mixed in small quantities with the gold sludge prior to smelting.

***Diesel***

Diesel will be delivered to site by bulk tankers and transferred to one of the two bulk storage tanks. The diesel will be used in the mine, the process plant, for backup generators and to refuel site vehicles.

Diesel may be pumped from the storage tanks to the plant day tank for use in the strip solution heater and carbon regeneration kiln.

***Reagents Storage***

Reagents will be received on site either in bulk (grinding media) or in shipping containers, with a minimum of one months' capacity stored on site, to ensure that supply interruptions due to port, transport or weather delays do not restrict production. Reagent containers will be offloaded from the delivery truck by the site crane and stacked in a lay-down area until required. Empty containers will be returned with the next delivery.

### 17.2.16 Services

***Raw Water***

Water will be pumped from the pit dewatering bores to a surge tank ahead of a treatment plant. Water from this surge tank will be pumped on demand to the plant raw water tank. The raw water tank will have sufficient capacity to minimise the impact of short term supply interruptions. Duty / stand-by water pumps will be provided for the raw water distribution to the plant.

***Fire Water***

Fire water for the process plant will be drawn from the raw water tank. Suctions for other water services fed from the raw water tank will be at an elevated level to ensure a fire water reserve always remains in the raw water tank.

The fire water pumping system will contain:

- an electric jockey pump to maintain fire ring main pressure
- an electric fire water delivery pump to supply fire water at the required pressure and flowrate
- a diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.

Fire hydrants and hose reels will be placed throughout the process plant, fuel storage and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

***Filtered Water***

Filtered water for the process plant will be produced by treating raw water in the filtered water treatment plant. The treatment plant will consist of clarification through flocculant addition, sand filtration, carbon filtration and biocide dosing. Filtered water will be report to the filtered water storage tank and will be distributed to the plant as required using duty / stand-by filtered water pumps.

***Gland Water***

Water from the filtered water storage tank will be distributed as gland service water using duty / stand-by gland water pumps.

***Mill water***

Overflow from the pre-leach thickener will feed the mill water tank. This water will be used for grinding circuit dilution and will be topped up with process water as required.

***Process Water***

Water will be pumped from the TSF decant to the plant process water tank. The plant process water will consist of TSF decant return water and raw water tank overflow. The process water tank will be located so that the raw water tank overflows to the process water tank allowing the process water tank to be kept full at all times.

Duty / stand-by process water pumps will be provided for the plant process water supply. A separate water pump will be provided for fluidisation water supply to the gravity concentrator.

Antiscalant will be added as required to condition the water and reduce fouling of pipelines, spray nozzles and screen decks.

***Bore Water***

Excess bore water from pit dewatering operations will be pumped from the bores to the bore water tank. This will feed a water treatment facility designed to produce water quality sufficient to be discharged into the Cavally River. The bore water tank will have sufficient capacity to minimise the impact of short term supply interruptions. Duty / stand-by bore water pumps will be provided for the bore water feed to the plant.

***Potable Water***

Filtered water will be supplied to the plant potable water treatment plant. The water treatment facility will include micro filtration, ultra-violet sterilisation and chlorination. Potable water will be stored in the plant potable water tank and will be reticulated to the site ablutions, safety showers and other potable water outlets. Transfer pumps will also feed water to a separate camp potable water tank for reticulation. Additional ultra-violet sterilisation units will be installed on outgoing potable water distribution headers.

***Plant and Instrument Air Supply***

Plant and instrument air will be supplied from duty / stand-by air compressors. The air will be filtered and dried before distribution with separate air receivers. A check valve on the instrument air supply will ensure the integrity of instrument air supply such that air from the plant air system serves as a back-up for instrument air, but plant air cannot draw down the instrument air system.

***Oxygen***

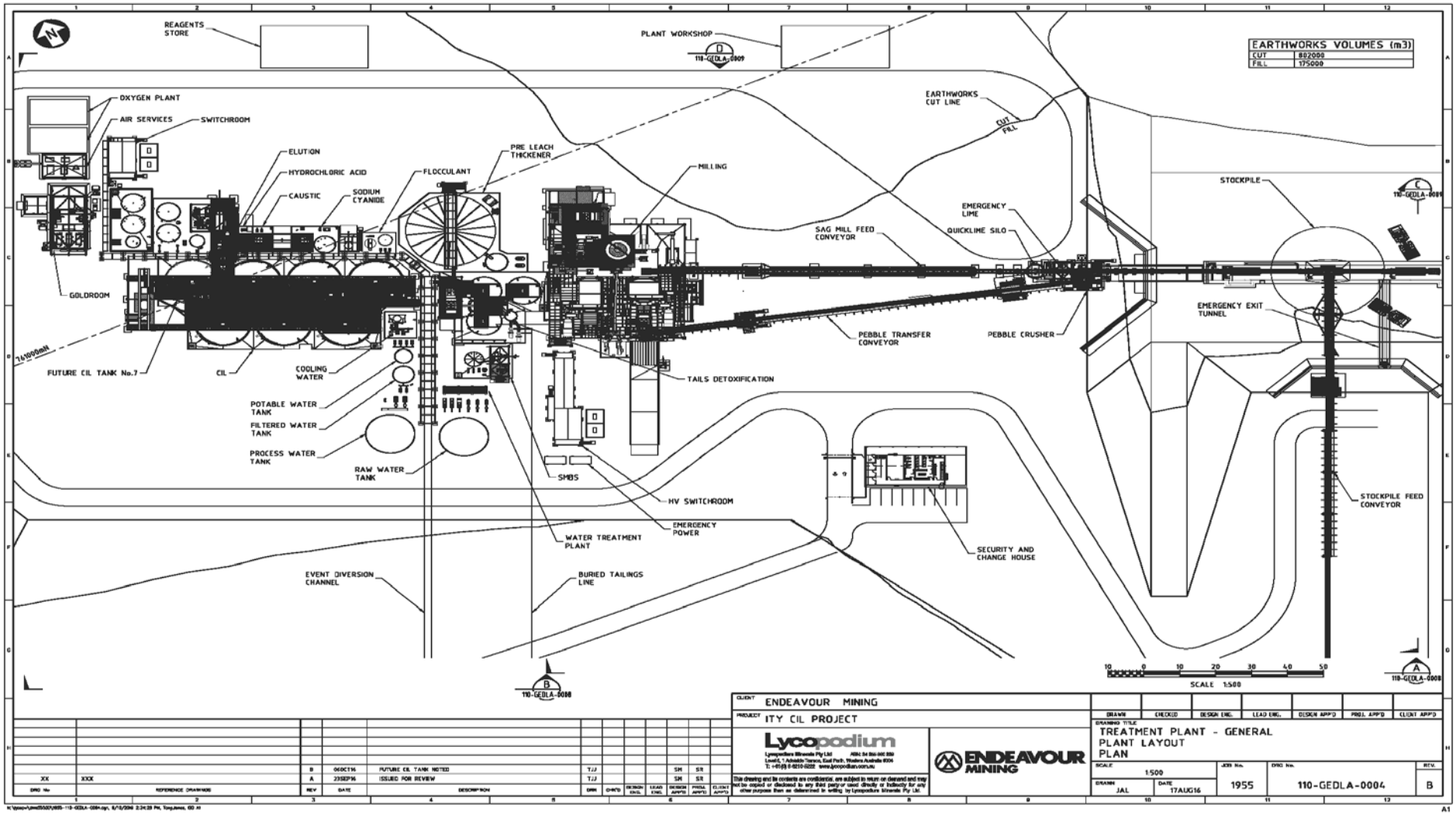
A Pressure Swing Adsorption (PSA) oxygen plant will be installed adjacent to the CIL tanks. The oxygen rich stream (90% $O_2$) will discharge to the oxygen receiver and then be distributed to the CIL tanks and the cyanide destruction tanks. The oxygen plant will be supplied with clean air from dedicated screw compressors. The compressed air will have water and oil removed prior to the PSA unit.

## 17.3 Plant Area Design

### 17.3.1 Introduction

The civil, mechanical and electrical design of the plant facilities is based on industry standard practice and Lycopodium's extensive experience in gold plant design and project implementation in West Africa and elsewhere. Where possible, commonality with Endeavour's Houndé project has been implemented. An overall General Arrangement drawing is provided in Figure 17.3.1. Flowsheet is shown in Figure 17.1.1.

**Figure 17.3.1 Overall General Arrangement**



### 17.3.2 General

The main process plant has been laid out to optimise use of the saddle in which the plant is positioned and to minimise footprint. It is relatively long and narrow with overall dimensions of 320 m x 130 m. To accommodate this, a number of process blocks from the Houndé process plant have been re-positioned. This configuration has resulted in a very compact layout which suits the available site dimensions.

The mechanical design of all process tanks will be in accordance with API 650 with due consideration of the seismicity applicable to the plant location, with appropriate materials selection and corrosion allowances.

Materials handling, containment and bunding in all plant areas has been designed to minimise the possibility of spillages to the environment and to meet legislative requirements.

Due consideration will be given to the nature of the various types of process slurries and reagents being pumped around the plant in terms of settling, wear and scaling.

### 17.3.3 Primary Crushing

The ROM pad will be located just south of the proposed Zia NE pit boundary and north of the existing Ity pit. This location was selected as it provides access for heavy vehicle movements from both the northern pits and the Daapleu pit via the river crossing. It also provides a location for the mine services area. This location will ensure dust picked up by the prevailing winds from the ROM pad will not be blown over other plant or infrastructure.

The primary crushing facility will be built into the ROM pad and will consist of a steel ROM bin fitted with a static grizzly mounted in a concrete vault. The primary apron feeder mounted below the bin will discharge onto the vibrating grizzly with oversize feeding the jaw crusher. The jaw crusher will be supported on a concrete platform over the crusher discharge conveyor.

A crusher maintenance hoist will be provided. Replaceable abrasion resistant linings will be provided in all chutes and conveyor skirtboards at all wear points. A concrete sump suitable for a submersible pump will be supplied to pump out rainfall and spillage that has been hosed to the sump.

A pulsed jet insertable dust collector will return dust to the discharge conveyor. Dust suppression sprays will also be used to control fugitive dust emissions.

### 17.3.4 Ore transfer and Stockpile

The local terrain has dictated that the plant site is approximately 800 m north-west of the primary crusher. A short 1,200 mm wide crusher discharge conveyor will exit the crusher vault to a transfer point. A self cleaning tramp magnet will be suspended over this transfer to pick up any steel from the crushed ore belt. The main ore transfer conveyor will be a low-level 1,200 mm wide conveyor approximately 700 m long which will run north west following the contours south of the decommissioned Aires leach pad to a transfer point 150 m south of the coarse ore stockpile where the ore will discharge onto the 1,200 mm wide stockpile feed conveyor. All conveyors will be fitted with pullwires and drift switches to ensure safe operation.

The coarse ore stockpile will be located over a concrete vault with two means of egress. Earthworks will be built up around the vault to provide a working platform for ore reclaim. The live stockpile will have 12 hours storage and will feed a single variable speed reclaim apron feeder which will be mounted above the SAG mill feed conveyor. Additional stockpile capacity can be created by pushing dead material away from the stockpile. A further two reclaim chutes will be provided outside the live stockpile to provide reclaim capability for stored material. These will be reclaimed by dedicated apron feeders providing a total of three feeders mounted over the SAG mill feed conveyor and SAG mill balls will be charged via these feeders on to the SAG mill feed conveyor. A lime silo will be provided on the mill feed conveyor. This will be filled pneumatically from bulk tankers. An emergency lime system will also be located adjacent to this using an electric hoist to lift one tonne bulka bags to the bag breaker on top of the hopper.

Forced ventilation will be provided for the reclaim tunnel. A sump pump will be located at the discharge end of the tunnel and will pick up spillage and rainfall from the lime area and pebble crushing area.

### 17.3.5 Grinding, Classification and Gravity Circuits

The grinding, classification and gravity separation circuits will be combined in a concrete and steel structure. Mill operations and maintenance will be catered for from a suspended concrete slab. The mills will be fitted with steel liners so a seven axis mill liner handler will be provided for mill relining.

Access to the main mill level platform will be via two stairways from ground level, one of these will also provide access up to the gravity circuit floor and screen / cyclone floor. Intermediate operating levels including the mill feed conveyor head platform and cyclone / trash screen platform may also be reached via a stairway from the ball mill discharge level. Secondary access will also be possible via the SAG mill feed conveyor walkway. Ball mill ball charging will be performed using a dedicated electric hoist and kibble discharging into the mill feed chute.

Servicing of the two mill drives, the mill discharge screen and cyclone feed pumps will rely on mobile craneage for all lifting requirements, road access for which will be available adjacent to the structure. The cyclone / trash screen platform will be serviced by a one tonne davit crane. Spillage handling will be provided by two sumps, one directly under the SAG mill feed chute and one at the SAG mill discharge. The SAG mill feed sump will be accessed by bobcat while the SAG mill discharge will use a front end loader. A sump pump will collect floor spillage in the vicinity of the discharge end of the mill, bunker drainage water and pump scuttling slurry and solids from the duty / stand-by cyclone feed pumps.

### 17.3.6 Pebble Crushing

Oversize from the pebble screen will discharge onto a single pebble transfer conveyor. In the event that this conveyor trips, an overflow chute will discharge pebbles into the SAG mill discharge drive-in sump where they will be reclaimed by FEL. A self-cleaning tramp metal magnet will be mounted over the conveyor and will remove tramp steel from the pebble belt. An operating platform will be provided for this magnet.

Pebble crushing will be housed in a separate building mounted over the SAG mill feed belt. The pneumatically operated deflector gate will allow pebbles to feed a surge bin or, in the event of fugitive steel being detected by the metal detector, will divert pebble feed to a bypass chute. The bypass chute will be fitted with wear liners and will feature a number of rock boxes to minimise impact before discharging directly onto the SAG mill feed belt.

The pebble surge bin will also have a wear lining and will discharge via a variable speed vibrating feeder into the pebble crusher. The pebble crusher will be mounted on a concrete foundation and will include rubber mounts to minimise transmission of vibration. The crusher will include its own ground-mounted lubrication skid. In addition, a hydraulic pump system will allow raising and lowering of the bowl and automatic tramp relief.

### 17.3.7 Gravity Recovery

Part of the cyclone underflow stream will be diverted to feed a gravity scalping screen mounted on the same floor as the trash screen. The undersize from this screen will feed a single centrifugal concentrator. The centrifugal concentrator will be located inside a secure area complete with mesh screening and a lockable door. Product from the concentrator will gravitate via a fully welded steel pipe to the intensive leach reactor, also located on the ground floor in a secure meshed area with a lockable door. A sump pump will service the leach reactor.

### 17.3.8 Trash Screening and Pre-leach Thickening

The trash screen will be located on the milling building upper level to allow height for gravity launder collection. Oversize material from the screen will be diverted to a bunker at ground level for disposal by FEL, undersize will flow by gravity into the pre-leach thickener feed box or directly into the CIL feed box if the thickener is offline.

The 30 m diameter above-ground pre-leach thickener will be located in a bunded area with mill water tank adjacent to grinding and classification. The thickener underflow pumps will serve as leach feed pumps and will be located outside the thickener wall for ease of maintenance and mobile crane access.

Two methods of access will be provided. The first access is directly from the cyanide destruction area and the second via a stairwell on the northern side of the thickener which will allow access from ground level to the thickener bridge. The concrete slab will be drained to a floor area sump pump for spillage collection. In the case of a thickener emergency dump, the bund will fill and overflow to the main plant drain running west to east and into the tailings storage facility.

### 17.3.9 Leach and Carbon Adsorption Circuit

The CIL plant will consist of six 2,900 m³ agitated tanks. A CIL feed box and launder diverter gates will be located at top of tank platform level. The carbon recovery screen and will also be located at top of tank level.

There will be two means of access to the top of tanks, stairway from ground level at the eastern end and a stairway through the elution area.

The top of tank equipment will be serviced by the CIL gantry crane. A 16 tonne crane will be provided for this and other maintenance duties. The intertank screens, carbon transfer pumps and launder gates will be located below the platform steel to reduce potential trip-hazards on the platform area above the tanks. Removable grating panels will provide safe access to platform levels below the main floor to inspect equipment and temporary handrails will make the CIL platform area safe.

The CIL tanks will be bunded for spillage and splash, but not for containment of a tank volume. Accidental or deliberate dumping of the contents of a CIL tank will cause the bund to overflow to the main plant drain and into the TSF compliant with ICMC guidelines. Two floor area sump pumps will be located in the bund at accessible positions adjacent the bund wall to collect and return spillage to the circuit. The CIL bund floor will slope to each of these sump positions.

### 17.3.10 Elution, Carbon Regeneration and Goldroom

The elution circuit will be located on the northern side of the CIL tank train adjacent to Tank 1. The circuit will be housed in a building supported directly off the ground and will include the loaded carbon screen and diesel tank on the top floor and the acid wash and elution columns running to the ground. A stairwell will run up this building with access at the bottom of the columns, the column manholes, the top of column valves and filters and the loaded carbon screen. The loaded carbon screen floor will lead directly onto the CIL top of tank floor. The column manhole floor will also provide access to the regeneration kiln.

The top floor will be under roof and wall cladding to protect the carbon screen. The regeneration kiln will also be protected to shield the shell from differential temperature changes and to keep instrumentation and control panels sheltered.

The carbon regeneration kiln and quench tank will be fitted with an overhead monorail hoist used regularly for new carbon addition but will also be available for maintenance of the kiln drive, shell and rollers. Additional lifting facilities will not be required in this area as maintenance of other equipment will be performed in situ.

The acid wash column will have a partitioned area of the area bund, separated such that any acid spillage cannot mix with cyanide solutions and will be separately disposed of by a dedicated sump pump.

The goldroom building and eluate tanks will be located inside a secure area for restricted access by authorised personnel and vehicles only. The secure area will have separate personnel and vehicle access gates. The strongroom will be enclosed by additional secure walls and roof to provide additional security. The building will be ventilated with fresh air, separate extraction of dust, fumes and heat from the smelting furnace and electrowinning cells. There will be stairway access to the upper level as well as an alternative emergency egress.

The electrowinning cells and anode support frame will be located on the upper level and serviced by an overhead gantry crane. The pregnant solution tank and strip solution tank will be located within a concrete containment bund.

### 17.3.11 Tailings Disposal

The carbon safety screen will be located on a steel structure adjacent to and below CIL Tank No 6 to allow for gravity feed to the screen. Access to the screen will be via a stairway from the top of CIL or ground level. The cyanide destruction tanks are located next to the screen. Screen undersize will gravity flow to the cyanide destruction feed launder whereas oversize will be diverted into the fine carbon bin situated at the base of the structure. The cyanide destruction tanks and arsenic precipitation tank are all located in the same area to allow gravity flow and to allow any of the tanks to be bypassed. The tails hopper has been located between the three tanks to allow bypass to tails in the event of a breakdown. A concrete slab and bund will contain any spillage and direct overflow to the main plant drain and TSF.

The tailings pumps will transport the tailings slurry via an HDPE pipe approximately 1.5 kilometres to the discharge point of the tailings storage facility located to the south of the plant.

All equipment will be readily accessible by mobile crane to conduct maintenance.

### 17.3.12 Reagents

Two reagent mixing areas will be provided. The reagent storage / mixing area for CIL and elution will be located on the western side of the CIL tanks. Hydrochloric acid, caustic and the flocculant mixing plant will be grouped on one common bunded area with appropriate partitioning for floor spillage, whilst a second separate bunded area will be used for sodium cyanide.

The reagent storage / mixing area for cyanide destruction and arsenic precipitation will be located adjacent to the cyanide destruction area on the eastern side of the CIL tanks and will be separately bunded.

***Cyanide***

Sodium cyanide briquettes in one tonne bulka bags will be laid down next to the cyanide mixing plant. A steel structure with access stairs from ground level will house the enclosed bag breaker, a mixing tank and a bag lifting hoist. The structure will be fitted with a roof to provide protection to the bag breaking operation from the weather. Mixed cyanide will be pumped to the cyanide storage tank.

Spillage in the area will be pumped via a sump pump into the leach feed box.

***Caustic Soda***

Caustic soda in bulk bags will be loaded into a bag breaker located on the top of the mixing and storage tank. The cyanide hoist will be used for loading caustic. A common platform for access to cyanide, caustic and hydrochloric acid tanks will be provided.

***Hydrochloric Acid***

The hydrochloric acid facility will be located within a partitioned area of the reagents bund and will have a dedicated sump pump which will dispose of any spillage to the tailings hopper.

Delivery of acid will be via 1,000 L isotainers and unloaded into the mixing and storage tank using a floor-mounted drum pump.

***Flocculant***

Dry flocculant bags will be transported by forklift from the reagents store to the flocculant mixing area adjacent to the pre-leach thickener. The bags will be loaded into a bag breaker located on the top of a small mixing and storage tank. Stair access to the top of tank will be provided.

The bag breaker and dry hopper structure will be partially clad for rain and wind protection.

***Sodium Metabisulphite***

Sodium Metabisulphite (SMBS) powder in one tonne bulka bags will be laid down next to the mixing plant. A steel structure with access stairs from ground level will house the enclosed bag breaker, a mixing tank and a bag lifting hoist. The structure will be fitted with a roof to provide protection to the bag breaking operation from the weather. Mixed SMBS will be pumped to the SMBS storage tank.

Spillage in the area will be pumped via a sump pump into the cyanide destruction tanks.

***Copper Sulphate***

Copper sulphate in 25 kg bags will be loaded manually into a bag breaker located on the top of the mixing tank. The telehandler will locate a pallet of the reagent bags next to the bag breaker. A needle tank will provide temporary feed to the cyanide destruction tanks while a new batch of copper sulphate is being mixed. A common platform for access to SMBS, copper sulphate and ferrous sulphate tanks will be provided.

***Ferrous Sulphate***

Ferrous sulphate in 25 kg bags will be loaded manually into a bag breaker located on the top of the mixing tank. The telehandler will locate a pallet of the reagent bags next to the bag breaker. A needle tank will provide temporary feed to the arsenic precipitation tank while a new batch of ferrous sulphate is being mixed.

### 17.3.13 Air, Oxygen and Water Services

***Air Services***

All compressed air services will be housed in a compressor station at the northern end of the CIL tank train consisting of a fully clad steel structure and concrete slab floor. The building will have one permanently open panel for forklift access should any of the package unit compressors need to be removed for major overhaul. Normal maintenance will be expected to be carried out with the units in situ.

A compact, packaged PSA oxygen plant will be located adjacent to the compressor station. Weather protection is not required.

***Water Services***

Water services include raw water, process water, fire water, potable water, bore water and filtered (non-potable quality). All tanks, pumps and the potable water treatment plant will be located on the southern side of the process plant and will be serviced via a common piperack.

***Filtered / Gland / Potable Water***

The potable water treatment plant will be located on the southern side of the process plant and will also serve the camp. No weather protection or permanent lifting facilities have been allowed for water services, although the potable water treatment plant will be containerised. Gland water will be pumped from the filtered (non-potable) water tank.

***Raw Water***

Raw water will be supplied to the raw water tank from the pit dewatering pumps via a storage tank located near the Daapleu pit. Transfer pumps will feed the plant on demand with excess bore water being treated through a containerised water treatment facility prior to release into the Cavally River.

***Fire Water***

The raw water tank will also hold a volume of 288 m³ of water reserved for firewater use. The fire protection skid of electric with backup diesel pumps and jockey pump will draw from the base of the raw water tank. The raw water distribution pump set will draw from higher up in the tank to prevent depletion of the fire water supply reserve.

***Mill / Process Water***

Mill water consists predominantly of pre-leach thickener overflow with top up from the process water system and is stored in a tank adjacent to the pre-leach thickener. Process water consists of TSF decant return and raw water tank overflow and is used for pressurised sprays, service points and reagent mixing.

***Piperacks***

The following piperacks have been allowed:

- A west-east piperack running from air services and the reagents MCC past the goldroom and joining the CIL piperack.
- A south-north piperack running between the CIL tanks and the reagents / elution areas up to the tailings area.
- A west-east piperack running from the pre-leach thickener, past tailings to the water services area.

### 17.3.14 Large Volume Spillage Containment

The drain under the central west-east piperack will act as the low point for plant spillage. In the event of a large spillage event the grinding area and cyanide destruction area will drain north to south and into a double-lined trench which will drain from the plant east and into the tailings storage facility. Similarly, the elution and CIL areas will drain north to south and into the same lined trench. No separate event pond will be provided due to space constraints.

### 17.3.15 Plant Area Buildings

These are described under Infrastructure and Services, Section 18.

### 17.3.16 Plant Site Drainage

This is described under Infrastructure and Services, Section 18.

## 17.4 Electrical Design

### 17.4.1 Installed Load and Maximum Demand

The plant site-wide electrical power requirements for infrastructure, mining and processing were calculated on the basis of preliminary equipment sizing. The installed load and maximum demand for the Project is summarised below.

**Table 17.4.1 Installed Load and Maximum Demand**

| Installed Load | | Maximum Demand | | Average Demand | |
|---|---|---|---|---|---|
| (MW) | (MVA @ 0.85pf) | (MW) | (MVA @ 0.85pf) | (MW) | (MVA @ 0.85pf) |
| 19.5 | 22.9 | 14.8 | 17.4 | 11.3 | 13.3 |

The maximum demand is based on an average load factor of 80% for all areas except the mill drive, which was given a load factor of 95%. The 4,000 kW SAG mill and 5,600 kW ball mill represents approximately 52% of the total load giving an overall average load factor of 85%. Further details of the plant electrical requirements are included in the electrical load list available on request.

### 17.4.2 Grid Power Supply

Power for the facilities will be sourced from the grid by an extension to the Danane substation and construction of 58 km of 90 kV overhead power line. The Ity substation, owned by CIE, will include a 90/11 kV transformer with an 11 kV feeder taken to the main plant switchboard.

### 17.4.3 11 kV Switchboards

One 11 kV switchboard has been allowed for in the plant and one will be supplied with the grid supply switchyard onsite power plant. The 11 kV switchboards will be fully withdrawable design complete with protection, metering and earthing facilities.

The design fault level and circuit breaker ratings adopted are:

- 11 kV switchboard busbar 2,000 A, 25 kA at 3 sec
- 11 kV circuit breakers 630 A

Protection will be provided by microprocessor based protection relays.

### 17.4.4 Electronic Variable Speed Drives and Soft Starters

LV variable speed drive (VSD) units and soft starter (SS) ratings range from 0.18 kW up to 315 kW. These are floor or wall mounted (dependent upon size) along the internal wall of the LV substation.

### 17.4.5 415 V Motor Control Centre

Three LV switchrooms will be included. The main switchroom will be located adjacent to the grinding areas to minimise cable length to the HV drives. This switchroom will be separated into an HV area for the 11 kV switchboard and an LV area for the 415 V MCCs. The second switchroom will be LV only and will be located adjacent to the elution and CIL areas. The third switchroom will be adjacent to the primary crusher area.

The MCCs will be double-sided (back to back) and housed in the LV switchrooms. Construction of all MCCs will have Form 4 segregation, Type 2 coordination. Starters in MCCs will be of demountable design and main incoming circuit breakers will be of withdraw-able design complete with protection. All motor starters will be equipped with smart overload relays as per Lycopodium engineering standard. The LV MCCs will supply power to the low voltage motors, low voltage variable speed drives and low voltage distribution boards.

### 17.4.6 Earth Fault Protection

Earth leakage protection will be applied to circuits with GPOs (General Purpose Outlets, i.e. power points) and for lighting circuits.

### 17.4.7 Fire Protection

The HV switchroom, LV switchroom and the plant control room will be provided with fire detection systems. Signals from the fire detection system will be wired to the respective fire indication panel (FIP) in the switchrooms and all signals will be monitored by a master fire detection panel (MFIP) in the security / emergency services control room. Each FIP will also be wired to a local siren with beacon to warn staff of the fire detection. The same fire and smoke activation alarm signals detected by the fire detection system will also be monitored in the plant control room.

