# NEW LIBERTY GOLD PROJECT, BEA MOUNTAIN MINING LICENCE SOUTHERN BLOCK, LIBERIA, WEST AFRICA, DEFINITIVE PROJECT PLAN

**REPORT PREPARED IN ACCORDANCE WITH THE GUIDELINES OF NATIONAL INSTRUMENT 43-101 AND ACCOMPANYING DOCUMENTS 43-101.F1 AND 43-101.CP.**

**Prepared For**

## AUREUS MINING INC.


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**Report Prepared by**

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# Table of Contents

# List of Tables

# List of Figures

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# NEW LIBERTY GOLD PROJECT, BEA MOUNTAIN MINING LICENCE SOUTHERN BLOCK, LIBERIA, WEST AFRICA, DEFINITIVE PROJECT PLAN

## 1 SUMMARY

### 1.1 Introduction

This Technical Report on the New Liberty Gold Project (New Liberty or the Project) within the Bea Mountain Mineral Development Agreement (Bea-MDA) property in Liberia, West Africa, has been compiled by SRK Consulting (UK) Ltd (SRK), for Aureus Mining Inc. (Aureus or the Company). Aureus, through its ownership of Bea Mountain Mining Corporation (Bea), has a 100% interest in the Bea-MDA in which the Project is located.

The Project is an advanced stage gold project which Aureus currently envisages will produce its first gold in May 2015.

This report has been compiled by SRK and describes the Project in its current stage of development, presents SRK's opinions on the Mineral Resource and Mineral Reserve and current production forecast and presents an updated economic model and cash flow forecast which reflects the recently completed work (which SRK has also reviewed).

### 1.2 History

The first exploration work at the property was carried out by Golden Limbo and comprised desktop studies, a review of satellite imagery, target selection and acquisition of a portfolio of possibilities. In 1997 Mano River Resources (Mano) collected preliminary channel samples across the artisanal workings, where primary rock was exposed. During reconnaissance work numerous targets for gold mineralisation were identified through geological mapping, supported by soil and stream geochemical sampling programmes.

Exploration by Aureus at the Bea-MDA property since 2011 has followed a systematic process of reconnaissance work, grab-sampling followed by soil geochemistry, mapping, trench sampling and eventually drilling. Aureus also completed a feasibility study in October 2012 and subsequent to this carried out additional work with a view to optimising the Project. This optimisation work was reported in the report titled New Liberty Gold Project, West Africa, Updated Technical Report, dated 3 July 2013.

Since 2013, the Company has continued to conduct further evaluation work at New Liberty, including grade control drilling to produce a better geological understanding of the orebody and also commenced construction. A revised more optimal mine plan has also been produced reflecting this grade control drilling and this report reflects the additional work and the current status of the Project generally.

## 1.3 Geology

The mineralisation being targeted by Aureus comprises typical Upper Archaean to Lower Proterozoic greenstone belt-hosted lode gold mineralisation. These deposits are often referred to as orogenic and are characterised by the presence of a combination of gold-quartz veins and disseminated mineralisation.

Specifically, drilling completed to date has allowed the delineation of a 'silicified metamorphosed ultrabasic suite' (SMUS) zone which contains the gold mineralisation. The SMUS strikes approximately 097° over the western half of its drilled extent while in the east it swings slightly towards the south (105°). Southerly dips are typically in the range 65°-80°.

The SMUS zone boundaries are more confidently defined near surface, with the benefit of higher drilling density and supported by surface mapping. At the 0 m RL elevation, horizontal thicknesses typically range from 40 m to 90 m, occasionally reaching 120 m. With increasing depth, however, the western half appears to widen significantly, reaching a horizontal width of around 250 m at approximately -400 m RL.

Intersections of anomalous gold grades occur in places across the full profile of the SMUS zone but elevated grade intersections of potentially economic interest are much more restricted in number and extent. Correlations between these higher grade intersections, typically above 0.5 g/t Au, reveal an orientation that is broadly aligned with the SMUS, although in some cases they drift slightly obliquely to the SMUS contacts in both strike and dip. The dimensions of these zones of elevated grade (mineralised zones) are strongly anisotropically planar, ranging in width between a few to sometimes 10 m – 15 m, while typically extending hundreds of metres in both strike and dip.

## 1.4 Mineral Resources

The most up to date Mineral Resource estimate for the Project was produced by AMC and reported in October 2012 according to CIM Standards and at a 1.0 g/t Au cut-off. This is presented in the table below.

**Table 1-1: AMC Mineral Resource (as at 1 October 2012)**

| | Measured | | | Indicated | | | Measured and Indicated | | |
|---|---|---|---|---|---|---|---|---|---|
| | Tonnes | Au | | Tonnes | Au | | Tonnes | Au | |
| **Minzone** | (Kt) | (g/t) | (Koz) | (Kt) | (g/t) | (Koz) | (Kt) | (g/t) | (Koz) |
| **M401** | 651 | 4.77 | 100 | 5,468 | 3.88 | 683 | 6,118 | 3.98 | 783 |
| **M402** | | | | 874 | 2.51 | 71 | 874 | 2.51 | 71 |
| **M501** | | | | 2,317 | 2.43 | 181 | 2,317 | 2.43 | 181 |
| **M503** | | | | 486 | 6.93 | 108 | 486 | 6.93 | 108 |
| **M504** | | | | | | | | | |
| **Total** | **651** | **4.77** | **100** | **9,145** | **3.55** | **1,043** | **9,796** | **3.63** | **1,143** |

| | Inferred | | |
|---|---|---|---|
| | Tonnes | Au | |
| **Minzone** | (Kt) | (g/t) | (Koz) |
| **M401** | 3,060 | 3.2 | 314 |
| **M402** | 130 | 3.6 | 15 |
| **M501** | 1,120 | 2.6 | 92 |
| **M503** | 1,300 | 3.6 | 152 |
| **M504** | 120 | 5.1 | 20 |
| **Total** | **5,730** | **3.2** | **593** |

| Key to Minzone Codes | |
|---|---|
| M401 | Larjor + Latiff + Kinjor main zone |
| M402 | Kinjor footwall zone |
| M501 | Marvoe main zone |
| M503 | Marvoe western hanging wall zone |
| M504 | Marvoe central hanging wall zone |

Notes 1.CIM definitions were used for Mineral Resources.
2. A cut-off of 1.0 g/t Au is applied for all zones.
3. Due to rounding, some columns or rows may not add up exactly to the computed totals.
4. Mineral resources that are not mineral reserves do not have demonstrated economic viability

While SRK notes that the Mineral Resource extends below the designed pit and may require to be developed by underground mining, SRK is confident that the Mineral Resource reported above reflects the available data and quality of this and the geological interpretations made and that it has been derived using appropriate and industry standard methods.

Aureus has completed a programme of grade control drilling since this Mineral Resource estimate was derived in October 2012. While the information obtained during this programme has not been used to update this estimate, SRK has reviewed this information and considers that the Mineral Resource remains robust when reviewed in the light of this. Further, SRK considers the information suggests that it may be possible to delineate more continuous zones with at least the same grade, if not slightly higher, as grade control data becomes available and as mining progresses.

## 1.5 Mineral Reserves

The most up to date Mineral Reserve estimate was derived for the Project by AMC in May 2013. This was based on the same geological block model generated by AMC that formed the basis of the Feasibility Study, and is summarised above, but took account of additional geotechnical data and slope designs developed by AMC subsequent to that.

The study focused on the open pitable portion of the Project and assumed that conventional open-pit gold mining techniques would be employed.

AMC prepared pit designs for the Project on the basis of pit optimisations carried out using the Mineral Resource estimate adjusted for dilution and ore loss, slope designs based upon the geotechnical assessments, and estimates of the contract mining costs, processing costs, site general and administration costs, metallurgical recovery. The pit optimisations only considered the Measured and Indicated mineral resources. All Inferred mineral resources were treated as waste.

While an updated mining plan has been developed for the Project, as commented upon later in this report, the final open pit design remains as derived in 2013 and consequently the reported Mineral Reserve also remains unchanged and is summarised in the table below.

**Table 1-2: AMC Mineral Reserve Estimate (as at 20 May 2013)**

| Reserve Category | Oxide / Fresh | Tonnes (Mt) | Au Grade (g/t) | Au Ounces (koz) |
|---|---|---|---|---|
| Proven | Oxide | - | - | - |
| | Fresh | 0.7 | 4.4 | 99 |
| Probable | Oxide | 0.3 | 2.3 | 18 |
| | Fresh | 7.5 | 3.3 | 806 |
| Total | Oxide | 0.3 | 2.3 | 18 |
| | Fresh | 8.2 | 3.4 | 905 |
| **Grand Total** | **Mineral Reserves** | **8.5** | **3.4** | **924** |
| Waste | Oxide | 13.3 | - | - |
| | Fresh | 118 | - | - |
| Total | | 131 | - | - |
| Strip Ratio | (W:O) (t/t) | 15.5 | - | - |

Notes: 1. CIM definitions were used for mineral reserves
2. A cut off of 0.8 g/t Au is applied for all zones
3. Due to rounding, some columns or rows may not add up exactly to the computed totals

SRK has reviewed the work completed to produce the above Mineral Reserve estimate and considers this to have been appropriately derived, to reflect the information currently available and the current mine plan and that it remains relatively robust to changes in costs and gold price.

## 1.6 Mining Plan

The Mineral Reserve is contained within an open pit which will be mined using conventional drill-and-blast, load and haul mining techniques.

A new mine plan has recently been developed by Aureus to extract the Mineral Reserve in a way which compensates for the delay in the commencement of processing operations and improves the Project's economics through the development of two starter pits (maximising operational face lengths and reduced haulage distances for waste).

SRK considers the updated mining plan to be a robust plan in terms of the production assumed and to appropriately reflect work completed by Aureus and additional information obtained plus the current delayed status of construction due to factors largely beyond the Company's control.

The revised mine plan assumes more gold is produced than in the previously reported plan over the early periods through the mining of more ore tonnes at a higher gold grade and processing more tonnage at a higher grade by selectively feeding from ROM stockpiles, thereby producing more ounces earlier in the mines life. The revised mine plan also reduces the Life of Mine (LOM) by some four months.

## 1.7 Mineral Processing

The proposed process flowsheet is an industry-standard arrangement consisting of two-stage crushing, ore stockpiling, milling and classification, gravity and CIL, cyanide detoxification, tailings disposal, acid wash, elution, electrowinning and gold room, carbon regeneration, reagent preparation, storage and dosing, oxygen, air and water systems.

Subsequent to the completion of the Feasibility Study metallurgical test work in 2013, Aureus employed DRA to scope and manage a testwork programme investigating the mobility of arsenic during the cyanide leach and Detox processes within the New Liberty Process Plant. Under the supervision of Digby Wells the Arsenic mobility on the Tailings Storage Facility (TSF) was also investigated and the geochemical test work was scoped by them. DRA facilitated the metallurgical preparation of these samples for testing at ALS Environmental.

This test work confirmed that a process which includes an $SO_2$/Air detox step in combination with 2.5kg/t ferric chloride addition on CIL tailings results in arsenic leaching with subsequent precipitation of a stable arsenate compound. A solids sample containing 1200ppm arsenic was treated using the optimum conditions (JR 1256) and has achieved an arsenic in solution value of 0.005ppm after 23 weeks in a kinetic column test.

Based on the kinetic column test results for test JR1256, the New Liberty process plant design was updated to allow for 1000m$^3$ tank in which cyanide destruction and arsenic leaching will take place. The additional leach tank is the same volume as the CIL tanks. In addition three 260m$^3$ tanks were added to treat the detox/arsenic leach product stream. These tanks allow for additional ferric sulphate addition and pH correction of tailings.

A large portion of the arsenic optimisation test work was conducted using ferric chloride reagent as a source of ferric ion. In test JR1256 the option of using ferric sulphate as opposed to ferric chloride was investigated and no difference in metallurgical response has been noted between the two reagents after 23 weeks in a kinetic column test. The plant operating cost estimate has been based on the use of ferric sulphate as a source of ferric ion.

## 1.8 Environmental Management and Permitting

An Environmental and Social Impact Assessment (ESIA) was undertaken from Q4 2010 to Q2 2012 to investigate the local environmental and social situation existing prior to the development of the Project, and to determine the likely positive and negative impacts of associated with its development.

The ESIA was completed using accepted international standards (notably those of the World Bank and World Health Organisation), best practice principles and techniques, and the Liberian EPA ESIA Procedural Guidelines (2006), and the Environmental Permit for the Project was granted by the Liberian Environmental Protection Agency in October 2012.

Subsequent to the completion of the ESIA and the granting of the Projects Environmental Permit, Aureus embarked upon an optimisation process which resulted in the relocation of various mining associated infrastructure. The revised infrastructure locations remain within the area permitted for mining and as a result of this, Digby Wells were engaged to revise and update the ESIA. Following a review of these documents by the IFC prior to their investment in the Company in 2014, an addendum to the updated ESIA was also produced and submitted to the EPA during March 2014.

The results of the impact assessment indicate that the management and mitigation of environmental and social impacts associated with the project are amenable to standard technical solutions. No issue has been identified that presents a technical challenge beyond that which is regularly encountered and resolved by comparable mining operations elsewhere in Africa.

Environmental management of the Project will be an evolving process over the life of the mine. In particular, the environmental management and mitigation measures and the monitoring programme outlined in this EIS will be updated annually for continual improvement to occur and for management practices to remain current and aligned with Liberian legislation and industry good practice.

In July 2014, following investment in the Company by the IFC, a Critical Habitat Assessment and Terrestrial Ecological Biodiversity survey was carried out by a team of international specialists at the New Liberty Project site. The report concluded that the overall study area comprises Natural Vegetation, with a smaller percentage of Modified Habitat, most of which was restricted to the mine footprint area. In total, 4.9% of the total Natural Habitat will be directly lost due to mine activities.

Development of the Project has required the resettlement of two relatively small villages, Kinjor and Larjor, which were located within the area of the proposed mine pit.

The Project Resettlement Action Plan (RAP) to address the above resettlement impacts was granted by the Liberian Environmental Protection Agency in March 2013, and the relocation of the local communities was completed in September 2014 to a temporary area within the RAP site. Construction at the RAP is nearing completion for the permanent houses within the village, and once complete, these will be handed over to the local community.

At the time of writing this report, the area of the old Kinjor village had been cleared, allowing for the commencement of mining operations and grade control drilling.

The Community Development Plan (CDP) was completed in December 2012 and was approved by the Liberian EPA in January 2013. Based on the development opportunities assessed, a number of potential CDP projects were identified, which include vegetable production, cashew nut production and coconut production as well as other further agricultural activities. A number of these initiatives have been started on site, including an agricultural cooperative producing vegetables and a woodworking and brickmaking cooperative producing construction materials. Where feasible the CDP will be expanded to incorporate the community development aspects of the Bea- MDA.

The development and operation of the Project will have both positive and negative impacts on the socio-economic structure of the project area and its environments. The development of the project will bring much needed investment and development opportunities with consequent impacts on the employment and the affected communities.

## 1.9 Construction Status

The construction of the Project infrastructure is now well advanced. Notably, the diversion dams and cuttings for the Marvoe Creek Diversion are essentially complete, the wall of the TSF is 75% complete and work is progressing well on the penstock line, with current forecasts to completion in line with the planned commissioning dates. The civil and earthworks for the process plant are complete, and steel, plate work and piping well advanced, with electrical and instrumentation following behind on schedule for planned commissioning dates.

The Project schedule envisages the first production of gold in May 2015. In SRK's opinion, while there remains a risk that construction may take longer than planned to complete, particularly given that the Ebola situation has not yet been fully resolved, and while there is further work to be done, the Project currently remains on schedule to achieve this.

## 1.10 Economic Analysis

Aureus has developed a financial model in order to evaluate the economics of the Project. SRK confirms that the inputs to the financial model have been appropriately derived from, and reflect the investigations of the various studies, as commented on in this report.

The financial model reflects pre-finance cashflows, allows for working capital and is based on a detailed analysis of gold processing throughput.

A net present value (NPV) has been calculated for the expected cash flows from commencement of commercial production (i.e. from 1 July 2015 and excluding all initial Project capital costs) through the application of Discounted Cash Flow (DCF) techniques to pre-financing cash flows derived from the inputs and assumptions presented in this report. All figures are presented in Q4 2014 real USD terms.

A government royalty of 3% of net revenue has been assumed. The financial model is reported on the basis of 100% of the Project, with no consideration of the free carried interest. The model assumes a corporation tax rate of 25% which is taken from the restated and amended Mineral Development Agreement.

A summary of cash flow modelling is presented below.

**Table 1-3: Cash Flow Model Summary**

| Description | Units | Project Totals/Averages |
|---|---|---|
| Recovered gold | Koz | 858 |
| Mill processing life | Years | 8 |
| Net smelter revenue (after royalty) | USD m | 1,079 |
| Operating costs | USD m | 594 |
| Net operating cash flow | USD m | 486 |
| Initial capital costs | USD m | 172 |
| Net post-tax cash flow | USD m | 401 |
| Post-tax NPV (5%)* | USD m | 328 |
| Average cash cost per ounce | USD/oz | 692 |
| Internal Rate of Return (IRR)+ | (%) | 21% |
| Payback Period+ | (years) | 3.7 |

*present value of expected cash flows from commencement of commercial production before debt servicing and repayment

+IRR and Payback Period include all capital cost cash flow from the start of construction. 72% of capital cost has been incurred as at December 31, 2014.

## 1.11 Conclusions and Recommendations

SRK has concluded that the Project is both technically feasible and economically viable.

There remain some risks to the Project. Notably, given that the Ebola situation has not yet been fully resolved, there may be further construction delays which could impact on revenues and capital costs and there is also a risk that some of the assumed reduction in mining costs may not be achieved. Notwithstanding this, the strong and experienced team on site has enabled Aureus to negotiate what has been a very difficult period and has minimised the impact of factors which have been outside of the Company's control and that assuming this team is retained, the risks identified above should be minimised.

The principal conclusion arising from this review of the Project is that the construction of the Project should continue and that Aureus should focus on this in order to ensure that the Project is constructed as envisaged and in the planned timeframe.

# 2 INTRODUCTION

This Technical Report on the New Liberty Gold Project (New Liberty or the Project) within the Bea Mountain Mineral Development Agreement (Bea-MDA) property in Liberia, West Africa, has been compiled by SRK Consulting (UK) Ltd (SRK), for Aureus Mining Inc. (Aureus or the Company). Aureus, through its ownership of Bea Mountain Mining Corporation (BEA), has a 100% interest in the Bea-MDA in which the Project is located.

The Project is an advanced stage gold project which was the subject of a Feasibility Study completed by Aureus which was reported in October 2012. Subsequent to this, additional work has been carried out with a view to optimising the Project. This optimisation work was reported in the report titled New Liberty Gold Project, West Africa, Updated Technical Report, dated 3 July 2013.

Since 2013, the Company has continued to conduct further evaluation work at New Liberty, including grade control drilling to produce a better geological understanding of the orebody and also commenced construction. A revised more optimal mine plan has also been produced reflecting this grade control drilling and this report reflects this additional work and the current status of the Project generally. Aureus currently envisages that the Project will produce its first gold in May 2015.

This Technical Report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgement on the CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

This report has been compiled by SRK and describes the Project as currently envisaged, presents SRK's opinions on the Mineral Resource and Mineral Reserve and production forecast as currently forecast and presents an updated economic model and cash flow forecast derived by Aureus and reviewed by SRK reflecting the recently completed work.

SRK is part of an international group (the SRK Group), which comprises over 1,500 professional staff offering expertise in a wide range of engineering and scientific disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. SRK has offices in UK, Sweden, Turkey, Russia, South Africa, North and South America, Kazakhstan, China, India and Australia. SRK has a significant amount of experience in undertaking technical-economic audits of, and monitoring of, mining and processing projects on behalf of banks and potential investors throughout the world and also in producing independent technical reports such as this in relation to the raising of equity or satisfying stock exchange listing requirements.

The Qualified Persons (QPs) who take responsibility for the Technical Report are Dr Mike Armitage BSc, MIMMM, C.Eng, C.Geol of SRK plus Robin Welsh Pr.Eng, MSAIEE, and Glenn Bezuidenhout FSAIMM of DRA Projects (Pty) Ltd (DRA), and Graham Trusler Pr.Eng of Digby Wells Environmental (Digby Wells). All of these people meet the requirements of a QP and are independent as defined in NI 43-101.

Specifically, Dr Mike Armitage takes responsibility as QP for Sections 1-12, 14-16, 19 and 21 to 27; Glenn Bezuidenhout for Section 13, Section 17 and part of Section 25; Robin Welsh for Section 18 and part of Section 25 and Graham Trusler for Section 20.

Dr Mike Armitage visited the site from 20-23 November 2012, Robin Welsh visited the site from 21–22 May 2012, and 21–25 January 2013, and 5–8 March 2013, and 14–17 May 2013, and 11–14 June 2013, and 10-15 March 2014 and 7-12 June 2014. Glenn Bezuidenhout visited the site between 21–22 November 2012. Graham Trusler visited the site 20–21 March 2013, and 13–17 May 2013 and 10-14 February 2014.

SRK's opinion, effective as of March 25$^{th}$ 2015, is based on information provided to SRK by Aureus and reflects various technical and economic conditions at the time of writing.

This report is based on technical information, which requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or affiliate of Aureus and neither SRK nor any affiliate of SRK has acted as advisor to Aureus or its affiliates in connection with the Project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

# 3 RELIANCE ON OTHER EXPERTS

SRK has confirmed that the Mineral Resources and Reserves reported herein are within the mining licence boundaries given below. However, SRK has not conducted any legal due diligence on the ownership of the licences. Rather, with respect to the Mineral Development Agreement (MDA) between The Republic of Liberia and Bea Mountain Mining Corporation (Section 4 of this report), SRK has relied on copies of documents provided by Aureus that confirm the terms of the Agreement.

With respect to the granting of a Class A Mining Licence to Bea Mountain Mining Corporation (Section 4 of this report), SRK has relied on copies of a document provided by Aureus that confirm the terms of the Licence.

Most of the factual text for this Technical Report covering Items 4–12 was originally prepared by Aureus and provided to AMC Consultants (UK) Limited (AMC) during 2013 for review. Aureus also supplied supporting technical documents which AMC used to verify this data where practical. This information has been subsequently reviewed, updated and verified by SRK for the purposes of this Technical Report. In addition, Sections 16, 19, and 21-24 were largely been prepared by Aureus and reviewed and verified by SRK for the purposes of this report. Notwithstanding this SRK has, where possible, independently verified the data provided, and has also undertaken a site visit to review the physical evidence for the deposit.

# 4 PROPERTY DESCRIPTION AND LOCATION

The property is located within The Republic of Liberia which is situated on the coast of the south-west corner of West Africa and bordered by Sierra Leone, Guinea and Cote d'Ivoire. Liberia lies between longitude 7º30' and 11º30' west, latitude 4º18' and 8º30' north, and covers a surface area of 111,369 km². The capital is Monrovia and, as of the 2008 Census, had a population of 3,476,600.

## 4.1 Location

The Bea-MDA property is situated 90 km north-west of the capital in Grand Cape Mount County, in the north-western portion of Liberia, approximately longitude 11$^{o}$ west, 7$^{o}$ north, as shown in Figure 4-1. The Project is situated within the Bea-MDA property, the UTM coordinates of which are shown in Table 4-1.


Source: *Aureus, 2012*

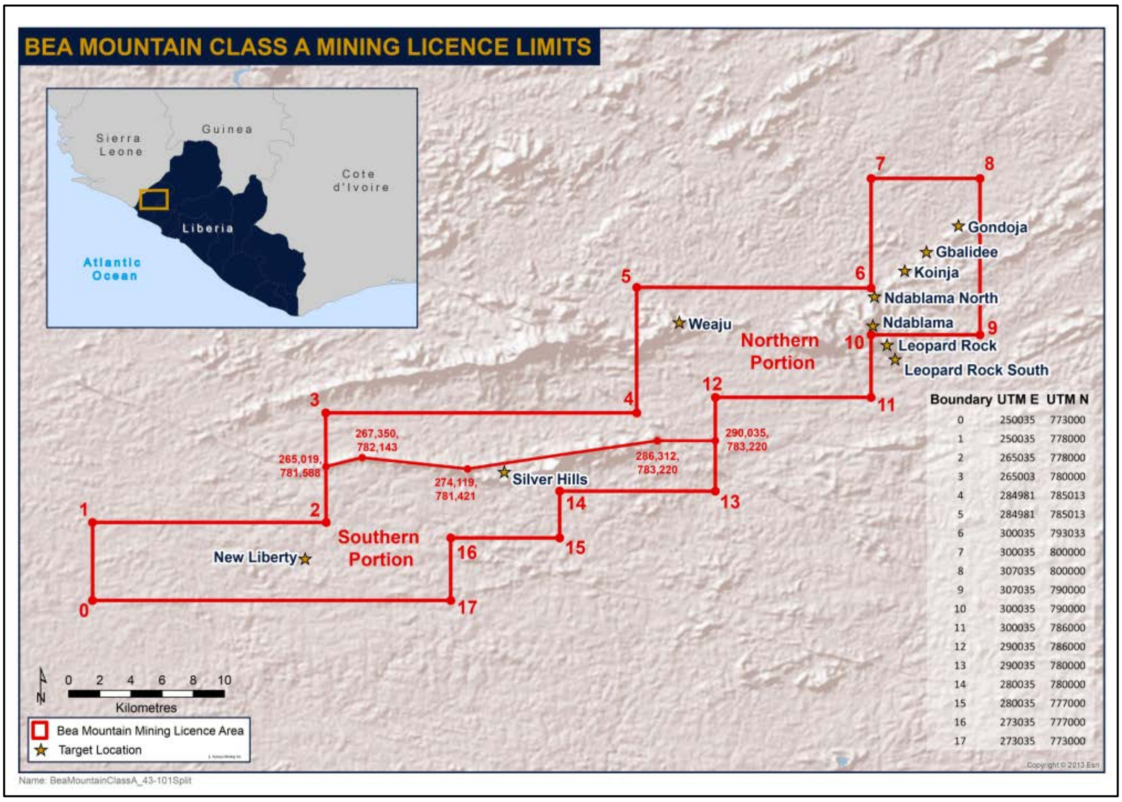
**Figure 4-1: Location of the Bea-MDA Property in Liberia**

## 4.2 Property Description

The Bea-MDA property covers an area of 457 km² with boundaries described by cadastral and cartographic survey in maps at the Ministry of Lands, Mines and Energy Republic of Liberia. The Project location is show in Figure 4-2, along with the other targets which are currently the subject of exploration by Aureus but which are not discussed in this report. The Bea-MDA property, which is covered by a Class A mining licence, has been reduced from a prior exploration lease which covered a total of 1,000 km².

This report relates to the Southern Block of the BEA-MDA property, the boundaries of which are illustrated in Figure 4-2. The Northern Block boundary with the adjacent Southern Block coincides with Silver Hills Mountains, trend EW and form a natural barrier between the blocks.

The UTM coordinates of the boundary between the blocks of the BEA-MDA property are as follows: 265019E, 781588N; 267350E, 782143N; 286312E, 783220N and 290035E, 783220N.




Source: *Aureus, 2014*

**Figure 4-2: Class A Mining Licence Limits**

**Table 4-1: WGS84 UTM Zone 29N Vertices of the Class A Mining Licence**

| Boundary | UTM E | UTM N |
|---|---:|---:|
| 0 | 250035 | 773000 |
| 1 | 250035 | 778000 |
| 2 | 265035 | 778000 |
| 3 | 265003 | 778000 |
| 4 | 284981 | 785013 |
| 5 | 284981 | 785013 |
| 6 | 300035 | 793033 |
| 7 | 300035 | 800000 |
| 8 | 307035 | 800000 |
| 9 | 307035 | 790000 |
| 10 | 300035 | 790000 |
| 11 | 300035 | 786000 |
| 12 | 290035 | 786000 |
| 13 | 290035 | 780000 |
| 14 | 280035 | 780000 |
| 15 | 280035 | 777000 |
| 16 | 273035 | 777000 |
| 17 | 273035 | 773000 |

## 4.3 Ownership

BEA has a 100% interest in the current Bea-MDA, which was signed with the Liberian Government in November 2001. BEA was previously a wholly owned subsidiary of African Aura Mining Inc.(African Aura), formerly called Mano River Resources Inc but is now a wholly owned subsidiary of Aureus. On April 13, 2011 African Aura completed a Plan of Arrangement ("Arrangement") under the Business Corporations Act (British Columbia) pursuant to which it transferred its gold assets, 30,792,770 shares in Stellar Diamonds plc and USD10.6 million cash (the "Transferred Assets") to Aureus and African Aura was renamed Afferro Mining Inc.

Under the Arrangement, among other things, the Transferred Assets were acquired by Aureus, and each participating shareholder received new common shares in Afferro and Aureus in exchange for the African Aura common shares held by such shareholder on the basis of one new Afferro common share and one Aureus common share for each African Aura common share held by such shareholder. Table 4-2 summarises the ownership history.

**Table 4-2: Ownership History**

| Date | Company | Comments |
|---|---|---|
| August 1995 | KAFCO | Assigned rights in area to Golden Limbo |
| 18 November 1996 | Golden Limbo | Assigned rights to BEA |
| 22 November 1996 | BEA | Approval received |
| 22 April 1998 | BEA | Bea-MDA defined as 1000 km² |
| 28 November 2001 | BEA | Bea-MDA reduction to 457 km² came into effect |
| 29 July 2009 | BEA | Granted a Class A Mining Licence |

## 4.4 Title

The mineral exploration and exploitation rights defined by the Bea-MDA originally became effective on April 22, 1998. Previously the ground was held by a Liberian entity known as KAFCO. In August 1995 KAFCO received government approval to assign its rights to the licence to Golden Limbo Rock Liberia Ltd (Golden Limbo). On 18 November 1996, Golden Limbo assigned its rights to the licence to BEA which was subsequently approved by the government on 22 November 1996. In April 1998, in anticipation of a new Mining Code, BEA replaced the existing licence and assignment, and entered into a specially-negotiated Exploration Agreement. Upon ratification of the new Mining Code in 2000, BEA, in keeping with the new law, reduced the size (acreage) of the licence and entered into the present governing Agreement. The Bea-MDA came into effect on 28 November 2001 and has an initial term of 25 years, which may be extended for successive 25-year terms.

Under the terms of the Bea-MDA, there is a 3% royalty payable to the Republic of Liberia calculated on a production basis. In addition, the Republic of Liberia is entitled to receive, free of charge, an equity interest on BEA's operations equal to 10% of its authorized and outstanding share capital without dilution (i.e. a 10% "carried interest"). African Aura through its subsidiary was required to pay the Republic of Liberia USD0.08 per acre per year as a rental fee for the Exploration Licence. Due to the civil unrest in the country, the Ministry of Land, Mines, and Energy suspended the exploration period as from July 2002 until 4 January 2005.

During the initial term of the Bea-MDA, BEA was required to make minimum exploration expenditures of USD1.40 per acre per year albeit that excess expenditures in a given year could be credited against succeeding years work requirements. The Bea-MDA provides BEA the right to free access to public land and will assist BEA in cases where access to private lands is necessary. Prior to the commencement of exploitation and production BEA is required to provide an Environmental Impact Statement to the Minister, detailing any adverse effects operations may have on the environment and along with plans to mitigate such effects. From time to time BEA is required to submit detailed plans "for the protection, correction and restoration of the water, land and the atmosphere".

BEA was granted a Class A Mining Licence (the Licence) on July 29, 2009. The annual licence fee for the Licence, based on the production area of 457 km² ("the Production Area"), amounts to USD0.80 per acre, which equates to USD90,146 per annum (1 km² = 247.1 acres). The Licence for the Production Area selected by the operator of the Project must remain valid and effective for the unexpired portion of the Bea-MDA and any extensions thereof. The Licence allows BEA to commercially exploit minerals found in the Production Area and all other activities incidental thereto, including the design, construction, installation, fabrication, operation, maintenance and repair of infrastructure, facilities and equipment and the mining, excavation, extraction, recovery, handling, beneficiation, processing, milling, stockpiling, transportation, export and sale of minerals.

BEA will need to apply for and acquire normal operating licences and permits for the mining operation, including licences associated with explosive storage and use, abstraction and discharge of water and construction. All licences required for the purpose of the operations to date have been applied for and granted.

## 4.5 Environmental Management

To the extent known, the area has only has only limited artisanal workings, and no historical environmental issues.

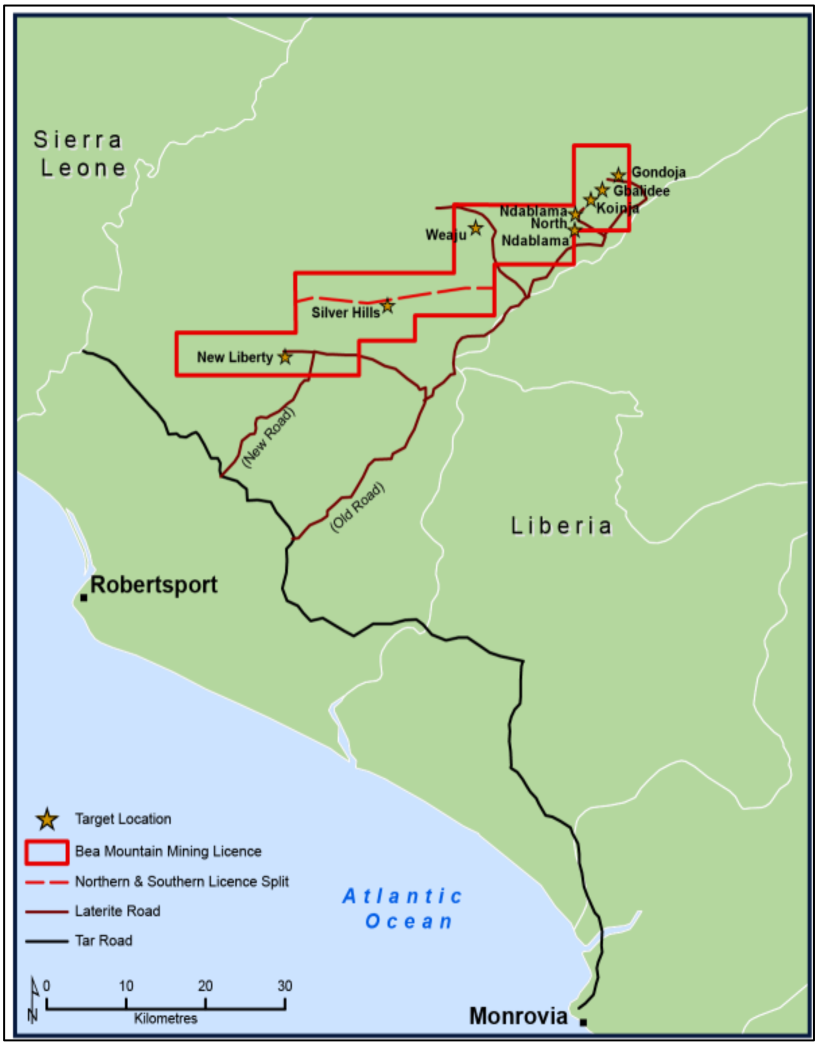
Baseline data collection for the ESIA was initiated in the fourth quarter of 2010 and was conducted during both the wet and dry seasons. The ESIA, as per Liberian legislation, included a Public Participation Process (PPP). An Environmental Impact Statement (EIS) was submitted to the Environmental Protection Agency of Liberia (EPA) in July 2012, which was approved by the EPA in October 2012. The approval of the EIS is required under the terms of the Agreement and is required prior to the commencement of exploitation and production.

Subsequent to the completion of the ESIA and the approval of the EIS by the EPA, a mine optimisation study was conducted in early 2013. Aureus then commissioned Digby Wells Environmental (Digby Wells) to undertake further detailed specialist studies and update the ESIA report. The updated ESIA report was submitted to the EPA in October 2013 as per the MDA requirements and all permits remained valid. Prior to the investment in the Company by the International Finance Corporation (IFC) in 2014, an addendum to the updated ESIA was also produced and submitted to the EPA during March 2014.

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

## 5.1 Accessibility

The Project is accessible by vehicle from Monrovia, with approximately 80 km of paved road to the town of Danieltown and a further laterite section of 20 km to the Project. Aureus has recently upgraded the laterite section of road and installed five new culvert-type bridges to facilitate the transportation of equipment to site. Secondary roads on the licence, built by Aureus, provide access across the property. The sandy nature of the roads allows all year round access, including during the height of the rainy season.




Source: *Aureus, 2012*

**Figure 5-1: Road Access to the Project**

## 5.2 Physiography

The Bea-MDA property contains both primary and secondary forest, as well as some grassland and farmland. The topography ranges from around 50 m above mean sea level (amsl) to a maximum of 600 m amsl with the majority of the licence area being composed of gently undulating plains which reach less than 200 m amsl. There are also two prominent east-west ridges of resistant rock units, termed the Bea Mountain Range and the Tokani Mountain Range respectively.

Vegetation consists of tropical trees which attain heights of 30 m to 40 m above the forest floor, with thick undergrowth common. The (primary rain forest is mainly in the mountainous area while gently undulating plains are mostly covered by secondary forest. In common with the majority of Liberia, deep lateritic soils limit rock outcrop to streams and the more rugged hill areas.

## 5.3 Climate

The equatorial climate is hot all year-round with heavy rainfall from May to October but with a short interlude between mid-July and August. During the winter months of November to March, dry dust-laden Harmattan winds blow inland. The average annual rainfall along the coastal belt is over 4,000 mm but this declines to 1,300 mm at the forest-savannah boundary in the north (Bongers, F et al, 1999). The temperatures range from the low 20 ºC's during the rainy season to warm (low 30 ºC's) during the dry season. Exploration and construction activities have been able to continue throughout the rainy season.

## 5.4 Infrastructure

The 1989-2003 civil wars in Liberia had a devastating effect on the country's economy, with neglect and damage during the civil strife resulting in much of Liberia's physical infrastructure being destroyed. Reconstruction began during 2003 and there has since been a recovery in critical infrastructure sectors such as power, water and transport.

The Liberian Electricity Corporation currently supplies 10MW in Monrovia, with private generators meeting the remaining requirement. The Port of Monrovia, which is privately run under a concession from the government, is one of four main ports in Liberia and is the only port with cargo and oil handling facilities and can accommodate third-generation container ships.

Liberia has approximately 10,600 km of road networks throughout the country, of which 650 km are paved highway. Some of the dirt roads in the interior of the country were constructed in the 1990s, chiefly by Asian timber companies. These roads were well built and maintained at the time.

The 490 km of rail line in Liberia was primarily constructed to haul iron-ore from interior mining areas to the ports. Much of the Bong Mine rail is still usable, while ArcelorMittal has renovated the Nimba Railway to the port of Buchanan which is located some 250km to the southeast of the Project.

Broadband internet services are available in Monrovia and in some smaller urban centres. The Aureus camps at the Project use 2560-1024 kbps and 512-512 kbps Vsat VOIP facilities. Cellular phone coverage in Liberia is good within the major urban areas and is widespread throughout much of the country. There are two cell towers which provide signal to the Project site.

The increasing presence of mining operations in Liberia is expanding the supply of mining personnel and mining services, such as drilling contractors, equipment rental and services, engineering services and a trained labour force. In addition, there is a mobile West African work force in the mining industry.

## 5.5 Local Resources

In the area around the Bea-MDA property, covering Grand Cape Mount County between the localities of Gbah and Gbesse, large tracts of land are devoted to rubber farms, however, these are located mainly outside the licence area. Closer to the Sierra Leone border the major farming activity is oil palm cultivation.

There are several small-scale artisanal alluvial diamond and gold operations within the BEA-MDA property.

# 6 HISTORY

The numerous artisanal mining sites that occur within the Bea-MDA property highlight the potential for local, 'source' gold mineralisation. At the Project, to the extent known, there are only limited artisanal workings, with the majority of miners seeming only interested in alluvial gold. Once these workings encounter bedrock or solid quartz, they are abandoned.

The first exploration work was carried out by Golden Limbo and comprised desktop studies, a review of satellite imagery, target selection and acquisition of a portfolio of possibilities. In 1997 Mano River Resources (Mano) collected preliminary channel samples across the artisanal workings, where primary rock was exposed. During reconnaissance work numerous targets for gold mineralisation were identified through geological mapping, supported by soil and stream geochemical sampling programmes. An overview of exploration activities across the licence is show in Figure 6-1.

Two previous, historical, mineral resource estimates were prepared for the Project, the first by ACA Howe International Ltd. (ACA Howe) in 2000 (Table 6-1), and the second by Lower Quartile Solutions (Pty) Ltd. (LQS) in 2006 (Table 6-2).

The ACA Howe estimate was prepared to "Australasian Institute of Mining and Metallurgy Joint Ore Reserve Committee's (JORC) code standards", and is presented here as an historical estimate. Estimates were completed for the three principal geological zones, and were based on relatively shallow drilling, with the deepest mineralised intercept reported at 104 m, and the resource quoted to a maximum depth of 150 m.

**Table 6-1: ACA Howe 2000 Historical Mineral Resource Estimate**

| Category | Tonnes (Kt) | Grade (g/t Au) | Gold (Koz) |
|---|---|---|---|
| Indicated | 1,078 | 5.23 | 181 |
| Inferred | 3,009 | 4.02 | 427 |

Notes:

1. Cross-section method employed.
2. No cut-off used, as mineralised zone taken.

The LQS estimate was produced in support of a study by MDM Engineering Group Limited (MDM), was reported according to CIM Standards and was based on significantly more drillholes than the ACA Howe estimate. This is summarised in Table 6-2 below.

**Table 6-2: LQS 2006 Historical Mineral Resource Estimate**

| Category | Tonnes (Kt) | Grade (g/t Au) | Gold (Koz) |
|---|---|---|---|
| Measured | 6,658 | 3.49 | 746 |
| Indicated | 6,875 | 2.88 | 637 |
| Total | 13,533 | 3.18 | 1,383 |

Notes:

1. A cut-off of 1.0 g/t Au is applied for all zones.

SRK has not reviewed the above estimates, and they are presented here for information only. To the extent known, there was no gold production on the Bea-MDA property by the previous licence holders.

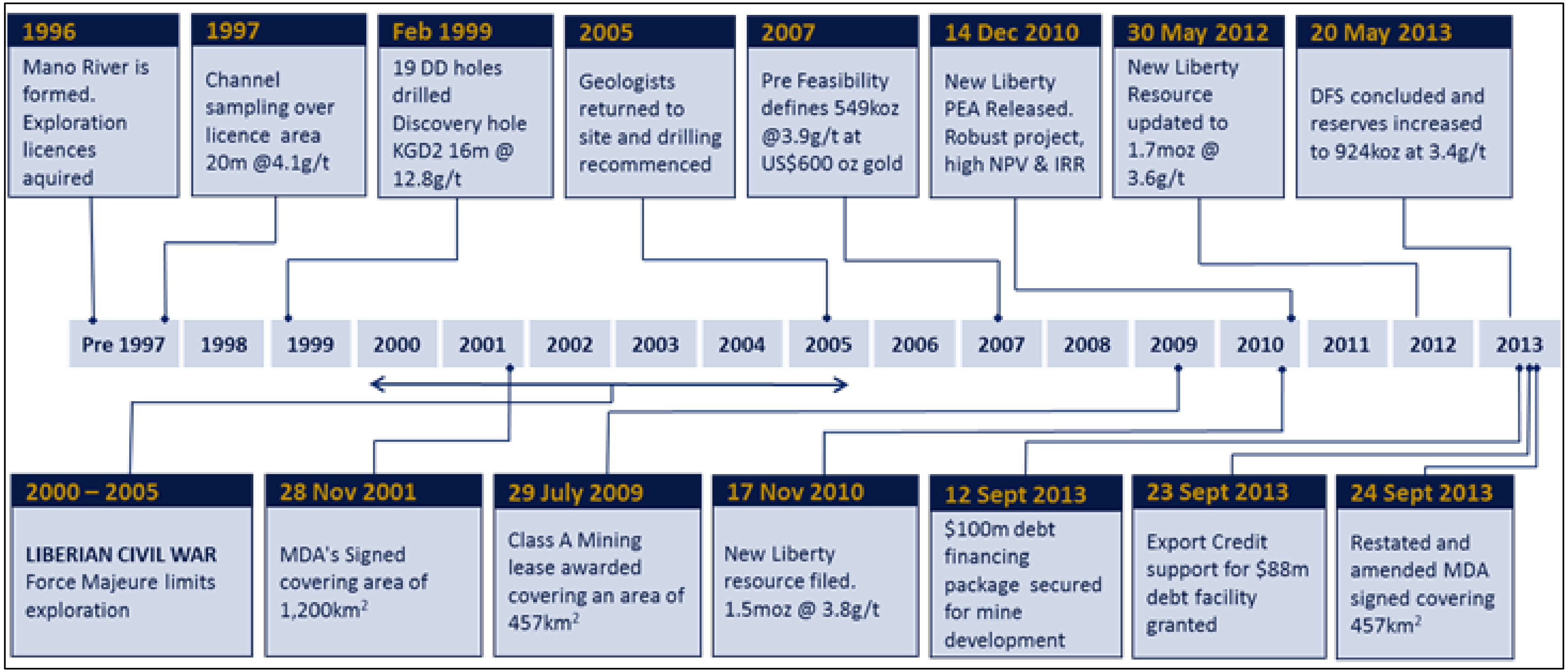
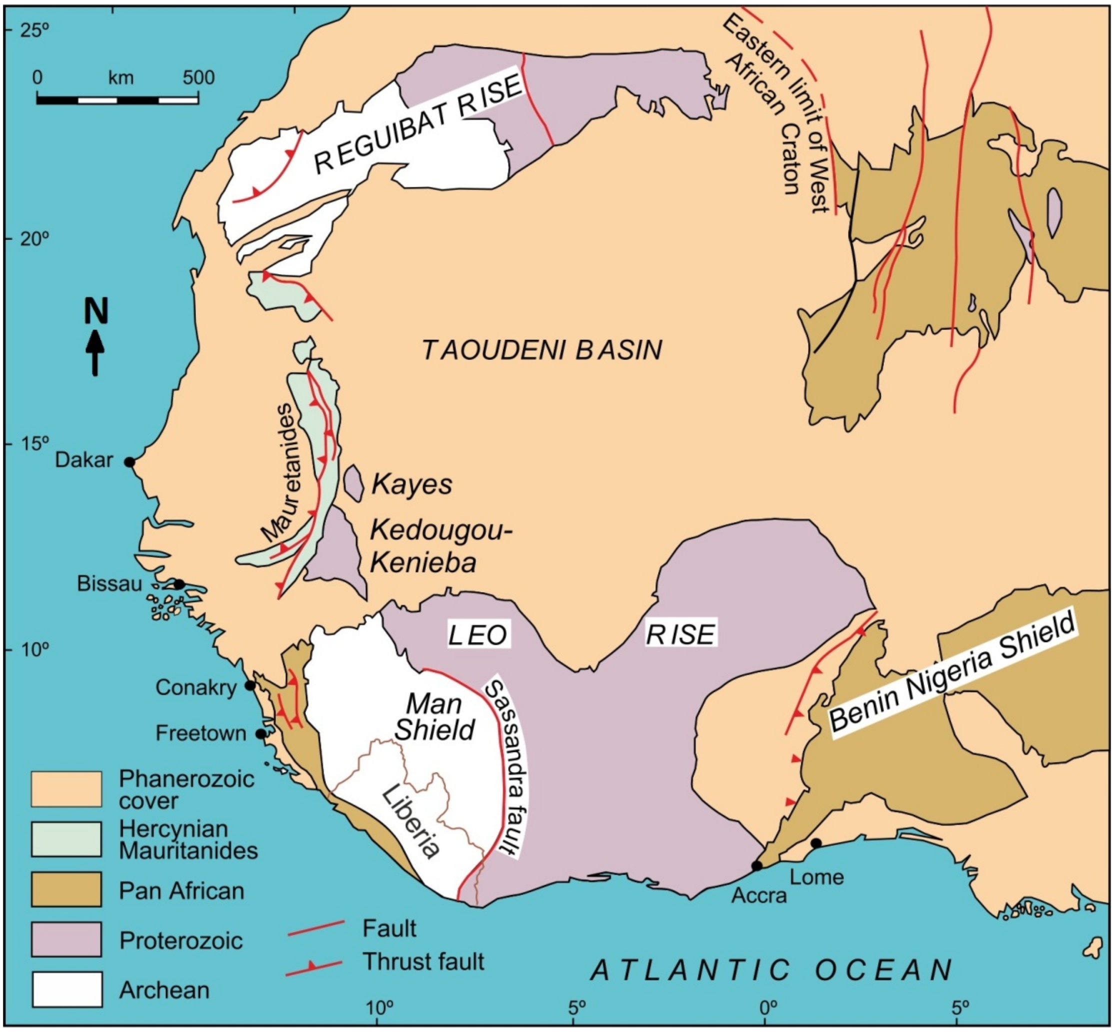



**Figure 6-1: History of Exploration at Bea Mountain Property**

# 7 GEOLOGICAL SETTING AND MINERALISATION

## 7.1 Regional Geology

Geologically, Liberia is situated within the West African Craton, which has remained stable since about 1.7 Ga. This craton consists of two major basement domains; the Reguibat Shield (in the north around Mauritania) and the Man Shield (3.0 to 2.5 Ga). The two shields are separated by the Taodeni Basin which is of Proterozoic to Paleozoic age, while the Man Shield lies to the west of the Proterzoic Birimian Belts. Liberia is located in the Man Shield (Figure 7-1).

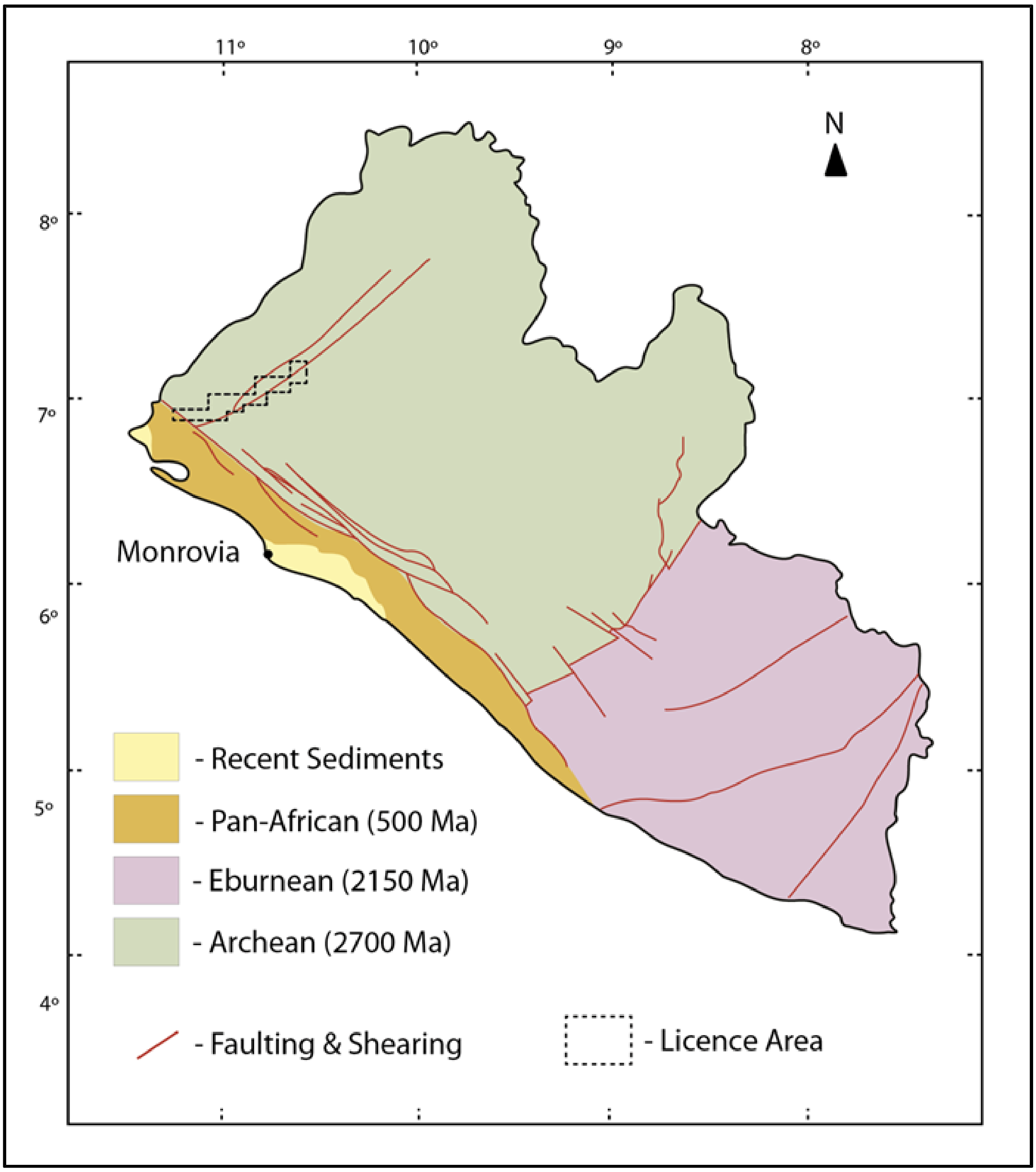



Modified from: *Milési et al. 1992*

**Figure 7-1: Regional Geological Setting**

To the east of Liberia is a Birimian-age (2.1 Ga) proto-continent that accreted onto Africa during the Eburnean Orogeny (Milési, J-P, et al 1992). Pan African units extend along the southern edge of the country, representing the formation of Gondwana (500 Ma). The west of Liberia is underlain by Archaean granites and granitic gneisses, as well as greenstone belts (metamorphosed mafic and ultramafic rocks, bounded by granites and gneisses suites representing the remains of volcanic belts), Figure 7-2. The Archaean rocks have been subjected to deformation and shearing, with the principal structures acting as conduits for mineralising fluids.

An Archaean mobile belt along the border between north-west Liberia and Sierra Leone represents a collision orogeny, with a north-east trend and a north-westerly directed closure. Oceanic crust, overlain by sediments, is preserved as tectonic inliers and forms the Bea Mountains, Kpo Range and associated greenstone belts. Later Eburnean (2.15 Ga) deformation is also found to the south-east. A major, crustal scale, north-westerly-trending shear zone in the south-western part of the country cuts across the regional trend of the Archaean mountain belt. The interference of these two tectonic elements produced complex structures with a strong rotational component of deformation and formed large and long-lived traps for mineralisation.



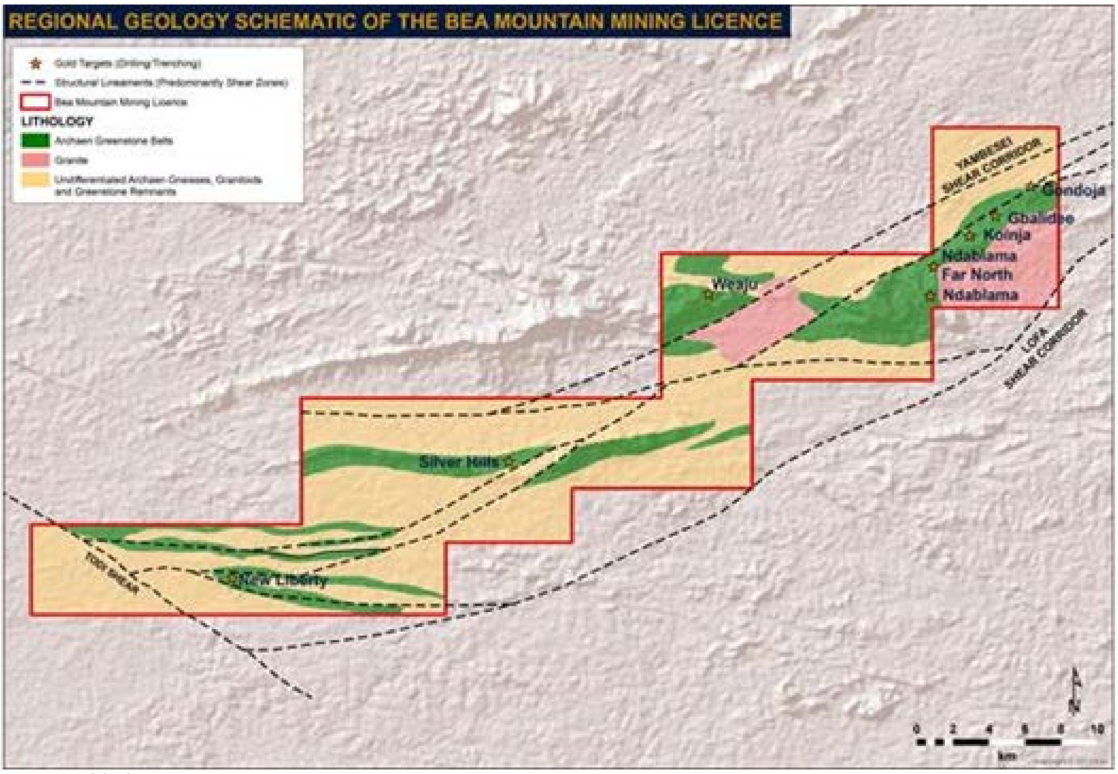

Source: *Hurley et al.*

**Figure 7-2: Age Province Map of Liberia**

## 7.2 Geology of the Bea-MDA Property

The Bea-MDA property contains a sequence of highly deformed discrete lenses of ultramafics and amphibolites, which represent relict Archaean greenstone belts, surrounded by granites and granodiorites. These rocks have been subjected to lower amphibolite grade metamorphism resulting in gneissose or schistose textures, depending on the rock competency.

The greenstone belts are elongated parallel to the regional strike, which is east-trending in the south, swinging to the north-east across a major shear in the north. Two sub-parallel arms of this greenstone unit have been mapped across the entire property; the northern arm represented by the Bea Mountain range, and the southern arm the Silver Hills. In the south of the Bea-MDA property, airborne geophysics has identified other, less clearly defined, east–west trending, units, which, in the case of New Liberty, have been confirmed by subsequent drilling.




*Aureus 2013*

**Figure 7-3: General Geology of the Bea-MDA Property Geology**

The Bea-MDA property contains several known areas of gold mineralisation, typical of Upper Archaean to Lower Proterozoic styles of metallogeny within greenstone belts. These are concentrated in major imbricate shear zones and possibly associated rotational fold hinges close to greenstone belt contacts, forming coevally with calc-alkaline granitoid intrusions. The shears and associated splays acted as structural channels for hydrothermal solutions, which deposited gold in suitable structures or chemical traps. This model is consistent with Archaean orogenic gold deposits described by Hagemann and Cassidy (2000), Richards and Tosdal (2001) Goldfarb, Groves and Gardoll (2001), Roberts et al (1998).

## 7.3 Project Geology

### 7.3.1 Stratigraphy

The Project is underlain by three main stratigraphic units (summarized in Table 7-1), which are further subdivided into minor zones of varying mineralogical assemblages. The geology is dominated by tremolite-chlorite-actinolite-talc ± magnetite rich meta-ultramafics, sometimes with phlogopite, and flanked by migmatitic gneisses.

**Table 7-1: Simplified Stratigraphic Succession**

| Main Stratigraphic Zones | Lithologies |
|---|---|
| Hanging Wall Complex (HWC) | Mafic and felsic gneisses |
| Silicified Metamorphosed Ultrabasics (SMUS) | Ultramafic schist which hosts the mineralisation. Often altered with silicification. |
| Footwall Complex (FWC) | Mafic and felsic gneisses and granites |
| **Subsidiary Stratigraphic Zones** | **Lithologies** |
| Contact Zone (GNgp) | Amphibolite gneiss with metasomatic granites. |
| Syn to late tectonic aplites, pegmatites and granitoids. | Granites varying mafic phases including tourmaline, biotite, phlogopite. |

The Hanging Wall Complex (HWC) consists of migmatite and gneisses. Amphibolite bands alternate with quartzo-feldspathic gneiss (Figure 7-4), repeating in fractals, from metre through to millimetre scales.

The Footwall Complex (FWC) rocks are similarly banded, but the bands have a wider zone of foliated leucratic gneiss (GNqf) and contain lesser but larger concentrations of hornblendic gneisses.

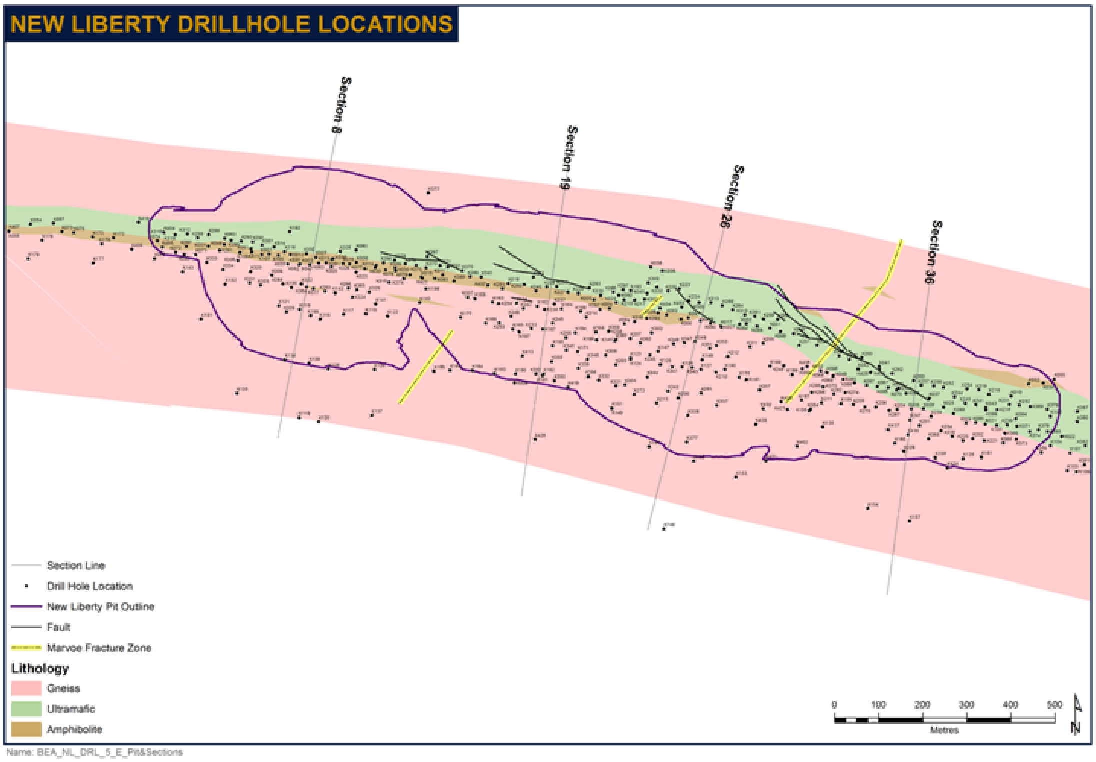
The silicified metamorphosed ultrabasic suite (SMUS) is the principal host to the gold mineralisation, and generally contains quartz, chlorite and amphibole, and a host of mafic minerals, including talc.

At the contact separating the HWC and FWC from the SMUS are transitional rocks, named here as garnet phlogopite ± actinolite gneiss (GNgp), which have a strong schistosity and coarse grain size (Figure 7-5). GNgp is also found within the ultramafic sequence.

Figure 7-6 shows a simplified plan view of the Project geology and Figure 7-7 a suite of cross-sections.


**Figure 7-4: Hanging Wall Gneiss Complex (HWC)**

**Figure 7-5: Almandine Garnet Porphyroblasts in HWC**




Source: *Aureus, 2013*

**Figure 7-6: Project Geology**

Syn-to-late tectonic aplites, pegmatites and granitoids that occur within the system are heterogeneous and show significant variations in deformation style relative to the host rocks. Greisens and pegmatitic granites intrude the ultramafics. The variable angles these granite contacts make with the units suggest that they were intruded both along the strike of the zone and into crosscutting fractures, faults and secondary shear zones. The relative ages of these intrusive bodies and their relationships to mineralisation are not known at this stage.

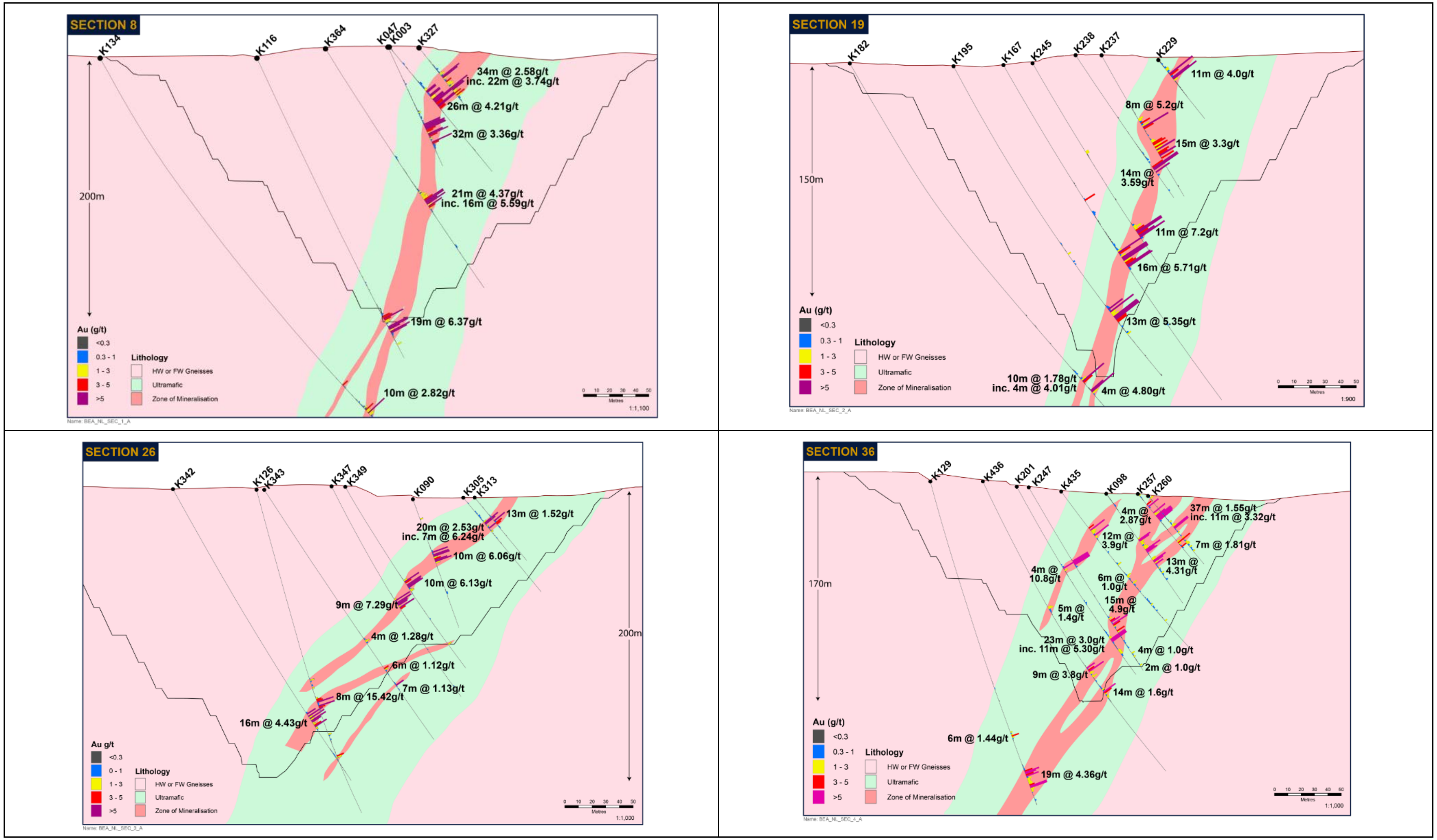



**Figure 7-7: Schematic South–North Cross-sections: New Liberty Geology**

### 7.3.2 Structure

The Project is positioned in a predominantly southerly-dipping schist belt, within a zone of high ductile shear strain oriented $287^0/72^0$, which served as the pathway for the migration of Au-bearing fluids into the host lithology. The ultramafic unit is bedded and cut by brittle faults and dolerite dykes. Parallel bands and linear basic bodies, interpreted as sills and mafic schists, have also been mapped locally to the north and south of the Project. The most prevalent fabric in the Project's ultramafic rock is a steeply dipping metamorphic banding that is well developed in sheared regions. Small scale folds (3 cm–5 cm) are common throughout the system.

Faults are difficult to detect on the surface due to the regolith and because some faults may be parallel to the regional strike, while others could have been annealed by granite veins and intrusions, again parallel to regional foliation. Thrust faults have been identified, with the hanging wall thrusting towards the north. Immediately adjacent to the gold mineralisation shearing increases in intensity until folding is no longer detectable.

## 7.4 Alteration

Within the ultramafic unit, silicification is found proximal to the mineralisation, within the immediate hanging wall and rarely in the footwall gneisses. Other alteration styles associated with the mineralisation include the presence of phlogopite as well as chlorite within the ore zone, and an associated bleaching of the rocks linked with the destruction of magnetite.

These features point to a pathway for the mineralising fluids which was active over a long period of time. The deposit shows the classic signs of sulphidation, with iron sulphides (mainly pyrrhotite) replacing the magnetite. It has a low sulphide content with sulphides forming between 0.1 and 1 % of the mineralised zones.

The four sections shown in Figure 7-8 highlight the coincident relationships between magnetite depletion, silicification, phlogopite alteration and gold mineralisation. The first Section A (top left) illustrates the relationship between gold grades, the mineralised zones and areas of magnetite depletion. This relationship is defined more clearly in Section B (top right) where the magnetite depletion zones are outlined. The coincidence of these zones and the gold mineralisation is evident. Section C (bottom left) shows the extent of the silicification and this again follows the magnetite depletion and the gold mineralisation outlines. Section D (bottom right) depicts the presence of phlogopite alteration which occurs along the upper contacts of the silicification zones.

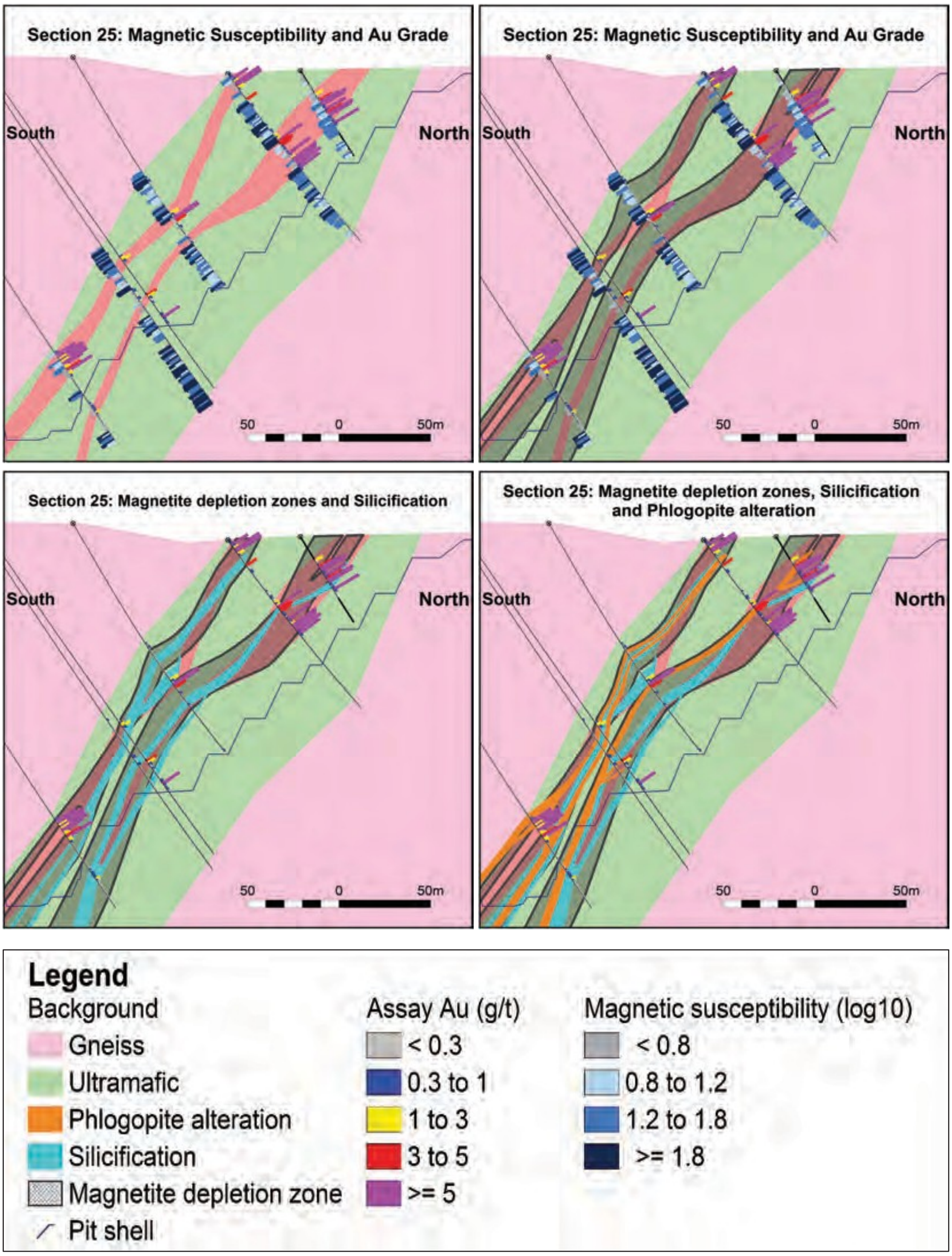



**Figure 7-8: Sections showing the relationship between gold mineralisation and magnetic susceptibility, magnetite depletion zones, silicification and phlogopite alteration**

Figure 7-9 shows the geochemical associations both in the mineralised zone and margins. Multi element analyses of cores have highlighted a clear association between gold and arsenic, sulphur, nickel and tungsten in the mineralised zones. Enhanced values of magnesium, sodium, potassium, rubidium and barium occur along the margins of the mineralisation. It is hypothesised that the gold-bearing metamorphic fluid may include a granitic component in its evolution.

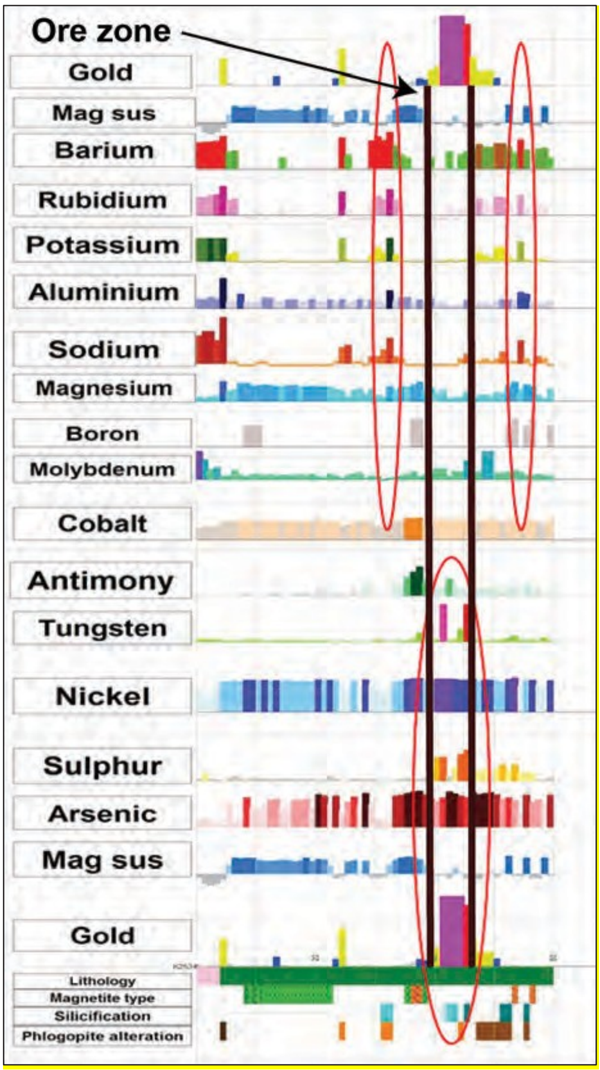



**Figure 7-9: Geochemical associations in the mineralised zone and the margins in the ultramafic host rock**

## 7.5 Mineralisation

The vast majority of the mineralisation at the Project is hosted within the altered parts of the sheared ultramafic rocks. Pyrrhotite and arsenopyrite are the main sulphides with occasional pyrite and rare chalcopyrite or pentlandite. The areas with the highest gold grades are associated with arsenopyrite. Metallurgical tests of the mineralised sections carried out by Lakefield Research Limited (Lakefield, 1999b) indicated that the gold is free in form. Gold mineralisation occurs in zones of variable thickness, with average widths of 10 m, and is nearly continuous along 2 km of strike.

Through the history of exploration at the Project, particular local concentrations of higher grade gold mineralisation have been identified, initially on the basis of apparent breaks in strike continuity at surface and subsequently through confirmation of strike discontinuity or at least variation at depth. For convenience, these zones have been named, from west to east as Larjor, Latiff (discovered in 2010 in what had been assumed to be a gap), Kinjor and Marvoe.

## 7.6 Metallogeny and Paragenesis

Gold at the Project is linked with an assemblage of sulphides and oxides in ultramafics and granite. Opaque minerals include trace to minor quantities of pyrrhotite, arsenopyrite, chalcopyrite, pentlandite, galena, sphalerite, magnetite, ilmenite and rutile. Sulphide growth may be in the form of vein fills, massive aggregates, clusters, blebs, stringers and fine or coarse disseminations in ultramafics or granite veins. There appears to be a progression from syntectonic to late-tectonic growth, with at least two phases of sulphide and oxide growth. The non-opaque minerals are amphibole, chlorite, mica, serpentine, talc and quartz. Pyrrhotite, arsenopyrite, coarse grained pyrite, chalcopyrite, sphalerite and minor pentlandite are the principal sulphides; the chief observation being (but not always) an increase in grain size and abundance, both absolute and relative, in host rocks near granite veins.

In Figure 7-10, pyrrhotite, arsenopyrite and pyrite are shown in cut and uncut ultramafic core, with the bulk of the sulphides aligned to the dominant cleavage.

**Figure 7-10: Mineralisation in Core**

## 7.7 Summary of Field Character of the Mineralisation

The gold mineralisation at the Project is associated with sulphides, hosted in metamorphosed ultrabasic rocks intruded by tourmaline-bearing granites that are closely associated with albitite dykes. The ultramafics consist of amphibole (tremolite, actinolite), chlorite, phlogopite, talc, some carbonate and the sequence is moderately to highly silicified.

The widespread silicification is accompanied by ubiquitous magnetite precipitation. The sulphide association is pyrrhotite, pyrite (the two alternating in dominance), arsenopyrite and minor-to-trace chalcopyrite, niccolite and gersdoffite. Magnetite and minor haematite are the main oxides.

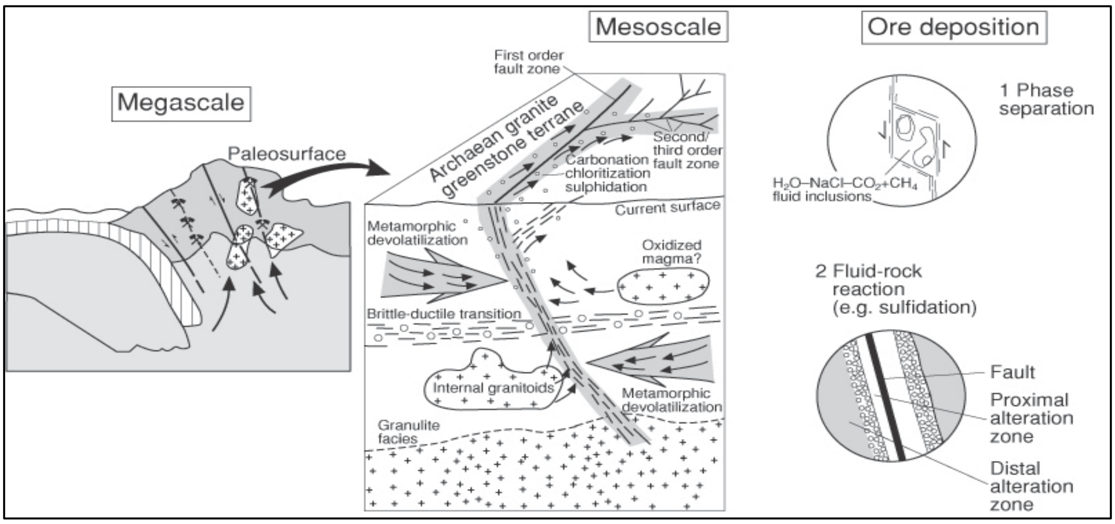
# 8 DEPOSIT TYPES

The mineralisation being targeted by Aureus comprises typical Upper Archaean to Lower Proterozoic greenstone belt-hosted lode gold mineralisation. These deposits are often referred to as orogenic and are characterised by the presence of a combination of gold-quartz veins and disseminated mineralisation.

Archaean orogenic deposits are typically hosted in greenstone belts comprising meta-volcano sedimentary supracrustal assemblages, together with coeval calc-alkaline granitoid intrusions. The gold mineralisation is typically hosted in moderate to steeply dipping quartz-dominated shear zones with associated extensional vein systems and is considered to be coeval with the syntectonic stages of the orogeny and related to periods of crustal shortening at 8 km -15 km depth. Structures are typically formed at, or close to, contacts between rock types of contrasting competencies, and mineralisation is often localized at bends or splay intersections in the shear system.

Mineralisation in Archaean orogenic deposits is typically associated with characteristic alteration styles (quartz-carbonate-sericite-biotite-sulphides) and often enriched in ‘lodes’ that plunge steeply. Gold deposits may occur in a variety of host rocks, which include granite, meta-volcanic rock (greenstones) and include mafic and ultramafic rock units and associated volcaniclastic, banded iron-formations and siliciclastic sediments, as observed within the Bea-MDA licence area. The schematic diagram (Figure 8-1) depicts a typical orogenic lode system with analogous geological settings for the deposit styles found on the Property.

The primary targets of Aureus’ mineral exploration programme in Liberia are shear zone-hosted gold systems, whether associated with quartz, granite veins, breccia zones or granitic bodies. A structural control to mineralisation is evident with areas of multiple structures intersecting. Gold mineralisation in these deposits is thought to have been emplaced by Au-bearing fluids flowing into dilatational zones formed by faults or fold hinges in high strain zones.




Modified from: *Hageman and Cassidy 2001*

**Figure 8-1: Schematic of Orogenic Gold Systems**

Gold within the system was introduced as gold sulphide complexes in hydrothermal solutions, which may in part have been sourced from underlying granitic plutons. The solutions reacted when they came into contact with the magnetite within the ultramafic rocks, causing the deposition of native gold and sulphide minerals. Prominent examples of such deposits, are: Golden Mile at Kalgoorlie, Australia, Kerr-Addison Mine in Ontario, Canada and Homestake Mine in the United States Groves et al. (2003), Robb (2005).

# 9 EXPLORATION

## 9.1 Introduction

Exploration by Aureus at the Bea-MDA property has followed a systematic process of reconnaissance work, grab-sampling followed by soil geochemistry, mapping, trench sampling and eventually drilling. Airborne and ground geophysics have also been conducted in situations where appropriate.

## 9.2 Methodology

### 9.2.1 Coordinates, Datum, Grid Control and Topographic Surveys

Geological and geographical information was first set out on a local grid using a baseline at 285° magnetic, which parallels the strike of the mineralisation. Early mapping of outcrop, trenches and streams was by tape and compass survey. This grid contained several errors, compounded by the magnetic effect of the ultramafic body. In 2009 survey control was re-referenced to UTM Zone 29N coordinates (map datum WGS84), and locations were obtained using GPS. In addition to re-surveying drillholes, a topographic map was created which included streams, roads and outcrop.

Surveys since 2010 for both drillhole collar pickup and topography were undertaken with reference to three control points, with two Trimble R3 receivers used for surveying in 2010. From October 2011, a Leica DGPS survey system was used to resurvey all the drillholes, while a new topographic survey is progressively being updated, with reference to the same three control points.

### 9.2.2 Geological Mapping

Aureus geologists have conducted several programmes of outcrop mapping. Outcrop is limited mostly to artisanal pits and trenches; therefore maps are progressively updated as more data from trenches and drilling becomes available.

### 9.2.3 Regional Stream and Outcrop Sampling

In the period 2005 and 2006, Mano acquired multi-element, stream sediment geochemical data from Western Mining Corporation (WMC) and undertook extensive regional outcrop and heavy mineral sampling programmes in Gola Konneh, Tewo and other districts.

Reconnaissance sediment surveys of small streams for gold and heavy mineral, in and around the Bea Mountain and Silver Hills ridges, have indicated the presence of several previously unknown gold occurrences in water courses flowing off the Bea Mountain ridge, and which require future investigation.

### 9.2.4 Soil Geochemistry

Soil sampling is undertaken on a set grid, with line spacing determined by the objectives of the individual programme. Samples are positioned using handheld GPS, with 1 kg of soil taken from a depth of 0.5 m.

### 9.2.5 Trenching

Trenches are staked out by geologists at an alignment that intersects the strikes of structures, and are then excavated to a depth of 1 m–4 m, depending on bedrock intersection depth. The trenches are surveyed and logged, followed by continuous channel sampling along each metre of the trench.

### 9.2.6 Geophysics

In May 2006, a high resolution helicopter-borne, combined magnetic gradient and gamma-ray spectrometer survey was conducted over the south-west and north-east sections of the licence area by New Resolution Geophysics (NRG). This was then complimented by a further survey, carried out by Geotech Airborne Limited in 2012 which covered the remainder of the Bea-MDA property, and the adjacent 'Archean' licence, which is also owned by Aureus. Sufficient overlap between the old and new survey and matching line spacing enable the surveys to be merged together. The survey parameters of both are summarized in Table 9-1. The datasets were merged by Geotech Airborne analysts and data quality control was undertaken by an independent consultant geophysicist. The radiometric spectrometry enables the demarcation of different lithology types, and the magnetics show both structure magnetic bodies, such as the ultramafic host rock at the New Liberty deposit (Figure 9-1).