### 17.4.8 Cable Ladders

Cable ladders will generally be laid horizontally, with vertical ladders used in areas where spillage may occur. Hot dip galvanised / epoxy painted type cable ladder will be used. However, stainless steel cable ladders will be used in areas exposed to corrosive particles / fumes / mists.

Cables of different voltage groups will be installed on separate ladders. If they need to be installed on the same ladder, then complete segregation of the ladders will be provided. Ladder routes will follow the mechanical piperacks.

### 17.4.9 Cables

Direct buried cables will be provided with armouring.

Cables up to 16 $mm^2$ will be PVC insulated and bigger cables will be XLPE insulated.

VSD cables will be multiple core 3 x phase and 3 x earth cables symmetrically laid out within an overall shielded cable.

Cables within the plant area will be installed above ground, on cable ladders and follow the mechanical piperacks wherever possible.

### 17.4.10 Lighting

All lighting around the process plant will be design in a fit for purpose manner to suit the operational requirement for each area.

### 17.4.11 Earthing System and Lightning Protection

The earthing system within the plant will be designed in accordance with relevant Australian Standards (i.e. AS 3000, AS 3007 and Australian Communications Authority (ACA)). The following method of system earthing will be implemented at various voltage levels:

11 kV Earthed via earthing transformers

415 V Solidly earthed system / Multiple Earthed Neutral (MEN) / T-N-C-S

Note: T – Terre (French for earth)

N – Neutral

C – Combined

S – Separate

Lightning protection will be provided for all plant building structures. Plant substations / switchrooms and structural high points will be fitted with lightning masts of sufficient height and quantity to ensure that all exposed points will be covered as per 'Rolling Sphere Method' of AS 1768. Lightning protection systems will have their own independent earthing electrodes and will be interconnected with the power earthing system.

## 17.5 Control System

### 17.5.1 General Overview

The general control philosophy for the plant will be one with an adequate level of automation and remote control facilities. Instrumentation will be provided within the plant to measure and control key process parameters and to provide a safe work environment.

The main control room, which will be located in the plant administration office, will house two PC based operator interface terminals (OIT). Both of the OITs will act as the control system SCADA servers as well as configuration / operator stations. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. An additional OIT will be provided in the crusher cubicle.

All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

The PCS that will be used for this plant is a PLC-based SCADA system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

There will be two modes for loop controlled Control Variables available in the PCS. These are 'Auto Mode' and 'Manual Mode'. In Auto Mode, the Control Variable will be predominantly controlled by the applicable PID loop based on the Process Variable set-point. Auto Mode control may include cascade control loops where the Process Variable set-point is provided by another control loop rather than being directly specified. In Manual Mode, the Control Variable may be entered manually on the SCADA OIT. Both Process Variable set-point and Control Variable set-point can be entered from the control loop pop up on the OIT.

In general, the plant process drives will, as a minimum, report their Ready, Run and Fault status to the PCS system and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will as a minimum, contain start and latch-off-stop (LOS) pushbuttons which will be hard-wired to the drive starter.

The OIT will allow drives / valves to be selected to Auto or Remote or Local modes via the drive / valve control popup on the OIT. Statutory interlocks such as emergency stops and overload protection are hardwired and will apply in all modes of operation.

- **Remote mode**: 'Remote' selection allows drives and valves to be controlled remotely from the OIT. This mode allows operators to start / stop or open / close a drive or valve manually by clicking on the relevant pushbutton on the OIT ('Manual mode') or controlled by an automatic sequence or control loop ('Auto mode'). If not in Remote mode, the drive or valve can only be controlled from local pushbuttons in the field ('Local mode').

- **Auto mode**: When all associated drives or valves are selected to 'Auto', this mode allows for the starting of automatic sequences for some systems, e.g. in the elution circuit, where the system may be group started when all associated drives or valves are selected to 'Auto'.

- **Local mode**: 'Local' selection allows each drive / valve to be operated in the field via a local pushbutton, which is connected to the PLC. All process interlocks will apply.

The control of most of the vendor packages will be done via the local control panel with no control or set-point changes from the PCS system. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally. Digital output can be transmitted from the PCS to the vendor packages where required. The function of these outputs is to enable the start permissive for the packages or to start these packages remotely. For some specific packages (elution heater, compressors) a remote set-point can be entered.

### 17.5.2 Drive Controls

In general, drives will be powered from starters installed in a Motor Control Centre (MCC) located in the low voltage switchrooms. The starter for a DOL drive will be controlled and monitored by a PLC using drive starters and marshalling panels. No 'smart MCCs' will be employed. Field inputs will be wired directly to the PLC and an output will be wired to the start relay of a DOL drive. Each DOL drive will have 'Running', 'Ready' and 'Fault' status displayed on the OIT as a minimum. A start / stop command can be sent from the OIT drive popup to the drive starter via the PLC.

There will be local start / E-stop pushbuttons for each DOL drive, which will be hard-wired to the drive starter circuit. There will be local start / stop / E-stop pushbuttons for each VSD drive, which will also be hardwired to the drive starter circuit. Start and stop pushbuttons will be momentary type and E-stop pushbuttons will be latched mushroom head type.

Variable Speed Drive (VSD) units will be of a Variable Voltage Variable Frequency (VVVF) type utilising Pulse Width Modulated (PWM) technology and mounted within the low voltage switchroom. VSD speed can be controlled via the PCS, in the case of process closed loop circuits, or the VSD faceplate where only manual speed control is required. There will be no provision made for speed control local to the drive.

Exceptions to this drive control philosophy will be drives which form part of vendor packages that are controlled from a vendor supplied local control panel.

Interlocks will be provided to prevent large loads from starting simultaneously. This will be implemented in the PLC program.

The starting of conveyors and other major rotating equipment, such as the mills, will be preceded by a start siren.

The PCS will provide run-time monitoring (run hours) on all equipment (including all equipment with drives larger than 55 kW and the carbon forwarding pumps).

### 17.5.3 Valve Control

Automatic valves will be controlled via the OIT. There will be no local control stations, except for the cyclone feed pumps, pre-leach thickener underflow pumps, tails pumps and elution. Isolation valves and manual control valves will be operated locally with no remote position indication available.

Local control stations will be provided for the valves on the large slurry pumps (the cyclone feed pumps, the pre-leach thickener underflow pumps and the tails pumps) to allow both auto and field selection of Dump / Start / Stop.

There will be a number of local control stations located in the acid wash and elution area to allow for the field control of the elution circuit valves.

Automatic control valves will be controlled by PID loops within the PLC. Manual operation of the valves, as required for maintenance purposes, will be from the OIT and there will be no provision made for local position control. Feedback of automatic control valve position will be available remotely.

Actuated valves for the cyclones will be able to be operated from the control room or locally from the vendor control panel.

### 17.5.4 Vendor Packages

The following equipment will be supplied with dedicated instrumentation and control:

- Primary crusher gap setting adjustment and LRS soft starter.
- SAG mill control including interlock protection, lubrication package, LRS for start-up, VVVF speed control and inching drive. An Ethernet connection to the PCS will provide mill weight, mill speed, mill power, bearing temperatures, gearbox temperatures, lubrication system fault status and grease system fault status. Trending will be provided for these parameters. Mill start-up will be from the field control panel.
- Ball mill including interlock protection, lubrication package, LRS for start-up, and inching drive. An Ethernet connection to the PCS will provide mill power, bearing temperatures, gearbox temperatures, lubrication system fault status and grease system fault status. Trending will be provided for these parameters. Mill start-up will be from the field control panel.
- Pebble crusher gap setting / bowl clamping.
- Self cleaning metal magnet.
- Centrifugal gravity concentrator operation and interlock control. The gravity concentrator will be controlled from a local control station. Interlocking with the ILR vendor package will prevent the gravity concentrator from dumping if the ILR is not ready or the concentrate feed hopper is full via a control permissive.
- Inline leach reactor (ILR) operation and interlock control. The ILR will be controlled from a local control station. The ILR control system will prevent the gravity concentrator from dumping if the ILR is not available or if the concentrate feed hopper is full by setting a control permissive.
- Pre-leach thickener rake drive and torque control.
- Flocculant mixing system.
- Plant and instrument air compressors including instrument air dryers. Remote access to pressure set-point will be provided via Ethernet.
- Stand-alone control of the oxygen PSA plant.

- Carbon regeneration kiln. Trending of the kiln temperatures will be provided on OIT.
- Induction furnace.
- Cell sludge pan filter vacuum system, including the vacuum pump. The cell sludge vacuum pump will be controlled from a local panel. The vacuum system is manually operated.
- Strip solution heater package. Remote access to temperature set-point will be provided via Ethernet.
- Electrowinning cell rectifiers – including local and remote display of voltage and current, and local and remote control of voltage.
- Fire water pump system including electric jockey pump, electric fire water pump and diesel fire water pump.
- Water filtration system.
- Potable water.
- Pit dewatering treatment.
- Sewage pumping stations.
- Sewage treatment system.

The above equipment will have its own sequencing and control provided by a local, stand-alone vendor supplied control system. A general system running and fault status will be reported to the PCS system. Any fault diagnosis will be done locally, except as specified (e.g. the mills).

### 17.5.5 Operator Interface

The main control room will contain two OITs each of which can run the SCADA, along with an OIT station in the crusher cubicle. Depending on access assignments, the OIT in the crusher cubicle will also be able to start and stop equipment in the crushing area. One of the OITs in the main control room will generally be configured for use as an engineering terminal for PCS configuration. Both OITs will be located in the control room in the plant administration building.

The SCADA OIT for the Ity Gold Project will include the following:

- Animated process mimics.
- Drive and valve status display.
- Drive and valve mode selection and start / stop or open / close control.
- Current active alarm display.

- Popup to display analogue ranges and set-points for analogue alarms. Analogue process values on the OIT are displayed using appropriate engineering units.

- Perform trending of all analogue values. The trend displays will enable the display of data from the past 30 days. The operator will be able to change the time scale and magnitude axis for each trend.

- Control loop pop-ups.

### 17.5.6 Emergency Stop Buttons (E Stops) / Safety Devices

Emergency Stop Buttons (E-Stops) will be hardwired into all the drive starter circuits. Every drive will have emergency stop buttons located in proximity to the drive. Emergency stop buttons will be of a latched mushroom head style.

In addition, there will be a number of emergency stop buttons in the plant to stop specific equipment:

- SAG mill – Four E-Stops in the field.

- Ball mill – Four E-Stops in the field.

- Pebble crusher – One E-stop vendor supplied.

- Centrifugal concentrator – One E-Stop vendor supplied.

- ILR - One E-Stop vendor supplied.

- Thickener rake drive control – One E-Stop supplied.

- Carbon regeneration kiln – One E-Stop vendor supplied.

- Feeders and conveyors will be equipped with lanyard type emergency stop pull wires.

- VSDs will be equipped with Soft Stop and Latch Off Stops.

In addition, all conveyors will have belt drift switches and under-speed sensors that will alarm on the OIT when triggered. The trigging of the under-speed sensors will shutdown the conveyor.

### 17.5.7 Cyanide Industry Standards

A number of alarms and interlocks have been included to meet cyanide industry standards.

- The cyanide mixing tank has dual independent level measurement tied to a shutdown of the fill device (water valve or transfer pump). This prevents overflowing of the tank.

- The process water tank has level indication which will alarm if the level rises above a preset point.

### 17.5.8 Control System Communications

Fibre optics will be used for communication between the switchrooms and the control room. Ethernet switches will be used at the plant LV switchrooms and the control room. Ethernet will be used as the communication backbone between PLC racks; and SCADA hardware. Fibre optics will be used to allow communication with the raw water harvesting, raw water storage and tails dam equipment.

### 17.5.9 Control System Access

The control system SCADA defaults to 'view only' access when no user is logged in. Each operator or engineer will be assigned with usernames and passwords in order to operate the plant via the SCADA. Access privileges will be assigned to usernames. For example, the operator user group will have access to drive controls but not to loop tuning. There are four levels of access privilege:

- **Viewers** will have view-only access with no access to drive / valve controls.
- **Operators** will have access to the day-to-day operation functions of the plant.
- **Engineering** users will have access to maintenance and/or diagnostic functions, e.g. loop tuning.
- **Administrators** will have access to all functions and can add additional / remove existing users.

### 17.5.10 Generic Control Description

Where instrument measurements are said to be only displayed locally, there will be no transmission of the signal from the field to the PCS system.

Where instrument measurements are said to be displayed on the OIT, they will first be transmitted to the PLC and then displayed on the OIT. Instrument measurements displayed on the OIT may also be displayed locally on the instrument or local transmitter depending on access.

Where the PCS system is said to control a device, e.g. VSD speed control, based on an instrument measurement, e.g. hopper level, there will be a PID controller programmed in the PLC. The instrument measurement will be displayed on the OIT and a process variable set-point can be entered by the operator on the OIT.

### 17.5.11 SAG Mill Start-up and Monitoring

The SAG mill will be supplied with a standalone control panel. The SAG mill will be controlled from this panel, including all protective interlocks and all control interlocks for start-up and shutdown. All SAG mill status and alarm conditions will be available to the plant PCS via a fibre optic Ethernet connection. The SAG mill will also include the following:

- Mill load measurement calculated from mill bearing lube system hydraulic pressure (output to plant PCS).
- Mill power draw from the HV starter metering module (output to plant PCS).
- Mill feed rate from the SAG mill feed weightometer (output to SCADA).
- An LRS to limit starting torque.
- A variable voltage, variable frequency drive with full speed adjustment (60 – 80% critical speed, output to SCADA). Mill speed display will be available on the SCADA.
- Bearing temperature measurement. Bearing temperatures will be available on the SCADA.
- Mill run hours.
- Locked charge timer.

### 17.5.12 SAG Mill Process Control

The PCS will allow remote control of SAG mill feed rate to a predetermined set-point via the surge bin apron feeders. No allowance for an automatic loop control of feed rate based on mill power or load has been made. A facility for remote adjustment of SAG mill speed from the control room will be provided.

SAG mill density will be controlled by addition of SAG mill feed water, which is proportional to SAG mill feed rate. SAG mill load will be monitored and SAG mill media will be charged manually by a FEL on to the SAG mill feed conveyor to maintain a target mill load. Mill power will be monitored and operations staff will need to adjust speed, charge and/or feed rate to maintain the target power.

### 17.5.13 Ball Mill Start-up and Monitoring

The ball mill will be supplied with a standalone control panel. The ball mill will be controlled from this panel including all protective interlocks and all control interlocks for start-up and shutdown. All ball mill status and alarm conditions will be available to the plant PCS via a fibre optic Ethernet connection. The mill will also include the following:

- Mill power draw from the HV starter metering module (output to plant PCS).
- An LRS for limiting start-up torque.

- Bearing temperature measurement. Bearing temperatures will be available on the SCADA.
- Mill run hours.
- Locked charge timer.

### 17.5.14 Ball Mill Process Control

Ball mill operation will aim to maintain a ball charge that fully utilises the available motor power. If the ball mill is under-utilised because the feed is fine or soft, the ball charge can be allowed to run down to reduce power draw. Generally, grinding variability will be addressed through the SAG mill speed control, as this is more flexible.

Ball mill ball charging will be manual with a kibble for charging media into the ball mill feed chute. A light will indicate to the hoist operator that the kibble is in the correct position for media charging into the ball mill feed chute.

Under normal circumstances, water will not be added to the ball mill feed. Ball mill feed density will be dictated by the cyclone underflow density, however provision to add mill water to the ball mill feed stream will be made, to cater for instances where a high slurry viscosity in the mill could cause the ball charge to 'float'. The ball mill feed water addition is manually adjusted.

### 17.5.15 Mass Flow Rate

Conveyed solids will be measured by weightometer to monitor the crushing and milling rates. Mass flow will be displayed on the OIT. Weightometers will have a 4 - 20 mA output for flow rate and a digital pulsed output for totalised flow.

Volume flow and slurry density will be measured on key streams to monitor the transport of material through the circuit.

The flow and density measurements will be displayed on the OIT. The mass flow rate of key streams will be calculated in the PLC using the measured flow rate and density. This value will also be displayed on the OIT.

### 17.5.16 Density Control

Cyclone feed density will be measured using a nucleonic density gauge and transmitter and displayed on the OIT.

Control of milling water addition will be by a PID loop in the PLC based either on cyclone feed density measurement or on the total milling circuit water addition. The total water addition to the cyclone feed hopper will be calculated in the PLC as the difference between the milling circuit water addition and the SAG mill feed water addition.

Pre-leach thickener underflow density will be measured using a nucleonic density gauge and transmitter and displayed on the OIT.

Tails density will be measured using a nucleonic density gauge and transmitter and displayed on the OIT.

### 17.5.17 Level Control

Level measurement of key hoppers and tanks will be made using ultrasonic level transmitters or differential pressure transmitters (for solutions) and displayed on the OIT. During normal operation PID loops in the PLC will control the level in these hoppers by regulating the discharge pump speed.

### 17.5.18 Flow Control

Flow control of key liquid streams will be monitored by the PLC using flow indicators and controlled by flow control valves or pump speed.

### 17.5.19 Thickener Control

A vendor supplied bed level indicator will be provided on the pre-leach thickener for measurement of the solid / liquid interface. Control of the bed level in the thickener will be by a PID loop in the PLC which will regulate the speed of the flocculant dosing pump.

A vendor supplied pressure indicator will be provided on the pre-leach thickener for measurement of the bed mass. Control of the bed pressure in the thickener will be by a PID loop in the PLC which will regulate the speed of the duty thickener underflow pump.

### 17.5.20 Pebble Crusher Control

The pebble cone crusher will be choke fed where possible to maximise size reduction and minimise overall crusher wear by ensuring choke conditions are maintained. A level detector in the feed canister of the pebble crusher will provide the feedback on the pebble load in the pebble crusher feed canister. Crusher current will also be monitored.

Hydraulic adjustment of the pebble crusher closed side setting will be provided.

The pebble crusher will be protected from tramp metal by one stage of magnetic metal removal using a self cleaning belt magnet on the pebble conveyor followed by a metal detector and feed bypass facility via a flop gate on the discharge of the pebble conveyor feeding the pebble surge bin.

## 17.6 Metallurgical Accounting

Weightometers will be located on the following conveyors throughout the plant:

- Crushed ore transfer conveyor will measure primary crushed ore tonnage.
- SAG mill feed conveyor will measure mill feed tonnes.
- Pebble conveyor will measure the pebbles being recirculated to the SAG mill feed.

- The tonnage of crushed ore reporting to the stockpile can be estimated from the difference between the crushed ore tonnage and the mill feed tonnes.

Manual sampling of the leach feed stream and the final plant tailings will allow composite shift samples for leach head grade and tails solution and residue grades.

Density and flow meters on the leach feed and tailings lines will allow the dry tonnage of solids pumped to the leach circuit and TSF to be determined as a cross check on the mill feed tonnage determined from the mill feed weightometer. In conjunction with the leach feed and plant tails samples, the mass flow measurements will allow the gold recovered in the leach / CIL to be calculated.

A dedicated electrowinning cell will be provided for recovery of the gold leached by intensive cyanidation of gravity concentrate and the recovered gravity gold sludge can be smelted separately. The plant head grade can be back-calculated from the gravity and leach head grade.

Regular gold 'in circuit' surveys will allow reconciliation of precious metals in feed compared to doré production.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the amount of reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates of cyanide, elution and detoxification reagents and diesel flow rates to unit operations will be measured (L/min) and accumulated ($m^3$) using flow meters.

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# 18.0 PROJECT INFRASTRUCTURE

## 18.1 Overall Site Development

The overall site development plan is shown in Figure 18.14.1. The drawing shows the major features of the Project and its infrastructure including the process plant, tailings storage facility, accommodation camp, roads, mine services area, mine open pit and mine waste dump.

The process plant and tailings storage facility are located on the western side of the open pit, just outside the 500 m blast zone. The ROM pad location has been selected for its centralised location close to major pits, as well as proximity to mine services area (MSA). The accommodation camp is located northwest of the process plant.

The main access road approaches the site from the west and the layout provides easy access for personnel and material movements.

The process plant will be fenced to prevent animal access and deter access by unauthorised persons. Monitored high security fencing will surround the process plant. Road access into the fenced area will be through a manned checkpoint. Security fencing will surround the accommodation camp and general site infrastructure.

## 18.2 Roads

### 18.2.1 Main Access Roads

The main access road will continue from the Ity village, with the entrance gate a further 3 km up at the process plant. The road will be sealed from the village to the entrance gate, and unsealed from this point.

### 18.2.2 Plant Roads

Plant internal roads will provide access between the administration area, plant site facilities, fuel storage and mine services area. These roads will generally be 9 m wide and will be constructed flush with the bulk earthworks pad to ensure that storm water sheet flow is achieved across the site, thereby avoiding the need for deep surface drains and culvert crossings within the plant area.

Access to the accommodation camp, located approximately 1 km northwest of the process plant, will be by a 9 m wide gravel road.

### 18.2.3 Access Tracks

A number of new tracks will be constructed to access infrastructure such as the tailings storage facility, sediment control structures and water bore pumps remote from the plant site. The access tracks will be cleared and graded natural earth tracks. These tracks will be constructed to carry mining and large earthmoving trucks. Exact routes will be determined during construction of the Project to best fit local terrain and vegetation density.

### 18.2.4 Site Access Roads

Site access roads were designed to facilitate the following routes:

- Existing public roads to Camp.
- Existing public roads to Plant Site.
- Plant Site to Tailings Storage Facility (TSF).
- Site haul roads to Emulsion Plant.
- Plant Site to mining area.

The site access roads will consist of two 4.5 m wide running lanes, for a total formation width of 9 m. A 200 mm laterite wearing course will be placed over the subgrade / general fill. Turnouts and level spreaders will be located as required along the road alignments to manage rainfall runoff and sediment control.

All culvert crossings for the site access roads were designed for a 1 in 20 year recurrence interval storm event, and will comprise corrugated steel pipe culverts or reinforced concrete box culverts for significant culvert crossings. The culvert inlets and outlets will comprise stone pitched headwalls.

### 18.2.5 Site Haul Roads

Site haul roads were designed to facilitate the following haulage movements during operation:

- Pit to waste dump.
- Pit to ROM Pad.
- Waste dump to TSF.

Waste haulage to the TSF embankments will be via the Ity waste dump, situated immediately downstream of the TSF. The site haul road vertical alignments were designed to balance cut to fill over the entire road length.

The site haul roads will consist of two 13 m wide running lanes, with two 1 m high safety bunds, for a total formation width of 30 m. A 200 mm laterite wearing course will be placed over the subgrade / general fill. Drainage ditches, turnouts and level spreaders will be incorporated into the site haul roads at the detailed design phase. Turnouts and level spreaders will be located as required along the road alignments to manage rainfall runoff and sediment control.

A box culvert crossing will be constructed over the Cavally River, situated within the diversion channel upstream of the Daapleu Pit. The crossing will comprise eight pre-cast concrete box culverts (3 m by 3 m), placed on a base slab, and six bridging slabs. The design width of the crossing is 40 m, to allow inclusion of a services corridor. The culvert crossing was designed for a 1 in 2 year recurrence interval storm event, with a floodway constructed away from the bridge crossing to accommodate flows from storm events up to 1 in 100 year recurrence interval.

All other culvert crossings for the site haul roads were designed for a 1 in 20 year recurrence interval storm event, and will comprise corrugated steel pipe culverts or reinforced concrete box culverts for significant culvert crossings. The culvert inlets and outlets will comprise stone pitched headwalls.

## 18.3 Airstrip

Currently, expatriate employees travel by aircraft from Abidjan to Man, then commute from the Man airport to site via bus. The national airline Air Côte d'Ivoire, currently operates flights from Abidjan to Man on Thursday and Sunday only. While the flight time is approximately 1 hour, the final bus trip from Man to site is limited by at times poor road conditions, with the commute to site a minimum of 2.5 hours.

The option of constructing an airstrip in the project area, if cost effective, will reduce the transit time to site and allow the engagement of a charter operator for direct flights on days and commute times that better suit operational requirements. Direct charter flights will improve security of personnel during the commute, as well as improving site safety and security for personnel evacuation from site if required.

A conceptual design for a site airstrip was completed by Knight Piésold. The detailed design of the site airstrip should be completed once a suitable location is confirmed. The airstrip was designed to Australian Civil Aviation Safety Authority (CASA) guidelines (Ref 1). The prevailing wind direction of the Ity project site is NW to SE (Ref 2). The airstrip will be orientated similarly to ensure optimal operability. In addition, site investigation of the proposed airstrip should be completed during the detailed design phase (once a suitable location is confirmed).

The conceptual airstrip design is summarised as follows:

- The runway running surface is 1,500 m long and 23 m wide. The surrounding runway strip is 90 m wide.
- The airstrip design was completed for a Beechcraft 1900D and King Air 350 as the design aircraft.
- It is likely that cut and fill operations will be required to achieve a design compliant with the CASA guidelines, in particular to achieve required longitudinal profiles.
- Localised widening at the ends of the runway running surface were included in the design, as the design aircraft cannot turn around within the design runway width.
- The apron is 100 m by 40 m and will comfortably park two design aircraft.

- The aircraft trafficked pavement design (runway, taxiway and apron) comprises two pavement layers constructed over a prepared sub-grade.

- External to the trafficked areas, the in situ material will be graded to line and level, and proof rolled.

## 18.4 Water Supply

### 18.4.1 Summary

Site water balance modelling was completed for the Ity Gold Project Plant Site, Tailings Storage Facility (TSF) and pit dewatering (groundwater and runoff), for average, and wet and dry climatic conditions.

Key objectives of the water balance modelling are summarised below:

- Determine supernatant pond volumes for average climatic conditions within the TSF throughout operation.

- Determine supernatant pond volumes within the TSF for design wet rainfall sequences and storm events, check TSF storm water storage capacity and confirm the suitability of the current TSF design philosophy.

- Determine staged embankment crest elevations, to ensure containment of tailings and design supernatant pond volumes.

- Determine the likelihood of recycle water shortfalls during average conditions and design dry rainfall sequences.

- Determine surplus pit dewatering volumes for average conditions and design wet and dry rainfall sequences.

- Assess risk factors for water balance modelling.

The water balance was completed using in-house Microsoft excel-based site-wide water balance model, based on a monthly time step.

### 18.4.2 Modelling Results

For water management modelling, various design rainfall conditions were modelled for selected operational years. The following rainfall sequences were modelled:

- Average conditions.

- 1 in 100 year recurrence interval, one year dry rainfall sequence.

- 1 in 100 year recurrence interval, one year wet rainfall sequence.

- 1 in 100 year recurrence interval, 72 hour duration storm event superimposed over average rainfall sequence (419 mm).

Water balance modelling results are summarised in Table 18.4.1.

**Table 18.4.1 Water Balance Modelling Results Summary**

| | Design Conditions | | | |
|---|---|---|---|---|
| | **Average Rainfall Conditions** | **1 in 100 Dry Rainfall Conditions** | **1 in 100 Wet Rainfall Conditions** | **Average Conditions + 1 in 100 year 72-hour storm** |
| Tailings Storage Facility | | | | |
| Peak Supernatant Pond Volume ($m^3$) (Year 2)[*1] | 199,000 | - | 344,000 | 521,000 |
| Peak Supernatant Pond Volume ($m^3$) (Final Year) | 927,000 | - | 1,400,000 | 1,400,000 |
| Average TSF Recycle Rate – Year 2 (% of water in slurry)[*2] | 68% | 62% | 71% | - |
| Minimum Monthly TSF Recycle Rate – Year 2 (% of water in slurry) | 41% | 41% | 41% | - |
| Maximum Monthly TSF Recycle Rate– Year 2 (% of water in slurry) | 82% | 80% | 83% | - |
| Average TSF Recycle Rate – Final Year (% of water in slurry)[*2] | 81% | 81% | 80% | - |
| Minimum Monthly TSF Recycle Rate – Final Year (% of water in slurry) | 77% | 77% | 76% | - |
| Maximum Monthly TSF Recycle Rate– Final Year (% of water in slurry) | 84% | 84% | 84% | - |
| Plant Site and Water Supply | | | | |
| Required Additional Water Supply[*3,4] ($m^3$) | 62 | 78 | - | - |
| Year 1 Annual Surplus Pit Dewatering Discharge Volume ($m^3$/year) | 1,835,000 | - | 1,971,000 | - |
| Year 1 Average Surplus Pit Dewatering Discharge Rate[*4] ($m^3$/hour) | 229 | - | 246 | - |
| Maximum Annual Surplus Pit Dewatering Discharge Volume ($m^3$/year) | 3,700,000 | - | 3,879,000 | - |
| Maximum Average Surplus Pit Dewatering Discharge Rate[*4] ($m^3$/hour) | 462 | - | 485 | - |

***1 Year 1 comprises November-December only.**

***2 Percentage of total annual volume of water required.**

***3 Available from existing pit dewatering bores post-mining.**

***4 Based on 8,000 hours/year of operation.**

Based on the modelling undertaken, the following conclusions were drawn:

- The TSF is designed to hold the tailings plus the design rainfall conditions, and thus has sufficient storm water storage capacity for all design storm events and rainfall sequences.
- Under average conditions, the TSF water balance is positive after Year 7 of operation (assuming that the pit dewatering continues to be discharged to the environment).
- Under average conditions, the TSF supernatant pond volume in the final 12 months of operation ranges from 238,000 $m^3$ to 927,000 $m^3$ and the water balance remains positive after decommissioning (increasing pond volume). Therefore, the supernatant pond should be removed (and treated if necessary) as soon as practicable after decommissioning.
- Under average and design dry conditions no plant water shortfall occurs at any stage of operation, providing that the Mont Ity pit dewatering bores remain operational subsequent to completion of mining (or an equivalent supply of 62 $m^3$/hour for average conditions and 78 $m^3$/hour for design dry conditions).
- Under average conditions, the maximum annual surplus pit dewatering discharge volume (including pit runoff removal) is 3.7 $Mm^3$, resulting in an average discharge (via treatment system if required) rate of 462 $m^3$/hour. Under design wet conditions, the maximum discharge volume is 3.9 $Mm^3$, resulting in an average discharge (via treatment if required) rate of 485 $m^3$/hour.

## 18.5 Tailings Storage Facility

### 18.5.1 Site Selection

Tailings storage facility (TSF) was designed for a total capacity of 46 million tonnes at an average annual throughput rate of 3 Mt/y. A site selection study was conducted to evaluate several sites relative to the SNC-Lavalin selected site designed for the previous FS. The following sites were evaluated:

- Option 1 – Location as per the previous TSF design (SNC Lavalin).
- Option 2 – Cross-valley TSF located approximately 900 m north-west of the proposed Plant Site, with a water / diversion dam upstream of the TSF (previous arrangement proposed by KP).
- Option 3 – Cross-valley TSF 200 m south-west of the proposed Plant Site, with a water / diversion dam upstream of the TSF.
- Option 4 – Cross-valley TSF located approximately 800 m south of the proposed Plant Site, with a water / diversion dam upstream of the TSF.
- Option 5 – A paddock TSF, approximately 2.9 km west south-west of the proposed Plant Site.

Embankment fill quantities and basin areas were calculated for all options, and estimated costs of these key items for each option were compared as part of the siting study.

In order to rank the TSF site options, each option was evaluated independently by scoring the option against seventeen criteria. Each criterion was assigned a weighting factor according to an assessment of its importance in the overall decision making process.

Based on these results, Option 3 was selected as the preferred option.

### 18.5.2 Design Parameters

Key design parameters for the Tailings Storage Facility (TSF) are summarised in Table 18.5.1.

**Table 18.5.1 Tailings Storage Facility – Key Design Parameters**

| Parameter | Detail |
|---|---|
| Throughput | 3 Mtpa |
| Storage Capacity | |
| • Stage 1 | 4 Mt (16 months) |
| • Stage 8 (Final) | 46 Mt (15.3 years) |
| Freeboard | Greater of:<br>(i) 0.5 m above maximum tailings elevation, or<br>(ii) 1.0 m above maximum required stormwater capacity elevation |
| Stormwater Capacity | |
| • Short duration | 1 in 100 year recurrence interval,24 hour storm event |
| • Long duration | 1 in 100 year recurrence interval,12 months wet rainfall sequence |
| Embankment Crest Width | 6.0 m |
| Side Slopes | |
| • Embankment (U/S) | 2.0H:1V |
| • Embankment (D/S) | 2.75H:1V (intermediate), 3.5H:1V (overall final) |
| Emergency Spillway (operation) | 1 in 100 year recurrence interval storm event, occurring when the supernatant pond is at the spillway inlet elevation. |
| Closure Spillway | Probable Maximum Flood (PMF) storm event. |
| Earthquake Loading | |
| • Closure | Maximum Design Earthquake (MDE) |
| • Operating | Operating Basis Earthquake (OBE) |
| Factors of Safety | |
| • Static (Operation) | 1.3 |
| • Pseudo – static (Operation) | 1.1 |
| • Static (Closure) | 1.5 |
| • Pseudo – static (Closure) | 1.1 |
| Slurry Characteristics | |
| • % Solids | 50% |
| • In situ Density (St. 1) | 1.10 $t/m^3$ |
| • In situ Density (Final) | 1.30 $t/m^3$ |
| • Tailings Beach Slope | 100H:1V (1%) |
| Closure Capping | Engineered fill cover, design to be determined based on field testing and trials conducted during operation. |

No design guidelines were identified for Côte d'Ivoire and therefore the following documents were used in the design:

- Australian National Committee on Large Dams (ANCOLD), 'Guidelines on Tailings Dams', May 2012.
- International Commission on Large Dams (ICOLD), 'Selecting Seismic Parameters for Large Dams. Guidelines, Bulletin 72', 1989.
- International Commission on Large Dams (ICOLD), 'Committee on Seismic Aspects of Dam Design', International Commission on Large Dams, Paris.
- International Commission on Large Dams (ICOLD), 'Tailings Dams and Seismicity – Review and Recommendations, Bulletin 98', 1995.
- Australian National Committee on Large Dams (ANCOLD), 'Guidelines for Design of Dams for Earthquakes', 1998.

### 18.5.3 Design Objectives

Key design objectives for the TSF are as follows:

- Permanent and secure containment of all solid waste materials.
- Maximisation of tailings densities using sub-aerial deposition.
- Removal and reuse of free water.
- Reduction of seepage.
- Containment of design storm events within the TSF.
- Ease of operation.
- Rapid and effective rehabilitation.

### 18.5.4 Design Summary

The TSF valley storage will consist of a single cell formed by three multi-zoned earthfill embankments, comprising a total footprint area (including the basin area) of approximately 77 ha for the Stage 1 TSF increasing to 160 ha for the final TSF. The TSF embankment will be constructed in stages to suit storage requirements.

The Eastern, Southern and Northern embankments comprise an upstream low permeability zone (Zone A), and a downstream structural fill zone (Zone C). Staged embankment crest elevations are presented in Table 18.5.2.

**Table 18.5.2 Staged Embankment Crest Elevations**

| Stage | Tailings Storage (Cumulative) (t) | Embankment Elevation[*1] (m RL) | Maximum Embankment Height (m) |
|---|---|---|---|
| 1[*2] | 4.0 | 273.2 | 11.2 |
| 2 | 7.0 | 277.6 | 15.7 |
| 3 | 10.0 | 281.3 | 19.4 |
| 4 | 13.0 | 284.7 | 22.8 |
| 5 | 16.0 | 287.8 | 26.0 |
| 6 | 19.0 | 290.6 | 28.8 |
| 7 | 22.0 | 293.2 | 31.4 |
| 8 | 25.0 | 295.7 | 34.0 |
| 9 | 28.0 | 298.0 | 36.3 |
| 10 | 31.0 | 300.2 | 38.5 |
| 11 | 34.0 | 302.3 | 40.7 |
| 12 | 37.0 | 304.2 | 42.6 |
| 13 | 40.0 | 306.1 | 44.5 |
| 14 | 43.0 | 307.8 | 46.3 |
| 15 | 46.0 | 309.5 | 48 |

*1 Includes a minimum freeboard and stormwater capacity for the greater of: (i) 1 in 100 year recurrence interval, 24 hour storm event occurring on an average conditions pond, or (ii) 1 in 100 year wet rainfall sequence pond.

*2 Stage 1 embankment designed for 16 month storage capacity.

Primary seepage control from the TSF will comprise a cut off trench excavated into foundation soils to competent foundation material, backfilled with low permeability fill to reduce seepage losses through the embankment foundations. The cut off trench will be constructed directly below the Zone A layer of each embankment, and excavated for the entire embankment length, to limit potential seepage at any level.

The design incorporates an upstream toe drain and basin underdrainage system in low-lying basin areas to reduce pressure head acting on the soil liner, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankment. The underdrainage system comprises a network of collector and finger drains. The toe drain and underdrainage system drain by gravity to a collection sump located at the lowest point in the TSF. Solution recovered from the underdrainage system will be released to the top of the tailings mass via submersible pump, reporting to the supernatant pond.

The construction methodology for the TSF basin preparation will be as follows:

- Underdrainage and borrow areas will be shaped to facilitate full drainage into the TSF, and the final surface graded and compacted to maximise potential runoff.
- External to underdrainage and borrow areas, the TSF basin will be cleared, grubbed and stripped of topsoil.

Supernatant water will be removed from the TSF via submersible pumps (designed by others) located within decant towers. The supernatant pond will be maintained within the northern part of the TSF basin throughout operation. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

As the supernatant pond migrates north, additional towers will be constructed higher up the valley, and raised during operation. Where required, decant towers will have an inlet trench to enable the tower to receive water when the supernatant pond is remote from the tower. Each decant tower will comprise a 25 MPa concrete base and 1.8 m diameter slotted precast concrete pipe sections. The decant tower will be surrounded by free-draining coarse rockfill (Zone G).

An operational emergency spillway will be available at all times during TSF operation, constructed in the Southern or Northern embankment abutment. The closure spillway will be excavated into the valley slopes to the north of the TSF basin, and will be constructed to ensure all rainfall runoff from the TSF will safely discharge upstream of the TSF (and report off site via the upstream diversion channel) after decommissioning of the TSF.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the Northern and Southern embankments, and the western perimeter of the TSF. Deposition points will be required on the Eastern embankment crest to promote supernatant pond control during the dry season. During the early stages of operation, when the risk of water shortages is highest, the deposition plan will be managed to achieve early water recovery. After initial establishment of the tailings beaches, a suitable cycle time will be determined in order to evenly deposit the tailings around the TSF.

A diversion channel will be constructed upstream of the TSF, to reduce the volume of water ponding upstream of the Northern embankment (and thereby reduce the amount of standing water in proximity to local settlements). The diversion channel will be excavated during Stage 1 construction and will be the primary source of embankment fill material for the starter embankments.

A preliminary dam break assessment was completed for the Ity TSF. Modelling indicated that significant settlements (Bietouo and Floleu) are not likely to be impacted in the event of a dam break, with only a small number of existing structures within any of the modelled impact zones. People at risk include permanent residences within the dam break catchment, road users as well as any mine staff completing works on the TSF. As it is difficult to know the frequency and number of people in the impacted area (particularly during operation), a PAR of 10 to 100 was assumed for the Ity TSF. The consequence category of the Ity project TSF is 'High C' (based on ANCOLD guidelines).

## 18.6 Surface Water Management

### 18.6.1 Cavally River Flow Analysis

KP undertook an assessment analysis of the flow of the Cavally River flow at the Ity Gold Mine, namely frequency analyses on daily flow data (from Ity gauge) to estimate the statistical likelihood of experiencing extreme flows at the project site. A large number of different probability distributions were fitted to the annual maxima of the daily flow data. The results of the analysis are provided in Table 18.6.1.

**Table 18.6.1 Extreme Design Flows**

| Annual Exceedance Probability (AEP) | Annual Recurrence Interval (ARI) (y) | Design Flow Rate ($m^3/s$) |
|---|---|---|
| 0.2% | 500 | 585 |
| 1% | 100 | 508 |
| 2% | 50 | 467 |
| 5% | 20 | 404 |
| 10% | 10 | 348 |
| 20% | 5 | 286 |
| 50% | 2 | 189 |

### 18.6.2 Design Summary

KP performed hydraulic modelling using the results of the Cavally River flow analysis to design the following structures:

- Cavally River diversion upstream of Daapleu Pit (including haul road crossings).
- Cavally River upstream of Walter Pit.
- Pit protection bunds for Zia NE, Walter, Gbeitouo and Daapleu pits.

All structures were incorporated in the hydraulic modelling. Flow levels at critical infrastructure locations are summarised in Table 18.6.2.

**Table 18.6.2 Design Flow Levels**

| ARI (Years) | Flow ($m^3/s$) | Maximum Water Surface Elevation (mRL) | | | | | |
|---|---|---|---|---|---|---|---|
| | | Zia NE Pit | Walter Pit | Gbeitouo Pit | Daapleu Pit (Upstream) | Daapleu Pit (Downstream) | Haul Road Crossing |
| 2 | 189 | Dry | 261.1 | Dry | 261.0 | 260.4 | 261.0 |
| 100 | 508 | 263.1 | 263.1 | 263.1 | 263.0 | 262.9 | 263.0 |

The Cavally River Diversion consists of a diversion channel with a 50 m base width upstream of the Daapleu Pit. The maximum diversion channel depth is approximately 10 m, with side slopes 2.5H:1V. The haul road crosses the diversion channel via a box culvert crossing, comprising eight pre-cast concrete box culverts (3 m by 3 m), placed on a base slab, and six bridging slabs. The culvert crossing was designed for a 1 in 2 year recurrence interval storm event, with a floodway constructed away from the bridge crossing to accommodate flows from storm events up to 1 in 100 year recurrence interval. Erosion protection rock fill will be installed within the diversion channel for its entire length, with reno mattress beneath and immediately upstream and downstream of the bridge crossing.

The Cavally River diversion channel upstream of the Walter Pit comprises a channel with a 25 m base, as that is the typical base width of the Cavally river at Ity, based on bathymetric survey data. The maximum diversion channel depth is approximately 8 m, with side slopes 2.5H:1V. Erosion protection will be installed on the pit side of the diversion channel.

Pit protection bunds were designed at the 100 year ARI flood levels (as shown in Table 8.4.2) plus 300 mm freeboard. The bunds will have a crest width of 6 m to allow vehicle access, and side slopes of 2.5H:1V. The protection bunds will be lined with rock fill for erosion protection on the flood side, and topsoil for revegetation on the pit side. In some places, the haul road will run immediately upstream of the pit protection bund at a lower elevation to reduce fill volumes.

## 18.7 Power Supply

ECG Engineering (ECG) has been engaged by Endeavour to investigate power supply options for the CIL Project. While a grid connected 6.6 kV power supply currently connects the existing site heap leach operations to the grid, this will not be adequate to supply the CIL plant power requirements of 25 MW.

Power for the CIL Project will be supplied via a connection to the national grid at Danané, approximately 58 km from the Project site. La Société des Energies de Côte d'Ivoire (CI-ENERGIES) owns the National Interconnected Transmission System (ITS) in Côte d'Ivoire, and Compagnie Ivoiriennne d'Electricite (CIE) manages the electricity generation and transmission network for the Government.

The project will require the following:

- Extension of the Danané Substation by extending the existing 90 kV bus and addition of a 90 kV transmission line feeder.
- Construction of 58 km of 90 kV single circuit lattice tower transmission line.
- Construction a substation at Ity site which will be owned and operated by CIE.

The Project would take a 90 kV tariff metered feeder, install a 90 / 11 kV transformer in the CIE substation and provide an 11 kV feeder to the Plant Main 11 kV Switchboard.

## 18.8 Power Distribution

The electrical system is based on 11 kV distribution and 415 V working voltage. For the process plant the 11 kV supply will be stepped down from 11 kV to 415 V at the main switchroom and distributed to two other switchroom at 415 V. The main switchroom will be separated into a HV area for the 11 kV switchboard and a LV area for the 415 V motor control centres (MCCs). The main switchroom will be adjacent to the grinding area to minimise cable lengths to the HV mill drive. One of other two switchroom will be located adjacent to the elution and CIL areas and the other switchroom will be adjacent to the primary crusher area.

Switchrooms in the crushing area and process plant area will house the 415 V MCCs. Outdoor control panels and distribution boards have been allowed for plant lighting and small power distribution and UPS power distribution.

There are 2 off 1 MVA stand-by emergency diesel generators to supply power to the critical plant process drives and the camp if there is main power supply failure.

The power distribution to the dewatering bores are from two 0.5 MVA diesel generators located close to the majority of the 42 dewatering pumps.

### 18.8.1 Electrical Buildings

To minimise site works electrical equipment will generally be procured pre-installed in prefabricated switchroom buildings:

- HV switchroom for process plant.
- Two LV switchrooms for the crushing and process plant areas.
- A plant control room with the control system equipment pre-installed and installed adjacent to the plant offices.

The electrical buildings will be installed with air-conditioners and suitably sealed to prevent ingress of dust.

### 18.8.2 Earth Fault Protection

Earth leakage protection will be applied to circuits with GPOs (General Purpose Outlets, i.e. power points) and for lighting ts.

### 18.8.3 Fire Protection

The HV switchroom, LV switchroom and the plant control room will be provided with fire detection systems. Signals from the fire detection system will be wired to the respective fire indication panel (FIP) in the switchrooms and all signals will be monitored by a master fire detection panel (MFIP) in the security / emergency services control room. Each FIP will also be wired to a local siren with beacon to warn staff of the fire detection. The same fire and smoke activation alarm signals detected by the fire detection system will also be monitored in the plant control room.

### 18.8.4 Cable Ladders

Cable ladders will generally be laid horizontally, with vertical ladders used in areas where spillage may occur. Hot dip galvanised / epoxy painted type cable ladder will be used. However, stainless steel cable ladders will be used in areas exposed to corrosive particles / fumes / mists.

Cables of different voltage groups will be installed on separate ladders. If they need to be installed on the same ladder, then complete segregation of the ladders will be provided. Ladder routes will follow the mechanical pipe racks.

### 18.8.5 Cables

Direct buried cables will be provided with armouring.

Cables up to 16 $mm^2$ will be PVC insulated and bigger cables will be XLPE insulated.

VSD cables will be multiple core 3 x phase and 3 x earth cables symmetrically laid out within an overall shielded cable.

Cables within the plant area will be installed above ground, on cable ladders and follow the mechanical pipe racks wherever possible.

### 18.8.6 Lighting

All lighting around the process plant will be design in a fit for purpose manner to suit the operational requirement for each area.

## 18.9 Pipelines

The overland HDPE tailings pipelines to and from the TSF will be placed in a lined trench. On sections where these pipes cross under the roads, they will be encased within larger pipes.

All diesel pipelines will be laid on a sand base and wrapped in denso wrap type material for corrosion protection, sealing and water proofing of the pipelines.

## 18.10 Fuel Supply

To minimise site works, fuel storage will comprise a vendor supplied package of two 1,200 $m^3$ double skinned self-bunded fuel storage tanks and pump skids to be located at the MSA. This provides sufficient fuel for the needs of the mining fleet and emergency power for three months if there is any disruption of fuel supply. The requirement to store fuel for the mining fleet is, however, dependent on the final contractual arrangements with the mining contractor.

## 18.11 Sewage and Solid Waste Management

Wastes will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

Waste lubricating oils will be returned to the supplier for recycling.

General solid wastes will be deposited into a landfill at the toe of the mine waste dump and promptly covered to deter vermin and scavengers.

Materials such as cyanide packaging will be decontaminated and buried, under supervision, on site beneath mine waste to prevent unauthorised use.

## 18.12 Explosive Storage and Handling

An emulsion plant for mixing and storage of ANFO (ammonium nitrate / fuel oil) will be established by contract with a reputable supplier. This cost has been included in the explosive supply cost in the operating cost estimate. The facility will be located south of the mine services area and accessed by an unsealed road. The facility will be fully fenced and will include monitored electronic security.

A high explosive magazine will be built and located separately. This will store high explosives and detonators and will include a blast berm and will be fully fenced.

## 18.13 Accommodation

### 18.13.1 Permanent Accommodation Camp

The 200 bed accommodation camp will be located approximately 1 km north west of the process plant and will provide accommodation for salaried and security staff not originating from the local area.

The camp will be constructed from blockwork and will consist of the following:

- Eight management units – two blocks of four units.
- 152 accommodation units – 19 blocks of eight units.
- 40 man fully furnished containerised units – relocated from construction camp.
- Dry mess with food storage and preparation, kitchen and dining facilities and camp administration office.
- Wet mess with a bar, TV area and store room.
- Combined laundry building / gymnasium incorporating ablution facilities for camp staff.

### 18.13.2 Temporary Construction Accommodation

The existing operations camp supplemented with containerised accommodation units will be used for pioneer accommodation pending early completion of the new permanent camp. These containerised units will be moved to the permanent camp once is it operational.

The permanent camp will then be used for Endeavour and EPCM contractor staff with any surplus capacity made available to senior contractor personnel if available.

Contractors will be required to provide their own accommodation for their construction workforce. An area will be set aside for temporary contractor camps adjacent to the construction site. Alternatively some contractors may choose to source temporary accommodation in nearby towns and villages.

## 18.14 Process Plant Facilities

Site buildings will be 'fit for purpose' industrial type structures. The workshop and warehouse will be constructed of a concrete slab on ground with structural steel frame and metal cladding. Offices and amenity buildings will be prefabricated structures.

The process plant and administration facilities are shown in the drawing below.

The following facilities will be located in fenced areas outside the secure process plant area:

- Gatehouse with turnstile and entry boom gate control.
- Main administration building.
- First aid / medical clinic.
- Administration junior staff mess.
- Warehouse and stores.
- Contract laboratory.
- Emergency power generators.
- Transformer / switchyard.
- Fuel storage facility.
- Mine services area.
- Emulsion plant

The following buildings will be located inside the process plant high security area:

- Security access building and change room including laundry.
- Plant offices and control room, training room, junior staff mess and ablutions.
- Plant workshop including small store, welding bay and overhead crane.
- Engineering offices and ablutions.
- Reagent storage area.
- Goldroom.

**Figure 18.14.1 General Overall Site Layout**

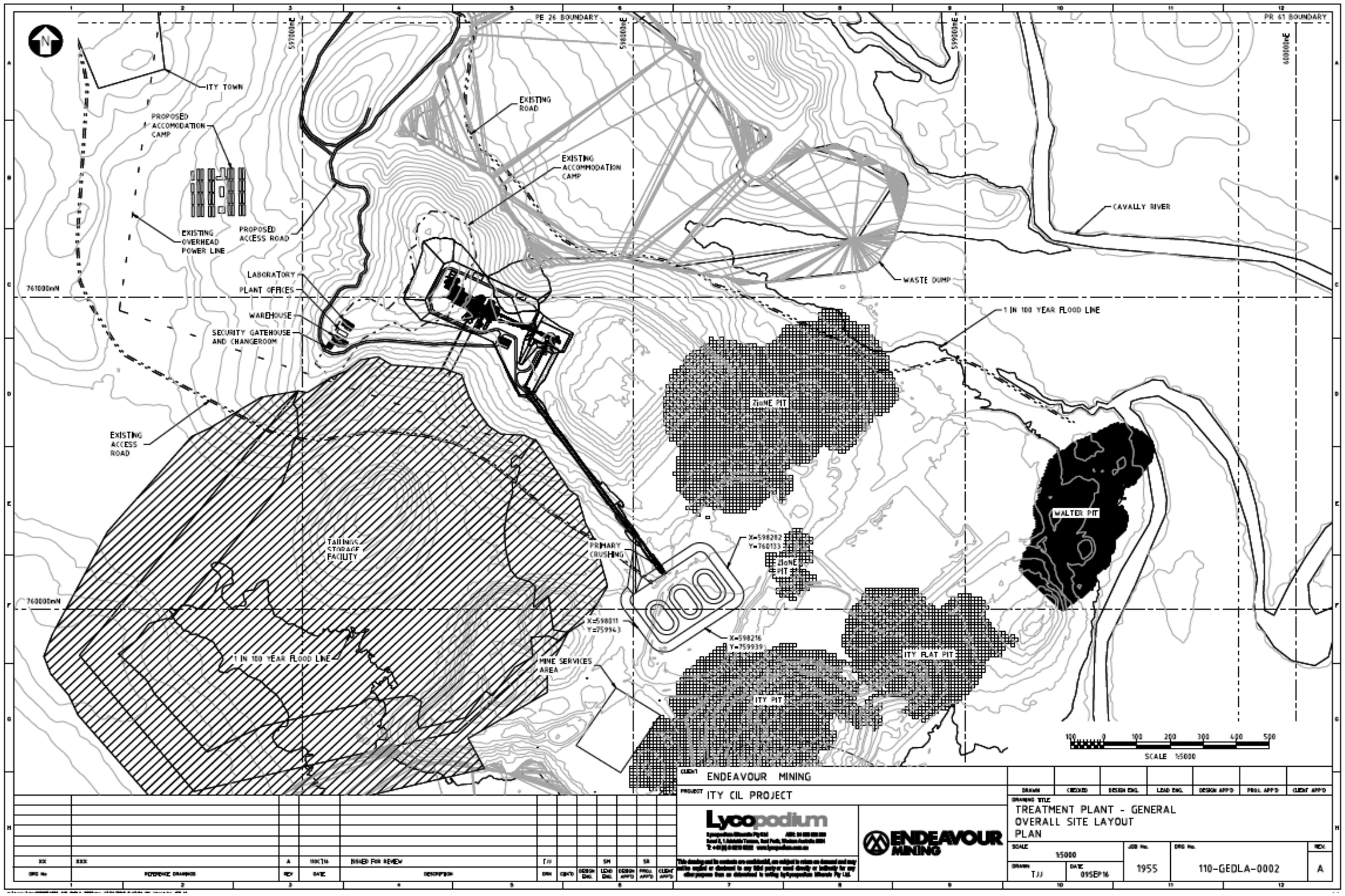
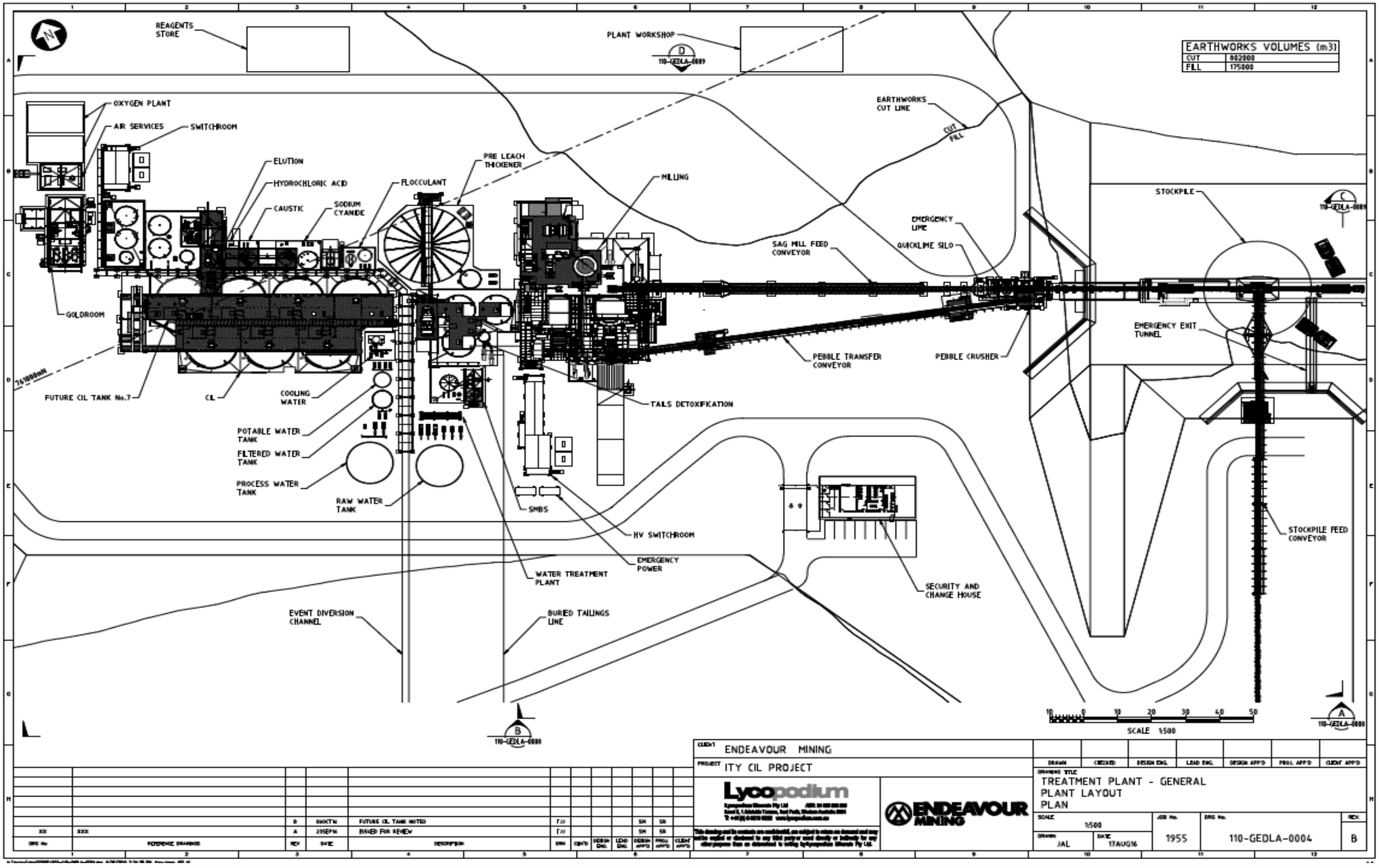


**Figure 18.14.2 General Plant Layout**



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# 19.0 MARKET STUDIES AND CONTRACTS

## 19.1 Market Studies

The Ity operations have been an ongoing concern for approximately 25 years, and additional marketing studies were undertaken for the CIL Feasibility Study. The planned CIL processing will produce gold doré bullion that is a fungible commodity for which an efficient global market exists. It is of high value density meaning that the realised price of the contained gold is insensitive to the ultimate location of the customer and refinery as freight costs are negligible in comparison to contained value.