**Table 9-1: Comparisons of 2006 and 2012 Airborne Geophysical Surveys**

| Company | Year | Survey Method | Data Acquired | Flight Elevation | Line Spacing | Positioning System | Line Flown (km) |
|---|---|---|---|---|---|---|---|
| New Resolution Geophysics | 2006 | Helicopter | Magnetics, spectrometry DTM | 30 m | 100 m with 1000 m tie lines | DGPS and radar altimeter | 2,200 |
| Geotech Airborne Limited | 2012 | Fixed wing | Magnetics, spectrometry DTM | 100 m | 100 m with 1000 m tie lines | GPS with WASS enabled and radar altimeter | 9,631 |

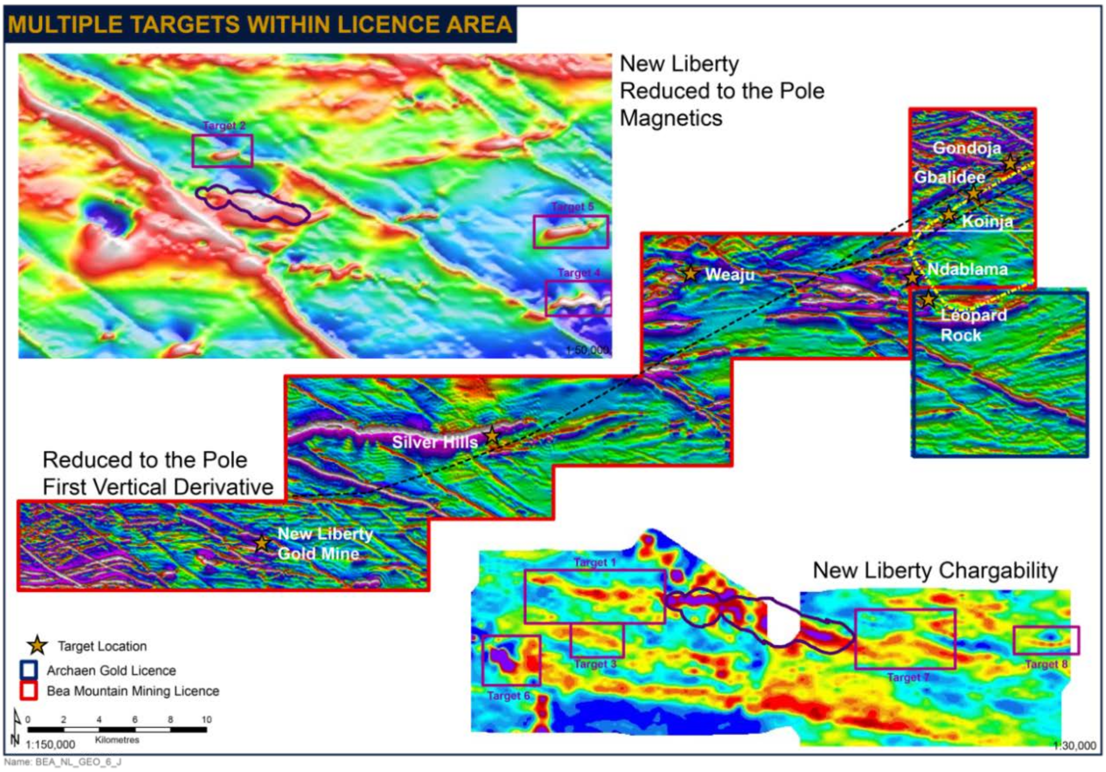
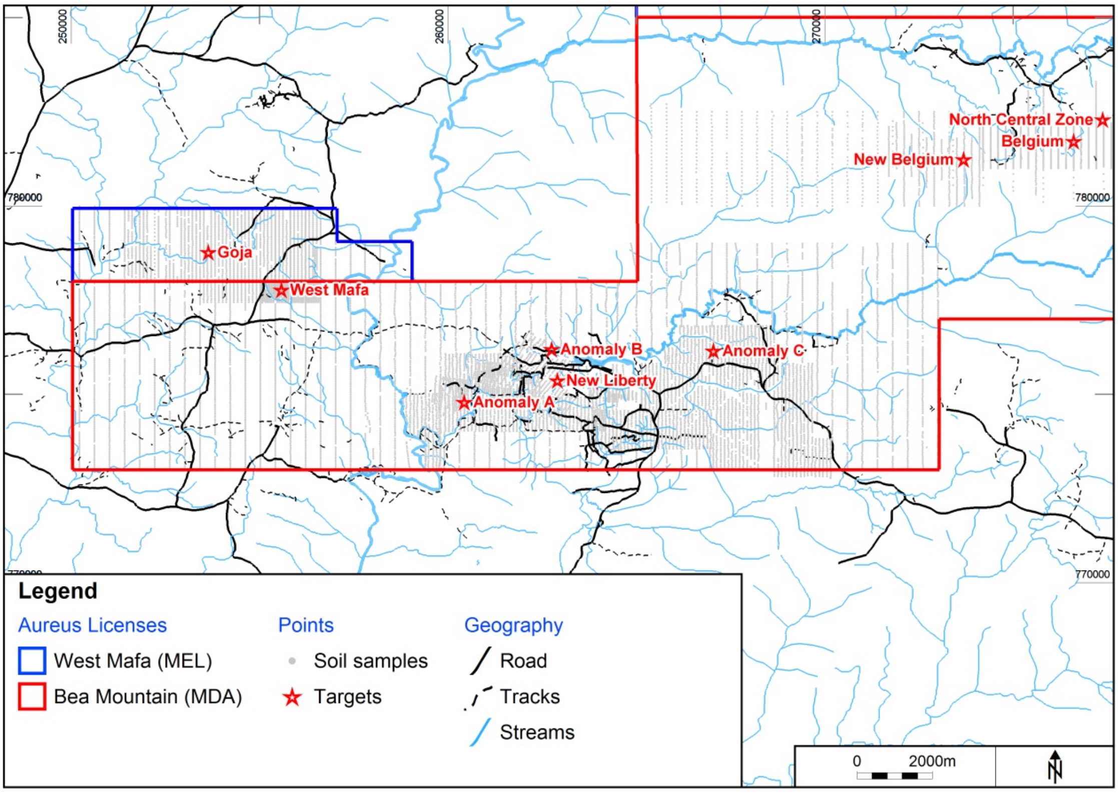



**Figure 9-1: New Liberty Geophysics Interpretation**

## 9.3 Regional Exploration

### 9.3.1 Soil Geochemistry

Geochemical soil sampling in 1999 on a 100 m by 20 m grid over 1 km each side of the known mineralization detected a strong anomaly over 200 m to the west and east. Further along-strike soil sampling in 2011 and 2012 extended the areas surveyed to the east and west, in conjunction with geophysics and exploration. During 2013 and 2014, further soil sampling occurred to the both the north-east of New Liberty at the Belgium targets (Silver Hills) and to the north-west at the West Mafa target, with the focus of locating near mine anomalies for further follow up exploration (Figure 9-2).




Source: *Aureus, 2015*

**Figure 9-2: Soil Sampling Coverage over the New Liberty Area**

### 9.3.2 Trenching

Following an encouraging channel sample programme of artisanal workings (Figure 9-3), which yielded intersections including 19.95 m at 4.06 g/t Au in the west and 13.1 m at 4.56 g/t Au in the centre of the system, trenches T1–T12 were excavated in 1997, each 3 m deep trench aligned approximately perpendicular to the east-west strike of the mineralisation, This covered an along-strike extent of 1,800 m (see trench example Figure 9-3). During 1998, trenches T13–T24 were completed at intervals of 100 m along the geological strike and 20 m–80 m long to depths ranging from 2.0 m to 4.0 m into saprolitic material (Figure 9-4). Later trenching (T27 and T28) was used for outcrop demarcation to assist in the positioning of borehole collars in poorly exposed terrain beyond the ultramafics and mineralisation.

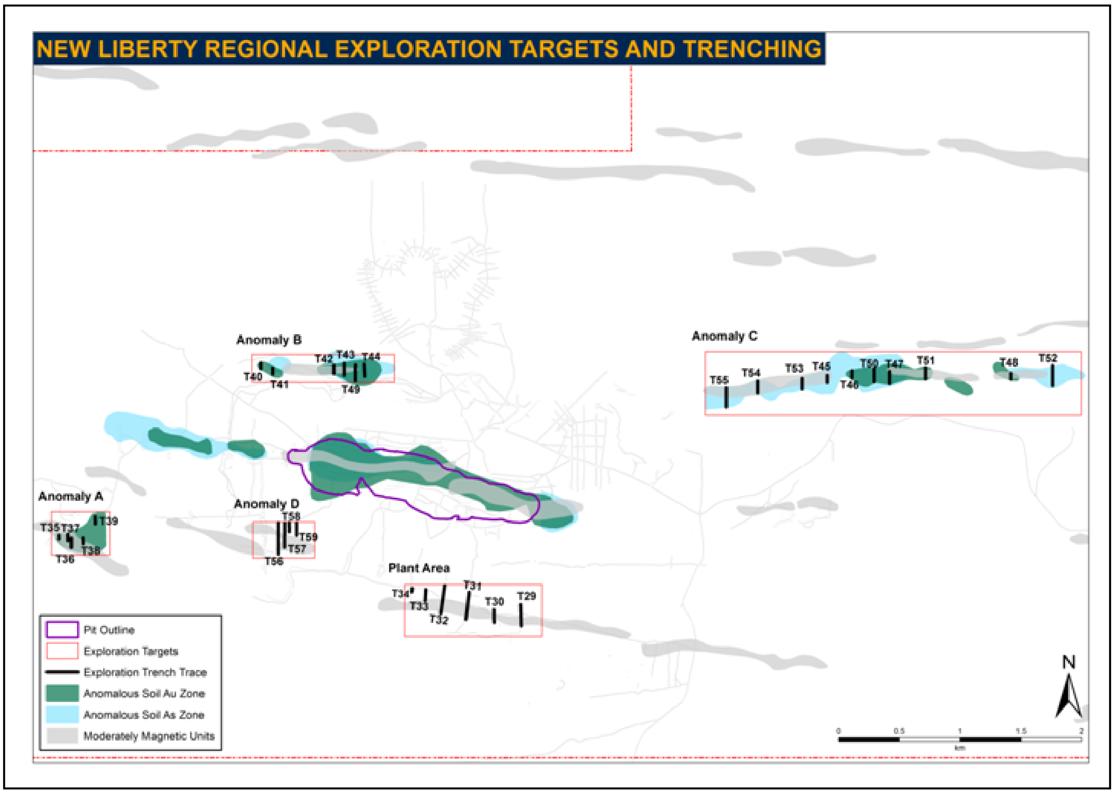
Further to this, during the 2012/2013 field season, a total of 29 trenches were dug across four key sites (totalling 3,241 metres, Figure 9-5). The trenches targeted anomalies represented by elevated soil gold and arsenic values coincident with geophysics anomalies.

All trenches were geologically mapped and channel sampled (metre-length samples). All samples were despatched to the SGS Laboratory in Monrovia for analysis for gold, and results are assessed as they are received.

**Figure 9-3: Artisanal Workings**

Source: *Aureus, 2012*

**Figure 9-4: Exploration Trench**

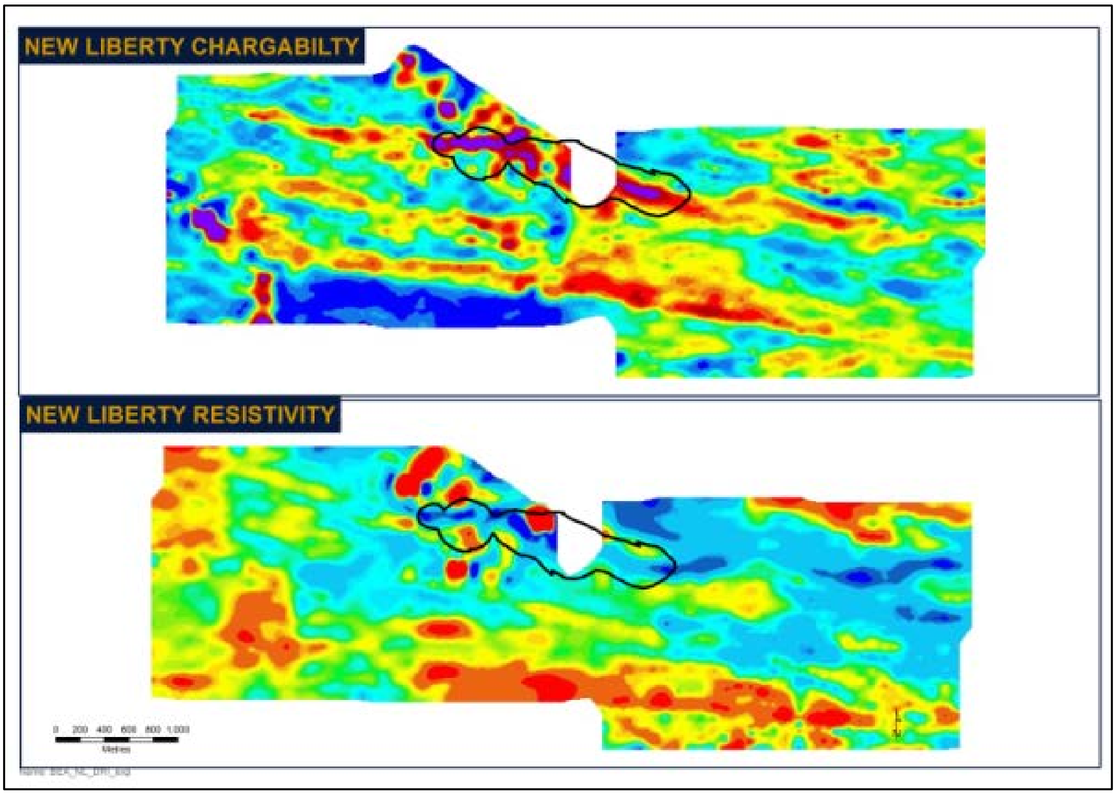



Source: *Aureus, 2013*

**Figure 9-5: Trench Coverage Around the New Liberty Project**

### 9.3.3 Geophysics

Following from the airborne survey, ground magnetic, induced polarization (IP) dipole-dipole lines and gradient array surveys were undertaken by international geophysics survey company, Fugro, in 2011 and 2012. Initially, the areas of known mineralisation were surveyed to gain an understanding of the signature of mineralisation, with areas outside then used to extrapolate to other features. Further investigation is based on the airborne magnetic data, and along-strike from the mineralisation. Fifty-two line kilometres of survey were completed for the ground magnetics and a further 15 km$^2$ for the IP grid and dipole-dipole. The IP detected a low resistivity corridor thought to represent a continuation of the mineralisation within the ultramafic unit (Figure 9-6). Regionally, a further 1.8 km$^2$ has been completed to the south of the Ndablama target.

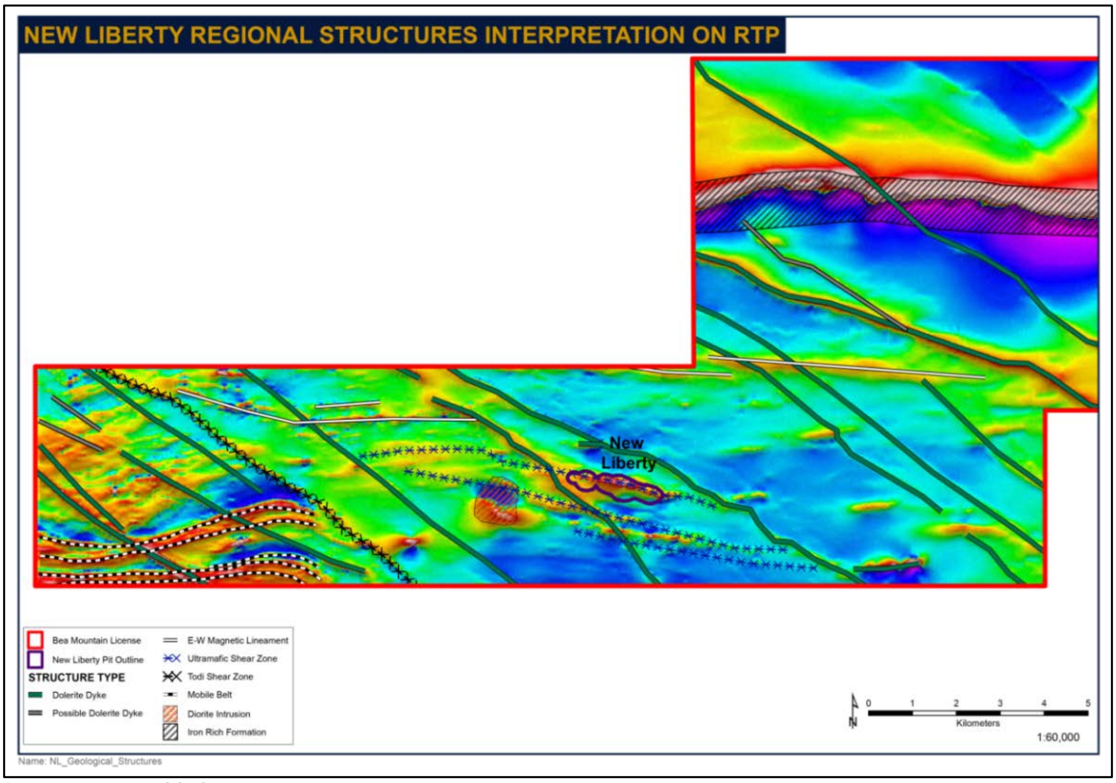



Source: *Aureus, 2012*

**Figure 9-6: IP Corridor at New Liberty**

## 9.4 Further Targets at the Project

Analysis of both the IP and the re-analysed airborne magnetic data has identified several targets around the Project worthy of further investigation (Figure 9-7). These are undergoing investigation with soil sampling, outcrop mapping and surveys to delineate potential targets for drilling.




Source: *Aureus, 2012*

**Figure 9-7: Aerial Magnetics Targets**

## 9.5 Other Targets in the Bea-MDA Property

### 9.5.1 Introduction

There are various other targets on the Bea-MDA property which are currently subject to exploration at various stages.

The information has been included here in the context of disclosing other activities on the Bea-MDA property, but these are unrelated to the purpose of this report.

### 9.5.2 Silver Hills

Silver Hills is situated approximately 14 km north-east of the Project. Soil sampling results have highlighted a zone 80 m long and 30 m wide, which was followed up by the excavation of three trenches (Table 9-2).

**Table 9-2: Silver Hills Trench Results**

| Trench ID | From (m) | To (m) | Length (m) | Au (g/t) |
|---|---|---|---|---|
| ST1 | 0 | 15 | 15 | 0.51 |
| Including | 2 | 8 | 6 | 1.08 |
| ST2 | **Unmineralised** | | | |
| ST3 | 1 | 29 | 28 | 0.36 |
| Including | 2 | 7 | 5 | 0.76 |
| | 48 | 66 | 18 | 0.18 |

### 9.5.3 Regional Targeting

As part of an ongoing exploration programme a geochemical and structural study of known areas of mineralisation is underway. This data will be merged with regional airborne magnetics and radiometrics datasets to identify structures and settings within the Bea-MDA property.

# 10 DRILLING

## 10.1 Introduction

Diamond drilling at the Project was conducted periodically between 1999 and 2012 (Table 10-1). The total meterage drilled in the exploration of the Project is now 65,187 m which has been completed in seven campaigns.

**Table 10-1: Summary of Drill Campaigns**

| Campaign | Year | Hole Numbers | Number of Holes | Metres |
|---|---|---|---|---|
| 1 | 1999 - 2000 | 1 – 19 | 19 | 1,947 |
| 2 | 2000 | 20 – 26 | 7 | 791 |
| 3 | 2005 | 27 – 61 | 35 | 3,024 |
| 4 | 2006 | 62 – 114 | 53 | 5,066 |
| 5 | 2008 | 115 – 130 | 16 | 4,485 |
| 6 | 2009 - 2010 | 131 – 175 | 45 | 12,423 |
| 7 | 2011 - 2012 | 176 – 438 | 252 | 37,451 |
| **Total** | | | **438** | **65,187** |

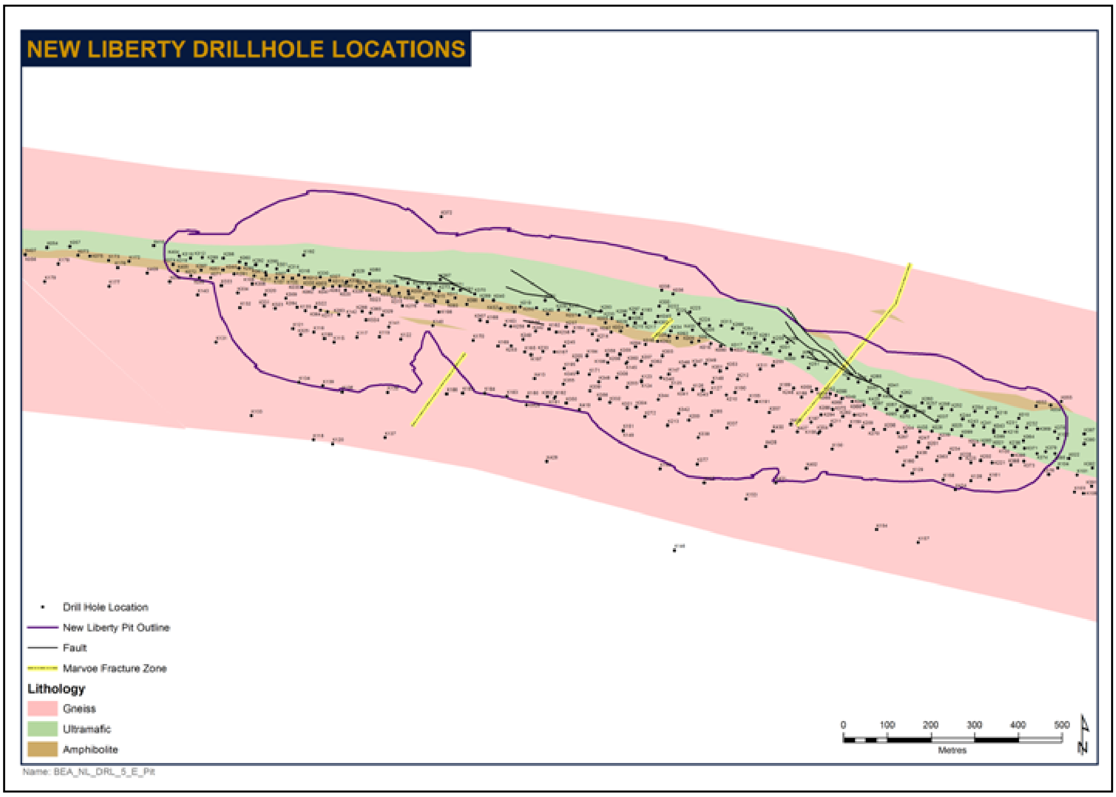
The drilling has been carried out in part by contractors and in part by Aureus. Campaigns 1–5 were done by UK-based firm Drillsure (later Envirodrill); Campaign 6 drilling was in part by Australian Exploration and Drilling Company (AEDCo), with the last eight holes being completed with in-house rigs and crews, using Aureus-owned Golden Bear and Hydrocore rigs. Campaign 7 was completed by Boart Longyear.

Drilling was conducted on a grid, with holes generally drilled on a 015° azimuth (magnetic) and inclined at between minus 45° and 70° to intersect the south-dipping zones. During drilling campaigns 6 and 7 a grid pattern was used. At times exceptions to the default bearing were introduced because of inaccessibility due to swampy conditions or because the distance to the target depth exceeded the capability of the rigs. This occurred in the case of six boreholes, K10, K32, K34, K55 in the Marvoe zone and K36 and K38 in the Kinjor zone. In each case the back bearing of 195° was used. Figure 10-1 is a plan view illustration of drillhole locations in the four mineralised zones.

The core sizes drilled varied over time as well as within holes, as shown in Table 10-2. The quarter core from the first 27 diamond drillholes and half core for the remaining holes are stored on site. Figure 10-2 shows a view of the core storage facilities at the time of the Campaign 6 drilling.

**Table 10-2: Drill Metres by Campaign**

| Campaign | HW/T6116> 90 mm | PQ 85 mm | HQ 63 mm | NQ 47 mm | NTW 45 mm | LTK 36 mm | BQ/T48 36.4 mm | AQ/DT48 27 mm |
|---|---|---|---|---|---|---|---|---|
| 1 | 143 | | 1,751 | | | | 52 | |
| 2 | 89 | | 615 | 86 | | | | |
| 3 | | | | | | 271 | 2,120 | 632 |
| 4 | | | | | 909 | 716 | 3,058 | 382 |
| 5 | 19 | | 309 | 1,363 | 248 | | 2,546 | |
| 6 | 251 | | 6,238 | 6,074 | | | | |
| 7 | | 2,329 | 17,463 | 17,659 | | | | |
| **Total** | **502** | **2,329** | **26,376** | **25,182** | **1,157** | **987** | **7,776** | **1,014** |




Source: *Aureus, 2013*

**Figure 10-1: Location of Zones and Drilling**

**Figure 10-2: Core Shed**

## 10.2 Drill Programme Campaigns

Campaign 1 comprised 19 holes drilled at HQ (68 mm), with the exception of hole K16, which was started at HQ and reduced to NQ (48 mm). The holes were drilled on 50 m centres and intersected mineralisation at depths ranging from 20 m to 30 m below surface along the length of the two mineralised zones. One hole, K10, was drilled some 500 m to the east of the Kinjor excavation to intersect mineralisation identified in trench T-11, in the area termed the Marvoe Creek Zone.

In early 2000, a second campaign of drilling was undertaken, with the aim of testing the mineralisation at greater depth under the Kinjor and Larjor artisanal workings, and to investigate the mineralisation in the Marvoe Creek Zone. K20 and K23 were drilled in the central part of the Larjor ore body and intersected mineralisation at some 50 m and 100 m below surface respectively. K21 and K22 were drilled on the Marvoe Creek Zone near hole K10.

The third diamond core drilling campaign, designed to close along-strike inter-hole distances to a maximum of 25 m started in January 2005. At the same time, selected holes were drilled at steeper angles in order to intersect the mineralisation at depth, as the deepest intersection at the time was 80 m below surface. The programme also aimed at further evaluating the eastern extremity of the Marvoe Creek Zone, which is indicated by aeromagnetic data to continue to the south-east.

A hiatus in drilling followed due to a period of unrest in the country.

Campaign 5 was completed between January and May 2008 and consisted of 16 NQ core drillholes, inclined at between -60° and -70°, drilled under the three known zones. Fourteen (14) of these holes tested the gold mineralisation at 300 m below surface elevation while two (both in Larjor) investigated and demonstrated that the Larjor zone mineralisation persists to -600 m level.

In 2009 (Campaign 6) a 10,730 m definition and extension drilling programme was initiated to satisfy two primary objectives:

- To better understand the local geometry of the mineralisation and confirm or otherwise the continuities implied in the interpretations then held.
- To assess the extent and continuity of the mineralisation beyond (down-dip of) the limits of the higher density drilled areas.

The drilling programme was flexible and dynamic, allowing changes to be implemented during the programme based on feedback from site, assay results received and to account for practical issues such as positioning of drill pads (Figure 10-3). One outcome of this was the discovery of the Latiff Zone from wildcat borehole K144 in the gap between the Larjor and Kinjor zones, which led to the revised drilling across the gap.

Four additional holes were drilled in the Latiff Zone through to August 2010 with all holes confirming continuity at depth of the mineralisation.

Campaign 7 was completed between 2011 and 2012. During the campaign, 252 diamond drill holes were drilled for a total of 37,451 m. The drilling was undertaken by Boart Longyear, with aims to increase definition within the orebody at all zones as well as to test for extensions along strike. During the drilling, Aureus used the results from logging and assaying to update the mineralisation model in order to optimise the drill programme. PQ drilling was used in the oxide, followed by HQ and then reducing to NQ.

**Figure 10-3: Diamond Core Drill Rig**

**Figure 10-4: Drill Core Showing Recovery**

## 10.3 Collar Coordinates

In 2009, a review of existing collar survey coordinates identified a number of uncertainties, and a full re-survey of collars was commissioned. The results of the subsequent August 2010 (DGPS) survey of all drill collars (described in Section 9) have not been directly verified by SRK. However, accumulated information regarding instrument quality and field procedures has indicated that the re-surveyed drill collar coordinate data can be accepted with confidence for the purposes of resource estimation.

Additional resurveying and validation of accessible pre-2011 collars were conducted in 2011 and all additional collars associated with the 2011 campaign were surveyed with the Leica DGPS survey procedures described in section 9.2.

## 10.4 Downhole Surveys

Downhole surveying practices varied through the different drilling campaigns. Some 96 of the 375 holes drilled have not been surveyed.

During the first and second drill campaign (1999/2000) the majority of the 26 holes were surveyed (approximately every 50 m), the results of which demonstrate minor downhole azimuth and dip deviations (less than 5° deviation over 100 m, and SRK understands that it was this observation of low deviation that influenced decisions relating to downhole surveying during subsequent campaigns.

Most of the holes from the 2005/2006 campaign, in which the maximum hole depth was 109 m, do not have downhole survey records. For the 2008 programme, multiple downhole surveys were conducted, but intervals between readings were relatively wide, typically between 50 m and 100 m. All holes drilled during the 2009/2010 and 2011/2012 campaigns were surveyed at short intervals (10 m and 5 m respectively) and constitute the best records of drillhole deviations for the Project. During the 2011/2012 campaign, initially 5 m intervals were used (up to and including K331 and K336), with the remainder at 10 m interval

Average recorded dip deviation over the full length of each hole is around 10°, but some deeper holes (more than 400 m) deviate more than 15°. Average azimuth deviation is around 5°, but some deeper holes deviate by more than 10°.

### 10.4.1 Acoustic Televiewer (ATV) Probe

In order to obtain additional high quality geotechnical baseline information from existing inclined diamond boreholes, Lim Logging were commissioned to undertake ATV surveys. A total of 19 holes were surveyed, between 2 and 15 of March 2013, for a combined total of 1,783 metres.

The data gathered by the ATV probe was processed on-site, generating an orientated acoustic image of the borehole wall (Figure 10-5) and provided to Australian Mining Consultants (AMC) for interpretation. The spatial orientation of each structure was determined by the amplitude of the sinusoidal curve in relation to the inclination of the borehole.

Each structure was assigned to a category, according to AMC's interpretation of the structure's origin: open fractures, closed fractures, s2 fabric, foliation and veins. This information was added to previous structural event data, generated from alpha/beta measurements of orientated core.

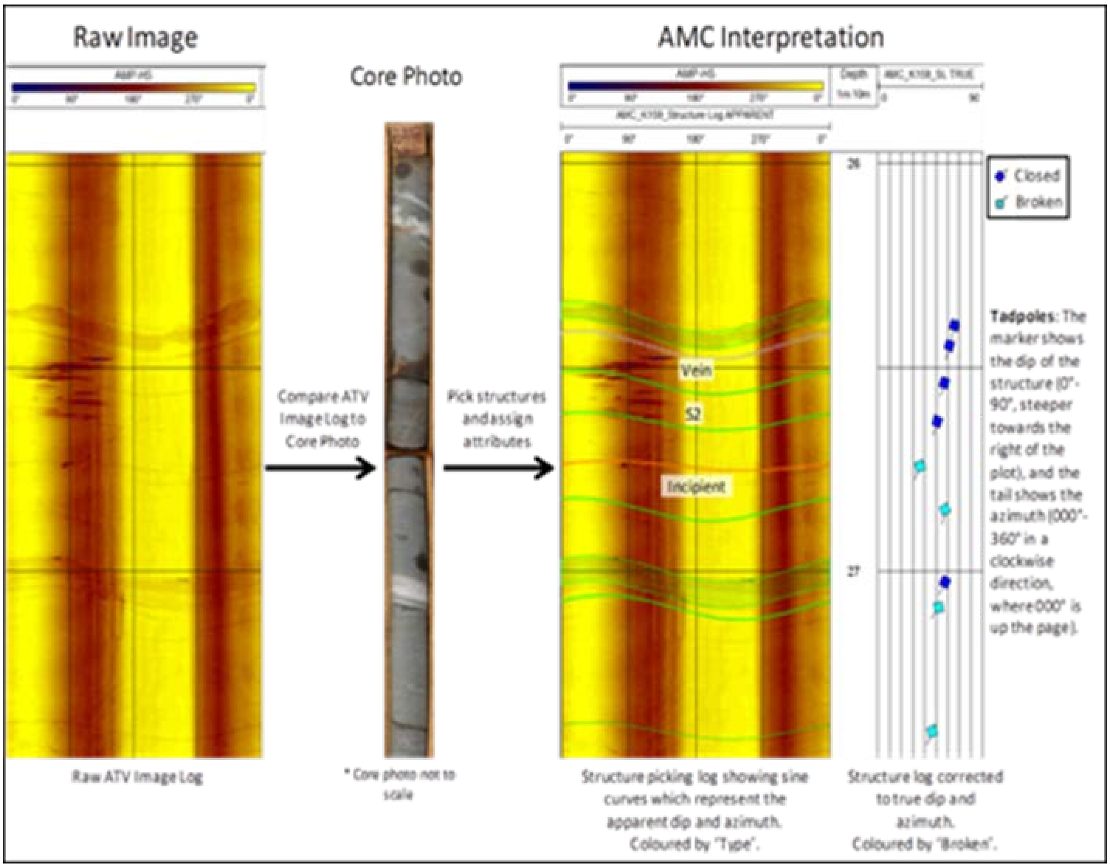



**Figure 10-5: Accoustic Image and Interpretation of ATV Survey**

Table 10-3 details holes logged during the ATV Probe Survey:

**Table 10-3: Holes Logged Using the ATV Probe**

| Hole ID | From (m) | To (m) | Interval (m) | Easting | Northing |
|---|---|---|---|---|---|
| K159 | 19 | 41 | 22 | 263781 | 775088 |
| K196 | 5 | 161 | 156 | 263227 | 775233 |
| K206 | 20 | 115 | 95 | 263860 | 775081 |
| K212 | 10 | 109 | 99 | 263524 | 775195 |
| K226 | 13 | 64 | 51 | 263185 | 775339 |
| K238 | 18 | 106 | 88 | 263108 | 775303 |
| K284 | 20 | 143 | 123 | 262514 | 775360 |
| K314 | 30 | 76 | 46 | 262495 | 775444 |
| K340 | 10 | 101 | 91 | 263348 | 775198 |
| K349 | 12 | 145 | 133 | 263450 | 775230 |
| K365 | 15 | 165 | 150 | 262683 | 775347 |
| K371 | 26 | 158 | 132 | 264181 | 775037 |
| K493 | 6 | 150 | 144 | 263707 | 775258 |
| K494 | 17 | 110 | 93 | 262806 | 775419 |
| K495 | 15 | 110 | 95 | 262806 | 775319 |
| MF001 | 5 | 98 | 93 | 263894 | 775339 |
| MF002 | 10 | 31 | 21 | 264025 | 775743 |
| MF004 | 5 | 115 | 110 | 262892 | 775133 |
| HYD002 | 5 | 46 | 41 | 263052 | 775304 |

## 10.5 Core Recovery

Drill core recovery was not recorded during the 1999/2000 drilling campaign but records from subsequent campaigns reveal very high recoveries, with most intervals returning values well above 90%. These recovery values are consistent with site observations of stored core as well as core photographs.

Figure 10-4 shows good core recovery in spite of the tendency for mineralised rock competencies to be lower than in adjacent unmineralised intervals.

## 10.6 Sterilisation Drilling

Some 6,810 m of sterilisation drilling has now been completed within the Project area. The 2013 drilling phase consisted of 12 diamond drill holes beneath the new plant site, the revised waste dump footprint and the new tailings dam. The details are shown in the table below, which also lists the details of the previous sterilisation drilling (Table 10-4).

**Table 10-4: New Liberty Sterilisation Drilling**

| Area | 2013 Phase | | Total | |
|---|---|---|---|---|
| | Number of Holes | Length (m) | Number of Holes | Length (m) |
| Waste dump | 2 | 320 | 13 | 1,935 |
| Tailings dam | - | - | 7 | 1,060 |
| New Tailings Dam site | 4 | 601 | 4 | 601 |
| Plant area | - | - | 6 | 659 |
| New Plant Area | 6 | 963 | 6 | 963 |
| Creek diversion | - | - | 10 | 1,577 |
| **Total** | **12** | **1,884** | **46** | **6,810** |

## 10.7 Grade Control Drilling

Reverse Circulation (RC) Grade Control drilling was undertaken during the period of July to December 2014 by Ore Search Drilling using a tracked EDM 2000 with Auxiliary Booster (Figure 10-6). The drilling was undertaken in two phases (see Table 10-5), the first of which focused on bringing the drill hole spacing in the Larjor End of Year 1 (EOY1) pit profile down to some 12m by 12m. The second phase, was undertaken in the Kinjor area of the main pit (Figure 10-6), In order to provide both 12m by 12m infill information and also to address any gaps in the resource model that may have arisen due to access issues with Diamond Rigs in the past, due to the presence of the Marvoe Creek running through this area of the pit.

**Table 10-5: 2014 Grade Control Drilling**

| Pit area | Number of. Holes | Length (m) | Start Date | End Date |
|---|---|---|---|---|
| Larjor EOY1 | 80 | 3,646 | 28th July | 8th Oct |
| Kinjor EOY1 | 56 | 2,595 | 9th Oct | 20th Dec |
| **Total EOY1** | **136** | **6,241** | | |

Figure 10-6: Ore Search EDM 2000 Drill Rig

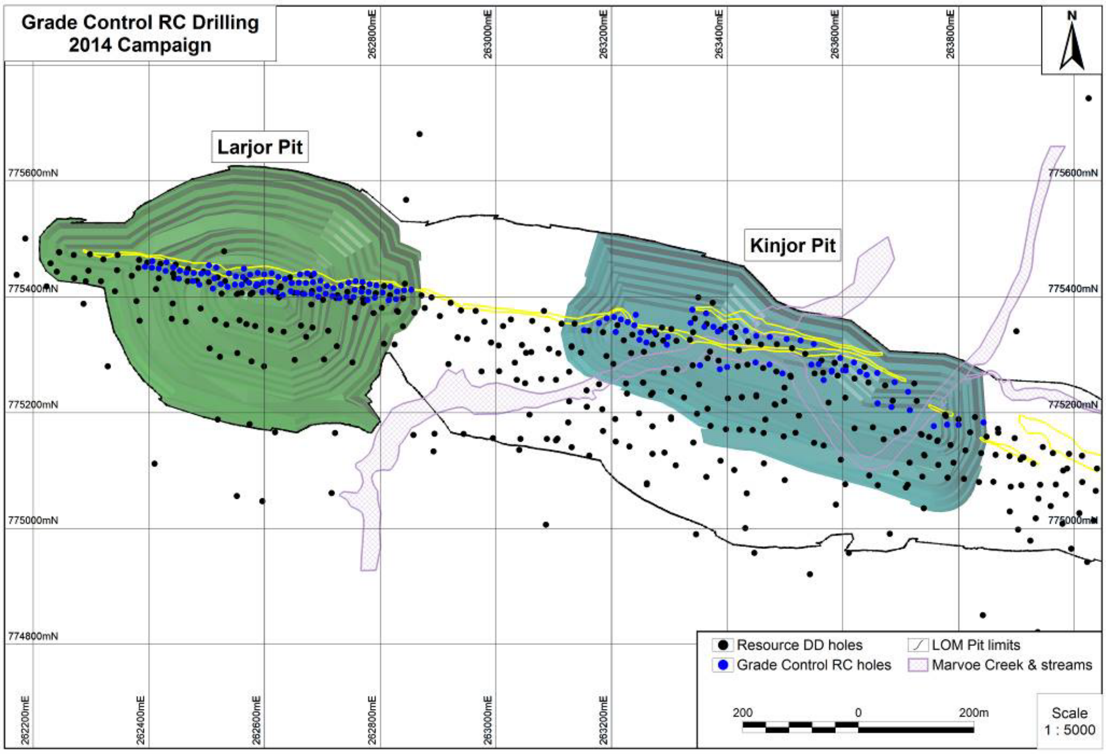



Figure 10-7: Grade Control Drilling Campaign

## 10.8 Drilling Near the Project

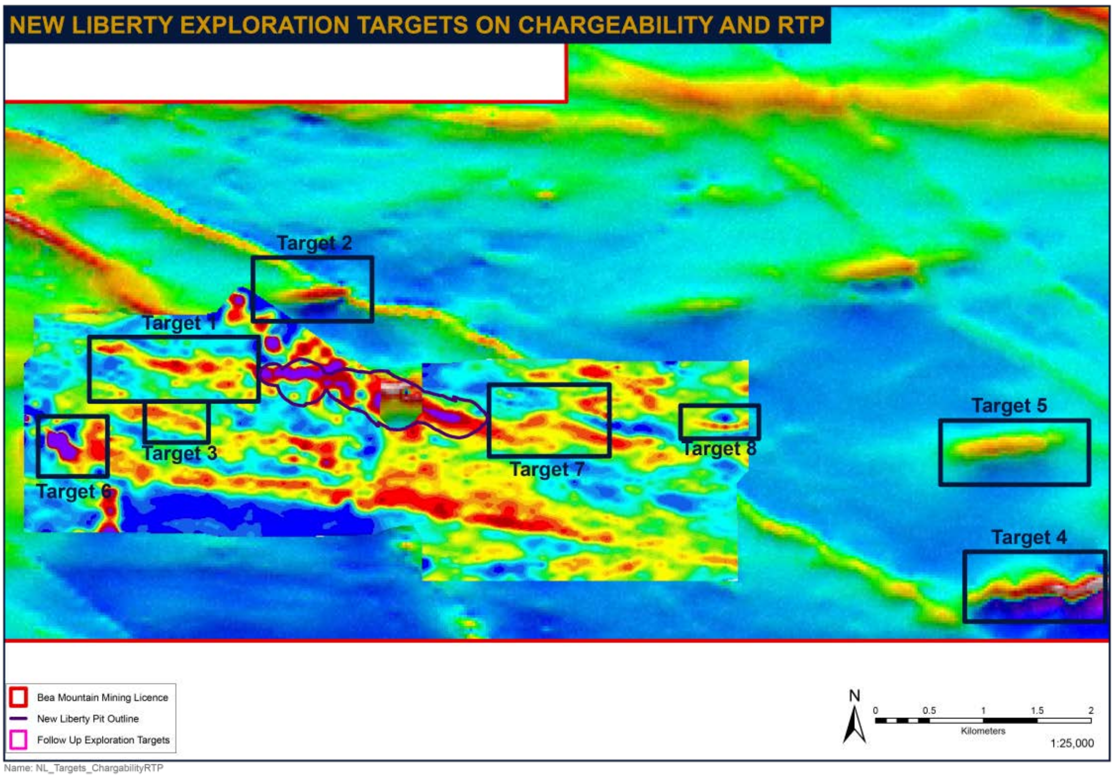



**Figure 10-8: Drill Targets Near to the Project**

Near to the Project, a further 12,153 m of drilling has been conducted on eight targets (Figure 10-8). This has identified continuations of the ultramafic host rock and parallel bands of the ultramafic units have been found to the north.

# 11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

## 11.1 Introduction

Sampling is carried out by project geologists in a manner consistent with mineral exploration procedures adhered to in other West African mineral exploration programmes. In total, 6,648 soil samples, 525 trench samples and 50,337 drill core samples have now been collected and submitted for gold assay from the Project.

## 11.2 Soils and Trenches

Soil samples have been collected from 0.5 m below the surface, in areas away from drainage channels, then coned and quartered to 1.5 kg–2.5 kg weights, and bagged for analysis.

In the trenches, one metre-long samples were systematically collected in saprolite material from 10 cm square channels cut into cleaned trench walls near the floor of trenches and across the strike of mapped structures. Some trenches (and channels) were excavated in separate segments to traverse around large boulders, trees and unstable artisanal workings, to give continuity across the zone.

All work has been carried out by Project crews and supervised by Aureus geologists.

## 11.3 Diamond Drillhole Samples

Diamond-drilling activity at the Project is also supervised by Aureus geologists. Core and core blocks are placed in core boxes by the driller. Upon reception in the core shed on site, core is cleaned or washed (if required) and core blocks are checked by Aureus staff. The core is then photographed, wet and dry, in a frame to ensure a constant angle to and distance from the photographer. Magnetic susceptibility readings are taken every metre. For unconsolidated core this is measured in situ and results recorded, in SI units (kappa), in the assay log sheet.

Geotechnical logging records casing size, bit size, depths, intervals, core loss/gain, core recovery with weathering index, RQD, fracture index, jointing and joint wall alteration and a simple geological description. Geotechnical logging covers holes up to K215 and K220, K226,K238,K239,K284,K304,K314,K320,K246,K248,K253,K256,K258,K263,K265K26, K289,K293,K303,K306,K317,K325,K327,K329,K339,K340,K349,K365,K371 and K464 to K495. Otherwise, only sulphides are recorded before the core is cut. For oriented core, additional point data is collected, as defined by depth and alpha and beta angles of fabrics (Figure 11-1).

Geological logging uses a from-to format to record depths, rock codes and brief descriptions of the lithological units and angles of contacts. Sample intervals are measured-off by the project geologists and a line drawn along the length of the core to indicate where the core must be cut. This line is chosen to be at 90° to the predominant structure so that each cut half of the core will be a mirror image.

Core cutting by diamond saw is conducted in a dedicated core saw shed while unconsolidated material is split using spoons or trowels, with half the diameter of the sample being removed for assay. Each sample interval is placed in a plastic bag with a sample ticket. The bag is labelled with the hole and sample numbers using a marker pen.

**Figure 11-1: Structural Core Logging Using Jig**

Early exploration samples were 2.0 m in length (holes K1-K18). For holes K21-K27, the 2 m sampling interval over suspected mineralised zones (rich in arsenopyrite and pyrrhotite) was maintained but sampling adjacent to the mineralised zone was extended to 4 m. Subsequently, from K27 to K40, 1 m samples were introduced for target intersections, retaining 2 m intervals over suspected weakly mineralised material. Thereafter, the adopted norm has been to sample boreholes uniformly at 1 m intervals for the entire ultramafic unit and within 20 m selvedges into the hanging wall and footwall gneisses.

### 11.3.1 Bulk Density Measurements

Bulk density readings are taken at 2 m intervals within the same lithology and on every lithological break. This is carried out by weighing samples in air and water with a balance. Porous samples are first wrapped in plastic. For drillholes K1-130, measurements were carried out on half core, i.e. post-sampling, but for subsequent holes whole was used. Measurements are recorded using a balance with top and under-slung measuring capabilities with detection limit of +/-1 gm (Figure 11-2).

**Figure 11-2: Measurement of Bulk Density**

The balance is regularly checked (recalibrated using certified weights). In lithological units of less than one metre thickness, a single sample is measured, while in thicker units, one sample every 2–3 m is measured. Density measurements are carried out using Archimedean principles for consolidated fresh core and mass/volume determinations on loose granular material. Density is computed from weights of small pieces of core (10 cm–15 cm).

For unconsolidated material, density is measured by filling to the brim a container of volume 180 cm and the density is the weight of the sample divided by 180.

The range of bulk densities by geological unit is shown in Table 11-1.

**Table 11-1: Dry Bulk Densities**

| Name/Unit | Code | Rock Description | Mean |
|---|---|---|---|
| Hanging Wall Complex – HWC | GNqf | Quartzo-feldspathic banded leucocratic gneiss | 2.70 |
| | GNa | Hornblende plagioclase gneiss in lit par lit repetition with GNqf | 2.98 |
| | QUl | quartz rich layer in migmatite of probable metasedimetary origin | 2.70 |
| Contact zone rocks on HWC/ FWC | GNgp | Garnet phlogopite ± actinolite gneiss | 2.96 |
| Silicified Metamorphosed Ultrabasic suite (SMUS)-ore zone | UMmt | magnetite-tremolite-chlorite schist | 2.91 |
| | UMtc | tremolite-chlorite –talc schist | 3.03 |
| | UMpt | phlogopite-chlorite-tremolite schist | 2.98 |
| Footwall Complex - FWC | GNqf | Quartzo-feldsparthic banded leucocratic gneiss | 2.66 |
| | Gna | Hornblende plagioclase gneiss in lit par lit repetition with GNqf | 2.89 |
| | QUl | quartz rich layer in migmatite probably metasedimetary band | 2.70 |
| Syn- to late-tectonic aplites, pegmatites and granitoids). | GRun | Undifferentiated biotite bearing granite | 2.72 |
| | GRpb | Phlogopite- biotite granite | 2.82 |
| | GRsv | Sulphide-rich phlogopite microcline Granite | 3.02 |
| | GRbr | Quartz biotite-orthoclase granite breccia | 2.70 |
| | GRto | Tourmaline ± beryl granite , tourmaline and albite veins | 2.71 |
| | QZv | Quartz and quartz tremolite intrusive vein | 2.79 |

The bulk density data in the database is used in the resource estimate and is discussed further in Section 14.

Quality assurance protocols have developed through several cycles, with various consultants contributing to the present status. QA/QC protocols were not very rigorous in the 1999-2000 campaign and simply involved the sparing use of core duplicates. Certified standard materials (CRMs) were first utilised in 2005-2006 campaigns. QA/QC procedures were then considerably tightened by AMC and core duplicates and assay pills were phased out.

Field samples collected from various projects are stored in a secure facility at the field base camp guarded by a private security firm SOGUSS prior to dispatch to the sample preparation laboratory where retained un-assayed duplicates are stored.

### 11.3.2 Preparation and Analysis

*1999–2000 Campaigns*

During this first drilling campaign, core samples were cut with a diamond saw and two metre samples were despatched to the SGS laboratory in Abidjan, Ivory Coast, for assay. Sample pulp check assaying was conducted through the OMAC laboratory in Ireland (OMAC). However, no standard or blank sampling was undertaken, nor any standard QA/QC procedures implemented.

*2005–2006, 2008 Campaigns*

In August 2005 a sample preparation facility managed by the Alex Stewart Group (OMAC) was opened in Monrovia, and from that point samples from the Project have been crushed, pulverised and split in Monrovia, and sample splits shipped by DHL to OMAC.

During the 2005–2006 and 2008 drilling campaigns some additional QA/QC procedures were introduced. Notably blanks and CRMs were together inserted into the sample stream at a rate of one in ten. The 19th and 20th samples were QA/QC samples, in which the 19th sample was a blank (1 kg of Monrovia sand) and the 20th was either an assay pill or Rocklab Ltd. standard (as 50g sealed sachets). Assay pills were crushed and inserted into a bag of 1 kg of Monrovia sand to make up a sample.

At the Monrovia sample preparation facility, the total sample (±3.5 kg) was dried to a core temperature of 110°C, jaw crushed to a nominal 2 mm, riffle split to 1 kg, then milled in an LM2 mill to a nominal 95% passing 75 mm. An analytical pulp of approximately 200 g was sub-sampled, of which a 100 g sub-sample was sent to Ireland for assay pulp and fusion in a lead collection fire assay. The resulting prill was dissolved in aqua regia, followed by an AAS finish.

*2009/2010 and 2011/2012 Campaigns*

Prior to shipment, final checking was carried out in the presence of a senior geologist and two field assistants to ensure sample identities were correct, samples intact and there were no omissions. Quality control standards and blanks samples were inserted at pre-determined intervals at this point. Samples were sent from site, on a complete-hole basis, directly to the OMAC preparation facility in Monrovia, along with documentation, which acted as a receipt and sign back. Sample transfer and delivery to the OMAC laboratory in Ireland was able to be monitored and tracked via the OMAC website, until assay results were released.

During 2011 the same sample preparation protocol was applied. However following the merger between OMAC and the ALS Group, ALS Chemex was no longer eligible for use as a referee company. Consequently SGS Canada Inc. (SGS) was commissioned as a reference lab. OMAC, ALS Chemex and SGS, including the Monrovia sample preparation facility, are independent of Bea and Aureus.

The flow chart in Figure 11-3 summarizes sample collection, sample preparation, assaying and QA/QC procedures adopted during the 2009/2010 and 20112012 drilling campaigns, including recommended modifications made after a site visit in December 2009 undertaken by AMC.

OMAC is accredited by Irish National Accreditation Board to ISO 17025 and fire assay is included in the Schedule of Accreditation. OMAC participates in inter-laboratory proficiency testing and certification programmes (round-robins).

On arrival of the prepared pulps at the laboratory, samples were checked against the submission sheet, logged into LIMS, and homogenized to prevent segregation that might have occurred in transit. Large consignments of samples (>300) were split into smaller sub-batches of 200 samples for convenience of processing.

Samples were weighed, mixed with flux and fused in clay crucibles. Lead buttons produced after fusion were cupelled, forming dore prills that were digested in aqua regia, and digests were analysed for gold using a Varian AA Spectrometer.

Samples were analysed in lots of 50 and include 44 original samples, four duplicates, one CRM and a blank.

For umpire assaying, pulps were taken from coarse rejects stored in the sample preparation laboratory of OMAC located in Liberia. Dry rejects were crushed entirely to 80% passing 2 mm using terminator jaw crusher. 1 kg crushed material splits were taken using a riffle splitter and milled using a LM2 mill to 90% passing 100 micron. 50 g portions of prepared pulp were packed in plastic mini-grip bags and couriered to the ALS Chemex laboratories in Canada. ALS Chemex is part of the ALS Minerals group which 'maintains ISO 9001:2008 and ISO/IEC 17025:2005 certifications' and operates a laboratory quality management system (QMS) involving both internal and external controls (e.g. round-robin programmes and proficiency tests).

Sample decomposition was again by fire assay fusion (FA-FUS03 and FA-FUS04 in the method coded Au-AA25), utilizing 30 g of sample followed by atomic absorption spectroscopy (AAS) finish.

*2014 Grade Control Drilling*

Pre-generated sample ID's and Standard QAQC inserts were developed on 1m intervals for all holes. Lab samples were taken for every interval (not just those perceived to be in/and around the ore zone), but only those intercepting the resource wireframe limits plus an additional 5m wide buffer zone where dispatched to the lab for both Au and As analysis. Duplicates were also taken within the ore zone for checks on reproducibility of assay results. Bulk samples were split on site using a Jones Riffle splitter down to a 2.5kg lab sample. These lab samples were double bagged with Sample ID tags inserted between the layers as well as ID's written on the outside before being sealed with zip ties.

Bulk sample values were recorded for each meter, and plotted alongside the returned assay values for sample support comparisons. (Table 11-2) details the statistics recorded for the bulk recoveries of both phases of drilling:

**Table 11-2: Grade Control Drilling Bulk Recovery Statistics**

| Pit area | Min Value (kg) | Max Value (kg) | Mean overall (kg) | Mean dry (kg) | % overall below 15kg |
|---|---|---|---|---|---|
| Larjor | 1 | 56.6 | 25.11 | 32.36 | 27 |
| Kinjor | 1.1 | 53.8 | 26.96 | 32.55 | 22 |

The percentage of recoveries below 15kg is representative of RC recovery figures in oxide materials during a drilling campaign that occurred mostly during the rainy season. Recoveries in the fresh (Mean dry) once holes had been collared off improved significantly as expected.

Laboratory samples from the first 41 holes of the Larjor phase of drilling were sent to SGS in Monrovia. SGS subsequently pulled out expatriate staff due to Ebola, at which time all remaining samples for the campaign were dispatched to ALS in South Africa. Comparative statistical work was undertaken to ensure that assay results from both labs could confidently be used in development of a Larjor Grade Control Block Model.

Comparative statistical work and block model analysis were also undertaken by external consultants to ensure the validity in combining both pre-existing DD assay data with the RC assay data before generation of both the Larjor and Kinjor EOY1 Grade Control Block Models. These two separate models were then later combined with a 2.5m re-blocked version of the resource model for those areas of the pit not yet drilled with RC to produce a hybrid Block Model incorporating both sets of information for use in short and long term planning scenarios. SRK notes that the resource estimate detailed in Section 14 of this report does not take into account this additional Grade Control Drilling and Aureus has developed additional block models incorporating this date for the purposes of mine planning but will update the resource block model in due course as additional information is obtained.

Aureus indicates that future Grade Control drilling will be undertaken using a Sandvik DR560 DTH drill owned by Aureus with convertible RC top drive. A 140mm hammer will be used for sampling 1m intervals to a max depth of 30m. Bulk samples will be reduced using a rotational cone splitter. This will enable 20RL of Benches to be drilled out at any one time with 5m of over drill. Once site establishment of the ALS laboratory has taken place, Grade Control drilling will be an ongoing activity with results constantly updating the hybrid model developed with additional information beyond the EOY1 pit limits already drilled out using the EDM 2000. If additional information is required beyond this, then an external contractor will be brought in to undertake any campaigns where holes beyond 30m are required.

Blast hole sampling will be used in conjunction with RC Grade control methods to confirm ore zone boundaries during mining but will not be used in the updating of future block models.

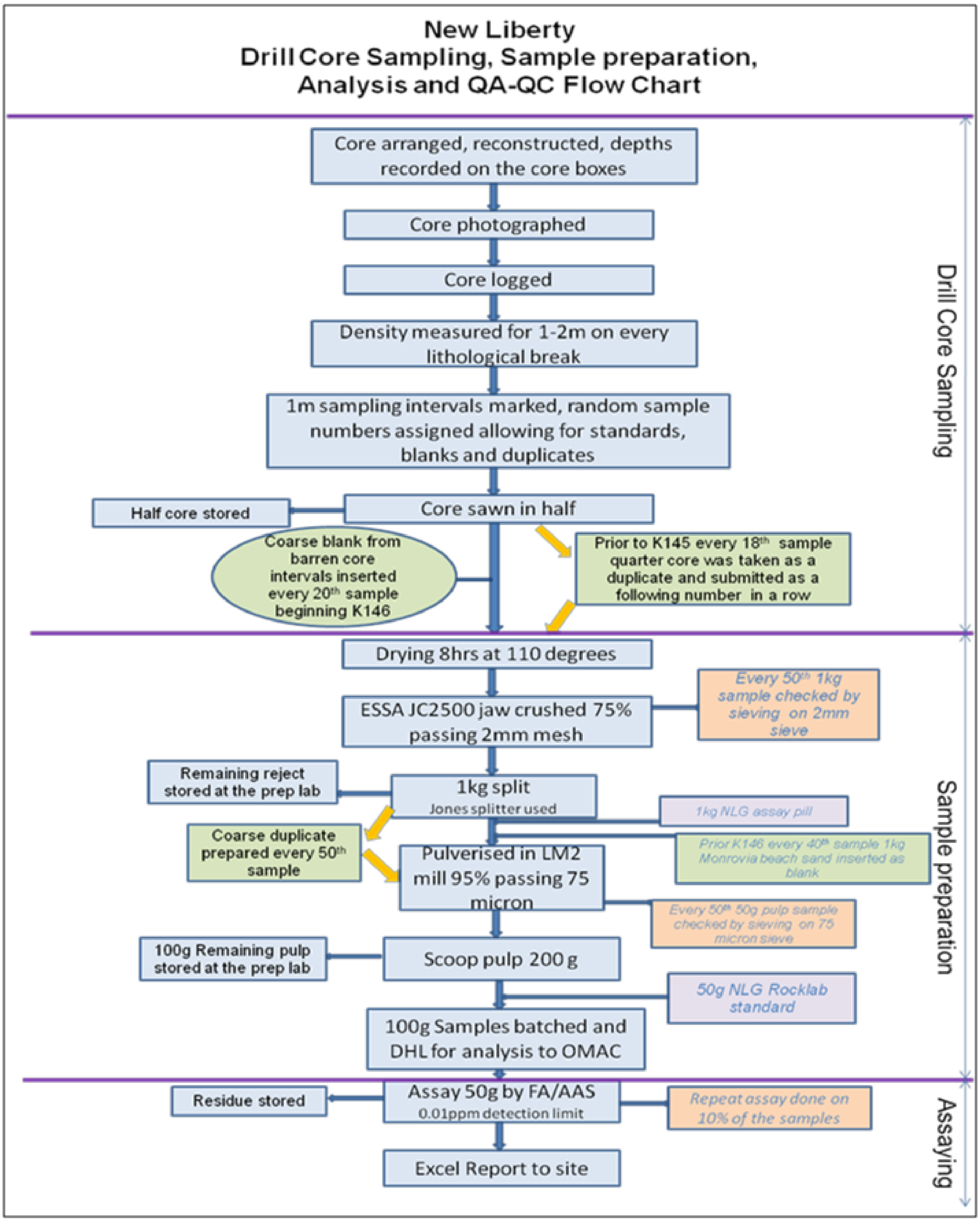



**Figure 11-3: Sample Preparation and QA/QC Flow Chart**

## 11.4 Assay QA/QC

For the discussion below, the drilling campaigns have been combined into three periods, since little QA/QC work was carried during in the early campaigns. AMC undertook detailed QA/QC analyses for the periods 2005–2008, 2009–2010 and 2011-2012; however only the 2009-2010 analyses are presented here in any detail, with the remaining periods covered in summary form only.

### 11.4.1 Period 1999–2000

*Field Duplicates*

Five quarter core samples from split core were collected by ACA Howe during its work in 2000 and sent for preparation and fire assay at OMAC laboratories. Table 11-3 compares the original samples and the Howe checks.

**Table 11-3: 1999–2000 Field Duplicate Comparison**

| Hole ID | From (m) | To (m) | Width (m) | Howe Check OMAC | Original Value SGS |
|---|---|---|---|---|---|
| KDG-2 | 32 | 34 | 2 | 16.4 | 23.0 |
| KDG-2 | 36 | 38 | 2 | 0.8 | 0.4 |
| KDG-8 | 34 | 36 | 2 | 4.9 | 18.2 |
| KDG-15 | 30 | 32 | 2 | 1.0 | 13.0 |
| KDG-18 | 44 | 46 | 2 | 0.4 | 1.0 |
| KDG-10 | 70 | 72 | 2 | 3.6 | 5.0 |

ACA Howe concluded that, although the sample-to-sample comparisons were poor, the results should be seen in the context of work by Lakefield Research (Lakefield Research, 1999a) which showed the presence of abundant free gold. Consequently, a strong nugget affect can be expected to influence the correlations.

### 11.4.2 Period 2005–2008

*Blanks*

A total of 368 blank samples were submitted to the OMAC laboratory during the 2005-2006 and 2008 campaigns. Generally, the assays performed as required (lower than three times detection limit), with four obvious high-grade outliers. The outliers are probably a consequence of sample mix-up, while a further five samples with higher than expected values could indicate laboratory contamination.

Forty pulp blanks were also routinely inserted into the sample stream, and analysis shows a good performance of assays against this blank, with one outlier recorded.

*Standards*

Eight different Rocklabs Ltd. standards were used during the 2005-2008 drilling campaigns, with certified gold values ranging from 0.2 g/t Au to 13.64 g/t Au, which suitably reflects the Project deposit gold grade range.

The notable features are the absence from the database of seven 2005 results against the 0.58 g/t standard, a number of outliers observed for 0.58 g/t the standard and a marked low bias for the 1.32 g/t standard. The assays for the 3.49 g/t standard performed within acceptable limits, but with a slight low bias.

During 2006 the assays performed much better against the standards, although some low bias is evident. The poorer standards performance in 2005 relative to 2006 is consistent with a common chronological trend, which typically reflects the bedding down of procedures at the start of a campaign.

The improved performance in 2006 was not sustained in 2008, even though the same standards were used, and a more marked low bias can be observed.

SRK is not aware of any control procedures in place during that period to check and react to QA/QC concerns, nor has any documentation been found that identifies possible contributory factors to the reduced standards performance during 2008. AMC considered it possible that the standards deteriorated in storage on site during the inter-campaign period and SRK also considers this to be a possibility.

The low grade standard (0.2 g/t Au) performed within acceptable limits but low bias can still be observed.

*Laboratory Repeats*

There were 832 laboratory repeats results recorded for the 2005-2008 drilling. Prior to statistical analysis, data with values below 15 times the detection limit and above 15 g/t Au were removed. Eight obvious outliers were also excluded, leaving 409 pairs.

Statistical summaries and charts indicate that, while there is good linear correlation between sample pairs, the point cloud shows a relatively wide spread. A precision value of 18.5% was achieved, in the context of a recommended precision for pulp pairs of less than 10%. This suggests that a high nugget effect is present.

### 11.4.3 Period 2009–2010

*Blanks*

Initially in this period (from drillhole K131) Monrovian beach sand was used to form blank samples, but from hole K146 onwards blanks were taken from barren hanging wall material, submitted as coarse samples which pass through all the preparation stages. The results included two outliers and five samples above three times the detection limit, while the remaining assays performed as expected (Figure 11-4). Pulp blanks recorded one outlier that most likely indicates a misclassification of a standard.

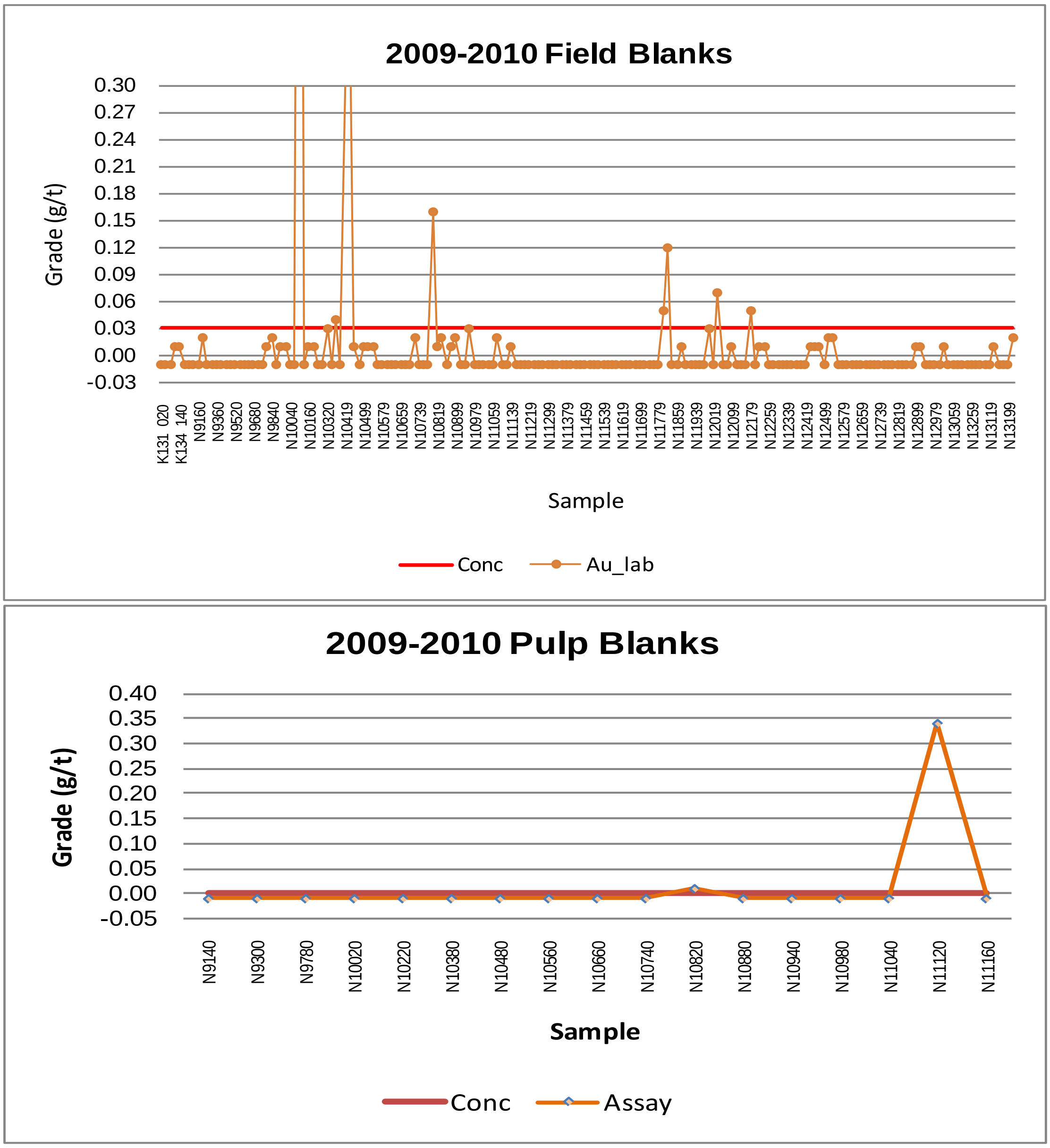



**Figure 11-4: 2009–2010 Blank Sample Analysis**

*Standards (CRM)*

A total of eight standards were used during the 2009/2010 drilling campaign, which had the following suppliers and gold values:

- Rocklabs: 0.20 g/t, 0.99 g/t,1.031 g/t and 5.911 g/t.
- Geostats Pty Ltd: 0.38 g/t, 0.99 g/t,1.52 g/t and 1.96 g/t.

The performance of assays against all the standards, from both sources, was very poor, most clearly reflected in a strong negative bias (Figure 11-5). In addition, a small number of outliers were also recorded, suggesting mislabelling during sample submission or sample preparation.

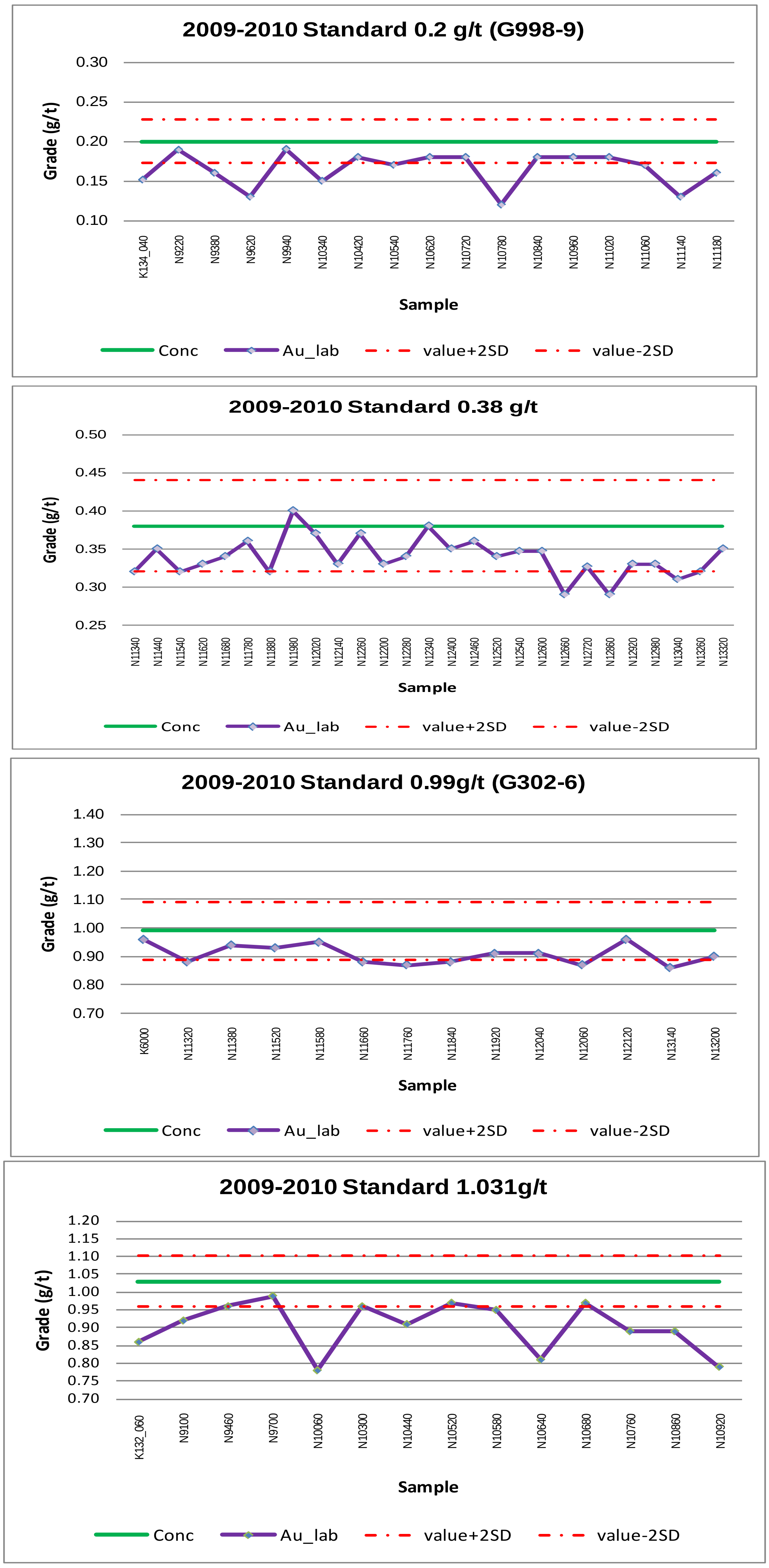
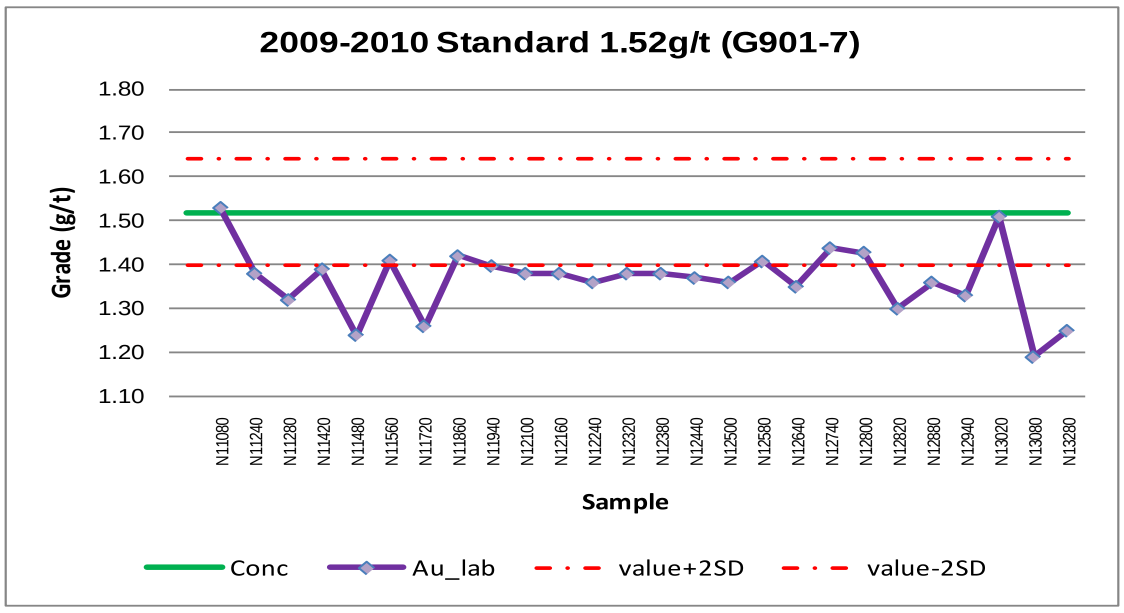
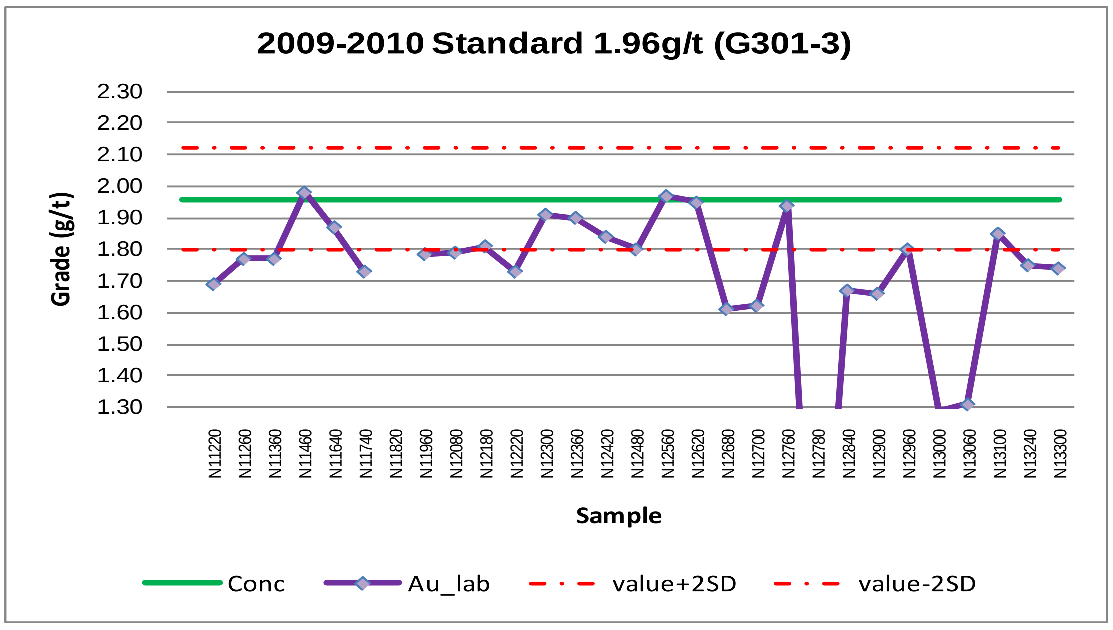



**Figure 11-5: 2009–2010 Standards Analysis**



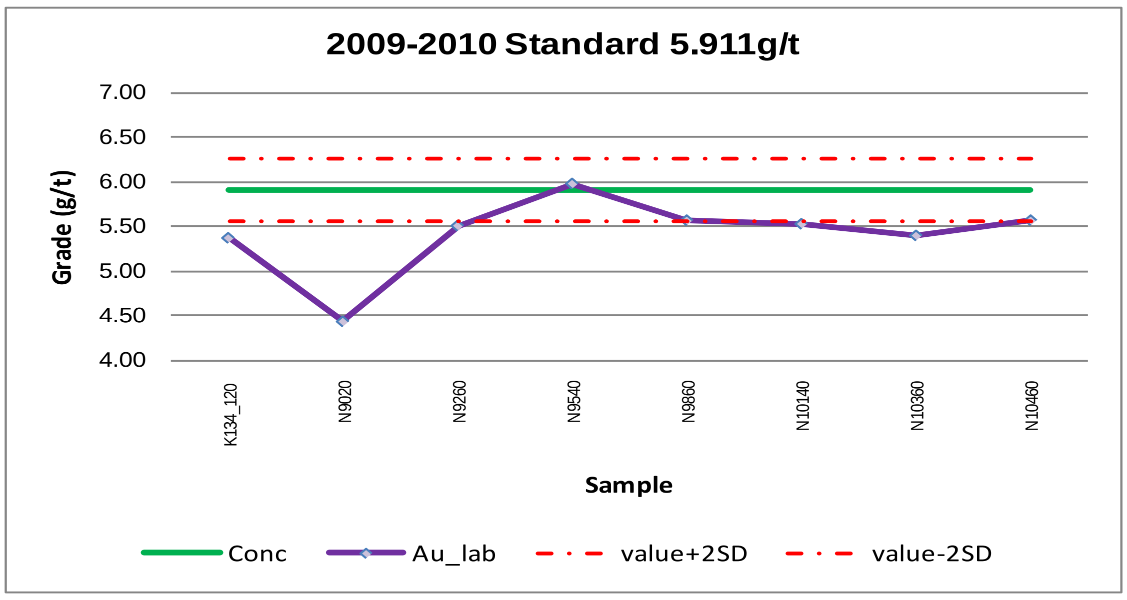




**Figure 11-5: 2009–2010 Standards Analysis (continued)**

Monitoring of standards data was not routinely followed during the drill programme, and this fact, combined with time lags between the drilling and sampling work and the receipt of sufficient standards results for analysis, meant that the biases described above were not fully recognized until the end of the main drilling programme.

The presence of a bias suggested either problems with the original CRM samples or systematic problems associated with assaying. In an attempt to better understand this matter, a re-assay programme was designed in which 10%-15% of the sample data, specifically focused on the mineralised intervals, was despatched for analysis at an umpire laboratory (discussed below).

*Drilling Duplicates*

### Quarter Core Duplicates

At the start of the 2009/2010 campaign, field core duplicates were produced every 18th sample using quarter core. On the basis of low sample volume and concerns that sampling errors could not be separated from intrinsic nugget effects, AMC recommended Aureus cease this practise and increase the number of crush duplicates.

### Crushed Duplicates

From drillhole K145 onwards only crushed sample duplicates were produced, but without the corresponding recommended increase in frequency of duplication, leaving the number of crushed duplicate produced and routinely split as approximately every 50th sample.

A total of 49 samples were reported as crushed duplicates, only 10 of which were located within a mineralised interval (Table 11-4).

**Table 11-4: 2009–2010 Crushed Duplicate Pairs Value**

| Hole ID | Sample | Au | C.dup Au |
|---|---|---:|---:|
| K136 | N9335 | 7.64 | 6.92 |
| K142 | N9981 | 5.76 | 6.26 |
| K140 | N9806 | 1.86 | 1.89 |
| K154 | N11715 | 0.38 | 0.35 |
| K146 | N10749 | 0.19 | 0.32 |
| K150 | N10450 | 0.30 | 0.30 |
| K153 | N11465 | 0.20 | 0.21 |
| K152 | N11215 | 0.20 | 0.18 |
| K152 | N11165 | 0.17 | 0.17 |
| K153 | N11417 | 0.10 | 0.12 |

*Drilling Laboratory Repeats*

A total of 503 laboratory repeat assays were undertaken by the primary laboratory, OMAC, of which only 138 exceeded ten times the detection limit. For statistical analysis of laboratory repeats, all assays below fifteen times the detection limit and grades in excess of 10 g/t Au were excluded.

The laboratory repeat assay results are presented in Table 11-5 where the 12.5% precision statistic is considered high, since the precision for laboratory repeats is expected to be well below 10% (see also Figure 11-6). The poor precision could be attributed to inherent high nugget effect or poor preparation procedures.

**Table 11-5: 2009–2010 Laboratory Repeats Statistics**

| Item | Au | Au r | Unit | Item | Value | Unit |
|---|---|---|---|---|---|---|
| Pairs | 91 | 91 | | Total mean | 0.77 | g/t Au |
| Mean | 0.79 | 0.76 | g/t Au | Absolute diff of means | 0.04 | g/t Au |
| Minimum | 0.15 | 0.16 | g/t Au | Regression slope | 1.08 | |
| Maximum | 4.82 | 4.41 | g/t Au | Av HARD (AMPD/2) | 4.14 | % |
| Variance | 0.85 | 0.70 | g/t Au sq. | Av HRD (half relative diff) | 0.93 | % |
| CV | 1.16 | 1.11 | | Precision (at 95%) | 12.5 | % |
| | | | | Absolute error (at 95%) | 0.10 | g/t Au |

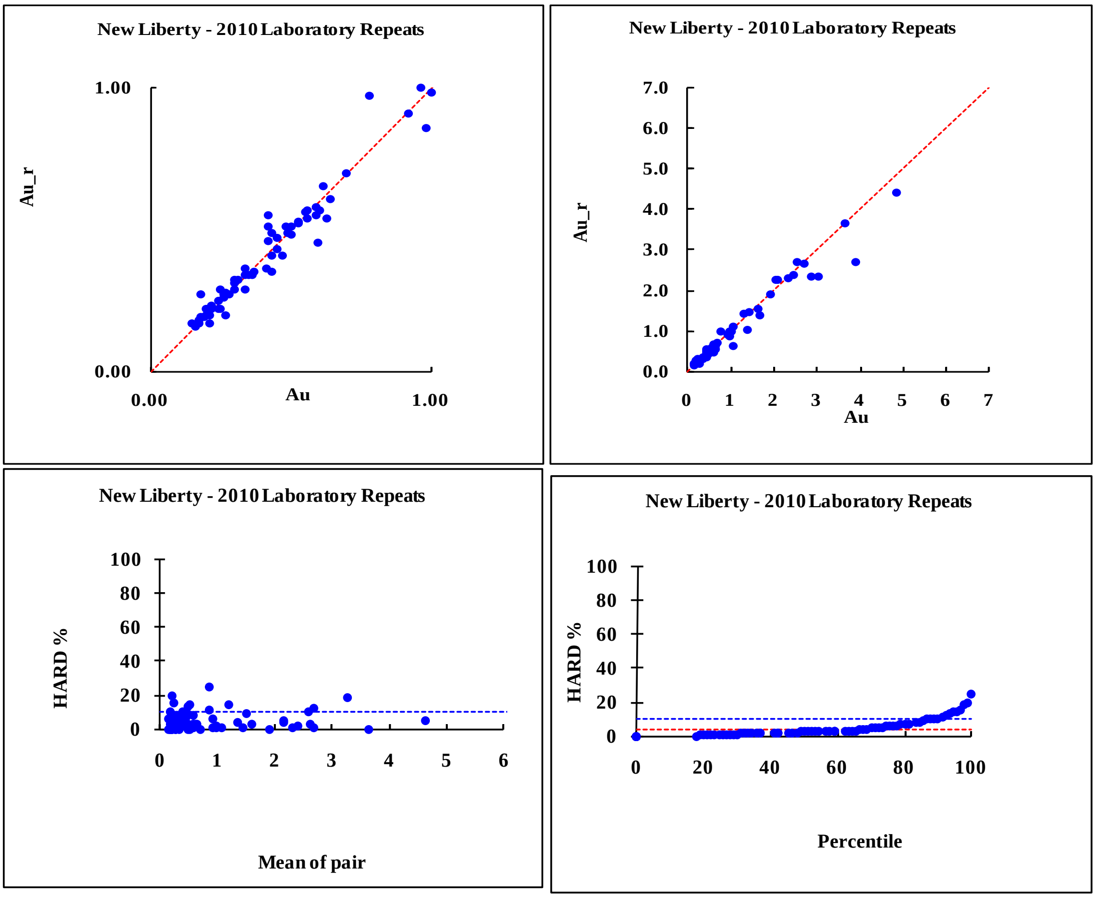



**Figure 11-6: 2009–2010 Laboratory Repeats Analysis**

*Drilling Re-assay Samples*

Some sample batches were submitted for re-assay because of concerns arising from a QA/QC review. Of the 180 results generated, thirty pairs remained for statistical analysis after removing samples below 0.15 g/t Au and above 5.0 g/t Au. A poor precision of 22.9% was achieved (Table 11-6 and Figure 11-7).

**Table 11-6: 2009/2010 Re-assay Samples**

| Item | Au | Au_r | Unit | Item | Value | Unit |
|---|---|---|---|---|---|---|
| Pairs | 30 | 30 | | Total mean | 1.01 | g/t Au |
| Mean | 1.00 | 1.01 | g/t Au | Absolute diff of means | 0.01 | g/t Au |
| Minimum | 0.17 | 0.15 | g/t Au | Regression slope | 0.96 | |
| Maximum | 3.74 | 3.44 | g/t Au | Av HARD (AMPD/2) | 7.88 | % |
| Variance | 1.11 | 1.16 | g/t Au sq. | Av HRD (Half Relative diff) | 0.46 | % |
| CV | 1.05 | 1.06 | | Precision (at 95%) | 22.9 | % |
| | | | | Absolute error (at 95%) | 0.23 | g/t Au |

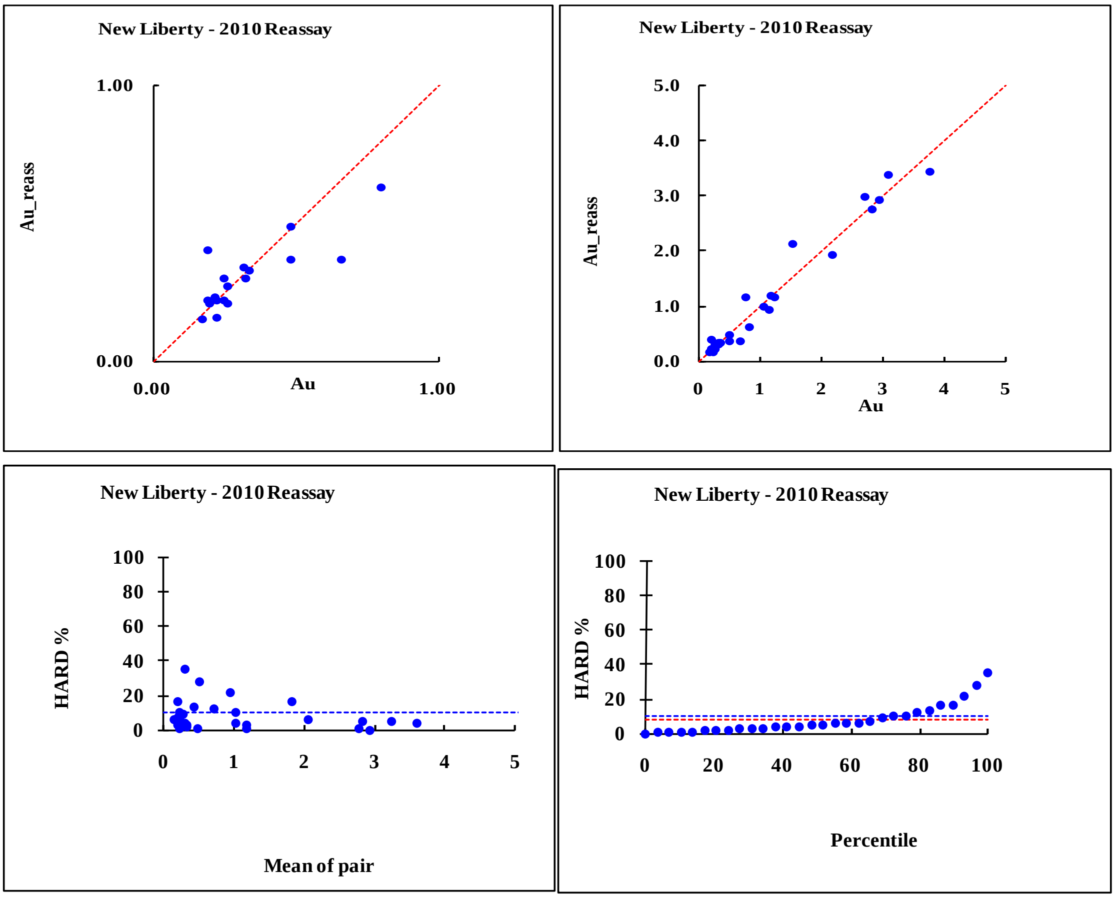



**Figure 11-7: 2009–2010 Re-assay Sample Analysis**

*Umpire Laboratory Check Assays*

Aureus selected ALS Chemex as an umpire laboratory, and for the programme a new set of standards was purchased from Rocklabs. ALS Chemex used a 30 g fire assay method compared to the 50 g used at OMAC.

A total of 1,051 selected pulp samples were despatched to ALS Chemex for assaying, including 52 blanks and 50 CRMs. After removing outliers and values below ten times the detection limit 732 pairs were available for inter-laboratory comparison.

The results of the comparison are presented in Table 11-7 and charted in Figure 11-8. Considering that the pulps were re-prepared from coarse rejects a precision of 19.3% is within acceptable limits. The OMAC results show some negative bias (1.05%) relative to the ALS Chemex values.

**Table 11-7: Inter-laboratory Comparison**

| Item | Au 1 | Au2 | Unit | Item | Value | Unit |
|---|---|---|---|---|---|---|
| Pairs | 732 | 732 | | Total mean | 2.01 | g/t Au |
| Mean | 2.01 | 2.00 | g/t Au | Absolute diff of means | 0.01 | g/t Au |
| Minimum | 0.10 | 0.10 | g/t Au | Regression slope | 0.99 | |
| Maximum | 26.85 | 28.40 | g/t Au | Av HARD (AMPD/2) | 6.27 | % |
| Variance | 14.16 | 14.00 | g/t Au sq | Av HRD (Half Relative diff) | -1.05 | % |
| CV | 1.87 | 1.87 | | Precision (at 95%) | 19.3 | % |
| | | | | Absolute error (at 95%) | 0.39 | g/t Au |

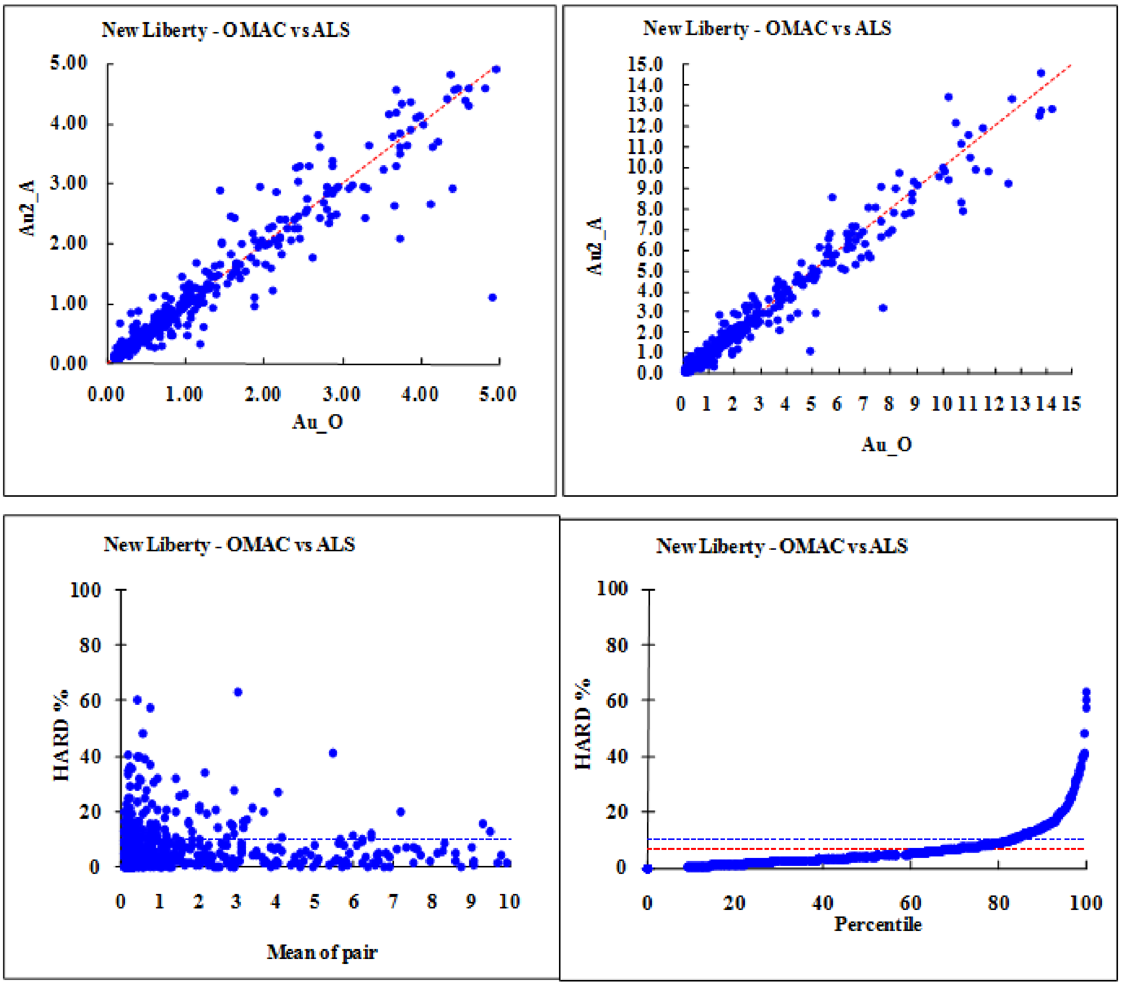



**Figure 11-8: 2009–2010 Inter-laboratory Comparison**

Prior to the umpire laboratory programme, the newly purchased standards were tested by sending five samples to each of ALS Chemex and OMAC. Low bias was observed in the results for both laboratories, with all OMAC values being outside the expected range.

The analysis of assay results from standard samples submitted during the ALS Chemex umpire laboratory programme show the presence of 3 outliers, as well as a consistent low bias (Figure 11-9), albeit less of a bias than in the original OMAC results.

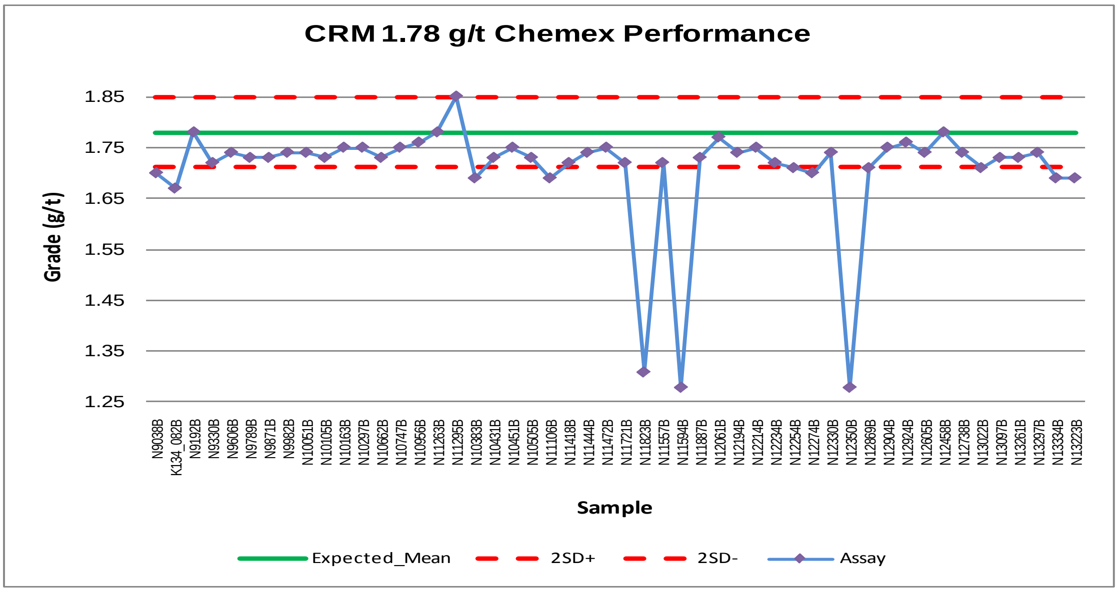



**Figure 11-9: 2009–2010 Umpire-laboratory Standards**

In the umpire laboratory programme, other than one identified outlier, blank samples performed well.

### 11.4.4 Period 2011-2012

Three sources of blank material (1,786 samples) and eight different CRMs (769 samples) were utilised during the 2011-2012 campaign, with the CRMs ranging in gold values from 0.606 g/t Au to 4.107 g/t Au.

From the analyses of blank sample assays it was clear that a small number of significant gold assays were related to sample insertion error where CRMs had been substituted for blanks. An additional set of assays that are above the expected values could not be explained by CRM swapping and either reflect mislabelling of non-QA/QC samples or laboratory problems. However, the very small percentage of these assays indicates that they are not material.

Similarly it was evident that a small number of CRM samples had probably been mislabelled, since the returned assays correspond closely to expected blank or other CRM values. More significantly, the CRM assays exhibit a similar persistent low bias relative to the expected values to that observed in the 2010 review.

Subsequent discussions with the primary laboratory and the CRM suppliers concluded that the apparent bias is probably not significant and it is notable that the umpire laboratory returned assays consistent with those from the primary laboratory.

A suite of 1,116 samples were submitted to umpire laboratory, SGS, for verification against the original OMAQ assay values. Analyses of the results were conducted on the total sample set and for the seventeen individual batches. The differences for the total set show some degree of spread, but no apparent bias, except for a subtle high tendency for of the umpire assays for the assays below 1.0g /t Au. For the individual batches, some cases may be inferred to show bias but this is not in a consistent direction or magnitude.

### 11.4.5 Observations

The standard of sample and assay QA/QC data collection and analysis has steadily improved over the various drilling campaigns, as better protocols have been introduced and lessons learnt from previous work. Nonetheless there remains a legacy of uncertainty associated with those data subsets where procedures were less comprehensive. The 2011/2012 infill drilling programme facilitated the inclusion of 'replacement' drillholes for some historical holes where doubt regarding assay quality exists.

Some evidence of sample mix-ups raises the concern that other less obvious cases may exist but go undetected, however, these appear to be relatively isolated cases.

Even with improved QA/QC procedures, there remains a problem (common to many exploration campaigns) that, as a consequence of time lags between the submission of samples and the receipt of sufficient results for analysis, drilling programmes may be well advanced before matters of concern are detected. The delay in detecting trends in the Project sampling results is exacerbated by the low proportion of routine QA/QC samples within mineralised material and the distance between the Project site and the laboratory.

There are remaining areas of concern relating to assay precision and apparent low bias, which have only been partially explained or resolved. Nonetheless AMC believed that the gold assay data was suitable for use for resource estimation at the confidence levels that have been assigned and SRK agrees with this conclusion.

# 12 DATA VERIFICATION

## 12.1 Source Data Verification

In 2010, AMC randomly selected a suite of drillholes representing approximately 20% of all drillholes used for the 2010 resource estimation, and checked the database-entered data against the original sources (hard copies). Hard copy documents of assays are limited to holes K1 to K56, as subsequent assays were received from the laboratory in digital CSV format only. For the set of assays checked, no database errors were detected.

AMC identified some differences in downhole survey records from various sources for the 2000-2006 drilling campaign. Aureus advised that these ambiguities related to early inconsistencies in the manner in which magnetic declination had been applied, as well as different recording formats. Following a review of the various datasets, Aureus supplied an accepted database of downhole survey records. Original physical downhole survey records from the early drilling campaigns are not available, due to deterioration in storage, and therefore database values could not be verified back to source.

During cross-checking of geological logging data, it was noted that modifications to the logging codes between drilling campaigns and re-logging of old holes has meant that hand written data does not always match database entries.

AMC also conducted a source data check of approximately 15% of holes drilled during the 2011/2012 drilling campaign. A small proportion of the database contained typographical and similar errors were detected and referred to the Aureus field team. These have been rectified in subsequent updates.

## 12.2 Database Field Integrity

A number of data validation tests of the sample data in the database were undertaken by AMC. A standard validation macro was used to test for such instances as duplicate samples, overlapping intervals, unmatched hole identifiers in collar, survey and assay files, and inappropriate downhole distances (such as negative values).

A separate macro was applied to generate basic statistics on all numeric fields in the desurveyed sample file to reveal any questionable values (e.g. negative lengths, out-of range coordinates). In addition, sample data was viewed graphically in 2D and 3D space, coloured on various code and grade fields, and critically assessed for any likely spatial or other problems.

AMC reported a limited number of queries to Aureus for correction or explanation, and the appropriate adjustments were made prior to further work by AMC.

## 12.3 Observations

As for the sample and assay QA/QC, procedures for data management and storage have improved over time. Remaining areas of uncertainty therefore relate mostly to the older data which cannot always be fully verified.

However, as the database grows it will not be as easy to maintain the same level of individual scrutiny of the data and AMC recommended that more rigorous data management and storage procedures be implemented to preserve a stronger audit trail for future data verification.

Aureus is in the process of employing a geological database manager and is organising a new geological database system to cope with larger datasets, and to improve QA/QC procedures in general.

Now that the Project is moving into its operational phase, Aureus is in the process of setting up a new Production database system, Maxwell Geosciences Datashed, in order to manage all the data that is generated and reporting required during the Production phase of the Project.

Aureus and Maxwell staff are currently undertaking the process of transferring all historical exploration data from the exploration database as well as data generated from the Grade Control drilling, from the previous database system into Maxwell Datashed.

SRK has reviewed the verification work done by AMC and the revised processes implemented by Aureus and is confident that the quality of the available data is sufficient to support the estimate of Mineral Resources presented later in this report.

# 13 MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction

### 13.1.1 Background

The details of previous metallurgical test work phases 1–7 are documented within the previous Technical Report on Updated Mineral Resources and Mineral Reserves dated 22 October 2012.

A more recent metallurgical test work programme has now been completed as part of the optimization phase of the Feasibility Study. The test work programme was undertaken by Australian, Perth based, ALS Laboratories (ALS).

The metallurgical scope of the optimisation phase was designed with the objective of completing all metallurgical test work required to finalise the process design criteria in order to finalise the process flowsheet, size mechanical equipment and determine plant capital and operating costs.

The test work programme included the following:

- An evaluation of Preg-Robbing (Composite samples).
- CIL Optimisation test work (Composite samples).
- Evaluation of plant recovery and reagent consumptions (Composite and Variability samples).
- Continuous cyanide destruction and Arsenic removal test work (Composite samples).
- Grinding test work, Levin and IsaMill testing (Composite samples).
- Arsenic remediation and kinetic column test work.

A flow chart for the optimisation phase metallurgical test programme is detailed in Figure 13-1 below.

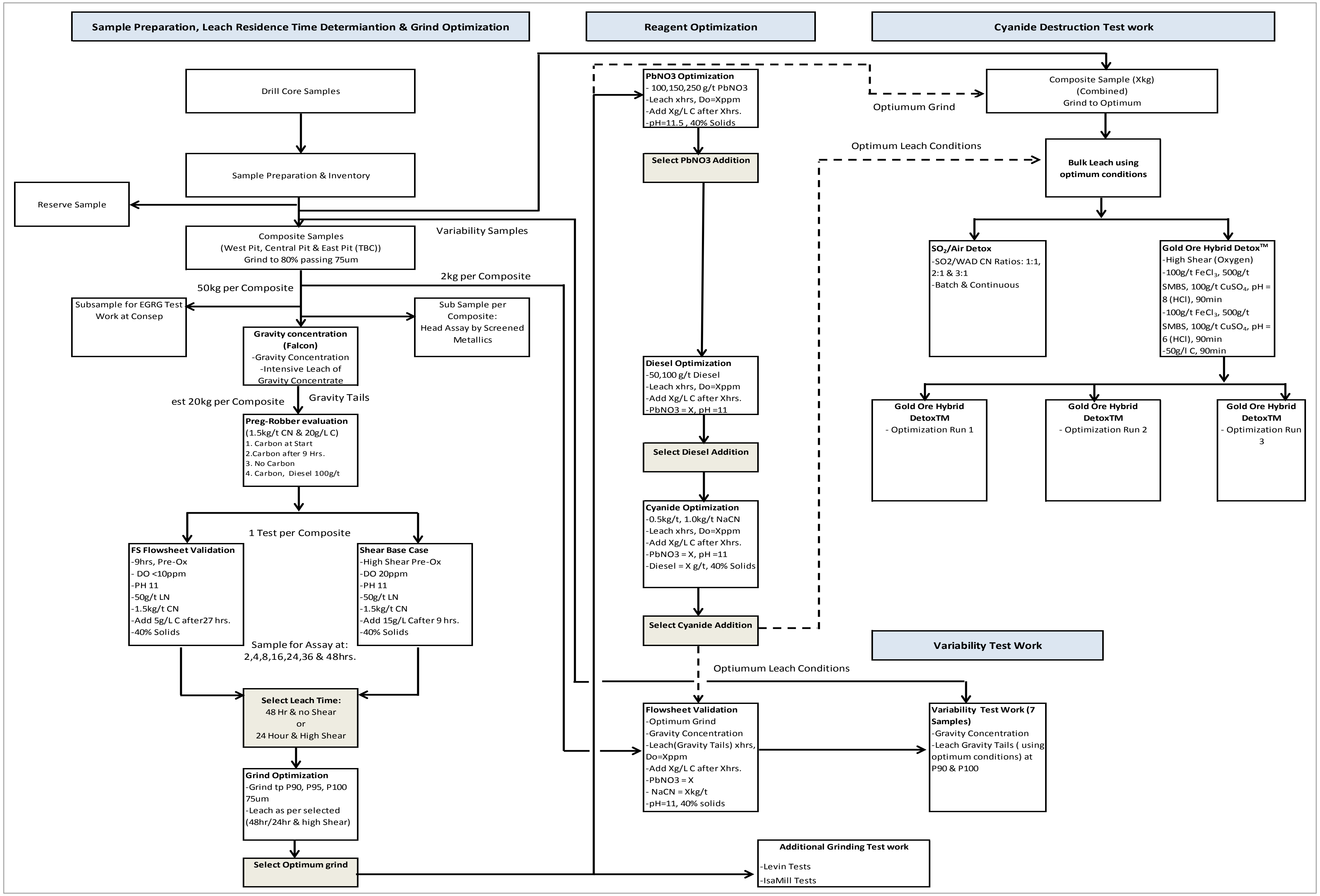



**Figure 13-1: Optimization Phase Test Work Flow Chart**

### 13.1.2 Test Work Samples

The metallurgical optimisation test work programme was performed on both composite and variability samples. A summary of the sample inventory list is presented in Table 13-1 below:

**Table 13-1: Optimisation Phase Sample Inventory List Summary**

| INVENTORY | | | |
|---|---|---|---|
| Composite ID | Drillhole Intervals (m) | JK Tech Comminution Samples | Mass (kg) |
| COMP #1 (K485A) | 24 - 40 | Assorted (see Appendix I for details) | 121.0 |
| COMP #2 (K485B) | 40 - 57 | | 152.0 |
| COMP #3 (K487) | 61 - 78 | | 129.6 |
| COMP #4 (K489) | 95 - 104 | | 75.2 |
| COMP #5 (K490) | 82 - 94 | | 91.1 |
| COMP #6 (K492) | 18 - 34 | | 117.4 |
| COMP #7 (K491) | 90 - 103 | | 98.5 |
| | | Sample ID | |
| K156 | 157 - 164 | N11881 - N11887 | 31.2 |
| K233 | 161 - 172 | N19782 - N19793 | 43.91 |
| K236 | 45 - 58 | N20014 - N20028 | 51.17 |
| K287 | 45 - 58 | N24001 - N24005 | 9.75 |

### 13.1.3 Chemical Analysis

The screened fire assays for the test work samples are presented Table 13-2 below:

**Table 13-2: Screened Fire Assay Results**

| Composite ID | +75 µm | | -75 µm | | | Calc'd Head Au (g/t) |
|---|---|---|---|---|---|---|
| | Mass (g) | SFA Au (g/t) | Mass (g) | SFA $Au_1$ (g/t) | SFA $Au_2$ (g/t) | |
| Master | 33.08 | 24.8 | 955.34 | 3.61 | 3.43 | 4.23 |
| Comp #1 | 26.23 | 13.7 | 946.8 | 2.52 | 2.32 | 2.72 |
| Comp #2 | 26.15 | 124 | 952.1 | 9.78 | 10.40 | 13.13 |
| Comp #3 | 25.79 | 6.27 | 967.0 | 4.57 | 4.75 | 4.70 |
| Comp #4 | 16.96 | 31.8 | 972.0 | 5.65 | 5.84 | 6.19 |
| Comp #5 | 25.92 | 6.52 | 962.5 | 3.22 | 2.81 | 3.11 |
| Comp #6 | 25.29 | 54.7 | 951.4 | 2.89 | 3.57 | 4.56 |
| Comp #7 | 23.26 | 22.8 | 971.6 | 1.81 | 1.89 | 2.34 |
| K485B | 25.49 | 121.0 | 965.8 | 7.65 | 7.40 | 10.44 |
| K490 | 18.75 | 32 | 972.7 | 2.13 | 2.05 | 2.66 |
| K156 | 25.59 | 61.60 | 973.9 | 4.69 | 4.75 | 6.18 |
| K233 | 20.51 | 11.4 | 964.8 | 4.42 | 4.89 | 4.80 |
| K236 | 26.60 | 5.72 | 960.9 | 4.12 | 4.31 | 4.26 |
| K287 | 22.70 | 41.2 | 964.2 | 3.75 | 3.99 | 4.73 |
| K492 | 12.61 | 166.0 | 965.1 | 3.12 | 2.61 | 4.97 |

### 13.1.4 Composite Samples

In October 2012, 800 kg of pre-crushed (12.7 mm) core sample material was delivered to ALS's mineral processing facility for metallurgical characterization test work. This material was the product material from comminution test work conducted on core samples at JKtech in September 2012. The sample material received from JKtech was composited to produce a 240 kg bulk master composite sample. This bulk sample was comprised of pre-crushed core samples from the western and central portions of the deposit at various depths. This composite represented the first six years of the mine schedule and did not contain material from the eastern portion of the deposit. The master composite had an assayed gold grade of 4.23 g/t as determined by screened fire assay (see Table 13-2 above).

The metallurgical optimisation test work programme conducted using the master composite samples, included the following:

- Evaluation of preg-robbing.
- Benefits of High Shear pre-treatment with oxygen (DO>16 ppm).
- Effect of lead nitrate addition on recovery and residence time requirements.
- Leach residence time requirements.
- Lime and Cyanide addition requirements for leaching.
- Determination of the optimum grind size.
- Evaluation of recovery.
- E-GRG test work at Consep.
- Continuous Cyanide destruction and Arsenic removal testing.
- Cyanide destruction and Arsenic removal reagent addition requirements.
- Additional comminution test work.

The distribution of the optimisation phase metallurgical composite test sample drill holes are presented in Figure 13-2 below (as circled in red).

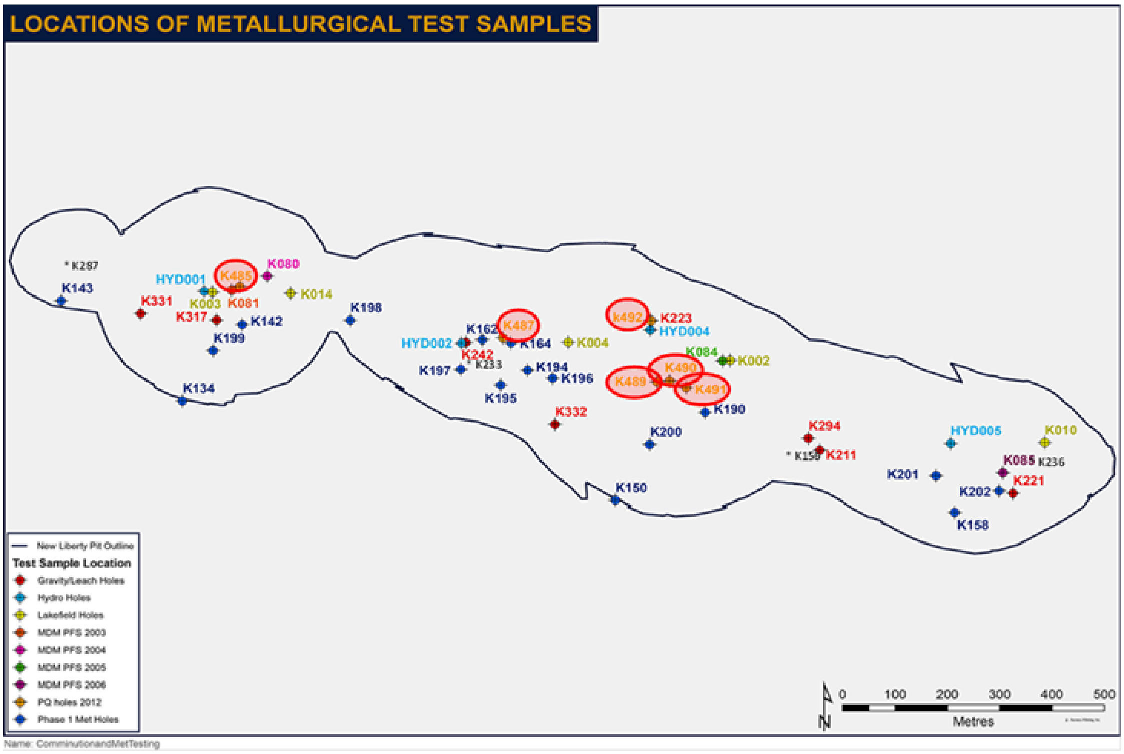



**Figure 13-2: Optimisation Phase Distribution of Composite Test Sample Drillholes**

### 13.1.5 Variability Samples

In addition to the test work conducted on the master composite sample, further variability test work was conducted using the optimised flowsheet and reagent consumptions. Seven (7) variability samples were prepared from the pre-crushed core samples as received from JKtech and four (4) additional ½ drill core samples which were delivered to ALS in December 2012. The four additional ½ drill core sample mass totalled 136 kg. The variability samples had a measured gold grade ranging from 4.26 g/t – 10.44 g/t. The ore variability samples represented various spatial locations distribution throughout the New Liberty deposit and included a sample from the eastern pit (K236).

The spatial distribution of the optimisation phase metallurgical variability test sample drill holes are presented in Figure 13-3 below (as circled in red).

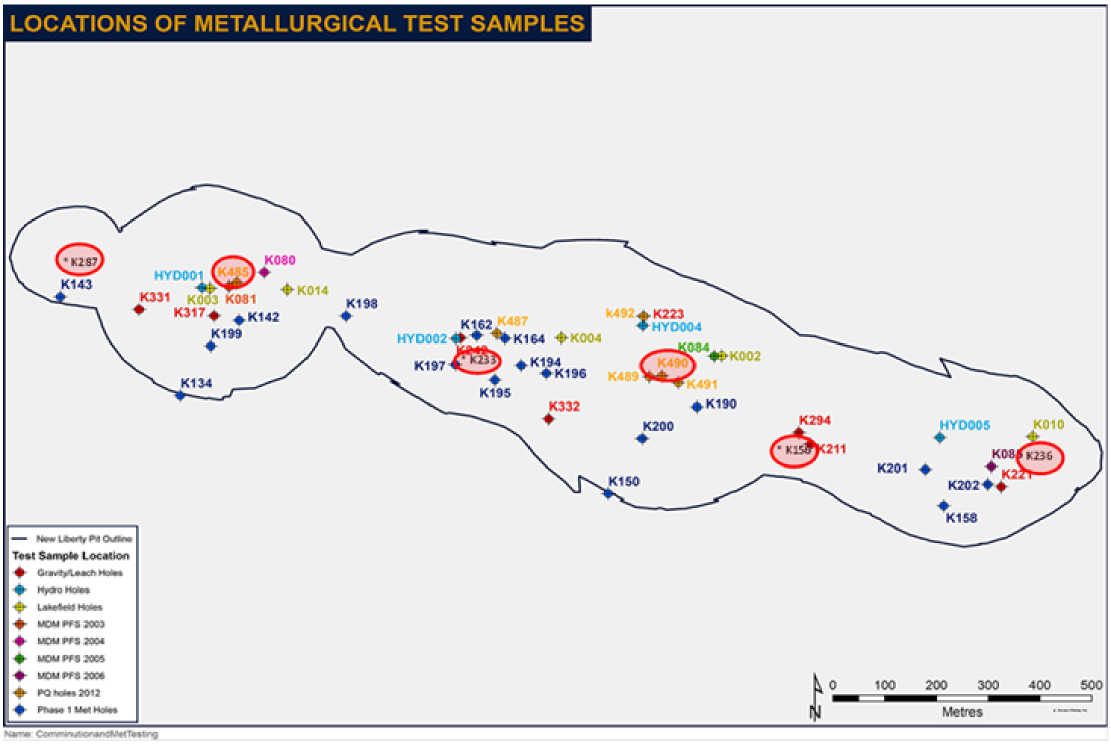



**Figure 13-3: Optimisation Phase Distribution of Variability Test Sample Drillholes**

## 13.2 Leach Optimisation Test Work on the Master Composite Sample

### 13.2.1 Introduction

Leach optimisation test work on the master composite was performed on gravity tailings samples. The test work conducted on the master composite sample was aimed at determining the objectives as listed in section 13.1.4 above. A summary and interpretation of the results from this phase of test work are presented below.

### 13.2.2 Evaluation of Preg-Robbing

Testing of preg-robbing can be assessed by a significant difference in the final residue gold assays when comparing conventional cyanidation, carbon-in-leach and carbon-in leach tests with diesel addition.

A Carbon in Leach (CIL) test was conducted on each composite at the target grind of 80% passing 75 µm. The comparison of the results of conventional cyanidation tests and carbon leach tests as presented in Table 13-3 and Figure 13-4 shows that the gold assays in the final residues for both methods were very similar for each composite. The similarity of the results suggests that there is no preg-robbing in these composites.

Based on the results of this evaluation, all further leach tests were conducted, with carbon addition at the start of the leach (0 hours).

**Table 13-3: Metallurgical Results for Optimisation Leach Tests to Evaluate Preg-Robbing**

| Test No | Feed | Test Conditions | Residue Au Grade (g/t) | Au Extraction (%) |
|---|---|---|---|---|
| JR133 | Master Composite | 1.5kg/t CN, 24 hrs. Add Carbon at 0 Hrs. | 0.49 | 88.34% |
| JR134 | Master Composite | 1.5kg/t CN, 24 hrs. Add Carbon at 9 Hrs. | 0.48 | 88.67% |
| JR135 | Master Composite | 1.5kg/t CN, 24 hrs. No Carbon | 0.49 | 88.42% |
| JR136 | Master Composite | 1.5kg/t CN, 24 hrs. Diesel and Carbon at 0 Hrs. | 0.49 | 88.12% |

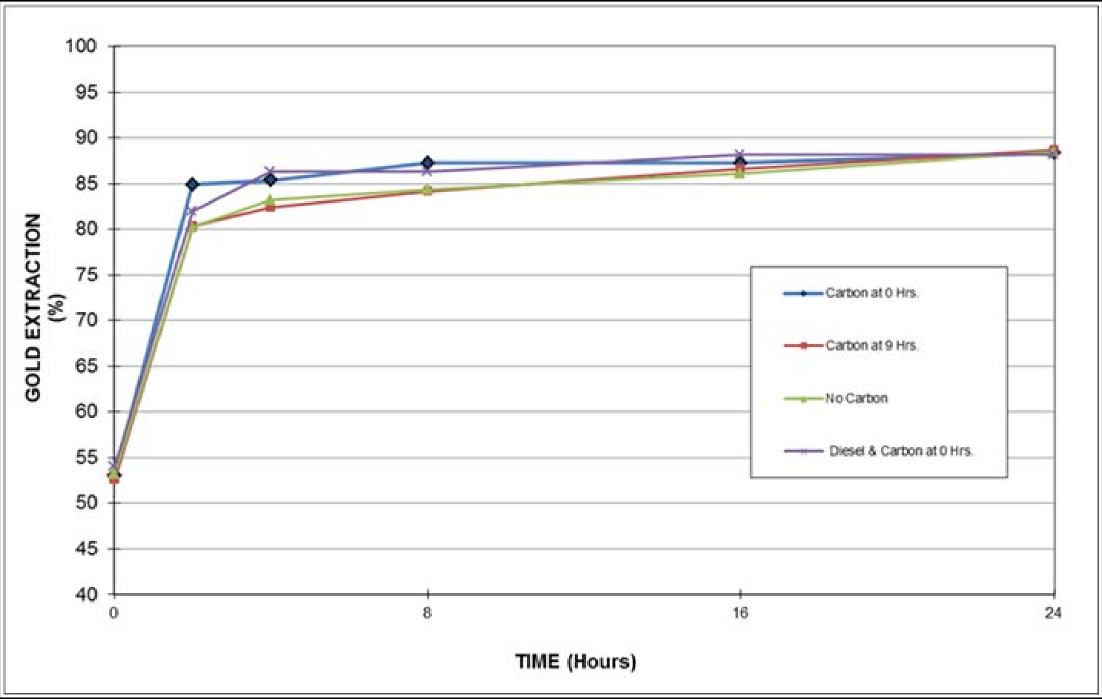



**Figure 13-4: Evaluation of preg-robbing for the New Liberty master composite sample at a grind of 80% passing 75 micron.**

### 13.2.3 Effect of high-shear, pre-treatment with oxygen (JR 177/183/184) in comparison to the feasibility flowsheet performance (JR 176).

The results of this work show that test JR184 achieved the highest gold recovery of 91.7% with 4 hours of high shear and 24 hours of CIL. The leach tails grades, however, were found to be 0.35 – 0.40 g/t for tests which included a high shear pre-oxidation step followed by 24 hours of CIL. This is in comparison to the leach tails grade of 0.53g/t which was achieved for the leach test conducted as per the feasibility flowsheet (JR 176) at 48 hours of CIL.

**Table 13-4: Metallurgical Results for Optimisation leach tests JR176, JR177, JR183 and JR184**

| Test No | Feed | Test Conditions | Calculated Au Head Grade (g/t) | Residue Au Grade (g/t) | Au Extraction (%) |
|---|---|---|---|---|---|
| JR 176 | Master Composite | Feasibility Flowsheet Validation, 48hr, 1.5kg/t CN, P80 | 4.28 | 0.53 | 87.63% |
| JR177 | Master Composite | Shear Base Case, 4 hrs High Shear Pre-Oxidation, 24 Hours, 1.5kg/t CN, P80 | 4.10 | 0.40 | 90.24% |
| JR 183 | Master Composite | 1.5kg/t CN, 50g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, P80 | 4.38 | 0.40 | 90.99% |
| JR 184 | Master Composite | 1.5kg/t CN, 250g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear & Diesel, P80 | 4.20 | 0.35 | 91.67% |

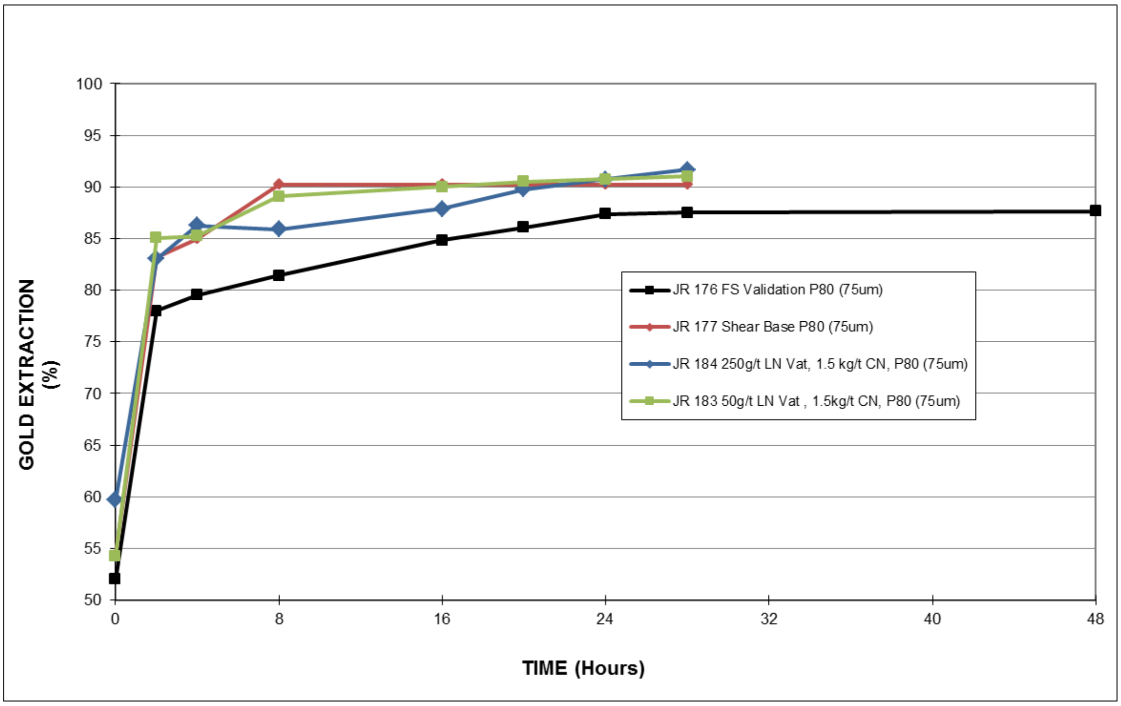



**Figure 13-5: Effect of high shear pre-treatment on gold recovery for the New Liberty master composite sample at a grind of 80% passing 75 micron.**

### 13.2.4 Optimisation of Cyanide Addition

The addition of a high shear, oxygen pre-treatment step increased the leach kinetics significantly and also resulted in improved overall gold extraction with a reduction in CIL residence time of 24 hours Optimisation of Cyanide Addition.

Once it was established that pre-treatment was required, the addition of cyanide was optimised by evaluating leach kinetics at cyanide addition of 1.5 kg/t, 1.0 kg/t and 0.5 kg/t.

Comparison of the leach kinetics for tests as presented in Table 13-5 and Figure 13-6 indicated that at a cyanide addition of 0.5kg/t (JR 178), similar recoveries and kinetics could be achieved as those obtained for test conducted at a cyanide addition rate of 1.0 kg/t ( JR 179) and 1.5 kg/t (JR177). The leach residue grades were in the range 0.40 g/t - 0.43g/t.

**Table 13-5: Metallurgical Results for Cyanide Optimisation Leach Tests JR177-179**

| Test No | Feed | Test Conditions | Calculated Au Head Grade (g/t) | Residue Au Grade (g/t) | Au Extraction (%) |
|---|---|---|---|---|---|
| JR177 | Master Composite | Shear Base Case, 4 hrs High Shear Pre-Oxidation, 24 Hours, 1.5kg/t CN, P80 75um, 50g/t Diesel | 4.10 | 0.40 | 90.24% |
| JR 178 | Master Composite | 0.5kg/t CN, 250g/t LN, 4hr. Pre-Ox, 24 Hr. CIL, P80 75 um, 50g/t Diesel | 4.06 | 0.41 | 89.90% |
| JR 179 | Master Composite | 1.0kg/t CN, 250g/t LN, 4hr. Pre-Ox, 24 Hr. CIL, P80 75um, 50g/t Diesel | 4.16 | 0.43 | 89.67% |

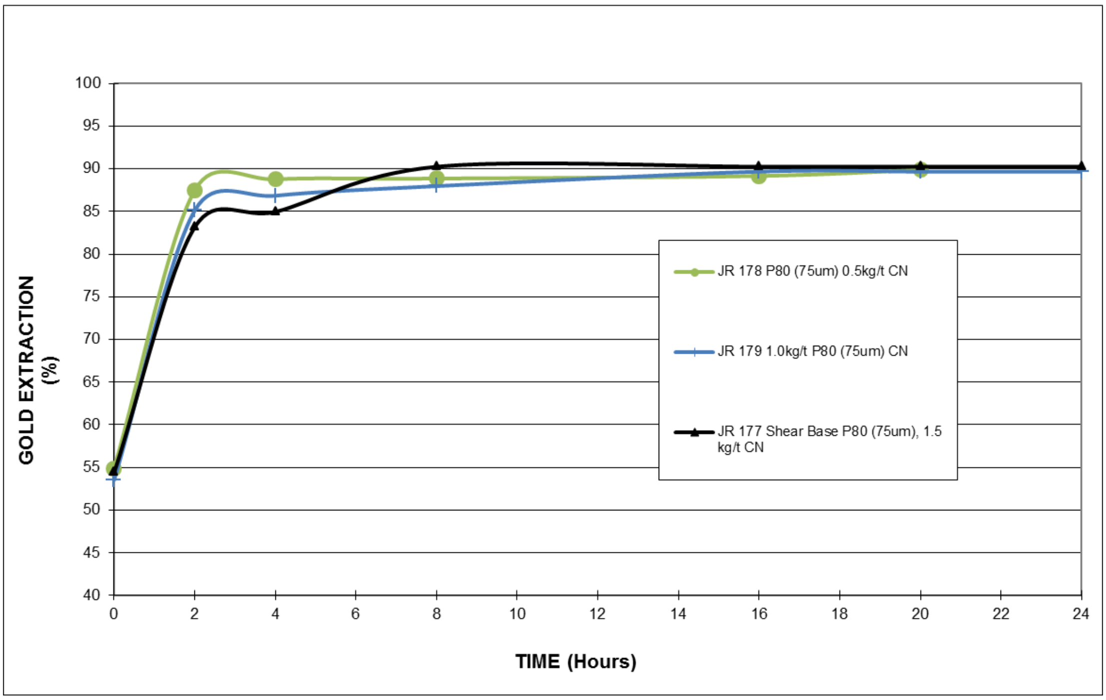



**Figure 13-6: Effect of cyanide addition on gold recovery for the New Liberty master composite sample at a grind of 80%passing 75 micron.**

### 13.2.5 Lead Nitrate Addition

The effect of lead nitrate addition was evaluated at dosage rates of 250 g/t, 150 g/t, 100 g/t and 50 g/t.

The results of these tests are presented in Table 13-6 and Figure 13-7. Comparison of the leach kinetics for test JR180- JR184, did not show any significant difference in final gold extraction for the lead nitrate addition in the range 50 g/t – 250 g/t. The gold content of leach residues was found to be 0.35 – 0.44 g/t. Based on the kinetic curves as presented in Figure 13-7 below a minimum lead nitrate addition rate of 25 g/t was selected on the basis that higher addition rates of lead nitrate did not provide an improvement in recovery or leach kinetics.

**Table 13-6: Metallurgical Results for Lead Nitrate Optimisation Leach Tests JR180 - JR184**

| Test No | Feed | Test Conditions | Calculated Au Head Grade (g/t) | Residue Au Grade (g/t) | Au Extraction (%) |
|---|---|---|---|---|---|
| JR 180 | Master Composite | 1.5 kg/t CN, 100g/t LN, 4 hr. Pre-Ox, 24 hr. BR, P80 75um | 4.14 | 0.41 | 90.09% |
| JR 181 | Master Composite | 1.5 kg/t CN, 150g/t LN, 4 hr. Pre-Ox, 24 hr. BR, P80 75 um | 4.14 | 0.43 | 89.62% |
| JR 182 | Master Composite | 1.5 kg/t CN, 250g/t LN, 4 hr. Pre-Ox, 24 hr. BR, P80 75 um | 4.26 | 0.44 | 89.67% |
| JR 183 | Master Composite | 1.5kg/t CN, 50g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, P80 75um | 4.38 | 0.40 | 90.99% |
| JR 184 | Master Composite | 1.5kg/t CN, 250g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear & Diesel, P80 75 um | 4.20 | 0.35 | 91.67% |

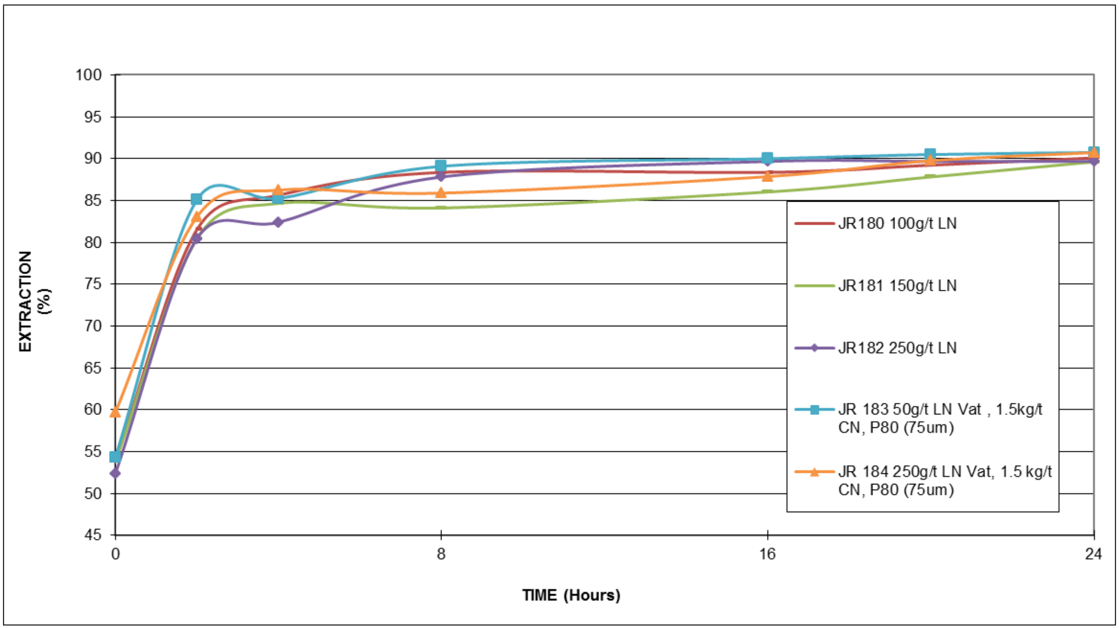



**Figure 13-7: Effect of lead nitrate addition on gold recovery for the New Liberty master composite sample at a grind of 80% passing 75 micron.**

### 13.2.6 Optimisation of Lime Addition

In the variability testing, it was noted that the lime consumption was excessive and would not prove economically feasible. The high lime consumption was attributed to the high target pH (in excess of 11) and the difficulties associated with measuring and maintaining such a high pH. It was thus decided to initiate lime optimisation tests, in which the pH control mechanism was adjusted as follows:

- Add lime to pH 11 prior to the pre-oxidation step
- Allow the pH to naturally decrease in the pre-oxidation and CIL process, with an allowance for lime addition should the pH drop to below 10.

A summary of the lime consumption for tests carried out in this manner is presented in Table 13-7 below. The lime consumption was found to be in the range 0.88 kg/t – 2.13 kg/t with an average consumption of 1.48 kg/t, which will be used as the optimised lime consumption.

**Table 13-7: Leach Tests Conducted at an Initial pH of 11, with Lime Addition to Maintain pH 10**

| Test No | Feed | Test Conditions | Residue Au Grade (g/t) | Au Extraction (%) | Lime Addition kg/t |
|---|---|---|---|---|---|
| JR 183 | Master Composite | 1.5kg/t CN, 50g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, P80 75um | 0.40 | 90.99% | 1.42 |
| JR 184 | Master Composite | 1.5kg/t CN, 250g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear & Diesel, P80 75 um | 0.35 | 91.67% | 2.13 |
| JR 298 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear, P95 75 um | 0.31 | 92.86% | 1.83 |
| JR 344 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um | 0.31 | 92.46% | 0.88 |
| JR 345 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL , Shear, P95 75 um | 0.29 | 92.91% | 1.14 |

### 13.2.7 Determination of Optimum Grind

The test work programme conducted to determine the optimum leach conditions was based on a target grind size of 80% passing 75 micron. Once the results of these tests became available it was apparent that the test work recovery for the master composite sample using the optimised leach conditions was in the range 89.7% - 91.7%. This was lower than the target recovery of 93%.

It was thus decided to do further testing in order to determine if there was an improvement in recovery at finer target grind sizes.

The metallurgical results of all the vat leach tests conducted at grind sizes of approximately 80% passing 74 µm, 60µm, 42µm and 50µm are presented in Table 13-8 below.

**Table 13-8: Metallurgical Results for leach tests on master composite samples to determine optimum target grind size.**

| Test No | Feed | Test Conditions | Grind P80 (µm) | Calculated HG Au Grade (g/t) | Residue Au Grade (g/t) | Au Extraction (%) |
|---|---|---|---|---|---|---|
| JR 183 | Master Composite | 1.5kg/t CN, 50g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, P80 75um | 74 | 4.38 | 0.40 | 90.99% |
| JR 184 | Master Composite | 1.5kg/t CN, 250g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear & Diesel, P80 75 um | 74 | 4.20 | 0.35 | 91.67% |
| JR 202 | Master Composite | 1.5kg/t CN, 250g/t LN 4hr. Pre-Ox, 24hr. Bulk Vat Leach MidShear, P90 75um, 15g/L C | 60 | 4.22 | 0.36 | 91.47% |
| JR 203 | Master Composite | 1.5kg/t CN, 250g/t LN 4hr. Pre-Ox, 24hr. Bulk Vat Leach MidShear, P90 75um, 20g/L C | 60 | 4.07 | 0.34 | 91.65% |
| JR 204 | Master Composite | 1.5kg/t CN, 250g/t LN 4hr. Pre-Ox, 24hr. Bulk Vat Leach MidShear, P90 75um, 30g/L C | 60 | 4.15 | 0.36 | 91.33% |
| JR264 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C | 43 | 4.30 | 0.24 | 94.42% |
| JR265 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C | 42 | 4.27 | 0.23 | 94.61% |
| JR266 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C | 42 | 4.27 | 0.24 | 94.38% |
| JR 206 | Master Composite | 0.5kg/t CN, 0g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um | 41 | 4.25 | 0.27 | 93.65% |
| JR 207 | Master Composite | 0.5kg/t CN, 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um | 41 | 4.28 | 0.26 | 94.04% |
| JR 208 | Master Composite | 0.5kg/t CN, 50g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um | 41 | 4.33 | 0.27 | 93.76% |
| JR285 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um Scalped, 15g/l C | 41 | 4.28 | 0.27 | 93.81% |
| JR 298 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear, P95 75 um | 48 | 4.34 | 0.31 | 92.86% |
| JR 344 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um | 51 | 4.11 | 0.31 | 92.46% |
| JR 345 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL , Shear, P95 75 um | 51 | 4.09 | 0.29 | 92.91% |

The results indicated the following:

- At a target grind size of 80% passing 75µm residue grades of 0.35 g/t-0.40 g/t were achieved.
- At a target grind size of 80% passing 60µm residue grades of 0.34 g/t-0.36 g/t were achieved.
- At a target grind size of 80% passing 50µm residue grades of 0.29 g/t-0.31 g/t were achieved.
- At a target grind size of 80% passing 42µm residue grades of 0.23 g/t-0.27 g/t were achieved.

As can be seen from Table 13-8, the average CIL recovery was found to increase at finer grind with an overall recovery increase of 2.8% when the fineness of grind increased from 80% passing 75 µm to 80% passing 42µm. Based on the results presented in Table 13-8 and the leach curves as presented in Figure 13-8 it was apparent that the recovery benefit was significant when the fineness of grind was increased to 80% passing 42 µm.

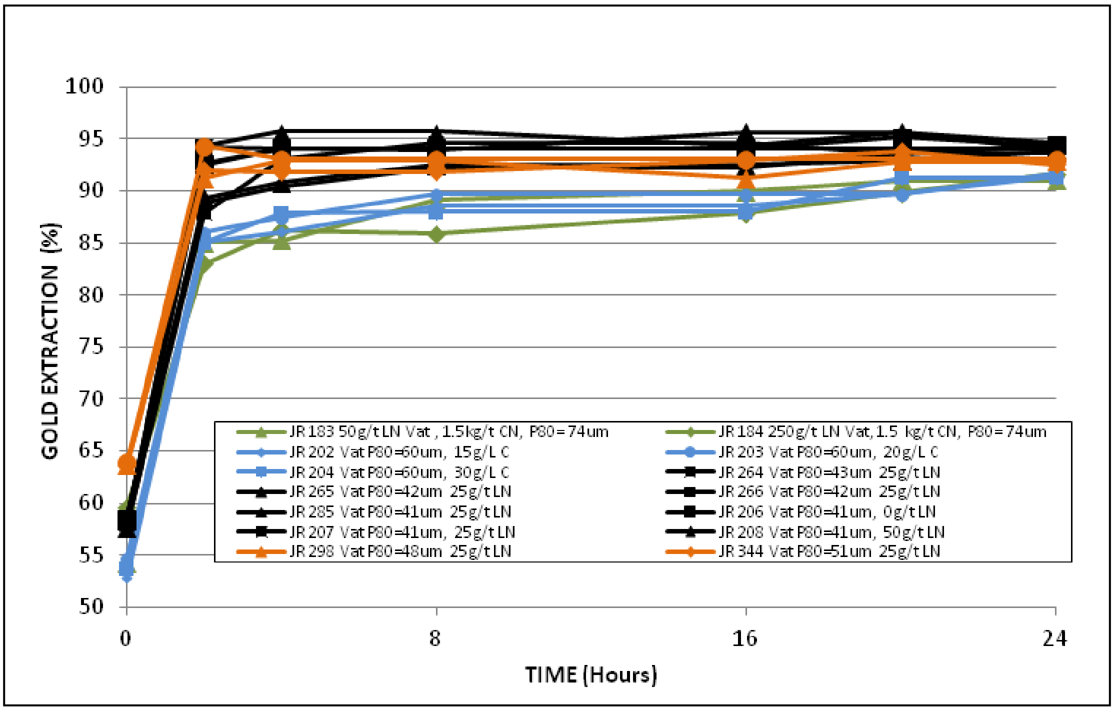



**Figure 13-8: Effect of Target Grind Size on Gold Recovery for the New Liberty Master Composite Sample**

The full particle size distributions for each target grind size are presented in Table 13-9 below, with the corresponding size distribution curves as presented in Figure 13-9.

**Table 13-9: Particle Size distributions for tests conducted on the master composite at various target grind sizes.**

| Test No | JR183/184 | JR203/204 | JR206/7/8 | JR 264 | JR 265 | JR 266 | JR 285 | JR 298 | JR 344/345 |
|---|---|---|---|---|---|---|---|---|---|
| Size (um) | | | | | | | | | |
| | | | | | | | | | |
| 106 | 96.0 | 96.7 | | | | | 99.6 | 99.0 | 99.9 |
| 90 | 88.6 | 94.0 | | | | | 99.1 | 97.0 | 99.6 |
| 75 | 80.8 | 89.0 | 97.7 | 96.3 | 96.5 | 96.5 | 97.7 | 94.6 | 97.9 |
| 63 | 72.0 | 82.6 | 94.9 | 93.3 | 93.8 | 93.7 | 95.3 | 90.2 | 92.7 |
| 53 | 61.8 | 74.3 | 90.5 | 88.5 | 89.7 | 89.6 | 91.2 | 84.5 | 82.6 |
| 45 | 50.0 | 63.3 | 84.9 | 82.5 | 83.3 | 83.6 | 84.8 | 76.8 | 73.7 |
| 38 | 44.8 | 53.0 | 75.6 | 71.3 | 74.2 | 74.6 | 75.4 | 65.9 | 62.0 |
| 25 | 36.8 | 42.4 | 62.4 | 59.6 | 60.3 | 64.2 | 64.6 | 54.6 | 48.2 |
| P80 | 73.9 | 59.9 | 41.3 | 43.4 | 42.5 | 42.2 | 41.4 | 48.3 | 50.7 |

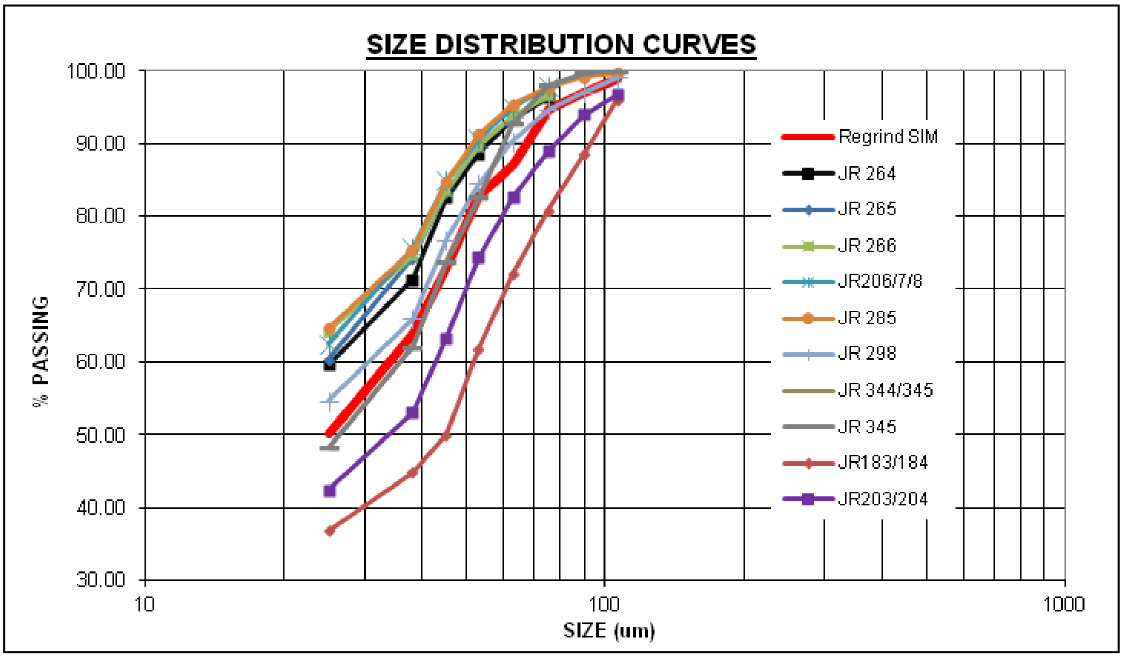



**Figure 13-9: Size Distribution Curves for the Grind Optimization Tests**

A series of scalped tests, were conducted using a re-grind test procedure that did not involve grinding all material to achieve a target grind. In this test the feed material was milled to a target grind of 80% passing 75 µm, the plus 75 µm fraction was then screened out and milled to achieve an overall grind of 80% passing 50 µm. It was felt that this procedure would provide a more accurate reflection of a plant scale re-grind application and would prevent generation of a large fines fraction. The particle size distribution achieved for the scalped test was found to have a 10-12% less in the minus 25 µm size fraction, than tests in which all the material was milled.

## 13.3 Additional Grinding Test Work

Additional grinding test work was undertaken to establish a work index for the master composite and determine power requirements for a regrind milling application.

*Determination of Bond Work Index*

The results of the Bond Work Index Test conducted on the master composite sample are presented in Table 13-10 below.

**Table 13-10: Results of the Bond Work Index Test on the Master Composite**

| Composite ID | Test Aperture Pi (µm) | Bond Ball Mill Work Index (kWh/t) |
|---|---|---|
| **Master** | 106 | 18.8 |

*Levin and IsaMill Testing*

Once it had been established that a finer mill grind size resulted in improved, recovery additional grinding test work was initiated. The initial target mesh of grind of 80% minus 75 micron was the starting point from which subsequent additional milling test work was conducted and was aimed at determining specific energy requirements for a regrind application and comprised of the following:

- Levin tests
- IsaMill tests

Levin tests are conducted in the standard Bond laboratory mill with standard ball charge, using an 'implied power' by assuming so many kW per revolution. This may then be converted to "applied" kWh/t; knowing the charge mass and number of revolutions used. The product size distributions are determined for a number of progressive milling tests.

An IsaMill signature plot is obtained from a standard series of multi-pass milling tests in a laboratory scale mill using bead media of known size distribution (5~6 mmØ), and measured power.

DRA had asked for the signature plot product particle size distributions to be determined in full, instead of just reporting product P80 so that a comparison could be made between conventional milling and a regrind application, could be determined by simulation. It is perceived that there will be different milling efficiencies, as a consequence of using different media, and power application. Very little information or literature is available regarding this comparison, apart from the knowledge of Metso using a de-rate of the Bond Work Index (0.65) for Vertimill applications. It was crucial to ascertain and benchmark specific energy requirements for New Liberty ore, given this degree of perceived variation.

*Results of Levin Tests*

**Table 13-11: Levin Test Results**

| PARTICLE SIZE DISTRIBUTION DETERMINATION | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| FEED | | | | Energy input kWh/t : 8 | | | | Energy input kWh/t : 18 | | | | Energy input kWh/t : 25 | | | |
| | | | | (µm) | 5.6 | Min | | (µm) | 12.6 | Min | | (µm) | 17.5 | Min | |
| | | | | | **393** | Rev | | | **884** | Rev | | | **1228** | Rev | |
| **(µm)** | **(g)** | **(%)** | **% <** | | **(g)** | **(%)** | **% <** | | **(g)** | **(%)** | **% <** | | **(g)** | **(%)** | **% <** |
| 250 | 1.9 | 0.19 | 99.81 | 250 | 0.5 | 0.05 | 99.95 | 250 | 1.0 | 0.10 | 99.90 | 250 | 0.4 | 0.04 | 99.96 |
| 212 | 3.1 | 0.31 | 99.50 | 212 | 1.3 | 0.13 | 99.82 | 212 | 1.7 | 0.17 | 99.73 | 212 | 1.2 | 0.12 | 99.84 |
| 180 | 6.3 | 0.63 | 98.87 | 180 | 4.2 | 0.42 | 99.40 | 180 | 2.7 | 0.27 | 99.46 | 180 | 1.9 | 0.19 | 99.65 |
| 150 | 9.1 | 0.91 | 97.96 | 150 | 5.8 | 0.58 | 98.82 | 150 | 5.1 | 0.51 | 98.95 | 150 | 3.8 | 0.38 | 99.27 |
| 125 | 15.9 | 1.59 | 96.37 | 125 | 11.2 | 1.12 | 97.70 | 125 | 9.8 | 0.98 | 97.97 | 125 | 7.7 | 0.77 | 98.50 |
| 106 | 38.8 | 3.88 | 92.49 | 106 | 17.2 | 1.72 | 95.98 | 106 | 12.4 | 1.24 | 96.73 | 106 | 9.8 | 0.98 | 97.52 |
| 75 | 123.5 | 12.35 | 80.14 | 75 | 72.8 | 7.28 | 88.70 | 75 | 48.9 | 4.89 | 91.84 | 75 | 36.2 | 3.62 | 93.90 |
| 53 | 182.6 | 18.26 | 61.88 | 53 | 137.3 | 13.73 | 74.97 | 53 | 114.5 | 11.45 | 80.39 | 53 | 99.6 | 9.96 | 83.94 |
| 45 | 95.4 | 9.54 | 52.34 | 45 | 76.1 | 7.61 | 67.36 | 45 | 70.9 | 7.09 | 73.30 | 45 | 52.7 | 5.27 | 78.67 |
| 38 | 79.4 | 7.94 | 44.40 | 38 | 87.3 | 8.73 | 58.63 | 38 | 85.6 | 8.56 | 64.74 | 38 | 80.9 | 8.09 | 70.58 |
| 25 | 92.6 | 9.26 | 35.14 | 25 | 96.2 | 9.62 | 49.01 | 25 | 97.2 | 9.72 | 55.02 | 25 | 94.1 | 9.41 | 61.17 |
| -25 | 351.4 | 35.14 | 0.00 | -25 | 490.1 | 49.01 | 0.00 | -25 | 550.2 | 55.02 | 0.00 | -25 | 611.7 | 61.17 | 0.00 |
| Total | 1000.0 | 100.00 | | | 1000.0 | 100.00 | | | 1000.0 | 100.00 | | | 1000.0 | 100.00 | |
| Calc'd P80 | ***75*** | | | | ***61*** | | | | ***53*** | | | | ***47*** | | |

*Results of IsaMill Tests*

**Table 13-12: Results of IsaMill Test Conducted 30 kWh/t**

| Calculated Data | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Pass # | Gross kW | Net kW | Q (m3/h) | % Solids | M (t/h) | E (kWh/t) | Cumul. E | P80 | P98 | CSI | |
| Feed | | | | | | | | 74.4 | | | |
| 1 | 1.47 | 0.73 | 0.189 | 48.2% | 0.132 | 5.5 | 5.5 | | | | |
| 2 | 1.67 | 0.93 | 0.171 | 47.4% | 0.118 | 7.9 | 13.4 | 44.7 | 100.5 | 2.2 | |
| 3 | 1.65 | 0.91 | 0.180 | 48.1% | 0.126 | 7.3 | 20.6 | 37.9 | 98.3 | 2.6 | |
| 4 | 1.63 | 0.89 | 0.180 | 48.1% | 0.126 | 7.1 | 27.7 | 27.9 | 89.1 | 3.2 | |
| 5 | 1.63 | 0.89 | 0.180 | 47.9% | 0.125 | 7.1 | 34.8 | 23.8 | 78.6 | 3.3 | |
| 6 | 1.61 | 0.87 | 0.180 | 48.1% | 0.126 | 6.9 | 41.7 | 22.0 | 74.0 | 3.4 | |
| 7 | 1.60 | 0.86 | 0.180 | 48.1% | 0.126 | 6.8 | 48.5 | 20.4 | 67.1 | 3.3 | |
| Target P80 Size (if applic.): | | 27 | kWh/t @ Target: | | 30.0 | Media Consumption (g/kWh): | | | | | 10 |

**Table 13-13: Results of IsaMill Test Conducted 40 kWh/t**

| Calculated Data | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Pass # | Gross kW | Net kW | Q (m3/h) | % Solids | M (t/h) | E (kWh/t) | Cumul. E | P80 | P98 | CSI | |
| Feed | | | | | | | | 74.4 | | | |
| 1 | 1.47 | 0.73 | 0.189 | 48.2% | 0.132 | 5.5 | 5.5 | | | | |
| 2 | 1.67 | 0.93 | 0.171 | 47.4% | 0.118 | 7.9 | 13.4 | 44.7 | 100.5 | 2.2 | |
| 3 | 1.65 | 0.91 | 0.180 | 48.1% | 0.126 | 7.3 | 20.6 | 37.9 | 98.3 | 2.6 | |
| 4 | 1.63 | 0.89 | 0.180 | 48.1% | 0.126 | 7.1 | 27.7 | 27.9 | 89.1 | 3.2 | |
| 5 | 1.63 | 0.89 | 0.180 | 47.9% | 0.125 | 7.1 | 34.8 | 23.8 | 78.6 | 3.3 | |
| 6 | 1.61 | 0.87 | 0.180 | 48.1% | 0.126 | 6.9 | 41.7 | 22.0 | 74.0 | 3.4 | |
| 7 | 1.60 | 0.86 | 0.180 | 48.1% | 0.126 | 6.8 | 48.5 | 20.4 | 67.1 | 3.3 | |
| Target P80 Size (if applic.): | | 23 | kWh/t @ Target: | | 40.0 | Media Consumption (g/kWh): | | | | | 10 |

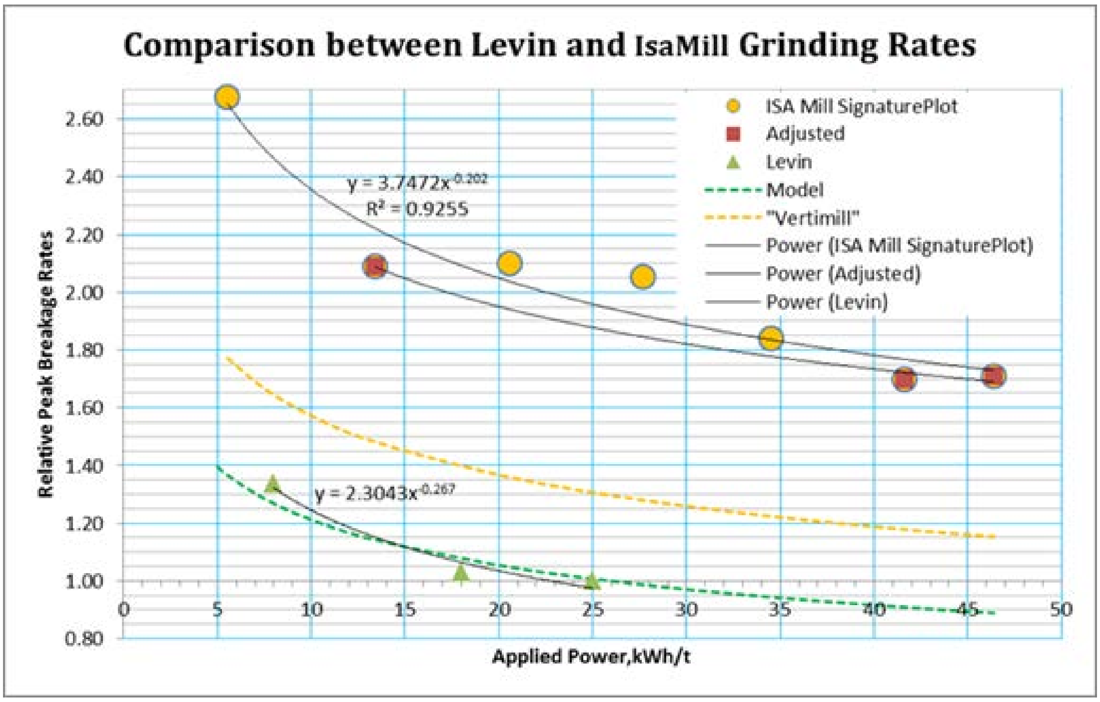



**Figure 13-10: New Liberty Summary of Levin and IsaMill Test Work Result**

Simulation of the Levin and IsaMill test work has shown significant improvement in milling efficiencies as compared to conventional mills. Figure 13-10 shows the relative difference in breakage rates (energy-based) on New Liberty regrind material. It can be seen that there are significant efficiency gains by 'matching media size' to feed size distribution. The Vertimill (using media 12.7 mm) estimated relative grinding rates have been shown in Figure 13-10.

The relative grinding rates derived from test work were used to estimate the performance of a VertiMill unit, this has indicated that to achieve the target grind of 80% passing 50 µm the energy requirement is 6.7 kWh/t.

## 13.4 Evaluation of Leach Feed Density

### 13.4.1 Introduction

At the end of the optimisation phase two confirmatory flowsheet validation tests were performed to evaluate the effect of CIL feed density on overall gold extraction. The results of these tests are presented in Table 13-14 below.

**Table 13-14: Evaluation of Overall Gold Extraction as a Function of Leach Feed Density**

| Test No | Feed | Test Conditions | Residue Au Grade (g/t) | Au Extraction (%) | NaCN Addition kg/t | Lime Addition kg/t |
|---|---|---|---|---|---|---|
| JR 344 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um | 0.31 | 92.46% | 0.50 | 0.88 |
| JR 345 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL , Shear, P95 75 um | 0.29 | 92.91% | 0.50 | 1.14 |

The 24 hour CIL test which was conducted at a feed density of 45% solids produced a final residue grade of 0.31g/t which was similar to the 0.29 g/t achieved for the 20 hour CIL test which was conducted at a feed density of 40% solids.

### 13.4.2 Diagnostic Leach Tests

Multi-stage sequential diagnostic gold leach test work was conducted on two 1 kilogram subsamples of the mater composite sample. The results of these diagnostic leaches are presented in Table 13-15 below.

**Table 13-15: Results of the Diagnostic Leach Tests Conducted on the Master Composite**

| Diagnostic Stage | Description | Master Composite | | | |
|---|---|---|---|---|---|
| | | $P_{80}$:75 μm | | $P_{90}$:75 μm | |
| | | Au Distribution | | | |
| | | (%) | (g/t) | (%) | (g/t) |
| 1 | Gravity Recoverable/Cyanidable Gold Content | 94.63 | 4.24 | 91.76 | 3.49 |
| 2/3 | Carbonate Locked Gold Content | 0.80 | 0.04 | 0.56 | 0.02 |
| 4/5 | Arsenical Mineral (Arsenopyrite) Locked Gold Content | 2.04 | 0.09 | 1.47 | 0.06 |
| 6/7 | Fine Disseminated Locked Gold (<20μm) Content | - | - | 5.51 | 0.21 |
| 8 (6) | Pyritic Sulphide and Iron Oxides Gold Content | 1.38 | 0.06 | 0.39 | 0.01 |
| 9 (7) | Silicate (Gangue) Encapsulated Gold Content | 1.14 | 0.05 | 0.30 | 0.01 |
| TOTAL GOLD CONTENT | | 100.00 | 4.48 | 100.00 | 3.81 |
| SCREEN FIRE ASSAY | | - | 4.23 | - | 4.23 |

The diagnostic leach tests indicated that in excess of 90% of the gold is free-milling and recoverable by cyanidation. The addition of a regrind step on the P90 sample indicated that an additional 5.5% of the gold can be liberated for cyanidation with the addition of a regrind step at a target grind of 80% passing 20 μm.

### 13.4.3 Whole Ore CIL Test

In order to provide an indication of the expected plant recovery without the gravity concentration circuit in operation a whole ore CIL test was performed at a target grind of 80% passing 50 µm. The original master composite sample had been depleted so a new composite was generated using the same material that was used to generate the original master composite sample. The results of this whole ore CIL test are presented in Table 13-16 below.

**Table 13-16: Results of the Whole Ore CIL Test**

| Test No | Feed | Test Conditions | Calculated HG Au Grade (g/t) | Residue Au Grade (g/t) | Au Extraction (%) | NaCN Addition kg/t | Lime Addition kg/t |
|---|---|---|---|---|---|---|---|
| JR 346 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um, No Gravity Concetration prior to CIL | 4.19 | 1.13 | 73.03% | 0.59 | 0.97 |

The feed material was assayed four times to obtain an indicated head grade of 3.62 g/t – 6.51 g/t. The calculated head grade was determined to be 4.19 g/t. The large amount of variation in the assayed head grade and discrepancy between the assayed and calculated head grade is indicative of the presence of coarse gold. The leach residue grade for this test was 1.13 g/t with an overall extraction of 73.0%. The decrease in recovery was attributed to the presence of coarse gold particles that were not removed by gravity concentration and illustrates the importance of the gravity recovery circuit.

Panning of the whole ore CIL residue showed in excess of 29 gold grains. The photograph below shows one of the largest flakes found in the concentrate (total 1.8 g out of ~47.0 g feed). The flake reaches 200 µm or 0.2 mm.

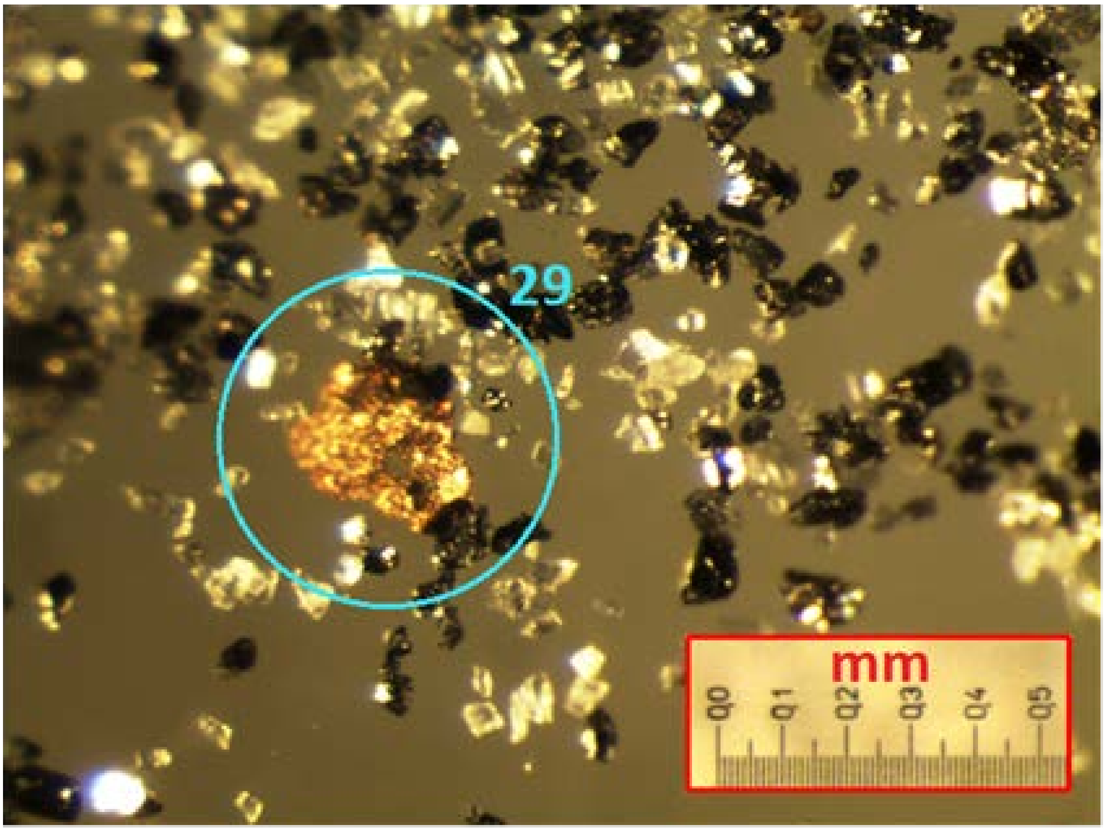



**Figure 13-11: Coarse Gold Flake in Gravity Concentrate**

## 13.5 Variability Test Work

### 13.5.1 Introduction

Gravity concentration and leach test work was performed at ALS on the variability samples collected from the deposit. The gravity test work was conducted at a target grind of 80% passing 75 µm the gravity tailings were then subjected to target grinds of 80% passing 50 µm and 80% passing <20 µm before CIL. At the time of making the decision to target a final mill grind of 80% passing 45 µm, the variability testing had already been completed and there was insufficient sample left to do further variability testing at this target grind.

The following optimised leach conditions were used:

- 4 hours of high shear pre-oxidation, followed by 24 hour CIL
- 25 g/t Lead Nitrate addition
- 15 g/L Carbon
- 0.5 kg/t Sodium Cyanide addition, with further incremental addition to maintain a solution cyanide concentration of 100 ppm up to 16 hours.
- Leach pH controlled at 11

During the variability test work, the presence of coarse nugget gold was evident from the comparison of the measured head assays (as presented in Table 13-2) and calculated head assays. For the purpose of determining the test recovery the calculated head grades were used, as this is the most accurate reflection of how plant feed grade and recovery will be determined for full scale plant operations. It is also a better calculation of recovery because it's based on more reliable product weights and assays.

### 13.5.2 Variability Test Work Results at Target Grind of 80% Passing 50 µm

The results of the variability tests conducted at a target grind of 80% passing 50µm (Refer P90 in ALS report) are presented in Figure 13-12 and Table 13-17 below.

**Table 13-17: Metallurgical results for Variability Testing Conducted at Grind of 80% Passing 50 Microns**

| Test No | Sampl ID | Test Conditions | Calculated HG Au Grade (g/t) | Residue Au Grade (g/t) | Gravity Extraction (%) | CIL Extraction (%) | Overall Au Extraction (%) | NaCN Addition (g/t) | Lime Addition (g/t) |
|---|---|---|---|---|---|---|---|---|---|
| JR 279 | K156 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 5.57 | 0.36 | 63.93% | 82.33% | 93.63% | 0.61 | 4.87 |
| JR 280 | K233 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 5.44 | 0.36 | 72.48% | 76.29% | 93.47% | 0.66 | 15.91 |
| JR 281 | K236 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 4.47 | 0.71 | 39.89% | 73.58% | 83.33% | 0.61 | 5.89 |
| JR 282 | K287 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 4.24 | 0.34 | 72.84% | 70.48% | 91.98% | 0.51 | 7.40 |
| JR 283 | K485B | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 7.66 | 0.22 | 78.98% | 86.65% | 97.19% | 0.58 | 4.55 |
| JR 284 | K492 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 3.69 | 0.30 | 72.22% | 71.22% | 92.01% | 0.54 | 13.43 |

The actual grind for the variability samples varied from 80% passing 47µm-71µm.The overall gold recovery for the six variability composite samples ranged from 83.3% - 97.2%. The overall recovery was comprised of the gravity circuit recovery and the cyanide carbon in leach (CIL) recovery. Gravity recovery ranged from 39.9% -79.0%, while the CIL recovery ranged from 70.5% -86.7%. The leach residue grades achieved ranged from 0.22 g/t to 0.71 g/t.

The residue grade of 0.71 g/t was achieved for test JR281 which was conducted on material from the eastern pit (K236). This residue grade was noted as being an outlier when compared to the variability testing conducted at P80 50µm. Further to this the result of this test was compared to previous Mintek phase I variability testing conducted at P80 75µm on material from the eastern pit which again confirmed this test as an outlier as can be seen in Table 13-18 below:

**Table 13-18: Comparison of Variability Test Results for Material from the Eastern Pit**

| Test No | Feed | Test Conditions | Target Grind | P80 | Calculated Head Grade (g/t) | Residue Au Grade (g/t) | Au Extraction (%) |
|---|---|---|---|---|---|---|---|
| Phase 1 | K201 | Whole Ore CIL | P80 | 75 | 4.33 | 0.19 | 95.61% |
| Phase 1 | K202 | Whole Ore CIL | P80 | 75 | 1.92 | 0.22 | 88.54% |
| JR281 | K236 | 25g/t LN, 4hr. Pre-Ox, 24hr. | P90 | 47 | 4.47 | 0.71 | 84.12% |

The average gravity tailings leach cyanide and lime addition rates were 0.59kg/t and 8.7/t respectively. It was noted that the lime consumption was excessive and would not prove economically feasible. The high lime consumption was attributed to the high target pH (in excess of 11) and the difficulties associated with measuring and maintaining such a high pH. It was thus decided to initiate lime optimisation tests, in which the pH control mechanism was adjusted as follows:

- Add lime to pH 11 prior to the pre-oxidation step
- Allow the pH to naturally decrease in the pre-oxidation and CIL process, with an allowance for lime addition should the pH drop to below 10.

The results of the lime optimisation testing are presented in Section 13.2.6.

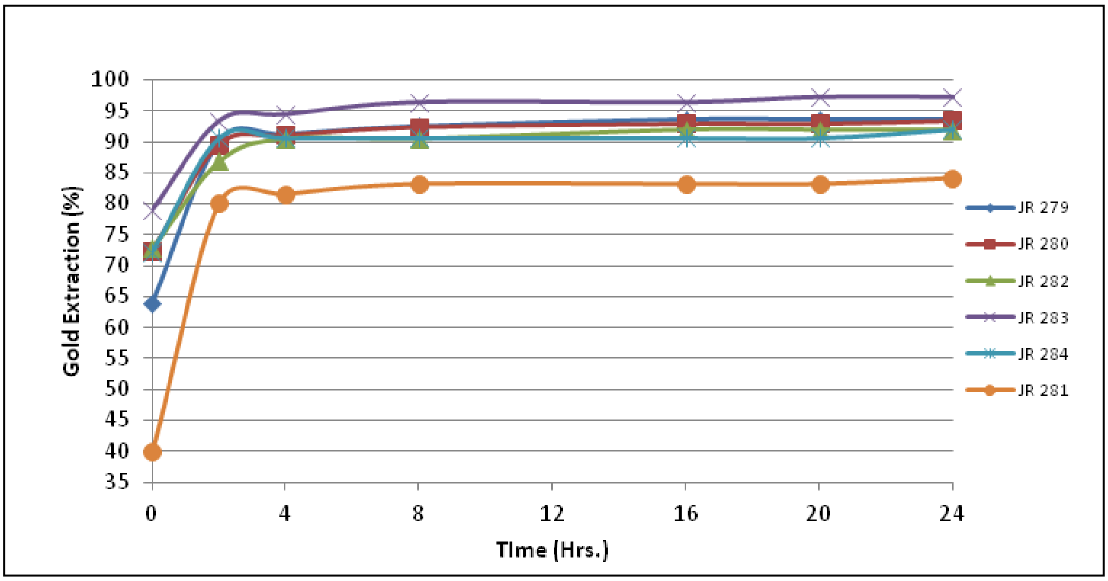



**Figure 13-12: Gold Recovery for New Liberty Variability Tests Conducted at a Target Grind Size of 80% Passing 50 µm**

In order to establish the feasibility of achieving the recoveries as determined by the variability test work conducted at a target grind of 80% passing 50 µm the size distribution curves for the feed material to each test were plotted. These results where compared to the simulated 80% passing 50 µm PSD for a regrind application based on the results of Levin test work conducted at ALS. These curves are presented in Figure 13-13 below. It is apparent that the size distributions for these variability tests can be replicated in a full scale regrind application, as can be seen by comparing the simulated PSD to the actual PSD.

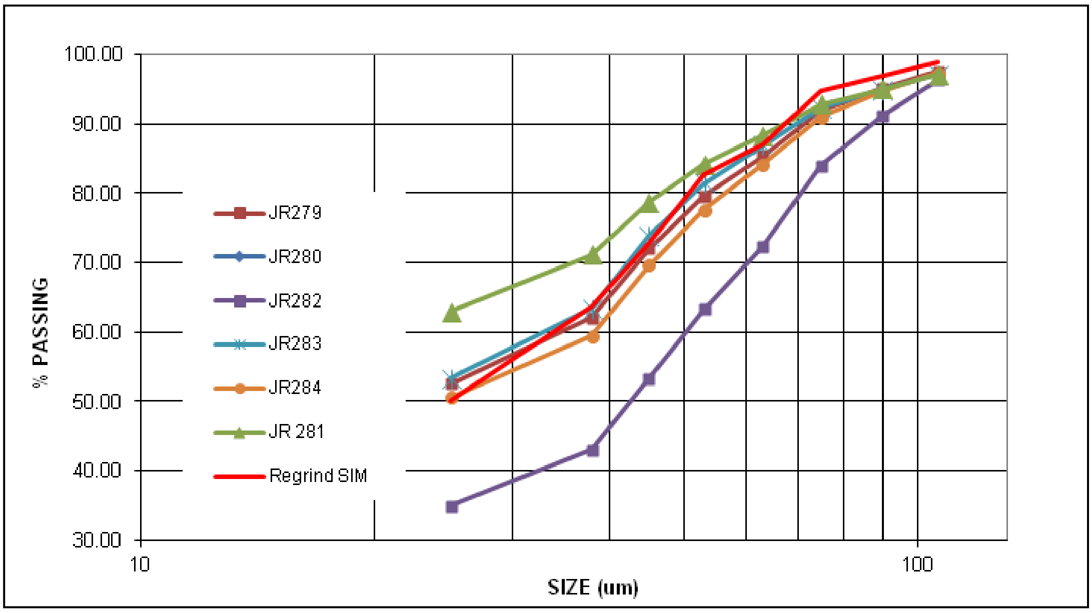



**Figure 13-13: Size Distribution Curves for Variability Tests Conducted at a Target Grind of 80% Passing 50 µm**

### 13.5.3 Variability Test Work Results at a Target Grind of 80% Passing 25 µm

The results of the variability tests conducted at a target grind of 80% passing <20 µm (Refer P100 in ALS report) are presented in Figure 13-14 and Table 13-19 below.