Refinery terms of 99.9% payable gold in doré bullion and a refining charge of $4.30 per ounce that were used in financial modelling are typical of current terms being offered for CIL produced gold doré bullion.

Gold output in the form of dore bars from the current operations is shipped to Europe (Switzerland), for refining at Metalor Technologies S.A. A refining contract is currently in place and is anticipated to be continued during CIL project operations.

## 19.2 Pricing

Based on review, the long term gold pricing forecast used for the design of the mining project at $1,250 per ounce is consistent with gold prices being used in similar publicly released studies. A common standard is to use the moving averages of the gold price. The five year moving average charts with 60 and 200 day moving averages is shown in Figure 19.2.1.

**Figure 19.2.1 Five Year Chart with 60 and 200 day Moving Averages**



The two year moving average chart is shown in Figure 19.2.2 with 30 an 200 day moving averages and a high of $1,366.

**Figure 19.2.2 Five Year Chart with 60 and 200 day Moving Averages**




The pricing for sensitivity options and economic evaluation utilised $1,000, $1,125, $1,250, $1,375 and $1,500 per ounce to be consistent with recent market fluctuations.

## 19.3 Contracts

No contracts for the sale of CIL production gold have been entered into, however Endeavour has existing contracts in place for current Heap Leach (HL) operations at Ity, and other operating projects in Africa. Gold output in the form of dore bars from the current operations is shipped to Europe (Switzerland), for refining at Metalor Technologies S.A. The refining contract currently in place between SMI and Metalor is valid for an unlimited period of time.

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# 20.0 REQUIRED PERMITS AND ENVIRONMENTAL CONSIDERATIONS

## 20.1 Overview

A number of environmental studies and investigations have been undertaken for the project by a range of consultants and experts including:

- Envitech-CI, Roche (Canada)
- 2D Consulting utilising experts from the Daloa University
- SNC Lavalin (Canada).

The studies commenced in 2013 and culminated in the publishing of the Environmental and Social Impact Assessment (ESIA) Final and Synthetic Report in March 2016. Prior to this, SMI engaged and conducted a number of studies and reports for current heap leach operations from 2000 onwards. These studies led to the development of Environmental Management and Rehabilitation Plans, which have served as the framework for environmental management on site. The Ministry of the Environment has also conducted environmental audits at the Project site.

Endeavour's goal is to adhere to both Côte d'Ivoire and IFC standards for the Social and Environmental activities and reporting associated with the Ity Project.

## 20.2 Environmental Studies and Permitting

Three separate ESIA have been submitted to date with three subsequent environmental permits granted as follows;

- The first ESIA was submitted for approval in September 2013 to cover the renewal of the PE26 mining permit, CIL plant and TSF to be constructed on PE26. This was approved by the Agence Nationale De l'Environnement (National Environment Agency) on 25 October 2013. The first Environment Permit for the CIL project was granted in December 2013.
- A second ESIA was submitted in August 2015, for Daapleu and Gbeitouo mining exploitation and was approved on 11 September, 2015, with the subsequent environmental permit granted in June 2016.
- The third ESIA was submitted in December 2015 to cover mining related infrastructure including bridge, Cavally River diversion, haulage roads and dyke protection bunds. This was approved on January 29, 2016, with the environmental permit granted in July 2016.

SNC Lavalin (Canada) were engaged by 2D Consulting to translate the French language reports and compile a synthetic report from the three ESIA reports in English to comply with international and IFC standards.

A revision will be required for the changes to CIL project layout during the FS, including new location of TSF and process plant, additional Cavally River diversion for the Walter pit (not required until year 4 of CIL operation), camp and access road.

### 20.2.1 Baseline Studies

Baseline studies for the ESIA collected data on the following:

- Climate.
- Air quality.
- Noises and Sound levels.
- Hydrology.
- Hydrobiology.
- Hydrogeology.
- Chemical and biological status of surface and underground waters.
- Fauna and flora.
- Soil classification.
- Archaeology and sacred sites.
- Cadastral surveys.
- Economic and social conditions.
- Community health.

The studies were conducted using appropriate scientific methods with data systematically collected on that basis.

Baseline samples were collected to cover both dry and rainy seasons to account for seasonal changes in climate. This was undertaken as follows:

- March to April 2013 by ENVITECH-CI field teams.
- August 2014, November 2014, February 2015 and August 2015 by Daloa University hydro biological data collecting team.

- June 2014 to January 2015 by 2D Consulting and Daloa University field teams on Daapleu and Gbeitouo deposits.

- August 2015 for ESIA infrastructure report field data collection.

## 20.3 Natural Environment

### 20.3.1 Biological

The biological environment contains a rich flora and fauna diversity. Eight special status flora species have been identified, seven of which are vulnerable and one Near-Threatened, according to the IUCN Red List. Based on existing literature, approximately 50 species of mammals (micro-mammals are excluded) are likely to be found in the study area. Combining the results of interviews with local hunters and of the reconnaissance surveys, the presence of 25 species of mammals in the study area can be confirmed. Five of these have a special status on the IUCN Red List. They are either near vulnerable (four species: the Giant Pangolin, the African white-bellied pangolin, the Hippopotamus and the African bush elephant), or endangered (one species: the chimpanzee).

Field reconnaissance and avifauna prospection were carried out in the project area. 1,545 individuals from 167 bird species distributed among 46 families were identified. These include four threatened species according the IUCN Red List, i.e. a vulnerable species - the Timneh parrot; and three near-threatened species namely the green-tailed bristlebill, the black-headed rufous warbler and the Copper-tailed glossy starling.

Twelve species of amphibians divided into eight genera and seven families were listed during surveys of various habitats in the study area. None of these is recorded as threatened or vulnerable on theIUCN Red List. Three distinct fish survey campaigns were carried out in the Study area in August 2014, November 2014 and May. Globally, considering the results of all survey campaigns carried out, 66 fish species from 35 genera, eight families and nine orders were sampled in the Cavally River in the study area.

Six of the species surveyed are threatened or near threatened according to the IUCN Red List: Two vulnerable species (Brycinus derhami and Chromidotilapia cavalliensis), one species in danger of extinction (Chrysichthys teugelsi) and three near-threatened species (Raiamas nigeriensis, Tilapiawalteri and Micralestes eburneensis).

### 20.3.2 Physical Environment

Côte d'Ivoire and the regions of Cavally and Tonkpi in particular, are subject to an equatorial-type transition climate. This climate is characterised by abundant rainfall, high humidity, high but not excessive temperatures as well as low temperature ranges.

The climate is punctuated by four seasons of unequal durations. In fact, Côte d'Ivoire's rainfall pattern is bimodal with two humid periods called rainy seasons intercalated with two periods of low rainfall called dry seasons. The Project area is located halfway between the mountain climate zone and the Baoulean climate zone.

The Cavally River presents an average modulated flow rate of 177 m³/s (15.29 million m³/day) at its Taï station, while this flow rate is only 43.7 m³/s (3.38 million m³/d) in Flampleu (Girard, 1974) about 40 km north of the current mine site in Ity.

A surface water quality measurement campaign commissioned by SMI was conducted by the firm Envitech in June 2013. Water samples were taken from the Cavally River, as well as wells and boreholes of the river watershed to understand the overall quality of water. It was noted that the Cavally River waters are under reducing conditions characterised by alkaline pH, with water from wells and bores tending to slightly acidic, however within WHO guidelines.

Approximately ten sampling points were identified, including two on the Cavally River. Exceedances of the Ivoirian Standards and/or the WHO Standards, where observed upstream and downstream for turbidity, biological oxygen demand (BOD), arsernic, cadmium, iron, lead and mercury. The baseline noise environment was established using noise measurement data collected in eight villages surrounding the SMI mining facilities. In all the villages, in calm periods, the ambient levels in daytime are higher than the night-time levels, reflecting the usual human activities in the villages during daytime.

### 20.3.3 Human Environment

The human environment is dominated by traditional community structures and a low level of development based on a cash crop and subsistence economy. The project's regional impact area is located in the Mountain Region of Côte d'Ivoire where parts of the sub-prefectures of Zouan-Hounien (Tonkpi region) and Bloléquin (Cavally region) are located.

The Cavally River forms a natural border between the two regions. Six villages are believed to be more likely to be affected, including Daapleu, Tieupleu, Kouègonopleu, Kouizonpleu, Samipleu, Téapkapleu and Gahapié.

Document review as well as surveys and consultations were carried out to produce statistical data on the residents of the direct area of the proposed CIL project for the gold deposits of Gbeitouo, Walter, Zia NE, Ity, Tontouo, and Daapleu in the departments of Bloléquin and Zouan-Hounien.

The Guéré are the indigenous people living in the Cavally region. They are mostly farmers and hunters. The main economic activities in the project area are agriculture and artisanal mining. The main source of household income in the project area is agriculture (68.62%), followed by gold panning practiced by 14.89% of the households surveyed.

## 20.4 Anticipated Environmental Costs – Operation

The anticipated operational costs would normally comprise the following four components:

- Costs associated with maintaining compliance of the operation with environmental legislation, regulations and licenses.
- Costs associated with monitoring according to international codes.
- This costs associated with management of incidental factors.

- Costs associated with community and social development and accommodation of loss of amenity to the community.

The costs have been developed on the following basis:

- Three full time environmental and social professionals will be retained for the project, as well as three environmental technicians.
- These personnel will be responsible for undertaking all monitoring associated with the implementation of the environmental management plan, and as required under conditions of permit and legislation.
- Sample analysis will be undertaken offsite with an accredited provider engaged to undertake both sampling and analysis.
- Field monitoring equipment will be purchased to complement existing facilities from Heap Leach (HL) operations.
- Works required to be undertaken associated with mitigation of amenity loss to the local community will be undertaken under guidelines to be defined in the Community Plan.

The overall costs associated with environmental management during the operational phase are estimated to be US$704,900/year, dependent on annual operation budgets based on the following:

- Staffing costs of $432,850.
- Environmental monitoring and compliance testing allowance of $120,000.
- Community development - $102,000 for community relations expenses, local projects and community scholarships, and based on historical contributions from HL operations.
- Additional allowance of $50,500 for permitting, regulatory inspections and environmental consultants.

These costs have been included in the operating cost general and administration (G&A) charges as described in Section 21.

## 20.5 Social and Community Impact

The mining operations are tightly and firmly linked to the local Yacouba community who live in eight villages that surround the project site. The people work predominantly as farmers and sometimes as artisanal miners and exceptionally as fisherman.

The CIL project area being developed by Endeavour covers two regions, Cavally and Tonkpi. The Cavally River is the natural and administrative line separating the two major communities, with Yacouba in the western part and Guere in the eastern part of the river.

The company has developed a strong social policy to maintain key relationships with the local communities, and has supported this by developing and providing significant local infrastructure over the projects mine life to date. This has allowed SMI to gain the support of local people, and integrate well into the local territories.

A large proportion of the gold deposits to be exploited in the CIL project will be mined from the Daapleu and Gbeitouo resource areas. These deposits are located in the Eastern PR 609 (formerly PR 61) permit area, and in the future Mining License ('PE on demand').

Relationships with communities will at all times be conducted in accordance with the legal recommendations contained in the laws relating to mining activities in Côte d'Ivoire. These laws include the mining, environmental and water codes, supported and guided by recommendations of completed SEIA's and SMI's policies and experiences of social and community management in the region.

### 20.5.1 Requirements and Guidelines for Community Relations in the Mining Law

The new Mining Code (Loin°2014-138 du 24 Mars 2014) and the Application Decree (Décret n° 2014-397 DU 25 Juin 2014) define rules and guidelines for mining companies to manage relationships with local communities. The most relevant sections of the Mining Code relating to community engagement and obligations include the following:

- **Chapter II, articles 121, 122, 123** mandates that "the state of Côte d'Ivoire guaranties respect, protection and management of human rights of local communities affected by mining activities". The state must ensure the societal responsibility of mining companies to respect, protect and promote human rights of local populations.

- **Article 124** recommends mining companies create a Community Development Plan, in consultation with local communities and administrative authorities, with objectives clearly defined and investment plans articulated. Article 124 also states "that the mining company must constitute an annual development fund which will be used to finance the local socio-economical project defined in the Community Development Plan".

- **Article 125** recommends that a Local Mining Development Committee be formed to coordinate social and economic development projects for the benefit of local communities. Mining companies must also "assist the local communities to improve their management capability in order to improve the efficiency in the management of the local fund from mining activities".

- **Chapter 1, Article 128** of the Application Decree of the Mining Code, states that the community development plan should include discussion and plans for:

  - infrastructure and equipment development.
  - social services and 'way of life' development.
  - promotion of local job opportunities.

- Local economic development plan.
- human capital or resources development.

- **Article 129** recommends that the relevant mining company must put in place a 'Local Development Fund' for the villages impacted directly by the mining activities as identified during SEIA studies.
- **Article 130** mandates that the local development fund will be used to finance and develop projects identified exclusively for local communities affected or impacted by the project. These projects must be approved by the Local Development Committee.
- **Article 131** directs that for each mining license or mining operation, 'both the Mining Ministry and the Territorial Administration Ministry must create one Local Mining Development Committee'. The committee's composition will include:
  - Prefect of the Department, **Committee President.**
  - Regional Council President, **Committee Vice-President.**
  - Sub-Prefect, Deputy, Mayors of localities affected.
  - Representatives of local communities and administrations.
  - Mining Company Representative, **Committee Secretary**.
- **Article 132** stipulates that the local fund is created and administered from a primary level bank and withdrawal of money is covered by the duel signatures of both the President of the Committee and the responsible representative of Mining Company.
- **Article 7** from the Fiscal Ordinance signed by the President of Côte d'Ivoire**,** mandates that 0.5% of total revenue from mining activities, with the exception of transportation and refinery costs, must be transferred by mining companies into the local community development fund. This amount is non negotiable.

### 20.5.2 Company and Community Interface Experience

During the last 25 years of operation SMI has developed a strong relationship with the communities living in the villages and department of Zouan-Hounien. SMI has at at all times strived to maintain harmony with the immediate environment and these local communities. The social multi-dimensional actions undertaken by SMI have involved the communities in promoting sustainable development and good welfare.

This social policy has relied upon strong, coordinated relationships and collaborative actions at all times between all local partners and mining management. SMI has instigated the establishment of a local committee to discuss and decide on projects to assist the local community. The committee's President was initially the Sub-Prefect and later the Prefect.

The membership of this development committee includes:

- local and administrative managers
- village managers (cadres and chiefs)
- youth and women's representative
- SMI representative.

SMI has coordinated and financed the committee which agrees and makes decisions on which projects to finance. SMI does not provide funds to people directly, instead funding projects selected by the committee, with finances controlled according to the schedule and budget approved and adopted by the committee. SMI has been successful in utilising local companies to build identified infrastructure projects including:

- schools and infirmaries
- housing for primary school teacher and nurse
- local markets
- wells for potable water
- electricity grid for all eight local villages
- roads
- resettlement of the Ity village
- Ebola prevention programmes.

SMI also provides daily free medical consultations and medicine to more than 80 people in the mine site infirmary. More than 13,000 people are annually received to the mine site infirmary for medical checks and assistance.

SMI also deployed considerable effort during the EBOLA crisis. This proved useful to local communities in monitoring and controlling an individual's temperature, promoting frequent hand washing and transferring people as required to the infirmary located at the mine site main gate. Specific equipment required to monitor the Ebola crisis in the area was paid for by SMI.

### 20.5.3 Financial Contributions to Local Communities

The financial assistance provided by SMI as part the current operation has increased over time since 1998, as follows:

- 50 million FCFA between 1998 and 2006.
- 80 million FCFA between 2006 and 2009.
- 100 million FCFA between 2009 and 2011.
- 200 million FCFA between 2011 and 2015, of which 100 million FCFA went to the eight villages surrounding the mine site, and 100 million for the Zouan-Hounien Department.

In total, SMI has contributed more than 2 billion FCFA (approximately US$4M) to assist the local communities in the eight villages surrounding the mine site to develop basis infrastructure in health, education, trade, transportation and communication.

SMI has also developed a recruitment policy to assist local youth and women to integrate into non-qualified jobs in the mine site, with 50-60% of the mine's current workforce coming from within the region. Local companies are also used on the minesite for building construction, cleaning, cooking of meals, and assisting field geology prospecting teams (especially for geochemistry, geological and geotechnical drillings campaigns).

These actions have resulted in the Côte d'Ivoire Government acknowledging the efforts of SMI with the company finishing second place in 2014, and first place in 2015 in the award of best performing mining company in Côte d'Ivoire.

These achievements have been possible through negotiation and collaboration of issues identified by local people, and efforts to responsibly react to any issues raised. It is evidence that even without an official legislated agreement between mining companies and the Côte d'Ivoire government, companies can develop and integrate plans to improve the surrounding environment.

With the direction of the new mining code and defined international standards, relations between mining companies and local communities are now clearly defined in law, and form the basis for actions to build a mutual and profitable plan for the sustainable development of local communities.

During the mining operations, financing for the Local Development Fund is 0.5% from the annual revenues of mining production for Daapleu and Gbeitouo Deposits, as mandated in Article 7 of the Mining Code Fiscal Ordinance.

The main focus of the local development plan will be to:

- develop basic infrastructure and equipment
- develop basic social services
- promote local employment

- develop local economic activities
- develop human capital or resources.

### 20.5.4 Community Relations Challenges for the Proposed CIL Plant

During 2013, public stakeholder consultations were conducted as part of the SEIA process for the renewal of operating licence PE26. This includes the villages of Ity, Daapleu, Bientouo, Floleu, Trogleu, Kroziale, Ouyatouo and Kouepleu. The results of the public consultation resulted in all villages surrounding the mine site agreeing and supporting the development of the CIL project, provided SMI considers the villages concerns in relation to the following:

- Land compensation provisions.
- Socio-economic infrastructure requirements.
- The creation of new plantations.
- Provision of youth employment.
- SMI financing of new activities.

In eastern Cavally, within areas to be covered by the new 'PE on demand' (forming part of exploration permit PR609 and containing the Daapleu and Gbeitouo deposits), the results of the public consultation conducted during the SEIA also concluded that all stakeholders consulted approved the development of the CIL Project, conditional on concerns of communities being taken into consideration.

The company is currently preparing a Local Development Plan to satisfy and remedy the concerns raised and will conduct mining activities in accordance with the recommendation of the Community Development Plan contained in the new mining code.

The results of the community surveys conducted in 2015 suggested that 89.9% of household heads support the Project while 6.9% are against it, demonstrating that these populations are clearly in general, receptive to the Project. The main reasons for support were:

- camps within the project area will no longer be isolated
- development of economic activities
- development of the camps in the project area
- easier traffic and transportation between Bloléquin and Zouan-Hounien and better access to health centres and evacuation of agricultural products.

The principal objections resulting in 6.9% of the heads of households being opposed to the CIL project was a fear that their plantations, arable lands, forest, sacred sites and cemeteries would be destroyed causing famine and unemployment.

Although the majority of the populations support the implementation of the Project, they have certain expectations concerning the improvement of their quality of life if they have to be relocated and will require compensation for loss of land and sacred places. As a result social management must be responsible, maintain and to improve sustainable relationships with the human, physical and social environment

The skills and experience required to manage this type of project exist to ensure the projects success. Endeavour has successfully implemented similar projects in West Africa, and has the experience required for successful resettlement of effected peoples.

## 20.6 Anticipated Land Acquisition and Relocation Costs

The Daapleu village and other associated camps will need to be relocated to a new site at Kouinzompleu, a resettlement operation which will be a key challenge to the new project's success. The key challenge for the project is to define the current ownership of the land comprising the Daapleu village, located on the eastern part of the Cavally River. The Daapleu villagers are Yacouba peoples, however land on the eastern side of the Cavally belongs to the Guere, with the Cavally River being the administrative boundary between the two regions.

As well as defining ownership of land, definition is required on which communities (either Yacouba or Guere or both) the company must engage with in negotiations for land compensation prior to project commencement. The successful resolution and definition of ownership is critical for ensuring harmony and reducing possible schedule impacts for project commencement.

The recommendation of social experts during the SEIA field survey was to manage this issue in consultation with local and administrative leaders. Ownership has not been clarified at this point, and will need to be prior to the CIL project commencing.

SMI is and will continue to engage all relevant local and national authorities to successfully manage these issues. This will include the different Prefects and Sub-Prefects, local and community leaders, local elected persons, as well as expert consultants.

Once issues relating to land ownership and negotiation for compensation are defined and understood, other requirements that flow from this will be clearer and easier to manage appropriately. This will include the following:

- Sacrifices and ritual actions required before transfer of people.
- Sacrifices and ritual actions required before transfer of cemeteries and human remains.
- Creation of the local committee for resettlement management.
- Creation of the local committee for cemeteries transfer.
- Payment for acquired plantations (agricultural compensation).
- Payment for land acquired (land compensation).

- Payment for local traders and local jobs that will be lost.
- Land acquisition for the new village site of Kouizompleu (14.5 ha).
- Tendering and construction of 80 new homes and supply of required equipment.
- New cemetery and transfer of human remains from different camps.
- Relocation of 401 people already identified for relocation.
- New land acquisition for compensation of destroyed plantations for Daapleu and closed camps and legal right of the new areas.
- New land acquisition for compensation for Ity former village after 2006-2007 resettlement and legal right of the new areas.
- Payment for transfer of effected people.

The cost of relocation and resettlement is estimated at US$2.4M, and is based on agreements between Endeavour, administrative authorities and the local community.

A Resettlement Action Plan (RAP) has been developed by Endeavour.

## 20.7 Anticipated Costs - Closure

A provision of US$5M has been included in the economic model for rehabilitation and closure. The biggest single cost will be the cost of closure of the tailings storage facilities.

The closure cost provision does not include provision for ongoing monitoring that may be required after closure and rehabilitation works have been completed. It is expected these could require ongoing monitoring of US$120,000/y, if required by the regulatory authorities prior to acceptance of relinquishment.

The objective of the mine closure will be to provide landforms that can be utilised by the community without liability to the community to generate an ecologically sustainable landscape post-mining. The present dependence of the community on cash crops and subsistence farming suggests that the closed mine should provide landforms which are capable of being grazed by animals or supporting annual or perennial crops. Water storage facilities should be retained for use by the community.

Buildings, infrastructure and access roads will be offered to the local community for use, and materials suitable for recycling will be sold.

All contaminated land will be remediated either by placement of a soil / rock cover system over the area (such as with the TSF) or through removal of the contaminated material to a designated landfill area and replacement of the soil with rock waste and a topsoil cover.

Waste rock dumps will be battered down and surfaces covered with topsoil collected during the construction and operations phase. Waste dumps will then be reseeded with local plant species to provide both a vegetated cover to control wind and water erosion and support any beneficial uses identified by the local community.

The pit voids will be left open but surrounded by a protective bund, outside the zone of instability, to ensure entry by vehicles will be hindered.

A new mining code adopted by the Parliament of Côte d'Ivoire in March 2014, requires mining companies to set up a bank account at of the beginning of the mining phase in order to cover environmental costs for rehabilitation and closure. This is analogous to a rehabilitation and reclamation bond. The fund is calculated on total mining material movement in the mine, with the rate applied on a $/t mined to be determined by agreement between the Government and Endeavour.

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# 21.0 CAPITAL AND OPERATING COST

## 21.1 Operating Cost Estimates

The operating cost estimate has been prepared with contributions as follows:

- Mining cost modelling by Calsta Pty Ltd (Calsta) for Mining Solutions.
- Process cost modelling and General and Administration (G&A) by Lycopodium.
- Contributions from Endeavour including salaries and current contracted reagent costs, as noted.
- Consumable prices from supplier budget quotations or the Lycopodium database.
- Crushing and grinding energy and consumables, using ore characteristics measured during the testwork.
- Reagent consumption and gold extractions based on laboratory testwork results.
- First principle estimates based on typical operating data.

The operating cost estimates are presented at summary and detail level. The estimate is considered to have an accuracy of ±15%, is presented in US dollars (US$) and is based on prices obtained during the third quarter of 2016 (3Q16).

### 21.1.1 Mining Cost Estimates

***Mine Operating Basis***

Mining operating costs were built-up on the assumption that all mining would be under an Owner Mining basis. Operating costs have been assigned as all ore and waste mined prior to and subsequent to ore processing. Table 21.1.1 provides the basis for the mining operating costs.

**Table 21.1.1 Mine Operating Cost Basis**

| Cost Area | Usage | Cost Basis |
|---|---|---|
| Direct Labour | Operator and maintenance personnel based on the equipment numbers related to assigned shifts as previously discussed. | Base salaries and on-costs determined by Endeavour. Additional allowances for operating consumables for category. |
| Indirect Labour | Quantities as previously discussed. | Base salaries and on-costs determined by Endeavour. Additional allowances for operating consumables for category. |
| Equipment maintenance parts and OEM labour | Based on life cycle accumulative hours matched to hours achieved in the schedule. | Life cycle rates for all major components and consumables as provided by OEMs. |
| Fuel | Quantity based on estimated fuel burn rates for respective mobile equipment types and duties. | Allocated $/L fuel rate ($1.00/L). |
| Tyres | Based on estimated tyre life for respective mobile equipment types and duties. | Respective unit tyre costs as provided by tyre suppliers. |
| GET | Ground Engaging Tools (GET) based on expected life, material abrasiveness and level of blasting. | GET component unit costs provided by suppliers. |
| Drill consumables | Consumables suited to the selected blasthole rigs, allocated typical useful lives. | Drill consumable unit costs as provided by Sandvik. |
| Excavator/Track dozer undercarriage | Based on information provided by the OEM's and from site visits, considering material abrasiveness, excavator walking and mining operator techniques for competency. | Undercarriage component costs provided by OEMs. |
| Blasthole Parameters | Drill parameters assume a target powder factor of 0.61kg/m3, blasting in fresh material only. Pit wall Presplit drilling limited to the Daapleu pit fresh rock material. | Cost drivers as covered (drill consumables, labour, maintenance, fuel). |
| Blasting | Emulsion product used throughout. Quantities of explosives as well as accessories based on the blasthole parameters driven by the mining schedule. | A Down-the-hole service arrangement is assumed based on RFQ pricing. Scope includes explosive product supply and hole charging. |
| Grade Control | • 5 m x 10 m drill pattern @ 60 degrees, 20% overlap<br>• Redrilling of 5%<br>• Sample interval of 1 m<br>• Scheduled 3-4 months ahead of ore mining by pit | Both drill and sample unit costs based on current Hounde contract rates. |
| Operating Overhead | The following overhead were provided:<br>• Mining consultants operating provision<br>• PPE and safety equipment<br>• General open pit operating consumables<br>• Geology and survey specific consumables<br>• Mining administration overheads | |

***Mining Operating Costs***

Table 21.1.2 provides the Operating Cost breakdown by activity to include ore and waste splits.