**Table 13-19: Metallurgical results for Variability Testing Conducted at a Target Grind of 80% Passing 20 Micron**

| Test No | Sampl ID | Test Conditions | Calculated HG Au Grade (g/t) | Residue Au Grade (g/t) | Gravity Extraction (%) | CIL Extraction (%) | Overall Au Extraction (%) | NaCN Addition (g/t) | Lime Addition (g/t) |
|---|---|---|---|---|---|---|---|---|---|
| JR245 | K287 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 4.72 | 0.21 | 77.77% | 79.99% | 95.55% | 0.50 | 7.12 |
| JR246 | K490 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 2.24 | 0.11 | 73.44% | 81.51% | 95.09% | 0.83 | 6.74 |
| JR247 | K492 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 4.59 | 0.14 | 81.42% | 83.58% | 96.95% | 0.92 | 15.92 |
| JR248 | K233 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 7.07 | 0.20 | 79.31% | 86.33% | 97.17% | 0.63 | 5.00 |
| JR249 | K236 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 4.15 | 0.27 | 37.66% | 89.56% | 93.49% | 0.75 | 7.89 |
| JR250 | K485B | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 9.43 | 0.11 | 59.38% | 97.13% | 98.83% | 0.64 | 5.32 |
| JR251 | K156 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 5.77 | 0.12 | 76.57% | 91.12% | 97.92% | 1.92 | 5.50 |

The actual grind for the variability samples varied from 80% passing 31µm to an estimated 15 µm for the remainder of the report this grind is referred to as 80% passing <20 µm. The overall gold recovery for the seven variability composite samples ranged from 93.5% - 98.8%. The overall recovery was comprised of the gravity circuit recovery and the cyanide carbon in leach (CIL) recovery. Gravity recovery ranged from 37.7% -81.4%, while the CIL recovery ranged from 80.0% -92.1%. The leach residue grades achieved ranged from 0.11 g/t to 0.27 g/t. The average gravity tailings leach cyanide and lime addition rates were 0.88 kg/t and 7.6 kg/t respectively. As previously noted the lime consumption was excessive and would not prove economically feasible.

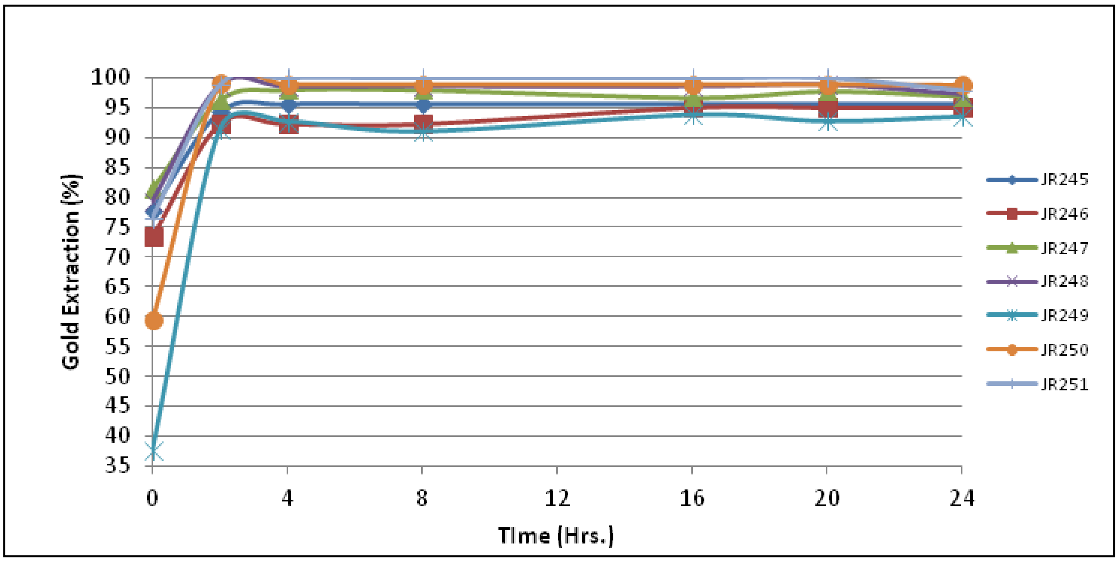



**Figure 13-14: Gold Recovery for the New Liberty Variability Samples at a Target Grind Size of 80% Passing 25 Micron**

In order to establish the feasibility of achieving the recoveries as determined by the variability test work conducted at a target grind of 80% passing <20 µm (P100) the size distribution curves for the feed material to each test were plotted. These results where compared to the simulated 80% passing 50 µm PSD for a regrind application based on the results of Levin test work conducted at ALS. These curves are presented in Figure 13-15 below. It was apparent that the size distributions for these variability tests would prove difficult to replicate in a full scale regrind application, due to the large proportion of fines present with more than 80% passing 20 µm as compared to the simulated 50% passing 25 µm. For this reason these recoveries will prove difficult to replicate as this grind almost represents pulverisation.

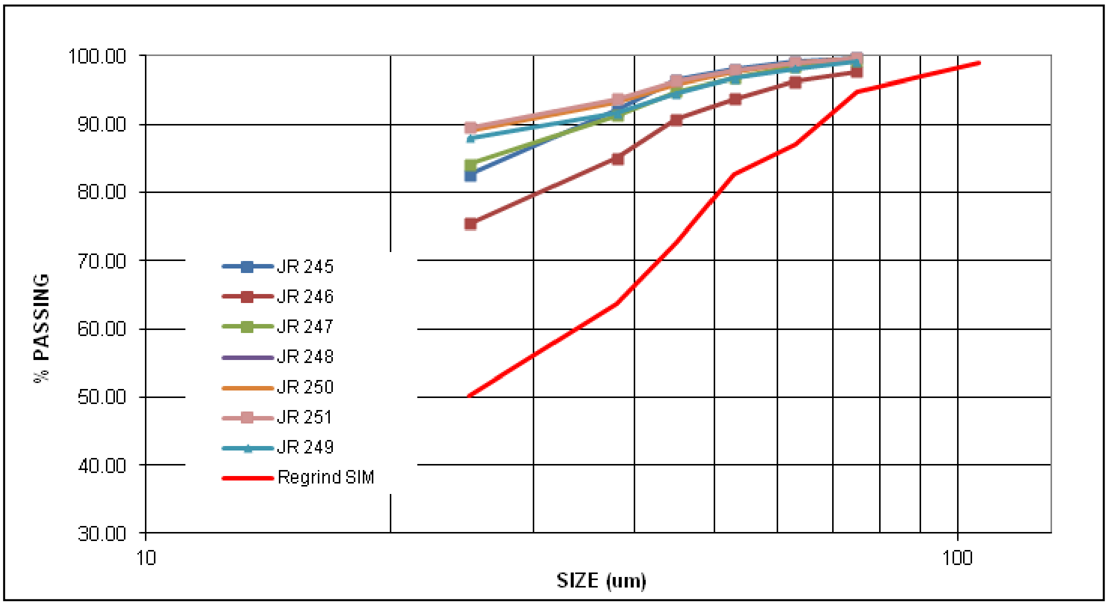



**Figure 13-15: Size Distribution Curves for the Variability Tests Conducted at 100% Passing 75 µm**

## 13.6 Selection of Mill Grind at 80% Passing 45 Microns

Based on the indicated recovery improvement as the mill target grind size was increased and results of the variability testing conducted, it was decided to target a mill grind of 80% passing 45 µm.

At the time of making the decision to target a mill grind of 80% passing 45 µm, the variability testing had already been completed and there was insufficient sample left to do further variability testing at this target grind. In order to obtain a better estimate of recovery at the target grind further tests were conducted on the master composite sample.

The simulated performance for the VertiMill with a power input of 6.7 kWh/t has been plotted relative to the PSD's achieved for all the tests conducted at an approximate grind of 80% passing 42 µm in Figure 13-16 below:

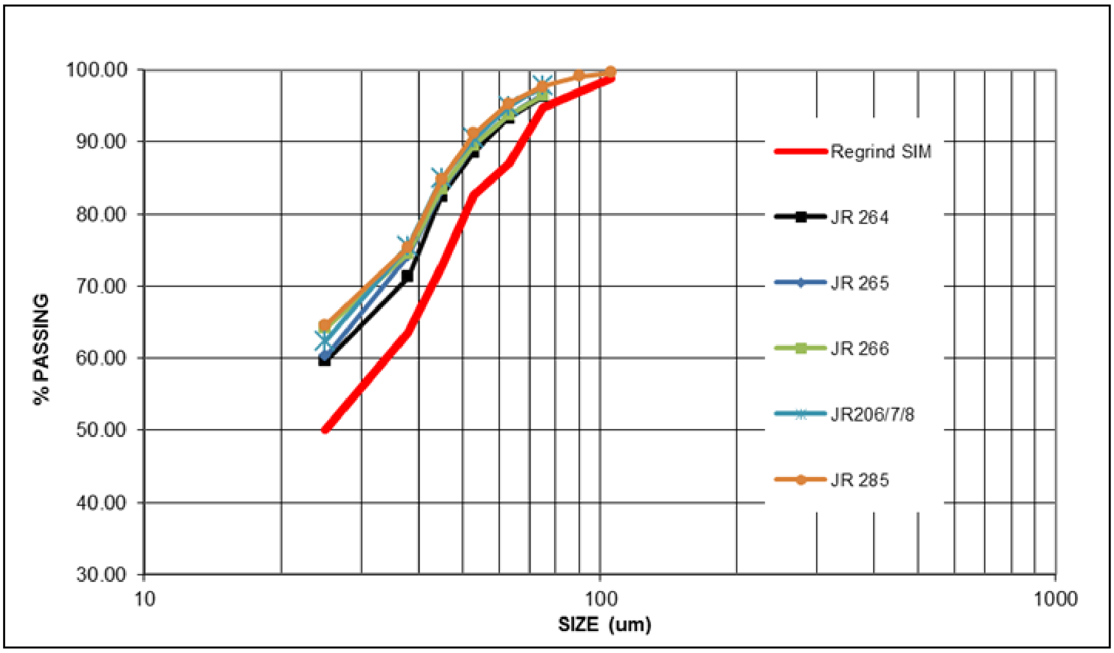



**Figure 13-16: Size distribution Curves for the Composite Tests Conducted at 80% Passing 42µm**

The simulated performance for the VertiMill with a power input of 6.7 kWh/t has been plotted relative to the PSD's achieved for all the tests conducted at an approximate grind of 80% passing 50 µm in Figure 13-17 below:

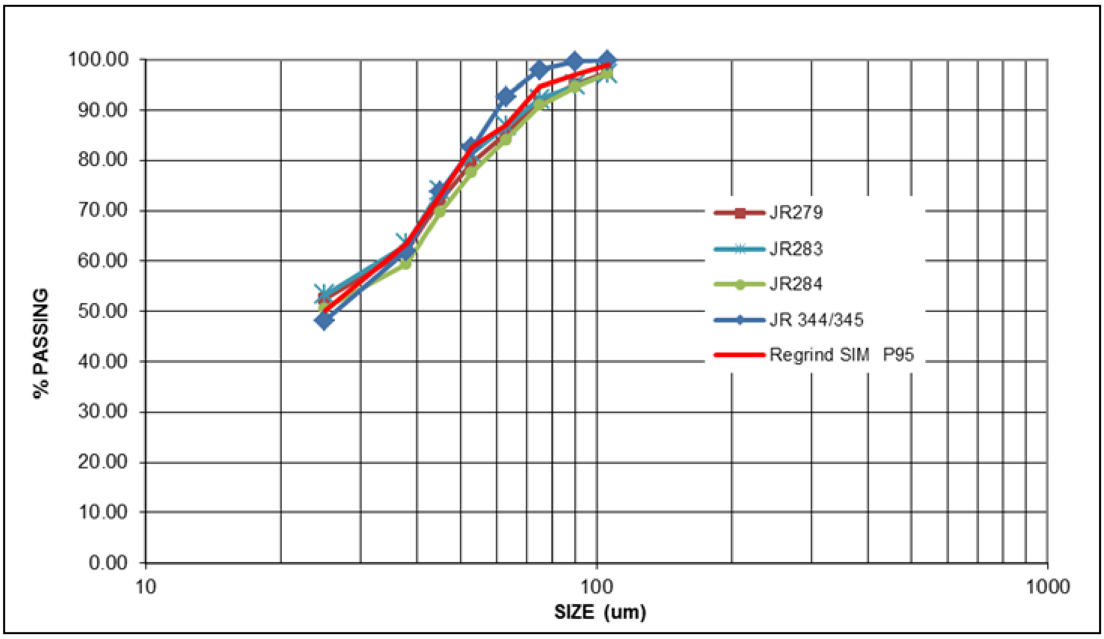



**Figure 13-17: Size Distribution Curves for the Variability and Composite Testing at a Grind of 80% Passing 50 Micron**

The graph above illustrates that these PSD's were similar to the simulated 80% passing 50 µm PSD's for a full scale Vertimill application.

Testing conducted at 80% passing 42µm is presented in Table 13-20, while testing conducted at 80% passing 50µm is presented in Table 13-21. A comparison of the two show a recovery differential of 0.60% between the average recoveries achieved, which is attributed to the finer grind with a higher percentage of 25 micron material in tests conducted at 80% passing 50 micron.

The diagnostic leach tests as presented in Table 13-15 show that, there is 5.51% fine disseminated gold locked in the minus 20 µm size fraction, which supports liberation at finer grind sizes and this would have to be achieved with a Vertimill.

**Table 13-20: Metallurgical Results for Composite Testing Conducted at a Grind of 80% Passing 42 Micron**

| Test No | Feed | Test Conditions | P80 | Calculated HG Au Grade (g/t) | Residue Au Grade (g/t) | Au Extraction Test Work (%) |
|---|---|---|---|---|---|---|
| JR 206 | Master Composite | 0.5kg/t CN, 0g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um | 41 | 4.25 | 0.27 | 93.65% |
| JR 207 | Master Composite | 0.5kg/t CN, 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um | 41 | 4.28 | 0.26 | 94.04% |
| JR 208 | Master Composite | 0.5kg/t CN, 50g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um | 41 | 4.33 | 0.27 | 93.76% |
| JR 264 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C | 43 | 4.30 | 0.24 | 94.42% |
| JR 265 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C | 42 | 4.27 | 0.23 | 94.61% |
| JR 266 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C | 42 | 4.27 | 0.24 | 94.38% |
| JR 285 | Master Composite | 0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um Scalped, 15g/l C | 41 | 4.28 | 0.27 | 93.81% |
| Average | | | | | | 94.11% |

**Table 13-21: Metallurgical Results for Variability and Composite Testing at a Grind of 80% Passing 50 Micron**

| Test No | Feed | Test Conditions | P80 | Calculated HG Au Grade (g/t) | Residue Au Grade (g/t) | Au Extraction Test Work (%) |
|---|---|---|---|---|---|---|
| JR 298 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear, P95 75 um | 48 | 4.34 | 0.31 | 92.86% |
| JR 344 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um | 51 | 4.11 | 0.31 | 92.46% |
| JR 345 | Master Composite | 0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL , Shear, P95 75 um | 51 | 4.09 | 0.29 | 92.91% |
| JR 279 | K156 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 54 | 5.57 | 0.36 | 93.63% |
| JR 283 | K485B | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 52 | 7.66 | 0.22 | 97.19% |
| JR 284 | K492 | 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11 | 57 | 3.69 | 0.30 | 92.01% |
| Average | | | | | | 93.51% |

It is noted that the selected target grind of 80% passing 45 µm would provide a PSD finer than for the tests at 80% passing 50 µm and thus the expected gold extraction at this target grind will more closely represent that for tests conducted at a target grind of 80% passing 42 µm.

## 13.7 Cyanide Destruction Test Work

### 13.7.1 $SO_2$/Air Cyanide Destruction Test Work

An initial series of $SO_2$/Air cyanide destruction test work was conducted on the product of bulk leach tests conducted on the master composite at a cyanide addition of 1.5 kg/t. These initial tests are presented in Table 13-22 below and indicated that at a CNwad level in the leach effluent stream of 162.8 ppm an $SO_2$: CN ratio of 4:1 was not sufficient to reduce CNwad levels in the cyanide destruction product stream to below 50 ppm.

**Table 13-22: Results of the Initial Scoping Tests Conducted for the $SO_2$/Air Cyanide Destruction Process**

| $SO_2$/AIR CYANIDE DETOXIFICATION RESULTS (TESTS D1 to D6) | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Test No. | Test Conditions | | | | | Solution Assays | | |
| | pH | Retention Time (minutes) | Reagents Used | | | Feed Effluent $CN_P$ (mg/l) | Treated Effluent $CN_P$ (mg/l) | Treated Effluent $CN_{TOT}$ (mg/l) |
| | | | $SO_2$ (g/g $CN_{WAD}$) | $Cu^{2+}$ (mg/l) | Lime (g/g $SO_2$) | | | |
| JR183 Leach Residue Slurry | | | | | | | | |
| D1 | 9.71 | 58.55 | 1.92 | 79 | 0.00 | 162.8 | 100.4 | 106.3 |
| D2 | 8.84 | 58.30 | 2.90 | 79 | 0.24 | 162.8 | 67.4 | 68.7 |
| D3 | 8.57 | 56.92 | 4.00 | 79 | 1.12 | 162.8 | 56.4 | 56.5 |
| D4 | 9.56 | 57.88 | 4.16 | 0 | 0.89 | 162.8 | 97.3 | 99.2 |
| D5 | 9.52 | 56.95 | 4.16 | 44 | 1.04 | 162.8 | 78.5 | 82.1 |
| D6 | 8.90 | 112.90 | 4.09 | 42 | 0.53 | 162.8 | 77.5 | 77.6 |

Once the optimum leach conditions had been established and a lower cyanide addition of 0.5 kg/t was determined to be optimum, a new $SO_2$/Air cyanide destruction test was conducted on the product of leach test JR298. The test as presented in Table 13-23 indicated that at a CNwad level in the leach effluent stream of 70.6 ppm an $SO_2$: CN ratio of 4:1 was sufficient to reduce CNwad levels in the cyanide destruction product stream to 10 ppm which is well below the 50 ppm target that is specified by the international cyanide code of practice.

**Table 13-23: Results of the $SO_2$/Air Cyanide destruction Tests Conducted for the on Leach Effluent Generated Using the Optimized Leach Conditions**

| $SO_2$/AIR CYANIDE DETOXIFICATION RESULTS (TEST D8) | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Test No. | Test Conditions | | | | | Solution Assays | | |
| | pH | Retention Time (minutes) | Reagents Used | | | Feed Effluent $CN_P$ (mg/l) | Treated Effluent $CN_P$ (mg/l) | Treated Effluent $CN_{TOT}$ (mg/l) |
| | | | $SO_2$ (g/g $CN_{WAD}$) | $Cu^{2+}$ (mg/l) | Lime (g/g $SO_2$) | | | |
| JR298 Leach Residue Slurry | | | | | | | | |
| D8 | 8.56 | 58.54 | 4.96 | 0 | 0.00 | 70.6 | 10.3 | 11.1 |

### 13.7.2 Hybrid SO2/Air Cyanide Destruction Test Work

An initial series of hybrid $SO_2$/Air (Includes a carbon contact stage) cyanide destruction test work was conducted on the product of bulk leach tests conducted on the master composite at a cyanide addition of 0.5 kg/t. These initial tests as presented in Table 13-24 indicated that for a 3 stage continuous test at a CNwad level in the leach effluent stream of 88.1ppm a reagent suite as indicated in the table below, was sufficient to reduce CNwad levels in the cyanide destruction product stream to below 2.5 ppm.

**Table 13-24: Results of the initial scoping tests conducted for the hybrid $SO_2$/air cyanide destruction process (conducted on leach effluent generated using the optimized leach conditions).**

| GOLD ORE HYBRID CYANIDE DETOXIFICATION RESULTS (TESTS H3 AND H4) | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Test No. | Test Conditions | | | | | Solution Assays | | |
| | Final pH | SMBS (kg/t) | Reagents Used | | | Feed Effluent $CN_P$ (mg/l) | Treated Effluent $CN_P$ (mg/l) | Treated Effluent $CN_{TOT}$ (mg/l) |
| | | | $CuSO_4.5H_2O$ (kg/t) | $FeCl_3$ (kg/t) | HCl (as 100%) (kg/t) | | | |
| JR229 Leach Residue Slurry | | | | | | | | |
| H3 | 5.97 | 0.53 | 0.28 | 0.27 | 4.44 | 88.1 | <0.2 | 0.24 |
| H4 | 5.83 | 0.57 | 0.25 | 0.26 | 3.72 | 88.1 | 2.50 | 2.68 |

Further to this a series of 4 stage batch hybrid $SO_2$/Air cyanide destruction tests were conducted on the leach effluent stream produced in each of the variability tests. The results of these tests are presented in Table 13-25 below. As can be seen from the results, low CNWad levels were achieved in all tests. It is worth noting that the feed material for these tests was produced from leaches conducted at a cyanide addition of 0.5 kg/t making these results incomparable to the results obtained for the initial series of $SO_2$/Air tests as presented in Table 13-22. These results are comparable to the results of the $SO_2$/Air test presented in Table 13-23.

**Table 13-25: Results of the Hybrid $SO_2$/Air Cyanide Destruction Tests Conducted on the Effluent Stream from the Leach Variability Tests**

| GOLD ORE HYBRID CYANIDE DETOXIFICATION VARIABILITY TEST RESULTS | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Test No. Stage No. | Leach Residue Slurry | Sample ID | Slurry pH | $H_2SO_4$ (as 100%) kg/t | Lime (as 60% CaO) kg/t | Feed Effluent CNp ppm | Treated Effluent CNp ppm | Treated Effluent $CN_{Tot}$ ppm |
| Variability tests with nominally 550g/t SMBS, 300g/t copper pentahydrate, 300g/t ferric chloride and 500g/t HCl (as 100% HCl equivalent) | | | | | | | | |
| **H5 - S3** | JR245 | K287 | 6.63 | 7.41 | 0 | 126.2 | 1.01 | 2.41 |
| **H5 - S4** | | | 9.00 | 0 | 0.51 | 1.01 | 0.99 | 2.39 |
| **H6 - S3** | JR246 | K490 | 6.95 | 8.90 | 0 | 204.2 | 0.99 | 2.39 |
| **H6 - S4** | | | 9.00 | 0 | 0.53 | 0.99 | 0.89 | 2.29 |
| **H7 - S3** | JR247 | K492 | 6.71 | 17.99 | 0 | 139.4 | 0.82 | 2.22 |
| **H7 - S4** | | | 9.00 | 0 | 0.76 | 0.82 | 0.67 | 2.07 |
| **H8 - S3** | JR248 | K233 | 6.28 | 6.31 | 0 | 72.2 | 0.47 | 1.87 |
| **H8 - S4** | | | 9.00 | 0 | 0.66 | 0.47 | 0.94 | 2.34 |
| **H9 - S3** | JR249 | K236 | 6.68 | 9.55 | 0 | 77.0 | 0.39 | 1.79 |
| **H9 - S4** | | | 9.00 | 0 | 1.35 | 0.39 | 1.61 | 3.01 |
| **H10 - S3** | JR250 | K485B | 6.95 | 8.97 | 0 | 104.4 | 0.62 | 2.02 |
| **H10 - S4** | | | 9.00 | 0 | 0.91 | 0.62 | 2.48 | 3.88 |
| **H11 - S3** | JR251 | K156 | 7.05 | 10.60 | 0 | 221.0 | 1.49 | 2.89 |
| **H11 - S4** | | | 9.00 | 0 | 0.81 | 1.49 | 4.44 | 7.24 |

The $SO_2$/Air process was selected as the basis of design for the New Liberty cyanide destruction circuit. For a leach product CNwad level of 70 ppm this process was found to produce a CNwad level in the effluent stream of less than 10 ppm. Based on the optimized leach conditions the leach effluent stream is expected to have a CNwad level of 50 ppm -100 ppm and based on the results of this test work, the SO2/Air process will be able to produce a cyanide destruction effluent stream CNwad level of less than 50 ppm as per the requirements of the international cyanide code of practice.

The $SO_2$/Air process does not use a shear reactor in the pre-oxidation phase and does not require high concentrations of activated carbon for cyanide removal. This makes the process less complex than the hybrid process and it has a lower associated operating cost. The reason for pursuing the hybrid $SO_2$/Air detox testing was due to the initial indication of the possibility for additional gold extraction, which was never quantified in the test work.

## 13.8 Arsenic Precipitation Tests

Arsenic precipitation test work for New Liberty consisted of initial exploratory testing which used ferric chloride to remediate arsenic on a solids sample containing 1000ppm arsenic. The solids content of 1000ppm was indicated to be representative of the average arsenic level for the orebody.

These initial scouting tests included evaluation of the following:

- A forced arsenic leach and precipitation of a stable ferric arsenate compound prior to CIL using 1kg/t ferric chloride addition, followed by a further 3.0kg/t ferric chloride addition in week 3 of the kinetic column in test JR 746. The total ferric chloride addition was 4.0kg/t for this test.
- CIL followed by a CCD simulation followed by precipitation of a stable ferric arsenate compound with 1kg/t ferric chloride addition in test JR 747.
- CIL followed by the $SO_2$/Air detox process with no further arsenic remediation in test JR 748.
- No further treatment of CIL product slurry (base case)in test JR 749.

The product stream from each of these tests was sent to a kinetic column in which solution arsenic levels were monitored over a 48 week period.

This test work showed that the conditions of test JR 746 with 4kg/t ferric chloride addition resulted in successful leaching of arsenic followed by precipitation of a stable ferric arsenate compound. The initial 1kg/t ferric chloride addition prior to CIL proved unsuccessful, however, the further addition of 3.0kg/t in week 3 of the kinetic column test resulted in a reduction in arsenic levels to below 0.1ppm.

Test JR 746 resulted in the lowest arsenic levels within the first 5 weeks and arsenic levels were found to remain below 0.1ppm between week 5 and week 35 of the column test. After week 35 soluble arsenic levels increased above the 0.1ppm target level, this was addressed as part of a further optimisation phase aimed at determining optimum reagent addition requirements.

The results of these initial scouting tests are presented graphically in Figure 13-18.

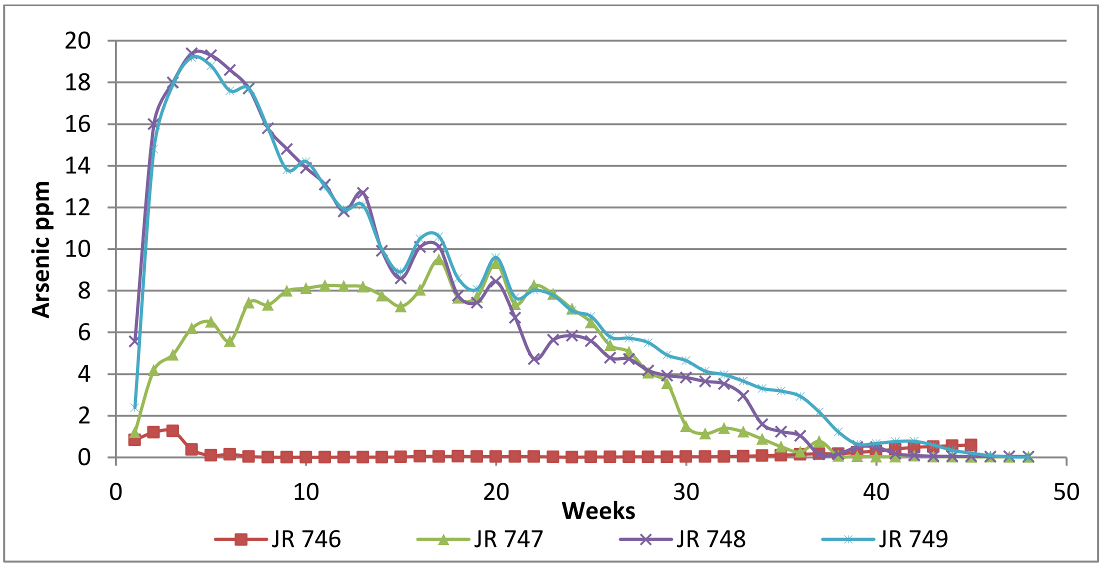



**Figure 13-18: Arsenic levels during column tests on arsenic remediation test product slurry for scouting tests JR 746-JR 749**

A further series of ferric chloride optimisation tests were initiated on gold leach product slurry from the sample containing 1000ppm arsenic. These tests were conducted using a hybrid of the conditions for test JR746 and JR 748. In these tests, the CIL feed slurry was treated with ferric sulphate for a forced arsenic leach and precipitation of a stable ferric arsenate compound at ferric chloride addition rates of 1.7kg/t – 4.2kg/t prior to CIL. The CIL product was then treated with $SO_2$/Air for cyanide detox. The product stream from each of these tests was sent to a kinetic column in which solution arsenic levels were monitored over a 48 week period.

In this test work an addition rate of 4.2kg/t in test JR 860 achieved the target arsenic level of 0.1ppm in the kinetic column within 5 weeks. At an addition rate of 2.9kg/t (JR 857) the target arsenic level of 0.1ppm was reached at 14 weeks which was comparable to the 12 weeks achieved for test JR 859 with an addition rate of 3.9kg/t. Test JR 855, JR 856 and JR 858 with an addition rate of 2.2, 1.7 and 3.2kg/t respectively achieved the target level after 30 weeks.

The results of these ferric chloride optimisation tests are presented graphically in Figure 13-19.

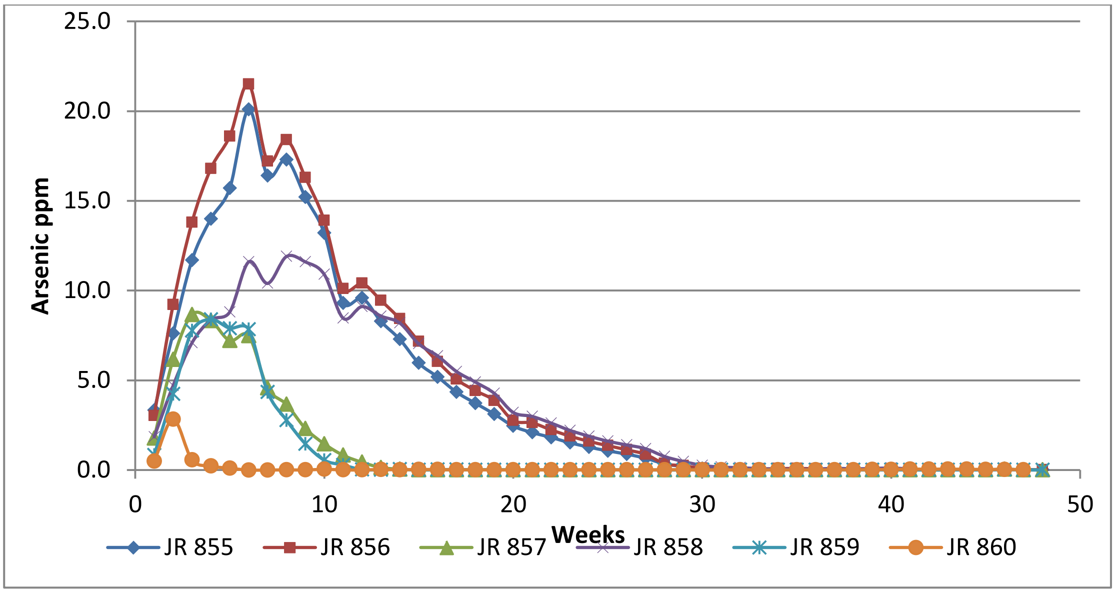



**Figure 13-19: Arsenic levels during column tests on arsenic remediation test slurry for tests JR 855 – JR 860**

The results of test JR 857 were promising and the similar result to test JR 859 indicated that a ferric chloride addition rate of less than 3.0kg/t was able to produce a stable arsenate compound. The result of test JR 858 with an addition rate of 3.2kg/t was inconsistent with the results achieved at addition rates of 2.9kg/t (JR 857) and 3.9kg/t (JR 859). Test JR 858 only achieved an arsenic level of less than 0.1ppm after week 33 of the kinetic column run which was comparable to tests JR 856 (1.7kg/t) and JR 855 (2.20kg/t). The reason for this is unclear.

The positive results at a total ferric chloride addition rate of 2.9kg/t indicated that there was opportunity to further reduce operating costs and the time taken to achieve a kinetic column arsenic level of less than 0.1ppm. At this point, a review of the test work flowsheet revealed that the use of a forced arsenic leach prior to CIL followed by $SO_2$/air detox on CIL tailings was resulting in a large number of pH variation during the process, resulting in the requirement to have four pH adjustments. A review of the process highlighted the opportunity to try and combine the forced arsenic leach and the detox process steps which have similar pH requirements. It was thus decided to focus further optimisation testing on a flowsheet that allowed for arsenic leaching and precipitation post CIL in combination with the detox process.

A further series of test work was then initiated on gold leach product slurry from a new sample which contained 500ppm arsenic in solids. In this testing the $SO_2$/Air detox was coupled with ferric chloride addition for forced arsenic leaching and precipitation. In this testing additional SMBS was added over and above the 0.77kg/t required for the $SO_2$/Air detox process. The reason of the additional SMBS was an attempt to provide additional oxidizing agent to assist in the leaching of the arsenic minerals prior to precipitation with ferric chloride.

The 500ppm arsenic level in solids for this test work was noted as being lower than the expected average of 1000ppm for the ore body, however this sample was used for further optimization test work while a new sample with a higher arsenic level was obtained. The test work conducted on the solids sample containing 500ppm arsenic was used to compare test performance and was not indicative of actual expected operational arsenic levels.

The following tests were conducted:

- JR 1044 – 2.5kg/t ferric chloride and 240g/t additional SMBS
- JR 1062 – 1.75kg/t ferric chloride and 2.2kg/t additional SMBS
- JR 1076 – 1.0kg/t ferric chloride and 3.5kg/t additional SMBS
- JR 1117 – 5.5kg/t additional SMBS

The product stream from each of these tests was sent to a kinetic column in which solution arsenic levels are being monitored over a 48 week period.

The tests on the solids sample containing 500ppm arsenic indicated that for this sample an arsenic level of 0.1ppm in the kinetic column could be achieved within 9 weeks with an addition rate of 2.5kg/t ferric chloride and 240g/t SMBS as per test JR 1044. Test JR 1062 with 1.75kg/t ferric chloride and 2.2kg/t SMBS addition produced a marginally better result than test JR 1044 reaching the target kinetic column arsenic level within 7 weeks. The result of test JR 1044 was considered optimal due to the resulting lower operating cost in the order of USD0.40/t based on the use of SMBS and an equivalent dosage of ferric sulphate as a source of ferric ion instead of the more expensive ferric chloride reagent.

Test JR 1117 has indicated that the addition of 5.5kg/t of SMBS with no ferric chloride addition in a combined cyanide destruction/arsenic precipitation stage could achieve similar results to testing conducted with ferric chloride addition. An operating cost comparison was conducted based on the use of the equivalent amount of ferric sulphate as a source of ferric ion. It was estimated that the use of 5.5kg/t SMBS would increase operating cost by USD0.80/t. This additional cost would have to be evaluated relative to the operational benefits associated with a simplified process.

The results of column tests conducted on arsenic remediation test slurries from tests conducted with ferric chloride and SMBS are presented graphically in Figure 13-20.

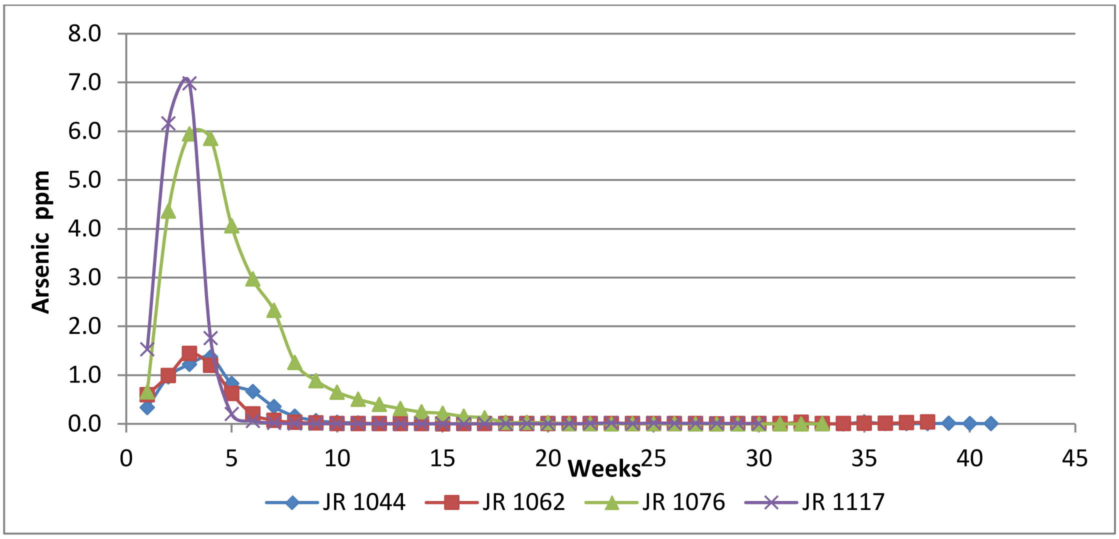



**Figure 13-20: Arsenic levels during column tests on arsenic remediation product slurry from tests with ferric chloride and SMBS addition**

In order to assess the long term stability of slurries, with an arsenic in solids content in the order of 1200ppm at an equivalent ferric chloride addition of 2.5kg/t, a final flowsheet validation test was conducted on a slurry post cyanide leaching and in combination with $SO_2$/Air detox (JR 1256). In order to better simulate intended plant design conditions, ferric sulphate was used as the source of ferric ion in this test.

The flowsheet validation test JR1256, which was used as the basis of the plant design, was run on a continuous basis to simulate intended in-plant treatment before discharge to the cyanide tailings facility and consisted of the following steps:

- A combined cyanide destruction and arsenic precipitation step incorporating shear, with a 4hr residence time.
- Two smaller arsenic precipitation stages of 1hr residence time each.
- A single pH correction stage of 1hr residence time.

Details of the test procedure and results are given in Table 13-26.

**Table 13-26: Procedure and Results for Arsenic Precipitation Test JR1256**

| Test JR1256<br>Arsenic Remediation with Ferric Sulphate and SMBS | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Test Step | Residence Time | FeCl3 Equiv. | CN Detox SMBS | As ppt SMBS | Lime | pH | ORP | As | CNWAD |
| | hrs | kg/t | kg/t | kg/t | kg/t | | mV | ppm | ppm |
| Feed | | | | | | 10.27 | | 23.6 | 86.62 |
| Stage 1 CN Detox + As ppt | 4 | 1.62 | 0.77 | | | 6.96 | 115 | 0.8 | 0.48 |
| Stage 2 As ppt | 1 | 0.55 | | 0.23 | | 5.91 | 160 | | |
| Stage 3 As ppt | 1 | 0.33 | | | | 5.62 | 214 | | |
| Stage 4 pH Correction | 1 | | | | 0.2 | 6.05 | 153 | 1 | 0.59 |

The Kinetic column test JR 1256 returned an arsenic in solution value of 0.034ppm after 8 weeks. At the time of writing this report, the 23 week arsenic in solution values are still well below the 0.1ppm target set for this work. The arsenic levels for the first 23 weeks of this column test are presented in Figure 13-21.

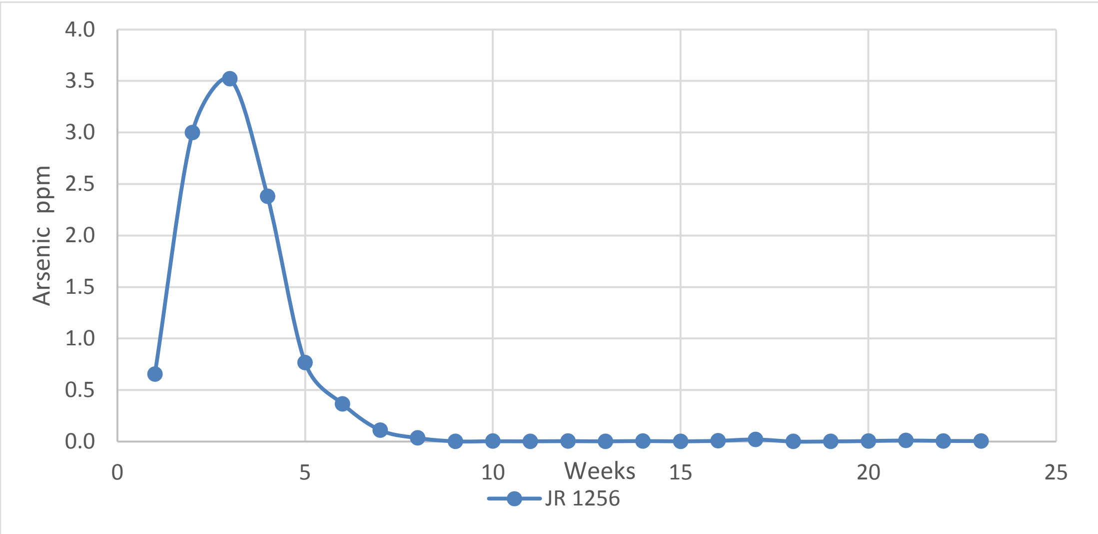



**Figure 13-21: Arsenic levels during column tests on arsenic remediation product slurry from test JR 1256 with ferric chloride and SMBS addition**

Based on the kinetic column test results for test JR1256, the New Liberty process plant design was updated to allow for $1000m^3$ tank in which cyanide destruction and arsenic leaching will take place. The additional leach tank is the same volume as the CIL tanks. In addition three $260m^3$ tanks were added to treat the detox/arsenic leach product stream. These tanks allow for additional ferric sulphate addition and pH correction of tailings.

Allowance has been made in the design for a total of 2.5kg/t equivalent Ferric Chloride addition as ferric sulphate and 0.23kg/t of SMBS addition. During operation, pH control is to be effected with SMBS and lime as required.

## 13.9 e-GRG Test Work Performed by Consep

As part of the optimisation phase e-GRG testing was carried out by Consep in Australia. A sample was screened and crushed to 100% passing 1.8 mm. The crushed product was then split to produce material for testing as follows:

- 1 kg for triplicate Au assay
- 3 x 2.5 kg portions for trial grinds
- 2 x 20 kg for eGRG/GRG testing

The results of the Gravity Recoverable Gold (GRG) test work are presented below.

**Table 13-27: e-GRG Test Work Head Assays**

| | Head Assay Au (g/t) |
|---|---|
| Assay 1 | 5.23 |
| Assay 2 | 13.45 |
| Assay 3 | 5.39 |
| Assay 4 | 7.46 |
| Assay 5 | 5.93 |
| Avg | 7.492 |

As can be seen in Table 13-28, there was a large amount of variation in the assayed head grade with an average reported value of 7.5 g/t. The average assayed head grade is higher than the calculated head assay of 3.80 g/t as presented in Table 13-28 and 4.54 g/t as presented in Table 13-29. This variation is indicative of the presence of coarse gold.

**Table 13-28: GRG Test Results**

| GRAVITY RECOVERABLE GOLD (GRG) RESULTS | | | | | | | |
|---|---|---|---|---|---|---|---|
| P80µm | Product | wt g | wt % | Au g/t | Au mg | Dist'n % | Cum Dist'n % |
| 690 | conc 1 | 85.7 | 0.43 | 190 | 16.26 | 21.3 | 21.3 |
| 220 | conc 2 | 93.6 | 0.47 | 225 | 21.03 | 27.5 | 48.8 |
| 75 | conc 3 | 161.2 | 0.80 | 32.3 | 5.21 | 6.8 | 55.6 |
| | final tails | 19762.5 | 98.3 | 1.72 | 33.93 | | |
| | totals (feed) | 20103 | 100.0 | 3.80 | 76.43 | | |
| | Knelson conc | 340.5 | 1.69 | 125 | | | |

duplicate Knelson tail (g/t Au): 1.84, 1.97

**Table 13-29: GRG with Concentrate Leaching Test Results**

| GRAVITY RECOVERABLE GOLD RESULTS WITH CONCENTRATE LEACHING | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| P80µm | Product | wt g | wt % | Au g/t | Au mg | Dist'n % | leached % | % recovery** |
| 690 | conc 1 | 95 | 0.47 | 223 | 21.16 | 23.17 | 95.47 | 22.12 |
| 220 | conc 2 | 94.6 | 0.47 | 285 | 27.00 | 29.58 | 95.10 | 28.13 |
| 75 | conc 3 | 156.7 | 0.78 | 40.8 | 6.40 | 7.01 | 90.72 | 6.36 |
| | tail | 19749.7 | 98.3 | 1.86 | 36.7 | 40.2 | -- | -- |
| | total | 20096 | 100.0 | 4.54 | 91.30 | 100.0 | | 56.6 |
| | Knelson conc | 346.3 | 1.72 | 158 | 55 | **59.8** | | |

duplicate Knelson tail (g/t Au): 1.84, 1.88

* concentrate grades calculated from leach tests

** recovery = gravity distribution to concentrates x leach dissolution

The overall GRG of the sample was determined to by 55.6%, with 21.3% recovered in the first pass and 27.5% recovered in the second pass. This indicated that the ore required a minimum grind of 80% passing 220 µm for the bulk of the GRG to be liberated. Intensive cyanidation recovery of the first, second and third pass concentrate was 95.5%, 95.1% and 90.7% respectively.

Further grinding to 80% passing 75 µm only liberated an additional 6.8% GRG, the intensive leach tests on this concentrate indicated that this material was relatively slow leaching which is indicative of the fact that the gold in this concentrate was not fully liberated.

The results of the eGRG tests were used to simulate the gravity recovery for full scale plant operations based on the distribution of GRG in the concentrate size fractions. The circuit was modelled based on a 300% circulating load in the ball mill with one third of the cyclone underflow reporting to the gravity concentration circuit.

Based on this the GRG contribution to gold recovery was simulated by Consep to be between 38% - 46% with an expected plant GRG recovery of 41%.

The test work on composite samples at ALS showed gravity recoveries ranging from 51% - 63%, and on variability samples the range was from 38% - 81%. Plant recovery estimates have been based on CIL residue grades and have not taken staged GRG recovery into account because the recovery of coarse gold will have been achieved in the milling circuit and subsequent intensive leaching.

## 13.10 Metallurgical Recovery Estimate

### 13.10.1 Introduction

In order to provide estimation for recovery based on test work, the following was undertaken:

- Evaluation of test work data to produce a correlation between head grade, grind and recovery.
- Monte Carlo probability distributions (derived from test results) to determine the 90% confidence interval for the residue grades achieved at each target grind.

In order to provide an estimate of the expected recovery for full scale continuous plant operations, the bench scale laboratory recoveries were discounted. This discount factor is used in order to account for process inefficiency and solution gold losses due to:

- Carbon fines losses to tailings.
- Solution Gold losses.
- Inefficiency of high shear oxygen addition in the pre-oxidation phase as compared to laboratory testing (Scale-up).

### 13.10.2 Derivation of a Correlation between Grade, Recovery and Mill Grind

The following test work data was used to derive a correlation between grade and recovery and mill target grind.

- Mintek CIL test work results for Phase 1, 2 and 7 from the previous feasibility study.
- ALS test work results on composite and variability samples.

It was decided to include Mintek results from phase 1, 2 and 7 as the test work conducted at ALS included very few tests conducted at a grind of 80% passing 75 µm. The most appropriate data from the Mintek test work campaign was determined to be that from Phase 1, Phase 2 and Phase 7. The Phase 6 CIL test work was excluded from this analysis due to the fact that the CIL recoveries achieved were significantly lower than in any of the previous test work phases and lower than the achievable recovery as indicated by diagnostic leach tests. It was noted that test work on this material was conducted without any input from DRA or MDS and has been considered not to be representative test work data. However this material was composited and re-tested in Phase 7 under the guidance of MDS. The recovery achieved in Phase 7 increased in line with that as indicated as achievable by the diagnostic leach tests.

The relationship between final CIL residue grade and head grade for a target grind size of 80% passing 75 µm, 60 µm, 50 µm, 42 µm and <20 µm is presented in Figure 13-22. With the exception of the three data points as highlighted it would appear that there was an increase in final residue grade at higher head grades. Upon evaluation of the data it was decided that the correlation between final residue grade and head grade for each grind could be adequately described by the following equation:

$$Final\ Residue\ \propto K(Head\ Grade)^{0.35}$$

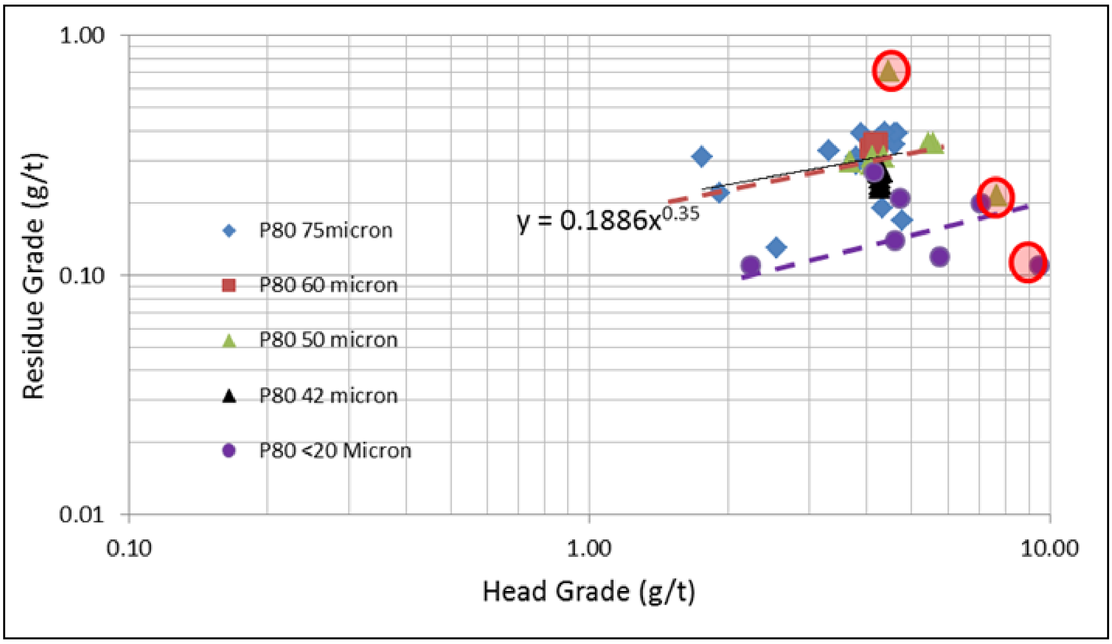



**Figure 13-22: Test Work Recovery as a Function of Grind**

The value for the constant K at each target grind size was then determined based on the test work data but excluding the result of test JR 281 which was considered an outlier for reasons as detailed in Section 13.4.1. The test results for test JR 283 and JR 250 were included in this analysis based on the fact that the residue grades of 0.22 g/t and 0.11 g/t being consistent with that achieved throughout the various test work programmes.

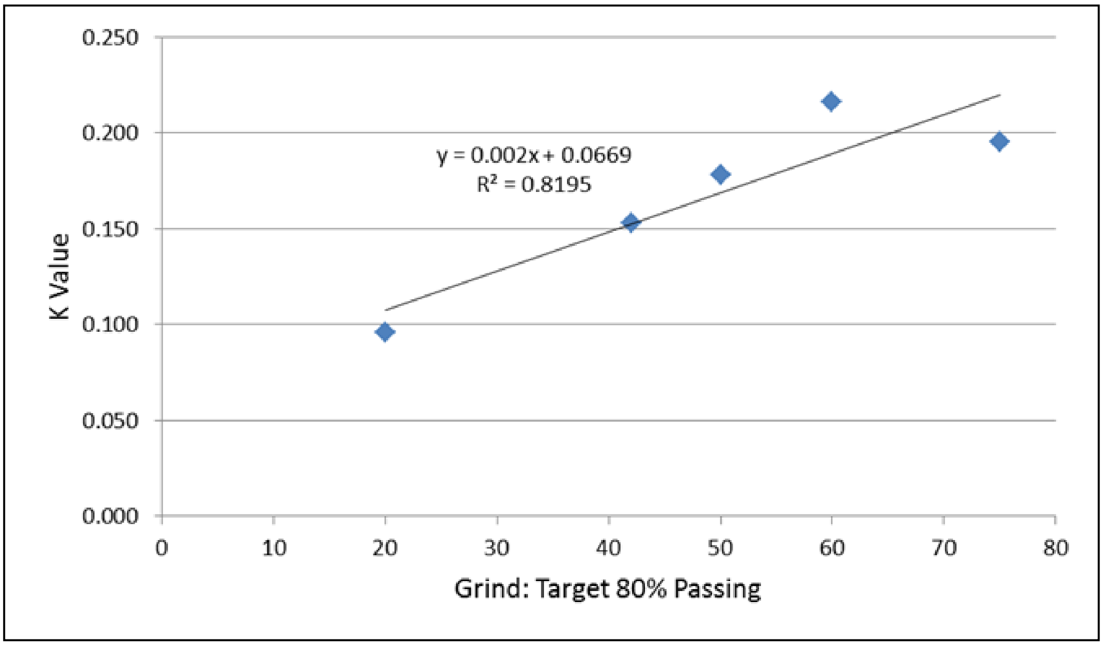



**Figure 13-23: Derivation of Correlation Constants for each Target Grind**

The correlation constants as determined from Figure 13-23 are presented in Table 13-30 below.

**Table 13-30: Correlation Constants for Each Target Grind Size**

| Grind | Constant (K) | |
|---|---|---|
| P80 | Test Work | Model |
| 75 | 0.195 | 0.217 |
| 60 | 0.216 | 0.187 |
| 50 | 0.178 | 0.167 |
| 42 | 0.153 | 0.151 |
| 20 | 0.096 | 0.107 |

Based on the constants as presented in Table 13-30 above the grade recovery correlation at each target grind size was found to be as follows:

- Final Residue (P80 75µm) = 0.217(Head Grade)^0.35
- Final Residue (P80 60µm) = 0.187(Head Grade)^0.35
- Final Residue (P80 50µm) = 0.167(Head Grade)^0.35
- Final Residue (P80 42µm) = 0.151(Head Grade)^0.35
- Final Residue (P80<20µm) = 0.107(Head Grade)^0.35

This modelled grade–recovery relationship is graphically presented in Figure 13-24.

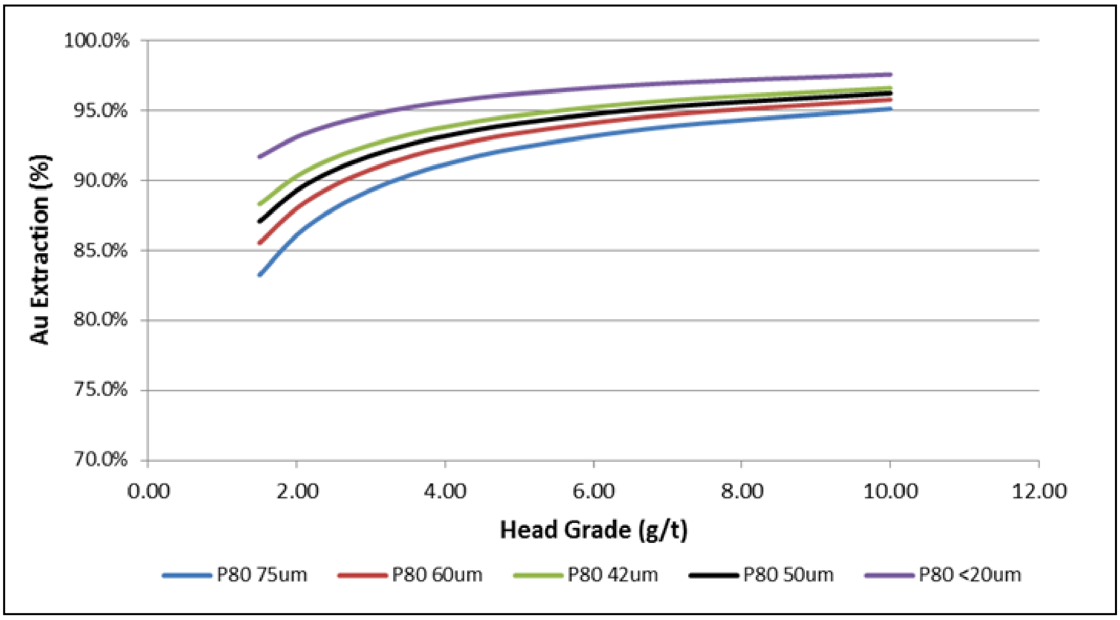



**Figure 13-24: Model Predicted Grade Recovery Curve at Each Target Grind Size**

The grade recovery relationship as predicted by the model as derived above was compared to the Mintek Phase 1, 2 and 7 and ALS test work results and found to be in good agreement as presented in Figure 13-25.

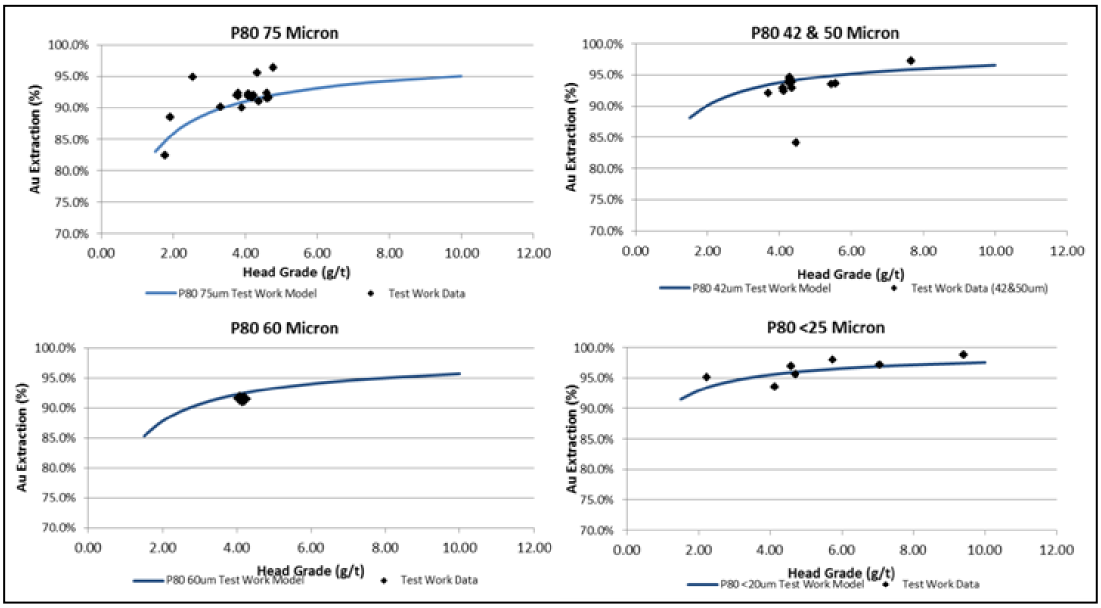



**Figure 13-25: Test Work Au Extraction Relative to Model Prediction**

### 13.10.3 Monte Carlo Analysis

A Monte Carlo analysis was performed to confirm the recovery as determined from the grade recovery models derived from test work as presented in Section 13.8.1. The following data was used for the Monte Carlo analysis:

- Mintek Phase 1, 2 and 7 from the previous feasibility study.
- ALS Optimization Phases Testing on composite and Variability samples (2012-2013).

*80% Passing 75µm Monte Carlo Distribution*

The probability distributions for 80% passing 75 µm target grind are presented in Figure 13-26 below. The results show a mean residue grade of 0.30 g/t. The 90% confidence recovery range for all samples is shown on the graphs, and can be seen to provide a residue range of between 0.16 g/t – 0.40 g/t over this confidence interval.

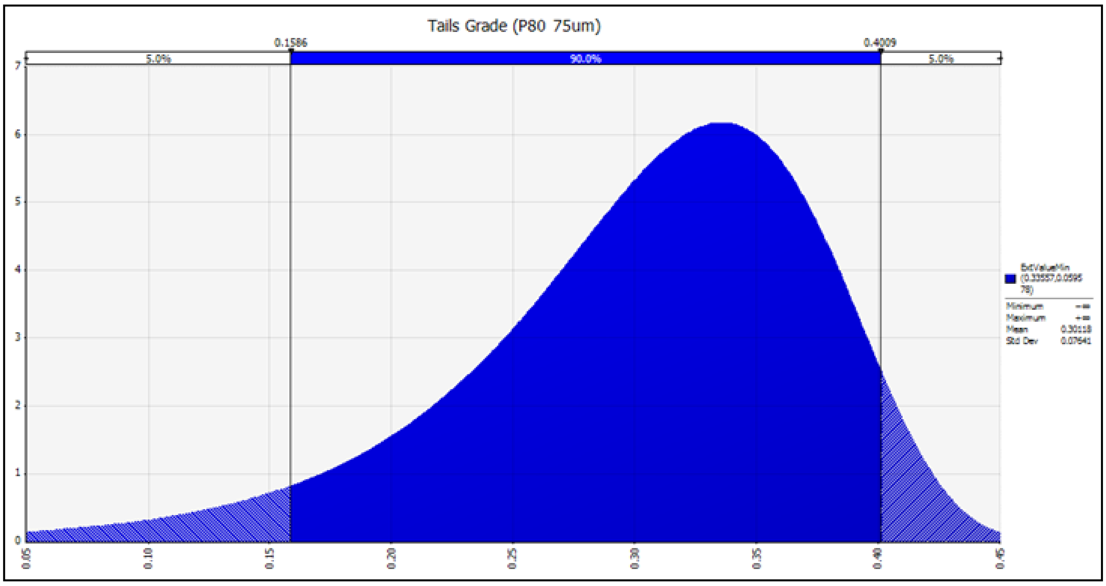



**Figure 13-26: Monte Carlo Analysis of Test Work Residue Grades at 80% Passing 75 Micron**

*80% passing 50µm Monte Carlo Distribution*

The probability distributions for 80% passing 50 µm target grind are presented in Figure 13-27 below. The results show a median residue grade of 0.33 g/t. The 90% confidence recovery range for all samples is shown on the graphs, and can be seen to provide a residue range of between 0.21 g/t – 0.54 g/t over this confidence interval.

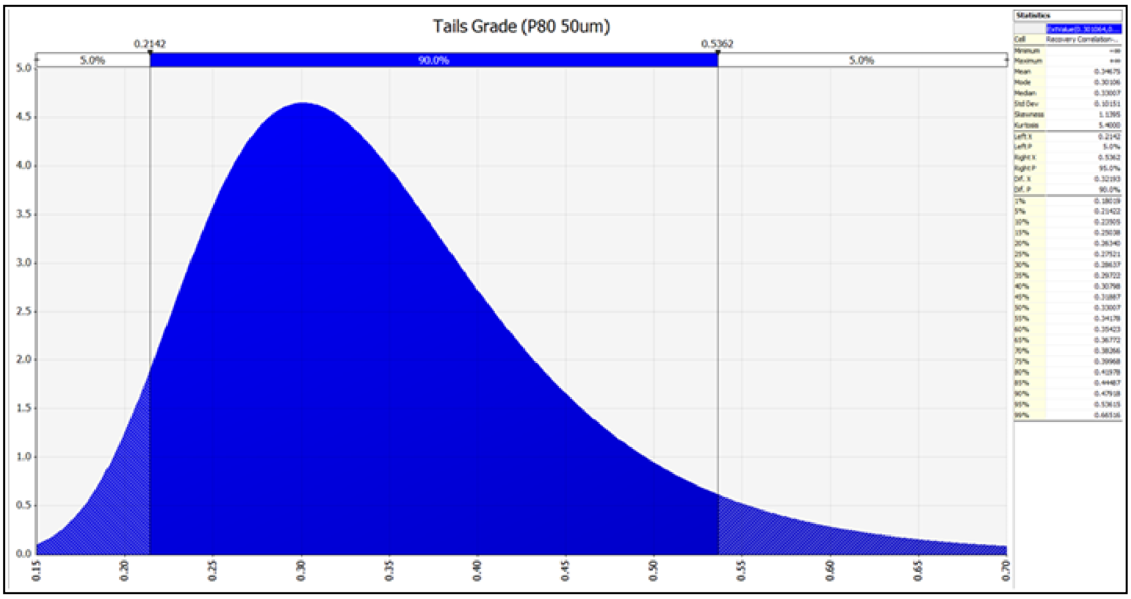



**Figure 13-27: Monte Carlo Analysis of Test Work Residue Grades at 80% Passing 50 Micron**

*80% Passing 42 µm Monte Carlo Distribution*

The probability distributions for 80% passing 42 µm target grind are presented in Figure 13-28 below. The results show a median residue grade of 0.25 g/t. The 90% confidence residue range for all samples is shown on the graphs, and can be seen to provide a recovery range of between 0.22 g/t – 0.27 g/t over this confidence interval. It should be noted that the recovery range provided here is based on all the composite tests conducted at a grind of 80% passing 41 µm - 43 µm. Because these tests were conducted on the composite samples this analysis is an indication of repeatability.

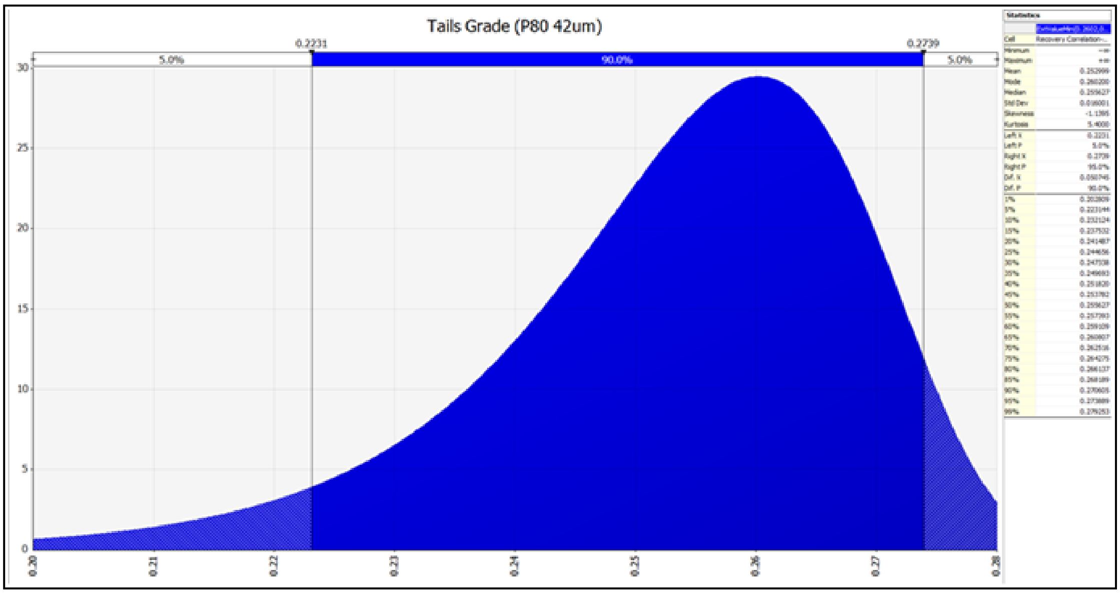



**Figure 13-28: Monte Carlo Analysis of Test Work Residue Grades at 80% Passing 42 Micron**

*80% passing <20 µm Monte Carlo Distribution*

The probability distributions for an 80% passing <20 µm target grind are presented in Figure 13-29 below. The results show a median residue grade of 0.15 g/t. The 90% confidence recovery range for all samples is shown on the graphs, and can be seen to provide a residue range of between 0.09 g/t – 0.27 g/t over this confidence interval.

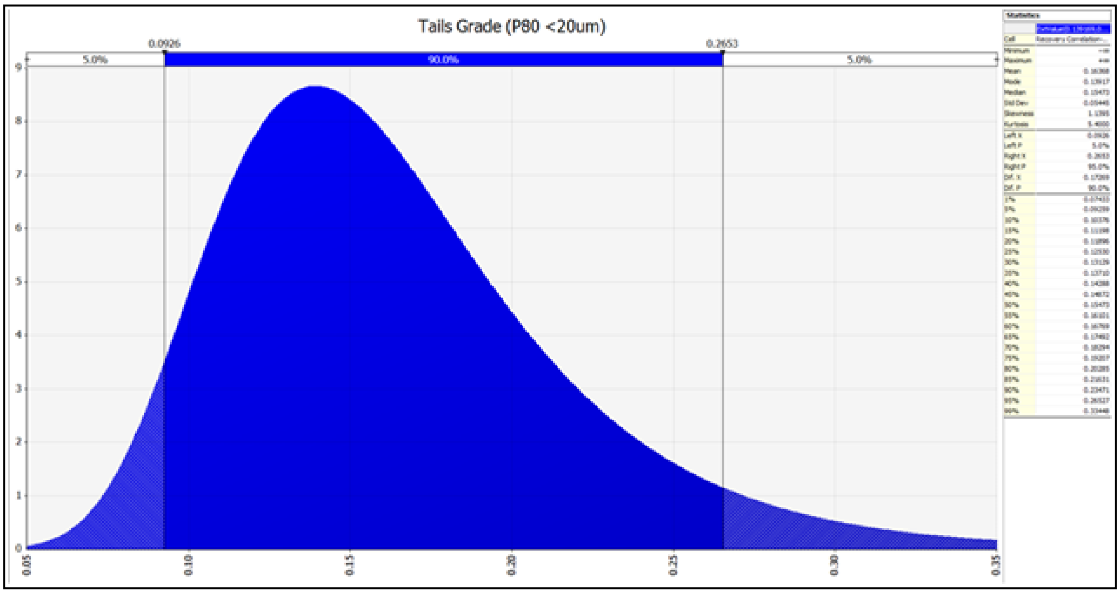



**Figure 13-29: Monte Carlo Analysis of Test Work Residue Grades at 80% Passing <20 Micron**

### 13.10.4 Head Grade, Grind, Recovery Correlation Compared to the Monte Carlo Distribution Results

The residue grade for each target grind size as determined by the Monte Carlo analysis was found to be in good agreement with the residue grade as predicted by the model presented in Section 13.8.1, for the range of head grades tested. This comparison is presented in Table 13-31 below.

**Table 13-31: Comparison of Modelled CIL Reside Grades and Residue Grades as Determined by Monte Carlo Analysis**

| P80 Grind | Test Work Head Grade (g/t) | Monte Carlo Reside Grade (g/t) | Modelled Reside Grade (g/t) |
|---|---|---|---|
| 75 µm | 1.32 - 4.77 | 0.16 -0.40 | 0.24 - 0.37 |
| 60 µm | 4.07 - 4.22 | Insufficient Data | 0.31 |
| 50 µm | 3.69 - 7.66 | 0.21 - 0.54 | 0.26 - 0.34 |
| 42 µm | 4.27 - 4.30 | 0.22 - 0.27 | 0.25 |
| <20 µm | 2.24 - 9.43 | 0.09 - 0.27 | 0.16 - 0.26 |

### 13.10.5 Conclusions

Based on the current proposed mine production schedule, the estimated plant recovery for New Liberty at a target grind of 80% passing 45 µm is presented in Table 13-32. The recovery estimate is based on the expected range for the residue Au grade as determined by the correlation between head grade, grind and recovery as presented in Section 13.8.1. The recovery estimate presented in Table 13-32 includes a recovery discount figure which was calculated based on the following:

- Fixed carbon losses of 25 g carbon loss per ton milled at an estimated gold grade of 50 g/t.

- Fixed solution losses based on a CIL feed density of 45% solids (w/w) and a solution gold content of 0.05 g/L in the plant tailings stream.
- An assumed recovery loss of 0.5% for inefficiency of high shear oxygen addition in the pre-oxidation phase as compared to laboratory testing to account for Scale-up to plant conditions.

The three recovery discount factors above were combined to formulate a combined recovery discount for full scale plant operations as presented in Table 13-32 below.

**Table 13-32: New Liberty Plant Recovery Estimate for a Target Grind of 80% Passing 45 Micron**

| 80% Passing 45um | | | | | |
|---|---|---|---|---|---|
| Year | Au g/t | Mtpa | Residue (g/t) | Recovery Discount | Modelled Avg |
| 1 | 3.10 | 1.00 | 0.22 | 0.77% | 91.99% |
| 2 | 3.60 | 1.10 | 0.24 | 0.73% | 92.70% |
| 3 | 3.20 | 1.10 | 0.23 | 0.76% | 92.15% |
| 4 | 4.00 | 1.10 | 0.25 | 0.71% | 93.16% |
| 5 | 4.00 | 1.10 | 0.25 | 0.71% | 93.16% |
| 6 | 3.50 | 1.10 | 0.23 | 0.74% | 92.57% |
| 7 | 3.30 | 1.10 | 0.23 | 0.75% | 92.30% |
| 8 | 2.00 | 0.80 | 0.19 | 0.92% | 89.46% |
| Average Year 1-4 | | | | | 92.51% |
| Average Year 1-5 | | | | | 92.64% |
| Average Year 1-6 | | | | | 92.63% |
| Average Year 7-8 | | | | | 91.10% |
| Average LOM | | | | | 92.29% |

Based on the results from the metallurgical test work an average of 93% gold recovery should be achievable for years one to six under steady state conditions, post commissioning and optimisation of recovery.

# 14 MINERAL RESOURCE ESTIMATES

## 14.1 Introduction

A Mineral Resource estimate for New Liberty was produced in January 2012 by AMC. Subsequently, following additional drilling mainly targeted between the Kinjor and Marvoe zones, an update of the mineral resource estimate was completed in April 2012 and reported in October 2012, also undertaken by AMC. Not all analytical tasks were repeated by AMC during the update (e.g. QA/QC and most statistical and variographic analyses), as the changes associated with the update drilling were not considered material to the manner in which the estimates were run. This updated mineral resource estimate (the 2012 Mineral Resource Estimate) remains the most up to date estimate available and forms the basis of the mineral reserve reported later in this report and the mining plan.

The 2012 Mineral Resource Estimate is based on interpretations from integrated geological and grade information recorded from diamond core logging and assaying. Apart from the initial sample data preparation, some aspects of the variography and intermediate spreadsheet processing, all of the mineral resource interpretation, modelling and estimation work was conducted using the Datamine geological and mine planning software package.

The Datamine 2D interactive and 3D visualization graphical environments were used to generate triangulated wireframe models, as well as for visual validation. Extensive use was made of the Datamine macro facilities for almost all of the data processing, as well as the analytical, cell modelling, estimation and reporting functions, and hence these macros constitute an audit trail for much of the work undertaken.

The deposits have been evaluated with reference to the UTM grid, and all directional references in the resource portions of this report are according to this grid.

## 14.2 Data Storage and Preparation

The sample database was provided to AMC by Aureus as a set of Microsoft Access database tables, as listed in Table 14-1. The final data available for use in the primary evaluation was received on 9 December 2011 and represented drilling up-to and including drillhole K375, whereas the updated database, received on 4 April 2012, represented drilling up to and including drillhole K438, as summarised in Table 14-2. Not all of the additional update holes were targeted on the resource mineralisation.

**Table 14-1: Sample Database Data Tables**

| Table | Records | |
|---|---|---|
| | December 2011 | March 2012 |
| Collar | 374 | 437 |
| Survey | 6168 | 7679 |
| Assay | 28962 | 36277 |
| Lithology | 18664 | 24003 |
| Alteration | 9050 | 9050 |
| Geotechnical | 15900 | 15900 |
| Density | 14065 | 14065 |

The database tables were subjected to standard validation procedures. Geochemical fields provided in the database are summarised in Table 14-2. In the case of gold grades, the primary assay field was extracted for use in the resource estimate. The trench data was considered adequate for spatial viewing purposes but not of sufficient quality for use in the resource estimation process.

**Table 14-2: Geochemical Fields**

| Field Name | Description | Used In Evaluation? |
|---|---|---|
| Au | Primary gold assay | Y |
| Au Rpt1 | First repeat gold assay | N |
| Au Rpt2 | Second repeat gold assay | N |
| Mean Au | Mean of primary and repeat gold assays | N |
| As | Arsenic assay (not populated) | N |
| Mag_sus | Magnetic susceptibility | N |

## 14.3 Geological Modelling

### 14.3.1 Orebody Geometry

A 'silicified metamorphosed ultrabasic suite' (SMUS) zone was identified which contains the gold mineralisation and this was therefore modelled using hard contacts. These contacts were determined as downhole drill pierce points based on logged lithological and stratigraphic unit codes.

The interpreted SMUS zone is continuous across, and is assumed to extend beyond, the strike of the drilled domain, and the full sub-surface extents have not yet been defined. The SMUS strikes approximately 097° over the western half of the deposits, while in the east it swings slightly towards the south (105°). Southerly dips are typically in the range 65°-80°.

SMUS zone boundaries are more confidently defined near surface, with the benefit of higher drilling density and supported by surface mapping. At 0 m RL elevation, horizontal thicknesses typically range from 40 m to 90 m, occasionally reaching 120 m.

With increasing depth, the western half appears to widen significantly, reaching a horizontal width of around 250 m at approximately -400 m RL.

### 14.3.2 Mineralisation

Intersections of anomalous gold grades occur in places across the full profile of the SMUS zone. However, within each drillhole, elevated grade intersections of potentially economic interest are much more restricted in number and extent. Correlations between these higher grade intersections, typically above 0.5 g/t Au, reveal an orientation that is broadly aligned with the SMUS, although in some cases they drift slightly obliquely to the SMUS contacts in both strike and dip. The dimensions of these zones of elevated grade (mineralised zones) are strongly anisotropically planar, ranging in width between a few to sometimes 10 m – 15 m, while typically extending hundreds of metres in both strike and dip.

From early in the Project exploration, at least three discrete zones (Larjor, Kinjor and Marvoe) were recognized, apparently separated along strike by surface mineralisation discontinuities. Drilling has subsequently confirmed that these zones continue as discrete entities at depth. An additional small, poorly mineralised, zone was identified some 200 m west along strike of Larjor, but has been excluded from the latest 2012 mineral resource estimate.

For each of the Kinjor and Marvoe zones, early detailed interpretations indicated the presence of two plane-parallel sub-zones. Subsequent re-interpretation led to the addition of a third sub-zone at the western end of Marvoe and a small second sub-zone in near-surface eastern end of Larjor. Furthermore, during the 2009/2010 drilling campaign, Aureus recognized that previous drilling had not satisfactorily sterilized the apparent gap in mineralisation between Larjor and Kinjor, and drill testing revealed the presence of what became known as the Latiff zone.

The 2011/2012 drilling campaign confirmed the previous interpretations in most respects, but the additional drillhole information formed the basis of further refinements, namely:

- Recognition that the eastern extent of the Latiff zone can be considered as a continuation of the Kinjor main hanging wall subzone.

- Justification for merging of Larjor with the western end of Latiff, although across the merged interval the mineralisation is thin and poorly developed.
- Subdivision of the western (hanging wall) Marvoe zone into two subzones based on markedly different geometric and grade distribution characteristics.
- Treatment of the previously defined dual and parallel Marvoe zones as a single entity.

Subsequently, the update drilling provided both confirmation and down-dip extension of the relatively thick, but limited strike length, zone positioned in the hanging wall of the gap between Kinjor and Marvoe.

To facilitate both interpretation of the mineralisation and subsequent resource modelling, a coding system, using the MINZONE field name, had been developed during previous modelling to distinguish the various zones and sub-zones. The assigned MINZONE codes have been modified to account for the changes listed above and are summarized in Table 14-3, along with the corresponding zone dimensions.

**Table 14-3: Mineralised Zone Codes (MINZONE Field)**

| Zone | Relative Position | MINZONE Field Code | Max Strike Extent (m) | Max. Dip Extent (m) |
|---|---|---|---|---|
| Larjor | Merged into Kinjor in 2012 | | | |
| Latiff | | | | |
| Kinjor | HW | M401 | 1460 | 600 |
| | FW | M402 | 460 | 500 |
| Marvoe | | M501 | 500 | 410 |
| | HW west | M503 | 100 | 170 |
| | HW central | M504 | 200 | 130 |
| SMUS uncorrelated | | BKGR | | |
| GNSS unmineralised | | WSTE | | |

No particular grade cut-off values were applied in the downhole definition of mineralised zone intersections. To define intervals of enhanced gold grade, boundaries commonly correspond with the first significant value above background, typically in the range 0.3 g/t Au to 2.0 g/t Au. In cases of ambiguity or gradational transitions between background and mineralisation, evidence of the sharpest change in grade was used as the primary criterion for boundary positioning. Many intersections can be relatively unambiguously defined; however others require some subjectivity and judgement.

Between drillholes, intersections were correlated on the basis of broad alignment with the SMUS zone contacts; however, ultimately, grade provided the primary basis for detailed correlation, as no other geological attributes were found to adequately define the mineralisation geometry.

The individual mineralised zones were initially wireframe-modelled as hanging wall and footwall surfaces, based on drillhole pierce points, combined with polygons representing strike and dip projections, and then formed into enclosing solids.

### 14.3.3 Oxidation

Interpretation of the weathered horizon has been based on geological logging from near-surface intersections in diamond core drilling, drawn on sections at 25 m intervals. The logged weathering codes record a progressive four-step diminution of weathering, as shown in Table 14-4, from completely weathered nearer to surface, down to fresh rock at depth.

Interpretation of weathering revealed that the partially or slightly weathered interval is commonly absent or, if present, seldom exceeds 5 m. Consequently modelling of weathering has been limited to a single horizon incorporating all weathered zones, the base of which is defined by the top-of-fresh rock.

**Table 14-4: Weathering Zone Codes (WEAZONE Field)**

| Description | Logged Code | Interpreted Surfaces | WEAZONE | Explanation |
|---|---|---|---|---|
| Completely weathered | CW | Base of complete oxidation | WEAT | Oxidized rock |
| Partially weathered | PW | | | |
| Slightly weathered | SW | | | |
| Unweathered | UW | Top-of-fresh | FRSH | Unoxidised rock |

A triangulated wireframe surface, representing the interpreted top-of-fresh rock, was constructed by linking the corresponding digitised section strings.

## 14.4 Topography

In December 2012, Aureus commissioned a LiDAR survey encompassing the Project Area. This data, together with the exploration borehole collar data, collected via a differential GPS (DGPS) and Total Station survey, was used as the database from which a highly detailed digital terrain surface was generated using AutoCAD software.

The resultant digital terrain model provides full coverage over the block model extents, and is confirmed as accurate by on-ground DGPS survey data.

The limited artisanal workings undertaken since this LiDAR survey have not been fully accounted for within the topographic survey, as these are of limited lateral and vertical extent and any volume differences associated with the workings are within the resolution of the topographic survey point distribution.

## 14.5 Cell Model Construction and Coding

A cell model of the project deposits was constructed from a suite of sub-models using a base configuration of 10 m (Easting) × 5 m (Northing) × 10 m (RL) parent cells, as shown in Table 14-5. The cell geometry was selected on the basis of the overall drill spacing in the relatively near-surface portions of the deposit (to 200 m depth).

**Table 14-5: Model Cell Parameters**

| Direction | Parent Cell Size (m) | Minimum Subcell (m) | |
|---|---|---|---|
| | | Mineral. | Topo/Weath |
| Easting | 10 | 5 | |
| Northing | 5 | 2.5 | |
| RL | 10 | 10 | 2 |

For the mineralised zone sub-model, each wireframe was filled with cells such that parent cells were permitted to split along bounding surfaces down to the minimum dimensions shown in Table 14-6. Cells were coded with the relevant MINZONE code.

An enclosing sub-model of the SMUS geological unit was constructed by filling the solid wireframe to a horizontal resolution of the parent cells and to a 2 m vertical subcell resolution (along topography). Cells within this stratigraphic sub-model were coded within the STRZONE field as 'SMUS'.

Sub-models representing both topography and weathered material were created by building cells above the respective triangulated surfaces, with horizontal and vertical subcell minima of 2 m. The weathering model was coded with the WEAZONE field set to 'WEAT'.

The mineralisation, stratigraphic, weathering and topographic (air) sub-models were combined to produce a unified and coded model consisting of mineralisation zones set within the SMUS unit, flagged by weathering code, and trimmed along the topographic surface. The various model code and attribute fields are listed in Table 14-6 and Figure 14-1 shows a schematic relationship of the mineralised zones within the SMUS unit.

**Table 14-6: Coded Model Field Descriptions**

| Coded | Field | Description |
|---|---|---|
| Pre-estimation | MINZONE | Mineralised zone (see Table 14.4) |
| | WEAZONE | Weathering zone (see Table 14.5) |
| | STRZONE | Division between gneiss and ore-hosting suite. |
| | SRCHZONE | Search ellipse orientation domain |
| Post-estimation | AU | Gold grade (g/t) |
| | DENSITY | Estimated/assigned bulk density ($t/m^3$) |
| | PASS | Search ellipse pass (1,2,3) |
| | NUMSAM | Number of samples used to estimate cell |
| | RESCAT | Resource classification codes |

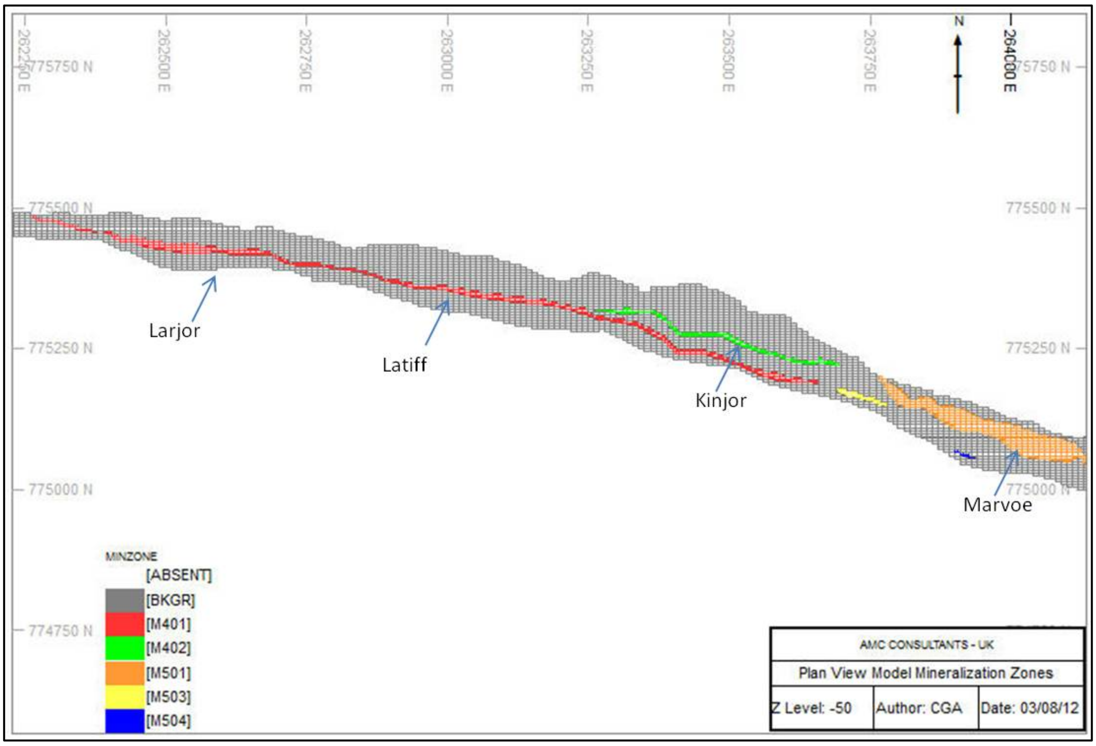



**Figure 14-1: Schematic Plan View of Model Mineralised Zones**

## 14.6 Sample Coding

Coding of samples according to mineralisation, stratigraphy and weathering zones followed a similar sequence of steps to the construction of the cell model.