**Table 21.1.2 Mine Operating Costs (US$ million)**

| Cost Area | Ore | Waste | Total |
|---|---|---|---|
| Pit load & haul | 40.57 | 86.94 | 127.51 |
| Ore rehandle | 33.75 | | 33.75 |
| Drill and blast | 31.70 | 58.29 | 89.99 |
| Grade control | 18.66 | | 18.66 |
| Management & Supt | 22.88 | 19.24 | 42.12 |
| Operating Overheads | 2.92 | 3.38 | 6.30 |
| **Total** | **150.49** | **167.85** | **318.33** |

### 21.1.2 Process Plant and Administration

Process plant operating costs for the Ity Gold Project have been developed with input from Endeavour.

The operating costs have been compiled from information sourced by Lycopodium and Endeavour:

- Manning levels and pay rates advised by Endeavour.
- Consumable prices from supplier budget quotations, Endeavour advice from Ity or the Lycopodium database.
- Cyanide and lime consumptions and gold extractions based on laboratory testwork results.
- Modelling by Orway Mineral Consultants (OMC) for crushing and grinding energy and consumables, using ore characteristics measured during the testwork.
- General and administration costs based on advice from Endeavour and typical allowances.
- First principle estimates based on typical operating data / standard industry practice.

The following exchange rates, as advised by Endeavour, have been used for the preparation of the operating cost estimate:

- US$1.00 = A$1.300 (Australian Dollar).
- US$1.00 = 593 XAF (CFA Franc).
- US$1.00 = 13.62 ZAR (South African Rand).
- US$1.00 = 0.891 € (Euro).

The operating cost estimates are presented at summary and detail level. The estimate is considered to have an accuracy of ±15%, is presented in US dollars (US$) and is based on prices obtained during the third quarter of 2016 (3Q16).

The detailed operating cost estimate is available on request.

The operating cost estimate includes all direct costs to allow production of gold bullion. All costs associated with mining are excluded from this estimate (and are summarised in Section 21.1.1 above) except mining personnel camp and catering costs and grade control laboratory costs, which are included in the general and administration cost and laboratory cost.

The treatment plant availability includes allowance for scheduled and unscheduled maintenance.

***Summary Battery Limits***

The battery limits for the process plant operating costs are as follows:

- Ore delivered to the ROM bin (ROM pad FEL costs are included in the mining operating costs Section 21.1.1).
- Tailings discharge from the tails pipeline to the tailings storage facility (TSF).
- Gold bullion in plant goldroom safe.

Operating costs have been developed for the seven ore sources (six pits with Daapleu split into oxide and primary ore).

Process plant and administration operating costs were developed using the parameters specified in the Process Design Criteria based on 'design' annual throughputs of 3,000,000 tpa using a blended feed as described in Mining Section 16.

The Life of Mine (LOM) costs are based on the pit schedule developed and described in Section 16.

The General and Administration costs in Table 21.1.3 exclude all head office and off site costs.

Costs were split into fixed and variable components to enable then to be used to derive annual costs for changing ore blends and/or throughput with further development of the mine plan. The split of fixed and variable costs by ore type are presented in Table 21.1.4.

**Table 21.1.3 Summary Operating Costs by Ore Type at 3 Mtpa Design Throughput**

| Cost Area | DAA Oxide | | Gbeitouo | | Ity Flat | | Mont Ity | |
|---|---|---|---|---|---|---|---|---|
| | US$/annum | US$/t | US$/annum | US$/t | US$/annum | US$/t | US$/annum | |
| Power (exc grinding) | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 6,018,257 | 2.01 |
| Power (grinding) | 3,240,750 | 1.08 | 4,677,197 | 1.56 | 4,302,635 | 1.43 | 4,190,358 | 1.40 |
| Operating Consumables | 13,035,449 | 4.35 | 16,729,300 | 5.58 | 19,194,345 | 6.40 | 21,335,548 | 7.11 |
| Maintenance | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 3,008,973 | 1.00 |
| Laboratory | 726,567 | 0.24 | 726,567 | 0.24 | 726,567 | 0.24 | 726,567 | 0.24 |
| Process & Maintenance Labour | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 3,002,248 | 1.00 |
| **Total Processing Cost** | **29,032,243** | **9.68** | **34,162,542** | **11.39** | **36,253,025** | **12.08** | **38,281,950** | **12.76** |
| Administration Labour | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 4,139,176 | 1.38 |
| General & Administration Costs | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 4,389,241 | 1.46 |
| Total General & Administration | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 8,528,418 | 2.84 |
| **Total Cost including G&A** | **37,560,661** | **12.52** | **42,690,960** | **14.23** | **44,781,443** | **14.93** | **46,810,368** | **15.60** |

| Cost Area | Walter | | Zia NE | | DAA Prim | | LOM Average * |
|---|---|---|---|---|---|---|---|
| | US$/annum | US$/t | US$/annum | US$/t | US$/annum | US$/t | US$/t |
| Power (exc grinding) | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 6,018,257 | 2.01 | 2.01 |
| Power (grinding) | 3,541,476 | 1.18 | 3,120,945 | 1.04 | 6,111,831 | 2.04 | 1.62 |
| Operating Consumables | 15,461,241 | 5.15 | 14,273,543 | 4.76 | 12,234,805 | 4.08 | 4.84 |
| Maintenance | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 3,008,973 | 1.00 | 1.00 |
| Laboratory | 726,567 | 0.24 | 726,567 | 0.24 | 726,567 | 0.24 | 0.24 |
| Process & Maintenance Labour | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 3,002,248 | 1.00 | 1.00 |
| **Total Processing Cost** | **31,758,761** | **10.59** | **30,150,532** | **10.05** | **31,102,681** | **10.37** | **10.72** |
| Administration Labour | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 4,139,176 | 1.38 | 1.38 |
| General & Administration Costs | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 4,389,241 | 1.46 | 1.46 |
| Total General & Administration | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 8,528,418 | 2.84 | 2.84 |
| **Total Cost including G&A** | **40,287,179** | **13.43** | **38,678,950** | **12.89** | **39,631,099** | **13.21** | **13.56** |

**Table 21.1.4 Fixed / Variable Operating Costs by Ore Type at 3 Mtpa Design Throughput**

| Cost Area | DAA Oxide | | Gbeitouo | | Ity Flat | | Mont Ity | |
|---|---|---|---|---|---|---|---|---|
| | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t |
| Power (exc grinding) | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 |
| Power (grinding) | 0 | 1.08 | 0 | 1.56 | 0 | 1.43 | 0 | 1.40 |
| Operating Consumables | 0 | 4.35 | 0 | 5.58 | 0 | 6.40 | 0 | 7.11 |
| Maintenance | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 |
| Laboratory | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 |
| Process & Maintenance Labour | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 |
| **Total Processing Cost** | **12,111,758** | **5.64** | **12,111,758** | **7.35** | **12,111,758** | **8.05** | **12,111,758** | **8.72** |
| Administration Labour | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 |
| General & Administration Costs | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 |
| Total General & Administration | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 |
| **Total Cost including G&A** | **20,640,176** | **5.64** | **20,640,176** | **7.35** | **20,640,176** | **8.05** | **20,640,176** | **8.72** |

| Cost Area | Walter | | Zia NE | | DAA Prim | | LOM | |
|---|---|---|---|---|---|---|---|---|
| | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t | Fixed US$/annum | Variable US$/t |
| Power (exc grinding) | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 | 6,018,257 | 0.00 |
| Power (grinding) | 0 | 1.18 | 0 | 1.04 | 0 | 2.04 | 0 | 1.62 |
| Operating Consumables | 0 | 5.15 | 0 | 4.76 | 0 | 4.08 | 0 | 4.84 |
| Maintenance | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 | 2,510,000 | 0.17 |
| Laboratory | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 | 581,254 | 0.05 |
| Process & Maintenance Labour | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 | 3,002,248 | 0.00 |
| **Total Processing Cost** | **12,111,758** | **6.55** | **12,111,758** | **6.01** | **12,111,758** | **6.33** | **12,111,758** | **6.68** |
| Administration Labour | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 | 4,139,176 | 0.00 |
| General & Administration Costs | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 | 4,389,241 | 0.00 |
| Total General & Administration | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 | 8,528,418 | 0.00 |
| **Total Cost including G&A** | **20,640,176** | **6.55** | **20,640,176** | **6.01** | **20,640,176** | **6.33** | **20,640,176** | **6.68** |

***Qualifications***

The operating cost estimates presented are exclusive of the following:

- Mining costs and other areas not specifically mentioned in the detailed estimate.
- All ore handling costs.
- All head office (off-shore) costs, including head office labour.
- Any impact of foreign exchange rate fluctuations.
- Any escalation from the date of the estimate.
- Any contingency allowance.
- All withholding taxes and other taxes (Import duties on consumables allowed).
- Tailings storage facility future lifts and site rehabilitation.
- Government monitoring / compliance costs except where noted.
- Gold refining and bullion transport and in-transit security of gold from site.
- Ongoing land / community compensation costs

The operating cost estimate includes the following:

- Import duties on consumable unit costs (in the consumables cost).
- Costs for the preparation and assaying of 100 mine grade control samples per day and routine laboratory tests on the site water samples (in the laboratory cost).

***Power***

The project will run from a grid power supply. This will necessitate expanding the Danane substation and running a new 90 kV overhead power line. ECG conducted a study which has indicated an electricity supply cost of US$0.1243/kWh.

The annual power cost for the ore types and blends is estimated to be US$11.5 million/annum or US$3.63 /tonne based on the LOM schedule.

***Operating Consumables***

Costs for operating consumables, including reagents, liners, fuels and process supplies have been estimated to be US$14.2 million/annum or US$4.73 /tonne based on the LOM schedule.

These costs include reagents, consumables, wear materials and grinding media.

***Maintenance***

The maintenance cost allowance has been factored from the capital supply cost using factors from the Lycopodium database and is estimated to be US$3.0 million/annum or US$1.00 /tonne based on the LOM schedule

The maintenance cost covers mechanical spares and wear parts, but excludes crushing and grinding wear components, media and general consumables (covered in the consumables cost estimate). The maintenance cost excludes all payroll maintenance labour (covered in the labour cost estimate) but includes an allowance for contract maintenance.

***Labour***

The labour rates, manning levels and rosters used to determine the labour operating cost estimate were agreed with Endeavour based on benchmarking within Cote d'Ivoire and similar operations. The shift manning levels are based on a three panel, 12 hour shift roster plus leave cover.

Manning numbers and annual labour costs are summarised in Table 21.1.5. More detail of the labour structure and costs is included in Appendix 12.1.

**Table 21.1.5 Manning Numbers and Annual Labour Cost**

| | Labour Cost US$/year | Expatriate | Ivorian National | Local | Total |
|---|---|---|---|---|---|
| Process Operations | 1,176,795 | 4 | 4 | 36 | 44 |
| Maintenance | 1,825,453 | 8 | 4 | 35 | 47 |
| Total Processing | 3,002,248 | 12 | 8 | 71 | 91 |
| Administration | 4,139,176 | 8 | 34 | 38 | 80 |
| **Total - Processing & Administration** | **7,141,424** | **20** | **42** | **110** | **172** |
| Mining Personnel | - | 11 | 42 | 134 | 187 |
| **Total Personnel (including Mining)** | | **31** | **84** | **243** | **359** |

Labour includes all personnel required for the Ity project, excluding off-shore head office personnel. This includes plant administration, operations, maintenance and mining owner and contract personnel.

***Laboratory***

A contract laboratory will provide sample preparation and assay services for plant and environmental samples and one hundred mine grade control samples per day and is estimated to be US$0.73 million/annum or US$0.24 /tonne

***ROM Rehandle***

The cost associated with rehandling the ROM ore for blending and plant feed is captured in the mining cost estimate (Section 21.1.1).

***General and Administration***

The general and administration costs were estimated by Endeavour and Lycopodium based on costs from existing Endeavour operations, supplier quotations and appropriate benchmarking and is estimated to be US$8.5 million/annum or US$2.84 /tonne

***Services and Utilities***

Mobile equipment costs provide for the fuel and maintenance of the mobile equipment fleet (excluding the mining fleet and mining light vehicles). The purchase cost of this equipment has been included in the capital cost estimate.

Water supply costs are based on operation and maintenance of pit dewatering bores around Daapleu and Ity Flat only as these two pits provide the first few years of process plant feed. The cost also allows for treatment of excess water prior to discharge to the river.

***Operations Pre-Production***

The costs incurred by the operations during the latter stages of construction and commissioning are included in the capital cost estimate under Owner's Costs, but are derived in the operating cost estimate. The total cost is estimated to be US$9.5 million. This is made up of:

- Pre-Production Labour
- Pre-Production Administration Expenses
- First Fill Reagents and Opening Stocks
- Vendor Representatives
- Training
- Working Capital

## 21.2 Capital Cost Estimate

### 21.2.1 Overview and Summary

The overall study capital cost estimate was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this study report. Additional estimate details for items costed are available on request.

Summary mine capital costs (developed by Calsta for Mining Solutions) are included in the estimate tables below.

Knight Piésold provided quantities for the tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds with rates applied by Lycopodium to derive the capital estimate.

All costs are expressed in US dollars ($) unless otherwise stated and based on 3Q2016 pricing. The estimate is deemed to have an accuracy of ±15%.

Where costs used in the estimate were provided in other than US dollars the following exchange rates were used:

- US$1.00 = A$1.300 (Australian Dollar).
- US$1.00 = 593 XAF (CFA Franc).
- US$1.00 = 13.62 ZAR (South African Rand).
- US$1.00 = 0.891 € (Euro).

The various elements of the Project estimate have been subject to internal peer review by Lycopodium and have been reviewed with Endeavour for scope and accuracy.

The initial project capital cost was estimated at US$306.66 million. The capital estimate is summarised in Table 21.2.1.

**Table 21.2.1 Initial Capital Cost Estimate Summary (US$, 3Q2016, ±15%)**

| Main Area | Initial Capital (US$M) |
|---|---|
| Treatment Plant | 63.3 |
| Reagents and Services | 9.2 |
| Infrastructure & Tailings | 46.2 |
| Mining | 58.6 |
| Construction Distributables | 24.5 |
| Subtotal | 201.8 |
| Management Costs | 15.8 |
| Owners Project Costs | 59.0 |
| Owners Operations Costs | 3.7 |
| Contingency | 26.4 |
| **Project Total** | **306.7** |

### 21.2.2 Mining Capital Costs

***Capital Cost Basis***

Mining capital costs covers all mining equipment as well as preparatory works. Mining equipment is purchased outright on an 'as required' basis.

Table 21.2.2 provides the basis for the mining capital costs (US$M).

**Table 21.2.2 Capital Cost Basis**

| Cost Area | Description | Cost Basis |
|---|---|---|
| Primary Mining Equipment | Covers all load, hauling, drilling and ancillary mining mobile equipment. | Based on OEM provided pricing. Covers:<br>• Equipment purchase<br>• Optional extras<br>• Freight and transport<br>• First fills<br>• Spares |
| Support Mining Equipment | Support equipment includes:<br>• Service & fuel trucks<br>• Rock breakers<br>• Lighting plants<br>• Maintenance tooling<br>• Light vehicles<br>• Miscellaneous spares | Recent Endeavour pricing |
| Preparatory Works | Covers the following:<br>• Clearing and grubbing for pit and waste dump footprints<br>• Establishment of a ROM pad base<br>• Waste dump dozing upon completion of mining as preparation for rehabilitation works | Clear and grub based on First Principals costing, checked to contract costs. ROM build based on budget contract pricing. |

### *Mining Capital Cost Estimate*

Estimated mining capital costs (excluding contingency allowances) are provided in Table 21.2.3.

**Table 21.2.3 Mining Capital Costs**

| | Cost – US$M | | |
|---|---|---|---|
| **Item** | **Preproduction** | **Sustaining** | **Total** |
| **Primary Mobile equipment** | | | |
| Excavators | 3.48 | 3.48 | 6.97 |
| Front End Loaders | 2.35 | 2.35 | 4.69 |
| Trucks | 11.18 | 3.44 | 14.62 |
| Track Dozers | 2.55 | | 2.55 |
| Wheel Dozers | 1.44 | | 1.44 |
| Graders | 1.21 | | 1.21 |
| Watercarts | 3.62 | | 3.62 |
| Blasthole drills | 2.04 | 2.04 | 4.09 |
| Tyre Handler | 0.58 | | 0.58 |
| **Subtotal Primary Mobile Equipment** | **28.46** | **11.31** | **39.77** |
| **Support Mobile Equipment** | | | |
| General Mine Mobile Equipment | 4.48 | 3.48 | 7.96 |
| Mine Operating Equipment | 1.86 | 1.84 | 3.70 |
| Mining First Fills & Spares | 0.18 | | 0.18 |
| Communication & System Controls | 2.02 | 1.21 | 3.23 |
| Light Vehicles & Busses | 1.63 | 2.18 | 3.81 |
| Maintenance Equipment Setup | 0.50 | 0.25 | 0.75 |
| **Subtotal Support Mobile Equipment** | **10.67** | **8.95** | **19.62** |
| **Total Mine Mobile Equipment** | **39.12** | **20.27** | **59.40** |
| **Owners Preparatory Works** | **1.87** | **2.20** | **4.07** |
| **Total** | **41.00** | **22.46** | **63.46** |

### 21.2.3 Process and Administration Capital Costs

***Scope***

The overall capital cost estimate includes the following scopes of work:

- Mine contractor mobilisation and pre-commissioning mining costs as described in Section 21.2.2.
- Process facility as described in Section 17.
- Infrastructure as described in Section 18.
- Tailings storage facility as described in Section 18.
- Installation costs, EPCM costs and contractor distributable costs.
- Owners costs including first fill and opening stocks of reagents and consumables as well as operating and insurance spares.
- Site earthworks, tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds.
- Mobile equipment.

Exclusions include the following:

- Project sunk costs.
- Import duties and taxes on the assumption that the Project will be exempt.
- Escalation.

***Plant and Infrastructure Capital Costs***

The capital cost estimate was prepared in accordance with Lycopodium's standard estimating procedures and practices. The basis and methodology are summarised in Table 21.2.4.

## Table 21.2.4 Capital Cost Estimate Basis

| Description | Basis |
|---|---|
| **Site** | |
| Geographical Location | Actual site |
| Maps and Surveys | Good topographical data available |
| Geotechnical Data | Preliminary |
| **Process Definition** | |
| Process Selection | Fixed |
| Design Criteria | Fixed |
| Flowsheets / Plant Capacity | Fixed |
| P&IDs | Not Required as suitable 'go-by' costs available from Lycopodiums database |
| Mass Balances | Fixed |
| Equipment List | Fixed |
| **Process Facilities Design** | |
| Equipment Selection | Selection based on duty and budget pricing provided by vendors. |
| General Arrangement Drawings | Fixed |
| 3D model | Preliminary to a level of detail suitable for FS |
| Piping Drawings | Not required |
| Electrical Drawings | HV SLD. LV drawings not required |
| Specifications / Data Sheets | Preliminary for BQRs |
| **Infrastructure Definition** | |
| Existing Services | Not relevant other than use of exploration camp for 'pioneer' accommodation |
| Design Basis | Fixed |
| Layout | Fixed |
| Bulk Earthworks – Plant-site, Mine Services & Camp | Volume estimated from 3D Models. |
| Tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds | Bulk Quantities provided by Knight Piésold |
| Detailed Earthworks | Allowances for under pad excavation and backfill to prepare site for concrete works |
| Concrete Installation | Estimated from the layout and similar projects of comparable scale. Concrete (wet) supply rates and installation rates applied from project specific BQRs. |
| Structural Steel | Quantities estimated from the layout and similar projects of comparable scale. Supply and install rates applied from project specific BQRs. |
| Platework & Small Tanks | Quantities provided in the mechanical equipment list. Large item quantities estimated from reference projects. Smaller items compared to database. Supply and install rates applied from project specific BQRs |
| Tankage Field Erect | Quantities provided in the mechanical equipment list. Supply and install rates applied from project specific BQRs. |
| Mechanical Equipment | Quantities provided in the mechanical equipment list. Costs from responses to BQRs from reputable suppliers for all equipment packages with a value nominally >$50,000. Costs for low value items taken from the Lycopodium database. |
| Mining Fleet | Refer mining cost estimate. |
| Power Supply | Estimate provided by ECG. |
| Conveyors | Concrete & structural estimated from reference projects and layout. Mechanicals supply pricing from current database and installation rates applied from responses to BQRs. |
| Plant Piping General | Factored off mechanical costs derived by completed project actuals. |
| Overland Piping | Size and specification based on engineering selection. Quantity based on site layout. Rates based on recent market inquiries. |
| Electrical General | Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs |
| Electrical HV | Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs |
| Commodity Rates – General | Appropriate rates from responses to project specific BQRs. |
| Installation Rates – General | Appropriate rates from responses to project specific BQRs based on preliminary contracting strategy. |
| Heavy Cranes | Requirements estimated based on largest lifts and likely duration. |
| Freight General | Combination of rates per freight ton & factors. |
| Contractor Mobilisation / Demobilisation | Appropriate rates from responses to project specific BQRs, adjusted to suit the final project scope. |
| Fencing | Costed based on measured length and rate. |
| EPCM | Scope and deliverables based estimate. |
| Vendor Representatives | Estimated. |
| **Owner's Costs** | |
| Site Establishment | Requirements estimated using base rates. |
| Construction Facilities | Allowance based on projects of a similar size. |
| Opening Stocks, First Fill Reagents and Consumables | Estimated from consumption rates and costs in operating cost estimate. |
| Working Capital | Estimated from costs in operating cost estimate and likely commissioning and ramp up schedule. |
| Spares | Historical data from similar sized plants. |
| Owner's Project Team | Client estimate. |
| Project Insurances and Permits | Client estimate. |
| Sterilisation Drilling | Assumed in sunk costs. |
| Land Compensation | Client estimate. |
| Community Relations | Client estimate. |
| Plant preproduction expenses | Estimated from operating cost estimate and likely commissioning / ramp up schedule. |
| Training | Estimated from operating cost estimate manning schedule |
| Owners Team Expenses | Client estimate. |
| Duties and Taxes | Excluded on the assumption the Project will be exempt. |
| Escalation | Excluded. |

***General Estimating Methodology***

The process plant was broken down into unit operation areas with quantity take-offs benchmarked against similar facilities from previous projects to provide the additional scope and level of confidence needed to confirm a FS level of estimate.

Overall plant layout and equipment sizing was prepared with sufficient detail to permit an assessment of the engineering quantities for the majority of the facilities for earthworks, concrete, steelwork, and mechanical items. The layouts enabled preliminary estimates of quantities to be taken for all areas and for interconnecting items such as piperacks.

Unit rates for labour and materials were derived from responses to BQRs sent to fabricators and contractors experienced in the scale and type of work in the region.

Budget pricing for equipment was obtained from reputable suppliers with the exception of low value items which were costed from Lycopodium's database of recent project costs.

For the accommodation camp, offices, workshops and similar items appropriate budget pricing was obtained from reputable suppliers of similar prefabricated designs with erection / installation costs derived from solicited contractor rates.

For the tailings storage facility, haul roads, site access roads, river diversions and pit protection bunds, bills of quantities were provided by Knight Piésold based on their preliminary designs.

All earthworks and concrete works have been costed on an Owners self perform basis with applicable rates and distributables included to support the work.

***Quantity Development***

Overall estimated quantities of key commodities are shown in Table 21.2.5.

The derivation of quantities within these categories by percentage is provided in the table weighted by value of the direct permanent works (i.e. excluding temporary works, construction services, commissioning assistance, EPCM, escalation and contingency).

Preliminary Engineering refers to quantities taken from data and layouts prepared specifically for the Project. Concept Engineering refers to quantities derived from similar project designs factored and/or modified to suit.

**Table 21.2.5 Derivation of Quantities**

| Classification | Quantity | Unit | Preliminary Engineering % | Concept Engineering % | Estimated % |
|---|---|---|---|---|---|
| Concrete (excluding blinding) | 11,044 | $m^3$ | 30% | 60% | 10% |
| Structural Steel | 1,218 | t | 90% | 10% | - |
| Chutes / Hoppers / Bins / Tanks | 399 | t | 90% | 10% | - |
| Field Erected Tanks | 1,154 | t | 80% | 20% | - |

***Pricing Basis***

Estimate pricing has been derived from a combination of the following sources:

- Budget pricing for all equipment and material packages valued over nominally $50,000.
- Recent historical pricing from low value equipment and materials.
- Quoted rates.
- Allowances.

Pricing has been identified by the following cost elements, as applicable, for the development of each estimate item.

***Plant Equipment***

This component represents prefabricated, pre-assembled, off-the-shelf types of mechanical or electrical equipment, either fixed or mobile. Pricing is inclusive of all costs necessary to purchase the goods ex-works, generally excluding delivery to site (unless otherwise stated) but including operating and maintenance manuals. Vendor representation and commissioning spares have been allowed for separately in the estimate.

***Bulk Materials***

This component covers all other materials, normally purchased in bulk form, for installation on the Project. Costs include the purchase price ex-works, any off-site fabrication, transport to site (unless otherwise stated), and over-supply for anticipated wastage.

***Installation***

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between direct labour, equipment and contractors' distributables.

The labour component reflects the cost of the direct workforce required to construct the Project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour to the contractor inclusive of overtime premiums, statutory overheads, payroll burden and contractor margin.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost also includes cranes, vehicles, small tools, consumables, PPE and the applicable contractor's margin.

Contractors' indirect costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, mobilisation and demobilisation, R&Rs, meals and accommodation costs, and the applicable contractors' margin.

***Temporary Construction Facilities***

Facilities will be capable of servicing the owners and EPCM teams.

Included in the estimate for construction facilities are the following:

- Construction offices.
- Computers and computing servers, telephones, printers, etc. and office furniture.
- Provision of services.

***Heavy Lift Cranage***

A heavy lift crane of 250 t capacity has been allowed for in the estimate for the duration of the installation of the mills.

***Contractor Distributables***

Costs for mobilisation / demobilisation of labour and equipment to / from the Project site were, where practical, adopted from budget quotation enquiries to contractors or adjusted from current tenders / contracts to reflect the Project location.

***Earthworks***

Quantities for plant site bulk earthworks have been estimated from the layout. Rates were derived from project specific budget quotation requests.

Quantities for the ROM pad (at the process plant) are limited to the engineered fill and drainage and site earthworks required around the ROM retaining wall. The cost of the balance of the ROM pad is included in the mining estimate.

***Concrete***

Quantities for concrete works were established using the following:

- Material take-offs from layouts prepared for the FS.
- Benchmarking against detailed drawings for similar sized projects completed by Lycopodium.

Rates for this estimate were based on an Owner self perform strategy, benchmarked against similar projects.

Rates and quantities were prepared on a composite per cubic metre basis. Mobilisation, demobilisation and indirect costs were separated to reflect contract methodology.

***Steelwork***

Quantities for structural steel were established using the following:

- The layout and equipment elevation drawings / sketches prepared for the FS.
- Benchmarking against detailed drawings for similar sized projects completed by Lycopodium.