Prior to coding, AMC excluded a number of holes, listed in Table 14-7, including some older ones which were rejected because of inconsistencies and which, based on more recent information, were indicative of data errors.

**Table 14-7: Rejected Drillholes**

| Drillhole | Reason for Exclusion |
|---|---|
| K004 | Does not fit with surrounding data |
| K010 | NS orientation (down dip) |
| K023 | Poor fit with several new holes |
| K032 | NS orientation |
| K034 | NS orientation |
| K036 | NS orientation |
| K038 | NS orientation |
| K044 | Unsampled |
| K055 | NS orientation |
| K070 | Fits interpretation but very low RL (-15 m) |
| K080 | Poor orientation, apparently unsampled |
| K082 | Poor fit, no min |
| K084 | Poor fit, no min |
| K085 | Unexplained displacement of mineralisation |
| K192 | Drilled in FW |
| K193 | Drilled vertical - no intersection |
| K367 | Drilled in FW |
| K370 | Drilled in FW |
| K372 | Drilled in FW |

The remaining drillhole samples were coded according to the relevant MINZONE codes using a tagging process which incorporated the same table of downhole intercepts used to define the mineralised zone wireframes. To achieve stratigraphic coding of the field STRZONE, samples were 'captured' within the wireframe solid of the SMUS geological unit. The top-of-fresh weathering surface was applied to the samples to code those which are located within the weathered horizon (WEAZONE field).

The resulting sample coding is consistent with the cell model codes shown in Table 14-7.

## 14.7 Bulk Density

Of the 13,547 bulk density measurements used in the evaluation, 1,361 fall within fresh mineralised zones. A review of the spatial distribution of values showed a clear distinction between weathered zone and fresh sample densities, but exhibited no particular trends or associations internally within these horizons. Figure 14-2 shows the variation with depth of densities, for both weathered and fresh rock (in the first 30 m below surface). Overlaps between the two clusters of points reflect difficulties at times in partitioning drill intervals according to weathering, as well as the natural variability in weathering intensity.

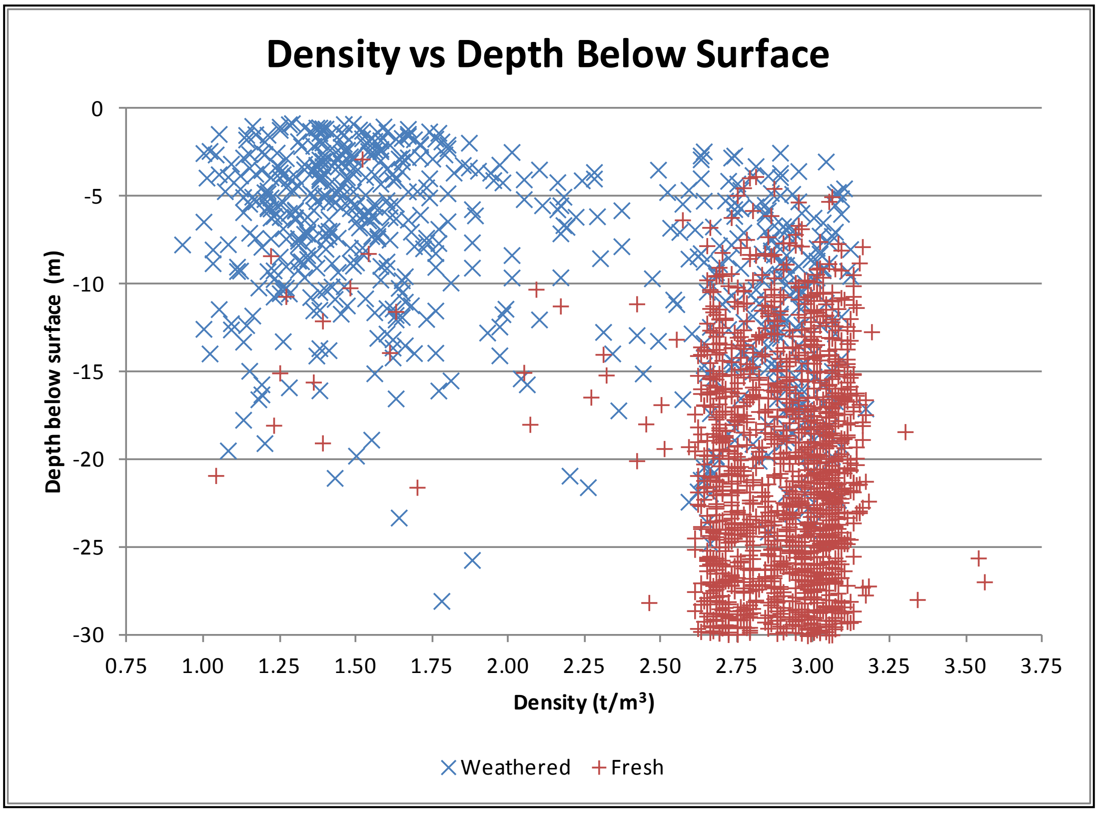



**Figure 14-2: Density Variation with Depth**

Mean density values by mineralisation zone and weathering horizon are summarised in Table 14-8.

**Table 14-8: Mean Bulk Density Values**

| WEAZONE | Zone | Bulk Density | |
|---|---|---|---|
| | | Number of samples | Mean (t/m$^3$) |
| WEAT | ALL | 641 | 2.01 |
| FRSH | M401 | 617 | 2.99 |
| | M402 | 130 | 2.94 |
| | M501 | 564 | 2.97 |
| | M503 | 28 | 2.96 |
| | M504 | 22 | 2.97 |
| | BKGR | 11,545 | 2.90 |

## 14.8 Sample Compositing and Statistics

### 14.8.1 Composite Selection

Approximately 95% of samples within the interpreted mineralised zones have lengths of 1.0 m. On this basis, and with consideration for the narrowness of the most of the zones, 1.0 m was selected as a composite length for mineralised zone statistical analysis, and estimation. Outside of the mineralised zones, 2.0 m composites were used because of the common presence of 2.0 m and sometimes 4.0 m composites in early drill campaigns.

### 14.8.2 Statistical Procedures and Characteristics

Table 14-9 shows the gold grade univariate statistics for 1 m composites and population characteristics for each of the mineralised zones. The corresponding sample distributions were plotted graphically as histograms, log histograms, and log probability charts. Figure 14-3 presents example charts for the dominant zones of Larjor, Latiff, Kinjor and Marvoe.

**Table 14-9: Summary Statistics within Mineralised Zones**

| Description | Field | Number | Min. | Max. | Mean | Variance | CoV |
|---|---|---|---|---|---|---|---|
| Weathered Minzones | Au | 359 | 0.01 | 23.04 | 2.22 | 12.6 | 1.3 |
| Fresh Minzones | Au | 4,540 | 0.01 | 86.40 | 2.84 | 31.1 | 2.0 |
| Minzone M201 | Au | 956 | 0.01 | 80.80 | 3.44 | 30.1 | 1.6 |
| Minzone M301 | Au | 486 | 0.01 | 74.24 | 4.22 | 37.1 | 1.4 |
| Minzone M401 | Au | 579 | 0.01 | 81.31 | 3.86 | 46.5 | 1.8 |
| Minzone M402 | Au | 422 | 0.01 | 34.40 | 2.05 | 15.7 | 1.9 |
| Minzone M501 | Au | 1,609 | 0.01 | 48.00 | 1.54 | 11.9 | 2.2 |
| Minzone M503 | Au | 297 | 0.01 | 53.76 | 5.15 | 64.1 | 1.6 |
| Minzone M504 | Au | 63 | 0.01 | 86.40 | 5.84 | 233.7 | 2.6 |

The grade distributions for all the data subsets shown in Table 14-9 display some degree of multimodality, mostly bimodal; however the inflections between subpopulations are transitional and the clarity of the bimodality has generally reduced as more drilling has been completed. The high-grade zone M503 displays the most marked bimodality, with a transitional inflection between 2.0 and 4.0 g/t Au.

In all cases the spatial distributions of the subpopulations were reviewed and AMC concluded that the lower grade portions represent narrow low grade or anomalous intervals within the overall defined zones that were not of a scale that would validly allow either partitioning from the interpreted mineralised zone or separate extraction during mining.

The mean gold grades for individual zones (excluding M504) are varied (1.54 g/t Au – 5.04 g/t Au). Nonetheless, the corresponding coefficients of variation (CoV) fall within a relatively narrow window (1.3 – 2.2), which may reflect a commonality of genesis for the different zones. The higher CoV of zone M504 is consistent with the high variability observed between drill intersections in that zone. In general the relative coherence of statistical grade distributions (good – M401 west, poor – M402) is consistent with the corresponding relative ease of zonal correlation during interpretation.

To assess whether there was any statistical distinction between gold grades from within the weathered zone and gold grades for fresh samples, a comparison was made between mineralised zone samples flagged as weathered and a corresponding set of mineralised samples within the interval 25 m below the interpreted base of weathering surface.

Overall, the grade distributions for the two (weathered and fresh) subsets exhibit similar characteristics; however the mean grade of the weathered population (2.22 g/t Au) is markedly lower than that of the near surface unweathered (2.57 g/t Au). Furthermore visualisation of gold grades across the boundary commonly shows differences in character that are not simply explained by the variability observed within the deposit as a whole. Consequently, the weathered and fresh assays were treated separately by AMC during variography and grade estimation.

**Larjor**

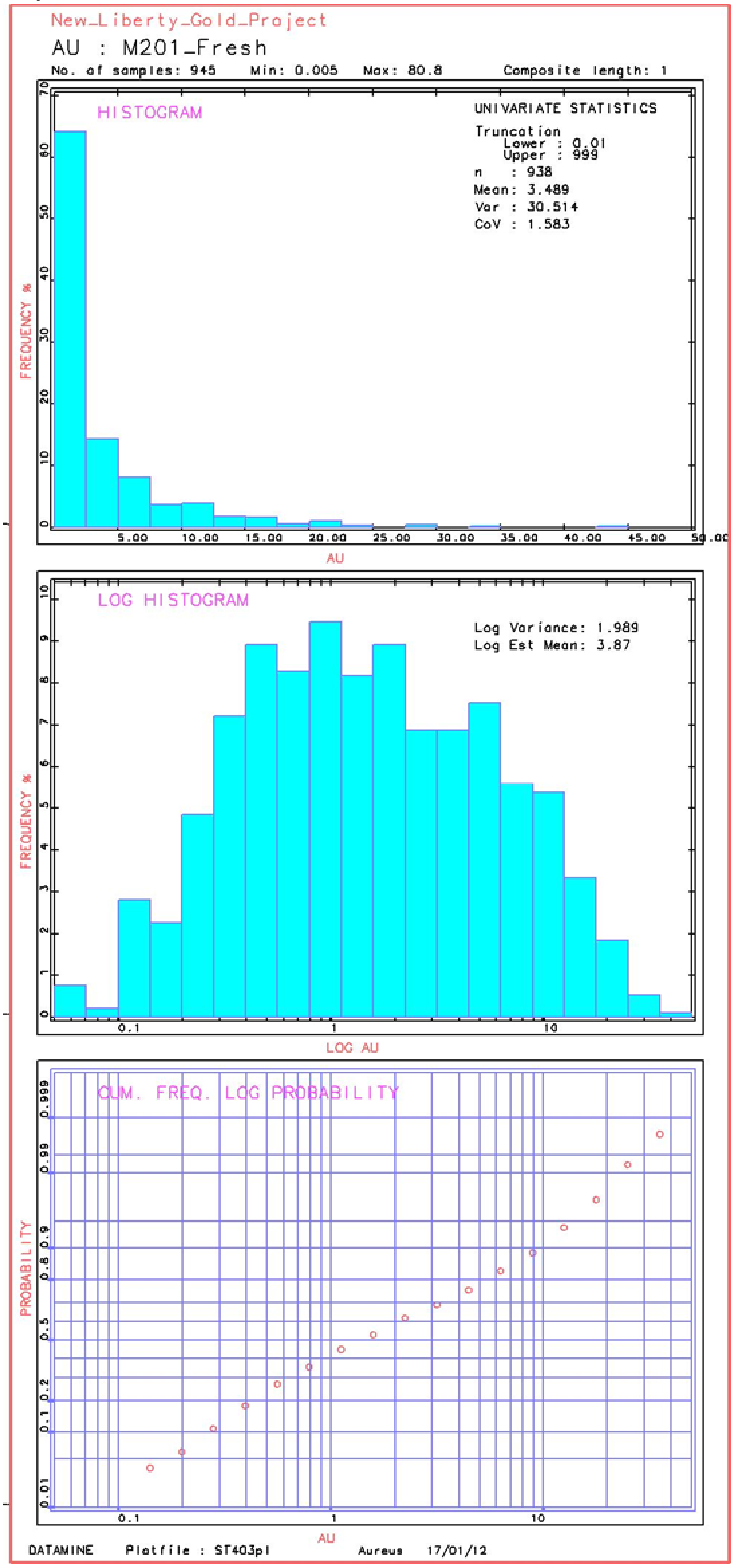



**Figure 14-3: Selected Statistical Charts**

**Latiff**

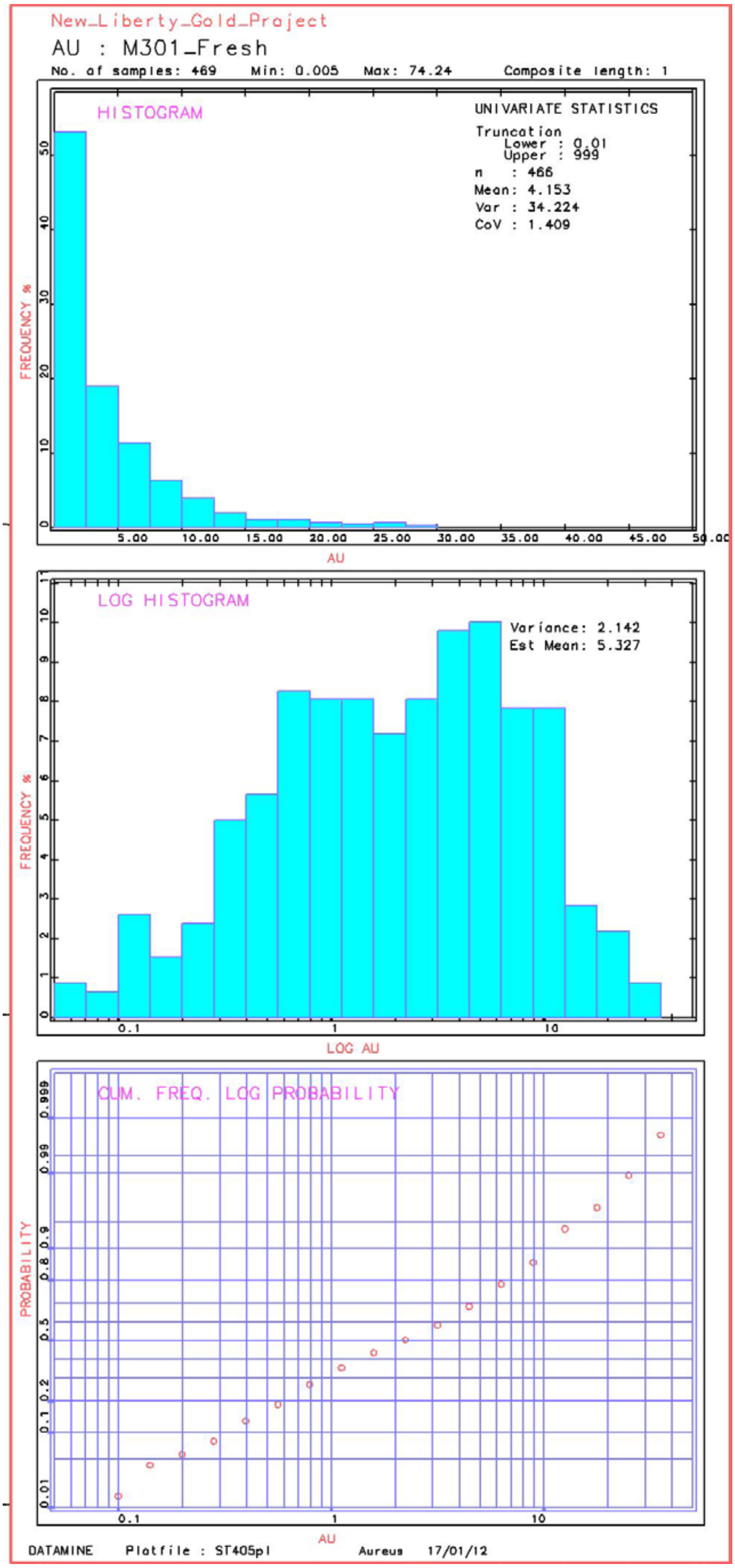



**Figure 14-3: Selected Statistical Charts (Continued)**

**Kinjor**

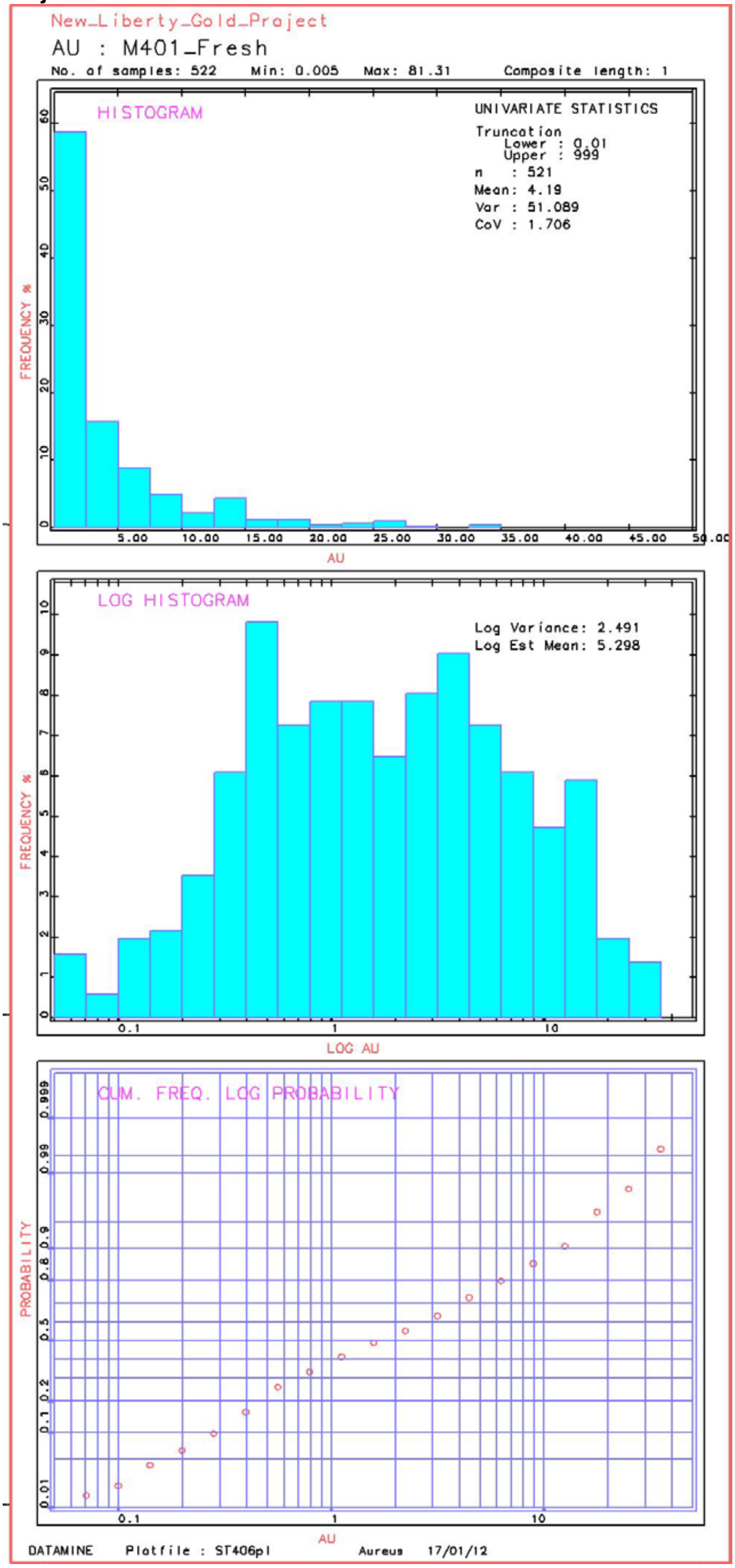



**Figure 14-3: Selected Statistical Charts (Continued)**

**Marvoe**

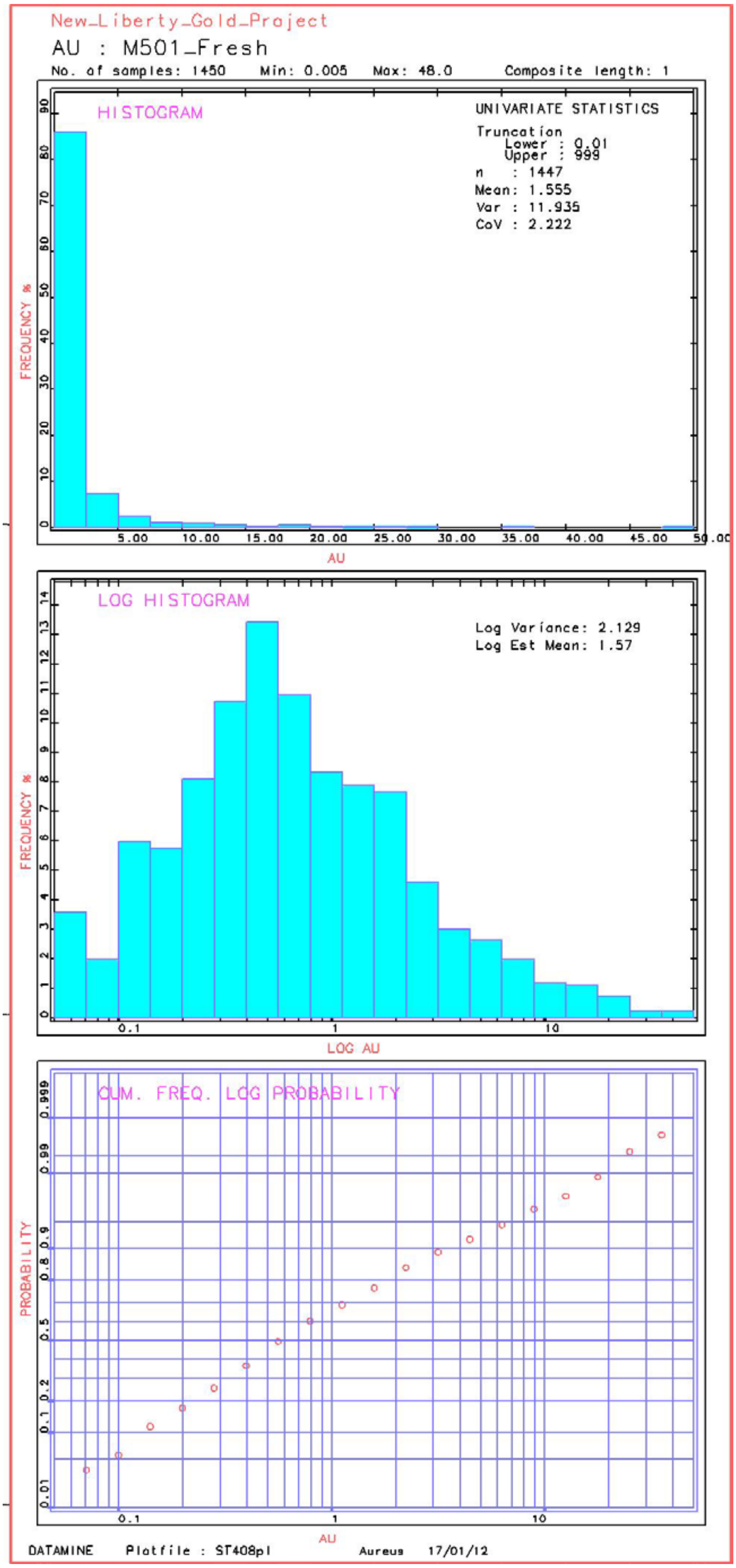



**Figure 14-3: Selected Statistical Charts (Continued)**

## 14.9 Grade Capping

Several steps were followed to assess whether there was a requirement for capping of high grades to reduce any undue influence that these grades might impart during grade estimation,

Initially, the relationships between population mean grades and variances, and the distribution of high grade values were reviewed. This included an analysis of gold grade log probability charts for each mineralised zone, in particular the relative frequency of higher grades (e.g. upper 5%).

Thereafter, composites for each mineralised zone were displayed in both 2D and 3D visualization environments, while highlighting those composites with potentially anomalously high grades. The positions of these high grades, both within individual intersections and between adjacent drill intercepts, were assessed, and careful consideration was given to the likely impact of the high grades during grade estimation.

The list of selected high grade caps is shown in Table 14-10. The grades in zone M503 were not capped.

**Table 14-10: Grade Capping**

| MINZONE | Cap-Grade (g/t Au) | Number of Affected Composites |
|---|---|---|
| M201 | 25 | 7 |
| M301 | 35 | 1 |
| M401 | 35 | 2 |
| M402 | 20 | 5 |
| M501 | 25 | 6 |
| M503 | - | - |
| M504 | 25 | 4 |

## 14.10 Variography

Variographic analysis was conducted for those mineralised zones with sufficient data points (i.e. excluding M504).

Experimental variograms were generated using pairwise relative procedures. Directions of preferred continuity were tested within the planes of each zone, and structures were obtained for each of the strike, down dip and across-plane orientations. In the case of M401 in the Kinjor area, the ‘dip’ orientation represents a south-westerly plunge direction of preferred gold grade continuity.

During variogram modelling (see example charts in Figure 14-4), the position of the nugget variance was fixed using the downhole variogram, and anisotropic variogram parameters were derived using two- or three-structure spherical models (Table 14-11).

In some cases very long ranges were invoked for the final structures to ensure that, where zonal anisotropy is evident, variogram models for all directions reach a common sill. These ranges are well beyond the search neighbourhood during estimation and therefore have no influence on the interpolation.

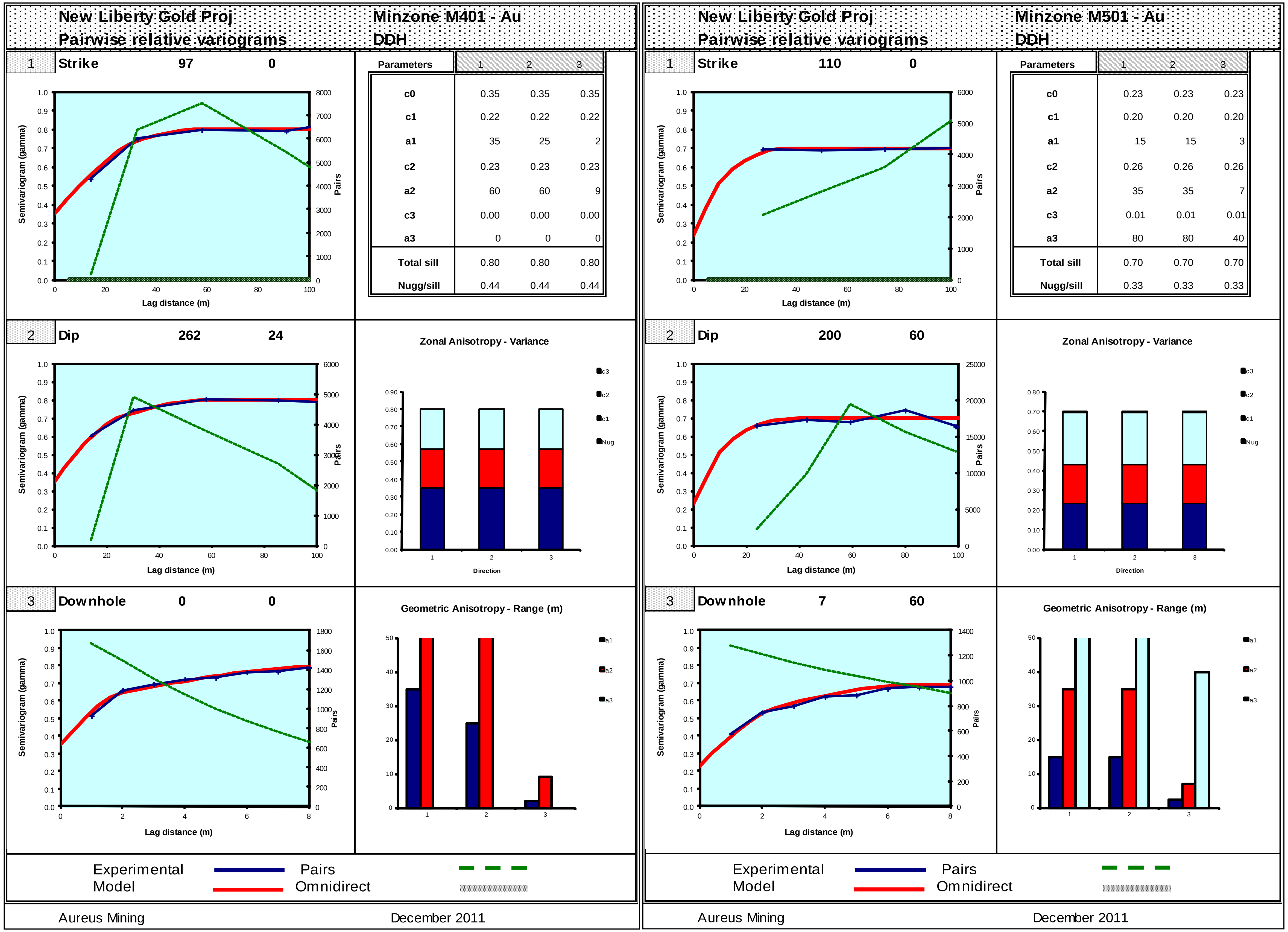



**Figure 14-4:** **Selected Variogram Charts**

The variogram charts show experimental pairwise semi-variance values, whereas the parameters used in the estimation and tabulated below are rescaled to normal variograms.

**Table 14-11: Variogram Parameters**

| Minzone | Value | Nugget Var. | Struct | Spatial Var. | Direction – Ranges (m) | | | Nugg/Sill |
|---|---|---|---|---|---|---|---|---|
| | | | | | Dip | Strike | X-str | |
| M401 | Au | 15.8 | 1 | 9.9 | 35 | 25 | 2 | 44% |
| | | | 2 | 10.4 | 60 | 60 | 9 | |
| M402 | Au | 6.4 | 1 | 6.8 | 35 | 35 | 2.5 | 38% |
| | | | 2 | 2.1 | 80 | 80 | 20 | |
| | | | 3 | 1.7 | 500 | 500 | 100 | |
| M501 | Au | 4.6 | 1 | 4.0 | 15 | 15 | 2.5 | 33% |
| | | | 2 | 5.2 | 35 | 35 | 7 | |
| | | | 3 | 0.2 | 80 | 80 | 40 | |
| M503 | Au | 35 | 1 | 43 | 20 | 20 | 3 | 37% |
| | | | 2 | 17 | 70 | 40 | 25 | |

## 14.11 Grade and Density Interpolation

Gold grades were interpolated from 1 m sample composites, using ordinary kriging for all fresh material zones, excepting zone M504 where inverse distance squared weighting (IDW) was applied. IDW was also used for all weathered material and unconfined SMUS mineralisation outside of the defined mineralised zones, the latter using 2 m composites.

Unsampled intervals in the sample set were assumed to represent intersections of very low probability of gold mineralisation and were therefore assigned a default grade of 0.05 g/t Au, and capping of high grades was applied by mineralised zone, as shown in Table 14.10.

Grade interpolation was conducted into parent cells under hard-bounded zonal control, using search ellipsoids aligned in the local plane orientation of each zone. The local plane orientation parameters of dip direction and dip were interpolated into the cell model by reference to the mineralisation hanging wall and footwall surfaces (Datamine Studio Dynamic Anisotropy function). Cell discretisation was facilitated by applying a 4 × 4 × 2 (XYZ) matrix.

In view of the similarity of the mineralised zone geometries and the distributions of drill intersections, a consistent 35 m × 60 m × 10 m (strike/dip/cross plane) search ellipsoid configuration was applied to all fresh mineralised zones, the exception being Marvoe zone M501, where a reduced dimension of 5 m across the plane of the mineralisation was used to minimise internal smearing within this broad zone.

Figure 14-5 shows a long. section view of the resource model, coloured on gold grades, while the four parts of Figure 14-6 present selected cross-sections through the resource model, one each for the main mineralised zones.

The high number of fresh density measurements facilitated the option to interpolate drillhole density values directly into the cell model. Analysis of the data had shown that there was no advantage in estimating densities separately within individual mineralised zones from those in the host units. In the case of the weathered zone, a single density value of 1.65 $t/m^3$ was applied.

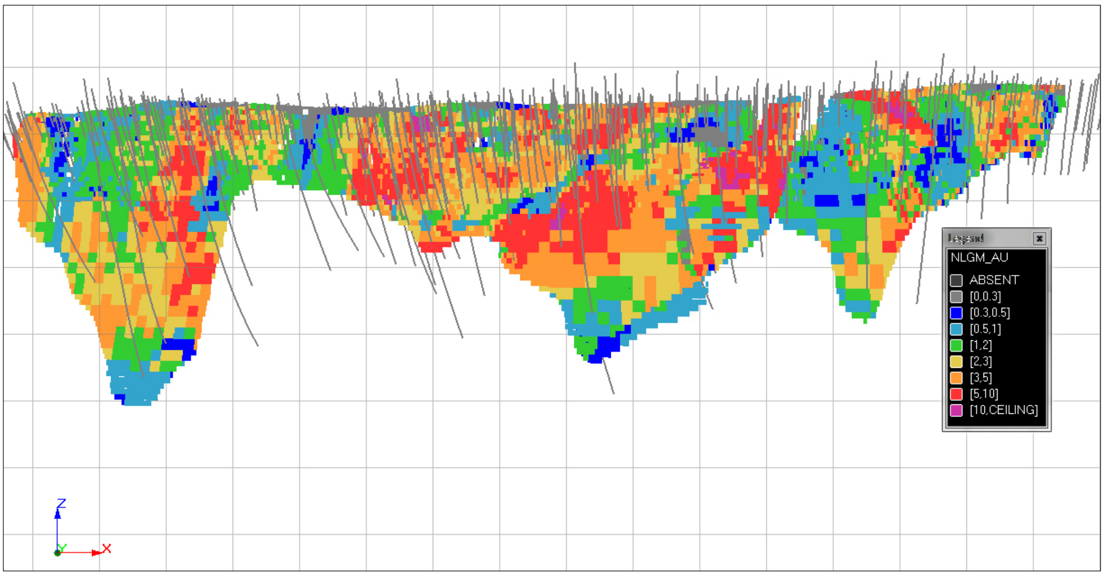



**Figure 14-5: Schematic Resource Model Long. Section Showing Gold Grades**

**Larjor**

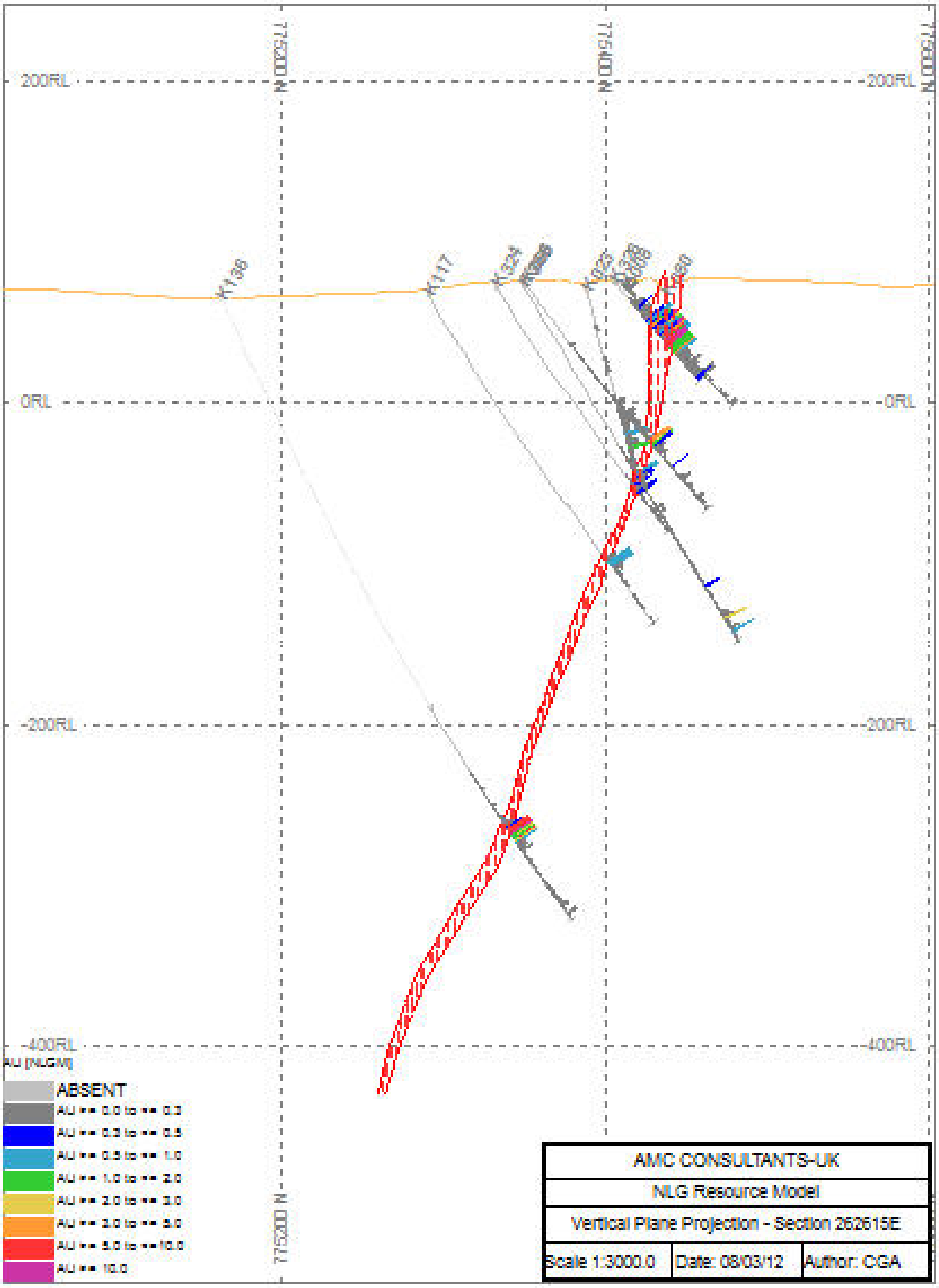



**Figure 14-6: Model Cross-sections with Drillholes Overlay**

**Latiff**

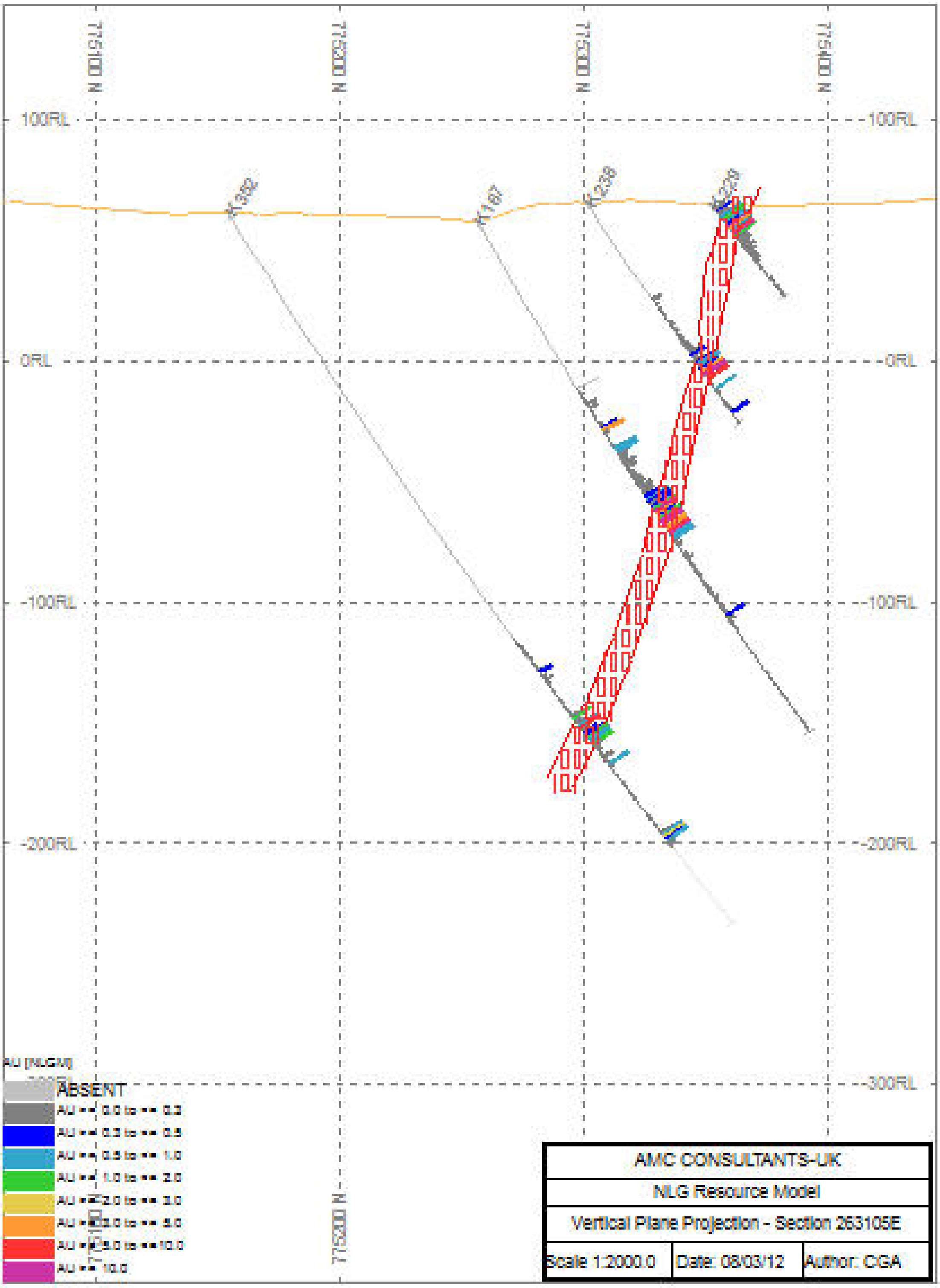



**Figure 14–6: Model Cross-sections with Drillholes Overlay (Continued)**

**Kinjor**

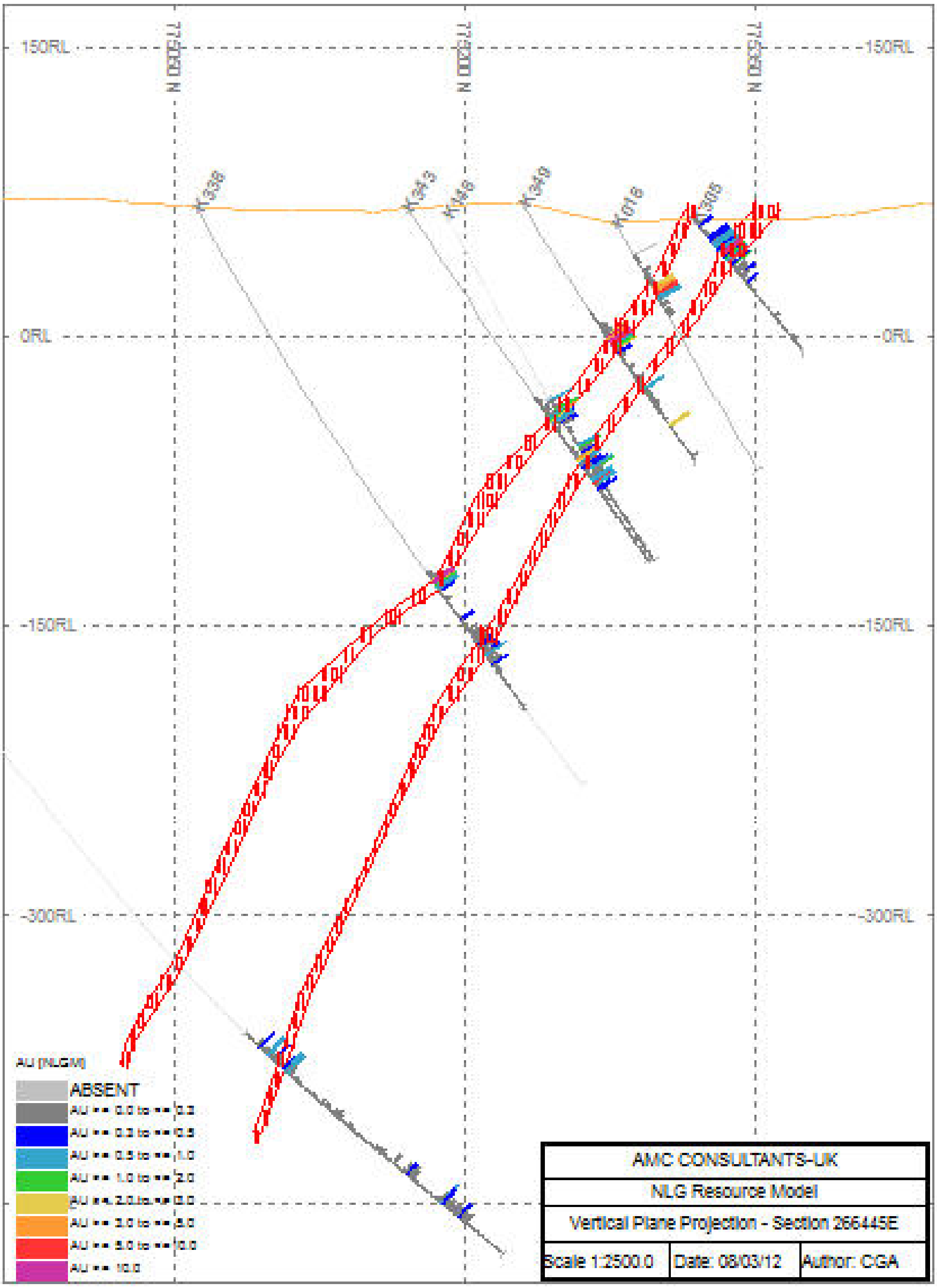



**Figure 14–6: Model Cross-sections with Drillholes Overlay (Continued)**

**Marvoe**

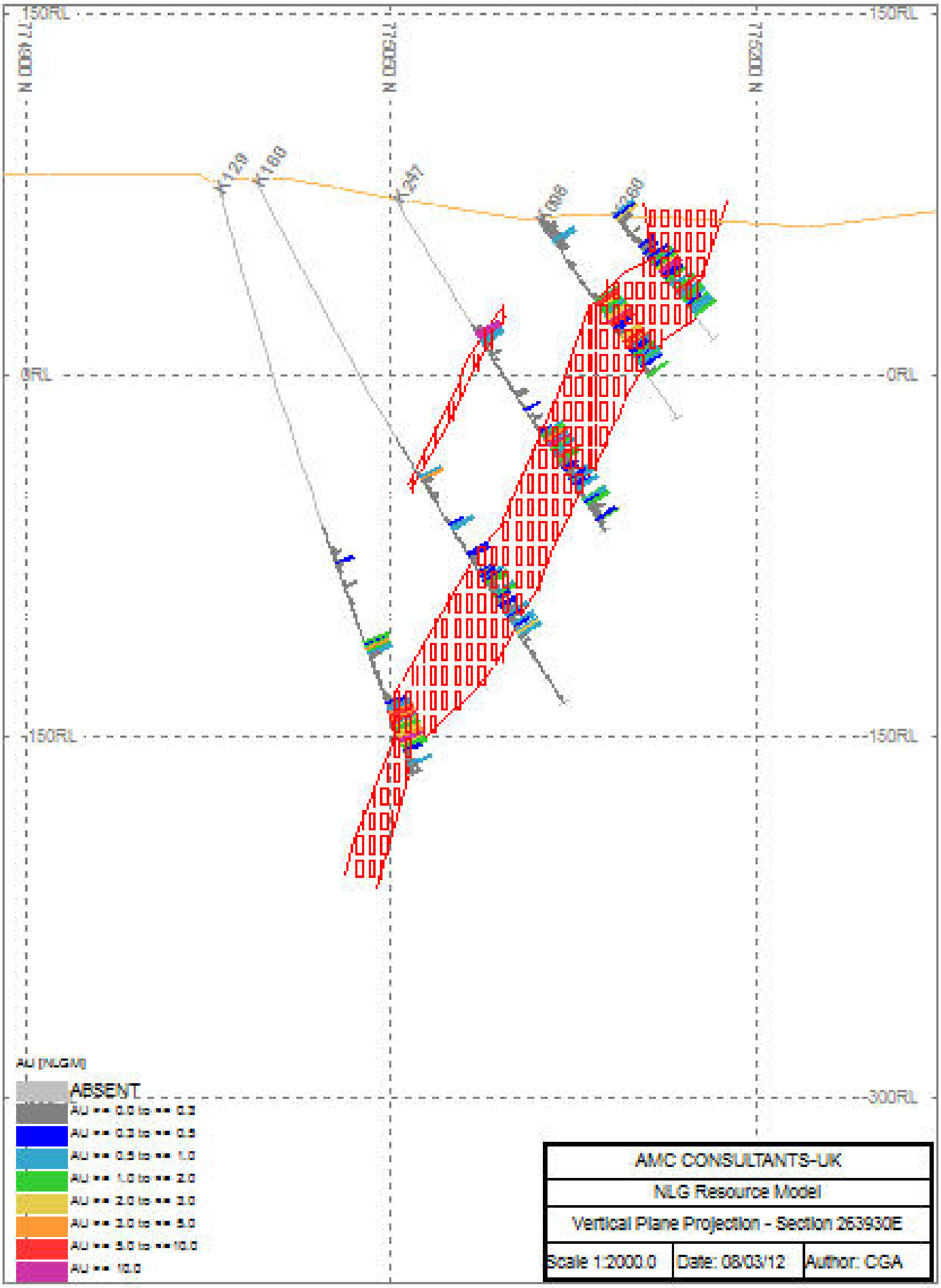



**Figure 14–6: Model Cross-sections with Drillholes Overlay (Continued)**

## 14.12 Resource Classification

Procedures for classifying the reported resources were undertaken within the context of CIM Standards. Notably this took account of:-

- The quality and reliability of raw data (sampling, assaying, surveying).
- AMC's confidence in the geological interpretation.
- The number, spacing and orientation of intercepts through mineralised zones.
- Knowledge of grade continuities gained from observations and geostatistical analyses.
- The likelihood of material meeting economic mining constraints over a range of reasonable future scenarios, and expectations of relatively low selectivity of mining.

In general, the drill spacing is more closely spaced near surface, progressively reducing in density with depth as the drilling tracks each mineralised zone down dip. On drill spacing alone, therefore, the level of confidence in the resource declines with depth.

Ultimately the default criterion used to classify material as Indicated was a maximum drill intersection spacing of 50 m both along strike and down dip. Consequently all candidate material for this classification is located relatively nearer to surface. The drill spacing guideline was, however, locally adjusted on the basis of knowledge of other factors such as data quality and evidence of grade continuity.

A similar procedure was undertaken to determine the presence of any measured mineral resource, in this case using a default reference drill spacing of 2 m × 25 m.

The process of coding the cell model by resource classification was facilitated by presenting the drill intersection data for individual mineralised zones in long section and digitising outline strings to represent the limits of areas considered to constitute, as appropriate indicated or measured mineral resources. The model cells were by default assigned an Inferred mineral resource classification (RESCAT field code 3), and those cells falling within the digitized strings were re-coded as Indicated mineral resources (RESCAT field code 2) or Measured mineral resources (RESCAT field code 1). The distributions of the different category materials are presented in Figure 14-7.

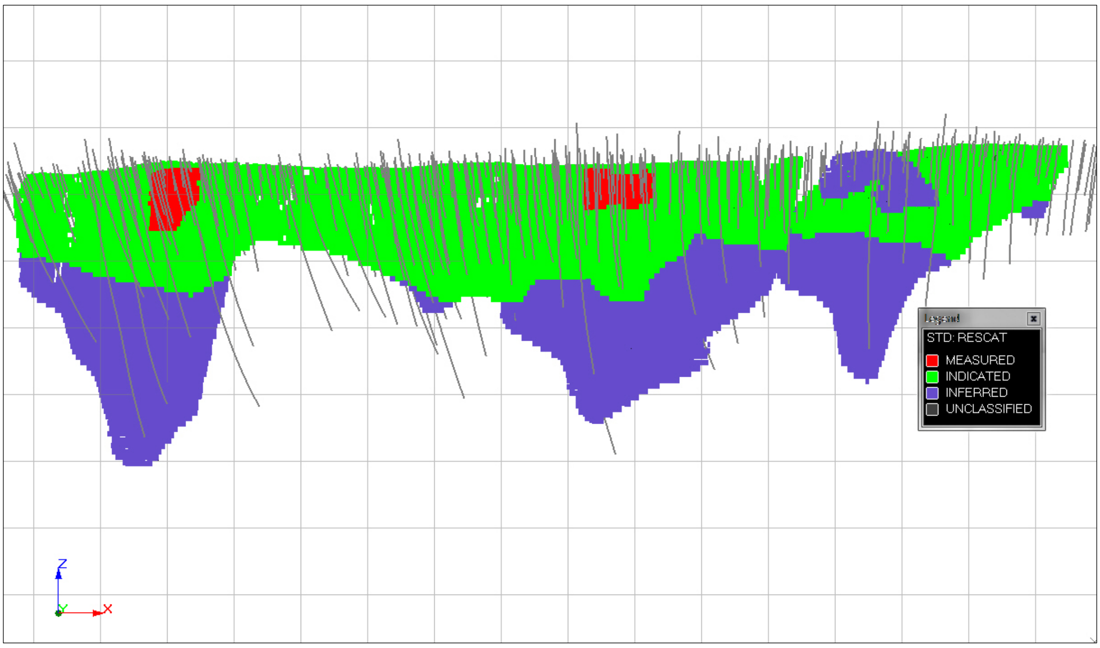



**Figure 14-7: Resource Model 3D Schematic Showing Classification Areas**

## 14.13 Model Validation

Statistics were generated to confirm that interpolated model cell grade field values fall within acceptable bounds. The grade estimates in the cellular models were scrutinized using graphical visualization utilities. Model and drillhole data were overlain and viewed in various 2D section and plan view slices, with colour highlighting of grade or zonal attributes.

Model grade spatial distribution patterns were also reviewed using 3D facilities, presented variously as section planes, point clouds and cell faces. These processes were undertaken repeatedly and continuously throughout the study, during which adjustments and refinements to the model were tested against the predicted consequences of these changes.

The model also progressed from an exclusively inverse distance squared weighting set of estimates to one where ordinary kriging was used on most zones. This evolution provided insight into the effects of different estimation techniques.

At key stages during the study the model was presented to and scrutinized by Aureus professionals, and the final model was also subjected to an AMC internal peer review process.

Almost all of the upper, intensively drilled, part of the model was estimated in the first ellipsoid pass, and the majority of the remaining model was estimated in the first or second pass.

## 14.14 Tonnage-Grade Reporting

Global estimates of tonnes and grade were calculated from the cell model using one method and verified using an alternative reporting procedure.

Aureus' current strategy for evaluation of the Project resources is in the context of open-pit mining. Consequently both the interpretation and the classification processes were conducted with this in mind, and the reporting of the tonnes and grade estimates at 1.0 g/t Au also reflects this line of thought. The near surface portion of the model, within a potentially open-pittable depth, consists dominantly of indicated mineral resources, with relatively small quantities of measured or inferred mineral resources.

It is less clear, what the likely cut-off or economic limits of the deeper, Inferred, classified material would be, or, given the continuous nature of the mineralisation, where the boundary between open pit and underground potential is likely to fall.

For consistency therefore, AMC reported all material at a 1.0 g/t Au cut-off, recognizing that the deeper, almost exclusively Inferred material, may require a higher cut-off grade when subjected to economic evaluation.

The Measured, Indicated and Inferred mineral resources for the project at a 1.0 g/t Au cut-off are presented in Table 14-12. Figure 14-8 summarises the grade and tonnage estimates at a series of cut-off grades.

**Table 14-12: AMC Mineral Resource (as at 1 October 2012)**

| | Measured | | | Indicated | | | Measured and Indicated | | |
|---|---|---|---|---|---|---|---|---|---|
| | Tonnes | Au | | Tonnes | Au | | Tonnes | Au | |
| **Minzone** | (Kt) | (g/t) | (Koz) | (Kt) | (g/t) | (Koz) | (Kt) | (g/t) | (Koz) |
| **M401** | 651 | 4.77 | 100 | 5,468 | 3.88 | 683 | 6,118 | 3.98 | 783 |
| **M402** | | | | 874 | 2.51 | 71 | 874 | 2.51 | 71 |
| **M501** | | | | 2,317 | 2.43 | 181 | 2,317 | 2.43 | 181 |
| **M503** | | | | 486 | 6.93 | 108 | 486 | 6.93 | 108 |
| **M504** | | | | | | | | | |
| **Total** | **651** | **4.77** | **100** | **9,145** | **3.55** | **1,043** | **9,796** | **3.63** | **1,143** |

| | Inferred | | |
|---|---|---|---|
| | Tonnes | Au | |
| **Minzone** | (Kt) | (g/t) | (Koz) |
| **M401** | 3,060 | 3.2 | 314 |
| **M402** | 130 | 3.6 | 15 |
| **M501** | 1,120 | 2.6 | 92 |
| **M503** | 1,300 | 3.6 | 152 |
| **M504** | 120 | 5.1 | 20 |
| **Total** | **5,730** | **3.2** | **593** |

| Key to Minzone Codes | |
|---|---|
| M401 | Larjor + Latiff + Kinjor main zone |
| M402 | Kinjor footwall zone |
| M501 | Marvoe main zone |
| M503 | Marvoe western hanging wall zone |
| M504 | Marvoe central hanging wall zone |

Notes
1.CIM definitions were used for Mineral Resources.
2. A cut-off of 1.0 g/t Au is applied for all zones.
3. Due to rounding, some columns or rows may not add up exactly to the computed totals.
4. Mineral resources that are not mineral reserves do not have demonstrated economic viability

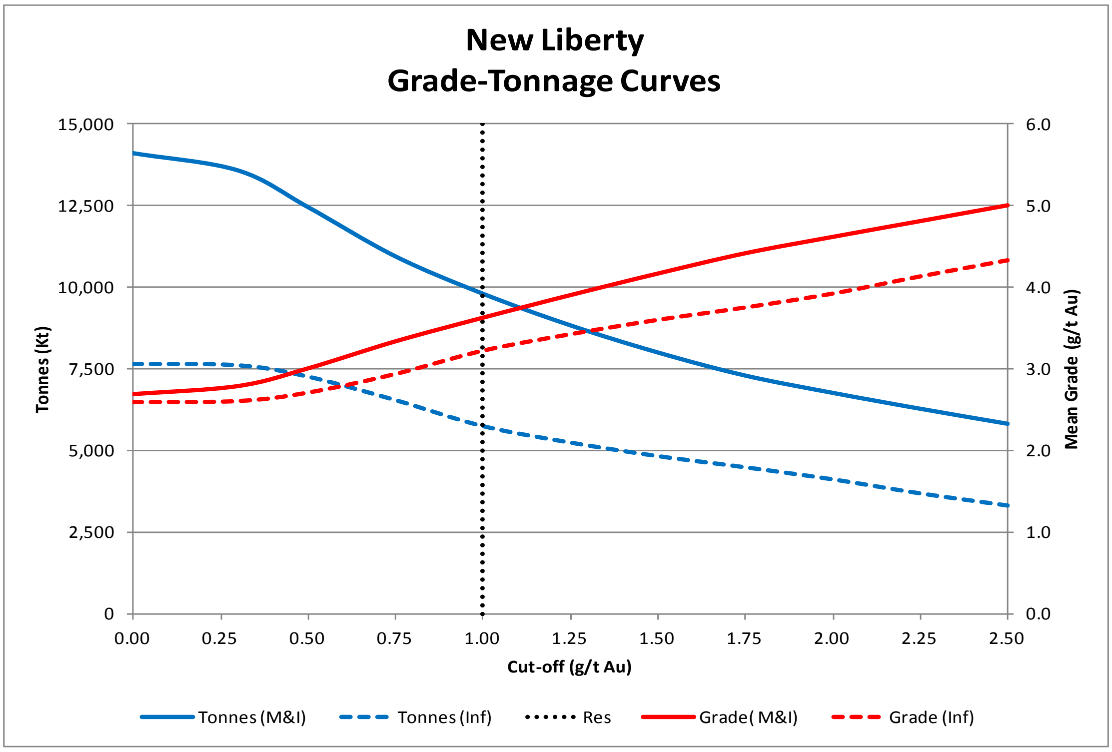



**Figure 14-8: Grade-Tonnage Profiles**

## 14.15 Concluding Remarks

While SRK notes that the Mineral Resource extends below the designed pit and may require to be developed by underground mining, SRK is confident that the Mineral Resource reported above reflects the available data and quality of this and the geological interpretations made and that it has been derived using appropriate and industry standard methods.

As commented earlier in this report Aureus has completed a programme of grade control drilling since this Mineral Resource estimate was derived in October 2012. While the information obtained during this programme has not been used to update this estimate, SRK has reviewed this information and considers that the Mineral Resource remains robust when reviewed in the light of this. Further SRK considers that the information suggests that it may be possible to delineate more continuous zones with at least the same grade, if not slightly higher, as grade control data becomes available and as mining progresses.

SRK is not aware of any environmental, permitting, legal, title, socio-economic, marketing, political or other issues that could have a material impact on the Mineral Resource as presented.

# 15 MINERAL RESERVE ESTIMATES

## 15.1 Mining Approach

The following section summarises the work carried out by AMC to derive a Mineral Reserve estimate for the Project in May 2013. This was based on the same geological block model generated by AMC that formed the basis of the Feasibility Study, and is summarised above, but took account of additional geotechnical data and slope designs developed by AMC subsequent to that.

The study focused on the open pitable portion of the Project and assumed that conventional open-pit gold mining techniques would be employed and that a mining contractor would be engaged to carry out the mining operation.

AMC prepared pit designs for the Project on the basis of pit optimisations carried out using the Mineral Resource estimate adjusted for dilution and ore loss, slope designs based upon the geotechnical assessments, and estimates of the contract mining costs, processing costs, site general and administration costs, metallurgical recovery. The pit optimisations only considered the Measured and Indicated mineral resources. All Inferred mineral resources were treated as waste.

The Whittle programming implementation of the Lerchs Grossman algorithm was used to examine the sensitivities of the pit limits. An optimization shell was selected as a basis for design, staged and final pits were designed and these formed the basis of the mining schedule.

While an updated mining plan has been developed for the Project, as commented upon later in this report, the final open pit design remains as derived in 2013 and consequently the reported Mineral Reserve also remains unchanged.

## 15.2 Geotechnical Assumptions

The geotechnical parameters from the Feasibility Study and Definitive Feasibility Study update were refined to reflect:

- The drilling of 4 new targeted geotechnical drillholes.
- ATV logging of 19 drillholes, confirming the S2 foliation trend, and contributing to the improved definition of joint sets. Analysis of this data also allowed for the measurement of fault orientations.
- Photologging of core to identify areas of poor ground, and assess the spacing of breaks along the foliation. As a result of the investigation, Marvoe was identified as having the poorest ground conditions, a factor which was considered in the wall and slope designs.

This new data was sufficient for AMC to develop a 3D structural model in collaboration with Aureus personnel. Geotechnical domains were redefined, based on four lithological domains (weathered material, and three fresh rock domains), and four structurally distinctive areas. In addition, a further domain was developed in the ultramafic, based on the alteration (high magnetite ultramafic and low magnetite ultramafic).

In order to integrate the new data into the revised geotechnical model, a review of the Aureus logging data was completed:

- Notably, following a report from Orefind (2013) that identified some cases where the orientation reference line had been poorly transferred onto the core, AMC filtered the Aureus database by removing all suspected poor quality data. The results of the data filtering process, confirmed the absence of S1 foliation fabric that was used during the FS study.
- Joint sets were assessed using the new database, indicating seven joint sets with a maximum of three joint sets in any particular domain.

Following the identification of errors and inaccuracies in the database of RQD provided by Aureus to AMC, a new RQD database was developed. Comparisons were drawn between lithologies and pit areas, showing consistency across the pit areas, with a greater amount of poor ground in the low magnetite ultramafic, and also the granitioids.

The variation in discontinuity condition ratings was analysed across the geotechnical domains, and a revised discontinuity strength was developed for each domain. Subequent numerical analysis considered the variation in strength across the geotechnical domains.

Finally, rockmass rating parameters were analysed throughout the different pit areas, and the lithological domains.

Bench scale kinematic and deterministic analysis showed:

- The factor-of-safety (FOS) stays above the acceptance criteria for planar failure analysis, when the bench face angle (BFA) is below 75°. Planar failure analysis of the southern walls shows design parameters to be acceptable throughout all of the domains.
- Toppling analysis indicated that whilst toppling failure may occur in localized walls with specific BFAs, it is not considered a major risk to bench scale slope stability.
- Wedge analysis shows that the FOS decreases below the acceptance criteria when the BFA exceeds 75° in the SW of Marvoe.

Berm capacity was analysed both at bench and inter-ramp scale. At bench scale, the only wedges that are likely to exceed the berm capacity have a FOS that is above the acceptance criteria. Inter-ramp wedge analysis indicated that most large wedges are successfully contained by a 15 m geotechnical berm. Those with a failure volume larger than the capacity, have a FOS above the acceptance criteria.

Using the bench scale and inter-ramp-scale kinematic and deterministic analysis, a bench configuration was developed. Numerical modelling of the overall slope stability from this bench configuration highlighted that in general, overall stability is above the acceptance criteria in drained conditions. However, some areas of particular slopes may be unstable under undrained conditions. The slope depressurisation programme will be confirmed following the completion of the on-going hydrogeological testing programme.

Numerical modelling has highlighted that inter-ramp stability is of concern if the following slopes are not depressurised:

- Kinjor northern and southern walls, with particular attention to the south wall in the east of Kinjor where there is a steep overall angle in the fresh rock (OSA=56°).
- Marvoe northern walls

Analysis of the waste dump designs indicated a stable design, assuming drained conditions and ± 20% of the assumed alluvium material strength, based on field logging.

## 15.3 Pit Optimisation

### 15.3.1 Introduction

The ore body model after adjustments to reflect dilution and ore loss was exported to the optimisation module where the optimisation analysis was conducted. The optimisation process evaluates the combinations of ore blocks and waste blocks which generate the highest cash surplus for any given set of economic and pit slope parameters. By varying an economic parameter, such as the gold price, a series of nested pits can be generated. The nested pit shells can be evaluated to test the deposits sensitivities to changes in economic parameters.

The optimisation process requires the input of certain data including the resource model, dilution, mining, process plant, general and administration costs, and a geotechnical assessment of slope angles at which the pit can be successfully mined. Appropriate unit costs and input data specific to the Project were provided by Aureus and/or the Project's retained consultants at the time the optimisation work was undertaken.

For the purposes of the optimisations, capital costs, depreciation, amortization, royalties, taxes and other finance charges have been excluded.

### 15.3.2 Dilution and Ore Loss

Due to the steeply dipping nature of the orebody it lends itself to controlled drill-and-blast methods and mining dilution control. For mine planning purposes the orebody was diluted to reflect the expected mining selectivity.

A minimum mining width of 2.5 m was applied to the resource model. Any zones of continuous ore grade blocks that fell below this specified minimum width were expanded into the highest grade adjacent material until the minimum width was achieved. The grade of this expanded width was compared to the cut-off grade, and included, or excluded as potential ore zones accordingly.

There are areas of the New Liberty resource where parallel zones of mineralisation occur. The areas of waste that lie between "ore zones" were tested to ensure that the middling of waste meets the minimum mining width conditions. If the area of waste between ore zones was below the 2.5 metres minimum width, then the ore zones on either side were combined to include the waste as internal dilution. The combined ore zone, including the internal dilution was compared to the cut-off grade, and included, or excluded as potential ore zones accordingly.

A dilution skin of 0.5 metres was added to all potential ore zone contacts to reflect the practicalities of grade control and mining with the size of the anticipated mining fleet.

The effect of dilution and ore loss varies locally within the deposit as wider zones are less affected than narrow areas of mineralisation. The effect on each mineralised zone is summarised in Table 15-1.

**Table 15-1: Dilution and Ore Loss Effect**

| | | Always Ore | | | Dilution | | | Ore Loss | | | In-pit Resource | | | Reserve | | | NET EFFECT | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **RESOURCE CATEGORY** | **MINERALISATION ZONE** | **Tonnes (kt)** | **Grade (g/t)** | **Au Ounces (koz)** | **Tonnes (kt)** | **Grade (g/t)** | **Au Ounces (koz)** | **Tonnes (kt)** | **Grade (g/t)** | **Au Ounces (koz)** | **Tonnes (kt)** | **Grade (g/t)** | **Au Ounces (koz)** | **Tonnes (kt)** | **Grade (g/t)** | **Au Ounces (koz)** | **Tonnes (kt) % Diff.** | **Grade (g/t) % Diff.** | **Au Ounces (koz) % Diff.** |
| Measured | M401 | 648 | 4.8 | 99 | 57 | 0.1 | 0 | 5 | 4.2 | 1 | 653 | 4.8 | 100 | 705 | 4.4 | 99 | 8% | -8% | 0% |
| | M402 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | M501 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | M503 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | M504 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Measured Total** | | **648** | **4.8** | **99** | **57** | **0.1** | **0** | **5** | **4.2** | **1** | **653** | **4.8** | **100** | **705** | **4.4** | **99** | **8%** | **-8%** | **0%** |
| Indicated | M401 | 4,102 | 4.0 | 531 | 506 | 0.1 | 2 | 80 | 1.8 | 5 | 4,182 | 4.0 | 536 | 4,608 | 3.6 | 533 | 10% | -10% | 0% |
| | M402 | 657 | 2.3 | 49 | 79 | 0.2 | 0 | 35 | 1.6 | 2 | 692 | 2.3 | 51 | 736 | 2.1 | 50 | 6% | -8% | -3% |
| | M501 | 1,812 | 2.3 | 137 | 133 | 0.3 | 1 | 49 | 1.5 | 2 | 1,862 | 2.3 | 139 | 1,945 | 2.2 | 138 | 5% | -5% | -1% |
| | M503 | 459 | 7.0 | 103 | 40 | 0.1 | 0 | 4 | 3.4 | 0 | 463 | 6.9 | 103 | 499 | 6.4 | 103 | 8% | -7% | 0% |
| | M504 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Indicated Total** | | **7,029** | **3.6** | **820** | **759** | **0.2** | **4** | **169** | **1.7** | **9** | **7,198** | **3.6** | **829** | **7,788** | **3.3** | **824** | **8%** | **-8%** | **-1%** |
| **Total Measured + Indicated** | | **7,677** | **3.7** | **919** | **816** | **0.2** | **5** | **174** | **1.7** | **10** | **7,851** | **3.7** | **99** | **8,493** | **3.4** | **923** | **8%** | **-8%** | **-1%** |
| Inferred | M401 | 31 | 1.3 | 1 | 7 | 0.2 | 0 | - | - | - | 31 | 1.3 | 1 | 38 | 1.1 | 1 | 24% | -16% | 4% |
| | M402 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| | M501 | 10 | 1.3 | 0 | 1 | 0.1 | 0 | - | - | - | 10 | 1.3 | 0 | 11 | 1.2 | 0 | 8% | -7% | 1% |
| | M503 | 27 | 3.7 | 3 | 1 | 0.3 | 0 | - | - | - | 27 | 3.7 | 3 | 28 | 3.5 | 3 | 4% | -4% | 0% |
| | M504 | 91 | 5.9 | 17 | 15 | 0.1 | 0 | 14 | 1.5 | 1 | 105 | 5.3 | 18 | 106 | 5.1 | 17 | 0% | -4% | -4% |
| **Inferred Total** | | **159** | **4.3** | **22** | **24** | **0.1** | **0** | **14** | **1.5** | **1** | **174** | **4.1** | **23** | **183** | **3.8** | **22** | **5%** | **-8%** | **-3%** |

The overall effect on the Measured and Indicated portion of the Mineral Resource was to add 10.4% in tonnes at a low grade which adds 0.5% to the contained metal. The ore loss amounted to 2.2% of the Measured and Indicated portion of the Mineral Resource. However this is largely from low-grade areas and hence only containing 1.1% of the contained gold. The net effect was to add 8.2% to the tonnage whilst losing 0.6% of the contained metal and hence the grade dropped by 8.1%.

### 15.3.3 Pit Slope Parameters

The approach taken for the geotechnical design was to develop stable overall slope angles and manageable bench and inter-ramp conditions on the assumption that the walls can be adequately drained. Geotechnical domains are based on the lithological interpretations, within the three domains identified:

- Weathered, near surface material;
- Northern Walls; and
- Southern Walls.

A summary of the mining bench parameters developed based on kinematic analysis is given in Table 15-2.

**Table 15-2: Pit Slope Design Domains**

| Domain | Bench Face Angle (°) | Batter Height (m) | Berm Width(m) |
|---|---|---|---|
| Weathered | 45° | 10 | 5 |
| Northern Walls | 70° | 20 | 8.5 |
| Southern Walls | 75° | 20 | 8.5 |

Numerical analysis of the overall slopes (Section 2.5.2) showed that 48° is an acceptable value for the upper limit of the overall slope angle.

The detailed design of the slopes were refined through the pit design process to reflect the slope design domains split to eight (8) different slope regions and to reflect the height of the slope and the integration of the access ramps required. The final design slope regions are summarized in Table 15-3 split according to the zones shown in Figure 15-1.

**Table 15-3: Final Pit Design Slope Design Parameters**

| Pit Region | Bench Face Angle (°) | Bench Height (m) | Berm Width (m) | Ramp Configuration * | OSA Fresh (°) | OSA Weathered (°) |
|---|---|---|---|---|---|---|
| Larjor North | 70 | 20 | 8.5 | Single and Double | 45 | 38 |
| Larjor South | 75 | 20 | 8.5 | Single and Double | 49 | 38 |
| Latiff North | 70 | 20 | 8.5 | Single and Double | 48 | 38 |
| Latiff South | 75 | 20 | 8.5 | Single and Double | 50 | 38 |
| Kinjor North | 70 | 20 | 8.5 | Single and Double | 46 | 38 |
| Kinjor South | 75 | 20 | 8.5 | Single and Double | 49 | 38 |
| Marvoe North | 70 | 20 | 8.5 | Single and Double | 44 | 38 |
| Marvoe South | 75 | 20 | 8.5 | Single and Double | 47 | 38 |

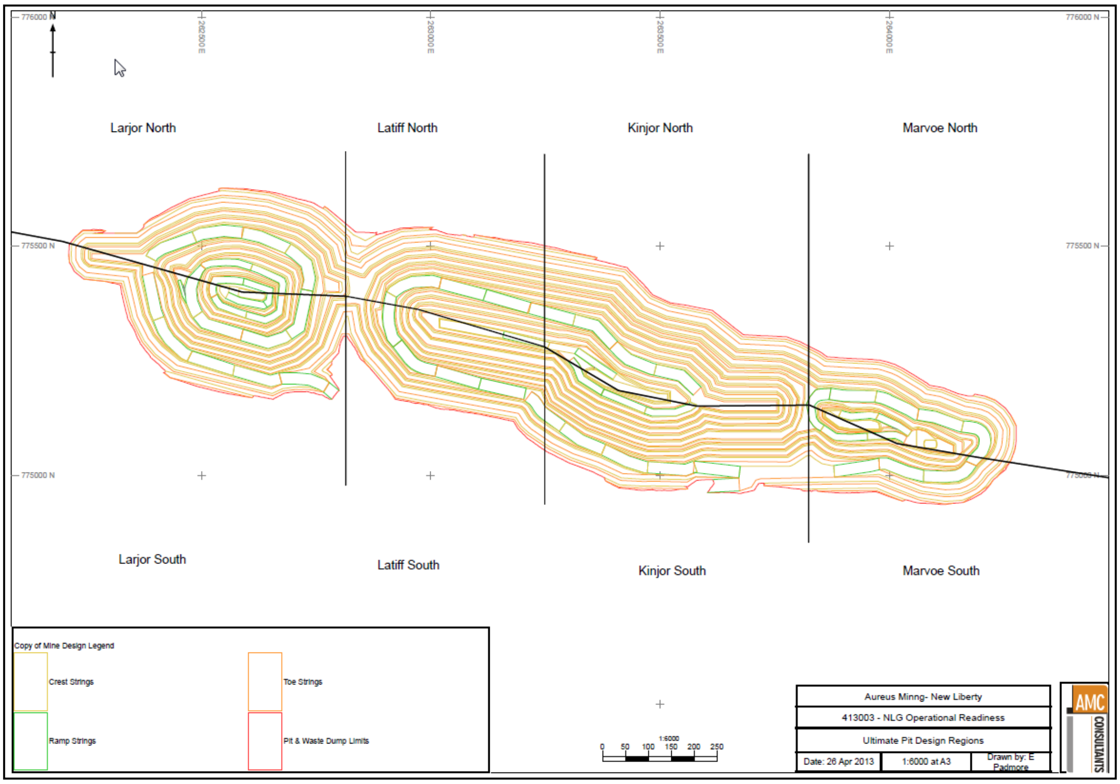



**Figure 15-1: Design Sectors for the Pit Design**

### 15.3.4 Mining Costs

The mine design, waste dump designs and mining schedule from the Definitive Feasibility Study and haulage studies for a revised waste dump design were used by Aureus to calculate a revised set of mining rates for the Project. Using these new mining rates the average cost of mining used in the optimisation was USD2.52/t mined.

### 15.3.5 Processing and General and Administration Costs

DRA Engineering (DRA) provided the metallurgical processing costs and recovery factors per tonne of ore, and Aureus provided the overall general and administration (G&A) costs based on a nominal throughput of 1.1 Mtpa of fresh ore. The recovery and cost factors were also assumed to apply to the weathered near surface ore which contributes 2% of the Project ounces. A breakdown of the costs used in the optimisation analysis is shown in Table 15-4.

**Table 15-4: Processing, General and Administration Costs for Pit Optimization**

| Cost Centre | Cost/milled tonne USD |
|---|---:|
| Processing | 22.57 |
| Administration | 6.25 |
| **Total** | **28.82** |

### 15.3.6 Gold Price

A base case gold price of USD 1,300 / ounce was used in the evaluation of pit optimisations and the setting of the cut-off grade. This value was selected in discussion with Aureus.

### 15.3.7 Cut-off Grade

A breakeven cut-off grade was determined based upon the estimates for metallurgical recovery, processing and general administration costs.

The key parameters applied in the determination of the cut-off grade were:

- A gold price of USD 1,300/ounce (USD41.80/gramme);
- A processing and administration cost of USD28.82 per tonne of ore treated; and
- A metallurgical recovery of 93%.

The break even cut-off grade is also commonly called the pit rim cut-off grade. This is the grade at which the value of the material in the back of a haul truck at the pit rim is the same whether it is delivered to the ROM pad or taken to the waste dump.

The calculation used is:

$$\text{Breakeven cut-off grade} = \frac{\text{Processing Cost+Administration Cost}}{\text{Gold Price*Metallurgical Recovery}}$$

$$= \frac{22.57+6.25}{41.80*93\%} = 0.8 \text{ g/t Au}$$

### 15.3.8 Optimisation Results

The results of the open pit optimization are shown in Figure 15-2 and Table 15-5.

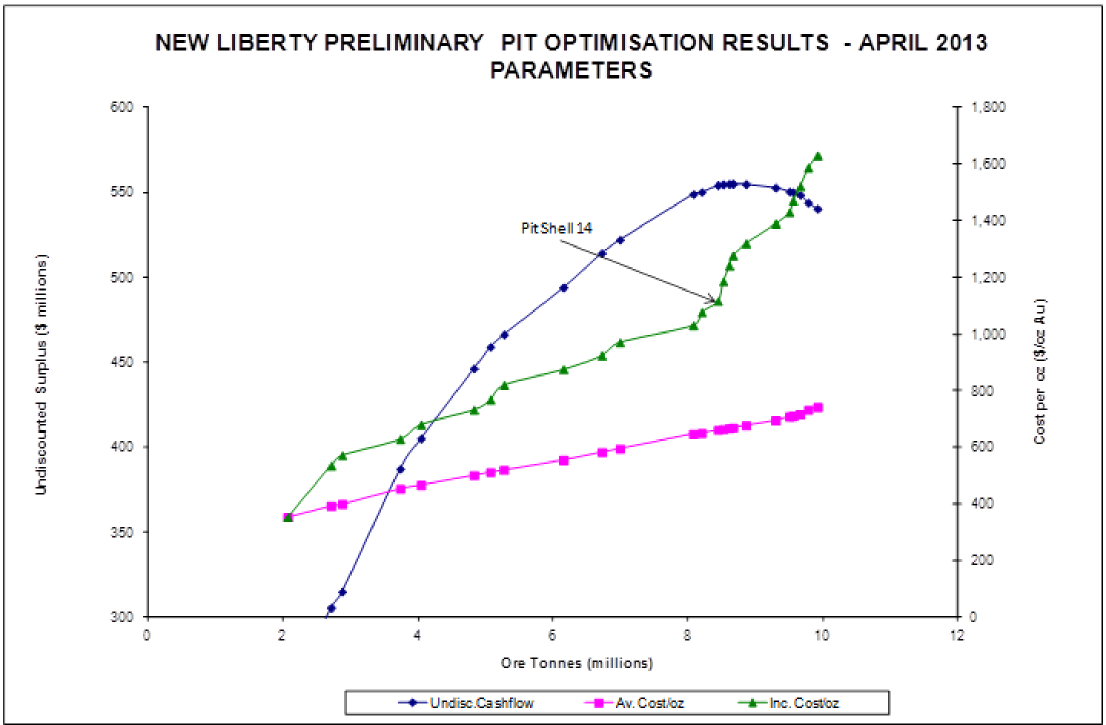



**Figure 15-2: Pit Optimisation Results**

**Table 15-5: Whittle Pit Optimisation Results**

| Evaluated at USD1,300/Au oz | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Base Shell Data | | | | | | | | | Value | | | | | | | | | | Incremental |
| Pit Shell | Metal Price | Bench | Total Ore | | | Waste | Total Rock | Strip Ratio | MCost Per Tonne of ore | Incr. MCost Per Tonne of ore | Rec. Au oz | Cost per Au Ounce | Processing Cost | Mining Cost | Revenue | Undiscounted Cash flow | Discounted Best Cash flow | Discounted Worst Cash flow | Cost per Ounce |
| | (USD/oz) | (m RL) | Tonnes (Mt) | Au (g/t) | Rec. Au (g/t) | Tonnes (Mt) | Tonnes (Mt) | W:O | (USD/t) | (USD/t) | (koz) | (USD/oz) | (USDm) | (USDm) | (USDm) | (USDm) | (USDm) | (USDm) | (USD/oz) |
| 1 | 500.0 | -90 | 2.07 | 4.26 | 3.96 | 10.6 | 12.7 | 5.1 | 15.3 | 15.3 | 263 | 352 | -61 | -32 | 342 | 250 | 216 | 216 | 352 |
| 2 | 550.0 | -130 | 2.71 | 4.15 | 3.86 | 18.1 | 20.8 | 6.7 | 19.1 | 19.1 | 336 | 392 | -80 | -52 | 437 | 305 | 259 | 257 | 535 |
| 3 | 600.0 | -130 | 2.88 | 4.06 | 3.78 | 19.0 | 21.8 | 6.6 | 18.9 | 18.9 | 349 | 399 | -85 | -54 | 454 | 315 | 266 | 262 | 571 |
| 4 | 650.0 | -150 | 3.74 | 4.09 | 3.80 | 34.9 | 38.7 | 9.4 | 25.8 | 25.8 | 457 | 453 | -110 | -96 | 594 | 387 | 316 | 306 | 628 |
| 5 | 700.0 | -150 | 4.05 | 4.02 | 3.73 | 38.8 | 42.9 | 9.6 | 26.4 | 26.4 | 486 | 466 | -119 | -107 | 631 | 405 | 327 | 315 | 680 |
| 6 | 750.0 | -150 | 4.83 | 3.87 | 3.59 | 50.1 | 55.0 | 10.4 | 28.3 | 38.0 | 558 | 501 | -143 | -137 | 726 | 446 | 352 | 331 | 732 |
| 7 | 800.0 | -170 | 5.08 | 3.84 | 3.57 | 54.2 | 59.3 | 10.7 | 29.1 | 40.7 | 582 | 511 | -150 | -148 | 757 | 459 | 359 | 336 | 768 |
| 8 | 850.0 | -170 | 5.28 | 3.79 | 3.52 | 56.7 | 61.9 | 10.7 | 29.3 | 41.7 | 597 | 519 | -156 | -154 | 776 | 466 | 363 | 337 | 819 |
| 9 | 900.0 | -170 | 6.16 | 3.60 | 3.35 | 68.0 | 74.2 | 11.1 | 30.1 | 36.7 | 662 | 554 | -182 | -185 | 861 | 494 | 378 | 340 | 874 |
| 10 | 950.0 | -190 | 6.73 | 3.57 | 3.31 | 80.3 | 87.0 | 11.9 | 32.4 | 41.5 | 716 | 582 | -199 | -218 | 931 | 514 | 388 | 338 | 924 |
| 11 | 1000.0 | -190 | 7.00 | 3.54 | 3.29 | 85.9 | 92.8 | 12.3 | 33.3 | 44.5 | 740 | 594 | -207 | -233 | 962 | 522 | 392 | 337 | 970 |
| 12 | 1050.0 | -210 | 8.08 | 3.48 | 3.23 | 112.2 | 120.3 | 13.9 | 37.6 | 51.8 | 839 | 646 | -239 | -304 | 1091 | 549 | 403 | 326 | 1030 |
| 13 | 1100.0 | -210 | 8.21 | 3.45 | 3.20 | 113.0 | 121.2 | 13.8 | 37.3 | 51.7 | 845 | 649 | -242 | -306 | 1098 | 550 | 403 | 325 | 1076 |
| **14** | **1150.0** | **-210** | **8.45** | **3.44** | **3.19** | **119.5** | **127.9** | **14.1** | **38.2** | **60.2** | **866** | **661** | **-249** | **-323** | **1126** | **554** | **404** | **319** | **1116** |
| 15 | 1200.0 | -210 | 8.53 | 3.42 | 3.17 | 120.3 | 128.8 | 14.1 | 38.1 | 59.5 | 870 | 663 | -252 | -325 | 1132 | 555 | 405 | 318 | 1185 |
| 16 | 1250.0 | -210 | 8.62 | 3.40 | 3.16 | 121.4 | 130.1 | 14.1 | 38.1 | 58.8 | 875 | 666 | -254 | -328 | 1137 | 555 | 405 | 317 | 1240 |
| 17 | 1300.0 | -210 | 8.67 | 3.40 | 3.15 | 122.4 | 131.1 | 14.1 | 38.2 | 46.6 | 878 | 668 | -256 | -331 | 1142 | 555 | 405 | 316 | 1276 |
| 18 | 1350.0 | -210 | 8.86 | 3.37 | 3.12 | 126.1 | 135.0 | 14.2 | 38.4 | 52.9 | 890 | 677 | -262 | -341 | 1157 | 555 | 405 | 310 | 1319 |
| 19 | 1400.0 | -210 | 9.30 | 3.29 | 3.05 | 132.7 | 142.0 | 14.3 | 38.5 | 41.3 | 912 | 694 | -275 | -358 | 1186 | 553 | 404 | 299 | 1389 |
| 20 | 1450.0 | -210 | 9.51 | 3.28 | 3.04 | 139.8 | 149.3 | 14.7 | 39.7 | 53.4 | 930 | 708 | -281 | -378 | 1209 | 550 | 403 | 293 | 1430 |
| 21 | 1500.0 | -210 | 9.56 | 3.27 | 3.03 | 140.4 | 149.9 | 14.7 | 39.7 | 53.8 | 932 | 710 | -282 | -379 | 1212 | 550 | 403 | 292 | 1471 |
| 22 | 1550.0 | -210 | 9.67 | 3.26 | 3.02 | 143.4 | 153.1 | 14.8 | 40.1 | 56.7 | 939 | 716 | -285 | -387 | 1221 | 548 | 402 | 288 | 1521 |
| 23 | 1600.0 | -230 | 9.79 | 3.27 | 3.04 | 151.7 | 161.5 | 15.5 | 41.8 | 74.5 | 955 | 731 | -289 | -409 | 1242 | 544 | 400 | 280 | 1587 |
| 24 | 1650.0 | -230 | 9.92 | 3.27 | 3.03 | 156.7 | 166.6 | 15.8 | 42.6 | 103.7 | 966 | 741 | -293 | -423 | 1256 | 540 | 399 | 274 | 1629 |

### 15.3.9 Selection of Optimum Pit Shell

The graph in Figure 15-2 shows the 24 optimisation shells that were generated using incrementally increasing revenue factors and hence the gold price used to generate the optimisation shell. All optimisation shells were then evaluated using the base costs and a gold price of USD1,300/oz. The optimisation pit tonnage and revenue estimate details are shown in Table 15-5.