Rates for this estimate were based on responses to BQRs from fabricators with experience on this kind of work and capacity to perform the works.

Site installation hours were based upon responses to BQRs from regional contractors with experience on this kind of work and capacity to perform the works.

***Platework / Tankage***

Platework and tankage quantities were determined using the sizing provided in the mechanical equipment list prepared for the FS as the basis.

Rates for this estimate were based on responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

***Mechanical Equipment***

The mechanical equipment list prepared for the FS provided the quantities and sizing for the cost estimate.

Budget pricing was obtained from reputable suppliers for the majority of mechanical equipment, based on equipment data sheets prepared for the FS.

Equipment installation hours were estimated based on responses to BQRs solicited from contractors and installation hours estimated by Lycopodium. For each individual item of equipment due allowances were made for retrieval from the storage location, handling, placing, installing and commissioning the equipment.

***Plant Pipework***

The supply and installation estimate for in-plant piping was derived using factors derived from previously built projects. These factors are a percentage of the mechanical equipment supply and installation costs, and are calculated per individual plant area. The plant piping costs allow for the supply and installation of pipe, fittings, mountings and manual valves.

***Overland Pipework***

The overland piping, i.e. tailings discharge line and decant water return line were estimated from first principle engineering. Quantities were based on material take-offs.

***Electrical Instrumentation***

The supply and installation estimate for electrical and instrumentation was estimated in detail and compiled using electrical equipment lists, loads lists, GA drawings and supplier pricing. .

***Erection and Installation***

Included in the discipline by discipline assessment of erection / installation costs detailed above, allowances were made for major construction cranage and equipment and construction costs such as site establishment, construction personnel meals, accommodation, flights and fuel usage, etc.

***Architectural / Buildings***

Budget pricing for prefabricated and steel frame buildings were sourced from reputable suppliers based on preliminary layout drawings.

***Transport***

The transport costs included in the estimate are a combination of rates per freight ton and factors on supply costs.

***EPCM***

The Engineering, Procurement and Construction Management (EPCM) estimate was based on a first principal build up of costs based on the assessed scope of work managed by the EPCM Engineer for the Project.

Expenses such as catering and accommodation for the Engineer's site personnel, as well as site telecommunications costs are included in the estimate.

***Vendor Commissioning***

The estimate was based on Lycopodium's experience for similar plant and equipment.

***Qualifications***

The estimate is subject to the following qualifications:

- All labour rates, materials and equipment supply costs are current at 1Q15. Contingency has been allowed based on the quality of the information, however no allowance for escalation has been included.
- Construction contractor rates include mobile equipment, vehicles, fuel, construction power and consumables for the duration of construction. Potable water and raw water supply will be provided by the Client and available at site for the use by contractors.

- Accommodation, meals and mobilisation / demobilisation / R&R flights of construction contractor personnel are incorporated in the contractor indirect labour rates on the basis of individual contractors making their own accommodation arrangements.
- Meals and accommodation for the owner and EPCM teams have been allowed in the estimate.
- Project spares have been based on other projects of similar size.
- A commissioning assistance crew is allowed for in the EPCM allowance.
- PLC programming for the process plant has been allowed for in the EPCM allowance.
- Site supply of power and raw water (for operations and construction), sewage removal and treatment, communications network for construction facilities are included in the infrastructure costs.

***Contingency***

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope. Contingencies do not include allowances for scope changes, escalation or exchange rate fluctuations.

Contingency is an integral part of an estimate and has been applied (after careful analysis) to all parts of the estimate, i.e. direct costs, indirect costs, services costs, etc.

### 21.2.4 Owners Costs

The following items are included in the Owner's costs:

- Owner's project management team.
- Owner's team costs including project insurances.
- Operating and capital spares holdings.
- First fill and opening stocks of reagents and consumables.
- Cost of preproduction labour, training and operational readiness.

### 21.2.5 Spares

Spares have been taken as an allowance of the capital costs and benchmarked against the spares expenditure on projects of a comparable scale. A minimalist approach has been assumed, with spares stocks progressively expanded during operations.

### 21.2.6 First Fill and Opening Stocks of Consumables

Quantities for opening stocks and first fill consumables have been assembled from basic principles and using the Project design criteria. Unit rates are based on budget quotations solicited from suitable suppliers.

### 21.2.7 Exclusions

The following is excluded from the overall project capital costs:

- Duties / taxes / fees.
- Project sunk costs (allowance made in the Cash Flow Model).
- Project escalation.

## 21.3 Project Implementation

The approach to project implementation outlined in this study and the summary Project Execution Plan (available on request) was used as the basis for the preliminary implementation schedule and the build up of the capital cost estimates. The proposed approach is to engage a suitable Engineering, Procurement and Construction Management (EPCM) contactor for design and construction management of the process plant and infrastructure, which will then be handed over to the Owner's operating team. The construction of the mining operations will be self-performed by the Endeavour operations team.

### 21.3.1 Owner's Team and Organisational Structure

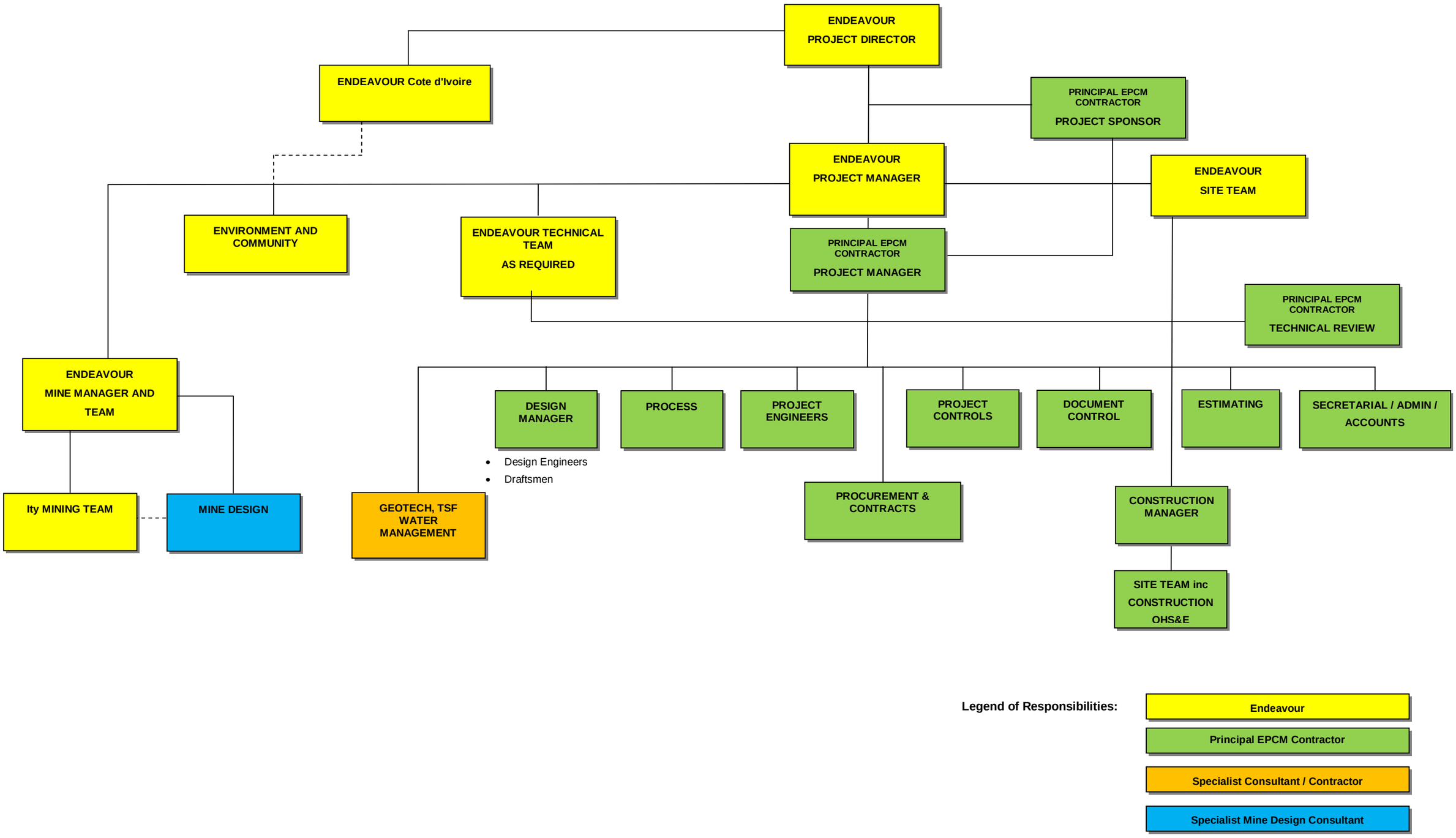
The Endeavour Owner's team will be progressively expanded to widen its skills and knowledge base to meet the needs of the Project. A 'snapshot' of the overall project development team structure is shown in Figure 21.3.1 at a point in time when offshore based engineering, design and procurement activity is high and site works are also underway.

An Endeavour project team will manage both the onshore and offshore activities of the principal EPCM contractor and specialist subcontractors as well as providing specialist technical input into the Project design.

Key onshore mine operations roles will be filled early to contribute to mine design and manage the early mine development works on site.

An Endeavour on site management, administration and services department will manage environment and community issues and prepare the site for the coming influx of operating personnel.

**Figure 21.3.1 Project Organisation Chart**



### 21.3.2 Logistics

West Africa has a well developed mining industry serviced by a network of air, sea and road routes, although within Côte d'Ivoire this has been less utilised in recent years than neighbouring Ghana, for example. The logistics needs of a project the size of Ity are, however, relatively modest on a regional mining scale and the port facilities at Abidjan have the capacity and facilities to act as the main gateway for importation of both construction equipment and materials, and ongoing operating consumables.

Road links within the country are generally good with significant investment in roads in Côte d'Ivoire in the past decade. Road conditions from Abidjan to the Ity site do vary however and can be summarised as follows:

- 230 km dual carriageway section from Abidjan to Yamoussoukro forms part of the Trans African Continental highway system and is in good condition.
- 243 km from Yamoussoukro to Duekoue via Daloa is paved, single lane and in good condition.

From Duekoue, two roads access the project site from north and south. The north access is from Man and then, on to Danané and Zounan-Hounein from where about 15 km of dirt road leads to the village of Ouyatouo. Southern access is through Guiglo and Toulépleu. The condition of both access roads is variable to poor in parts.

### 21.3.3 Mine Development

The mine planning study is based on the Owner operated mining operation. The mining activities will be managed by the Owner’s mining team. It is possible that the mining fleet will be used to undertake some early stage bulk earthworks for the Project.

A mining services area has been located close to the ROM pad which will house support facilities and infrastructure such as offices, workshop, wash bay and workforce facilities.

The current mine plan anticipates mining commencing three months before first ore to the mill, with a total mine life of eight and a half years and with the mill operating on material recovered from stockpiles for a further five months after completion of mining.

### 21.3.4 Process Plant and Surface infrastructure

For the process plant and surface infrastructure, the implementation strategy adopted is an EPCM approach where an Engineer (an internationally accredited engineering company) will be responsible for managing all aspects of the design, procurement and construction activities under the broad direction of an Owner's team.

A comprehensive Project Execution Plan (PEP) incorporating a schedule and control budget will be developed for the Project detailing the overall management methodology for the delivery of the Project, including engineering, procurement, construction, commissioning and handover.

The PEP will include strategies for all aspects of project management and control across all the Project functions and phases.

Overall responsibility for health and safety, scope, schedule, budget, and quality within the boundaries of the EPCM contract will rest with the Engineer's Project Manager. The Engineer's Project Manager will be supported at corporate level by the Engineer's Project Sponsor and peer review team who will act as coordinators and advisors regarding the Engineer's corporate quality requirements.

The Engineer's offshore design and procurement team and onshore construction team will both report to the Engineer's Project Manager. The Project Manager will be based initially in the design office but will relocate to site at an appropriate time once offshore work is essentially complete. The Engineer's management team including Design Manager, Lead Project Engineer, Contracts Engineer, Senior Procurement Officer, Health and Safety Manager, and the Construction Manager will all report to the Engineer's Project Manager.

### 21.3.5 Health and Safety

All aspects of the Project will be required to comply with relevant local legislation and current international industrial practice.

***Health and Safety Principles***

The Project's HSEC objective is to develop a culture and install processes to ensure the safety and health of all employees, contractors and stakeholders. In striving for this objective, the following principles will apply:

- All contractors shall demonstrate commitment to high HSEC standards to be eligible to work on the site.
- No business objective will take priority over health and safety.
- At all times, the safety of all personnel is to be ensured.
- The Project will target zero harm.
- All incidents and injuries shall be reported, investigated, and actions taken to prevent reoccurrence.
- Accountability for providing a safe work environment rests with every employee.
- All individuals have the responsibility and accountability to identify and eliminate or manage hazards / risks associated with their workplace.
- Legal obligations are the minimum requirements for safety and health standards.
- All employees (including those of contractors) are to be trained and equipped to have the skills and facilities to enable an injury free workplace.

- All employees (including those of contractors) are to undergo Project specific inductions prepared by the Engineer and Endeavour, as appropriate to their site work requirements.

Ensuring health and safety is the responsibility of each Project participant. Consistent with this position, each individual will be expected to actively contribute to the promotion of constructive health and safety behaviours and in the implementation of HSEC management plans.

***HSEC Management Plan***

In consultation with Endeavour, the Engineer will prepare a HSEC Management Plan for the Project.

The HSEC Management Plan will be issued to all contractors tendering for site work as part of the enquiry document. Each contractor will be required to demonstrate a satisfactory prior commitment to safety and present a site specific plan for their proposed involvement in the Project.

### 21.3.6 Design Management

A Project Engineering Plan will define the principles and execution guidelines to be adopted by the Engineer's design team during the design phase of the Project. It will also describe the handover of various engineering deliverables at procurement, tender, construction, commissioning, project close-out and handover stages.

To achieve as short as possible timeline for the overall project development without commitment to full expenditure the schedule has allowed for a period of front end engineering and design (FEED) immediately post completion of the FS. This period will concentrate on the identification of long lead items and the development of engineering deliverables relating to the tendering and award of early contracts.

The Engineering Plan is the principal document detailing the Engineer's major design activities and outlines the philosophy of how key activities will be implemented, the key aspects of each activity and the associated procedures that will be followed to ensure successful outcomes.

The objectives of the Engineering Plan are to:

- define the engineering organisational structure
- nominate key engineering personnel
- define project roles and responsibilities
- outline key work processes required to complete the design and identify procedures and documentation
- incorporate the intent of applicable Endeavour specifications into the Engineering Plan and associated procedures

- identify the performance measures used to monitor the completion of all tasks and activities to gauge the success of the design.

The majority of design outputs are produced during the FEED and detailed design process. These take the form of equipment specifications, datasheets, drawings and purchase requisitions. Support documentation includes lists, material take-offs, calculations, check prints, vendor data and field installation checklists for construction and commissioning.

The Engineering Plan details the specific design, design review, verification and checking procedures to ensure successful outcomes.

### 21.3.7 Project Controls

Effective project controls are critical to the successful completion of a project providing relevant and consistent budget, costs and schedule reporting to the Project team. This provides the tools to efficiently manage the Project at the level of detail necessary to meet project cost and schedule objectives.

The Project Controls Plan (PCP) addresses cost control, planning, progress measurement, project reporting, asset capitalisation and close-out. The scope of the PCP is to provide a framework of the work processes, work flows and information relating to the standard project controls and accounting interface tools, systems and procedures that will be utilised during the execution of the Project.

***Project Schedule and Implementation***

The implementation strategy is structured into four broad stages:

- Detailed design of the process plant and infrastructure.
- Procurement.
- Construction.
- Commissioning and handover.

The estimated durations of key project activities are provided in Table 21.3.1. A preliminary project implementation schedule is is available on request

The dates and durations shown in Table 21.3.1 below are preliminary and are based on receipt of the relevant authorisations to proceed and availability of project funding.

**Table 21.3.1 Key Project Milestones**

| Milestone Description | Planned Completion |
|---|---|
| EPCM Award Date | Week 0 |
| Mill Contract Award | Week 8 |
| Bulk Earthworks and Civil Contractor Mobilisation | Week 10 |
| Camp Contractor Mobilisation | Week 20 |
| Plantsite Civil Contractor Mobilisation | Week 20 |
| SMP Contractor Mobilisation | Week 45 |
| E&I Contractor Mobilisation | Week 49 |
| Mills Mechanical Installation Commencement | Week 58 |
| Power On | Week 71 |
| First Gold Pour | Week 85 |

Several items and activities are potentially on or close to the schedule critical path for design and construction including procurement, manufacturing, transport and installation of long lead mechanical and electrical equipment items for the process plant. Their criticality will be investigated further during early engineering.

***Cost Control***

The capital cost estimate developed during the FS will be updated at key design milestones and used as the control budget for the Project.

Costs will be measured and reported by activity in accordance with the Project Work Breakdown Structure developed during the FS and contract award phases. This will serve to keep the Project informed on a timely basis about risk associated with cost and time.

Monthly cost reports will be prepared to show original budget, approved changes, revised budget and current forecast costs. Committed and incurred costs and paid expenditures will also be included in the report.

### 21.3.8 Quality

The Project Quality Plan (PQP) will cover all work to be undertaken and all services provided by the Engineer and its subconsultants including the provision of EPCM services, subconsultants and other suppliers contracted to undertake work on the Project.

The PQP sets out the quality objectives for the Project and provides the framework for effective quality management during execution of the Project. The document also sets out the measures by which the achievements of the Project can be assessed against project KPIs.

### 21.3.9 Procurement and Contracting

The PEP will address the major procurement and contracting activities and detail the strategies, methodology, procedures and controls that will be adopted during the delivery of the Project.

Packages will be competitively tendered to achieve competitive pricing, and an effective negotiating position to provide value for money to Endeavour. Packages to be sole sourced will be duly justified and first agreed with Endeavour.

Equipment suppliers will be selected on the basis of technical compliance, previous performance and availability to supply relevant equipment. Contractors for site works will be selected on the basis of their safety record, their IR record, previous experience with similar type projects, cost, schedule, availability and capability to perform the work.

A logistic services provider will be engaged to consolidate all project freight, provide sea passage to Abidjan, arrange port and Customs clearance, and arrange road transport to site. With existing mines in operation in the region no insurmountable logistical issues are anticipated.

Local contractors and suppliers will be encouraged to tender for all project works and contracts for which they are qualified to undertake and will be assessed based on their ability to meet the required conditions. It is planned that direct negotiations will be undertaken with smaller local business groups with specific contract packages to encourage local sourcing of project requirements.

Construction contracts will generally be tendered as horizontal packages, that is, by discipline of construction work (e.g. concrete, erection) although in some cases, for example the power station, a vertical design / supply / install contract may be considered.

### 21.3.10 Construction Plan

The Construction Management Plan details the strategies and resources required to construct the works. The plan defines the responsibilities of all parties during construction activities to ensure they are undertaken in a safe and organised manner.

As previously identified the construction strategy is based on an EPCM approach.

Bulk earthworks for the plant will be undertaken by an earthworks contractor. The contractor will initially focus on areas required for temporary facilities and priority buildings to allow these facilities to be established prior to the major plant packages commencing. Temporary facilities include the Engineer's construction offices and project laydown areas.

Concrete works will commence in areas identified to provide earliest access to install major structural steel and site erected tanks.

Structural, mechanical and piping (SMP) and electrical and instrumentation (E&I) installation packages will be structured to provide maximum overlap of activities with preceding disciplines, but without causing excessive interface issues.

Construction activities will be prioritised and managed to facilitate an orderly handover for pre-commissioning activities which will then lead into dry commissioning as operable sections of the plant and infrastructure become available.

Handover to operations for wet commissioning will be on an area by area basis to facilitate the early commencement of operational activities and transition to the operations phase.

### 21.3.11 Project Commissioning

A Commissioning Execution Plan will be prepared for the Project.

These documents outline the plan for pre-commissioning and wet commissioning of the process plant and infrastructure. They also outline the plan for process (or load) commissioning of the plant followed by ramp up to design capacity and execution of performance tests.

The Engineer will provide commissioning services and facilities to ensure the proper execution of the various commissioning phases, and the bringing of the Project into service in a controlled and timely manner to the satisfaction of Endeavour.

Assistance will be provided to Endeavour, if required, for developing and implementing its operational readiness plan.

The Engineer will assist with initial commissioning runs to ensure that plant performance is in accordance with the specified design / performance criteria and to provide such additional supervision and expertise as is required to identify and rectify defects and thereby enable the plant to operate at its specified parameters.

Formal performance trials will be carried out to confirm the completed plant meets its key performance criteria.

### 21.3.12 Project Closeout

At the completion of all construction and commissioning activities, the Engineer will provide the following close out information to Endeavour:

- As built drawings covering buried services, P&IDs, electrical, and others as agreed.
- Testing and commissioning data and records including instrument calibration sheets.
- Documentation for the discharge of vendor and contractor bank guarantees and warranties.
- Project close out report.
- Quality records.

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# 22.0 ECONOMIC EVALUATION

## 22.1 Introduction

An economic model has been used to collate the study results in order to estimate and evaluate project cash flows and economic viability. The evaluation method takes into account milled tonnages and grades for the ore and the associated recoveries, gold price (revenue), operating costs, bullion transport and refining charges, government royalties and capital expenditures (both initial and sustaining). The project has been evaluated on a 100% ownership basis, with no debt financing beyond an expected $25.5 million mining equipment lease, as discussed in Section 14.3.8.

The economic evaluation presents results for Net Present Value ('NPV'), Internal Rate of Return ('IRR'), and payback period. The economic model allows for sensitivity evaluations for changes in gold price, operating costs, and capital costs to determine their relative importance in evaluating the investment decisions. The economic assessment has been prepared with the input from Endeavour, Mining Solutions Consultancy Pty Ltd (mining and processing schedules), Calsta Pty Ltd (mining costs prepared for Mining Solutions), Lycopodium (processing plant, site G&A and infrastructure), and Knight Piésold (tailings management facility and water storage). Lycopodium, with assistance from Endeavour, undertook the economic evaluation and report compilation.

## 22.2 Summary

The results of the economic model show robust results. Applying a long term gold price of $1,250/oz on a flat line basis from the commencement of production, the after-tax NPV5% is $410.9 million, IRR is 35.9% and project payback period is 2.1 years. The life of mine average cash cost per ounce is $528, net of silver credits, and with the addition of royalties and sustaining capital, the life of mine average AISC/oz is $603. Over the first five years of production, the average AISC/oz is $507.

Project gold production averages 113,656 ounces per year over the life of the project with an average of 164,989 ounces per year over the first five years.

The project has low sensitivity to capital and operating costs, but is typically sensitive to gold prices. At $1,250/oz gold price the IRR is 35.9%, and the project maintains an IRR of 19.5% at $1,000/oz gold price.

The summary of the cash flow model (Condensed Annual Cash Flow) is included in Section 14.5.

Table 22.2.1 presents a summary of the production information on which the cash flow model is based and the key project economic measures.

**Table 22.2.1 Summary of Economic Analysis Results**

| Economic Summary | Units | Value |
|---|---|---|
| LOM tonnage ore processed | kt | 41,042 |
| LOM strip ratio | w:o | 2.1 |
| LOM feed grade processed | Au g/t | 1.42 |
| LOM gold recovery | % | 83.1% |
| LOM gold production | oz | 1,561,902 |
| Production period | Years | 13.7 |
| Upfront capital cost | $M | 281.2 |
| | | |
| Life of Mine average: | | |
| Gold, average annual production | oz | 113,656 |
| Cash costs per ounce, net Ag credits | $/oz | 528 |
| AISC per ounce | $/oz | 603 |
| | | |
| Project years 1 to 5 average: | | |
| Gold, average annual production | oz | 164,989 |
| Cash costs per ounce, net Ag credits | $/oz | 446 |
| AISC per ounce | $/oz | 507 |
| | | |
| Project years 1 to 9 average: | | |
| Gold, average annual production | oz | 143,848 |
| Cash costs per ounce, net Ag credits | $/oz | 475 |
| AISC per ounce | $/oz | 542 |
| | | |
| Using $1,250/oz gold price: | | |
| Internal rate of return (pre-tax) | % | 37.3% |
| Net present value - 0% discount rate (pre-tax) | $M | 695.2 |
| Net present value - 5% discount rate (pre-tax) | $M | 466.2 |
| Net present value - 10% discount rate (pre-tax) | $M | 314.5 |
| | | |
| Internal rate of return (after-tax) | % | 35.9% |
| Net present value - 0% discount rate (after-tax) | $M | 607.2 |
| Net present value - 5% discount rate (after-tax) | $M | 410.9 |
| Net present value - 10% discount rate (after-tax) | $M | 278.3 |
| Payback period | Years | 2.1 |

## 22.3 Principal Assumptions

The economic evaluation of the Ity Gold Project was based upon:

- Capital cost estimates prepared by Lycopodium, Calsta (for Mining Solutions), Knight Piésold, and cost estimates from Endeavour's Houndé Gold Project and Agbaou Gold Mine. The Agbaou Gold Mine is located in Côte d'Ivoire and was constructed by Endeavour with commercial production declared in January 2014. The Houndé Project is located in Burkina Faso and is currently in construction.
- Mine schedule and mining operating cost estimates based on an owner-operated mining fleet, prepared by Mining Solutions and Calsta (for Mining Solutions), respectively.
- Process operating and general and administration (G&A) cost estimates prepared by Lycopodium.
- Metallurgical performance characterised by testwork conducted on composite samples from the Ity deposits.
- Sustaining capital cost estimates for the mining operation prepared by Calsta (for Mining Solutions) and for infrastructure and process plant by Lycopodium, Knight Piésold and Endeavour.
- Owners capital cost estimates prepared by Endeavour using recent project data.
- Royalty, tax, discount rates and other model inputs provided by Endeavour. The cash flow analysis excludes any effects due to inflation and all dollars are expressed as real.
- A gold price of $1,250/oz, with +/-20% gold price sensitivity analysis ($1,000 to $1,500/oz).
- Construction is completed over an 18 month Pre-Production period and operations commence in Project Year 1.
- The economic assessment has been undertaken in United States Dollars (US$).

### 22.3.1 Basis of Estimate

- Quarterly tonnage, strip ratio and head grade have been based on the mining schedule as discussed in Section 4.
- The mining, processing and administration costs are based on the operating cost estimate discussed in Section 12.
- The overall recovery figures are based on testwork. Metallurgical recovery is discussed in Section 5.

- The capital cost estimate used as basis for the cash flow model is discussed in Section 13.

- The treatment of depreciation and company taxes are based on the understanding of current Côte d'Ivoire tax law.

### 22.3.2 Depreciation

- Provision has been made for depreciation using a straight line method for a period of 10 years for initial construction capital and for a period of five years for sustaining capital and leased mining equipment.

### 22.3.3 Company Tax

- Corporate tax rate of 25% of gross profit has been used. The Ity Gold Project is comprised of two corporate entities. The first entity, SMI, includes the Walter, Ity Flat, Mont Ity, and Zia NE deposits. SMI is expected to incur all of the upfront capital expenditures and on-going sustaining capital expenses. The second entity, SMD, includes the Daapleu and Gbeitouo deposits. SMD is expected to employ SMI to conduct its operating activities (mining and processing) on a cost-plus basis. SMD is expected to be tax-exempt during an initial five year production period.

### 22.3.4 Refining Costs

- A refinery gold payable rate of 99.90% is assumed.

- The refining charge of $4.30 per ounce of gold is based on Endeavour experience and actual costs incurred from current operating mines in Côte d'Ivoire, and includes the cost of transport and insurance of the gold from site to the refinery.

### 22.3.5 Silver Credits

- Silver to gold ratios typically range from 3.5 to 1 to as little as 2 to 1. For the economic model, payable silver in the doré ingots has been estimated on a ratio of 2 to 1. Silver credits have been calculated using a flat, long-term silver price of $15 per ounce.

### 22.3.6 Royalties

- The State of Côte d'Ivoire is entitled to production royalties as follows.

| Royalty | Gold Price |
|---|---|
| 3% | Up to $1,000 |
| 3.5% | $1,000 – $1,300 |
| 4% | $1,300 – $1,600 |
| 5% | $1,600 and Over |

### 22.3.7 Working Capital

- Working capital, first fills and spare parts have been included in the upfront capital amount, with ongoing working capital movements estimated based on maintaining a rolling quarterly balance equivalent to one month of total operating expenses. Working capital is returned to a nil balance at the end of the mine life.

### 22.3.8 Mining Equipment - Leasing

- The upfront capital cost excludes $25.5M in mine fleet costs which are expected to be lease financed. These costs have been shown in the Condensed Annual Cash Flow Model in Section 14.5 as 'Lease payments - mine equipment', which includes lease finance costs.

### 22.3.9 Closure Costs

- In the economic model, demolition and closure activities occur in Year 15 after mining and processing activities have ceased, with an estimated cost of $5.0 million.

### 22.3.10 Other

- The cash flow model is based on 100% project ownership.
- No provision has been made for interest or cost of capital.
- No provision has been made for escalation or inflation.
- No provision has been made for additional taxation or costs related to the repatriation of funds from Côte d'Ivoire.
- The NPV calculation is based on payments occurring at the end of each period. The cost of the first year is discounted, based on the assumption that there will be some delay in spending the capital from present.
- Allowance has been made in the capital costs for Import Duties with an amount of $1.97 million.
- The cash flow calculations have been prepared on a quarterly basis, and have been totalised to annual amounts based on Project years in the Condensed Annual Cash Flow included in Table 22.4.1.

## 22.4 Outcomes

The results of the economic model show robust results. Applying a long term gold price of $1,250/oz on a flat line basis from the commencement of production, the after-tax NPV5% is $410.9 million, IRR is 35.9% and project payback period is 2.1 years. The life of mine average cash cost per ounce is $528, net of silver credits, and with the addition of royalties and sustaining capital, the life of mine average AISC/oz is $603. Over the first five years of production, the average AISC/oz is $507. A summary from the economic model output is included as Table 22.4.1.

## Table 22.4.1 Condensed Annual Economic Model

| Item | Unit | LOM total / Average | Pre-production | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mining schedule** | | | | | | | | | | | | | | | | | | |
| Total material moved | kt | 128,564 | 3,116 | 15,501 | 15,332 | 17,394 | 16,064 | 15,740 | 15,727 | 12,454 | 11,626 | 5,610 | 0 | 0 | 0 | 0 | 0 | 0 |
| Total waste moved | kt | 87,522 | 2,435 | 10,113 | 10,558 | 13,053 | 9,822 | 10,744 | 11,536 | 8,810 | 7,462 | 2,989 | 0 | 0 | 0 | 0 | 0 | 0 |
| Total ore mined | kt | 41,042 | 680 | 5,388 | 4,774 | 4,340 | 6,243 | 4,996 | 4,190 | 3,644 | 4,165 | 2,622 | 0 | 0 | 0 | 0 | 0 | 0 |
| Stripping ratio | w:o | 2.1 | 3.6 | 1.9 | 2.2 | 3.0 | 1.6 | 2.2 | 2.8 | 2.4 | 1.8 | 1.1 | 0 | 0 | 0 | 0 | 0 | 0 |
| Au grade - ore mined | g/t | 1.42 | 1.53 | 1.54 | 1.85 | 1.42 | 1.51 | 1.40 | 1.35 | 1.06 | 1.16 | 1.27 | 0 | 0 | 0 | 0 | 0 | 0 |
| Contained gold - ore mined | oz | 1,879,948 | 33,477 | 265,913 | 284,370 | 197,787 | 303,757 | 224,970 | 182,196 | 124,419 | 155,812 | 107,246 | 0 | 0 | 0 | 0 | 0 | 0 |
| **Processing schedule** | | | | | | | | | | | | | | | | | | |
| Total ore processed | kt | 41,042 | 0 | 2,815 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 3,000 | 2,227 | 0 |
| Au grade - ore processed | g/t | 1.42 | 0 | 2.37 | 2.64 | 1.74 | 2.26 | 2.01 | 1.62 | 1.16 | 1.37 | 1.22 | 0.69 | 0.69 | 0.69 | 0.69 | 0.68 | 0 |
| Contained gold - ore processed | oz | 1,879,948 | 0 | 214,471 | 254,320 | 167,545 | 218,450 | 194,183 | 156,608 | 111,802 | 131,794 | 117,700 | 66,167 | 66,167 | 66,167 | 66,167 | 48,409 | 0 |
| Au recovery | % | 83.1% | 0 | 75.8% | 75.7% | 80.2% | 84.8% | 77.4% | 85.1% | 92.5% | 94.0% | 92.7% | 85.1% | 85.1% | 85.1% | 85.1% | 86.6% | 0.0% |
| Recovered gold | oz | 1,561,902 | 0 | 162,517 | 192,517 | 134,441 | 185,172 | 150,297 | 133,276 | 103,413 | 123,927 | 109,071 | 56,340 | 56,340 | 56,340 | 56,340 | 41,912 | 0 |
| Payable gold | oz | 1,560,340 | 0 | 162,355 | 192,324 | 134,306 | 184,986 | 150,147 | 133,143 | 103,309 | 123,803 | 108,962 | 56,284 | 56,284 | 56,284 | 56,284 | 41,870 | 0 |
| **Cash flow summary** | | | | | | | | | | | | | | | | | | |
| Gross revenue | $M | 1,950.4 | 0.0 | 202.9 | 240.4 | 167.9 | 231.2 | 187.7 | 166.4 | 129.1 | 154.8 | 136.2 | 70.4 | 70.4 | 70.4 | 70.4 | 52.3 | 0.0 |
| Plus: Silver credits | $M | 46.9 | 0.0 | 4.9 | 5.8 | 4.0 | 5.6 | 4.5 | 4.0 | 3.1 | 3.7 | 3.3 | 1.7 | 1.7 | 1.7 | 1.7 | 1.3 | 0.0 |
| Less: Royalties | $M | 68.3 | 0.0 | 7.1 | 8.4 | 5.9 | 8.1 | 6.6 | 5.8 | 4.5 | 5.4 | 4.8 | 2.5 | 2.5 | 2.5 | 2.5 | 1.8 | 0.0 |
| Less: Refining & transport charges | $M | 6.7 | 0.0 | 0.7 | 0.8 | 0.6 | 0.8 | 0.6 | 0.6 | 0.4 | 0.5 | 0.5 | 0.2 | 0.2 | 0.2 | 0.2 | 0.2 | 0.0 |
| Net revenue | $M | 1,922.3 | 0.0 | 200.0 | 236.9 | 165.5 | 227.9 | 185.0 | 164.0 | 127.3 | 152.5 | 134.2 | 69.3 | 69.3 | 69.3 | 69.3 | 51.6 | 0.0 |
| Operating costs | | | | | | | | | | | | | | | | | | |
| Mining | $M | 315.3 | 0.0 | 35.0 | 33.7 | 34.8 | 44.2 | 40.1 | 33.2 | 18.0 | 23.2 | 24.1 | 6.0 | 5.9 | 5.3 | 6.2 | 5.5 | 0.0 |
| Processing and maintenance | $M | 433.4 | 0.0 | 29.3 | 31.7 | 32.3 | 33.7 | 31.7 | 32.5 | 32.1 | 31.7 | 31.2 | 31.0 | 31.0 | 31.0 | 31.0 | 23.0 | 0.0 |
| Site G&A | $M | 114.9 | 0.0 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 8.4 | 6.3 | 0.0 |
| Total operating costs | $M | 863.6 | 0.0 | 72.7 | 73.8 | 75.5 | 86.2 | 80.2 | 74.0 | 58.5 | 63.2 | 63.7 | 45.4 | 45.3 | 44.7 | 45.5 | 34.7 | 0.0 |
| Operating margin | $M | 1,058.7 | 0.0 | 127.3 | 163.1 | 90.0 | 141.7 | 104.7 | 90.0 | 68.8 | 89.3 | 70.6 | 23.9 | 24.0 | 24.6 | 23.8 | 16.9 | 0.0 |
| Construction capital | $M | 277.0 | 277.0 | | | | | | | | | | | | | | | |
| Lease payments - mine equipment | $M | 32.3 | 0.0 | 6.5 | 6.5 | 6.5 | 6.5 | 6.5 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| Sustaining capital - mine equipment | $M | 19.6 | 0.0 | 1.7 | 0.0 | 2.2 | 0.9 | 2.5 | 4.4 | 0.0 | 0.7 | 1.5 | 4.2 | 1.6 | 0.0 | 0.0 | 0.0 | 0.0 |
| Sustaining capital - TSF lifts | $M | 29.6 | 0.0 | 0.0 | 1.4 | 1.6 | 2.0 | 2.0 | 2.0 | 2.4 | 2.4 | 2.4 | 2.4 | 2.8 | 2.8 | 2.8 | 2.8 | 0.0 |
| Closure | $M | 5.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 5.0 |
| Working capital | $M | 0.0 | 4.2 | 2.6 | -0.4 | 0.7 | -0.4 | -0.5 | -0.4 | -1.1 | 0.8 | -1.1 | -0.7 | 0.1 | -0.1 | 0.0 | -3.4 | -0.3 |
| Net project cash flow before tax | $M | 695.2 | -281.2 | 116.5 | 155.7 | 79.1 | 132.8 | 94.3 | 84.0 | 67.4 | 85.4 | 67.7 | 18.0 | 19.6 | 22.0 | 21.0 | 17.5 | -4.7 |
| Taxes | $M | 88.0 | 0.0 | 0.0 | 0.0 | 2.7 | 1.0 | 14.4 | 0.0 | 11.7 | 7.9 | 14.0 | 11.5 | 5.3 | 5.3 | 5.3 | 5.3 | 3.8 |
| **Net after-tax cash flow** | **$M** | **607.2** | **-281.2** | **116.5** | **155.7** | **76.4** | **131.8** | **79.9** | **84.0** | **55.7** | **77.5** | **53.7** | **6.5** | **14.3** | **16.8** | **15.7** | **12.2** | **-8.5** |
| Cumulative after-tax cash flow | $M | | -281.2 | -164.6 | -9.0 | 67.4 | 199.2 | 279.1 | 363.2 | 418.9 | 496.4 | 550.1 | 556.6 | 571.0 | 587.7 | 603.4 | 615.7 | 607.2 |
| Cash cost per ounce (net Ag credit) | $/oz | $528 | | $422 | $358 | $536 | $440 | $509 | $530 | $541 | $485 | $559 | $781 | $779 | $768 | $783 | $804 | |
| All-in sustaining cost per ounce | $/oz | $603 | | $477 | $409 | $608 | $500 | $582 | $622 | $608 | $554 | $638 | $943 | $900 | $861 | $876 | $913 | |

| Project IRR and NPV (after-tax) | | |
|---|---|---|
| Internal rate of return | % | 35.9% |
| Net present value - 0% discount rate | $M | 607.2 |
| Net present value - 5% discount rate | $M | 410.9 |
| Net present value - 10% discount rate | $M | 278.3 |
| Payback period | Years | 2.1 |

## 22.5 Sensitivity Analysis

A sensitivity analysis was performed on the after-tax cash flows by varying key variables (gold price, Capex, Opex) to +/- 20% of the base case cash flow and each sensitivity was performed independently. The results are summarised in

Table 22.5.1, Table 22.5.2 and Table 22.5.3 and Figure 22.5.1 and Figure 22.5.2 show the sensitivity analysis on the after-tax NPV5% and IRR economic indicators.

**Table 22.5.1 Gold Price Sensitivity**

| Gold Price ($/oz) | NPV ($M) | | | IRR |
|---|---|---|---|---|
| | 0% | 5% | 10% | |
| $1,000/oz | 285 | 171 | 92 | 19.5% |
| $1,125/oz | 446 | 291 | 185 | 28.0% |
| $1,250/oz | 607 | 411 | 278 | 35.9% |
| $1,375/oz | 758 | 524 | 366 | 43.0% |
| $1,500/oz | 918 | 643 | 458 | 50.3% |

**Table 22.5.2 Capital Costs Sensitivity**

| Capex Change (%) | NPV ($M) | | | IRR |
|---|---|---|---|---|
| | 0% | 5% | 10% | |
| -20% | 665 | 466 | 331 | 47.6% |
| -10% | 636 | 439 | 305 | 41.2% |
| +0% | 607 | 411 | 278 | 35.9% |
| +10% | 578 | 383 | 252 | 31.6% |
| +20% | 549 | 356 | 225 | 27.9% |

**Table 22.5.3 Operating Cost Sensitivity**

| Opex Change (%) | NPV ($M) | | | IRR |
|---|---|---|---|---|
| | 0% | 5% | 10% | |
| -20% | 755 | 518 | 359 | 42.0% |
| -10% | 681 | 464 | 319 | 39.0% |
| +0% | 607 | 411 | 278 | 35.9% |
| +10% | 533 | 357 | 238 | 32.8% |
| +20% | 459 | 304 | 198 | 29.5% |

**Figure 22.5.1 NPV5% Sensitivity**

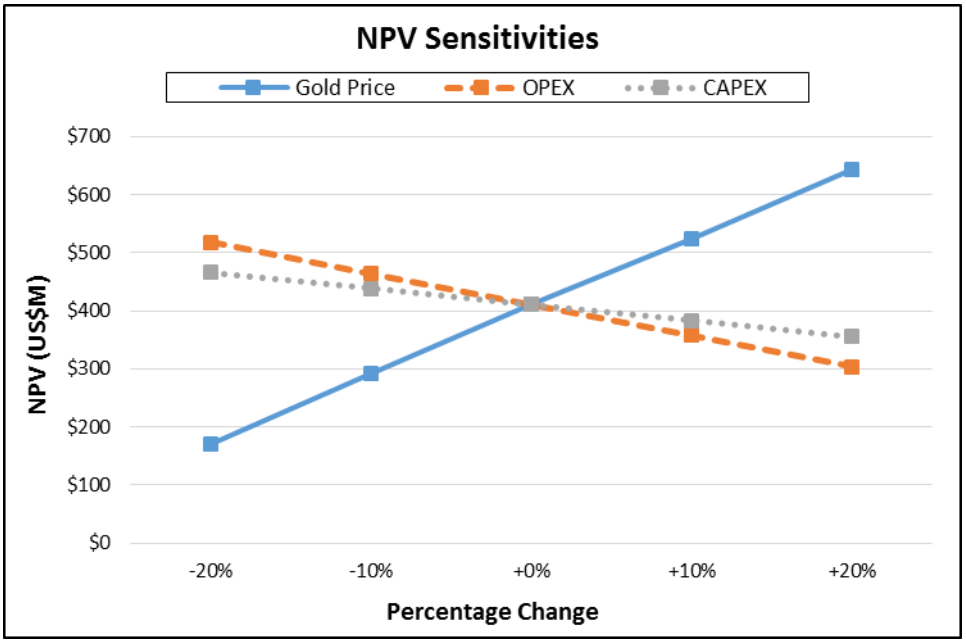
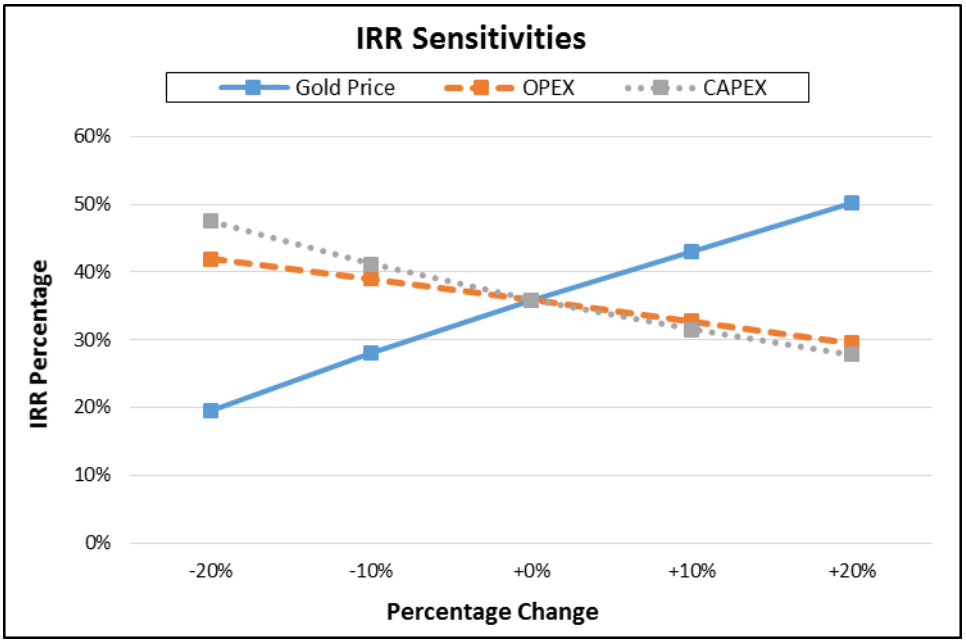



**Figure 22.5.2 IRR Sensitivity**



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**23.0 ADJACENT PROPERTIES** **23.1**

# 23.0 ADJACENT PROPERTIES

There are adjacent properties.

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# 24.0 OTHER RELEVANT DATA AND INFORMATION

## 24.1 Risks

### 24.1.1 General

***Route Survey***

A detailed transport route survey should be undertaken prior to detailed design. Road conditions vary considerably in Côte d'Ivoire, and vary from good to poor between Man and site.

***Risk Reviews***

No formal risk reviews have been undertaken during the FS. With a compressed timeline and concurrent activities, these reviews will be completed either prior to or during detailed design. It is recommended that the following occur:

- Hazop / Hazid – During detailed design with design and layout revised from FS.
- Design and Constructability – Initial post FS, detailed during detailed design phase.
- Project Risk Review – Post FS and prior to detailed design.

### 24.1.2 Resource and Geology

- Successful selective mining of the Mineral Resource will depend on the ability to discriminate between ore and waste at elevated cut-offs within the mineralised envelope. Visual selectivity is likely to be limited and diligent grade control practices including RC grade control and geological mapping of the benches will be critical to accurately defining material above the selected cut-off grades. For this style of mineralisation and the required ore block definition, the risk of possible dilution and/or ore loss of the resource without industry standard grade control procedures is high.
- For the rock waste dumps Teckraie and Verse Ouest and discontinued Aires heap leach pad, the assigned in situ bulk density was based on the available density determination measurements. Although a slightly conservative approach has been adopted for determining the assigned stockpile densities, there is a risk of sample bias and a resultant overstatement of in situ stockpile tonnes. Cube would recommend a program of large sample bulk density determinations to confirm the assigned stockpile densities.
- Additional gold recovery data should be collected from the shear hosted deposits such as Daapleu and Gbeitouo. There is a requirement to improve the understanding of the distribution of material with poor metallurgical recovery. Geo-metallurgical domaining within these deposit styles is critical and needed in order to increase the confidence in the estimation of this important modifying parameter for mine planning and processing options.

- All historical and modern drilling data has been validated and migrated into a recently introduced AcQuire database. Some outstanding data issues were identified during the estimation and require further investigation and possible adjustment in the live database. In addition, current QAQC procedures should be continued with minor improvements as recommended in the technical report in order to ensure the quality of data is sufficient for future Mineral Resource updates.

### 24.1.3 Mining

***Mine Development***

- The location of proposed low grade and waste stockpiles has been undertaken in large part in isolation of current site heap leach mining and stockpiling activities. Planned stockpile locations for current operations need to be considered.

- Sterilisation drilling should be undertaken to reduce the risk of stockpile placement on prospective areas

- Monitoring of the production and regular reconciliations can identify the potential issues earlier to rectify the production schedule. The vertical advance rates are particularly low for the largest main Daapleu open pit. Any production shortfall during operations can be sourced from the existing stockpiles if necessary.

- Underperformance of the grade control system might be an issue at the start of the operations. It is expected that the grade control system will settle after mining and processing several benches and reconciling the production for each deposit.

- The risk of higher water inflow rates would be controlled in the first instance by construction of a quality surface drainage system. Regarding in pit pump capacity, the DFS has used conservative instantaneous allowances for in pit pump rates. In the event the rates are higher than allowed for, additional pumps could be mobilised quickly and with relative ease to manage dewatering demand spikes.

- The DFS assumes that the mining surface roads will be built to a high standard, including design of a quality drainage system to manage the high rain in the area. Selected ancillary equipment resourcing and operating hours in the DFS allows for ongoing road maintenance (e.g. graders, wheel dozers, watercarts).

- The DFS allocated equipment productivity rates assume a level of reasonable local operator skill. This risk is managed in the DFS through allocation of specialist trainers, together with an upfront productivity ramp-up factors and the fact that such skills are currently available onsite.

- Poor management could result in delayed equipment delivery and commissioning timing, hence delayed mine commencement. This risk would be managed by engagement of competent OEMs who would be working under a detailed mining Execution Plan to be defined by Endeavour as a post DFS activity.

***Pit Wall Geotechnical***

Ity base case pit design parameters are derived using interpretations based on available data and could require adjustment due to variability of geological / geotechnical conditions.

It is considered that base case design parameters presented may be subject to the following geotechnical risks:

- Borehole spatial distribution and direction bias may have resulted in some structural discontinuities being under-represented / not represented in the data sets considered.
- Unknown geological structures/ units, if such exist, may negatively impact base case design parameters.
- The distribution of poorer quality rock units may be greater than currently anticipated.
- Rock weathering depths may be locally deeper than current interpretations.
- Unfavourable hydrogeological conditions, or sub-optimal results from wall depressurisation measures, may result in greater than anticipated destabilisation groundwater pressures in pit wall clays and rock at Ity.

It is expected that Endeavour will gain a better understanding of exposure to geotechnical risks once further project geotechnical investigation work is completed and open pit mining allows geotechnical pit wall mapping/ inspections to be carried out.

### 24.1.4 Process Plant

***Plant Location***

The plant site location has moved during the FS. The following is recommended prior to detailed design.

***Site Geotechnical***

- While detailed geotechnical information was available for the original plant site, only preliminary drilling has occurred during the FS. This preliminary drilling has been conducted late in the FS, and therefore the results have not been available for consideration during the study design. This program should be reviewed immediately post FS and a detailed site geotechnical program should be conducted with results available for commencement of detailed design.

***Sterilisation Drilling***

- Site sterilisation drilling for process plant and infrastructure should be completed as soon as practicable to reduce the risk of siting plant and infrastructure on a potential resource.

***Plant and Equipment***

- The process plant design has been developed concurrently with metallurgical testwork. This should be revised to identify and eliminate design risks related to equipment selection and sizing.

***Site Layout***

- Location of site infrastructure requires review post FS to ensure optimum locations are selected for both operability, functionality, as well as to minimise construction earthworks costs.

### 24.1.5 Metallurgy and Process Development

***Metallurgical Testwork***

- The metallurgical testwork program conducted during the FS should be reviewed and any gaps in data or quality of results identified. Metallurgical testwork is not finalised and the key issues remaining are the soluble copper and arsenic and the potential for further treatment of Daapleu primary ore to improve extraction.

- Any additional testwork required should be identified and be undertaken to allow results to be available for detailed design. This is of particular relevance to consider design for treatment of refractory ore to improve process plant recovery.

### 24.1.6 Infrastructure

***Camp Location***

- The existing accommodation village on site will need to be replaced due to its age, size and proximity to CIL plant location. The selected site for the new camp requires review to ensure alternative locations are considered.

***Water Treatment***

- Currently available water analysis data for FS was limited and further baseline dewatering bore analysis is required to ensure the current provisions for dewatering bore treatment pre release are correct. This should include analysis of water from current pit dewatering, as well as baseline sampling from the Cavally River both upstream and downstream of the project site. Water should be tested for basic physio-chemical parameters, nutrients and metals.

***Power***

- Power supplies in Africa can pose risks for reliability, which has forced operating mines to install additional diesel generation to maintain plant availability. CIE Energies in Côte d'Ivoire considers their power supply to be reliable. An alternative 225 kV OHPL from Man, should be considered as an alternative to the currently selected 90 kV from Danane during the detailed design phase. This option is potentially more reliable, with significantly more load capability. The additional cost of this option is considered low at this point.

***Tailings Storage Facility***

- No geotechnical data is available for the TSF site due to the location being moved during the FS. While data is available for areas adjacent to the TSF, a detailed investigation and drilling program is required prior to detailed design to reduce risks related to TSF construction and confirm suitability of selected location.

### 24.1.7 Environmental and Community

- A number of changes have occurred during the FS that conflict with information in previous ESIA submissions. This includes relocation of TSF and process plant, ROM pad, Cavally River diversion, additional river diversion required to accommodate Walter pit and new site access road. A revised ESIA will be developed to consider the impacts of these changes and will require submission to Government. This is a potential schedule impact for any required re-permitting.

### 24.1.8 Project Execution

- Execution of CIL project construction activities will have potential impacts on current heap leach operations. This requires careful consideration and planning to reduce risks associated from interaction of these activities.

### 24.1.9 Operations

- The development of the CIL project on site while the current HL operations are ongoing will require careful planning to reduce risks from both a cost, schedule and safety perspective.

## 24.2 Opportunities

### 24.2.1 Resource and Geology

- Drilling down dip and along strike of known mineralisation is recommended to further define the mineralising systems and to potentially increase the Mineral Resource base.

- In-fill and extensional drilling of the poorly defined areas of Inferred Resources would increase confidence in the grade continuity, better define the extents of the mineralisation and lead to potential upgrading of the Mineral Resource classification.

- Infill drilling of the Verse Ouest Inferred rock dump is required and will increase confidence in the global grade and allow an upgrade of the Mineral Resource classification.

- Deeper drilling in the project area will assist in testing the limits of both skarn and shear styles of mineralisation and may define new zones of mineralisation down-dip.

- Numerous near mine exploration targets such as Bakatouo, Colline Sud, and Samuel are likely to be converted to Mineral Resources with sufficient definition drilling.

- It is very possible that continued near mine exploration is likely to discover new mineralisation systems which will further provide the opportunity to increase the Mineral Resource inventory.

### 24.2.2 Mining

- The development costs for placement of waste material to the ROM pad could be reduced if existing waste stockpiling from HL operations was redirected to the final ROM location, reducing rehandling costs during CIL project development.

- The pit designs are significantly larger than the optimal pit shells due to the inclusion of the ramps and mining width. The final operating cost estimate is significantly lower than the initial estimate used in pit optimisations. The economics of the Project can potentially be improved through revised pit optimisations and pit redesign process.

- The DFS assumes 28 days lost per annum due to weather. Operationally there is potential to improve on this assumption with quality road construction and ongoing best in class road maintenance. This would have a flow on affect in areas such as increased tyre life and reduced operating costs.

- Loading and haulage engine hours can be reduced as a result of optimising tray and bucket capacities. Further, truck productivity can be improved by increasing payloads to plus 10% within the OEM recommendations through operational optimisation.

- Based on currently available data it has been estimated that 100 bores will be required over the project life. Opportunity exists to reduce this number should higher yielding aquifers exists in the final pit locations. This will be assessed post FS following a review of the most recent pit scale geological data, any additional available drilling logs, block models and geophysical data.

- A further site visit by Knight Piésold could also assist in reducing costs and effort required in current operations to dewater existing pits with improved use of available technologies such as airborne geophysics.

### 24.2.3 Metallurgy and Process Development

- Initial testwork on the Daapleu primary ore has indicated an opportunity to produce a flotation concentrate and grind this finer which may yield higher gold extraction.

### 24.2.4 Infrastructure

***Power***

- The current FS power supply assumes a grid connect via connection at Danane and 58 km 90 kV OHPL to site. An alternate opportunity exists for a 225 kV OHPL from Man which may provide a more reliable supply and increased load capacity, at similar cost.

***Airstrip***

- A conceptual design was developed by Knight Piésold for the FS. While the location of the airstrip requires further consideration, an airstrip located within the project area provides potential benefits to improve flexibility for site operations, access to site and improved security options.