The results of undiscounted and discounted cash flow analysis are used to select the optimum pit for design purposes. After analysis and discussion with Aureus Pit shell 14 (revenue factor = 0.9) was chosen as the optimal pit for design purposes as the incremental production cost rises steeply after this pit shell.  Pit shell 14 produces:

- 8.5 Mt at 3.4 g/t
- A stripping ratio of 14.1
- Total waste tonnes of 119 Mt
- 866 koz of recovered gold (at 93% recovery)

## 15.4 Open Pit Design

The ultimate pit design comprised two "joined" pits, the Larjor pit to the west, and the Latiff and Kinjor zone (Latkin) in the centre and connected with the Marvoe zone to the east. The Larjor pit is separated from Latkin pit by a poorly mineralized area which forms a saddle between the two. A less significant saddle exisits between Kinjor and Marvoe zones at depth and the pits are accessed via separate ramps.

The final pit design is shown in Figure 15-3.

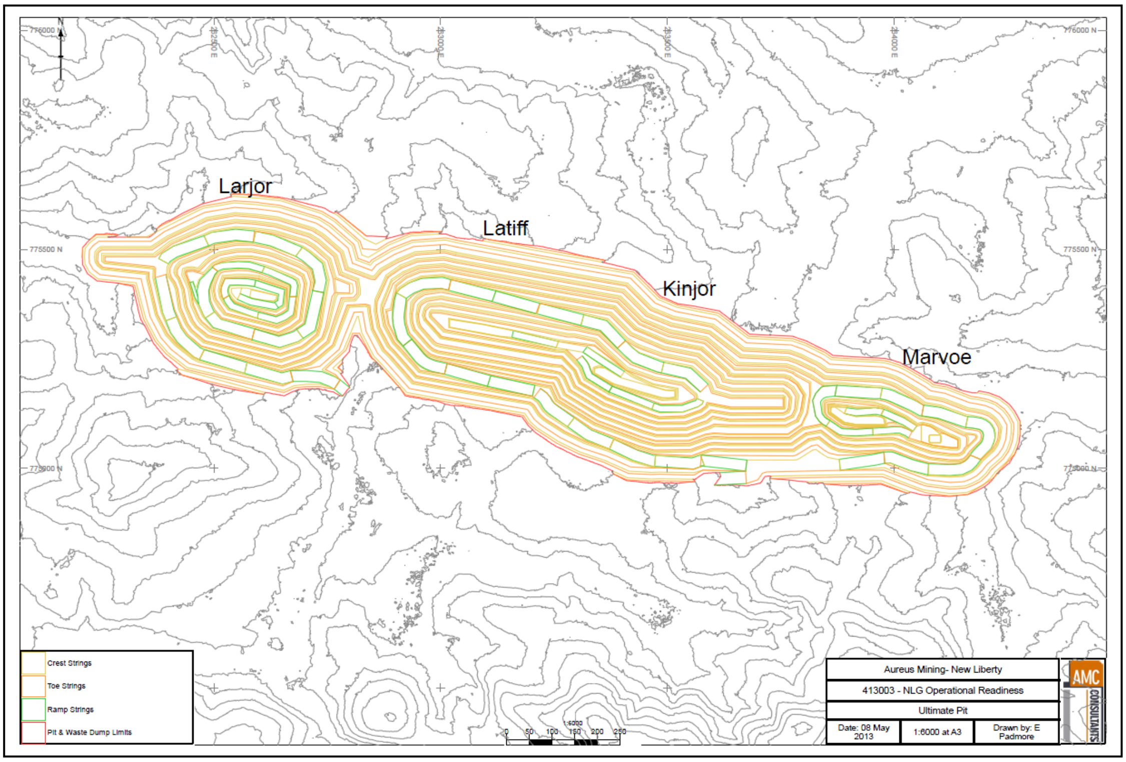



**Figure 15-3: Final Pit Design**

Table 15-6 shows the achieved design in comparison with the optimisation pit shell 14.

**Table 15-6: Optimised Pit and Designed Pit Comparison**

| | Ore (Mt) | Au (g/t) | Waste Rock (Mt | Total Rock (Mt) | Strip Ratio W:O | Rec. Au (koz) |
|---|---|---|---|---|---|---|
| **Optimized Pit Shell 14** | 8.4 | 3.4 | 119 | 128 | 14.1 | 866 |
| **Pit Design** | 8.5 | 3.4 | 132 | 140 | 15.5 | 859 |

In the initial stages of the pit operations it is planned to mine two interim starter pits in the Larjor and Kinjor areas. This will be followed in the mining sequence by the Larjor pit and a second cutback to the Latiff-Kinjor pit to take the pit to its final limits. Marvoe has a small starter pit to access some low stripping ratio ore earlier in Year 4, thereafter it deepens to its final limits. Figure 15-4 shows the main staging elements of the pit design, although the mining sequence does not necessarily follow the numerical order of these stages.

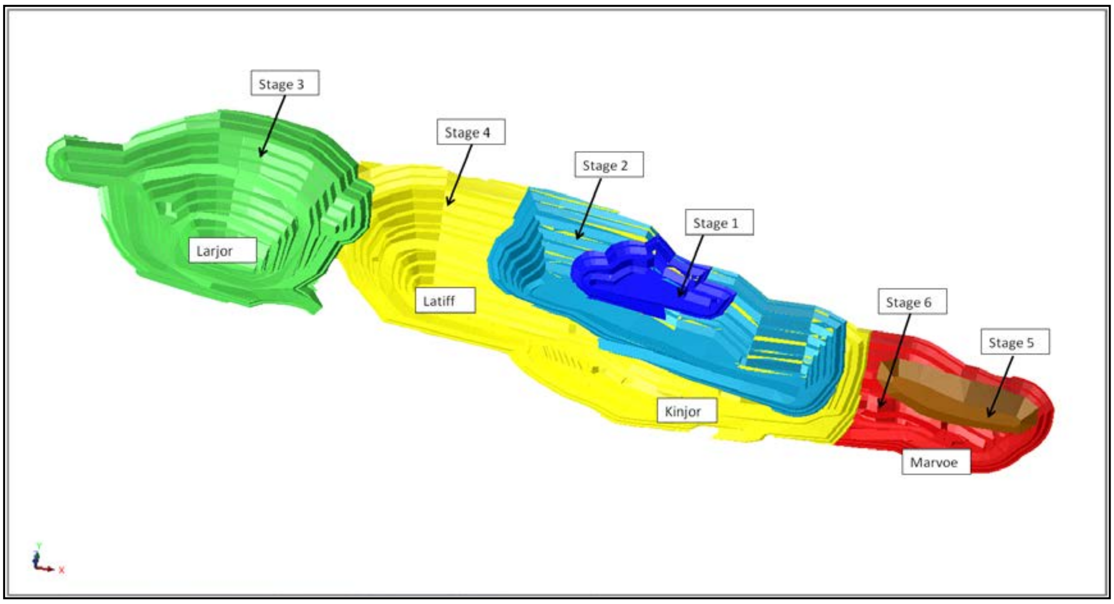



**Figure 15-4: Staging of Pit Design**

## 15.5 Mineral Reserve Statement

The pit design was evaluated using the diluted block model to estimate the mineral reserve of the Project. The mineral reserve estimate is summarised in Table 15-7.

The design generates a Proven and Probable mineral reserve of 8.5 million tonnes of ore with an average grade of 3.4 g/t, giving a total contained gold ounces of 924 koz at a strip ratio of 15:5.

**Table 15-7: AMC Mineral Reserve Estimate (as at 20 May 2013)**

| Reserve Category | Oxide / Fresh | Tonnes (Mt) | Au Grade (g/t) | Au Ounces (koz) |
|---|---|---|---|---|
| Proven | Oxide | - | - | - |
| | Fresh | 0.7 | 4.4 | 99 |
| Probable | Oxide | 0.3 | 2.3 | 18 |
| | Fresh | 7.5 | 3.3 | 806 |
| Total | Oxide | 0.3 | 2.3 | 18 |
| | Fresh | 8.2 | 3.4 | 905 |
| **Grand Total** | **Mineral Reserves** | **8.5** | **3.4** | **924** |
| Waste | Oxide | 13.3 | - | - |
| | Fresh | 118 | - | - |
| Total | | 131 | - | - |
| Strip Ratio | (W:O) (t/t) | 15.5 | - | - |

Notes: 1. CIM definitions were used for mineral reserves
2. A cut off of 0.8 g/t Au is applied for all zones
3. Due to rounding, some columns or rows may not add up exactly to the computed totals

The pit optimisations suggest that the pit size is relatively insensitive to changes in the costs, recoveries or metal prices. The pit is constrained by the high stripping ratio and the fact that it is reaching the base of the Measured and Indicated mineral resources in some areas.

SRK has reviewed the work completed to produce the above Mineral Reserve estimate and considers this to have been appropriately derived, to reflect the information currently available and the current mine plan and that it remains relatively robust to changes in costs and gold price.

# 16 MINING METHODS

## 16.1 Introduction

Aureus has entered into a partnership agreement with a specialist West African focused heavy equipment supplier, MonuRent (Liberia) Limited (MonuRent) to provide and maintain the mining fleet for the Project. Aureus will employ the equipment operators and plan, manage and utilise the equipment to undertake open pit mining operations.

MonuRent was requested to submit prices for a new fleet of mining equipment required for the open pit mining plan. Aureus has worked closely with the MonuRent senior management team to develop the equipment and mining costs for the Project. MonuRent will supply all of the capital mining fleet requirements (for the purchase of a new mining fleet) including pumps for pit dewatering and other support equipment, and also be responsible for the maintenance of the fleet. As a part of this agreement, MonuRent has guaranteed Aureus an equipment availability of at least 85%.

Aureus will undertake and manage all mine planning and technical aspects, including geology, grade control, mine planning, drill-and-blast planning, operational scheduling, mining and operator training and will manage the HME supplier. At the time of this report, apart from the senior mine geologist (a replacement of the previous incumbent who has recently left the Company), the mining team are already working at New Liberty.

MonuRent has the relevant experience in supplying and maintaining Heavy Mining Equipment (HME), and will utilise its established infrastructure in West Africa to service its existing Liberian business in Monrovia as well as its business at New Liberty. MonuRent carries an extensive part and spares inventory across its business, which it will use to service this Contract. MonuRent will also provide training facilities for the fleet operators.

The operating and capital costs associated with the Aureus employees and fleet rental model were estimated as part of the general and administration costs and mining costs respectively and discussed in Section 21.

The mining operations are based on conventional drill-and-blast, load and haul mining techniques. Loading of the ore and waste rock following drill-and-blast will be undertaken by hydraulic excavators in backhoe configuration into nominal 100 tonne Komatsu HD 785 haul trucks with 50% of the ore reporting directly to the crusher tip and 50% to the ROM ore pad, and waste to the designated waste dump areas. It is has been assumed that the weathered portion of the ore and waste is free dig or requires some local ripping and as it transitions to fresh rock, blasting will be required.

Pit access roads have been designed and installed as required and are constructed to a nominal width of 25 m inclusive of safety berms and water drainage controls. A ROM pad stockpile area is under construction adjacent to the process plant crusher station/tipping point and all ore stockpiled will be sorted under a stockpile grade control management scheme. Stockpile reclaim will be undertaken and fed to the crusher as required.

All waste material will report either to the surface waste dumps which wrap around the open pit or be backfilled in the mined-out pit areas of the Larjor pit, depending on which facility is closest and available at the time of mining, and to suit the efficient scheduling of the mining fleet. Before any waste is backfilled into Larjor, an underground mining study will be undertaken to assess the potential of mining the deeper, down plunge high-grade ore shoots. In the early stages of the Pre-Strip and mining phases, waste rock will be used to construct a protective drainage berm, which wraps around the open pit between the waste rock dumps.

In the mining of the designed pit ‘mineralized waste’ will be mined from areas in that grade category as defined in the grade control and resource models, and from other resources that become diluted below the cut-off grade. All such material will be identified during the mining operations and, subject to grade control, will be dumped and stored in locations, enabling it to be retrieved for treatment as marginal ore if and when future financial conditions allow. It is estimated that some 600 kt of mineralized waste from the Measured and Indicated resource categories at an average grade of 0.7 g/t will be mined throughout the life of the Project.

## 16.2 Waste Dump Design

The waste rock dump has been designed to form a protective barrier between the Marvoe Creek Diversion and the open pit. In addition to this a protective drainage berm is under construction between the pit and the toe of the waste rock dumps from weathered saprolite generated during the early phases of mining. This will be compacted to form a waterproof barrier preventing surface water flow from the waste dump areas into the pit. The design of the Waste Rock Dump, drainage berm and haul roads are shown in Figure 16-1.

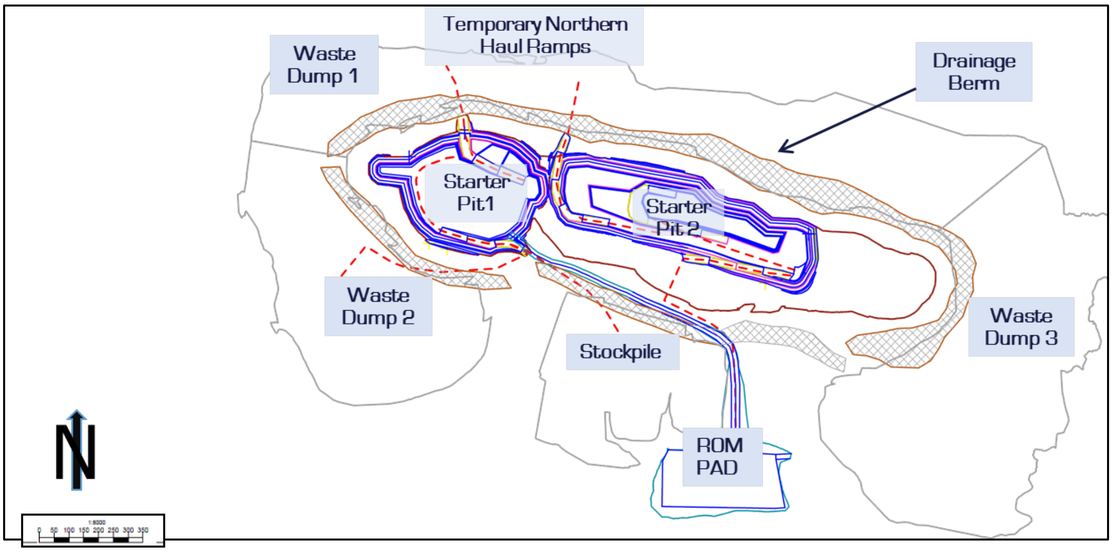



**Figure 16-1: Pit and Waste Dump Design**

The waste dumps were designed to have a capacity of 52 million cubic metres and to wrap around the open pit forming a protective barrier. The gap between the dumps to the south of Larjor (Starter pit 1) follows the previous natural course of the Marvoe Creek. This gap is for the accommodation of sedimentation ponds. The gap in the dump south of Marvoe (Starter pit 2) is to provide haul road access to the ROM pad.

The upper surface of the dump is profiled to shed water away from the pit catchment area and the wrap around design also provides additional flood protection around the open pit, forming a robust barrier between the pit and Marvoe Creek Dams.

The mining plan proposes backfilling the Larjor Pit with mined waste. It is estimated that approximately 10M BCM of broken waste material will be backfilled into the Larjor pit. This backfilling of waste will not take place until later in the mining schedule and after the completion of an underground mining study.

## 16.3 Mine Production Schedule

**Revised Mining Plan**

Despite the Ebola crisis, the Company remained funded for its development plan as set out in the previous Technical Report titled “New Liberty Gold Project, Liberia, West Africa Updated Technical Report” dated July 3, 2013. However, when taking into account the delay to the commencement of processing operations the previously proposed mining schedule does not optimise returns in the current gold price environment.

Since the last Technical Report was completed in 2013, the Company has continued to conduct further evaluations including grade control drilling to produce a better geological understanding of the orebody. Utilising this information, an optimal new mine plan has now been produced and it is this plan that forms the basis of the updated valuation presented later in this report.

The new mine plan detailed in this chapter compensates for the delay in the commencement of processing operations and improves the Project's economics through the development of two starter pits (maximising operational face lengths and reduced haulage distances for waste). It reduces costs over the life of mine (LOM) and generates stronger cash flows, particularly during the ramp-up period and first six months of production.

The principal benefits of the new mine plan relative to the previously published plan are:

- Stronger cash generation, particularly in the early stages of the Project, which will provide more cash for exploration and working capital
- Greater operational flexibility through the creation of two starter pits, providing increased face length and stockpile management and giving greater confidence that production targets will be met
- Increased Run of Mine ("ROM") ore stockpiles ensure against any unforeseen production disruption
- Reduced mining cash costs based on more efficient mining by utilizing the layout of the mining infrastructure, such as the waste dumps

The new mine plan involves a revised mining sequence, now running from East to West, which utilises two shallower starter pits at Kinjor and Larjor. This provides increased operational flexibility due to the increased workable face lengths and allows access to areas of high-grade ore earlier in the LOM. A drainage berm surrounding the open pit, which is constructed from waste rock, has also been incorporated into the new mine plan. The construction of this berm not only shortens the haulage distance for waste rock, but also lowers the Project execution risk in the wet season by safely reducing water ingress into the pit, minimising the pumping required to keep the pit fully operational throughout the wet season.

The new mine plan also incorporates increased efficiencies in the mining fleet and schedule, including "hot seat change over" at the start and end of shifts and the use of temporary haulage ramps to the north of the pit to minimise waste haulage distances. This enables more waste rock to be removed and ore to be mined and processed earlier in the mining schedule. The new mine plan also has the incremental benefit of the current low fuel prices, which helps to reduce the overall mining cost.

The incorporation of two starter pits combined with the increased mining and trucking efficiencies has allowed the Company to develop a more refined stockpile strategy than that outlined in the previous technical report. The increased early tonnage in the new schedule allows the Company to create a larger stockpile of ore on the ROM Pad, thus enabling higher ore grade material to be blended and fed to the process plant earlier than in the original schedule. This also provides an additional safeguard to any potential problems in the pit by having more ore material available for processing. The stockpile blending strategy facilitates a consistent grade of ore to be fed to the process plant. Additional oxide material will be blended with fresh ore during dry season, improving plant throughput by some 15%.

The change in the mining schedule results in the completion of mining operations four months earlier than planned in the DFS (AMC, 2013. New Liberty Gold Project, Liberia, West Africa Updated Technical Report) when maintaining all other material parameters. The revised production profile is more appropriately aligned to the current gold price environment and the basis of the new plan is that 10% more ore material is mined and 35% more gold is produced in 2015 than in the delayed DFS mine plan. These revisions more than adequately compensate for the delay in processing operations caused by the Ebola outbreak.

Although the overall mining schedule has been revised since the DFS plan, as detailed in the previous paragraphs, it should be noted that there has been no change to any other parameters such as the resource model (produced by for the 2013 DFS study) or to the pit and waste dump designs.

A pre-strip period of 6 months was scheduled to establish the pits and to build an ore stockpile ahead of plant commissioning. The schedule steps up an average material movement rate of approximately 70,000 tonnes per day from the end of the second quarter of Year 1.

In this schedule, processing commences in July of 2015 (Year1), treating the initial ore stockpile and pit ore production. Open-pit material movement achieves a steady state annual production of 1.1 Mt ore and 25.3 Mt total movement by June 2016. This steady state production continues until December 2019 (Year 5). The total movement declines after June 2021 as the strip ratio declines from around 25:1 to around 10:1 towards the end of the mine-life.

The Larjor pit is completed at the end of June 2019 (Year 4), and waste may be backfilled into the pit from this period until the scheduled end of the operation in 2023.

Table 16-1 summarises the annual production from the open pit and the treatment schedule.

The mining schedule reflects a review of a number of scheduling options with the focus on maximizing the grade mined to be treated in the early life of the mine whilst deferring mining costs. The change in the mining schedule results in the completion of mining operations four months earlier than planned in comparison to the DFS schedule when maintaining all other material parameters.

Table 16-1 details the mill treatment and gold production schedule, which assumes a nominal treatment rate for ore of 1.1 million tonnes per annum. The mining and treatment schedules are also shown in Figure 16-2 to Figure 16-4.

**Table 16-1: Mining and Treatment Schedule**

| | Units | Total | Year 0 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore | Mt | **8.5** | 0.01 | 0.8 | 1.1 | 1.0 | 1.3 | 1.1 | 1.0 | 1.1 | 0.8 |
| Grade | g/t | **3.4** | 2.3 | 3.4 | 3.0 | 3.4 | 3.7 | 3.9 | 3.3 | 3.5 | 2.4 |
| Contained Gold | k oz | **924** | 0.7 | 94.4 | 110.3 | 113.6 | 161.4 | 143.6 | 108.6 | 126.7 | 64.2 |
| Waste | Mt | **131** | 0.97 | 18.47 | 24.79 | 25.19 | 24.32 | 21.05 | 9.42 | 6.10 | 1.3 |
| Total Material | Mt | **140** | 1.0 | 19.3 | 25.9 | 26.2 | 25.7 | 22.2 | 10.4 | 7.2 | 2.1 |
| Strip Ratio | t w:o | **15.5** | 96.8 | 21.6 | 21.4 | 24.4 | 18.1 | 18.4 | 9.2 | 5.5 | 1.6 |
| | | | | | | | | | | | |
| Ore Milled | Mt | **8.5** | - | 0.5 | 1.2 | 1.2 | 1.1 | 1.1 | 1.1 | 1.1 | 1.0 |
| Grade | g/t | **3.4** | - | 3.8 | 3.1 | 3.3 | 3.5 | 4.2 | 3.3 | 3.7 | 2.3 |
| Contained Gold | k oz | **924** | - | 65.9 | 123.4 | 122.2 | 126.7 | 154.0 | 122.4 | 136.1 | 72.9 |
| | | | | | | | | | | | |
| Recovery | % | **93%** | | 92.2% | 93% | 93% | 93% | 93% | 93% | 93% | 93% |
| Gold Produced | k oz | **858** | - | 60.7 | 114.8 | 113.6 | 117.9 | 143.2 | 113.8 | 126.6 | 67.8.0 |

The mining schedule has been generated on a monthly basis for all production periods. The schedule has been based on a practical sequence of mining of ore and waste blocks. All material below the cut-off grade 0.8 g/t and above 0.65 g/t has been considered to be mineralized waste. The mineralized waste has been identified in the mining schedule as a separate stockpile to allow for possible future treatment of such material as financial conditions dictate and probably towards the end of the LOM.

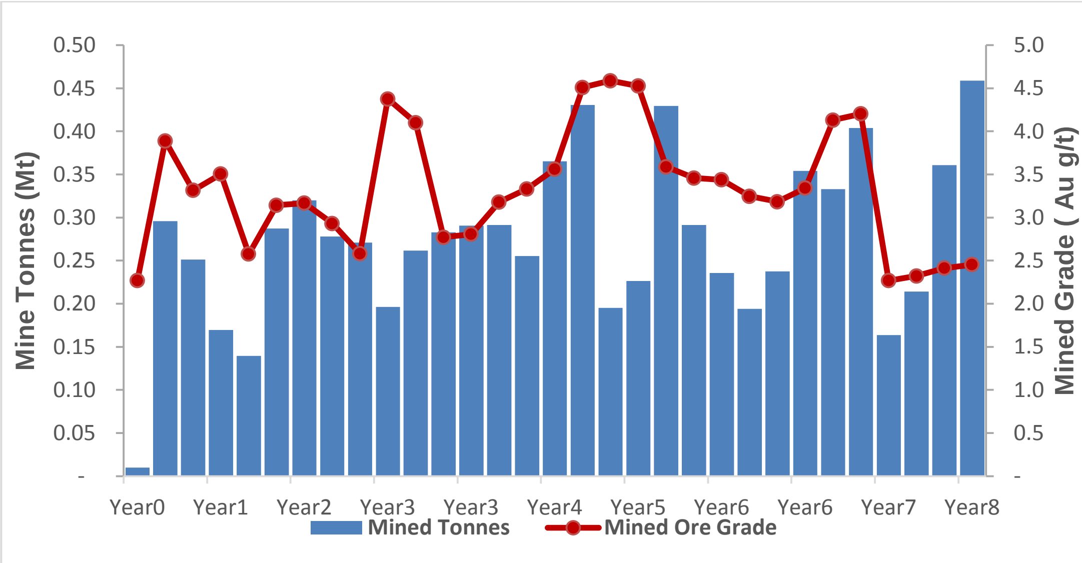



**Figure 16-2: Mined Tonnes and Grade**

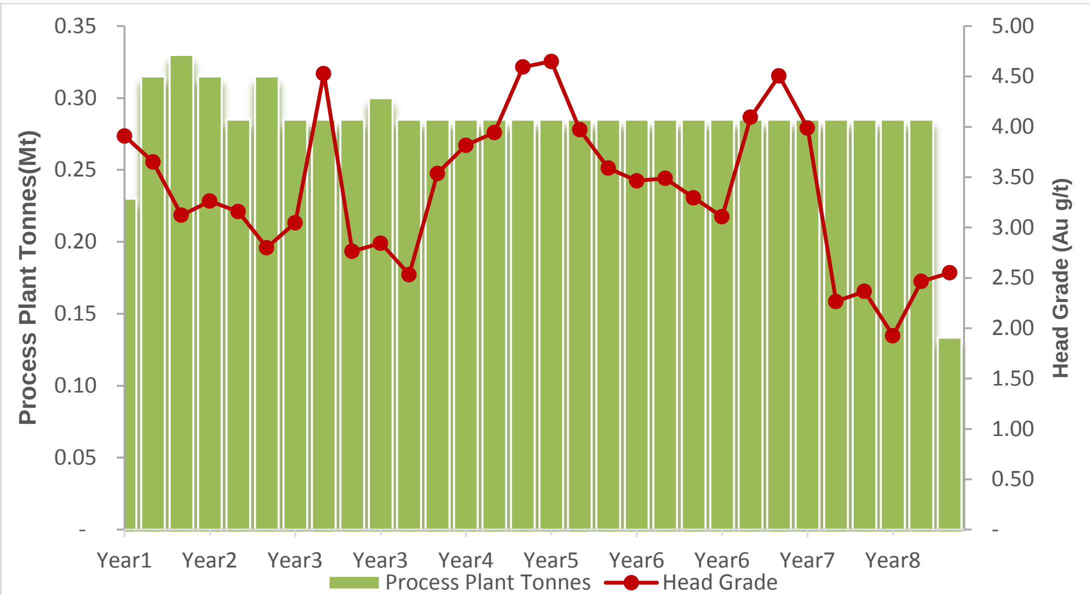



**Figure 16-3: Process Plant Tonnes and Head Grade**

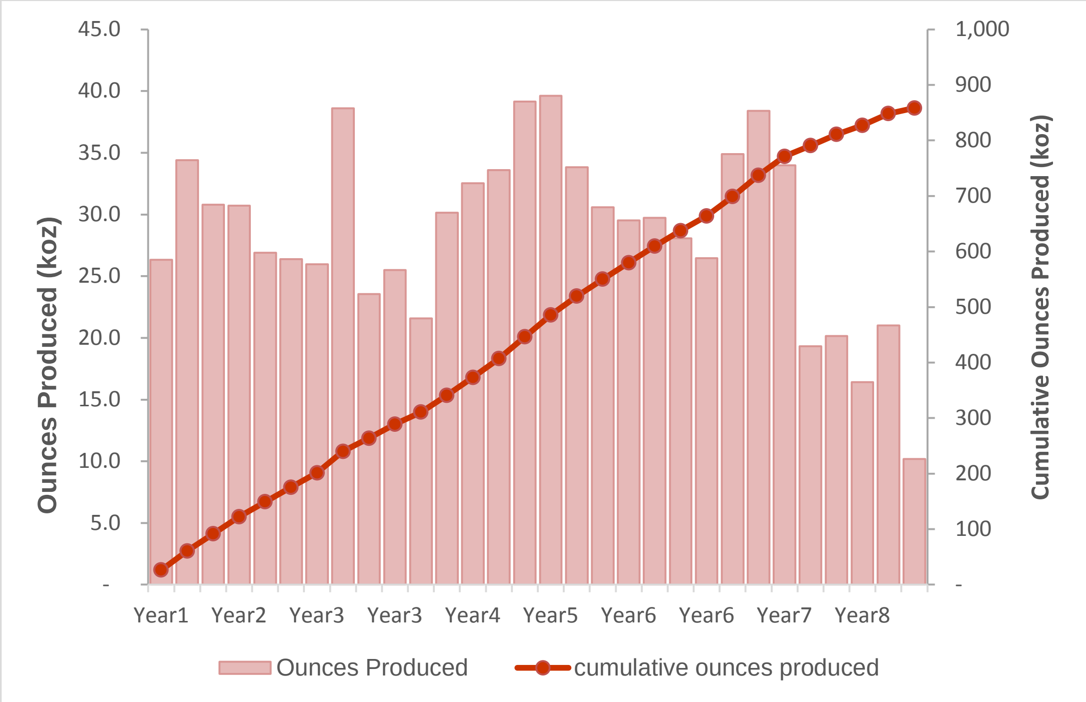



**Figure 16-4: Ounces Produced and Cumulative Ounces Produced**

## 16.4 Stockpile Strategy

One of the key differences between the previously published mine plan and the revised mine plan is that over the Life of Mine (LOM) it is clear that there is considerably more stock on the ROM pad in 2015 and 2016, approximately 200 Kt more than was the case in the DFS, giving additional security and enabling higher grades to be fed to the mill earlier in the schedule. The stockpiles are lower in the latter years of the mine's life. The LOM stockpile strategy is shown below in Figure 16-5.

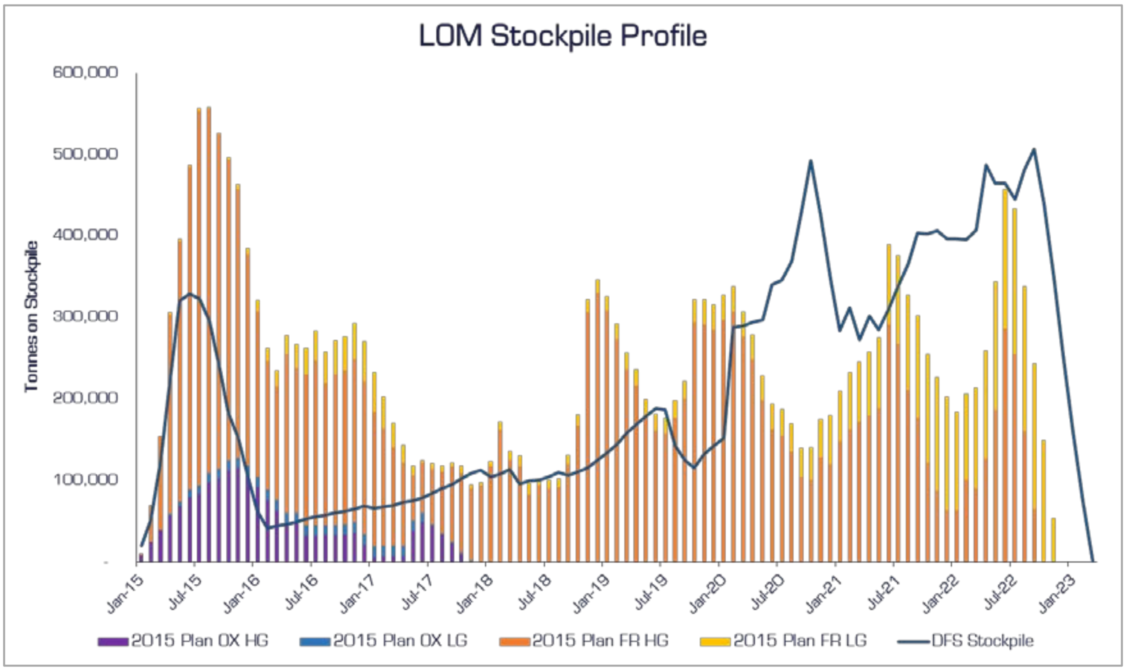



**Figure 16-5: Stockpile Strategy**

## 16.5 Mining Equipment Requirements

The mining fleet equipment reflects the scale of the operation at peak mining rate from the production schedule, this is summarised in Table 16-2. There is a planned phased build up in mined tonnage to reflect both commissioning requirements of the process plant and the mobilisation and training requirements for the mining operators.

The MonuRent owned and maintained mining fleet is composed of a combination of new mining equipment supplied by both Komatsu and Caterpillar. Under the Mining Fleet Supply contract, MonuRent guarantee Aureus fleet availability of at least 85% on a rolling three month basis. If this condition is not met, there are penalties applied to the contractor, including the purchase of replacement units at the contractors cost. MonuRent have agreed to supply spare swing units available for use in the event of a breakdown or equipment failure at no additional cost to Aureus. Table 16-2 provides an inventory of the MonuRent supplied fleet.

**Table 16-2: Mining Fleet**

| Equipment | No. of Pieces |
|---|---:|
| Excavator 12 $m^3$ bucket | 1 |
| Excavator 6 $m^3$ bucket | 2 |
| Excavator 2 $m^3$ bucket | 1 |
| Haul trucks 100 t | 16 |
| Dozers | 4 |
| | |
| Graders 16 foot | 2 |
| ROM pad loader | 2 |
| Water carts | 2 |
| Fuel and lube trucks | 4 |
| Drills | 7 |
| Light vehicles | 29 |
| Buses | 6 |
| Pumps, lighting towers, tyre handler etc. | Various |

Aureus is employing previously trained operatives as one-on-one operator/trainers from within the Economic Community of West African States (ECOWAS) region, who are providing training on a rotational basis. They will be replaced by local Liberian operatives as soon as the local operators reach the required safety standards. MonuRent has also supplied a state of the art operator training simulators for Dump Truck, Excavator and drill rig operator safety training, to aid the training of local operators at New Liberty.

## 16.6 Mine Work Schedule

Figure 16-6 and Table 16-3 summarise the mine work schedule planned for the Project. The mine is scheduled to operate 338 days per year, which includes the time for inclement weather. Holidays will be worked on an overtime basis and form part of the two-shift rotation system. Ore will be preferentially mined on day shift to allow greater supervision on ore mining, with waste mining being undertaken during the night shift. The mine will operate two 12-hour shifts per day, which allows 338 available shifts per year for ore and 676 shifts for waste. In order to achieve this, a three-crew system will be adopted for all mining personnel, one shift day-shift, one shift night-shift and a roster off-shift. The operating time per shift will be the actual time during the shift that the equipment is ‘’productively’’ working at its rated capacity and is equal to the total scheduled time less all scheduled and unscheduled delays.

**Table 16-3: Scheduled Working Periods**

| Parameter | Units | Value |
|---|---|---|
| Calendar Days | Days | 365 |
| Days per week | Days | 7 |
| Holidays | Days | 0* |
| Weather | Days | 27 |
| Scheduled days | Days | 338 |
| Shifts/day ore | Shifts | 1 |
| Shifts/day waste | Shifts | 2 |
| Annual work shifts ore | Shifts | 338 |
| Annual work shifts waste | Shifts | 676 |
| Hours per shift – Ore | Hours | 12 |
| Hours per shift - Waste | Hours | 12 |
| Scheduled hours Ore | Hours | 4,056 |
| Scheduled hours Waste | Hours | 8,112 |

**FULCO system i.e. pay overtime for holidays*

| Annual Calendar | Units | Value |
|---|---|---|
| Calendar Hours per year | Hrs | 8,760 |
| Example Equipment Availability | % | 0.85 |
| Total Down time per year | Hrs | 1,314 |
| Total Available time per year | Hrs | 7,446 |
| Total operating standby per year | Hrs | 1,212 |
| Use of Availability (Utilisation) | % | 84% |
| Working Hours Per annum | Hrs | 6,234 |
| **Daily Calendar** | **Units** | **Value** |
| Schedule Hours | Hrs | 24.0 |
| Time lost due 85% Availability | Hrs | 3.6 |
| Time lost due 84% Utilisation | Hrs | 3.3 |
| Hours Available per day | Hrs | 17.1 |

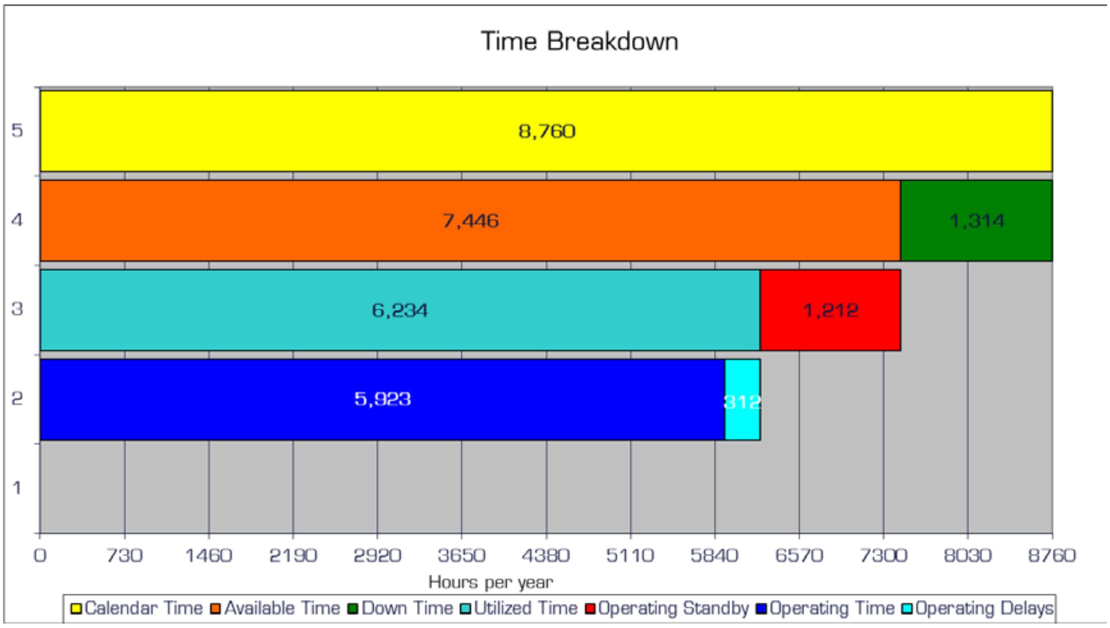



**Figure 16-6: Production Hours Scheduling Assumptions**

## 16.7 Open-Pit Dewatering

The open pit dewatering strategy is based on a study by RPS Aquaterra in June 2013. The total average monthly inflows have been estimated for the LOM and show significant seasonal fluctuation, primarily due to fluctuations in rainfall runoff. The inflows for the entire pit typically range from approximately 25 l/s in the drier months, derived primarily from groundwater, to over 100 l/s in the wet months where the water is derived primarily from rainfall runoff. The pit pumps will be sized to handle double this amount of water.

The water management plan for the open pit will have two focus areas, firstly managing surface runoff water and secondly pumping groundwater from sumps inside the pit.

A water diversion berm will be constructed around the open pit to prevent the inflow of surface water and shed it into the Marvoe Creek (Figure 16-7). As previously detailed, this berm will be located between the open pit crest and the toe of the waste dump, and also serve as a haul road around the edge of the pit. It will also significantly reduce the pumping costs associated with the potential surface water inflows. All rainwater run-off will be directed away from the pit, minimising the pumping requirements.

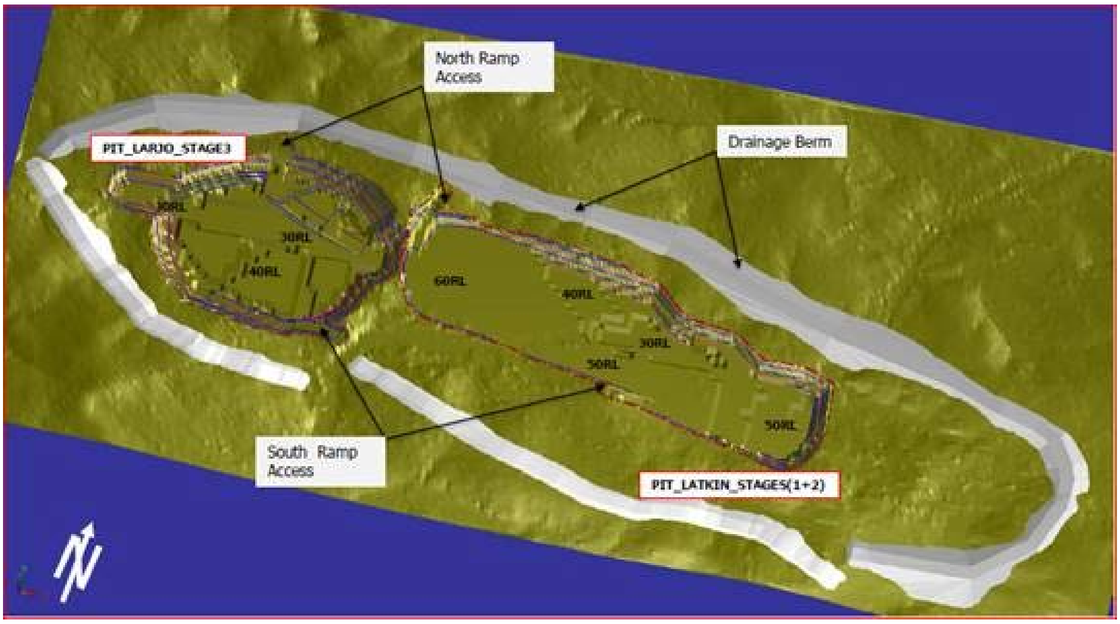



**Figure 16-7: Plan Showing the Diversion Berm and Pit Outline at the End of Year 1 (Dec 2015)**

A drain and berm will be also constructed just below the weathered/fresh rock interface on the 50m elevation. This will capture the water seepage at the aquifer at the weathered / fresh rock interface, collect it in strategically placed sumps and pump it back over the surface berm. The water from these will be collected and diverted to a Walled Sedimentation Basin (WSB), which will allow sediment to settle before the water is released. A series of dewatering boreholes situated in the eastern and western limits of the pit will also be drilled if required where the main orebody shear zone can be intersected and dewatered.

The second aspect of the pit dewatering relates to rain/stormwater events, which will be managed by dewatering pumps capable of handling the expected maximum ingress as described earlier. Diesel driven pumps will be used to pump to a sump inside each pit from where the water will be pumped out to the WSB by means of a pump station equipped with electric pumps. Successive pumping stations will be established as the pit extends deeper to ensure adequate pumping capacity at all times. MonuRent will be responsible for the maintenance of the entire pumping system, whilst Aureus personnel will be responsible for their operation.

## 16.8 Mining Manpower

The Aureus mining personnel will provide all the mining related labour and supervision as well as the managerial and technical services function to the operation. These include mine management, survey, geology, drill-and-blast planning and grade control, mine planning as well as all the supervisors and operators. MonuRent will supply the labour required for the maintenance and repair of the mining equipment.

## 16.9 Mine Infrastructure

The mining infrastructure is detailed in Section 18 as part of the general site plan.

A layout of in-pit and surface access roads has been developed. These roads allow access between the pits, process plant, mine laydown area and workshops, explosives magazine, ROM stockpile and waste dumps area covering all the work activities associated with the mine operations. These roads are currently being constructed by the mining team and the civil works contractor.

## 16.10 Underground Potential

Substantial mineralisation exists below the current designed pit bottom elevations used in the pit optimisation models. Aureus will be extending its exploration-drilling programme to further explore the lateral and depth extents of the currently defined mineralised zones. Future studies will assess the viability of potential deepening of the open pits or potential underground mining operations during the operational phase of the mine.

## 16.11 Concluding Remarks

SRK considers the updated mining plan to be a robust plan and to appropriately reflect the work recently completed by Aureus and additional information obtained plus the current delayed status of construction due to factors largely beyond the Company's control.

As already commented, the revised mine plan assumes more gold is produced than the optimised DFS over the early periods through the mining of more ore tonnes at a higher gold grade and processing more tonnage at a higher grade by selectively feeding from ROM stockpiles, thereby producing more ounces earlier in the mines life. This shortens the LOM by four months and improves the overall Project economics.

In addition, as noted earlier in this report, the results of the close spaced drilling have suggested that if anything the estimates of tonnes and grade used as the basis of the LOM Plan may be slightly conservative.

Given the above, and notwithstanding the fact that the updated mining costs are lower than previously forecast and that there is a risk that not all of the envisaged savings will be realised, SRK considers that this addresses the impact of the slippage in start-up caused by construction delays and improves the New Liberty business plan particularly in the early years.

# 17 RECOVERY METHODS

## 17.1 Plant Design Criteria

### 17.1.1 Introduction

Metallurgical testwork results and industry norms, where necessary, were used to define the process design criteria for the New Liberty Gold Project. The process plant is designed to treat 1.1 million tonnes per annum.

The proposed process flowsheet is an industry-standard arrangement consisting of two-stage crushing, ore stockpiling, milling and classification, gravity and CIL, cyanide detoxification, tailings disposal, acid wash, elution, electrowinning and gold room, carbon regeneration, reagent preparation, storage and dosing, oxygen, air and water systems.

Several references have been used to derive data used in the process design criteria. These are the following:

- Aureus Mining;
- Metallurgical testwork;
- Calculated data;
- Vendor data or recommendations;
- DRA standards or practices;
- Handbook (engineering handbook);
- External consultants.

**Table 17-1: Process Plant Design Criteria**

| **Base Data** | **Units** | **Value** |
|---|---|---|
| Plant feed rate | dmt/annum | 1,100,000 |
| Plant feed rate | dmt/month | 91,667 |
| Maximum lump size | mm | 700 |
| Moisture content | % w/w | 10.0 |
| Ore true density | tonne/m³ | 2.8 |
| Ore bulk density | tonne/m³ | 1.88 |
| ROM head grade (Year 1 – Year 7) | ppm Au | 3.10 – 4.00 |
| Overall Recovery Target | % | 92.0% – 93.2% |
| **Operating Times** | **Units** | **Value** |
| ROM delivery | days/annum | 350.0 |
| ROM delivery (by truck or loader) | hours/day | 18.0 |
| ROM delivery utilization | % | 76.5 |
| ROM delivery | hours/annum | 4,820.0 |
| Crushing | days/annum | 350.0 |
| Crushing | hours/day | 18.0 |
| Crushing utilization | % | 76.5 |
| Crushing | hours/annum | 4,820.0 |
| Concentrator | days/annum | 350.0 |
| Concentrator | hours/day | 24.0 |
| Concentrator utilization | % | 90.0 |
| Concentrator | hours/annum | 7,560.0 |
| **Feed Particle Size Differentiation** | | **Value (mm)** |
| Primary crusher feed PSD | $P_{100}$ | 700 |
| (typical design values) | $P_{80}$ | 433 |
| (typical design values) | $P_{50}$ | 209 |
| (typical design values) | $P_{25}$ | 70 |

## 17.2 Ore Characteristics

Gold mineralisation at New Liberty occurs in zones of variable thickness and is nearly continuous along 1.8 km of strike length. The ore characteristics are shown in Table 17-2.

**Table 17-2: Ore Characteristics**

| Base Data | Units | Value |
|---|---|---|
| Ore Source | type | Open pit |
| Maximum lump size (F100) | mm | 700 |
| Maximum lump size (F80) | mm | 433 |
| Maximum lump size (F50) | mm | 209 |
| Maximum lump size (F25) | mm | 70 |
| **ROM Characteristics** | | |
| Moisture content | % | 10.0 |
| Ore SG | t/m³ | 2.80 |
| Bulk density of crushed ore | t/m³ | 1.88 |
| **Raw Water Analysis** | | |
| Source | - | Marvoe Creek |
| Magnesium | ppm | <0.5 |
| Sulphate | ppm | <1 |
| Chloride | ppm | 0.900 |
| Calcium | ppm | 1.00 |
| Total cyanide | ppm | <0.01 |
| WAD cyanide | ppm | <0.01 |
| Conductivity | mS/m | 1.70 |
| pH | - | 5-5.2 |
| TDS | mg/l | 9-12 |

## 17.3 Operating Schedule

Table 17-3 summarises the data used in compiling the operational schedule of the processing plant.

**Table 17-3: Operating Schedule**

| Operating Schedule | Units | Value | |
|---|---|---|---|
| | | Nominal | Design |
| **General** | | | |
| Annual tonnage treated | Mtpa | 1.10 | 1.10 |
| Ore Processing tonnes per month | t/month | 91,667 | 91,667 |
| **ROM Delivery** | | | |
| Days per annum | days | 350 | |
| Overall utilisation | % | 76.5% | |
| Hours per day (loading by truck or loader) | h | 18.0 | |
| Operating hours per annum | h/a | 4,820 | |
| **Crushing Circuit** | | | |
| Days per annum | days | 350 | |
| Hours per day | h | 18.0 | |
| Overall utilization | % | 76.5% | |
| Operating hours per annum | h/a | 4,820 | |
| Primary crusher feedrate | t/h | 228 | 228 |
| Screening throughput | t/h | 492 | 492 |
| Screen O/Size % of new feed | % | 115% | 115% |
| Secondary crusher throughput | t/h | 263 | 263 |
| **Milling and Concentrating** | | | |
| Days per annum | days | 350 | |
| Hours per day | h | 24.0 | |
| Overall utilisation | % | 90.0% | |
| Operating hours per annum | h/a | 7 560 | |
| Milling throughput | t/h | 145.5 | 145.5 |

## 17.4 Plant Recovery

The assumed overall plant recoveries are shown in Table 17-4.

**Table 17-4: Plant Recovery**

| Recovery | Units | Value | |
|---|---|---|---|
| | | Nominal | Design |
| Head grade | g/t | 3.10 | 4.00 |
| Head gold content | g/h | 451 | 582 |
| Head gold content (maximum) | g/day | 10 825 | 13 968 |
| **Gravity Concentration** | | | |
| Estimated Gravity circuit feed grade | g/t | 2.48 | 2.96 |
| Gravity circuit feed gold | g/h | 372 | 444 |
| Gravity recovery (% Au of gravity concentration unit feed)* | % | 58.2% | 62.9% |
| Gravity recovery from test work (% Au of head grade)* | % | 48.0% | 48.0% |
| Gravity concentrate gold | g/h | 238 | 305 |
| Dissolution reactor recovery (% Au of unit feed) | % | 95.0% | 95.0% |
| Dissolution reactor gold recovery | g/h | 226 | 290 |
| Overall gravity concentration recovery (% Au of head grade)* | % | 45.6% | 45.6% |
| **CIL** | | | |
| CIL feed grade | g/t | 1.69 | 2.18 |
| CIL feed gold | g/h | 245 | 317 |
| Solution tailings | ppm | 0.005 | 0.005 |
| Solid tailings | g/t | 0.22 | 0.25 |
| Estimate for Gold in CIL tailings (solids+solution+carbon fines) | g/h | 33.9 | 37.0 |
| High Shear Scale-up Recovery Discount | % | 0.50% | |
| Leach rate constant for modelling | t/h.g | 0.800 | |
| Absorption rate constant for modelling | h-1 | 0.020 | 0.020 |
| Equilibrium constant used for modelling | g/t | 10 000 | 10 000 |
| Freundlich exponent used for modelling | - | 0.700 | 0.700 |
| | | | |
| **Total Overall Recovery (% Au of head grade)** | **%** | **92.0% – 93.1%** | |

*Denotes non- discounted recovery estimates

## 17.5 Process Plant Design

### 17.5.1 Ore Receipt and Crushing

ROM open pit ore received from trucks will be treated in a primary crushing circuit comprising of a ROM bin (100-BN-001) fitted with a static grizzly, apron feeder (100-FD-004) and primary jaw crusher (100-CR-010) operated in open circuit. A dust suppression system will be installed.

The primary crusher product and apron feeder dribblings will gravitate onto the jaw crusher product conveyor (120-CV-014).

Secondary crusher product is combined with primary crusher product on the jaw crusher product conveyor (120-CV-014) which feeds the crushing circuit sizing screen (120-SR-016).

The primary crusher product conveyor (120-CV-014) is fitted with a weightometer (120-ZM-040), which will be positioned before the introduction of secondary crusher product.

Circuit screen oversize is weighed (120-ZM-603) and conveyed (120-CV-019) to the secondary crushing circuit comprising of a bin (120-BN-023), a vibrating feeder (120-FD-024) and a secondary cone crusher (120-CR-026), in closed circuit to produce a crushed product stream with a $P_{100}$ of 22 mm. It should be noted that only one crusher will be installed but space provision allows for the installation of a second cone crusher. A dust suppression system will be installed. Circuit screen undersize is conveyed to a 3500 t mill feed stockpile (200-SP-050) via the stockpile feed conveyor (120-CV-034).

The key parameters of the crushing circuit are summarised in Table 17-5.

**Table 17-5: Ore Receipt and Crushing**

| **Primary Crushing** | **Units** | **Value** | |
|---|---|---|---|
| | | Nominal | Design |
| **General** | | | |
| Crusher work index (CWi) used for crusher sizing | kWh/t | 20.5 | |
| Uniaxial compressive strength (UCS) | MPa | Not Known | |
| Abrasion index (Ai) | | 0.332 | |
| **Primary Crushing** | | | |
| Method of feeding ROM bin | Type | Truck | |
| Truck type | Type | Komatsu 785 | |
| Truck load size | T | 100t | |
| Primary jaw crusher | Type | C125 | |
| Number of primary crushers | # | 1.00 | |
| Crusher CSS | mm | 100 | 100 |
| Primary crusher product size ($P_{100}$) | mm | 192 | 192 |
| Primary crusher product size ($P_{80}$) | mm | 128 | 128 |
| Primary crusher product size ($P_{35}$) | mm | 60 | 60 |
| **Primary Crusher Feeder** | | | |
| Type | Type | Apron feeder | |
| Drive type | Type | VSD | |
| Capacity | t/h | 228 | |
| **Combined Primary and Secondary Crusher Discharge Conveyor** | | | |
| Dry capacity | t/h | 492 | |
| Maximum lump size (F100) | mm | 192 | |
| Moisture content | % | 10.0% | |
| Nominal wet capacity | t/h | 546 | |
| Bulk density | t/m³ | 1.88 | |

| Secondary Crushing | Units | Value | |
|---|---|---|---|
| | | Nominal | Design |
| **Secondary crusher feed** | | | |
| Circuit screen type | type | 2.4 m wide x 6.1 m long, double deck, decline, dry | |
| Screen aperture (top) | mm | 55x55 | |
| Screen aperture (bottom) | mm | 22x55 | |
| Circuit screen feedrate | t/h | 492 | |
| Screening oversize % of new feed | % | 115% | |
| **Secondary Crusher Feed Bin** | | | |
| Type | type | Bin | |
| Total bin volume | m³ | 25.0 | |
| Selected bin capacity | min | 5.00 | |
| Emergency overflow | Yes/No | No | |
| Withdrawal method from bin | Type | Vibratory pan feeder | |
| **Secondary Crusher Feeder** | | | |
| Drive type | type | VSD | |
| Capacity | t/h | 263 | 263 |
| **Secondary Crushing** | | | |
| Secondary crusher type | type | HP500 Std, medium liners | |
| No. of Secondary crushers required | # | 1.00 | |
| Selected secondary crusher CSS | mm | 24.0 | |
| Secondary crusher feed at crusher loading | t/h | 263 | |
| Final crushing product (P100) | mm | 22.0 | |
| Final crushing product (P80) | mm | 18.0 | |
| **Secondary Crusher Feed Conveyor** | | | |
| Dry capacity | t/h | 263 | |
| Maximum lump size (F100) | mm | 192 | |
| Moisture content | % | 10.0% | |
| Wet capacity | t/h | 292 | |
| Bulk density | t/m³ | 1.88 | |

### 17.5.2 Milling

A ball milling and regrind vertimill circuit will treat crushed ore at a design feedrate of 146 t/h dry solids producing a final product with a nominal $P_{80}$ of 45 micron and $P_{60}$ of 25 micron.

The mill will be fed from the mill feed silo using belt feeders (200-FD-053/054) and into the milling circuit by a conveyor (200-CV-057) fitted with a weightometer (200-ZM-091). Milk of lime is added to the mill feed to control the CIL-circuit operating pH.

A 17.5ft x 22ft EGL ball mill (200-ML-059) will be operated in closed circuit to produce a target grind of 80% passing 212 µm. The mill feed will consist of fresh crushed ore, a portion of the classification cyclone underflow, gravity concentration classification screen oversize and a recycle stream from the gravity concentration circuit tailings. The mill density is controlled at ±70-75% solids by mass, by the addition of process water to the mill inlet. A trommel screen (200-SR-070) is installed on the mill discharge to remove scats from the mill discharge slurry. A scats bunker will collect the scats and a mechanical reclaim will dispose of scats.

Mill product will report to the mill discharge sump (200-TK-072) where it is diluted to 50%-55% solids (under normal conditions) before being pumped to the classification cyclone cluster (200-CL-075) operated at 60-100 kPa so as to achieve an overflow product of 80% passing 75 µm at an estimated 38-40% solids by mass.

Cyclone underflow will be routed to the gravity concentrator feed at a rate of 140 t/h dry solids, with any excess re-cycled directly to the mill feed.

The cyclone overflow is gravity fed to the secondary cyclone feed sump (200-TK-640) where it is joined by the product from the regrind mill and diluted to 50% solids before being pumped to the classification cyclone cluster (200-CL-076) operated at 110 kPa so as to achieve an overflow product of 80% passing 45 µm and 60% passing 25 micron.

The cyclone underflow will be fed to the Vertimill. Provision has been made to allow the Vertimill to be bypassed. The vertimill product will join the primary classifier overflow for combined secondary classification.

Secondary classifier cyclone overflow is gravity fed to the pre-leach thickener (260-TH-120) via a linear trash screen (200-SR-078) and a primary cross cut and secondary vezin sampling system (200-SA-124/125). Trash screen oversize is dewatered in a basket (200-SR-081) situated in the mill sump and drains directly to the mill-area spillage pump.

Spillage in the milling circuit bund will be collected and pumped to the mill discharge sump.

The key parameters of the milling circuit are shown in Table 17-6.

**Table 17-6: Milling**

| **Milling and Classification** | **Units** | **Value** | |
|---|---|---|---|
| | | Nominal | Design |
| **Stockpile Feed Conveyor** | | | |
| Dry Capacity | t/h | 228 | |
| Maximum Lump Size (F100) | mm | 22.0 | |
| Moisture Content | % | 10.0% | |
| Wet Capacity | t/h | 254 | |
| Bulk Density | t/m³ | 1.88 | |
| **Mill Feed Stockpile** | | | |
| Quantity | # | 1.00 | |
| Silo Live Capacity | hrs | 24 | |
| Silo Live Capacity | t | 3,500 | |
| **Mill Feeders (stockpile Reclaim)** | | | |
| Quantity of Feeders | # | 2.00 | |
| Drive Type | type | VSD | |
| Capacity | t/h | 145.5 | |
| **Mill Feed Conveyor** | | | |
| Dry Capacity | t/h | 145.5 | |
| Maximum Lump Size (F100) | mm | 22.0 | |
| Moisture Content | % | 10.0% | |
| Wet Capacity | t/h | 162 | |
| Bulk Density | t/m³ | 1.88 | |
| **Milling Circuit Selection** | | | |
| Circuit Type | type | Closed Circuit Ball Milling | |
| **Primary Milling** | | | |
| Ball Mill Work Index Range (BWi) | kWh/t | 14.0-22.1 | |
| Rod Mill Work Index Range (RWi) | kWh/t | 14.7-18.3 | |
| JK Tech Drop Weight Ranges | A | 51.7 – 89.1 | |
| | b | 0.27-0.53 | |
| | A x b | 21.3-30.5 | |
| | ta | 0.21-0.59 | |
| Quantity of Mills Installed | # | 1.00 | |
| Wet Or Dry Milling | type | Wet | |
| Open Or Closed Circuit | type | Closed | |
| Overflow Or Grate Discharge | type | Grate Discharge | |
| Feed Size ($F_{80}$) | mm | 18.0 | 18.0 |
| Mill Discharge Product Size Prior to Classification ($P_{80}$) | µm | 212 | 212 |

| Milling and Classification | Units | Value | |
|---|---|---|---|
| Discharge Slurry Density | %Sw/w | 70%-75% | |
| Scats Removal | type | Bunker | |
| Mill Discharge Screen | type | Trommel | |
| Aperture Size | mm | TBC | |
| **Selected Mill Size** | | | |
| Diameter (Inside Shell) | ft | 17.5 | |
| Effective Grinding Length (EGL) | ft | 22.0 | |
| Mill Speed as % of Critical Speed | %Cs | 75.0% | |
| Drive Type | type | Single Pinion and Reversible Girth Gear | |
| Design Mill Ball Load (Process) | % | 25.0% | |
| Ball Material (High Cr, Cast Or Forged) | type | Forged | |
| Mill Liner Material (Steel, Rubber, Polymet) | type | Polymet | |
| Mill Ball SG | t/m³ | 4.80 | |
| Selected Ball Size | mm | 90 mm | |
| Mill Power (Gross) | kW | 2,951 | |
| Motor Power (installed) | kW | 3,500 | |
| **Mill Discharge Sump** | | | |
| Cyclone Feed Density | %S w/w | 50.0% | 55.0% |
| Design Residence Time | min | 2.00 | |
| Cyclone Feed Grade (Estimated) | g/t | 2.22 | 2.65 |
| **Primary Milling Classification** | | | |
| Type of Classification | type | Hydrocyclone Cluster | |
| Quantity of Clusters | # | 1.00 | |
| Cyclone Operating Pressure | kPa | 85 | |
| Circulating Load ( Vertimill Operating) | % | 96.0% | |
| Circulating Load Vertimill (By-Passed) | % | 200% - 250% | |
| Cyclone Overflow Density | %Sw/w | 38%-47% | |
| Overflow Product Size ($P_{80}$) | µm | 75.0 | |
| Estimated Gold Upgrade Ratio in Underflow (Relative to cyclone feed) | - | 1.12 | |
| **Regrind Mill** | | | |
| Mill Type | type | VertiMill | |
| Quantity of Mills Installed | # | 1.00 | |
| Selected Mill Size | type | VM 1500 | |
| Wet Or Dry Milling | type | Wet | |
| Mill Feed Slurry Density | %Sw/w | 70%-75% | |
| Circuit Feed Rate | tph | 146 | |

| Milling and Classification | Units | Value |
|---|---|---|
| Mill Feed Size ($F_{80}$) | µm | 75 |
| Open Or Closed Circuit | type | Reverse Closed Circuit |
| Ball Material (High Cr, Cast Or Forged) | type | Forged |
| Mill Liner Material (Steel, Rubber, Polymet) | type | Steel |
| Mill Ball SG | t/m³ | 4.80 |
| Selected Ball Size | mm | 19 mm |
| Mill Power (absorbed) | kW | 1,000 |
| Motor Power (installed) | kW | 1,120 |
| **Regrind Mill Classification** | | |
| Type of Classification | type | Hydrocyclone Cluster |
| Quantity of Clusters | # | 1.00 |
| Cyclone Operating Pressure | kPa | 85 |
| Cyclone Tonnage U/F Relative to Mill Feed | % | 70% |
| Cyclone Overflow Density | %Sw/w | 30.0% -35.0% |
| Overflow Product Size ($P_{80}$) | µm | 45 |
| Circulating Load | | 200%-250% |
| **Trash Screening** | | |
| Type of Screen | type | Linear |
| Screen Feed Density | %Sw/w | 30.0%-35.0% |
| Aperture Size | µm | 600 |
| Underflow Slurry Transport Method | type | Gravity |

### 17.5.3 Gravity Concentration

The gravity concentrator feed is pre-screened on a vibrating screen (220-SR-100) to remove oversize material not suited for the concentrator. The screen sprays will be used to dilute the feed to the concentrator to 60%-65% solids by mass while also increasing screening efficiency. Screen oversize will return to the mill, with the underflow gravitating to the gravity concentrator (220-GS-104). Concentrator tailings will gravitate to the ball mill feed while the concentrate will report to a batch dissolution reactor. Fluidizing water will be supplied to the concentrator, which will be fenced for security reasons.

Concentrate from the gravity concentrator is collected in the Acacia concentrate de-watering cone (220-ZA-111). De-watering cone underflow at ±60% solids by mass is fed to the intensive-dissolution reactor on a batch basis, once every 24 hours, while the overflow is returned to the ball mill discharge sump. A batch solution of cyanide and caustic is made up and this leach solution is used to achieve a target gold extraction in excess of 95%. The resultant pregnant leach solution is then pumped to the dissolution circuit, electrowinning cell, feed tank (520-TK-323). The solids in the reactor are rinsed with water after the batch leach. The rinse water with the solids is then pumped to the milling discharge sump.

Spillage in the dissolution area will be collected and pumped to either the mill discharge sump or the pre-oxidation tank feed box. The dissolution reactor will be enclosed with a fence for security reasons.

The parameters of the gravity concentration section are summarised in Table 17-7.

The layout has space has space provision for the installation of 2nd gravity concentrator in future.

**Table 17-7: Gravity Concentration**

| Gravity Circuit | Units | Value | |
|---|---|---|---|
| | | **Nominal** | **Design** |
| **Scalping Screen** | | | |
| Type of Screen | Type | Vibrating | |
| Aperture Size Design | Mm | 1.6 x 13 – Slotted | |
| **Gravity Concentrators** | | | |
| Gravity Feed Grade | g/t | 2.48 | 2.96 |
| Type of Gravity Unit | Type | Falcon | |
| No of Gravity Units | # | 1 | |
| Unit Feed | t/h/unit | 140 | |
| Unit Feed Density | %Sw/w | 65-70% | |
| Unit Flushing Water Requirement | $m^3/h$ | 12 | |
| Concentrate Mass Pull (% of Unit Feed) - Estimated | % | 0.100% | 0.150% |
| Concentrate SG | $t/m^3$ | Not Known | |
| Concentrate Density | %Sw/w | 20.0% | |
| **Gravity Concentrate Treatment** | | | |
| Type of Concentrate Treatment | Type | Acacia | |
| Quantity of Units | # | 1.00 | |
| No of Leaches per Day | # | 1.00 | |
| Dissolution Reactor Volume Model | | CS 2000 | |
| Pregnant Solution Per Batch | $m^3$ | 5.00 | |

### 17.5.4 Thickening, Pre-Oxidation, Pre-Leach and CIL

Cyclone overflow gravitates to the pre-leach thickener (260-TH-120), where it is thickened to produce an underflow density of 45% solids by mass. The thickener underflow is pumped to the 1,000 $m^3$ pre-oxidation tank 300-TK-164 and spillage will be pumped (260-PP-123) back to the thickener feed box.

There is a step height of 600 mm between the pre-oxidation tank and the first CIL tank to ensure that slurry is transferred from one tank to the next by gravity. The pre-oxidation tank has a bypass facility to ensure continuity in production if the tank is taken offline for maintenance. Oxygen will be introduced to the bottom of the pre-oxidation tank by spargers and a pump with high shear reactor will increase the dissolved oxygen content. Lead nitrate will be added to aid the process and milk of lime for pH control.

The CIL circuit consists of 6 × 1000 $m^3$ (300-TK-127/128/129/130/131/132) tanks in series, with slurry transferred between tanks by Kemix MPS (P) inter-tank screens and launders. All tanks have a bypass facility to ensure continuity in production if a tank is taken offline for maintenance. Carbon concentrations of 12-15 g/L will be maintained in each tank, with counter-current carbon flow. In the CIL circuit, cyanide is added to the first and/or second CIL tank so as to effect the leaching of gold. Barren electrowinning solution and elution spillage is recycled to the first CIL tank to boost cyanide levels. The CIL circuit will be operated to achieve a carbon loading of 1,100-1,500 g/t, with daily inter tank carbon transfers to achieve constant carbon distribution. This translates to a loaded carbon batch size of ±5 tons per day. The CIL circuit will be operated to achieve a desired gold grade of less than 0.22 – 0.25 g/t in the tailings (dependant on mill feed grade) and a target solution gold tenor of 0.005 ppm.

The slurry from the $6^{th}$ CIL tank is then pumped to the Detox/ As leach tank before entering the Arsenic precipitation tanks via the carbon safety screen (300-SR-156).

Loaded carbon from the first CIL tank will be pumped to a vibrating screen for washing. (300-SR-168). The screen oversize (washed loaded carbon) will gravitate to the elution circuit acid-wash tank (500-TK-255) and the undersize (slurry) will report back to the first CIL tank.

A spillage pump is installed in the CIL bund (300-PP-159). Intensive leach reactor spillage and CIL spillage are recycled to the pre-oxidation tank.

Hydrogen cyanide and ammonia gas detection will be installed on both the CIL and Detox circuits, together with cyanide and pH control equipment in required locations. A safety shower will be installed in the area. A maintenance bay with a screen frame and washing facilities will be supplied for the cleaning of the inter-stage screens as well as a tower crane (300-XL-161) for maintenance.

The key parameters relating to thickening and CIL are summarised in Table 17-8.

**Table 17-8: Thickening, Pre-Oxidation, and CIL**

| Thickening, Pre-Oxidation and CIL | Units | Value | |
|---|---|---|---|
| | | Nominal | Design |
| **Pre-Leach Thickening** | | | |
| Type of Thickener | Type | High Rate | |
| Thickener Feed Density | %Sw/w | 35% | |
| Thickener Feed Solid Flowrate Design | t/h | 145.5 | |
| Thickener Feed Slurry Flowrate Design (Regrind Mill Classification Cyclone Product) | m³/h | 322 | |
| Specific Settling Area | t/m²/h | 0.5 | |
| Underflow Density | %Sw/w | 40% - 45% | |
| Selected Thickener Diameter | m | 21.0 | |
| Flocculant Addition Rate | g/t | 30.0 | 40.0 |
| **Pre-Oxidation and CIL** | | | |
| Leach Feed Grade | g/t | 1.69 | 2.18 |
| Leach Feed Density | %Sw/w | 40.0% - 45.0% | |
| Leach Feed Slurry Flowrate | m³/h | 230 – 270 | |
| No of Pre-Oxidation Tanks | # | 1 | |
| Selected Pre-Oxidation Tank Volume | m³ | 1,000 | |
| Calculated Pre-Oxidation Residence Time | hrs | 3.59 – 4.22 | |
| No of CIL Tanks | # | 6 | |
| Required Total CIL Residence Time | hrs | 24 | |
| Calculated Total CIL Residence Time | hrs | 21.5 – 25.3 | |
| Selected CIL Tank Volume | m³ | 1,000 | |
| Tank Volume Loss | % | 3. 0% | |
| Carbon Concentration Design | g/L | 12 – 15 | |
| Gold Loaded on Carbon | g/day | 5,075 – 6,710 | |
| Carbon Movement Per Day | t | 5.00 | |
| Carbon Loading | g/t | 1,015 – 1,342 | |
| Final Carbon Loading | g/t | 1,065 – 1,092 | |
| Carbon Residence Time/Tank (Modelled) | hrs | 72.5- 81.5 | |
| Carbon Movement Rate | t/h | 0.208 | |
| Mass of Carbon per Stage | t | 15.1 – 17.0 | |
| Time Required to Move Loaded Carbon | hrs | 6.7 - 8.3 | |
| Mass of Carbon to Elution per Batch | t | 5.00 | |
| CIL Carbon Transfer Rate | t/h | 0.60 - 0.75 | |
| Slurry Flow During Loaded Carbon Transfer | m³/h | 50 | |
| Time Required to Move per Inter-tank Flow | hrs | 6.7 - 8.3 | |
| Carbon Inter-tank Transfer Rate | t/h | 0.60 - 0.75 | |
| Slurry Flow During Inter-tank Carbon Transfer | m³/h | 50 | |
| **CIL Interstage Screen** | | | |
| Interstage Screen Type | type | MPS(P) | |
| Interstage Screen Selected | - | MPS(P) 400 | |
| Aperture Size | µm | 630 | |
| Leach Slurry Flow | m³/h | 230-270 | |
| Inter-tank Carbon Transfer | m³/h | 50 | |
| Eluted Carbon Transfer | m³/h | 50 | |
| Total Flowrate | m³/h | 280 – 320 | |

| Thickening, Pre-Oxidation and CIL | Units | Value |
|---|---|---|
| Open Area | % | 22.26 |
| Screen Volumetric Flowrate (Flux) Nominal | $m^3/m^2/h$ | 57.5 - 67.6 |
| Screen Volumetric Flowrate (Flux) Max | $m^3/m^2/h$ | 85.0 |
| Maximum Flowrate Available For Carbon Transfer | $m^3/h$ | 70.0 – 110.0 |
| Selected Flowrate For Loaded Carbon Transfer | $m^3/h$ | 50 |
| Selected Flowrate For Inter Tank Carbon Transfer | $m^3/h$ | 50 |
| Quantity of Interstage Screens | # | 7.00 |
| Dry Lifting Mass | t | 2.13 |
| Typical Operating Mass | t | 3.12 |
| Maximum Lifting Mass | t | 4.03 |

### 17.5.5 Acid Wash and Elution

The elution circuit processes one ±5 dry tonne batch per day of gold-laden carbon for subsequent gold recovery by electrowinning and smelting. Elution is based on the Anglo-American Research Laboratory (AARL), split-circuit process. The circuit essentially consists of loaded-carbon acid washing, elution and carbon regeneration.

**Acid Wash**

Hydrochloric acid (HCl) at a concentration of 33% w/w is transported in 1 $m^3$ IBCs (intermediate bulk containers) to the plant by road in bulk containers. Upon removal from the containers, the IBCs are stored in the chemical shed. An IBC container will be moved to the elution area as required. Hydrochloric acid at 33% w/w is pumped using a peristaltic pump (500-PP-462) as required to the dilute acid make-up tank (500-TK-286), where it is diluted to 3% w/w HCl for acid washing of the carbon.

Loaded carbon recovery screen oversize is delivered at a rate of ±5 t per daily cycle to the acid-wash hopper (500-TK-255) located directly above the elution column (500-ZM-256). Dilute hydrochloric acid is pumped (500-PP-287) into the acid-wash hopper. The loaded carbon is soaked for ±1 hour in the acid solution. The acid solution is then drained to the spent acid tank (500-TK-289). Wash water is then passed through the hopper and the carbon is washed until a neutral pH is achieved, with the rinse effluent also draining to the spent acid tank.

**Elution**

The neutralised carbon is dropped into the elution column. The solution used for the first two steps in the elution cycle, will be drawn from the intermediate solution tank, which was obtained from the previous cycle's wash and cooling steps. Cyanide and caustic will be added to this tank yielding a solution with 1%w/w cyanide and 3%w/w caustic respectively. The volumetric basis on which the elution cycle is based is as follows:

- 1 bed volume soak
- 5 bed volumes elution
- 5 bed volumes rinse
- 1 bed volume cooling

*Pre-Heat Step*

The elution solution in the intermediate tank is circulated using a single centrifugal pump through the elution heating circuit until a temperature of 125°C is reached. The outlet temperature of 125°C will initiate the next step.

*Soak Step*

The soak cycle is initiated once a temperature of 125°C is reached. One (1) bed volume will be pumped through the elution column via the heating circuit. The solution will then be cooled and pumped to the pregnant solution tank. The column pressure is maintained at 3BAR using pressure control valves and the temperature at 125°C using the heating circuit. Heat recovery will produce a final temperature of 60°C to the pregnant solution tank.

*Elution Step*

After the soak cycle the main elution is initiated. A total of five (5) bed volumes is pumped through the elution column. As per the soak cycle, the solution will be drawn from the intermediate solution tank, passing through the heating circuit and into the column. The solution leaving the column will again pass through the heating recovery circuit before being routed to the pregnant solution tank. The column pressure is maintained at 3BAR using pressure control valves and the temperature at 125°C using the heating circuit. Heat recovery will produce a final temperature of 60°C to the pregnant solution tank.

*Rinse Step*

After the completion of the elution cycle, the carbon is rinsed. This is done by adding softened water to the elution solution tank, and pumping to the intermediate solution tank, via the heaters, elution column and cooling section. The column pressure is maintained at 3BAR using pressure control valves and the temperature at 125°C using the heating circuit. A total of five (5) bed volumes will be transferred into the intermediate tank completing the rinse cycle. This step's water will form part of the soak and elution cycles discussed above.

*Cooling step*

The final step in the elution cycle is cooling down the carbon before transferring to the regeneration kiln. Again softened water is added to the elution solution tank, and pumped to the intermediate solution tank. During this step, no heating is required, and the heaters are switched off before the water is passed through the column. A total of one (1) bed volume is transferred to the intermediate tank. Also this water is used in the soak and elution steps.

#### Carbon Regeneration

On completion of the cold rinse cycle, the carbon within the column is transported to the carbon regeneration kiln feed sieve-bend via a T-piece on the outlet at the bottom of the column. Pressure is then applied to the elution column in order to supply adequate energy to the column contents and once the pressure set-point is reached, the bottom outlet is opened and the eluted carbon is effectively pressure-educted from the column and transported to the carbon regeneration kiln feed sieve-bend. Sieve-bend oversize gravitates to the kiln feed hopper (500-TK-270) which discharges into the kiln feeder (500-FD-291). The water drained from the sieve bend and any additional drainage water from the kiln hopper or feeder reports to the carbon quench tank (500-TK-273). The carbon is regenerated in the rotary kiln (500-XF-272) and then discharged into the barren carbon quench tank, from where it is pumped to the CIL barren-carbon dewatering screen (500-SR-279) via the screen feed box.

Make-up carbon is delivered in bulk bags and is added to the carbon quench tank when required.

#### Spillage Handling and Services

Spillage in the discretely concrete-bunded elution area’s spillage sump is pumped to the 1st CIL tank; spillage from the acid tank bund is pumped to the spent acid tank; whereas spillage accumulated in the discretely concrete-bunded carbon regeneration kiln area is pumped to the carbon quench tank. A safety shower supplied with potable water is located within the elution area.

The key parameters relating to acid wash and elution are summarised in Table 17-9.

**Table 17-9: Acid Wash and Elution**

| Acid Wash and Elution | Unit | Value | |
|---|---|---|---|
| | | Nominal | Design |
| **Loaded Carbon Screen** | | | |
| Type of Screen | type | Vibrating | |
| Aperture Size | mm | 0.630 | |
| Feed Slurry Flowrate | m³/h | 50 | |
| Feed Slurry Density | %Sw/w | 40.0% -45.0% | |
| Feed Slurry Density | %Sv/v | 22.1% | |
| Underflow Slurry Transport Method | type | Gravity | |
| **Acid Wash** | | | |
| Type of Acid Wash Vessel | type | Conical Tank | |
| Material of Construction | type | MSRL | |
| **Acid Wash Tank** | | | |
| Minimum Acid Wash Volume | BV | 2.00 | |
| Acid Wash Time / Recirculation | min | 30.0 | |
| Acid Wash Solution Strength - HCl | % | 3.00% | |
| **Rinsing** | | | |
| Rinse Volume | BV | 2.00 | |
| Rinse Time | min | 30.0 | |
| Cone Emptying Method | type | Gravity | |
| **Elution** | | | |
| Elution method | type | Pressure split AARL | |
| Material of construction | type | Stainless steel | |
| Operating temperature | ºC | 125-140 | |
| Operating pressure | kPa | 300 | |
| Carbon transfer method | type | Hydraulic | |
| Required quantity of elution batches per day | # | 1 | |
| Elution batch size | t | 5.00 | |
| Elution column selected size | t | 5.00 | |
| Design barren carbon loading | g/t | 50.0 | |
| Bed volume | $m^3$ | 10 | |
| Elution pump type | | Helical Rotor | |
| Pre-soak flowrate | BV/h | 1.00 | |
| Pre-soak volumes | BV | 1 | |
| Cyanide strength in solution | % | 1.00% | |
| Caustic strength in solution | % | 3.00% | |
| Pre-soak solution volume | $m^3$ | 10 | |
| Elution flowrate | BV/h | 2.00 | |
| Elution cycle time | min | 150 | |

| Acid Wash and Elution | Unit | Value | |
|---|---|---|---|
| Elution volume | $m^3$ | 50 | |
| Elution solution | type | Recirculation Tank solution | |
| Rinse flowrate | BV/h | 2.00 | |
| Rinse cycle time | min | 150 | |
| Rinse volume | $m^3$ | 50 | |
| Rinse solution | type | Softened water | |
| Cooling flowrate | BV/h | 2.00 | |
| Cooling cycle time | min | 30.0 | |
| Cooling volume | $m^3$ | 10 | |
| Total Eluate volume | $m^3$ | 60 | |
| Minimum Eluate tank volume | $m^3$ | 66.7 | |
| **Elution Heating** | | | |
| Elution heating medium | type | Thermic oil | |
| Elution heaters type | type | Diesel | |
| Elution heating required | kW | 1 750 | |
| **Regeneration** | | | |
| Regen feed hopper | | | |
| Dewatering means | type | Sieve Bend | |
| Feed hopper capacity | t | 6.00 | |
| | | Nominal | Design |
| Feed hopper capacity | $m^3$ | 13.0 | |
| **Regeneration Kiln** | | | |
| Reactivation capacity | kg/h | 300 | |
| Type of kiln | type | Horizontal | |
| Type of kiln heating | type | Diesel fired | |
| Temperature control | type | Automatic | |
| Regen. temp | ºC | 750 | |
| Filling ratio | % | 10.0% | |
| Kiln angle | ° | 0.700 | |
| Residence time at temperature | min | 15.0 | |
| Residence time at total | min | 25.0 | |
| Feed moisture | %w/w | 50.0% | |
| L:D ratio max | | 6.00 | |
| Regen. kiln running time per day | hrs | 18.0 | |
| Type of regen. carbon storage tank | type | Open tank | |
| Carbon transfer method | type | Hydraulic | |

### 17.5.6 Electrowinning and Gold Room

The electrowinning plant processes both intensive-leach reactor and CIL pregnant solutions to recover gold for downstream smelting.

The CIL pregnant solution tank (520-TK-320) and Intensive leach pregnant solution tank (520-TK-323) that supply pregnant feed liquor from the CIL elution circuit and ACACIA reactor to the electrowinning cells, as well as barren solution tank (520-TK-331) that recycles barren effluent back to the plant, are situated in a discrete, concrete-bunded area immediately adjacent to the gold room.

High-security gold room processing comprises electrowinning of gold from the pregnant solutions, followed by drying and fluxed smelting of the resultant gold sludge to a final doré bar product to be transported to the refinery. The gold room layout design accommodates both full security guard surveillance and second-level surveillance by remote control CCTV cameras with viewing facilities in the process manager and security foreman offices.

Toilet and crib-room facilities are provided within the secure area to minimize the frequency of human-traffic passage. Gold room ingress and egress are controlled and monitored via a proximity card and turnstile system.

Of the four identical electrowinning cells, two are dedicated to the CIL circuit (520-EC-326/327), and one is dedicated to the gravity circuit (520-EC-329). The fourth cell is operated as a common standby unit (520-EC-328). The cells are equipped with stainless steel anodes and stainless steel wool cathodes. A direct current is passed through the cells between the electrodes, and the electrolytic action results in the gold in solution plating out onto the cathodes. The electrowinning cells are provided with a fume extraction fan and associated hoods and ducting which expel fumes generated during the process to atmosphere.

CIL pregnant solution is pumped from the CIL pregnant solution tank to the CIL electrowinning cells, and recirculated to the CIL pregnant solution tank for the duration of the 18 hour process. Similarly, ILR pregnant solution is recirculated from the ILR pregnant solution tank via its electrowinning cells to the ILR pregnant solution tank for the same duration. On completion of the electrowinning cycle, the barren solution from both circuits is directed, using the same recirculation pumps, to the barren solution tank (520-TK-331). Dissolution reactor flushing water is also received into the barren solution tank. Barren solution is pumped back to the CIL circuit tanks 1 and 2. Manual addition of sodium hydroxide to the two pregnant solution tanks from the sodium hydroxide manifold is provided for. Spillage accumulated in the electrowinning area bund is pumped into the Barren solution tank (520-PP-336).

The loaded cathodes are manually hoisted from the electrowinning cells and taken to the cathode wash table (540-ZM-351) where the gold sludge is removed from them by high pressure water blasting, with the sludge reporting to the cathode wash sludge settling tank (540-TK-352) which also receives loosened sludge from the EW cell drains. Excess water is decanted from the settled sludge which is then dried in a drying oven (540-XT-353), prior to direct-smelting with flux in a furnace (540-XF-356) to produce doré bars for further refining. The furnace is provided with a hood, and appropriate ducting to deliver furnace gases to atmosphere. The doré bars are stored in a safe while awaiting delivery to the refinery.

Gold room spillage accumulates in a dedicated sump within the area and is pumped (540-PP-365) via a gold trap to the cathode wash sludge collection tank.

A safety shower is located in the gold room area.

The key parameters relating to electrowinning and the gold room and summarised in Table 17-10.