### 24.2.5 Project Execution

***Project Timing***

- Prompt development of the Project at a time when construction activity in West Africa and elsewhere is below recent peaks offers the opportunity to negotiate competitive prices for equipment supply and construction contracts. At the very least project timing reduces the risk of cost escalation and overruns, reducing the risk of exceeding project contingency allowances.

***Construction Materials***

- Limestone waste is currently being stockpiled at site and is potentially of use for construction. An opportunity exists to campaign crush and screen this material for use in CIL project construction.

Construction sands can also potentially be won from the Cavally River in areas close to the plant site. While some potential areas were identified during FS site visits, a more detailed investigation prior to detailed design, and campaign screening/washing and stockpiling on site will benefit construction.

## 24.3 Other Relevant Data

There is no other data relevant to the Ity Project that has not already been discussed in the report.

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# 25.0 INTERPRETATION AND CONCLUSIONS

## 25.1 Interpretation and Conclusions

The Ity CIL Project is to be developed based on conventional open cut mining techniques and a carbon in leach processing facility. Mining has been undertaken on the project site for approximately 25 years, and is currently operated using open pit mining and heap leach (HL) processing.

Sovereign risk is understood and mitigated by the relatively short payback period of the Project. Residual environmental and social issues are low and manageable.

The Ity FS Project is an economically viable Project at current gold prices, with strong economic viability as detailed in Section 22.

## 25.2 Mineral Resource

The Mineral Resource model on which the Project is based is considered to be robust, with the key outcomes being:

- Resource classification is based on information and data provided from the Endeavour Mining database. The drilling techniques, survey, sampling, sample preparation, analytical techniques and database management / validation are well within industry standard. The database represents an accurate record of the drilling undertaken in the Project.

- Geological continuity is understood with reasonable confidence. The classification of the Mineral Resource reflects the level of confidence.

- The estimation technique and parameters are appropriate for the style of mineralisation.

The Verse Ouest stockpiled ore has been classified as Inferred by Cube Consulting during the FS, While the stockpile had been partially drilled by diamond drilling (DD) on approximately 50 x 50 m spacing, additional drilling was recommended to allow consideration and reclassification of the stockpiled ore as an Indicated resource. The additional drilling has been completed subsequent to completion of resource modelling for the FS. Cube believe that this ore (5.88 Mt at 0.99 g/t Au for 187 koz) can be reclassified as Indicated in a future revision of the resource estimate, which is likely to improve the Project value.

## 25.3 Mining

Mine planning has demonstrated the Project to be a viable and attractive prospect for development. The key outcomes of the Study are as follows:

- A 14 year mine life with progressive open pit mining of six pits completed in Year 9, and rehandling from stockpiles to feed the process plant from Years 9 to 14.

- A focus on mining and processing of higher grade ore initially, with low grade ore stockpiled for Years 9 to 14.
- Higher grade ore mined in Years 1 to 9 will be supplemented with ore stockpiled from existing heap leach operations.
- Total material movement and vertical rates of advance are readily achievable.

## 25.4 Mineral Reserves

The mineral reserve estimate on which the Project is based is considered to be robust, with the key outcomes being:

- Total mineable ore of 41.0 Mt at 1.42 g/t and a strip ratio of 2.1:1.
- Annual gold production is variable due to grade, ranging from 41.9 koz in Year 14, to 192.5 koz in Year 2, with an average of 111.6 koz per annum over LOM.
- 14 year mine life.

## 25.5 Metallurgy and Process Plant Selection

The testwork programme completed achieved its objectives of defining a process flowsheet and engineering parameters that can be used for design to allow estimation of the plant capital cost component for the feasibility study as well as defining the metal recoveries and operating consumables to allow estimation of project revenues and process operating costs.

Comminution testwork results were somewhat variable showing competency of the ore varied with degree of alteration. The recent testwork was used as the basis to provide a robust comminution flowsheet which will process the proposed ore blend at the proposed throughput rate.

The bench leaching testwork was considered reliable with a high degree of repeatability. Data indicated gold and silver extractions varied across the pits with location and with degree of alteration. The overall reagent consumption and metal recoveries were weighted by proportion of ore type to achieve overall values for the project.

Ancillary testwork on slurry thickening is now complete and supports the design values input in the design criteria.

The presence of elevated copper and arsenic in the Daapleu ores may warrant further investigation.

Testing of Daapleu primary ore to improve metal extraction was in progress at the time of writing and while interim results and conclusions are in section 13, a final report will be issued separately.

Finalisation of the plant site selection and infrastructure will be undertaken during the next phase of development.

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# 26.0 RECOMMENDATIONS

## 26.1 Geology and Mineral Resources

The Verse Ouest stockpiled ore has been classified as Inferred by Cube Consulting during the FS based on approximately 50 x 50 m spaced diamond drilling (DD). Additional drilling was recommended to allow consideration and reclassification of the stockpiled ore as an Indicated resource. The additional drilling has been completed subsequent to completion of resource modelling for the FS. Cube believes that this ore (5.88 Mt at 0.99 g/t Au for 187 koz) can be reclassified as Indicated in a future revision of the resource estimate.

Several additional targets at different stages of exploration have been identified. The additional resources when fully defined have the potential to increase the CIL project mineral resources and mineral reserves base.

Additional gold recovery data should be collected from the shear hosted deposits such as Daapleu and Gbeitouo. Geo-metallurgical domaining within these deposit styles to improve the understanding of the distribution of material with poor metallurgical recovery is critical and needed in order to increase the confidence in the estimation of this important modifying parameter for mine planning and processing options.

Outstanding data issues were identified during the estimation and require further investigation and possible adjustment in the recently introduced AcQuire database. In addition, current QA/QC procedures should be continued with minor improvements as recommended in the technical report in order to ensure the quality of data is sufficient for future Mineral Resource updates.

Successful selective mining of the Mineral Resource will depend on the ability to discriminate between ore and waste at elevated cut-offs within the mineralised envelope. Visual selectivity is likely to be limited and diligent grade control practices including RC grade control and geological mapping of the benches will be critical to accurately defining material above the selected cut-off grades.

## 26.2 Metallurgy and Process Plant

### 26.2.1 Metallurgy

While the metallurgical program was sufficient to develop the flowsheet and engineering design criteria for the study, the outstanding testwork data should be reviewed to determine if further flowsheet optimisation is required

Final results of the cyanide detox and arsenic precipitation work and the thickener testwork should be incorporated in any future design.

Review of the refractory testwork conducted on the primary ore should determine if any changes to the flowsheet are required.

### 26.2.2 Process Plant

It is recommended that further geotechnical investigation is undertaken to resolve key uncertainties associated with the prediction of Plant Site structure settlements. The following needs to be confirmed:

- The depth to groundwater. The duration of settlements below the water table will be longer than those above the water table.
- The permeability of the ground. In situ and laboratory testing results currently indicate widely different permeabilities.
- The ground conditions in the vicinity of the primary crusher are uncertain and particularly details of the infilled pit which is located very close by. Much of the ROM pad will be located over the infilled pit.
- Shrinkage and swelling of foundation materials.
- A review of predictions of ground response to structural loading, should be compared to analysis of response of existing site structures at current heap leach (HL) operations in similar ground conditions

The plant layout should be optimised as part of the detailed design to take maximum advantage of the terrain.

## 26.3 Mining and Mineral Reserves

The reported pit designs are relatively large (+20%) compared to the optimal pit shell quantities due to the inclusion of ramps, mining width and slope design factors. A review and update of the pit optimisations for the overall design slope angles and change in the initial assumptions (particularly lower mining and throughput costs in the final estimates) is recommended. This may in turn require adjustments in the pit design limits.

The resultant pit designs are for feasibility study evaluation purposes and needs to be reviewed for the variable local geological and geotechnical conditions before the implementation. It is recommended to review and update the pit designs as necessary following further geotechnical investigations and reassessment of slope angles for all the pits.

The current pit designs are based on dual lane ramp width for the full depth of the open pits. After further geotechnical investigation and pit optimisations, the ramp width for the last 30 – 40 m depth of the pit designs can be reduced to single lane (with passing bays) to decrease the waste mining requirements.

The current production schedule and reserves assume an end of heap leach pit surface for the start of the CIL operations. This assumption needs to be reviewed for the timing of the transition between heap leach and CIL operations. The planning of heap leach and CIL operations in parallel will be required if both operations are to coexist in the transitional period.

## 26.4 Pit Wall Geotechnical

Geotechnical investigation boreholes are required to be drilled at Ity to increase the level of confidence and reliability of geotechnical assessment. Geotechnical investigation boreholes should be designed to provide coverage of all sectors of the proposed pits and to refine preliminary base case wall design parameters.

Data obtained from further drilling would be combined with existing data to enable more reliable interpretation of conditions and derivation of appropriate wall angles. Geotechnical data obtained from future drilling should be cross-checked with findings and assumptions presented in this assessment report.

It is recommended that Endeavour carries out further geotechnical investigation drilling prior to committing to final Daapleu, Gbeitouo, Zia NE, Mont Ity, Ity Flat and Walter open pit designs.

It is considered essential that ongoing design re-assessments, based on information obtained using observational techniques are made during mining. It may be necessary to make local design adjustments during pit development.

## 26.5 Tailings Storage Facility

Several potential locations were identified and assessed during the FS. The final FS location was selected based on its convenient proximity to the processing facility, as well as considerable potential capital and operating cost advantages in comparison to other options.

A geotechnical site investigation for the proposed TSF location should be completed during the detailed design phase, including location of suitable construction materials. A seepage assessment should be completed based on the findings of the geotechnical investigation, and the stability analysis updated to incorporate both the findings from the geotechnical investigation and result of the seepage assessment.

The TSF design is based on an average tailings beach slope of 1% (100H:1V). However, the beach slope is heavily dependent on the grind size and the ore blend. Thus, small changes in plant performance or design, ore type, or the ore blend have the potential to change the tailings beach slope. A sensitivity analysis of a gentler beach slope (e.g. 0.4% (250H:1V) on the Stage 1 TSF configuration should be completed during the detailed design phase.

The supernatant was found to be of a poor quality, exceeding the reference water quality guidelines for release and livestock drinking water for fifteen parameters, with several found to be many times the threshold values. Cyanide was also found to be extremely high as the sample is understood to be pre-detox. Therefore, it is recommended that post-detox tailings solids and supernatant samples be analysed prior to final design.

The detox sample may reduce the concentration of certain metals together with the cyanide. However, on the basis of the current results, the proposed seepage control measures need to be reviewed with particular emphasis on control at seepage in the operating supernatant pond areas during the subsequent design phase.

## 26.6 Site Roads

A geotechnical investigation should be completed prior to construction to determine suitable sources of wearing course material for site roads construction (haul roads and general site access roads).

## 26.7 Airstrip

Detailed design of the site airstrip should be completed once a location is nominated. A geotechnical investigation should be completed subsequent to locating the airstrip to determine suitable sources of pavement material for airstrip construction (haul roads and general site access roads).

## 26.8 Water Management

A dewatering bore installation schedule has been developed based on the current mine plan. This includes the installation of an additional 100 bores over the LOM. The bore numbers are based on a median yield of 11 $m^3/h$ obtained from the existing Ity pit dewatering bores. The number of dewatering bores could be reduced by the installation of higher yielding bores.

A more detailed analysis of the existing geological data is recommended for the refinement and improvement of the proposed bore locations and pit inflow predictions. Potential sources of additional information include the existing block geological model, additional borehole logs and geophysical data. Additional field investigations and testing should take place in a phased manner and successful bores could be adopted as long term dewatering bores. The groundwater model should be updated as additional investigation and monitoring data are obtained.

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# 27.0 REFERENCES

## 27.1 Supporting Documents

The following documents have been developed during the course of the Feasibility Study and support the designs and schedules included in this report. They are available on request from Endeavour.

### 27.1.1 Cube Consulting Resource

- Endeavour_Ity_QAQC Analysis_October 2012 to June 2016_Final.

### 27.1.2 Mining Solutions

- Endeavour Mining_Ity CIL Project_Mining Resourcing and Costing Consultant Report (Calsta)_October 2016.
- Ity Pit Optimisation Inputs and Results_October 2016.
- Ity Pit Waste Dump Designs_October 2016.
- Ity Preliminary Production Schedules_October 2016.
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- Ity Final Production Schedule v2c_October 2016.

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- Geotechnical Assessment Open Pit Mining Daapleu,Gbeitouo,Zia NE,Mont Ity, Ity Flat & Walter Deposits_September 2016_V 1.0.

### 27.1.4 Lycopodium Minerals

- SNC Lavalin Draft Study Report (628121-0000-30RA-0001 Feb 2016).
- ALS metallurgical testwork report (17202).
- OMC Comminution Report (7753).
- Site contour data from EDV.
- Reference KMZ files for site location.

### 27.1.5 Knight Piésold

- Feasibility Study – Geotechnical, Tailings and Water Management Report_November 2016_Rev B.

### 27.1.6 Endeavour Mining

- Roche_SEIA Final Report Validation October 25th 2013.
- 2D Consulting_SEIA Daapleu Gbeitouo La Mancha CI 2015.
- 2D Consulting_SEIA Infrastructures SMI 2015.
- SNC Lavalin_SEIA Synthetic Report 2016.

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### 27.2.1 Cube Consulting

- Abzalov, M.Z., 2006. Localised Uniform Conditioning (LUC): A New Approach for Direct Modelling of Small Blocks, Mathematical Geology38 (4), p. 393-411.
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### 27.2.3 Peter O Bryan & Associates

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**28.0 DATES, SIGNATORIES AND CONSENT OF QUALIFIED PERSONS 28.1**

# 28.0 DATES, SIGNATORIES AND CONSENT OF QUALIFIED PERSONS

## CERTIFICATE OF QUALIFIED PERSON

I, *Adriaan Almero (Attie) Roux* (Pr.Sci.Nat.) as an author of this report entitled "Ity CIL Project NI 43-101 Technical Report" (the Technical Report), prepared for Endeavour Mining and dated 9 December 2016, do hereby certify that:

1) I am *the Chief Operating Officer for Endeavour Mining*. My office address is *Croisement Boulevard Latrille et Rue du Lycée Technique, Abidjan, Cote D'Ivoire.*

2) I graduated with a Higher National Diploma in Metallurgical Engineering from the University of Johannesburg, South Africa, in 1980.

3) I am a Registered Professional Natural Scientist (Pr.Sci.Nat.) with the South African Council for Natural Scientific Professions (SACNASP) since 2004 – Membership No. 400156/04.

4) I have since graduation worked as a Metallurgical Engineer, General Manager and currently, the Chief Operating Officer and have over 40 years' experience in the mining industry. My relevant experience includes both Project and Operational work at Anglo American, De Beers Consolidated, AngloGold Ashanti, Adamus Resources and Endeavour Mining Corporation and includes extensive project and operational experience in gold, uranium, sulphuric acid and diamonds processing and in roles of Metallurgist, Operational Management, Technical Head, General Management, Consulting Metallurgist, Director and currently, Chief Operating Officer with area of control spread over multiple countries.

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

6) I have visited the Ity Project site many times during the past year and recently in October 2016.

7) I am responsible for the preparation of sections 1.15, 19 and 20 of the Technical Report and jointly for section 27.1.

8) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

9) I have been involved in the current operations of the Ity mine but I have not been involved in the previous PFS Report on the Ity CIL Project.

10) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

11) At the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th day of December, 2016



Adriaan Roux

**CONSENT OF QUALIFIED PERSON**

9 December, 2016

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

Dear Sirs/Mesdames:

**Re: Filing of Technical Report by Endeavour Mining (the "Issuer")**

I, **Adriaan Roux**, am responsible for preparing or supervising the preparation of all or a portion of the technical report entitled "**[Endeavour Mining. NI 43-101 Technical Report Ity CIL Project Cote D'Ivoire]**" dated effective **9 December**, 2016 (the "**Technical Report**").

Pursuant to Section 8.3 of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, this letter constitutes my consent to the public filing of the Technical Report with each of the British Columbia Securities Commission, the Alberta Securities Commission, the Ontario Securities Commission and the Toronto Stock Exchange and I acknowledge that the Technical Report will become part of the Issuer's public record.

DATED this **9** day of **December**, 2016.

Yours truly,



**Adriaan Roux**

**Lycopodium Minerals Pty Ltd**
ABN: 34 055 880 209

Level 5, 1 Adelaide Terrace
East Perth, Western Australia 6004
Australia

PO Box 6832
East Perth, Western Australia 6892
Australia

T: +61 (0) 8 6210 5222
www.lycopodium.com.au

## CERTIFICATE OF QUALIFIED PERSON

I, David John Gordon, as an author of this report entitled Ity CIL Project, Cote D'Ivoire NI 43-101 Technical Report (Revision B), prepared for Endeavour Mining and dated x December 2016, do hereby certify that:

1) I am Group Manager - Process, Lycopodium Minerals Pty. Ltd. My office address is Level 5, 1 Adelaide Terrace, East Perth, Western Australia, 6004.

2) I am a graduate of the Western Australian Institute of Technology with a B. App. Sc. in Engineering Metallurgy, 1983.

3) I am a Member of the Australasian Institute of Mining and Metallurgy, membership number 108413 and registered as a Fellow with that Institute. I have worked as a metallurgist, operations manager, laboratory manager and process engineer/manager for a total of thirty years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - managed and interpreted results from numerous leaching and mineral processing testwork programs on gold and silver ores
   - involved in the process design of treatment plants for over 14 years.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Ity Project site.

6) I am responsible for all preparation of Sections: 2, 3, 13, 17, 18.1, 18.7-14, 21.3, 22, 25.5, 26.2 and jointly responsible for Sections 1, 5, 21.1, 21.2, 24, 25, 26 and 27 of the Technical Report.

7) I am an independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report on the Ity Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this ninth day of December 2016.



**DAVID JOHN GORDON**
**GROUP MANAGER - PROCESS**

**Lycopodium Minerals Pty Ltd**
ABN: 34 055 880 209

Level 5, 1 Adelaide Terrace
East Perth, Western Australia 6004
Australia

PO Box 6832
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T: +61 (0) 8 6210 5222
www.lycopodium.com.au

9 December 2016

British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

Dear Sirs/Mesdames

**FILING OF TECHNICAL REPORT BY ENDEAVOUR MINING (THE "ISSUER")**

I, **David John Gordon**, am responsible for preparing or supervising the preparation of all or a portion of the technical report entitled **"[Endeavour Mining. NI 43-101 Technical Report Ity CIL Project Cote D'Ivoire]"** dated effective 9 December, 2016 (the **"Technical Report"**).

Pursuant to Section 8.3 of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, this letter constitutes my consent to the public filing of the Technical Report with each of the British Columbia Securities Commission, the Alberta Securities Commission, the Ontario Securities Commission and the Toronto Stock Exchange and I acknowledge that the Technical Report will become part of the Issuer's public record.

DATED this ninth day of December, 2016.

Yours truly



**DAVID JOHN GORDON**

mining solutions

**Mining Solutions Consultancy Pty Ltd**
**ABN 16 082 728 836**
5 Salter Point Parade, Salter Point, Western Australia 6152

Phone (+61 8) 9450 3399
Fax (+61 8) 9450 3377
www.miningsolutions.com.au

## CERTIFICATE OF QUALIFIED PERSON

I, *Tamer Dincer*, as an author of this report entitled Ity CIL Project, Cote D'IvoireNI 43-101 Technical Report, prepared for Endeavour Mining and dated 9 December 2016, do hereby certify that:

1) I am *a Principal Mining Consultant with Mining Solutions*. My office address is *5, Salter Point Parade, Salter Point, WA, 6152 Australia*.

2) I am a graduate of the Middle East Technical University *with a Masters in Mining Engineering*.

3) I am a Member of the *Australasian Institute of Mining and Metallurgy (AusIMM)*, membership number 202560 and registered as a Fellow. I have worked as a mining engineer and mining consultant for a total of thirty years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Mine planning including pit optimisation, mine design, mine scheduling, estimation of mining costs and project economic performance evaluation.
   - Feasibility study experience, over a period of 25 years,with a range of open pit projects located in Australia and Africa.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have visited the Ity Project site.

6) I am responsible for all of preparation of Sections 1.7 and 1.8, 15, 16 (except 16.2 and 16.3), and jointly for 1.12, 1.13, 1.17, 21, 24, 25, 26 and 27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report on the Ity Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 9th day of December 2016



Tamer Dincer

**CONSENT OF QUALIFIED PERSON**

9 December, 2016

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

Dear Sirs/Mesdames:

**Re: Filing of Technical Report by Endeavour Mining (the "Issuer")**

I, **Tamer Dincer**, am responsible for preparing or supervising the preparation of all or a portion of the technical report entitled "**[Endeavour Mining. NI 43-101 Technical Report Ity CIL Project Cote D'Ivoire]**" dated effective 9 **December**, 2016 (the "**Technical Report**").

Pursuant to Section 8.3 of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, this letter constitutes my consent to the public filing of the Technical Report with each of the British Columbia Securities Commission, the Alberta Securities Commission, the Ontario Securities Commission and the Toronto Stock Exchange and I acknowledge that the Technical Report will become part of the Issuer's public record.

DATED this **9** day of **December**, 2016.

Yours truly,



**Tamer Dincer**

# Cube Consulting Pty Ltd

## CERTIFICATE OF QUALIFIED PERSON

I, Mark Zammit, as an author of this report entitled Ity CIL Project, Cote D'Ivoire NI 43-101 Technical Report, prepared for Endeavour Mining and dated 9 December 2016, do hereby certify that:

1) I am a Principal Consultant Geologist with Cube Consulting Pty Ltd. My office address is Level 4, 1111 Hay Street, West Perth, WA 6005, Australia.

2) I am a professional geologist having graduated with a BSc (Hons) in Geology, from the University of Western Australia(1992), a Post Graduate Diploma in Business (Management Studies) from Edith Cowan University (2005) and a Graduate Certificate in Geostatistics from Edith Cowan University (2008).

3) I am a Member of the Australian Institute of Geoscientists, membership number 3843. I have practiced my profession as a geologist for the past 24 years in the mineral resources sector and engaged in the assessment, development and operation of mineral projects both within Australia and overseas.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I personally inspected the Ity Gold Project between 18th June 2016 and 22ndJune 2016.

6) I am responsible for all of preparation of Sections 1.3 to 1.6, 4, 6, 7, 8, 9, 10, 11, 12 and 14 and contributed to Sections 1, 24, 25, 26 and 27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in previous Technical Reports on the Ity Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 9th day of December 2016



Mark Zammit

Cube Consulting.Pty Ltd

www.cubeconsulting.com
PO Box 877 West Perth WA 6872 Australia
Level 4, 1111 Hay Street, West Perth WA 6005Australia
Phone +61 8 9442 2111 Fax +61 8 9442 2110
ABN 84 094 321 829

**CONSENT OF QUALIFIED PERSON**

9 December, 2016

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

Dear Sirs/Mesdames:

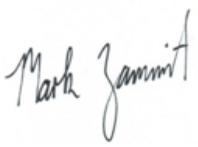
**Re: Filing of Technical Report by Endeavour Mining (the "Issuer")**

I, **Mark Zammit**, am responsible for preparing or supervising the preparation of all or a portion of the technical report entitled "**[Endeavour Mining. NI 43-101 Technical Report Ity CIL Project Cote D'Ivoire]**" dated effective **9 December**, 2016 (the "**Technical Report**").

Pursuant to Section 8.3 of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, this letter constitutes my consent to the public filing of the Technical Report with each of the British Columbia Securities Commission, the Alberta Securities Commission, the Ontario Securities Commission and the Toronto Stock Exchange and I acknowledge that the Technical Report will become part of the Issuer's public record.

DATED this **9** day of **December**, 2016.

Yours truly,



**Mark Zammit**

**CERTIFICATE OF QUALIFIED PERSON**

I, Peter O'Bryan, as an author of this report entitled Ity CIL Project, Cote D'Ivoire NI 43-101 Technical Report, prepared for Endeavour Mining and dated 9 December 2016, do hereby certify that:

1) I am a Principal Geotechnical Engineer with Peter O'Bryan & Associates. My office address is43 York Street, Subiaco, WA 6008, Australia.

2) I am a professional Geotechnical Engineer having graduated as a Bachelor of Engineering (Mining) from the University of New South Wales in 1981and a Master of Engineering Science (Rock Engineering) from James Cook University of North Queensland in 1990.

3) I am a Member and Chartered Professional of the Australasian Institute of Mining and Metallurgy (Member 20335). I have practiced as a Geotechnical Engineer for the past 35 years in the mineral resources sector and engaged in pre-mining and operational geotechnical assessment and review of mineral projects both within Australia and overseas.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Ity Project site.

6) I am responsible for all of preparation of Section16.3and contributed to Sections 1, 24, 26 and 27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in previous Technical Reports on the Ity Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 9th day of December 2016



**Peter O'Bryan**

Peter O'Bryan & Associates.

**CONSENT OF QUALIFIED PERSON**

9 December, 2016

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

Dear Sirs/Mesdames:

**Re: Filing of Technical Report by Endeavour Mining (the "Issuer")**

I, **Peter O'Bryan**, am responsible for preparing or supervising the preparation of all or a portion of the technical report entitled "**[Endeavour Mining. NI 43-101 Technical Report Ity CIL Project Cote D'Ivoire]**" dated effective **9 December**, 2016 (the "**Technical Report**").

Pursuant to Section 8.3 of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, this letter constitutes my consent to the public filing of the Technical Report with each of the British Columbia Securities Commission, the Alberta Securities Commission, the Ontario Securities Commission and the Toronto Stock Exchange and I acknowledge that the Technical Report will become part of the Issuer's public record.

DATED this **9th** day of **December**, 2016.

Yours truly,

**Peter O'Bryan**

Knight Piésold
CONSULTING

## CERTIFICATE OF QUALIFIED PERSON

I, *David John Toomey Morgan* as an author of this report entitled Ity CIL Project, Cote D'Ivoire NI 43-101 Technical Report (Revision B), prepared for Endeavour Mining and dated 9 December 2016, do hereby certify that:

1) I am *the Managing Director of Knight Piésold Pty Ltd.* My office address is *Level 1, 184 Adelaide Terrace, East Perth, Western Australia 6004.*

2) I am a graduate of the *University of Manchester, (BSc, Civil Engineering, 1980) and the University of Southampton (MSc, Irrigation Engineering, 1981).*

3) I am a Member of the *Australian Institute of Mining and Metallurgy (Australasia)*, membership number *202216* and registered as a Chartered Professional Engineer and member of the Institution of Engineers Australia (Australia, 974219). I have worked as a Civil Engineer for over 35 years. My relevant experience for the purpose of the Technical Report is:

   - Project Director – Akyem Gold Project.
   - Project Director – Geita Gold Mine.
   - Project Director – Ahafo Gold Mine.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have visited the Ity Project site.

6) I am responsible for all of preparation of Section Numbers 16.2, 18.2-18.6 and contributed to Sections 1.17, 5.6, 21, 24, 26 and 27 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report on the Ity CIL Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Knight Piésold
CONSULTING

Dated this 9th day of December 2016

DAVID J T MORGAN

The Institution of Engineers, Australia

David J T Morgan
MIEAust CPEng
Chartered Professional Engineer
Membership No. 974219

**CONSENT OF QUALIFIED PERSON**

9 December, 2016

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
Toronto Stock Exchange

Dear Sirs/Mesdames:

**Re: Filing of Technical Report by Endeavour Mining (the "Issuer")**

I, **David John Toomey Morgan**, am responsible for preparing or supervising the preparation of all or a portion of the technical report entitled "**[Endeavour Mining. NI 43-101 Technical Report Ity CIL Project Cote D'Ivoire]**" dated effective **9 December**, 2016 (the "**Technical Report**").

Pursuant to Section 8.3 of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects*, this letter constitutes my consent to the public filing of the Technical Report with each of the British Columbia Securities Commission, the Alberta Securities Commission, the Ontario Securities Commission and the Toronto Stock Exchange and I acknowledge that the Technical Report will become part of the Issuer's public record.

DATED this **9** day of **December**, 2016.

Yours truly,



**David John Toomey Morgan**