**Table 17-10: Electrowinning and Gold Room**

| Electrowinning | Units | Value | |
|---|---|---|---|
| Cell Type | type | Atm Sludging | |
| Mode Of Cell Operation/Feed | type | Parallel | |
| **Electrolyte source** | | **Gravity** | **CIL** |
| Volume to Be Treated per Batch | $m^3$ | 5.00 | 60 |
| Pregnant Liquor Au Concentration | mg/L | 1266 | 96 |
| Gold Extraction per Pass (Design) | % | 15.0% | 40.0%-50.0% |
| Total Design Gold Extraction per Electrowinning Batch Cycle | % | 98.0% | 98.0% |
| Barren Solution Grade (Au) | mg/L | 25.3 | 1.93 |
| Electrowinning Batch Cycle Time | Hrs. | 18.0 | 18.0 |
| Electrowinning Circulating Flowrate Required | $m^3/hr$ | 7.24 | 26.1 – 32.0 |
| Recommended Specific Eluate Flow Rate | $m^3/h/m^2$ | 30.0 | 30.0 |
| Rectifier Sizing | A | 1 000 | 1 000 |
| Selected Quantity of Cells | # | 1.00 | 3.00 |
| Cell Solution Temperature | ºC | 40.0 | 60.0 |
| Type of Cathode | type | S/Steel Mesh | |
| Type of Anode | type | S/Steel | |
| Quantity of Cathodes per Cell | # | 6 | |
| Cathode Sludge Removal | type | High Pressure Wash | |
| Total Barren Solution | $m^3$ | 65.0 | |
| Barren Solution Grade (Au) | mg/L | 2-4 | |
| Barren Tank Capacity | $m^3$ | 82 | |
| **Sludge Handling** | | | |
| Type | type | Sludge/Decant Tank | |
| Sludge Moisture | % | 50.0% | |
| **Drying and Smelting** | | | |
| Oven Type | type | 80L | |
| Quantity of Ovens | # | 1.00 | |
| Quantity of Trays | # | 5.00 | |
| Installed Power in Oven | kW | 30.0 | |
| Smelting Furnace Type | type | Diesel Fired | |
| Installed Power | kW | - | |
| Bullion Mould Capacity | - | 800 Oz | |
| Type of Crucible | type | A200 | |
| Cascade Trolley/ Cascade Mould Trays | # | 4 Bullion and 2 Slag | |

### 17.5.7 Cyanide Detoxification and Arsenic Leaching

The 1,000m$^3$ Detox/Arsenic Leach Tank (300-TK-234) serves a dual purpose, firstly destroying the cyanide in the slurry to below the required 20ppm WAD, using the SMBS/Air process, and secondly to leach the arsenic present in the ore before precipitating it in the 3 x 260m$^3$ tanks downstream of the detox tank. The SMBS/Air process is based upon conversion of CNWAD (weak acidic dissociable cyanides) to cyanate using a mixture of $SO_2$ and air, in the presence of a soluble copper catalyst at a controlled pH.

With the ore being Nickel rich, detox is carried out at pH levels of between 5.0 – 6.0 as opposed to the conventional pH of 8.0 – 9.0.

The addition of SMBS in this process has a dual purpose, firstly taking part in the detoxification reaction, and secondly lowering the pH (due to the formation of sulphurous acid) to such an extent favouring the leaching process.

Ferric Sulphate is also added to this tank to aid leaching of Arsenic. It has to be noted that precipitation of Arsenic is also encountered in this tank, but to a lesser extent when compared to the Arsenic precipitation tanks downstream.

A reactor pump circulates the slurry in the tank through a single high shear reactor which contacts the slurry with oxygen under high shear conditions. Additional oxygen is injected at the bottom of the tank using spargers. A MSP(P) interstage screen is then used to transfer the slurry via a linear screen to the precipitation circuit for further treatment before discharging to the tailings dam.

### 17.5.8 Arsenic Precipitation

Tailings that have undergone cyanide detox and Arsenic leaching as described above are transferred to the first of three agitated vessels (400-TK-178, 400-TK-187, and 400-TK-185) which make up the Arsenic precipitation and conditioning circuit.

The Arsenic precipitation circuit has 3 x 260m$^3$ tanks. The first two tanks, (400-TK-178/187), allow enough residence time for the leached arsenic to precipitate, whilst the purpose of the third tank (400-TK-185) is to allow for any pH corrections to be made before being pumped to the CTSF. Air is introduced to the bottom of each tank through a bubble cap. The air is supplied in excess, creating an environment suitable for oxidising the $As^{3+}$ to $As^{5+}$, making it stable to precipitate as an iron-arsenic complex.

Provision has been made to add lime to these tanks to allow for pH correction. The precipitation of arsenic is favoured at a pH range between 5.0 – 6.0. The pH of the material fed from the Acidic Leach Tank (300-TK-234) is already within this range, suggesting that the lime will only be used in extreme cases where the pH drops to levels below 5.

Ferric Sulphate which is used as an oxidant is also added to this tank.

The slurry from Arsenic Precipitate tank #2 (400-TK-187) gravity flows to the Conditioning Tank (400-TK-185) where the final pH correction with Lime is made to above 6.0, as required by legislation, before it is pumped to the final tailings disposal facility. This tank is also used for additional residence time for the precipitation of the $As^{+5}$ ion.

The precipitated solution gravitates to the final tailings disposal tank (400-TK-190) via a primary cross-cut/secondary vezin sampler system. Plant tailings are comprised of the following streams; detoxification/As precipitation circuit tails, carbon fines, spent acid and acid spillage. Plant tailings are pumped to the tailings storage facility (400-DM-197) by a tailings pump train comprising of two pumps operating in series (400-PP-191/192). The design caters for the installation of a standby tailings pump train (400-PP-193/194).

The gland water requirements for the tailings pumps are serviced by high pressure GSW supply pumps. The tailings GSW supply pumps are designed to deliver 1 bar above the delivery pressure of the secondary pump.

A spillage pump and safety shower will be installed in the detox/tailings disposal area.

Return water is pumped (400-PP-199/200) backed to the process water pond via a silt trap. A spillage pump is installed at the tailings return water pumping station (400-PP-201).

The key parameters relating to the cyanide detoxification and Arsenic Precipitation circuit are summarized in **Error! Not a valid bookmark self-reference.**.

**Table 17-11: Cyanide Detoxification and Arsenic Precipitation**

| Cyanide Detoxification and As Precipitation | Units | Value |
|---|---|---|
| **Cyanide Detox and Arsenic Leach** | | |
| Cyanide Detoxification Method | type | $S0_2$/Air |
| Total Detox Air Addition Rate | $Nm^3$/hr/tank | 500 |
| Required residence time for detox | min | 90.0 |
| Actual residence time of last leach tank | min | 216 – 233 |
| pH Set-Point | pH | 5.0-6.0 |
| Quantity of tanks | # | 1 |
| Detox selected tank volume | $m^3$ | 1,000 |
| $CuSO_4$ addition | g/t | 65 |
| SMBS addition | g/t | 770 |
| $Fe^{3+}$ Addition | g/t | 688 |
| Cyanide in final detox tailings | ppm | <20ppm WAD |
| **Arsenic Precipitation** | | |
| Quantity of Tanks | # | 3 |
| Selected tank volume | $m^3$ | 260 |
| SMBS addition | g/t | 230 |
| $Fe^{3+}$ Addition | g/t | 172 |

### 17.5.9 Reagents

**Caustic Soda Make-up and Storage**

Caustic soda (NaOH) is delivered to the plant in 1,000 kg bulk bags in “pearl” form. The bags are lifted by the reagent area overhead crane and delivered as required to the caustic soda bag-splitter cabin (700-ZA-440) located above the mechanically agitated caustic soda mixing tank (700-TK-441). The hoist lowers the bag rapidly onto the caustic bag-splitter, and the contents discharge into the mixing tank, where they are diluted with reagent water to solution strength of 20% w/v solution. The caustic soda solution is pumped into the caustic soda storage tank. The solution is pumped to required points of use (cyanide make-up, intensive leach reactor, strip solution make-up tank and electrowinning) using fixed speed helical screw pumps (700-PP-445/446) (operating and standby) as required.

Sodium hydroxide and sodium cyanide make-up share a common, discrete, concrete bund. Area spillage gravitates to a dedicated sump, and are pumped to the detoxification circuit (700-PP-456).

**Sodium Cyanide Make-up and Storage**

Sodium cyanide (NaCN) is delivered to the plant in 1,000 kg bulk bags contained in a wooden box. The bags are lifted by the dedicated 2 t cyanide bag hoist and delivered as required to the cyanide bag-splitter cabin (700-ZA-447) located above the mechanically agitated sodium cyanide mixing tank (700-TK-448). The hoist lowers the bag rapidly onto the cyanide mixing tank feed hopper bag-splitter, and the contents discharge into the mixing tank, where it is diluted with reagent water to solution strength of 20% w/v solution. The cyanide solution is pumped into the cyanide storage tank, from where it is pumped to required points of use. Dosing pumps (700-PP-452/453) feed cyanide to the CIL circuit via a pressure relieved manifold, while the intensive leach reactor and the column elution strip solution make-up receives batched cyanide via a fixed-speed pump (700-PP-454) as required.

The safety showers in this area are equipped with a high-flow switch which will alarm when said shower is in use and alert the control room operator to investigate in the cause of activation.

**Copper Sulphate Make-up and Storage**

Copper sulphate is delivered to the plant in 1,200 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the copper sulphate screen-hopper positioned above the agitated copper sulphate mixing tank. The reagent is washed through a screen into the mixing tank (720-TK-470) and mixed to a 20% w/v solution with reagent water prior to contents being dropped into the storage tank (720-TK-473). From here it is pumped to the cyanide detoxification circuit by a variable speed dosing pump (720-PP-474).

Copper Sulphate make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

**SMBS Make-up and Storage**

SMBS is delivered to the plant in 1,200 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the SMBS hopper positioned above the mechanically agitated SMBS mixing tank (720-TK-477). The reagent is washed into the mixing tank and mixed to a 20% w/v solution with water prior being pumped into the SMBS storage tank (720-TK-481). From here it is pumped to the detoxification and precipitation circuits via a variable speed dosing pumps (720-PP-482/240/311). A safety shower supplied with potable water is strategically located within the area.

SMBS make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

**Lead Nitrate Make-up and Storage**

Lead Nitrate is delivered to the plant in 1,000 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the Lead Nitrate hopper positioned above the mechanically agitated Lead Nitrate mixing tank (720-TK-671). The reagent is gravity fed into the mixing tank and mixed to a 20% w/v solution with water prior to being pumped into the Lead Nitrate storage tank (720-TK-676). From here it is pumped to the CIL circuit via a variable speed dosing pump (720-PP-677). A safety shower supplied with potable water is strategically located within the area.

Lead Nitrate make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

**Ferric Sulphate Make-up and Storage**

Ferric Sulphate is delivered to the plant in 1,000 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the Ferric Sulphate hopper positioned above the mechanically agitated Ferric Sulphate mixing tank (720-TK-661). The reagent is washed into the mixing tank and mixed to a 30% w/v solution with water prior being pumped into the Ferric Sulphate storage tank (720-TK-664). From here it is pumped to the detoxification and precipitation circuits via a variable speed dosing pumps (720-PP-665/241. A safety shower supplied with potable water is strategically located within the area.

Ferric Sulphate make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

**Flocculant Make-up and Dosing**

Flocculant is delivered to the plant in 25 kg bulk bags and manually loaded into the flocculant powder feed hopper (730-ZA-497). From here it is fed by the flocculant screw feeder (730-FD-501) into a blower/venturi system for transfer to a wetting head (730-ZM-509/510) and mixed with water to achieve 0.25% solution strength and fed directly into the mechanically agitated flocculant transfer tanks (730-TK-503/507) From here it is pumped to the pre-leach thickener using a variable speed pump (730-PP-506).

Flocculant area make-up area spillage is pumped by the spillage pump (740-PP-519) to the detoxification circuit feed tank.

**Hydrated Lime Make-up and Distribution**

Hydrated lime is delivered in 1,000 kg bulk bags which are transported to the plant by road in containers. The bags are lifted by the reagent area overhead crane and lowered rapidly onto the milk of lime addition bag cabin (740-ZM-511) as required, with the contents discharging into the feed hopper. The feed hopper is equipped with a vibrating system and variable speed rotary feeder (740-FD-514).

The rotary feeder meters the hydrated lime into the agitated milk of lime mixing tank (740-TK-516) where it is diluted to 20% w/v slurry. The lime slurry is pumped to the mill feed and the cyanide detoxification circuit (740-PP-517/518) in a ring-main arrangement. A facility is installed to allow for the addition of lime to the pre-leach thickener and CIL tank No. 1.

At the mill feed, the rate of primary dosage of lime into the CIL feed slurry is controlled by a /integrated-timer interface which operates a solenoid-controlled valve on the lime ring-main off-take.

Lime make-up area spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank. A safety shower provided with potable water is strategically located within the Lime make-up area.

**Table 17-12: Reagents**

| Reagents | Units | Value |
|---|---|---|
| **Lime** | | |
| Delivery method | type | Bulk bags |
| Delivery size | kg/bag | 1,000 |
| Type of lime | type | Hydrated lime |
| Equivalent % $Ca(OH)_2$ | % CaO | 90.0% |
| CIL consumption as 100% | kg/t | 1.8 |
| Detox consumption as 100% | kg/t | 0.0 |
| Consumption as 100% | kg/h | 310 |
| Consumption at 90% activity | kg/h | 340 |
| Physical form | type | Powder |
| Lime solids SG | $t/m^3$ | 2.25 |
| Lime addition system | type | Milk of lime Ringmain |
| Lime make-up strength | %S w/v | 20.0% |
| Number of make-up per day | # | 1.00 |
| Number of make-up/dosing tanks required | # | 1.00 |
| Total storage and dosing tank capacity | hrs | 34.0 |
| Selected storage tank capacity | $m^3$ | 50.0 |
| Lime Consumption | $m^3/hr$ | 1.71 |

| Reagent | Unit | Value |
|---|---|---|
| **Sodium Cyanide (NaCN)** | | |
| Delivery method | type | Bag in box |
| Delivery size | kg/box | 1 000 |
| Physical form | type | Briquettes |
| CIL consumption | kg/t | 0.65 |
| Elution consumption | kg/Batch | 100 |
| Dissolution | kg/Batch | 50 |
| Cyanide make up strength | %S w/v | 20.0% |
| Number of make-up per day | # | 1.00 |
| Number of make-up tanks required | # | 1.00 |
| Number of dosing tanks required | # | 1.00 |
| Total dosing tank capacity | days | 2. 47 |
| Selected storage tank capacity | $m^3$ | 30.0 |
| Selected dosing tank capacity | $m^3$ | 30.0 |
| Cyanide dosing method | type | Ring main |
| Velocity required in ring main (max) | m/s | 1.50 |
| Calculated consumption (nominal) | $m^3/h$ | 0.51 |
| **Caustic Soda (NaOH)** | | |
| Delivery method | type | Bags |
| Delivery size | kg | 1,000 |
| Physical form | type | Pearls |
| Caustic consumption elution | kg/Batch | 150 |
| Caustic consumption dissolution | kg/Batch | 12.0 |
| Caustic consumption electrowinning | kg/Batch | Intermittent |
| Caustic make up strength | %S w/v | 20.0% |
| Number of make-up tanks required | # | 1.00 |
| Number of storage/dosing tanks required | # | 1.00 |
| Total dosing tank capacity | days | 7.41 |
| Selected mixing tank capacity | $m^3$ | 5.5 |
| Selected dosing tank capacity | $m^3$ | 6.0 |
| Caustic Soda dosing method | type | Direct dosing |
| Velocity required in pipeline | m/s | 1.50 |
| Calculated consumption (nominal) | $m^3$/day | 0.81 |
| **Hydrochloric Acid (HCl)** | | |
| Delivery method | type | IBC |
| Delivery size | kg/Iso -container | 1 185 |
| Physical form | type | Solution |
| HCl delivered strength | % | 33.0% |
| 33% HCl SG | $t/m^3$ | 1.20 |
| Acid wash consumption at 100% strength | kg/Batch | 328 |
| Hydrochloric Acid dosing method | type | Direct from IBC |
| Calculated consumption at 33% strength (nominal) | $m^3$/batch | 0.829 |
| **SMBS** | | |
| Delivery method | type | Bags |
| Delivery size | kg/bag | 1,200 |
| Physical form | type | Granular |

| Reagent | Unit | Value |
|---|---|---|
| SMBS consumption (Nominal) | g/t | 1000 |
| Number of make ups per day | # | 1.00 |
| Make up strength | %S w/v | 20.0 |
| Number of make-up tanks | # | 1.00 |
| Selected make-up tank volume | m³ | 8.0 |
| Number of dosing tanks | # | 1.00 |
| Selected dosing tank volume | m³ | 15.0 |
| Dosing method | type | Direct dosing |
| Velocity required in pipeline | m/s | 1.50 |
| SMBS solution consumption (Nominal) | m³/h | 0.728 |
| **CuSO4** | | |
| Delivery method | type | Bags |
| Delivery size | kg/bag | 1,200 |
| Physical form | type | Granular |
| $CuSO_4$ consumption (Nominal) | g/t | 65 |
| Number of make ups per day | # | 1.00 |
| Make up strength | %S w/v | 20.0 |
| Total dosing capacity | days | 7.05 |
| Number of make-up tanks | # | 1.00 |
| Selected make-up tank volume | m³ | 8.0 |
| Dosing method | type | Direct dosing |
| Number of dosing tanks | # | 1.00 |
| Selected dosing tank volume | m³ | 8.0 |
| Velocity required in pipeline | m/s | 1.50 |
| $CuSO_4$ solution consumption (Nominal) | m³/h | 0.05 |
| **Lead Nitrate** | | |
| Delivery Method | type | Bags |
| Delivery Size | kg | 1 000 |
| Physical Form | type | Pearls |
| Leach Consumption | g/t | 25 |
| Lead Nitrate Make Up Strength | %Sw/v | 20.0% |
| Number of Make-Up per Day | # | 1.00 |
| Number of Make-Up Tanks Required | # | 1.00 |
| Number of Storage/Dosing Tanks Required | # | 1.00 |
| Dosing Tank Capacity | days | 13.70 |
| Selected Mixing Tank Capacity | m³ | 5.50 |
| Selected Dosing Tank Capacity | m³ | 6.00 |
| Dosing Method | type | Direct Dosing |
| Calculated Consumption (Nominal) | m³/day | 0.44 |
| **Ferric Sulphate** | | |
| Delivery Method | type | Bags |
| Delivery Size | kg/bag | 1 000 |
| Physical Form | type | Granular |
| $Fe^{3+}$ Consumption | kg/t | 0.86 |
| Number of Make Ups per Day | # | 1.00 |
| Make Up Strength | %Sw/v | 30.0% |
| Number of Make-Up Tanks | # | 1.00 |

| Reagent | Unit | Value |
|---|---|---|
| Selected Make-Up Tank Volume | m³ | 8.00 |
| Dosing Method | type | Direct Dosing |
| Number of Dosing Tanks | # | 1.00 |
| Selected Dosing Tank Volume | m³ | 15.00 |
| Dosing Tank Capacity | hrs | 8 |
| Velocity Required In Pipeline | m/s | 1.50 |
| $Fe^{3+}$ Consumption | m³/h | 0.42 |
| **Flocculant** | | |
| Delivery method | type | Bags |
| Delivery size | kg/bag | 25.0 |
| Physical form | type | Granular |
| Flocculant consumption | g/t | 30.0 – 40.0 |
| Total Flocculant consumption | kg/h | 4.4 – 5.8 |
| Number of make ups per day | # | 1.00 |
| Flocculant make up strength | %S w/v | 0.250% |
| 0.25% Flocculant solution density | t/m³ | 1.00 |
| Total storage and dosing capacity | hrs | 18.0 - 24.0 |
| Number of make-up/dosing tanks | # | 2 |
| Selected make-up/dosing tank volume | m³ | 21.0 |
| Dilution factor | factor | 1:10 |
| Flocculant dosing method | type | Direct dosing |
| Velocity required in pipeline | m/s | 1.50 |
| 0.25% Flocculant solution consumption | m³/h | 1.75 – 2.34 |
| **Activated Carbon** | | |
| Delivery method | type | Bags |
| Delivery size | kg/bag | 600 |
| Type of carbon in use | Mesh | 8 × 16 |
| Type of carbon in use | mm | 1.68 × 2.38 |
| Carbon bulk density | t/m³ | 0.480 |
| Carbon dry SG | t/m³ | 0.800 |
| Carbon wet SG | t/m³ | 1.37 |
| Consumption rate | g/t | 25.0 |

### 17.5.10 Water

#### Process Water

Return water from the tailings storage facility return water pond is pumped to the plant process water tank (600-TK-400) where it is joined by the overflow from the pre-leach thickener. Process water is supplied to the plant with two dedicated process water pumps (600-PP-402/403) (operational and standby).

Make-up water from the river system can be added to meet process water demand requirements.

**Clean Water**

Clean water is supplied to the plant from the clean water tank (600-TK-405), which receives make-up water from the river system via pumps 600-PP-420/422. The clean water tank provides for plant clean water and gland service water requirements with two dedicated clean water pumps (600-PP-406/407) (operational and standby) via a clean-water supply line.

**Potable Water**

Clean river water is supplied to the water treatment plant and potable water is supplied from the water treatment plant to the plant potable water tank (600-TK-412). Potable water is supplied to the plant via a dedicated potable water supply pump (600-PP-413).

**Fire Water**

Fire water is supplied from the bottom section of the clean water tanks (600-TK-405) to the plant via a dedicated vendor package fire water supply pump (600-PP-419), fire water jockey pump (600-PP-417) as well as fire water diesel pump (600-PP-416) in case of power failure.

The water usage is summarised in **Error! Not a valid bookmark self-reference.**.

**Table 17-13: Water**

| Water Services | Units | Value |
|---|---|---|
| **Return Water** | | |
| Return water % of tailing disposal (wet season) | % | 100% |
| Return water % of tailing disposal (dry season) | % | 50.0% |
| **Process Water Supply** | type | Tank |
| Capacity | $m^3$ | 200 |
| **Process water storage** | type | TSF pond |
| Capacity (nominal) | $m^3$ | 60 000 |
| **Clean Water Storage** | type | Tank |
| Capacity (total) | $m^3$ | 500 |
| Capacity (total) excl reserve | $m^3$ | 300 |
| Capacity (total) excl reserve | hrs | 4.3 |
| **Plant Potable Water Storage** | type | Tank |
| Selected size | $m^3$ | 30 |
| **Fire Water Supply** | type | Clean water tank reserve |

### 17.5.11 Plant Services

**High Pressure Air Services**

The compressed air purification system consists of high efficiency filters (610-FL-419/435/421) and a dryer (610-DR-420) which is installed between the filters. The filters remove contaminants such as water, oil and solid particles from the compressed air stream. The dryer is responsible for the removal of moisture within the air delivering high-purity air to the receivers.

The instrument air receiver is designed to hold 5.0 $m^3$ of air at 7.5 bar.

### Oxygen Plant

A Vendor-package, Pressure Swing Adsorption (PSA) oxygen plant, provides piped oxygen to the Pre- Oxidation and CIL tank spargers as well as the Pre-Oxidation high shear reactor.

### Low Pressure Air Services

A low pressure air circuit is comprised on two low pressure air blowers (610-HA-647/648) which supply low pressure air to the cyanide detoxification circuit at a minimum pressure of 171 kPa.

The plant services are summarised in **Error! Not a valid bookmark self-reference.**.

**Table 17-14: Plant Services**

| Plant Services | Units | Value |
|---|---|---|
| **Compressed Air** | | |
| Plant general | $Nm^3/h$ | 2.5 |
| Workshops | $Nm^3/h$ | 2.50 |
| Instrument air | $Nm^3/h$ | 7.00 |
| Instrument air receiver | $m^3$ | 5.0 |
| Total compressed air | $Nm^3/h$ | 12.0 |
| Air pressure (maximum) | kPa | 750 |
| **Oxygen Requirement** | | |
| Usage | t/day | 5.0 |
| Oxygen Supply Pressure Required | kPa | 650 |
| PSA plant capacity | t/day | 5.8 |
| **Low Pressure Air** | | |
| Air Supply Rate | $Nm^3/h$ | 1,500 |
| Air Supply Pressure Required (Minimum) | kPa | 170 |

A process plant block diagram is shown in Figure 17-1 and a schematic of the process flow is shown in Figure 17-2.

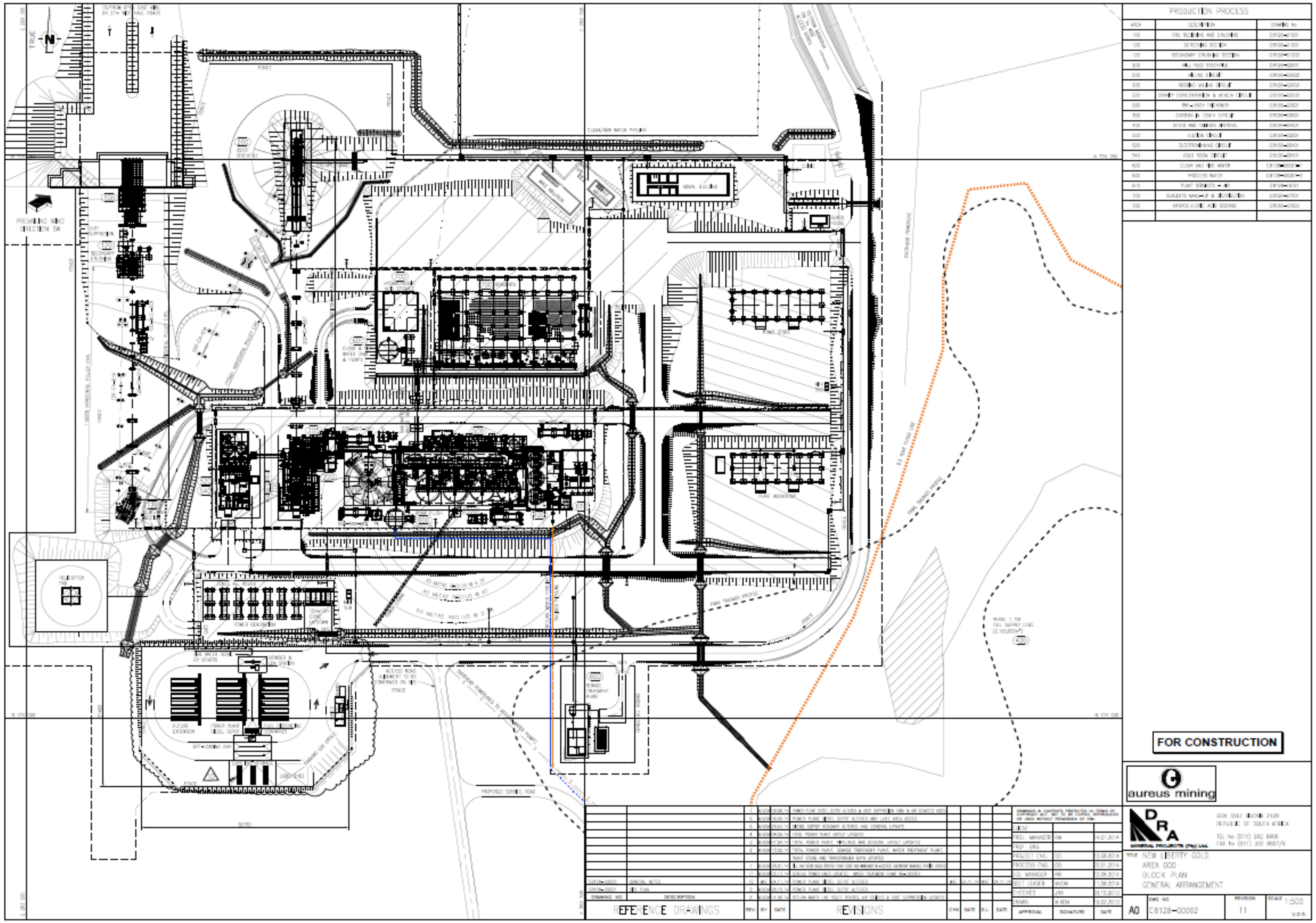



**Figure 17-1: Plant Layout**

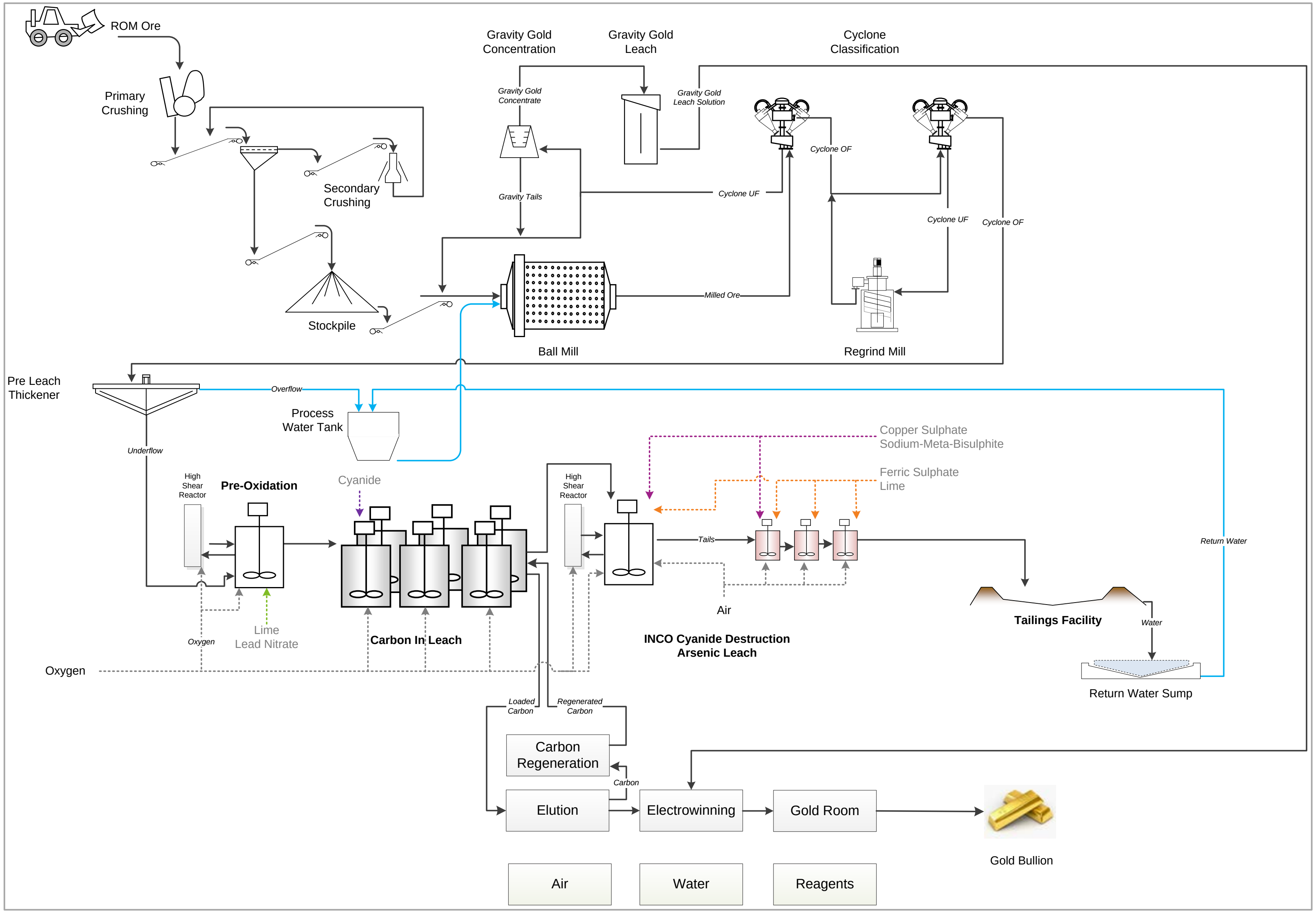



**Figure 17-2: High Level Process Flow Diagram**

# 18 PROJECT INFRASTRUCTURE

## 18.1 Introduction

The Project is located approximately 100 km north–north-west of the Liberian capital, Monrovia. The Freeport of Monrovia, a deep-water port which can accommodate third generation container ships, is privately run under a concession from the government, is one of four main ports in Liberia and is the only port with cargo and oil handling facilities.

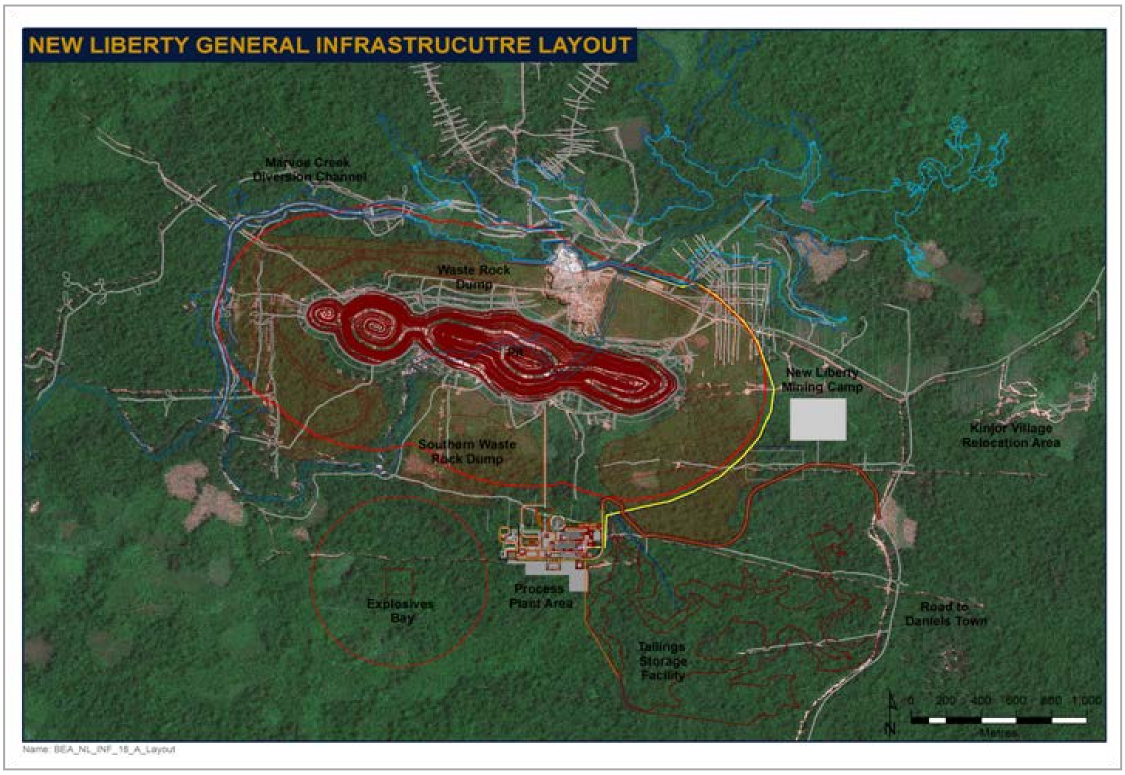
The current road to the Project site allows for easy access for larger cargo as the project infrastructure grows, and to date the ball mill and crushers and all other major infrastructure, including elements of the mining fleet have all been carried to site successfully. Aureus has widened and re-graded this laterite road, made improvements to road drainage, and upgraded and installed concrete culvert type bridges. Secondary roads on the Bea-MDA licence area, built by Aureus, provide access across the property. Due to the laterite nature of the roads, access is all year round, including during the height of the rainy season.

The original infrastructure consisted of an exploration camp that has offices, staff accommodation, messing facilities, core storage facilities. The exploration camp is within the 500 m blast radius of the open pit operations and will therefore need to be closed. The exploration camp is currently being used but will be evacuated in favour of Camp David once the construction of the process plant is complete.

The infrastructure currently being constructed at New Liberty will support the mining, plant and construction operations and can be summarized as follows:

- Mining infrastructure,
- Process Plant
- Operational Accommodation Facilities
- General Mine site infrastructure
- Power supply and distribution
- Process tailings management - tailings storage facility (TSF)
- Marvoe Creek Diversion Channel (MCDC)
- Waste rock dump.

This section details the facilities that are currently under construction at the Project. The site layout below shows the position of the process plant, tailing storage and water storage dams relative to each other and the surrounding topography.




**Source:** ***Aureus, 2013***

**Figure 18-1: General Infrastructure Layout**

## 18.2 Mining and General Infrastructure

### 18.2.1 Introduction

The mining infrastructure has been designed and constructed to provide adequate support for the duration of the LOM and includes the following:

- Main workshop and repair facilities for the maintenance of the mining fleet and major mining equipment.
- Explosive storage, which will be located away from the main facilities as per relevant international codes of practice.
- Change house, administration office and security office.

### 18.2.2 Mining Equipment Workshop

This workshop is situated on the western edge of the ROM pad and will be fully equipped with all the facilities to maintain the mining fleet and ancillary equipment. Provision has been made for 6 service bays with enough head clearance to allow the servicing of 100 tonne dump trucks. These service bays will be serviced by a 20 tonne overhead crane and will have suitably engineered flooring (200 mm) to provide for heavy loads. One of the bays will be demarcated for use by tracked vehicles and excavators, and the floor will be suitably strengthened by means of rail sections. The drill rigs will have a dedicated area in the workshop. Other facilities include a hydraulic and hose repair room, tyre repair bay, washing bay, workshop office and ablutions. The fenced area outside the workshop will be suitably compacted and provision made for adequate drainage in order to ensure that heavy vehicles can negotiate the terrain in wet conditions. Parking areas for equipment awaiting service and collection is provided for inside the fenced area.

### 18.2.3 Fuel Storage Area

Diesel is required to operate the power generators which provide the power to the processing infrastructure and the camp as well as for the mining fleet and associated workshop. The forecast monthly consumption of diesel is approximately 2 million litres.

The fuel storage and the fuelling facility for both diesels and lubricants are provided by an external contractor, which will include a fire suppressant system on the Modular Pump House and 100kg dry and foam chemical extinguishers for all other areas. The fuel farm and lubricant storage area will be managed by the external contractor who will be responsible for the following:

- Storage facility
  - Design, construction and commissioning of the storage facility
  - Design, construction and commissioning of loading bay
  - Supply of products from Monrovia to the fuel storage facility at the Project
- Operations
  - Supply of diesel to the storage facility on the mine site
  - On-site offloading of fuels and lubricants
  - Transfer operations for the trucks to the main storage
  - Handling of used diesel and oil and general management of hydrocarbons
- Filtration
  - Filtration solution implementation
  - Filters and filtration equipment maintenance
- Technical support
- HSEQ – to specific guidelines.

The storage facilities will include the following:

- 12 x P75 tanks (Main Fuel Farm - 869,400lts), 2 x P69 tanks (Workshop Fuel Farm - 134,240lts) – Grand Total = 1,000,364lts / 1000.4$m^3$ across the project site
- Bunded bulk lubes storage
- All civils, including a concrete bund (All areas are lined underground with drainage that is attached to sumps that lead to an oil separator)
- Distribution piping and filtration equipment
- Bunded old fuel and oil storage area
- Fire fighting equipment.

Based on the diesel consumption rates, the fuel in the storage facility will be able to run the processing plant and camp for approximately 18 days before refilling (plus 9 days of safety stock). A fleet of trucks will be used by the external contractor to deliver product from the port at Monrovia to the Project site.

### 18.2.4 Explosives Storage

Bulk explosives will be supplied and stored by an external contractor. The Explosives Magazine will be located in an area to the south-west of the pit, which is outside the pit blasting zone. Care has been taken to place all other infrastructure outside a 500 m radius of the explosives storage magazine. The Explosives Magazine will be used by the explosives contractor (Manex) to store detonators and boosters. The area will be securely fenced and guarded and provision has been made for adequate lighting at night.

A site has also been established for the manufacture of emulsion and this will similarly be securely fenced and lit. This site is outside the 500m blast zone, adjacent to the plant access road.

### 18.2.5 Change house, Administration Office and Security Office

A male and female change house will be constructed of equal size.

An administration office will be constructed consisting of a few offices for senior people and an open plan area for the majority of the staff.

Access security facilities will be set up at strategic points around the mine.

## 18.3 Processing Plant and Administration Facilities

### 18.3.1 Introduction

The processing plant and administration facilities include the following:

- Access roads within the plant site area
- Process Plant and Plant administration buildings; including, but not limited to, security office, change house, workshop, main administration offices, medical facility, assay laboratory and warehouses.
- Sewage treatment and disposal
- Water treatment plant
- Water services including of raw water abstraction, potable water and fire water.
- Accommodation
- Security
- Communications

### 18.3.2 Access Roads within Processing Plant

The roads within the plant area will be stripped of organic material and compacted, which will facilitate access to the requisite areas within the plant. Drainage ditches and culverts will be created in accordance with the requirements for site drainage.

### 18.3.3 Process Plant

The process plant will consist of the following main elements:

- ROM Tip

- Primary Crusher
- Screening
- Secondary Crusher
- Milling
- GRG and Dissolution Reactor
- Pre-Leach Thickening
- CIL
- Acid Wash, AARL Elution & Carbon Regeneration
- Electrowinning and Gold room
- As and CN Destruction
- Tails Pumping

### 18.3.4 Plant Buildings

The following buildings will be constructed:

- Security Office
- Plant Change Houses – male and female
- Plant Control Room
- Process Plant Equipment Workshop and Offices
- Plant administration building
- Plant Store
- Reagents Building

*Security Gatehouses*

The security gatehouses will be located at the main site entrance, the Process Plant entrance and also the entrance to Camp David. The gatehouses consist of the following:

- Protection services office
- Protection services search area

These gatehouses will control all vehicles in and out of the mine area. The security search zone will provide the security personnel with the means to conduct individual body searches, isolation rooms and general scanning. The main access gate to the plant will have a security office for the control of vehicle access to the plant

*Change Houses*

Ablutions for 50 males and 10 females. The plant will operate in shifts. There will not be more than 60 people per shift and the ablutions only need to accommodate one shift at a time.

The female ablution will have 26 lockers and the male ablution will have 92 lockers. Employees will be supplied their own locker 1,800 mm high x 300 mm wide x 450 mm deep and there will be a laundry area to wash overalls.

*Plant Administration Buildings*

Buildings located in the plant area consist of security offices, change house, process plant equipment workshop, control room, general administration offices, medical facility, assay laboratory, reagent warehouses and spares / store warehouse.

*Plant Control Room*

A dedicated plant control room is to be located in a double container arrangement. The top container will house the control room and the bottom container will house one of the MCC units. The control room will house the SCADA system and provide operators with an elevated view of the entire plant.

*Process Plant Equipment Workshop*

A workshop with an area of 480 $m^2$ has been constructed adjacent to the process plant to enable repair of plant equipment. The workshop consists of a steel framed building equipped with a 3 tonne overhead crane and will have bays for servicing light vehicles. The workshop will have separate areas for mechanical and electrical repairs. Provision has been made for oil separation of any water leaving the facility. Offices for supervisory, workshop store, maintenance and planning personnel will be provided in the form of a modular building situated next to the workshop.

*Mine Administration Buildings*

The administration building is a 'Hydrafrom' block building. The building will include general areas for engineering, administration personnel and offices for the general manager, mining manager, plant superintendent, administration superintendent, technical manager, chief geologist, plant maintenance superintendent, chief mining engineer, SHEQ Manager and the chief security officer.

*Plant Store Building*

A store with an area of 480 $m^2$ has been constructed adjacent to the process plant. The store consists of a steel framed building.

*Assay Laboratory*

The assay laboratory will be in the form of a containerized lab supplied to the project and managed by an independent third-party laboratory service provider. This laboratory will conduct all of the onsite test work for the processing plant and the grade control and will be located adjacent to but outside from the entrance of Camp David.

*Medical Facilities*

Aureus will provide an equipped medical facility, which will allow for the treatment of any injuries during the construction and operational phases, as well as treatment of sick personnel. This will be located adjacent to but outside from the entrance to the mine accommodation camp, Camp David.

### 18.3.5 Sewage Treatment and Disposal

Sewage from the plant will be treated with a sewage treatment system.

### 18.3.6 Water Treatment Plant

A water treatment plant will be installed to ensure potable water is available in areas such as the change houses and plant administration building.

### 18.3.7 Water Services

The Project is located in a net water surplus climate. To minimize the volume of non-contact surface rainfall run-off reporting to the TSF or the open pits, water diversion channels and ditches will be constructed. The preliminary water balance developed by DRA for the site indicates that there will be a surplus of contact water that will require discharge to the environment. This surplus contact water will be discharged to the Tailings Storage Facility.

*Raw Water Supply Dam*

The design of this facility is based on meeting or exceeding agreed design criteria which comply with World Bank and other international standards. It has been incorporated in the design of the MCDC, under Epoch's scope of work.

A water balance was developed and was used as the basis for sizing the water storage dam and the raw water requirements. Raw water stored in the water supply dam will be pumped to the process plant for make-up operations during the plant start-up and during periods where the return water from the tailings storage facility is insufficient to meet the requirements of the plant.

*Potable Water Distribution*

Raw water will be supplied to the potable water treatment plant from the plant raw water tank and in turn through a pipe line from the raw water dam.

Potable water will be reticulated to all areas of the plant including a safety shower header tank.

In respect of potable water for human consumption, dedicated potable water supply boreholes will be drilled. This will form part of the open pit dewatering strategy. Current groundwater quality indicates that the resource is suitable for potable use according to the WHO Guidelines for Human Consumption.

*Fire Water Distribution*

The fire water system is a dual power system that can use electricity and diesel. The electric powered pump is used in the event of a fire and the diesel pump is used in the event of a fire where electrical supply is unavailable – the pump is also available as a standby unit. A jockey pump will be provided to maintain the pressure in the fire water header during normal plant operation.

The fire water system consists of a fire water distribution system with hydrants strategically positioned around the process plant.

### 18.3.8 Operational Accommodation Facilities

The mine camp "Camp David" has been constructed to house up to approximately 500 individuals during the construction phase and will be scaled back to be able to house approximately 129 individuals for the production phase. This camp includes the following infrastructure:

- Kitchen has been constructed using modular construction materials
- Camp dining room has been constructed using Hydraform bricks
- Entertainment area and gym has been constructed using Hydraform bricks
- Laundry
- Potable water plant
- Sewage disposal plant
- Communal TV room
- Gym
- Volley ball court/mini soccer field
- Administration offices for the Catering Contractor
- Gate House with search rooms and toilet
- Medical facility

The mine accommodation camp is designed as a combination of a permanent wooden structures and temporary "traditional earth wall" accommodation.

129 Individual Rooms have been constructed using local timber. This is part of the permanent accommodation. Some rooms have en-suite bathrooms while the majority have shared ablutions.

A ten room guest house has also been constructed to accommodate senior visitors as well as the plant commissioning engineers. The mine manger also has an accommodation building.

Over 40 traditional houses have been constructed with concrete floors and cement plastered walls. Each of the houses accommodates 8 people. They have communal ablutions. The sewage plant will treat the water in accordance with South African DWEA General Limits for the release of treated water into the environment.

### 18.3.9 Security

The plant site will be enclosed within a security fence. Access to the plant area will be via gates located on access roads to the site. Additional fencing will be provided for further safety and security within process plant areas, such as power plant, fuel storage, gold room area, transformers and substations, as required.

CCTV cameras will be installed at strategic locations in the plant for surveillance purposes. The cameras will be integrated with the plant's overall network, which will be the responsibility of the security manager.

### 18.3.10 Communications

The Project site is currently serviced with a private company phone and internet services via satellite link from an international service provider—this has recently had an upgrade for additional bandwidth to service the commissioning activities. A dedicated satellite system has also been installed at the new plant area for operations.

There is also mobile phone coverage at the Project site from a cell tower located at the junction of the laterite Daniels Town access road and the Project Site, and a second mobile telephone service provider was at the time of this report erecting a further cell tower adjacent to the process plant site to give added network flexibility.

### 18.3.11 Access Road to Site

The Project site is 100 km from Monrovia, 80 km of which is on a tarmac road with the remaining 20 km on a new laterite road from Daniel's Town. Aureus has widened and re-graded the laterite road in order that the transportation of the large volume of infrastructure required to be transported from the port at Monrovia to the Project site can be undertaken. Concrete culvert bridges and cross drains have also been constructed to manage rain water.

All the main plant and mine equipment for the project has been delivered to site. This includes the mine vehicles and mill.

## 18.4 Power Supply and Distribution

### 18.4.1 Power Supply

An external contractor will provide the power generating capability at the Project site which will be used for both the processing plant and the mine camp. Liberia has a limited power grid in Monrovia which does not serve the local district around Project, and as such, Aureus is responsible for generating its own power.

The external contractor will provide an 11 kV, 10.8 MW, diesel driven, build, own, operate and transfer (BOOT) power station at the Project. The generators will be housed in 12 m shipping containers.

The power plant has been designed to be self-sufficient and shall have its own fence line to allow for potential maintenance and servicing agreements to be executed with minimal disruption to the main processing facility. The 11kV feeds from the generators will be run via cables to the plant main 11kV substation. Synchronization will be performed at the generator alternator circuit breakers, with control and protection of the supplies being performed by the power plant contractor. Real estate has been allowed for in respect of the future inclusion of additional generator sets for power plant expansion should this be required.

The supply of diesel to the power plant shall be via the bulk diesel storage facility located adjacent to the power plant fence line. The bulk diesel storage facility shall be built and operated by the diesel supply contractor, with diesel fuel being free-issued to the power plant contractor. A diesel day tank has been allowed for within the power plant area.

Other power plant infrastructure includes: a local control room, a workshop, an oil change station and a transformer for supply of power to auxiliary loads. Power factor correction equipment is not required as no connection to a utility is being made and diesel consumption is not dependant on the power factor.

The workings for the 6-genset power station configuration can be summarised as shown in Table 18-1.

**Table 18-1: Power Station Configuration**

| | |
|---|---|
| **Genset Units** | |
| Prime power output | 1,965 kW (m) |
| Prime power output | 1,827 kW (e) alternator terminals |
| Unit prime power output | 1,800 kW (e) alternator terminals |
| **Output** | |
| Average demand | 7,505 kW (e) |
| Guaranteed power MD | 7,200 kW (e) (99.5% availability) |
| Installed units | 6 |
| Installed capacity | 10,800 kW (e) at 11kV busbar |
| 1h overload capacity | 11,880 kW (e) at 11kV busbar |
| **Fuel Consumption** | |
| g/kWh (m) | 196 (mechanical at crankshaft) |
| Alternator efficiency | 96.0% |
| g/kWh (e) | 204.2 (electrical at alternator terminals) |
| Transformer losses | 0% (alternators at 11kV) |
| Fuel consumption | 204.2 g/kWh (at 11kV) |
| Fuel consumption | 0.234 L/kWh |
| **Power Levels with 6 Gensets Installed** | |
| 11,880 kW | 1 hour overload capability when all gensets available |
| 10,880 kW | installed prime power capacity |
| 7,200 kW | maximum continuous running capacity of 4 gensets (normal operation) |
| 8,100 kW | continuous average load capability of 5 gensets (75% of 9,000 kW) |
| 5,940 kW | capacity of 3 remaining gensets if 1 genset trips during normal genset operations (3 x 110%) |
| 7,200 kW | guaranteed power output for 99.5% of the time |
| 6,424 kW | annual average load (=4,021,826 kWh pm) |

There may be further opportunities in the future to optimise the power generation capability.

### 18.4.2 Power Distribution

Power from the power plant shall be transferred at 11,000V, 50Hz via individual feeders from each of the generators to the plant main 11kV substation.

Medium voltage electrical power shall be distributed throughout the main processing facility via 11,000V XLPE cable. Low voltage electrical power distribution shall be distributed to loads (motors, distribution panels, light fittings etc.) via 1000/600V PVC cable, which shall generally be run above ground on cable ladder, or buried where the use of cable ladder is not appropriate.

The main electrical power consumer shall be the 11,000V 3.5MW ball mill and 1.1MW Vertimill motors, which shall be supplied from the plant 11,000V substation. In order to mitigate against the substantial increase in electrical current normally associated with motor starting events, the mill motors shall be of the wound rotor type and shall utilize a liquid resistance starter during the mill starting sequence.

Motor control centres shall nominally operate at 525V, 50Hz, for supply of electrical power to low voltage motors. Other lighting and small power loads shall be rated for 380V (3 phase) and 220V (single phase), 50Hz.

Power to the plant infrastructure and the New Liberty accommodation camp shall be supplied from the plant 11,000V substation via overhead line.

Pit dewatering and raw water intake pumps will be diesel powered.

Emergency electrical power has not been provided for within the main plant area but has been allowed for in the accommodation camp where the diesel generator used during the construction phase shall be retained and act as the emergency backup generator.

The specification and selection of electrical equipment has been in accordance with South African Standards (SANS Standards).

### 18.4.3 Process Tailings Management – Tailings Storage Facility Introduction and Design Criteria

As a part of the construction of the project, Epoch Resources (Pty) Ltd (Epoch) was appointed to undertake an Optimization Study and Detailed Design of the Tailings Storage Facility (TSF) associated with the project.

Epoch's brief for the optimization study was to:

- Review the Golder's Feasibility Study TSF option;
- Evaluate the terrain for alternative sites south of the open pit;
- Develop a proposed design philosophy for the selected TSF option; and
- Compile a BoQ and capital cost for the selected TSF option

There are no Liberian guidelines related to the design, operation and closure of tailings storage facilities and water diversion systems. In the absence local guidelines, the following internationally recognized publications and industry standards were used to develop site specific design criteria:

- International Financial Corporation guidelines (IFC, 2007);
- International Committee on Large Dams (ICOLD) - Various Manuals and Bulletins;
- Canadian Dam Association (CDA) - Dam Safety Guidelines (CDA, 2007);
- Mining Association of Canada (MAC) - A Guide to the Management of Tailings Facilities (MAC, 1998), and Developing an Operations, Maintenance and Surveillance Manual for Tailings and Water Management Facilities (MAC, 2003);
- Australian Committee on Large Dams (ANCOLD) (1999); "Guidelines on Tailings Dam Design, Construction and Operation";
- South African Committee on Large Dams;
- Department of Mines and Petroleum, Western Australia (1999); "Guidelines on Safe Design and Operating Standards for Tailings Storage";
- SANS Code of Practice for Mine Residue Deposits (SANS 10286); and

- The Cyanide Code Standard of Practice.

In addition to the above, cognisance of the following measures have been accounted for in the TSF design:

- DRA designed the processing plant and detox circuit to conform to the Cyanide Code - Standard of Practice 4.4 by implementing measures to limit the concentration of WAD cyanide in the TSF to a maximum of 50 mg/l; and
- The TSF was designed and constructed to conform to the:
    - Cyanide Code Standard of Practice 4.3 and 4.5;
    - The IFC Emissions and Effluent Guidelines of the Environmental, Health, and Safety Guidelines – MINING. This will be achieved by implementing measures to limit discharges to surface waters not exceeding 0.5 mg/l WAD cyanide and 0.1 mg/l of Arsenic (95% of the operating time for IFC standard).

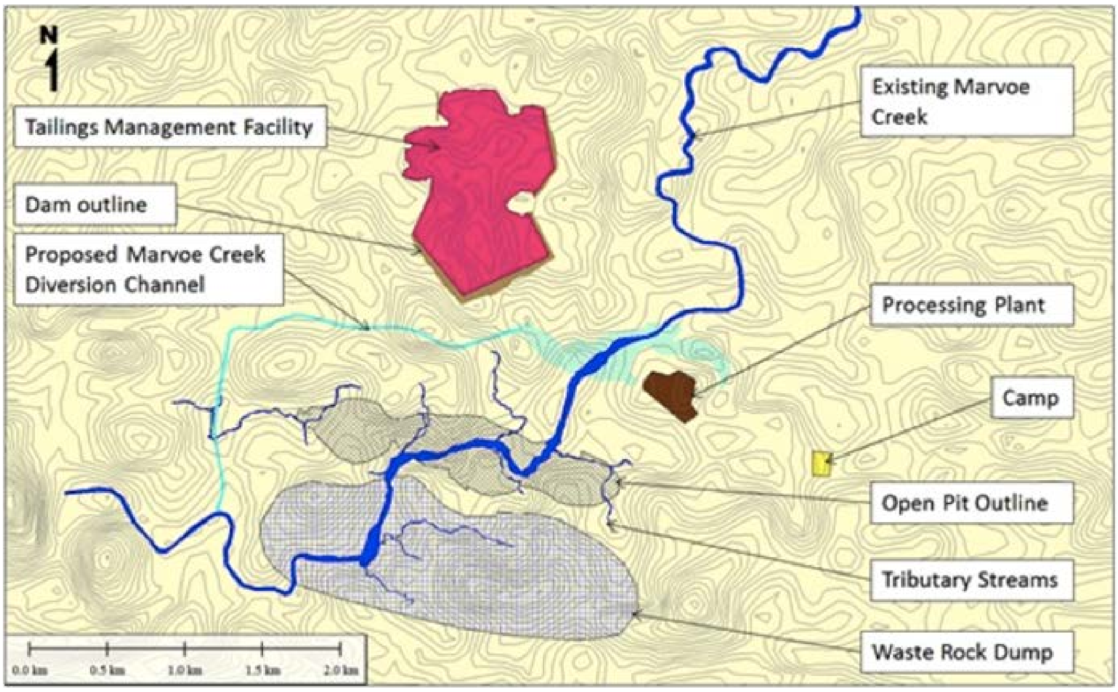
The mining process is expected generate a total of approximately 9.4 Mt of tailings over the life-of-mine. The dry density of the deposited tailings has been estimated at 1.36 t/m$^3$, requiring storage of a total of 6.9Mm$^3$ of tailings. The design criteria and assumptions developed for the TSF are as summarised in Table 18-2 below.

**Table 18-2: TSF Design Criteria**

| Parameter | Design criteria |
|---|---|
| Life-of-mine | 8.5 years |
| Tailings production rate | 1.1 Mt/annum |
| Specific gravity of tailings | 2.8 |
| Void ratio of tailings (assumed) | 1.0-1.3 |
| Dry density of deposited tailings | 1.36 t/m$^3$ |
| Slurry density | 1.36 t/m$^3$ |
| Tailings PSD | 90% passing 75 micron |
| Percentage solids | 41% |
| Basin lining | Unlined |
| TSF dams slope stability | 1.36 t/m$^3$ |
| Percentage solids | 41% |

### 18.4.4 TSF Site Selection and Location

A Feasibility study was undertaken by Golder, which identified a preferred TSF site and associated design. The study identified a site north of the open pit as the preferred TSF site, refer to Figure 18-2.




Source: *Epoch, 2013*

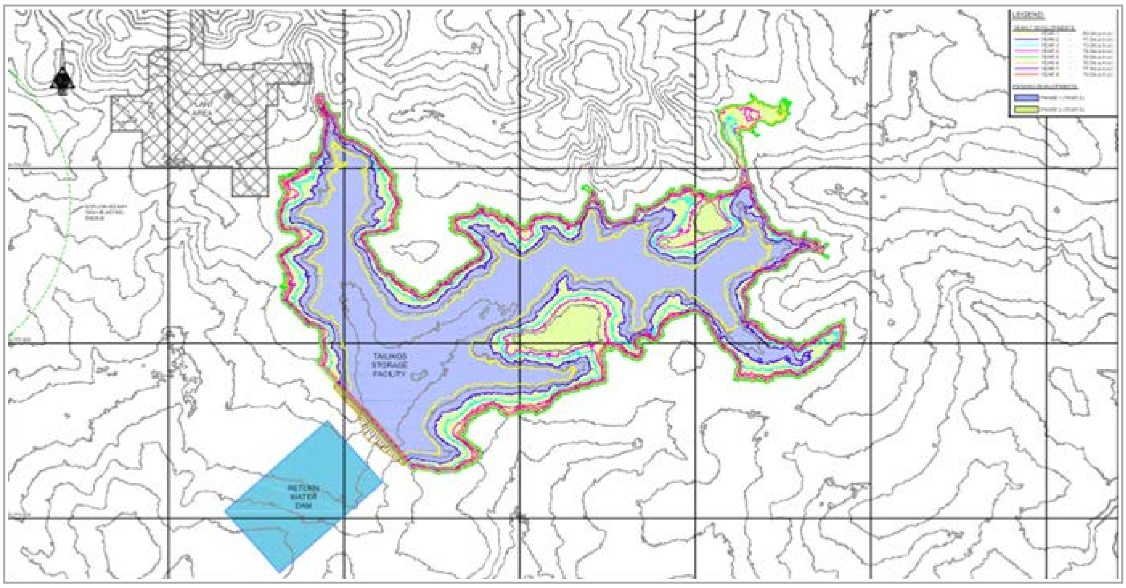
**Figure 18-2: TSF Layout as per Golder's Feasibility Study**

During the optimization study various aspects of the mine layout changed in terms of location of infrastructure, including the process plant, waste rock dumps and TSF. TSF sites located in the south of the site and open pit had been considered during the Feasibility Study but were considered less favourable options, with some sites falling outside of the mining lease area. It should be noted that the survey information used during the Feasibility Study was not of a sufficient level of accuracy.

For the optimization study, an airborne LiDAR survey dataset of the mine site was made available, and due to this increased level of accuracy and the changes in the mine infrastructure layout, a more favourable TSF site from a layout, proximity to plant, risk and capital cost perspective compared to the Feasibility TSF site was identified in the south of the mine lease area. Figure 18-3 shows the location and overall development of the southern TSF site over the life-of-mine (LOM).

The main advantages this site offered over the Feasibility Study site are:

- Valley dam TSF with a single embankment/starter wall of significant less earth fill volumes. The tailings dam is constructed as a self-raising upstream dam in year 3.5 onwards as the rate of rise is below 1.35 m/yr and typically around 1.0 m/yr, refer to Figure 18-4;
- Close proximity to and downslope of the plant, thus less pumping distance and slurry piping;
- TSF is located downstream of the open pit and creek diversion, resulting in a reduced potential risk profile should a TSF failure occur; and
- The slurry pipeline does not cross the creek diversion, thus the environmental risk due to potential pipe bursts and leakages into the creek diversion are nullified.




**Source:** ***Epoch, 2013***

**Figure 18-3: Southern TSF Site Following the Optimization Study**

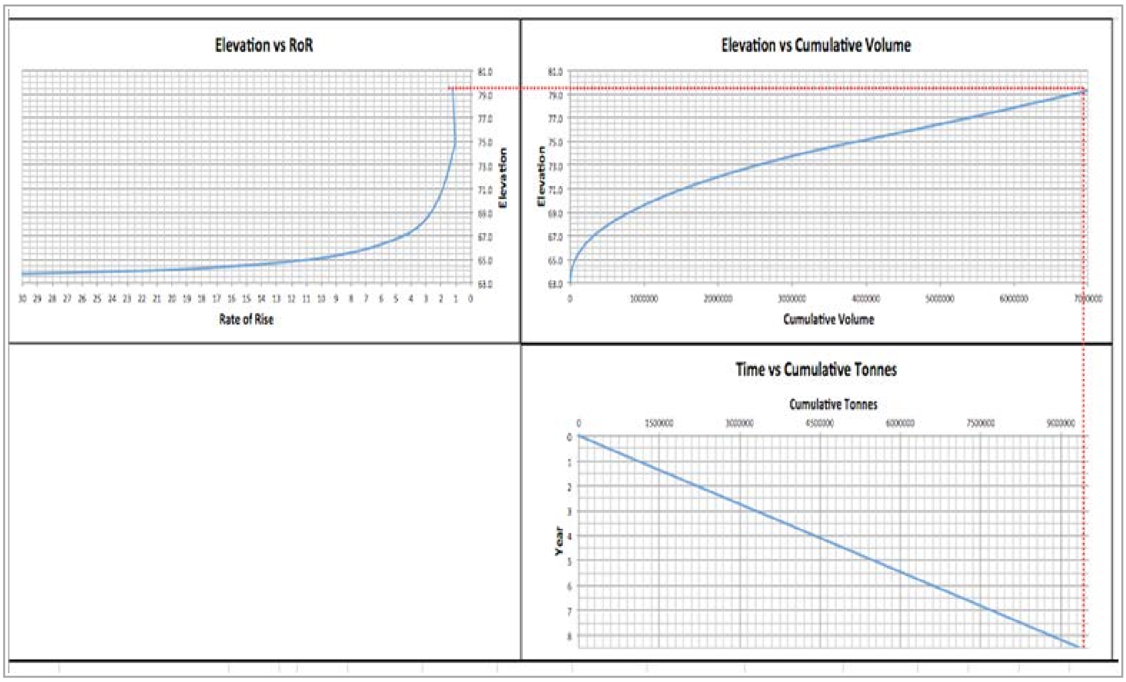



**Figure 18-4: Stage Capacity Curve for the Southern TSF Site Following the Optimization Study**

## 18.4.5 TSF Design

The TSF has been designed to accommodate a volumetric storage capacity of 9.4 million dry tonnes over an 8.5 year LOM. The general arrangement of the TSF is shown in Figure 18-3. The key design features of the TSF is it is a valley single embankment dam with a compacted earth embankment wall catering for the first 3.5 years of operations when the rate of rise is greater than 2.5 m/year. Subsequent development of, and raising of the TSF embankment is by means of the upstream self-raising paddock system. The TSF has a total footprint area of 120 ha, a maximum elevation of 82.0 m amsl (a maximum height of 20.0 m) and an average rate of rise of ±1.1 m/year above the elevation of the earth embankment.

### 18.4.6 Stage Capacity Curve

The stage capacity curve for the TSF, reflecting the relationship between tailings elevation, rate of rise, storage volume, footprint area, cumulative tonnage and time is shown in Figure 18-4. The initial 14.0 m high compacted earth starter embankment corresponds to a crest wall elevation of 76.0 m amsl, at which point the average rate of rise of the TSF decreases to below 1.5 m/year, in year 3.5 of operation. The rate of rise continues to decrease with time and TSF height, until a rate of rise of 0.95 m/year is reached at closure. The indicated TD footprint has the potential to accept additional tailings beyond year 8.5 of operation. The staged development of the TSF is shown in Figure 18-4.

### 18.4.7 TSF Dam Preparatory Works

The preparatory works associated with the TSF comprise the following:

- Topsoil stripping to a depth of 300 mm over the entire TSF footprint;
- A box cut to a depth of 500 mm beneath the starter wall embankment;
- A vertical curtain drain located within the compacted earth embankment wall;
- A compacted clay key below the starter wall embankment with the following dimensions:
    - 5.0 m deep;
    - 4.0 m wide base; and
    - 1V:1H side slopes.
- A compacted earth starter wall embankment, constructed with suitably sourced material from within the immediate vicinity of the TSF, with the following dimensions:
    - 14.0 m high (i.e. crest elevation of 76.0 m amsl);
    - 6.0 m wide crest;
    - 1V:2H internal side slope; and
    - 1V:3H external side slope.
- The self-raising portion of the TSF is battered at an overall side slope of 1V:3H;
- A 5.0 m wide elevated toe drain located on an elevated platform. The toe drain is positioned along the inside toe of the starter wall embankment and comprises the following:
    - 160 ND slotted HDPE Drainex pipe;
    - Suitably graded filter sand;
    - 6.7 mm stone;
    - 19 mm stone; and
    - Non-woven geo-fabric.
- A 3.0 m wide NGL toe drain extending the length of the inside toe of the starter wall embankment above an elevation of 65.0 m amsl. The toe drain is comprised of the following:
    - 160 ND slotted HDPE Drainex pipe;
    - Suitably graded filter sand;
    - 6.7 mm stone;
    - 19 mm stone; and

  - o Non-woven geo-fabric.
- 160 ND non-slotted HDPE Drainex pipes, spaced at 50 m intervals and positioned along the perimeter of the elevated and NGL toe drains. Seepage water emanating from the drains is collected and channelled into a decant manhole and pumped back to the plant, his water will not go to the Return Water Dam.
- An energy dissipater for the collection of supernatant water from the penstock outfall pipe. The water is pumped back to the plant as make up water;
- A 1.0 m high catchment paddock wall extending the perimeter of the TD;
- A storm water diversion channel with its associated cut-to-fill berm wall with the following dimensions:
  - o 1.0 m deep;
  - o 4.0 m wide base; and
  - o 1V:1.5H side slopes.
- A seepage interception drain downstream of the TD to an average depth of 5.0 m below NGL. Water from the drain will be collected in a manhole and pump back to the plant, this water will not go to the Return Water Dam.
- A buried 900 ND penstock pipeline comprising single intermediate intakes and a double final vertical 750 ND precast concrete penstock ring inlets;
- A lined Return Water Dam/Decant sump from where water is pumped back to the process plant for re-use or discharged downstream under surplus water conditions; and
- A slurry deposition pipeline along the length of the TSF starter wall embankment.

### 18.4.8 Tailings Dam Depositional and Operational Methodology

The proposed depositional methodology for the TSF is by means of the “self-raising upstream day-wall” system above the crest of the starter wall embankment. During the initial commissioning stage of the project, it remains crucial that the tailings not be deposited directly onto the elevated toe drain as this would lead to erosion and possible blinding of the toe drain system. Tailings shall be deposited into the basin of the TSF by means of an open-ended deposition technique whereby flexible hosing, positioned at approximately 30 m interval off-takes, is utilised. Prior to the tailings reaching the elevated toe drain, coarse tailings are to be used to cover the drains.

Open-ended deposition shall continue above the covered elevated toe drain, until the tailings elevation is within 1.0 m from the elevation of the starter wall embankment. Thereafter, the forming of 0.3 m high tailings paddocks, extending the length of the inner perimeter of the initial containment embankment, should commence so as to facilitate the shift in depositional methodology from an open-ended depositional strategy to a self-raising paddock system. Figure 18-5 illustrates the typical construction of a day-wall paddock system.

Supernatant and storm water collected on the TSF shall be decanted through vertical penstock inlets and a buried penstock pipeline to the energy dissipater/collection sump form where is shall be pumped back to the plant as make up water or discharged. During commissioning and initial development of the TSF, decanting occurs through single temporary/intermediate penstock inlets located along the migrating path of the pool from the starter wall embankment up towards the final location of the pool. It is at this final location where both the double penstock inlets and the pool wall/wing walls shall be situated. The intermediate inlets are progressively sealed as the pool is relocated to an adjacent upstream inlet.

### 18.4.9 Wetland Area

Aureus is currently considering the installation of a wetland system downstream of the TSF to reduce levels of arsenic and cyanide in any potential discharge from the TSF and the tailings sump circuit.

At the time of writing this report, the current conceptual wetland design is comprised of seven walls along its length, as well as four laterite walls. The laterite will function as a source of iron for removal of arsenic from solution. Walls should be comprised of inert waste rock with a reno mattress, and should be 1m high. The walls will separate compartments including: a hydroxide sump, open water to allow for iron reaction processes, a macrophyte bed, a second compartment of open water for aeration and lastly, the oxidation cascade compartment. With the proposed conceptual design, a retention time of three (minimum) to 80 (maximum) days has been estimated, depending on the time of year and amount of rainfall. Between June and October, retention times will be between 3-5 days.

Aureus is also currently working with other consultants to consider other designs for the wetland so the final wetland area may differ from the concept detailed above.

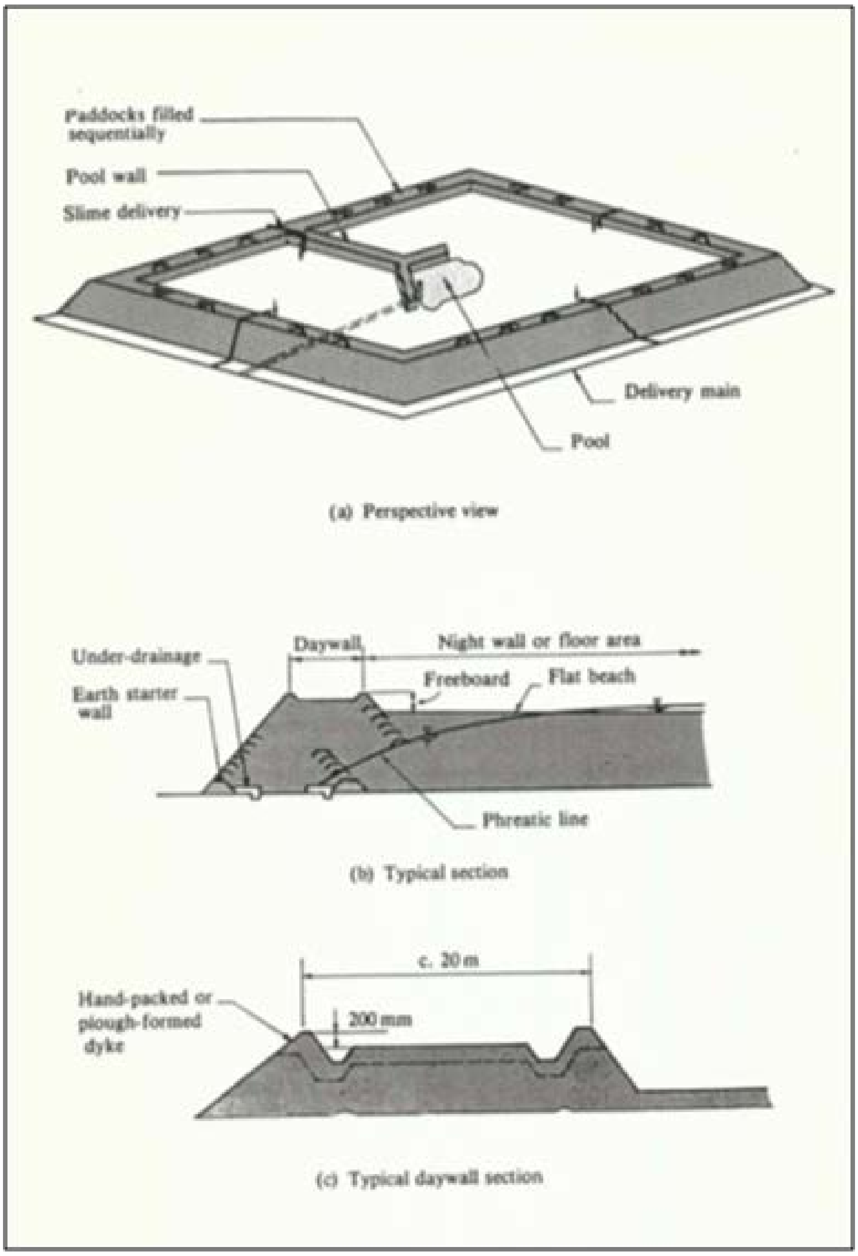



**Figure 18-5: Typical Construction of a Day-Wall Paddock System**

## 18.5 Marvoe Creek Diversion

### 18.5.1 Background

The Marvoe Creek is the dominant drainage feature in the Project area. It is fed by numerous small tributaries and is itself a tributary of the Mafa River, which lies 5 km south-west of the Project. The creek diagonally bisects the Project site and the alignment is such that it passes through the proposed open pit and waste dump sites. As a result, a permanent diversion channel has been constructed to route the Marvoe Creek around the open pit and the waste rock dumps. The drainage area of the creek upstream of the proposed diversion is approximately 109 km$^2$. Where it passes through the Project site, the Marvoe Creek is approximately 30 m wide with a mild slope.

### 18.5.2 Guidelines on Safety in Relation to Flood for Dams

For this study the SANCOLD Safety Evaluations of Dams, Report No. 4, Guidelines on Safety in Relation to Flood, published by the South African National Committee on Large Dams, December 1991 was adopted.

*Classification and Categorization of the Dam*

The dams have a total storage capacity of approximately 4.8 million cubic metres, of which 2.6 million cubic metres is flood storage and a maximum vertical height of 11 metres; therefore both are classified as small size dams.

The following information is available to determine the hazard rating of the dam:

- The design life of the dam will exceed the design life of the mine, which is 8 years.
- The dam will be designed with a design life of 100 years.
- The mine pit is situated downstream of the dams, but will be protected by means of a waste-rock dump placed between the pit and the dams.
- A dam failure after mine closure will be into the pit, which will collect sediment and attenuate the flood.
- Both the spillway and the cutting between the two dams can be lowered to drop the full supply level in the dams with mine closure. This will reduce the long-term risks significantly.

The adopted hazard rating of the dam is low; this means that the potential loss of life is zero. Although there is a significant potential in economic loss to mining production should the dams fail when the waste-rock dump is still at a low level, this risk will reduces as the waste rock dumps progress. The dams are therefore category I dams of small size and have a low hazard.

*Requirement in Respect of the Spillway Design*

The requirements in respect of the spillway are dependent on the dam size and its hazard rating. In this instance the Recommended Design Flood (RDF) is the 100 year flood event and the Safety Evaluation Flood (SEF) is the Regional Maximum Flood (RMF). The Recommended Design Flood (RDF) must pass safely through the spillway with available freeboard and the Safety Evaluation Flood (SEF) must pass safely through the spillway with zero freeboard available.

### 18.5.3 Deterministic Flood Evaluation – (Rational Method)

The Rational Method was used to calculate the flood peak – frequency relationships for the Marvoe Creek. The Arial Reduction Factors (ARF) was used as recommended by Prof. D.C Midgley in the HRU 1/72 report, to accommodate the phenomena of convection storms which have high intensities but cover relative small areas.

The recommended flood peak – frequency relationships are listed in Table 18-3 below.

**Table 18-3: Flood Peak and Volume Estimation Results for the Rational Method**

| Return Period (Years) | Marvoe Creek Flood Peak ($m^3/s$) | Marvoe Creek Flood Volume (Million $m^3$) |
|---|---|---|
| 2 | 91 | 4.95 |
| 5 | 135 | 7.34 |
| 10 | 176 | 9.59 |
| 20 | 220 | 12.01 |
| 50 | 284 | 15.51 |
| 100 | 342 | 18.67 |
| 200 | 409 | 22.29 |

### 18.5.4 Regional Maximum Flood Evaluation

The maximum observed floods since 1850 in West Africa were plotted with their catchment areas on the “x”- axis and the flood peak on the “y” - axis. The following countries were included in the evaluation; Liberia, Ivory Coast, Mali, Benin, Ghana, Guinee, Sierra Leone, Togo and Burkina Faso.

Only two observations were done in Liberia, but seven were done in Sierra Leone. The largest flood recorded in Liberia was in the Mano River at Mano Mine. The catchment area of the river at the observation point is 5,540 $km^2$, the maximum flood peak recorded was 1,610 $m^3/s$, with a corresponding “K” value of 3.6.

In neighbouring Sierra Leone the highest recorded flood had a “K” value of 4.68, which is also the highest in West Africa. Of the total of seventy-four floods evaluated, eight floods (11%) had a “K” value of equal or greater than 4.0, and two (3%) a “K” value greater than 4.5.

Since Liberia is one of the countries in West Africa to receive the highest annual rainfall, we adopted a “K” value of 4.5 for the estimation of the Regional Maximum Flood.

The calculated Regional Maximum Flood (RMF) for the Marvoe Creek is 525 $m^3/s$. If plotted on the same graph as the flood peak – frequency relationships from the Rational Method, the selected Regional Maximum Flood has an approximate return period of 650 years.

Figure 18-6 indicates the maximum observed floods in West Africa, the red line is the line associated with a “K” value of 4.5. The red dots represents floods observed in Sierra Leone and the green dots floods observed in Liberia.

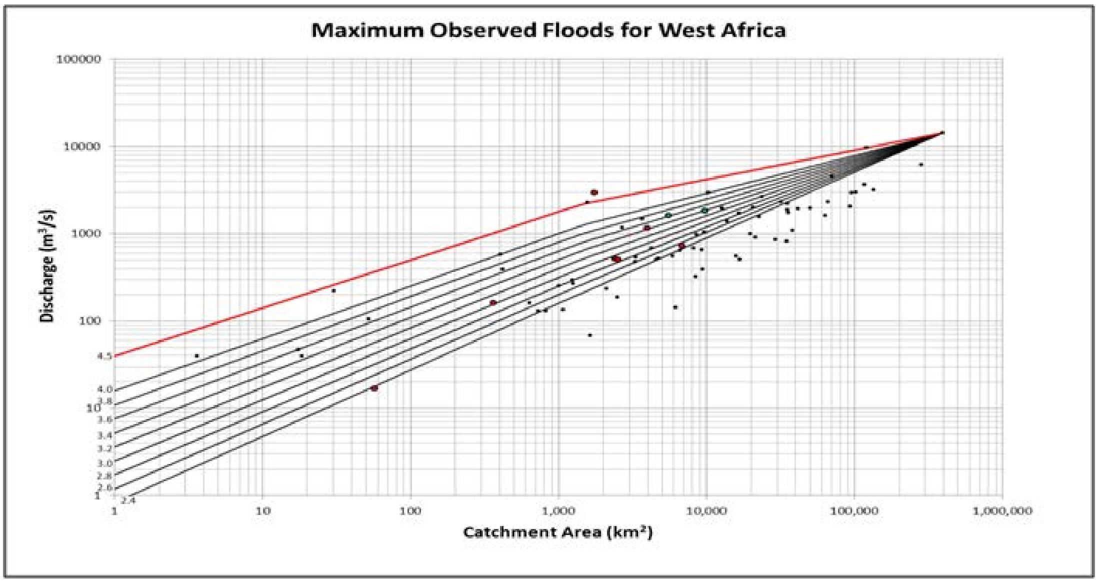



**Figure 18-6: Maximum Observed Flood in West Africa**

### 18.5.5 Reservoir Routing

The Marvoe Creek diversion system has two flood control dams / reservoirs, referred to as “Dam 1” and “Dam 2” on the drawings. “Dam 1”, is the first dam that was constructed in the Marvoe Creek and “Dam 2”, was then constructed in a tributary of the Marvoe Creek. A trench has been cut between the two dams to ensure “Dam 1” overflows into “Dam 2”.

For the purpose of routing floods through the dam, it has been assumed that the dams will be at full supply level by the time the flood wave arrives. The full supply level (FSL) of the dam is 68.50 m amsl. (based on the local benchmark on site). The invert level of the cutting between the two dams is 2 metres below FSL, at 68.00 m amsl. The expected water level difference between the two dams during the Regional Maximum Flood event (RMF) is 200 mm based on the energy principal. For the purpose of flood routing it has been assumed that the water level in both dams to be the same during any flood.

The dams have a total storage capacity of approximately 4.8 million cubic metres, of which 2.6 million cubic metres is flood storage and a maximum vertical height of 11 metres. The flood storage capacity relates to the temporal storage between full supply level (FSL) at 68.500 m amsl. and non-overspill crest level (NOC) at 73.00 m amsl.

Reservoir Routing was carried out using the level pool routing method. The attenuated flood peaks are tabulated below in Table 18-4.

**Table 18-4: Expected Flood Peak Reductions Due to Flood Attenuation**

| Flood Event | Peak Inflow ($m^3/s$) | Peak Outflow ($m^3/s$) | Percentage Reduction (%) | Energy Head Required (m) |
|---|---|---|---|---|
| 2 year flood | 91 | 72 | 21 | 0.85 |
| 100 year flood | 342 | 282 | 18 | 2.05 |
| RMF | 525 | 441 | 16 | 2.71 |

### 18.5.6 Hydraulics for Diversion Channel

*Design Philosophy*

The design philosophy of the diversion channel was simply to mimic the existing natural environment the Marvoe Creek functioned within before the existence of the mine. It was intended to create a system that has an inner main channel with a floodplain wherein an ecosystem can develop over time.

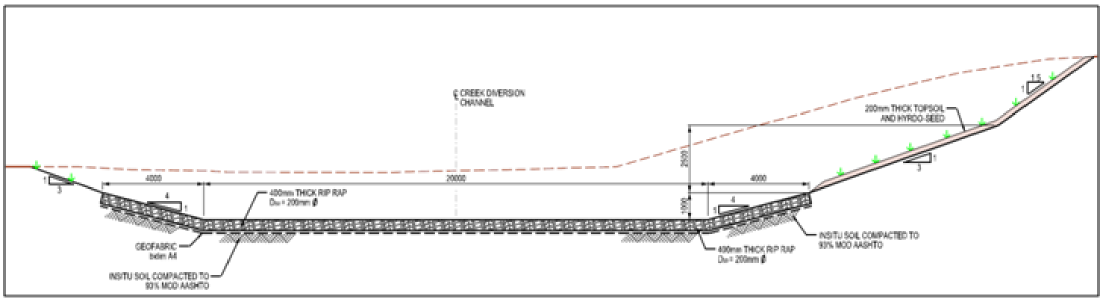
The design has two main objectives namely; to safely and cost effectively divert the Marvoe Creek around the proposed pit and secondly to do it in such a manner that its impact on the local environment is limited.

In terms of cost saving the approach was to design a system with the smallest possible footprint, least excavation requirements and require minimal scour protection. This was to be done within the confines of good hydraulic design principles, which includes large radius curves to avoid standing waves and maintaining sub-critical flow conditions with Froude numbers close to critical conditions. The reason for maintaining sub-critical flow conditions with Froude numbers close to critical conditions is to find a hydraulic efficient section that will produce a stable water surface profile and that can accommodate directional changes without the formation of standing waves or hydraulic jumps.

The bush clearing was also be limited into the flood plain, this not only saved costs but has been beneficial in creating a stable floodplain environment that can resist scour through providing flow resistance in the form of established vegetation.

*Description of the Typical Cross-section*

The typical cross-section has an inner main channel which has a 400 mm thick rip-rap lining underlay by a geotextile to act as a filter medium. The main channel has a bottom width of 20 metres, a depth of 1 metre and side slopes of 1:4 (V:H). The main channel was designed to be on average 1 metre below existing ground level and follow the average longitudinal ground slopes in general. Where the main channel passes though cut areas, excavations are extended at a slope of 1:3 to 3.5 metres above main channel invert level and thereafter a side slopes of 1:1,5. The reason for the 1:3 side slopes is that on the average flow depth of the 100 year flood event is 3.5 metres.

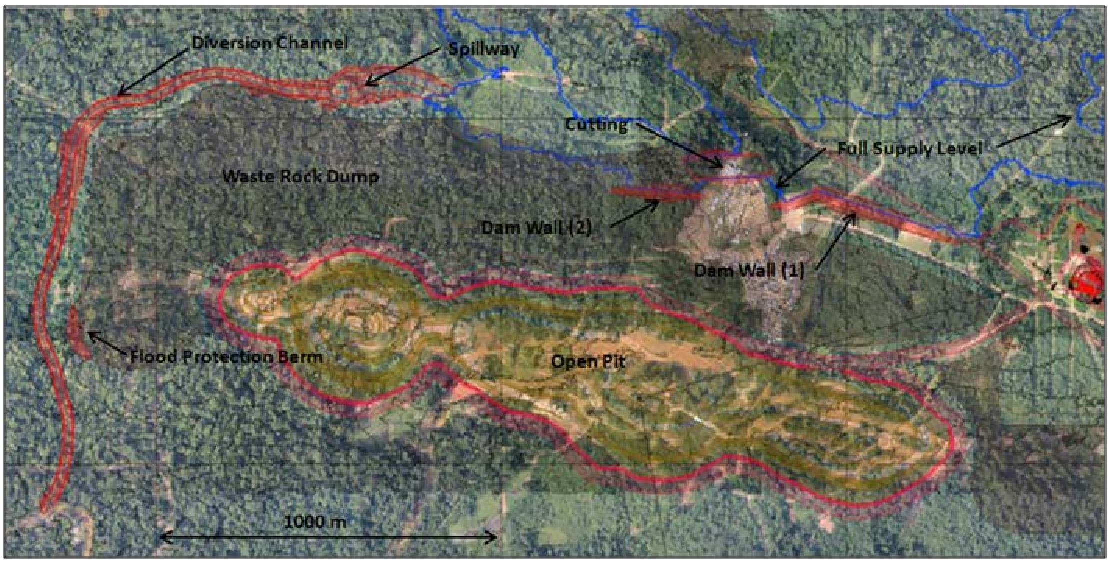



Source: *Epoch, 2013*

**Figure 18-7: Typical Diversion Channel Cross-section**

*Typical Layout and Three-Dimensional Views of the Proposed System*

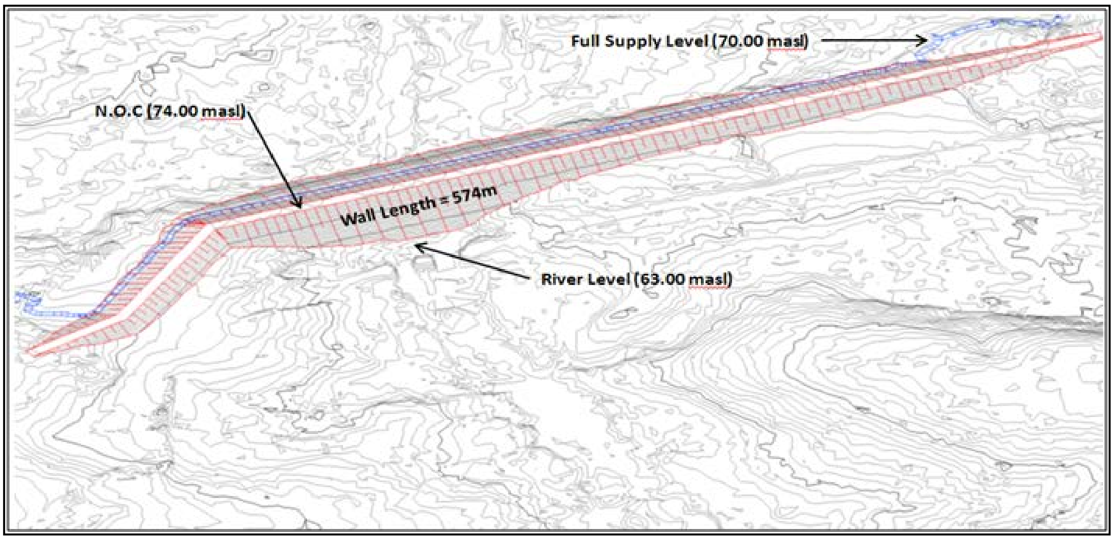
The general arrangement of the Marvoe Creek diversion system is shown in the figure below, and it consists of two flood control dams with a cutting connecting the two dams, a By-wash spillway, diversion channel and flood control berms. The figure in this section is indicative with the purpose to provide the reader with an understanding of the system.




Source: *Epoch, 2013*

**Figure 18-8: General Arrangement of Marvoe Creek Diversion System**

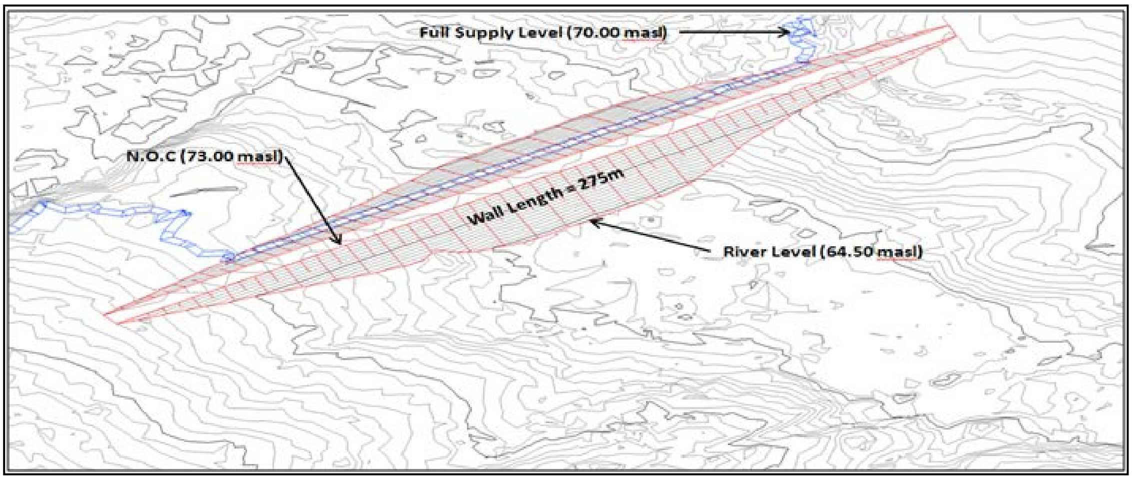
Figure 18-9 below, shows the layout of Dam Wall "1", The design and construction drawings development for both dam walls fell under Epoch Recourses' scope of works. The minimum allowable NOC level based on Safety Evaluation Discharge (SED) build up in the dam is 73.21 m amsl.




Source: *Epoch, 2013*

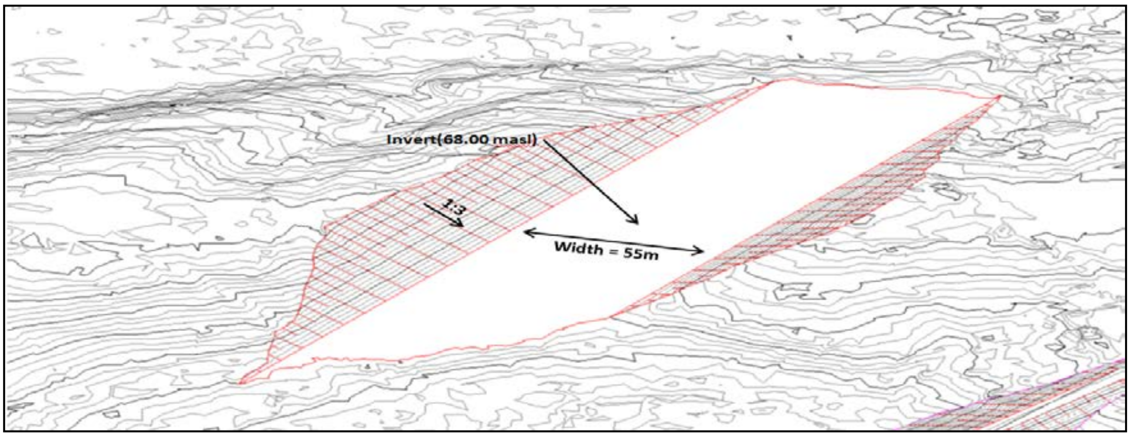
**Figure 18-9: 3D – View of Dam 1**

Figure 18-10 below, provides an overview of the configuration of the Dam Wall "2. The non-overspill crest level (NOC) isthe minimum allowed based on Safety Evaluation Discharge (SED) build up in the dam, which is 72.71 m amsl.



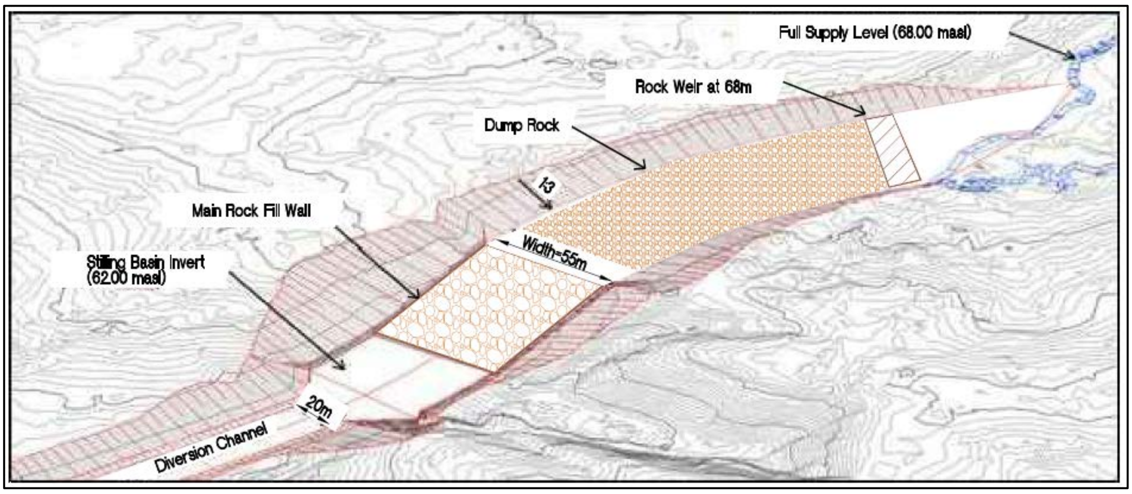

Source: *Epoch, 2013*

**Figure 18-10: 3D – View of Dam 2**



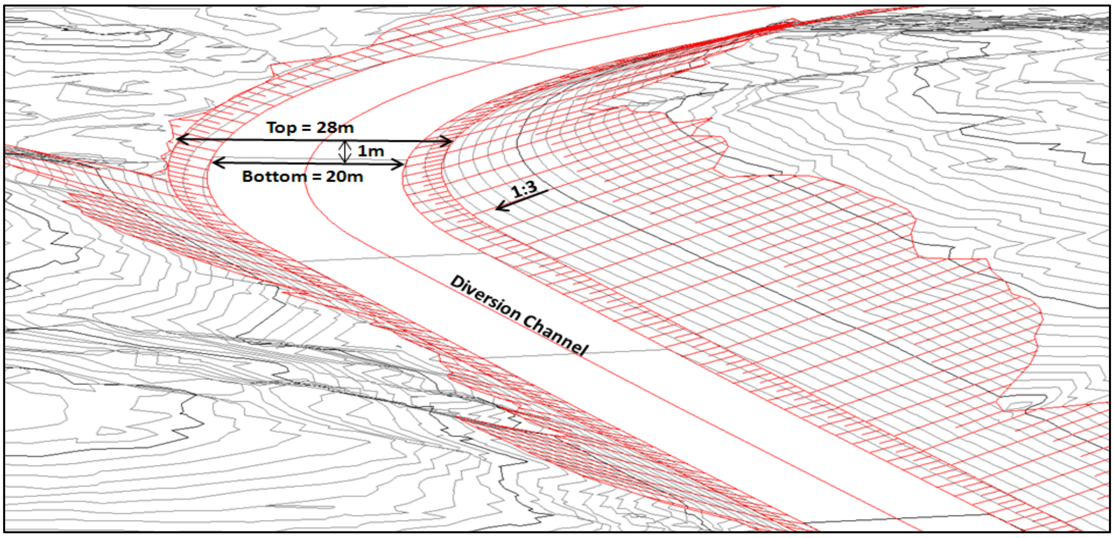

Source: *Epoch, 2013*

**Figure 18-11: 3D – View of Cutting between Dam 1 And Dam 2**



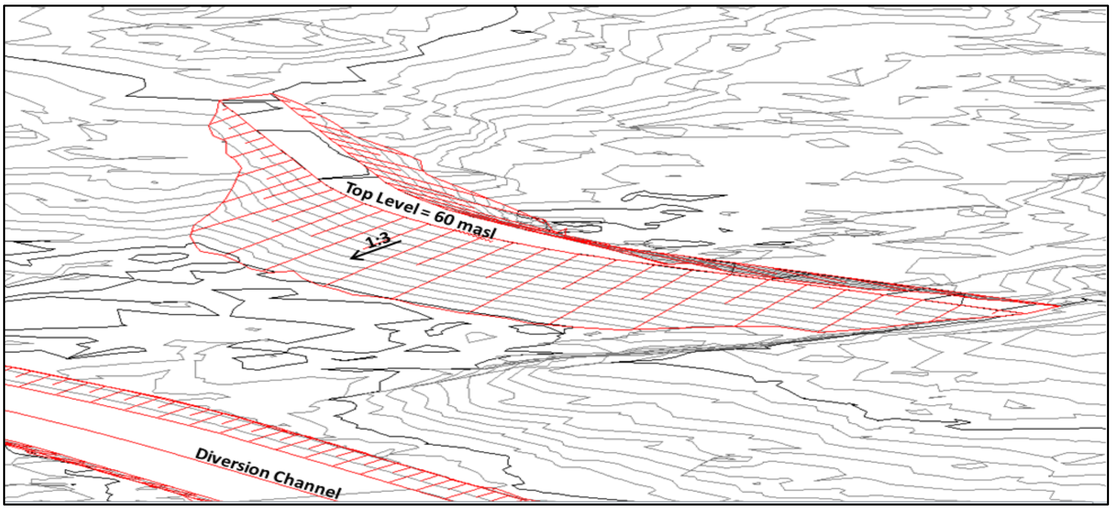

Source: *Epoch, 2015*

**Figure 18-12: Spillway with Energy Dissipation System**




Source: *Epoch, 2013*

**Figure 18-13: Typical 3D – View of Diversion Channel**




Source: *Epoch, 2013*

**Figure 18-14: Typical 3D – View of Flood Protection Berm**

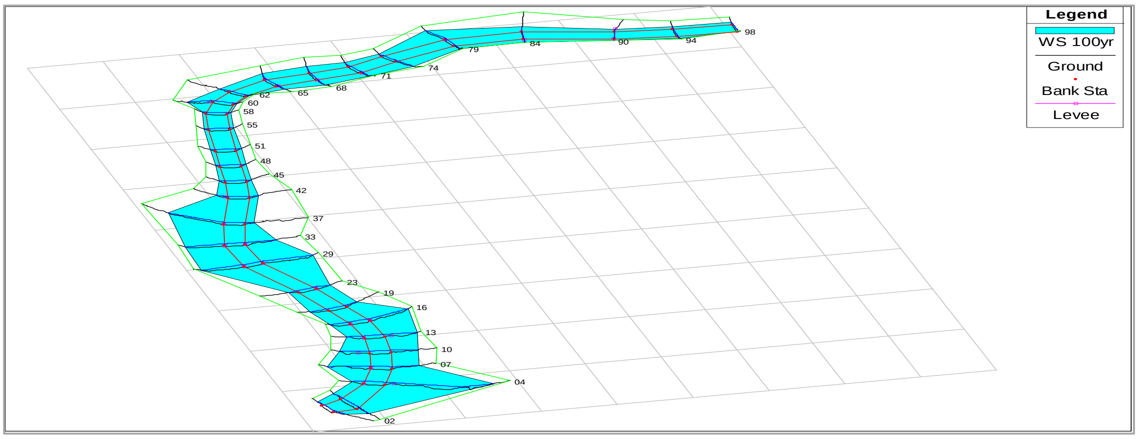
Except for Dam "1" and Dam "2", the rest of the 3D model as shown in the figures in this section was utilized for the final development of the construction drawings. On the deep cuttings for the spillway and diversion channel the side-slopes were made 1:1.5 where possible to save costs.

*HEC-RAS Results*

The results discussed in this section are only the most important outcomes that will confirm the specifications as shown on the construction drawings.

A three-dimensional CAD drawing was produced from which cross-section were extracted to build the hydraulic model in HEC-RAS. The results from HEC-RAS confirmed that flows less than the 2 year flood peak will remain within the main channel and larger floods will flow into the floodplains. The diversion channel's upstream section has a steeper longitudinal slope than the downstream section. The upstream section's slope is 1:130 and the downstream section's slope is 1:350.

The average velocity in the upstream section of the main channel is between 3.0 m/s and 4.2 m/s during a 100 year flood event and in the downstream section between 2.0 m/s and 3.0 m/s. The shear forces in the upstream section of the main channel are between 120 N/m$^2$ and 200 N/m$^2$, for the downstream section between 30 N/m$^2$ and 120 N/m$^2$.




Source: *Epoch, 2013*

**Figure 18-15: Three-Dimensional View of the Diversion Showing the 100 Years Flood Event**

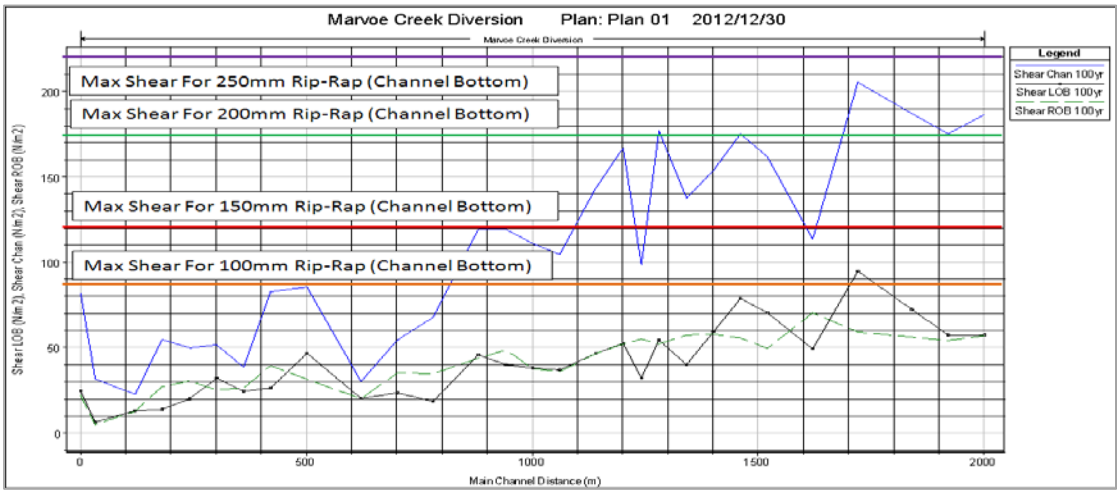



**Figure 18-16: Minimum Scour Protection Stone Sizes Along the Diversion for the 100 Year Flood**

### 18.5.7 Costing

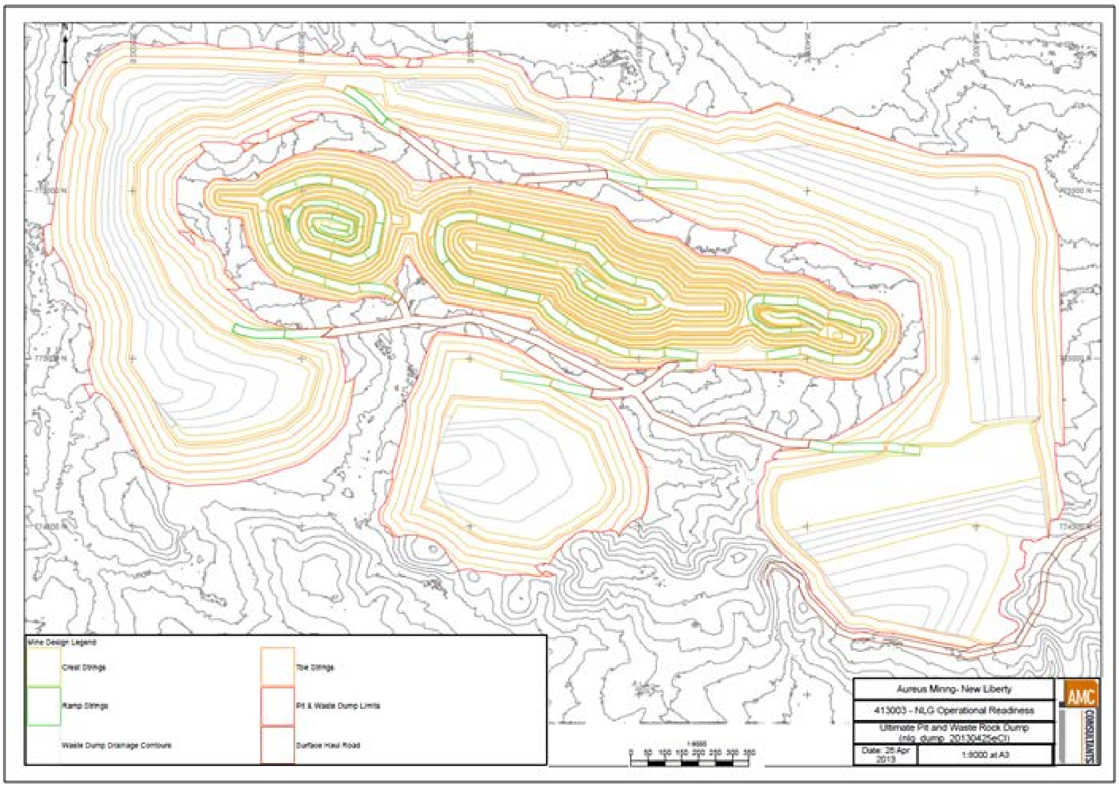
Detailed quantities were taken from the three dimensional CAD model and construction drawings and forwarded to DRA for pricing. The design discussed in the report resulted in major savings from the budget allowed for by Golder during the pre-feasibility design.

## 18.6 Waste Rock Dumps

Figure 18-17 and Figure 18-18 show the final waste dump design for the Project. The volume of material from the pit in bank cubic metres (BCM) is 48.3 million BCM. (mBCM). Considering an expansion factor of 35% the total waste dump volume required is 65.3 million m$^3$.

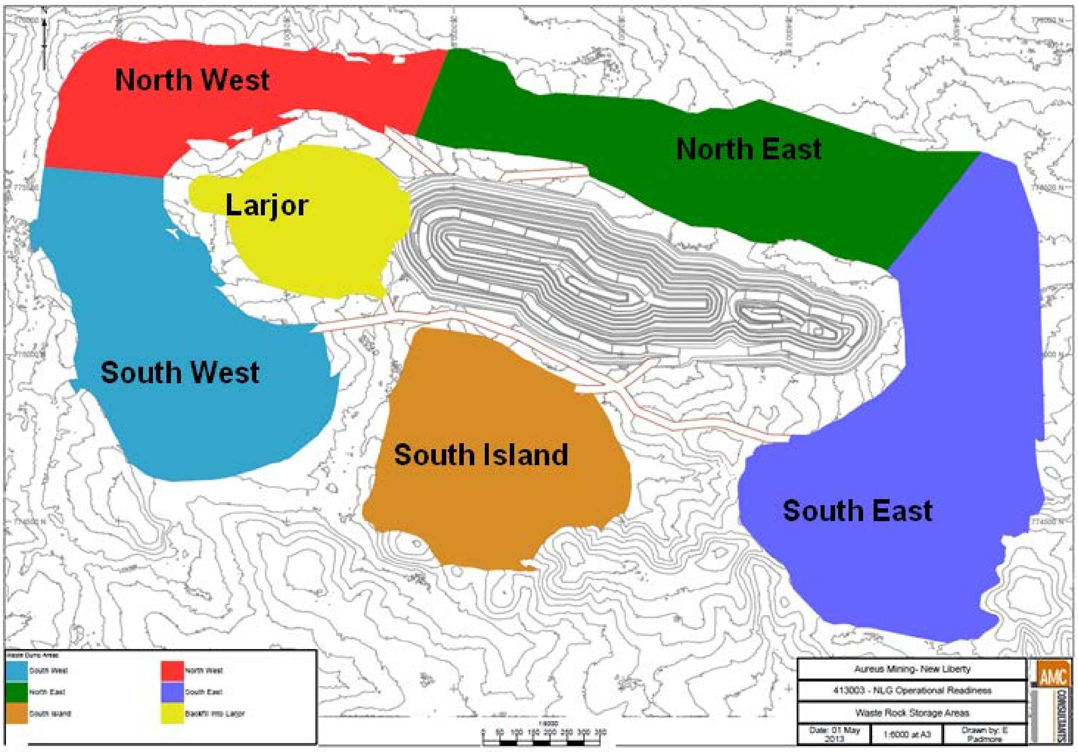
The following changes were incorporated into the design:

- The design now has two separate dumps. The gap between the dumps to the south of Larjor follows the existing course of the Marvoe Creek. This gap is for sedimentation ponds.
- The upper surface of the dump is profiled to shed water away from the pit catchment.
- The southern edge of the South East dump is positioned adjacent to the access road and at the drainage divide between the open-pit catchment to the north and the tailings facility catchment to the south.
- The northern toe of the dump has been positioned to abut the dams walls constructed to form the water storage dam and the Marvoe Creek Diversion Channel.
- An allowance has been made for a culvert under the western dump to facilitate drainage of the western open-pit catchment under the waste dump to the Marvoe Creek Diversion Channel.




Source: *AMC, 2013*

**Figure 18-17: Final Waste Dump Design**




Source: *AMC, 2013*

**Figure 18-18: Waste Dump Areas**

In addition to the construction of the waste dump, the project will use in-pit waste dumping. Once the Larjor pit is completed waste material will be placed in to it, rather than hauling to the waste dump south of the pit.

Table 18-5 shows the specifications of the waste dump.

**Table 18-5: Waste Dump Design Specifications**

| Property | |
|---|---|
| Overall capacity (million $m^3$ ) | 52.6 |
| Overall Slope angles (degrees) | 18 |
| Lift height (m) | 15 |
| Berm width (m) | 25 |
| Batter angle(degrees) | 33 |
| Footprint area (Ha) | 244 |
| Elevation of top lift (m RL) | 110 |
| Maximum height (m) | 45 |
| Minimum height (m) | 15 |

The waste dump surface has been designed with a gradient of approximately 1:40, draining away from the open pit catchment. The top surface of the dump is at 110 m Reduced Level (RL or elevation) grading to 100 m RL.

The dump has been designed to reduce infiltration and to minimize surface runoff towards the pit. Interim dump stages will be capped by saprolite to reduce infiltration and the dump will be graded to facilitate water flow away from the pit.

A waste schedule was constructed giving consideration to maintaining the shortest haul distances, where possible. The schedule takes into account the material required in Phase 1 of filling up the South East waste dump area. Table 18-6 shows the origin and destination of waste volume by year. The detailed schedule can be seen in Figure 18-18.

**Table 18-6: Waste and Backfill Schedule**

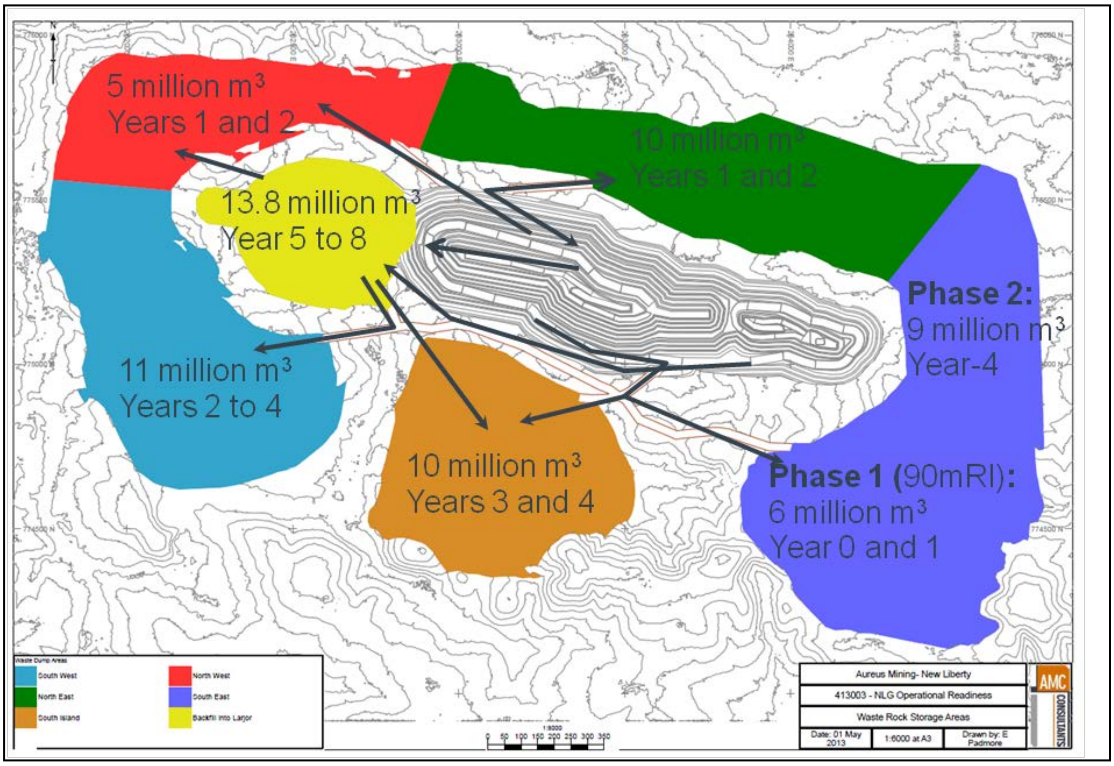
| Source | Destination | Units | Totals | Year 0 Q4 | Year 1 Q1 | Year 1 Q2 | Year 1 Q3 | Year 1 Q4 | Year 2 H1 | Year 2 H2 | Year 3 Y | Year 4 Y | Year 5 Y | Year 6 Y | Year 7 Y | Year 8 Y |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Stage 1** | **Waste Dump** | **Mt** | **1.11** | **-** | **0.85** | **0.26** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** |
| **Stage 2** | **Waste Dump** | **Mt** | **29.17** | **-** | **-** | **0.57** | **1.88** | **1.97** | **7.02** | **4.14** | **12.60** | **0.99** | **-** | **-** | **-** | **-** |
| **Stage 3** | **Waste Dump** | **Mt** | **36.37** | **0.97** | **3.41** | **3.75** | **2.78** | **3.01** | **3.49** | **2.91** | **5.41** | **10.64** | **-** | **-** | **-** | **-** |
| **Stage 4** | **Waste Dump** | **Mt** | **54.18** | **-** | **-** | **-** | **-** | **-** | **1.68** | **5.55** | **6.10** | **12.68** | **21.05** | **5.16** | **1.95** | **-** |
| **Stage 5** | **Waste Dump** | **Mt** | **10.78** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **1.08** | **-** | **-** | **4.26** | **4.14** | **1.30** |
| **Sub Total** | | **Mt** | **131.61** | **0.97** | **4.26** | **4.58** | **4.66** | **4.98** | **12.19** | **12.60** | **25.19** | **24.32** | **21.05** | **9.42** | **6.10** | **1.30** |
| **Stage 1-4** | **Waste Dump** | **Mt** | **115.67** | **0.97** | **4.26** | **4.58** | **4.66** | **4.98** | **12.19** | **12.60** | **25.19** | **24.32** | **21.05** | **0.89** | | |
| **Stage 4&5** | **Larjor** | **Mt** | **15.93** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **8.53** | **6.10** | **1.30** |

Volumes assume a 35% swell factor from in situ to broken volume.

The capacity of different areas of the waste dump and the volume scheduled to each area is shown in Table 18-7.

**Table 18-7: Capacity of Waste Dump Areas**

| Waste Dump Area Destination | Volume into Dump (million $m^3$) | Capacity of Dump Area (million $m^3$) |
|---|---|---|
| South East – Phase 1 | 6.0 | 6.0 |
| South East – Phase 2 | 9.0 | 9.8 |
| North West | 5.2 | 5.0 |
| North East | 10.2 | 10.8 |
| South West | 10.7 | 10.6 |
| South Island | 10.3 | 10.5 |
| Larjor Pit backfill | 13.8 | 14.0 |
| Total | **65.3** | **66.6** |




Source: *AMC, 2013*

**Figure 18-19: Waste Dump Areas and Volumes**

The total area of top soil to be removed around the pit and waste dump area is 321 Ha. This is the area covered by the pit, dump and surface haul roads. Table 18-8 shows the respective surface areas. Assuming an average thickness of top soil of 0.3 m, a total of 1.22 million $m^3$ will require stockpiling as topsoil for later use in rehabilitation. This figure does not take into account topsoil from any other infrastructure such as the mine camp, process plant, MCDC or the tailings storage area.

This topsoil volume will form a significant dump if stored in one location. It is the intention to store topsoil on the south island waste dump location, until it is required for progressive rehabilitation of the main waste dumps. Topsoil will then be moved from here, onto the main rehabilitated waste dumps, clearing this area for the final waste dump.

**Table 18-8: Areas of Topsoil Stripping**

| Topsoil | Surface Area (Ha) |
|---|---:|
| Main dump | 204.5 |
| South Island Dump | 39.7 |
| Haul Road north of Pit | 1.6 |
| Haul Road south of Pit | 4.6 |
| Open Pit | 70.8 |
| **Total** | **321** |

## 18.7 Waste Controls

Industrial and domestic waste will be recycled where practical. Any combustible waste will be burnt and the ashes buried with any non-combustible waste in the mine waste dump.

Treatment facilities for sewerage disposal form part of the site accommodation requirements.

## 18.8 Closure Plan

### 18.8.1 Closure Objectives

The primary closure plan objectives are:

- To ensure long-term physical and chemical stability of the Project components (e.g. the residue management area and waste rock storage facilities) remaining on-site at closure;
- To minimize long-term care and maintenance requirements; and
- To minimize the health and safety hazards posed by the site with regard to local residents and their livestock.

### 18.8.2 Process Plant

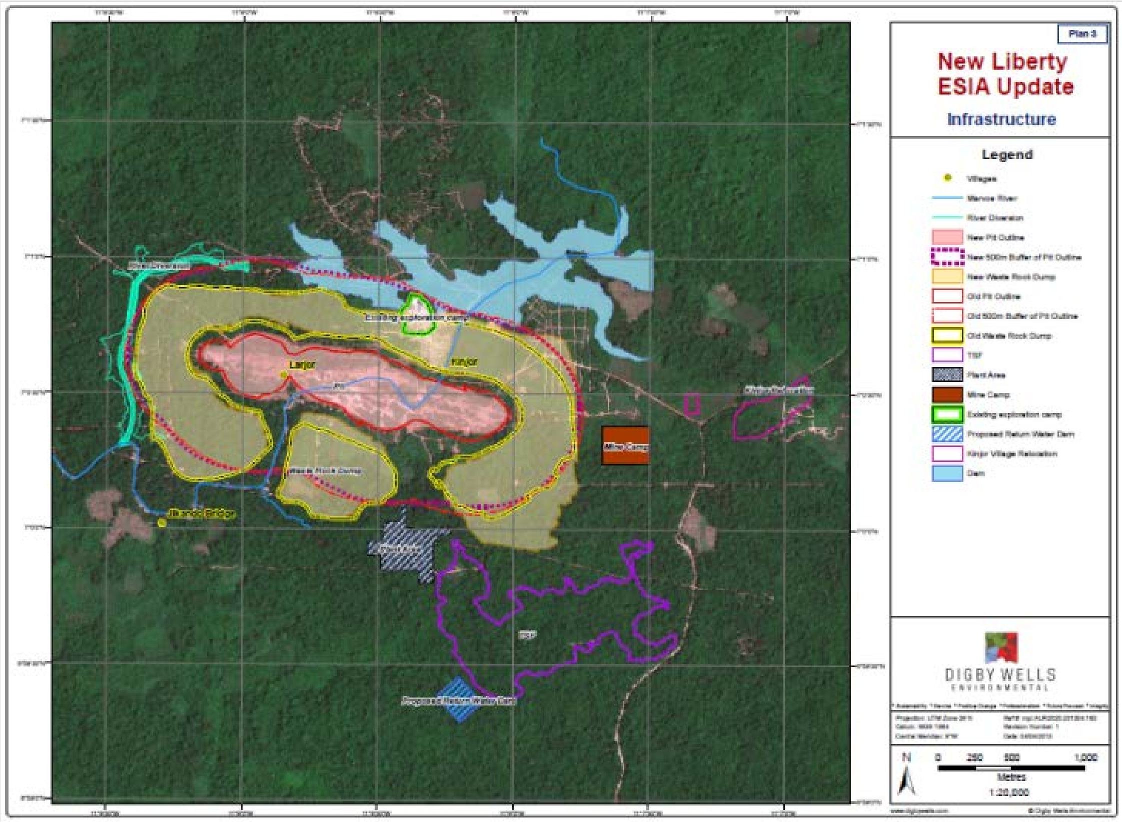
The closure objectives for the Process Plant site are as follows:

- Decommission unnecessary materials, equipment and infrastructures;
- Remove and remediate soil and restore original land use to the extent feasible;
- Handover some infrastructures for local use, if appropriate; and
- Encourage re-vegetation.

The closure measures, which will be undertaken in accordance with international accepted practices and the Bea Mountain MDA, can be summarized as follows:

- Conduct environmental site assessment/risk analysis to determine clean-up requirements;
- Drain and store all valuable reagents for resale;
- Flush and clean all equipment to ensure that no residual toxic materials are left;
- Dismantle equipment and sell for re-use if possible or sell as scrap. Non-saleable material will be disposed at an appropriated disposal facility;
- Dismantle unnecessary buildings and resell. Non saleable material will be disposed of at an appropriated disposal facility;

- Remove impacted/contaminated soils and dispose at the TSF (beneath the saprolite/ cover. Treat or cover the impacted area to enhance satisfactory mitigation;
- Where feasible, promote re-vegetation of the footprint area of the demolished facilities;
- Scarify non-essential internal roads and other surface areas to encourage infiltration and natural re-vegetation;
- Culverts will be removed where necessary and the disturbed areas re-graded to allow for unobstructed drainage; and
- Where appropriate, construct diversion ditches/channels to route run-off to the open pit.




Source: *Digby Wells, 2013*

**Figure 18-20: Project Closure Plan**

## 18.8.3 Open Pit

The closure objectives for the open pit are:

- Ensure physical stability;
- Ensure chemical stability; and
- Limit human and animal access.

The closure measures for the open pit are:

- Decommission pipelines, pumps, and electrical lines;
- Stop open pit dewatering;
- Block access ramps;

- Evaluate the stability of the pit slopes and where required, re-shape or stabilize pit slopes;
- Construct a 3 m high safety berm around open pit with warning signs;
- Construct spill structures and ditch for the floodwater to spill to the MCDC; and
- Continuously assess the water quality in the open pit and if required, provide water treatment system (e.g. pit lake treatment, constructed wetland, water treatment plant, etc.).

### 18.8.4 Tailings Storage Facility

The closure objectives for the TSF are:

- Creation of a stable landform;
- Ensure chemical stability;
- Minimize erosion and dust generation; and
- Minimize rainfall infiltration to extent practically possible.

The closure measures for the TSF are:

- Construct a permanent spillway to ensure physical stability of the facility during storm events;
- Fill the tailings pond area to eliminate water ponding (no water should pond on the surface of the TSF post closure);
- Place the tailings cover (which includes from top to bottom- vegetation, 0.5 m thick topsoil, and 1 m (minimum) thick saprolite layer);
- Reshape the downstream berms and encourage re-vegetation; and
- Maintain access road for continuous monitoring.

Note: Current designs for the tailings storage facility is a single cross-valley impoundment design.

### 18.8.5 Marvoe Creek Diversion Channel

The closure objectives for the MCDC are:

- The MCDC to remain at closure; and
- The MCDC to become stable and self-sustaining.

The closure objectives for the MCDC are:

- Inspect the MCDC and natural channel and make required repairs and upgrades to improve hydraulics and long-term morphologic stability;
- Post-closure, open pit overflow will report to the diversion; and
- Additional armouring may be required to handle the additional flows.

### 18.8.6 Waste Rock Dump

The closure objectives for the waste dump are:

- Creation of a stable landform
- Confirm chemical stability
- Minimize erosion and dust generation
- Minimize rainfall infiltration to the extent practically possible; and
- Apply stripped topsoil and vegetate or allow natural colonisation of vegetation. Sufficient topsoil will be stripped and stockpiled during mine construction for use in rehabilitation of the waste dump as detailed in Section 18.7.

The closure measures are:

- Reshape the crest (2%) of the waste dump and allow drainage channels on the side slope for the run-off to flow to the open pit;
- Where necessary, reshape the side slopes of the waste dump to 3H:1V for ease of cover placement and provide 10 m wide benches on the side slope for water management; and
- Place the waste rock dump cover (which includes vegetation and 0.5 m topsoil).

# 19 MARKET STUDIES AND CONTRACTS

## 19.1 Markets

Liberia allows for the direct export of gold doré to refiners. The Government of Liberia has the right, but not the obligation, to purchase a portion or all of the production at fair market value. As such, it has been assumed that all gold shall be sold, after refining, on the open market. For the economic evaluation (Chapter 22) Aureus has used a flat gold price of USD1,300/oz.

There is currently no gold refining capability in Liberia.  As such, the New Liberty operation will produce gold doré which will be air freighted from site to refineries in Europe.

## 19.2 Contracts

105 contracts/orders have been placed with the overall procurement progress being fully complete. Only minor amendments are still continuing on the current contracts/orders. A few highlights are:

- In June 2013 an order was placed on NCP International Ltd (NCP) for the supply, delivery and installation of the ball mill; this is a long lead key plant item.
- In November 2013 DRA Projects (Pty) Ltd (DRA) was appointed as the engineering, procurement and construction management (EPCM) contractor. This follows on the successful relationship Aureus has built over three years with DRA whom assisted Aureus with the Preliminary Economic Assessment, the Feasibility Study, the Definitive Feasibility Study and the Front End Engineering Design. DRA has a long history of working on gold projects across Africa.

- In November 2013 Group Five Projects (Pty) Ltd (G5) was appointed as the structural, mechanical, platework and piping supply and installation contractor. G5 is an old and well established construction company whom operates in over 20 countries and has vast experience in Africa with many a project successfully completed in West Africa.
- In April 2014 Aureus appointed MonuRent (Liberia) Ltd (MonuRent) to supply and maintain a new mining fleet in support of the owner mining operations over the life of mine (LOM) at New Liberty. MonuRent has provided a fleet, under a separate contract, over the past two years for the clearing, civils and earthworks at New Liberty during the construction phase. They have a well-established West African business and network infrastructure which will allow them to maintain the fleet and to guarantee a minimum fleet availability during the operational phase.
- In April 2014 Group Five Projects (Pty) Ltd (G5) was appointed as the electrical and instrumentation supply and installation contractor.
- In February 2015 Aminata and Sons, the largest Liberian importer of petroleum, was awarded a fuel contract for the supply of petroleum product for the duration of the mining operations at New Liberty.
- In February 2015 Aureus appointed Manex Limited for the importation, manufacture and supply of industrial explosives and accessories for use at the New Liberty Project.

# 20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

## 20.1 Introduction

The following section provides an overview of the legislative process surrounding the Project, the permitting requirements of the Project, and the environmental impacts and implications of the Project together with Aureus’ commitments to the environment and local community.

An Environmental and Social Impact Assessment (ESIA) was undertaken from Q4-2010 to Q2-2012 by Golder Associates (Ghana) Ltd (Golder) to investigate the local environmental and social situation existing prior to the development of the Project, and to determine the likely positive and negative impacts of the Project. The timing, extent, intensity and cumulative effects of these impacts were investigated. The ESIA also identified the necessary management measures required to mitigate the identified impacts. These form the basis of the Environmental Management Plan (EMP), Resettlement Action Plan (RAP) and accompanying Community Development Plan (CDP). The ESIA was approved by the Liberian Environmental Protection Agency in October 2012 and the RAP and CDP during April 2013.

Subsequent to the completion of the original ESIA, a mine optimization study was conducted during Q1 2013, with the focus on optimising the layout of the plant, tailings dam, mine accommodation and other infrastructure, and also address risks from design, operational and environmental perspectives. The newly-identified positions for infrastructure all fall within the area permitted for mining.

However based on the optimisation work and independent review, Digby Wells Environmental ("Digby Wells") was appointed to undertake further detailed specialist studies in these areas of changed infrastructure positions to establish baseline conditions, and update the ESIA report for submission to the Liberian Environmental Protection Agency (EPA). The updated ESIA was subsequently submitted to the Liberian EPA during October 2013 as per the MDA requirements and all permits remained valid.

Following a review of these documents by the International Finance Corporation (IFC), prior to their investment in the Company in July 2014, an addendum to the updated ESIA was also produced and submitted to the EPA during March 2014. This document detailed updated Environmental Management Plans to ensure ongoing compliance with the Equator Principles, observance to IFC Performance Standards and the World Bank Group Environmental, Health and Safety guidelines.

## 20.2 Liberian Legislation and Guidelines

In November 2002, the Liberian Government adopted the National Environmental Policy, the Environment Protection Agency Act and the Environment Protection and Management Law (EPML). The three documents became law in April 2003.

Under Part III of the Act creating the EPML of the Republic of Liberia (2002), an ESIA Licence or Permit is required from the Environmental Protection Agency (EPA) prior to commencement of activities specified under Annex 1 of that Law. Consideration of the Project's Project Brief (submitted to the EPA in November 2010) by the EPA identified the need for a full ESIA.

**Terms of the Mineral Development Agreement**

The Mineral Development Agreement (MDA) between Aureus Mining and the Liberian Government is of relevance to the Scoping Report and reads "*16.1 Environmental Impact Statement [EIS]: The Parties recognise that Operations may result in some pollution, contamination or other environmental damage to land, water and the atmosphere within the Contract Area and elsewhere. The EIS illustrates the adverse effect operations will have on the environment and review plans to mitigate such effects."*

The MDA and its commitments were taken into account during the development of the ESIA.

**International Standards and Best Practices**

The ESIA refers to relevant international standards, guidelines and best practice documents notably those of The World Bank Group (WBG), including the International Finance Corporation (IFC) Performance Standards, the Equator Principles (EP's) and the World Health Organisation (WHO) Guidelines.

**International Finance Corporation Standards for the ESIA Process Followed**

The IFC Performance Standards on Social and Environmental Sustainability address social and environmental issues and potential impacts associated with project development. The Performance Standards require that social and environmental impacts and risks of a project are identified and assessed in the early stages of project development and continue to be managed throughout the life of the project.

The Environmental, Health and Safety Guidelines provide general and industry-specific best practice guidance and numerical limits for occupational and community health and safety, noise, gaseous emissions, effluent discharges and other waste products.

There are eight IFC Performance Standards. Of particular importance is Performance Standard 1 of the IFC on Social and Environmental Sustainability. This standard requires the Project proponent to assess the social and environmental impacts of the proposed Project and to ensure the continued management of social and environmental performance throughout the lifecycle of the Project. The ESIA has been completed and subsequently approved. Procedures for long-term monitoring and reporting on the effectiveness of the risk management measures are also required in terms of meeting Liberian and international standards. Performance Standard 5 on Land Acquisition and Involuntary Resettlement is also relevant in terms of the Project's RAP for the villages of Kinjor and Larjor, which have now been undertaken.

**Equator Principles**

The 'Equator Principles' are a set of standards for determining, assessing and managing environmental and social risks in project finance transactions. It comprises a set of principles adopted by the Equator Principles Financial Institutions (EPFI's) to ensure that the projects they finance are developed and implemented in a manner that is socially responsible and environmentally sound. These principles apply to all project-financing by EPFI with total project capital costs of USD10 million or more, and across all industry sectors.

**Minerals and Mining Act**

The Minerals and Mining Act (2000) states that minerals on the surface of the ground or in the soil or subsoil, rivers, streams, watercourses, territorial waters and continental shelf are the property of the Republic of Liberia. This act allows for the establishment of a Minerals Technical Committee consisting of: Ministry of Lands, Mines and Energy, Ministry of Justice, Ministry of Finance, Ministry of Planning and Economic Affairs, National Investment Commission, Ministry of Labour, Council of Economic Advisors, and Central Bank of Liberia. This committee has the power to negotiate agreements for Class A Mining Licences.

**Project Permitting Process**

The primary permit/licence required for the development of the NLGM Project is an Environmental Permit issued by the Minister of Environment. This was granted for the Project in October 2012 and is valid for three years subject to an annual renewal by the Liberian EPA. Various other operational permits from other governmental departments will be required (e.g. construction of buildings etc.) but are relatively procedural in nature.

In order to gain the Environmental Permit, the following permitting process steps were undertaken:

- Application for an Environmental Impact Assessment Licence by BMMC;
- Notice of Intent completed;
- Development of a Project Brief submitted to the EPA (carried out in November 2010);
- Conducting the Scoping Phase of the ESIA and submitted a Scoping Report to the EPA (carried out in April 2012);

- Conducting the Impact Assessment Phase of the ESIA, notably the applicable specialist assessments and baseline studies;
- Based on the findings of the specialist assessments, the EIS was developed and was submitted to the EPA at the end of July 2012; and
- The EPA and other relevant Liberian governmental agencies have reviewed the EIS and the Environmental Permit was granted for the Project in October 2012.
- In order to ensure that the Environmental permit remains valid, subsequent revisions of the ESIA and relevant EMPs have been undertaken and submitted to the EPA for approval, as have the MDA required annual environmental reports.

## 20.3 ESIA Study Area

The New Liberty Project is located in the Grand Cape Mount County of Liberia, approximately 90 km north-west of the capital, Monrovia. Liberia lies almost entirely within the Upper Guinea forest block, which forms the western part of the West African Guinean Forest.

There are numerous hills, valleys and watercourses in this zone. Vegetation in Grand Cape Mount County and in the eastern part of the country consists mostly of forests. Most of the private agricultural concessions are located in this belt where both agriculture and forestry are favoured by the prevailing topographical and climatic conditions.

The forest ecosystems can be divided into closed forest and transition or secondary forest. The closed forest can further be sub-divided into evergreen and semi-deciduous forest.

The vegetation surrounding the New Liberty Project can be broadly classified into four vegetation communities, namely:

- Pristine primary forest;
- Impacted primary forest;
- Secondary forest; and
- Transformed landscapes.

In July 2014, following the investment in the Company by the IFC, a Critical Habitat Assessment and Terrestrial Ecological Biodiversity survey was carried out by a team of international specialists at the New Liberty Project site. The survey aimed to capture the biodiversity values for the Project and the likely impact on these values. In addition, a Critical Habitat Assessment was conducted in order to categorise the habitat associated with the Project.

The findings of this survey were recorded in a report titled "Terrestrial Ecology Assessment for the New Liberty Gold Mine, Liberia", and the report concluded that the overall study area comprises Natural Vegetation, with a smaller percentage of Modified Habitat, most of which was restricted to the mine footprint area. In total, 4.9% of the total Natural Habitat will be directly lost due to mine activities.

As a part of this assessment, a total of 264 plant species were recorded. Of these, the vast majority were indigenous species and 21 species were classified as "weeds". Of the indigenous species recorded from the study site, 82 species are upper Guinea Endemics, and 2 species are endemic to Liberia.

In total, 28 amphibian species were recorded from the study area during this study. In terms of IUCN listed species, those of conservation concern include Allen's Slippery Frog (Conraua alleni) (Vulnerable), as well as eight Near Threatened species. There were several Upper Guinea endemics in the study area (thirteen) as well as several West African endemics (fourteen). Overall the amphibian diversity was classified as high and comprises species occurring in forest and stream habitats.

In terms of the reptiles recorded from the study site, this study recorded 16 species. Important to note was the presence of the African Dwarf Crocodile (Osteolaemus tetraspis), which is listed as Vulnerable by the IUCN.

In terms of bird species, 139 species were recorded in total in habitat ranging from forest, to swamps to young bush to rocky areas. Most of the avifaunal species recorded are those expected to be found in such areas, with some clear disturbance-preferring species moving into the construction areas, including the mine camp. Of conservation concern are the IUCN Vulnerable Yellow-casqued Hornbill (Ceratogymna elata), Timneh Parrot (Psittacus Timneh) and Yellow-bearded Greenbul (Criniger olivaceus).

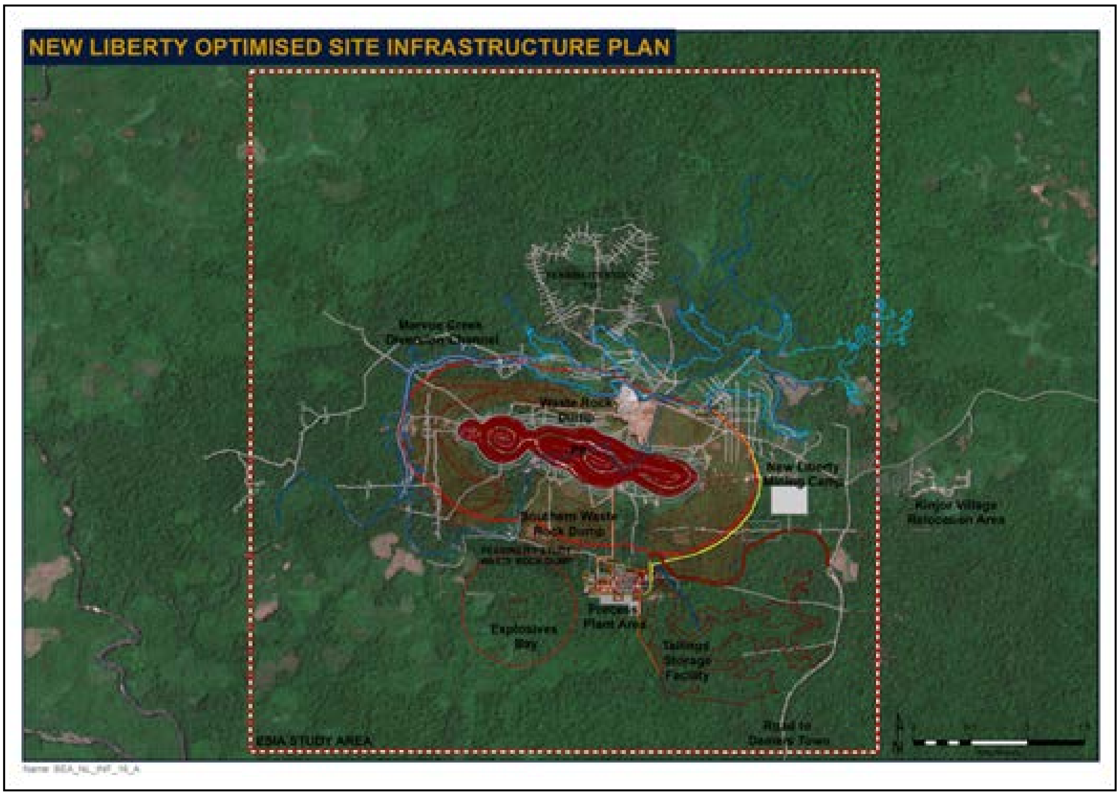
Thirty mammal species were recorded directly within the study area. These species included two bat species, two rat species, four different squirrel species, several antelope species as well as several primate species. All thirty species were observed in their habitat, as recently caught bush meat with the potential to come from the area, or were recorded from sound or signs such as scats and tracks. Of these thirty species, three are of conservation concern, all of which are listed as Vulnerable by the IUCN.

With respect to land use, the area is mainly characterized by tall tropical forests or disturbed forest area. A significant portion within the study area has experienced impacts from artisanal mining. About 30% of the area is arable, with the remaining proportion of the either marginal or unsuitable for agricultural purposes.

The demographic and socio-economic conditions of the project area, as well as its political and traditional structures, are extensively documented in the RAP. The project area includes two affected villages (Kinjor and Larjor); the main economic activities of local communities are subsistence agriculture and artisanal mining.

The study area for the ESIA consisted of the footprint of the proposed Project (approximately 8 $km^2$); upstream and downstream areas; local topography (studied as part of the infrastructure design process and fed into the ESIA); the directly-affected villages of Kinjor and Larjor; neighbouring villages within a radius of approximately 5 km of the site; and the 20 km gravel road via Daniels Town in the South that has been re-graded and serves as the Project's main access road. In addition, consideration was given to the wider geographical context where applicable.

As mentioned above, a project optimization study was undertaken subsequent to completion of the original ESIA, and new locations selected for the plant, tailings dam and mine village. These changes necessitated modification to the areas subject to detailed baseline studies. The findings of these studies were submitted as an addendum to the approved ESIA. Figure 20-1 shows the areas of infrastructure and of detailed baseline investigations.




Source: *Aureus, 2013*

**Figure 20-1: New Liberty ESIA Study Area**

## 20.4 Environmental and Social Impact Assessment

The EIS details the findings of the environmental and social baseline studies and specialist studies conducted during the ESIA; it also presents the Environmental Management Plan (EMP). The format for the EIS is taken from and aligned with the Liberian EPA "Environmental Impact Assessment Procedural Guidelines" (2006) and commitments are in line with international standards and practices.

The key factors considered in the ESIA were as follows:

- The existing/baseline environment, and obtaining relevant baseline data;
- The potential direct and indirect environmental and socio-cultural impacts of the proposed Project;
- Measures that are technically and economically feasible and that would mitigate any significant adverse environmental or social impacts related to the proposed project;
- Project alternatives that are technically and economically feasible; and
- Closure objectives and costing associated with the Project.

The following specialist studies were undertaken as part of the ESIA:

- Soils and land use;
- Surface water, also including surface water quality;
- Groundwater;
- Geochemistry;

- Air quality;
- Greenhouse gas emissions;
- Noise;
- Terrestrial ecology;
- Aquatic ecology (water quality);
- Socio-economic;
- Cultural heritage;
- Visual aesthetics; and
- Closure objectives and closure costing.

The results of the impact assessment indicate that the management and mitigation of environmental and social impacts associated with the Project are amenable to standard technical solutions. No issue has been identified that presents a technical challenge beyond that which is regularly encountered and resolved by comparable mining operations elsewhere in Africa.

## 20.5 Water management

### 20.5.1 Water balance and baseline conditions

The New Liberty Gold Mine (NLGM) project is situated in a high rainfall area (3.5m/annum) with a drier period occurring during December to February and the wet season from July to September. With management systems as currently planned there will be a positive water balance with discharge being necessary in the wet season (March to November) and none during the dry months (July to September).

Currently arsenic levels in the rivers are already at high levels with concentrations between 0.002 and 0.038 mg/L. The discharge of water along with potentially high arsenic levels is one of the project issues requiring management attention.

As part of the geochemical assessment, both static and kinetic testing of representative waste rock samples were undertaken (60 samples for static and 1 combined sample for kinetic testing). Arsenic is present in the ore body and thus the investigations into levels in the waste rock. The kinetic testing showed a steadily declining level of arsenic in solution. At all times the arsenic concentration was below the discharge levels recommended in the IFC EHS Guidelines for Mining. The results further indicated that the waste rock's acid neutralizing capacity is greater than its ability to generate acid, confirming the low potential for acid rock drainage.

### 20.5.2 Waste rock testing conducted

*Golder Programme*

Golder Associates (hereafter Golder) conducted a geochemical characterisation to answer if any of the geological and mine waste materials will generate acid mine drainage (AMD) and metal leachates (ML). They also provided management plans to any risks of AMD and ML.

- 50 samples were selected from diamond core holes, the sampling and identification was done by Aureus. This was done in agreement with Golder. The 50 samples came from 5 different boreholes.
- Analysis was done by SGS laboratories.

The tests conducted are summarised in the table below:

**Table 20-1: Summary of the NLGM geochemical characterisation programme by Golder**

| Type of Test | Specification | Samples |
|---|---|---|
| Sample preparation | Crushing to <6.3mm and further as required for specific analysis | All samples – 50 rock sample |
| Whole Rock Analysis (Total Elemental) | XRF (Lithium Metaborate Fusion) Metals with multi-acid digestion and ICP-MS | 50 waste rock samples |
| Acid Base Accounting | Sobek NP<br>Paste pH<br>Total-S<br>Sulphate-S<br>Sulphide-S<br>Total-C<br>TIC | 50 waste rock samples |
| Leaching Tests | Shake flask tests using 1:3 Solid: liquid ratios. Supernatant to be analysed for: pH, EC, TDS, Alkalinity, Acidity, CN, $NH_4^+$, NO3, F, Cl, SO4, and cations by ICP-MS | 25 waste rock samples and 1 tailings sample |
| NAG Tests | Single addition NAG tests with peroxide leach subsample to be analysed for the same parameters as leaching tests above | 8 waste rock samples |
| Mineralogy | Rietveld XRD | 10 waste rock samples and 1 tailings sample |
| Water analysis | | 1 sample |
| Kinetic tests | Humidity cell | 1 tailings sample |

Golder concluded that:

- The waste rock and tailings are unlikely to generate acid since the sulphide content is low (<0.18%) and is associated with increased alkalinity (pH>9).
- The waste rock material could potentially leach Al, Fe, P, As, Cr, Mn and Ni.
- The tailings material has a potential to leach As, B, Ca, Co, Cr, Cs, Cu, Fe, Mn, Ni and Zn.

*DRA Programme*

Ten waste rock samples were tested for Total S, ANC, NAG, TAPP, NAPP, pH and Conductivity.

Ten waste rock samples underwent multi element analysis.

*Digby Wells Programme (Waste rock and tailings)*

One kinetic test was conducted on a mixed waste rock sample.

A preliminary summary of the available tailings material test results indicate:

- That there is a level of arsenic (3.7 mg/L and 7.7 mg/L) entering into solution in the tailings samples analysed in leach tests;
- The Arsenic levels (3.7 mg/L and 7.7 mg/L) are above IFC discharge standards (0.1 mg/L), with the implication that focus must be directed towards managing potential impacts that could arise from Arsenic contamination; and
- These and previous Arsenic leach test results will be used as an input parameter in a numerical groundwater flow model to simulate possible control measures to manage impacts from the tailings material and to assist in selecting the best options for the design of both the tailings dam construction and closure scenarios.

A number of column kinetic test have been conducted to primarily evaluate the effectiveness of arsenic treatments on tailings material. Some of the tests have been completed and others are in progress. All the tests are run by ALS laboratory in Australia while the environmental interpretations are conducted by Digby Wells. Results are interpreted in line with the IFC guidelines;

Initially the kinetic tests were conducted on untreated tailings and waste rock samples that were representative of the site geochemistry. The results showed that neither the waste rock nor the tailings material was acid generating.

No metals leached from the waste rock were in excess of the IFC guidelines.

The test also showed that arsenic is the only element of concern as it exceeded the recommended IFC limits (0.1 mg/L). The arsenic in the untreated tailings decreased from a maximum of 20.8 mg/L to 8.8 mg/L during the 26 test weeks.

Following this outcome, a number of additional testwork programmes have being conducted to identify the optimal ferric dose that could treat arsenic to the recommended limits. The amount of ferric chloride ranges between 0.5 kg/t to 6 kg/t.

Recent kinetic laboratory test work undertaken by DRA Minerals Projects shows that the addition of ferric chloride in the process circuit successfully precipitates out the arsenic, resulting in a reduction in arsenic in solution to some 0.148mg/l. Definitive tests to replicate the addition of ferric chloride in the process plant are currently underway and giving very promising results. The aim is to replace the ferric chloride with more cost effective ferric sulphate and this has thus-far been shown to be just as effective at precipitating the arsenic.

Additionally, a comprehensive Water Management Plan for the Project has been developed and agreed with the IFC. This plan will be implemented throughout all phases of the project, for both effluent discharge and drinking water quality. All water qualities will be classed against the WHO drinking water standards as well as Liberian EPA Class 2 and 3 guidelines and recommendations with effluent monitoring to comply with IFC Discharge Standards and Guidelines.

The positive water balance at the Project dictates that a large volume of water needs to be recycled, discharged and managed within the mining and environmental circuit. Therefore a comprehensive surface and ground water monitoring programme has been undertaken to build up a record of baseline values, and to serve as an early warning system to any potential contamination from the mine activities.

Discharge of As containing water from the TSF is anticipated, and if not managed appropriately, this discharge could be above the WHO and IFC standards for drinking water and effluent discharge respectively, which may impact on the local communities drinking water and aquatic ecosystems downstream of the NLGM. It must be noted that current natural (background) levels for Arsenic (0.036mg/l) are above the recommended WHO Standards (0.01mg/l) and NLGM needs to manage the discharge from the operation to ensure that these natural levels are not exceeded.

The potential for metals to leach to groundwater from the TSF has been taken into account in the groundwater modelling exercise conducted and for the TSF design.

Cyanide (CN) will be used at the New Liberty Gold Mine (NLGM) to process and extract gold from the ore, after which some of the CN will be discharged along with the tailings material. The fact that the mine will discharge this tailings material into a valley fill Tailings Storage Facility (TSF), and that water discharges are necessary from this facility, require that cyanide in the tailings is destroyed prior to discharge from the plant into the TSF. Accordingly NGLM is intending to subject the tailings to the INCO cyanide destruction treatment system before being placed on the TSF. This involves the chemical oxidation of cyanide with sodium metabisulphite (SMBS) and $SO_2$/Air. Expert investigations and testwork have been conducted to provide an optimal solution for the management of cyanide to enable the discharge of the chemical to acceptable levels and in line with the IFC requirements.

## 20.6 EMP Commitments

Aureus is committed to limiting the negative impacts of operations at the proposed Project site on the environment. The EMP was developed in line with Liberian legislation and international good practice standards and principles, in order to put this commitment into practice. The EMP translates the findings and recommendations of the ESIA into measures for management and monitoring of impacts of the proposed Project activities. The mitigation measures that have been identified in the EMP are for the exploration, construction, operational and decommissioning and closure phases of the Project.

Aureus will implement this EMP, and will update it every three years after its approval, as per Liberian legislative requirements. Environmental management of the Project will be an evolving process over the life of the mine. In particular, the environmental management and mitigation measures and the monitoring programme outlined in the EIS will be updated annually for continual improvement and for management practices to remain current and aligned with Liberian legislation and industry good practice. An annual report is to be submitted to the Liberian EPA and is a condition of the Environmental Permit.

The key recommendations highlighted in the revised 2013 ESIA document include:

- To mitigate against the expected impacts associated with the generation of NOx from the power generation plant, regular generator maintenance and tuning of the combustion equipment, reduce engine idling/start-up times and reduce residence time for power generation plant will be needed;
- Development of an agricultural programme to reduce the impacts of the local community on natural resources. The programme should focus on reducing slash and burn agriculture, keeping domestic animals to reduce reliance on bush meat and the possible establishment of woodlots;
- A biodiversity offset strategy should be to implemented for the NLGM project to compensative for loss of biodiversity, potentially resulting in an overall positive impact for NLGM;
- Assist the government in managing the safety risks along the Daniels Town public road;
- Resettle Larjor and Kinjor according to the agreed RAP
- To ensure that Acid Rock Drainage ("ARD") does not occur during the decommissioning, closure, and post closure phases. If ARD does occur, to ensure that proper measures are taken to monitor and manage ARD, so as to ensure that minimal damage to the environment (e.g. soil, surface and groundwater systems) occurs.

## 20.7 Progress with the Actions Outlined by the ESIA

Since the approval of the ESIA, the following progress has been made in terms of implementing the recommendations listed above:

- The RAP and the CDP have been completed and were approved by the Liberian Government in April 2013 (discussed in more detail below);
- A comprehensive Stakeholder Engagement Plan (SEP) for the project as a whole has been developed and is currently being rolled out across the project;
- The residents of Kinjor and Larjor have been relocated to the temporary accommodation area of the RAP site and Kinjor village has now been cleared;
- The project has undertaken an in depth biodiversity study and is currently in discussions with a number of consultants to act in a role as Biodiversity Advisors to the company and help develop and offsets programme;
- An agricultural cooperative has been developed and is operational, with plans to sell produce for consumption by workers at the Project site. At the time of writing this report, 17 hectares of land have been cleared and 40 members of the local community have been employed by the partnership. First harvest is expected in December 2014, and there are plans to double both staff numbers and the area of agricultural land;
- Data collection for Air quality modelling studies is underway; and
- Numerous management plans have been developed to manage the identified impacts.

## 20.8 Resettlement Action Plan

The Project involved the relocation and resettlement of two villages (Kinjor and Larjor) encompassing 325 property owners and their households, as well as the relocation of some households along the access road. Relocation of the latter is primarily motivated by potential safety impacts associated with increased traffic volumes caused by the Project, although the undertaking of this relocation is the responsibility of the Government of Liberia. International best practice for resettlement related to private sector projects is guided by the IFC's Performance Standards on Social and Environmental Sustainability, and particularly defined by the IFC's Performance Standard 5: Land Acquisition and Involuntary Resettlement.

The RAP has been developed utilising this Performance Standard to specify how relocation and compensation for affected assets and livelihoods will be undertaken. In accordance with international best practice, the RAP identifies all laws of Liberia that are applicable to land acquisition and involuntary resettlement, including relevant local customs and traditions that govern affected communities.

The legal framework lays the foundation for the four key elements of the RAP:

- Approach to Land Access and Management
- Establishing rates of compensation
- Determining eligibility for compensation and resettlement assistance, including livelihood initiatives.; and
- Establishing mechanisms to resolve grievances among affected persons related to compensation and eligibility.

The purpose of developing a RAP was is to outline the framework for execution of the NLGM Resettlement Project as early as possible within the project development cycle. This allowed for effective disclosure to key stakeholders, and subsequent feedback and inputs, prior to completion of resettlement negotiations and implementation.

As indicated above, the RAP has been approved by the Liberian EPA and the relevant owners' compensation packages approved. The RAP stakeholder engagement process expands on the EIS process in that it includes the regulatory authorities that are expected to be involved in the development and implementation of the RAP. The primary focus of the RAP stakeholder engagement process is on the directly affected property owners in the Kinjor and Larjor communities and along the main access road, as well as their representative bodies (including the traditional authorities concerned).

In order to involve affected communities in the resettlement planning and implementation processes, a Resettlement Working Group (RWG) was established and comprises representative members of the resettlement-affected communities and households, as well as the relevant local government structures, town councils, traditional authorities, women's group, the youth, and non-governmental organizations. The RWG is chaired by Aureus and the chairperson nominated is available at all formal RC meetings.

### 20.8.1 Resettlement Site Selection

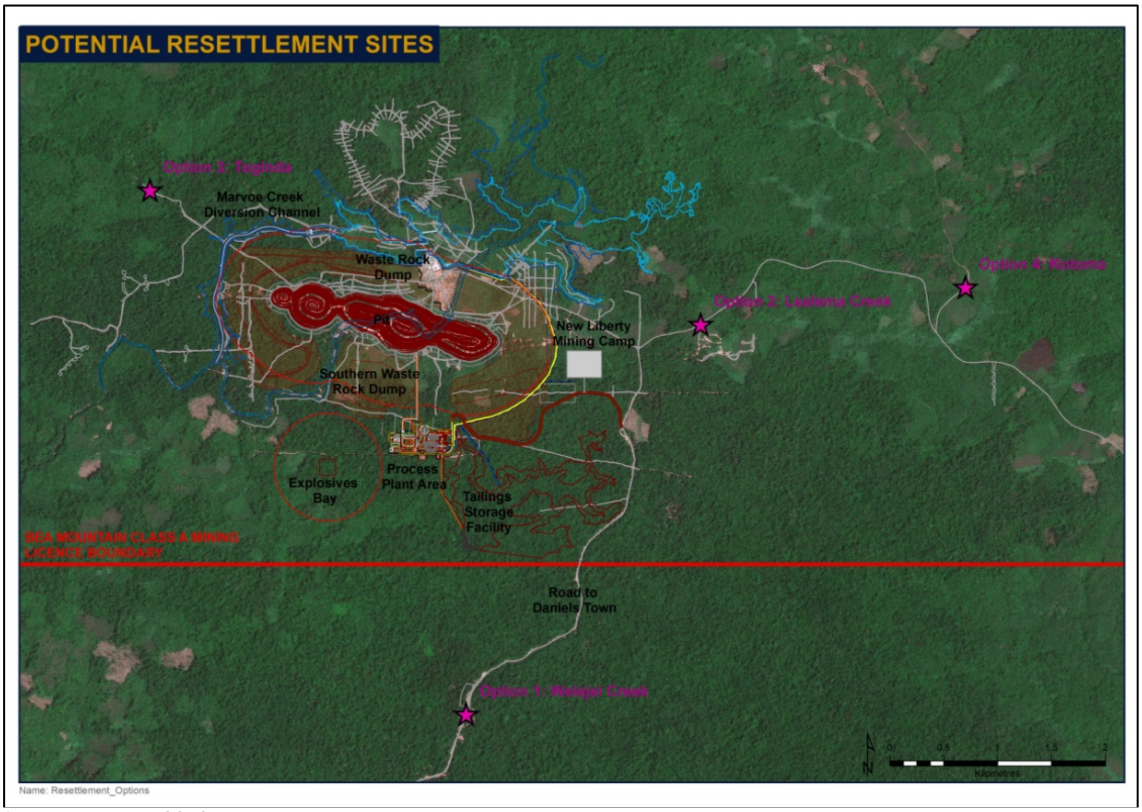
The following general criteria were applied in the selection of a preferred resettlement site:

- Availability of land and the quality/production potential of the land;
- Current land ownership and land acquisition conditions;
- Location of resettlements in relation to the licence area;
- Proximity to the NLGM in order to:
    - Accommodate local recruitment processes;
    - Minimize travel distances to the mine for the local workforce;
    - Prevent/control human settlement in too close proximity to the mine camp;
- Proximity/access to public infrastructure and community facilities;
- Availability of land to develop agriculture-based livelihood restoration projects;
- Settlement patterns and land use at the resettlement site (integration potential);
- Availability and quality of natural resources (particularly water); and
- Compatibility with the socio-cultural practices and economic activities of host community.

During an initial resettlement site selection workshop with the RWG, four site options were identified as shown in Figure 20-2. Site Option One was the preferred option of the RWG based on the outcome of a workshop on site options. However, this site is located on an adjacent concession licence area, which would have significantly complicated the land acquisition process. During discussions with the RWG, it was agreed that Site Option One should be discarded based on these limitations. Site Option 2 (Lealema), was subsequently selected by the RWG as the preferred site. Lealema is located approximately 4.5 km from Kinjor and 3 km from the proposed new Project mine camp.

The acquisition of the replacement land from the owners was finalised in November 2014. The relocation of the local communities was completed in September 2014 to a temporary area within the RAP site. Construction at the RAP is nearing completion for the permanent houses within the village, and once complete, these will be handed over to the local community. As a part of the RAP agreement those eligible for relocation under the RAP are being handed the title deeds to two properties, first their temporary relocation residence and secondly, on completion, their permanent residence.

At the time of writing this report, the area of the old Kinjor village had now been cleared, allowing for the commencement of pre-strip mining operations and grade control drilling.




Source: *Aureus, 2013*

**Figure 20-2: Potential Resettlement Sites**

## 20.9 Community Development Plan

Aureus is committed to developing measures and action plans in line with its policy to conduct its business activities in a manner that promotes sustainable development and social welfare in the areas in which it operates. Moreover, Aureus has committed to mitigate and manage the resettlement impacts associated with the development of the Project in accordance with the requirements of IFC PS with regard to the following:

- Improve, or restore, the livelihoods and standards of living of displaced persons;
- Ensure that affected vulnerable people receive additional assistance, if required; and
- Provide opportunities to displaced people to derive appropriate development benefits from the project.

Aureus commissioned independent consultants Digby Wells to develop a Community Development Plan (CDP) for the resettlement of affected households and communities. This plan was completed in December 2012 and was subsequently approved by the Liberian EPA in January 2013.

Based on the development opportunities assessed, a list of potential CDP projects was identified, including:

- Vegetable production (This co-operative has commenced and currently employs 40 members of the local community);
- Seedling and expanded plant nursery;
- Vegetable pack house;

- Lowland rice production, drying and milling;
- Cassava production and value-adding demo;
- Mobile cassava grater business;
- Corn production and milling demonstration unit;
- Chicken broiler unit;
- Plantain and banana production;
- Abaca and banana fibre production;
- Pineapple production and value-adding;
- Cashew nut production and value-adding; and
- Coconut production.

Where feasible, the final CDP will be expanded to incorporate the community development aspects of the BMMC MDA. The MDA requires that operations shall be carried out in a manner that is consistent with the continuing economic and social viability of centres of population that have formed, and/or may form as a result of the NLGM operations during the term of the MDA.

The development and operation of the Project will have both positive and negative impacts on the socio-economic structure of the Project area and its environments, as well as impacts at a District and National level.

The positive impacts will relate mainly to the economic advantages which will have immediate and long-term benefits for the socio-economic environment. This will be achieved in various ways at National, District and Local levels through the payment of taxes and royalties, increased employment opportunities, training, purchase of goods manufactured and supplied in Liberia, cash compensation for farms, commercial opportunities and an improvement in local infrastructure by the establishment of the Resettlement site/town. The development of the Project will bring much-needed investment and development opportunities with consequent positive impacts on employment and the affected communities.

Negative impacts relate to the disruption of the local social dynamics and increased pressure on local infrastructure and resources, mostly due to the influx of people to the area. At the time of writing this report, Aureus is currently working with Digby Wells to develop an Influx Management Plan to mitigate against these affects.

## 20.10 Rehabilitation and Closure

A closure study was undertaken for the Project as part of the ESIA. This study focused on developing closure objectives for the proposed mine site, as well as producing a high level closure costing table for the site.

The overall closure goal for the Project is to progressively re-instate native forest areas that are safe, stable and non-polluting, mimicking the current land use, and taking into account the unavoidable remaining mining residue and/or disturbances towards leaving behind a positive post-mining legacy. The Closure costs also reflect the conditions spelled out in the MDA, whereby the Government of Liberia take ownership of all associated project infrastructure.

# 21 CAPITAL AND OPERATING COSTS

## 21.1 Introduction

The following section summarises the capital and operating costs of the Project and assumes a conventional open-pit mining operation, a two-stage crushing process, ball and secondary milling, a CIL circuit followed by AARL elution. The plant design has been based on the treatment of 1.1 million tonnes per annum of ore.

The capital and operating costs presented below form the basis of the economic analysis performed in Section 22.

## 21.2 Operating Cost Estimate

### 21.2.1 Accuracy and Basis of Estimate

The process operating cost estimate was completed from a zero base. All labour, materials and consumables have been included in this estimate. Estimates were generated by Aureus based on inputs from the Aureus Owners Team and DRA in turn based on the work of ALS Metallurgy, using quotations from Afrilog the appointed reagents and consumables supplier. General and administration costs were determined by the Aureus Owners Team.

The mining operating costs estimate was completed by the Aureus Owners Team and includes costs provided by MonuRent for the costs of providing the mining fleet.

This estimate excludes the cost of transporting product materials from site.

### 21.2.2 Base Date

The base date of this operating estimate is December 2014.

### 21.2.3 Definitions of Costs

The costs in this estimate can be defined as all costs that will be incurred in the life cycle of the operation.

Fixed Cost: This component can be defined as the costs that will be incurred irrespective of production rates. These costs would typically include the following:

- Labour
- Lease Costs
- Environmental and Social

Variable costs can be defined as costs that are only incurred during production. It can be said that these costs are based on rates per tonnage and the total costs are incrementally incurred as production rates increase.

The costs would typically include the following:

- Reagents
- Maintenance Spares
- Diesel

- Tyres
- Oil
- Consumables
- Power
- Water
- Explosives

### 21.2.4 Processing Costs

This section provides a description of the operating costs for the New Liberty metallurgical processing facility. The following are examples of those items excluded from this estimate:

- All VAT, import duties and/or any other statutory taxation, levies and/ or national and local institutions
- Contributions to social programmes
- All owner's budget costs, head office, administration charges, payroll etc.
- Contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements
- All costs associated with grade control, blending and stockpile management. These costs are excluded from the plant estimates and included in the mining costs
- Final product transport, marketing and sales agreement costs.

The basis for the estimated operating cost is given in each of the sections to follow. All costs are provided in United States dollars.

*Labour*

The labour costs have been based on the organogram for the plant, as presented Figure 21-1 below. The cost for labour has been determined based on a labour cost model derived for the project using information from the owner's team.

**Table 21-1: New Liberty Process Plant Labour Requirements**

| Personnel Complement Breakdown | Section | Responsibility | Grade | No. | Local/ Expat |
|---|---|---|---|---|---|
| **Management** | | | | | |
| Engineering Manager | Management | Plant | 2 | 1 | Expat |
| Maintenance Manager | Management | Engineering | 2 | 1 | Expat |
| Metallurgical Manager | Management | Process | 3 | 1 | Expat |
| | | **Total** | | **3** | |
| **Administration** | | | | | |
| Officer, Safety | Administration | Safety | B | 1 | Local |
| | | **Total** | | **1** | |
| **Laboratory** | | | | | |
| Services of ALS to be contracted in | | | | | |
| | | **Total** | | **0** | |
| **Process Supervision** | | | | | |
| Metallurgical Foreman | Process Supervision | Plant | 5 | 2 | Expat |
| Metallurgical Foreman | Process Supervision | Plant | 5a | 2 | Local |
| Shift Foreman | Process Supervision | Plant | 5 | 3 | Expat |
| Shift Foreman | Process Supervision | Plant | 5a | 3 | Local |
| | | **Total** | | **10** | |
| **Process** | | | | | |
| Control Room Operators | Process | Plant | 6 | 1 | Expat |
| Control Room Operators | Process | Plant | B | 3 | Local |
| Crusher Operators | Process | Crushing | C | 4 | Local |
| Mill Operators | Process | Milling | B | 3 | Local |
| Gravity and Thickening Operators | Process | Leaching | B | 3 | Local |
| CIL Operators | Process | CIL & CND | B | 3 | Local |
| Elution Operators | Process | Elution | B | 3 | Local |
| Tailings Dam Foreman | Process | Tailings Facility | 6a | 1 | Local |
| | | **Total Process** | | **21** | |
| | | **Day Shift Process** | | **4** | |
| | | **Total per shift** | | **9** | |
| **Maintenance** | | | | | |
| Maintenance Planner | Engineering Maint | Plant | 5 | 1 | Expat |
| Maintenance Planner | Engineering Maint | Plant | 6a | 1 | Local |
| Mechanical Foreman | Engineering Maint | Mechanical | 4 | 1 | Expat |
| Mechanical Technician | Engineering Maint | Mechanical | 6a | 1 | Local |
| Boilermakers | Engineering Maint | Mechanical | 5 | 1 | Expat |
| Boilermakers | Engineering Maint | Mechanical | 6a | 1 | Local |
| Boilermaker Assistants | Engineering Maint | Mechanical | A | 2 | Local |
| Fitters | Engineering Maint | Mechanical | 5 | 1 | Expat |
| Fitters | Engineering Maint | Mechanical | 6a | 2 | Local |
| Fitters Assistants | Engineering Maint | Mechanical | A | 3 | Local |
| Rigger / Crane Operator | Engineering Maint | Mechanical | A | 1 | Local |
| Rigger Assistant | Engineering Maint | Mechanical | B | 2 | Local |
| Generator operators | Engineering Maint | Mechanical | 6a | 3 | Local |
| Maintenance Operative | Engineering Maint | Mechanical | A | 4 | Local |
| Electrical Foreman | Engineering Maint | Electrical | 4 | 1 | Expat |
| Electrical Technician | Engineering Maint | Electrical | 6a | 2 | Local |
| Instrumentation Foreman | Engineering Maint | Instrumentation | 4 | 1 | Expat |
| Instrumentation Technician | Engineering Maint | Instrumentation | 6a | 1 | Local |
| | | **Total Maintenance** | | **29** | |
| | | ***Total Staff Complement*** | | ***64*** | |
| | | **Expat Complement** | | **15** | |
| | | **Local Complement** | | **49** | |

The total cost for labour includes all production and engineering staff directly associated with the processing plant and laboratory, and covers a complement of 64 people. The total cost shown is a total cost to the Company inclusive of all allowances, medical contributions, insurances, training and expat contingent travel costs as provided by the Company. The processing plant organogram is presented in Figure 21-1, where the number of employees and proposed grading band is shown.

**Table 21-2: New Liberty Process Plant Labour Requirements**

| Employee Group | Complement | Annual Cost (USD) |
|---|---:|---:|
| Management | 3 | 616,768 |
| Administration | 1 | 11,180 |
| Process Supervision | 10 | 523,750 |
| Process | 21 | 274,545 |
| Engineering Maintenance | 29 | 928,855 |

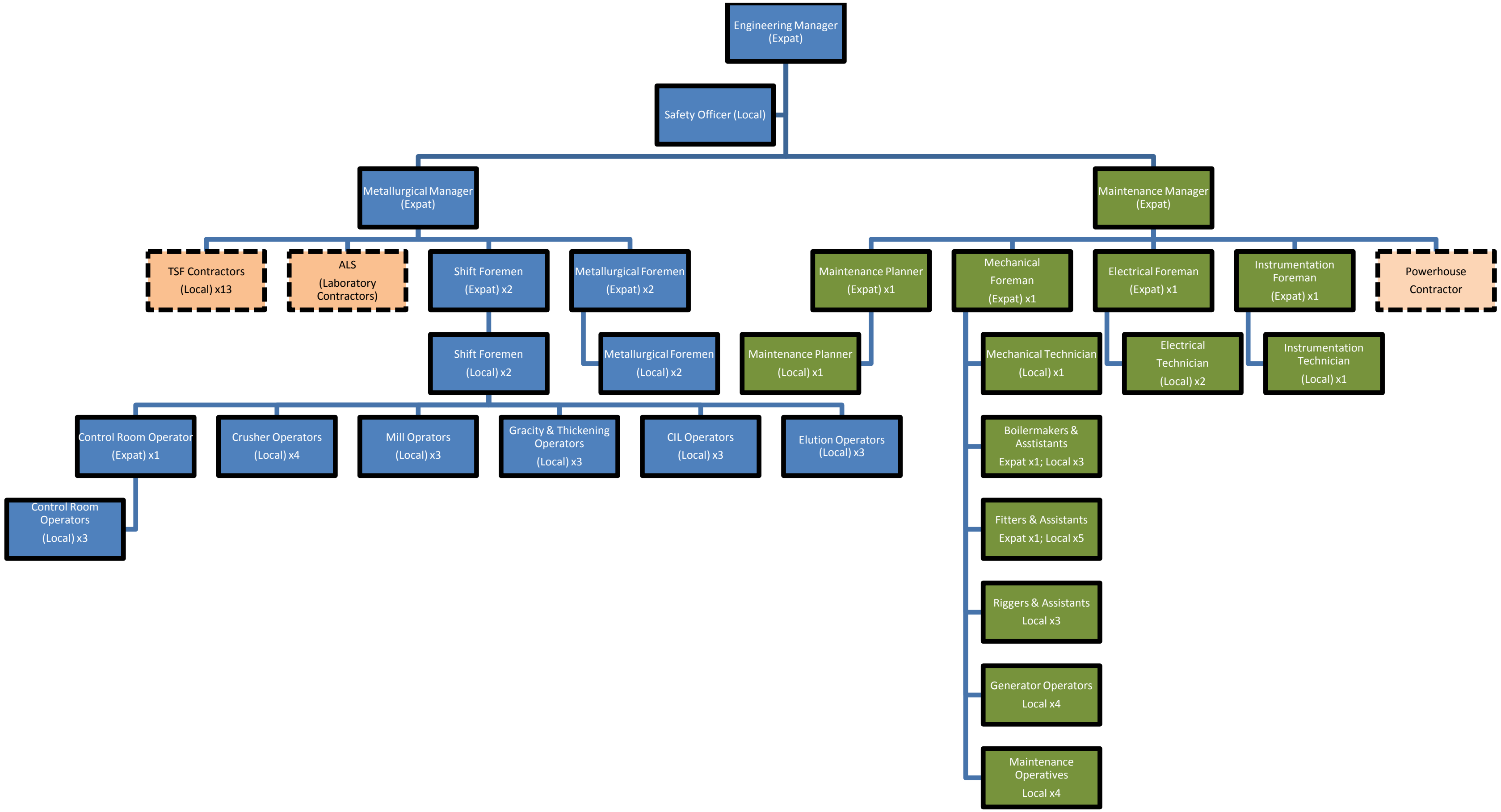



**Figure 21-1: New Liberty Process Plant Organogram**

*Power*

The total connected electrical load, inclusive of all standby units in the plant, is approximately 12.25MW by summation of all the equipment specified on the mechanical equipment list. The estimated average running load has been calculated using expected power draw from the equipment sizing calculations. The design operating power draw for the plant has been estimated at 8.1MW.

The plant power is to be generated by a diesel power generation plant. The power cost (USD/kWh) for plant operations was calculated based on the quoted diesel consumption figures and pricing as presented in Table 21-3 below. The estimated calculated power costs are summarized in Table 21-4.

**Table 21-3: Basis for Determination of Plant Power Costs**

| **Basis for Determination of Power Costs** | | |
|---|---|---|
| Diesel Cost (Excluding amortization of capital) | USD/L | USD0.80-98 |
| Diesel Consumption | g/kWh | 201.1 |

**Table 21-4: Power Cost Estimate (at USD0.98 /litre)**

| **Power Requirements** | | **Cost** |
|---|---|---|
| Primary Crusher | kWh/t | 0.37 |
| Secondary Crusher (2 x HP500) | kWh/t | 1.01 |
| Ball Mill | kWh/t | 22.00 |
| Regrind Mill | kWh/t | 6.32 |
| Plant Other | kWh/t | 18.32 |
| Total Power | kWh/t | 48.02 |
| **Total Power** | **USD/t** | **11.33** |

*Water Cost*

Water consumption was calculated using the nominal mass balance as indicated on the Process Flow Diagrams (PFDs). Water can be sourced from a local river for the New Liberty project and there was no cost associated with this water supply. The water treatment costs (reagents and flocculants) for the treatment of potable water is insignificant and has been excluded from the analysis. The pumping costs for supplying water from the various storm water impoundments and the river supply system are included in the plant power costs.

*Reagent Consumption and Costs*

Expected nominal reagent consumption rates were estimated based on results obtained from the laboratory scale test work, vendor specifications and mass balancing. The reagent prices include freight to site, and include all clearance charges and but no taxes that may be incurred. All reagent costs were quoted on the basis of deliveries to site being made ex Monrovia. A minimum of 60 days of stock will be held in a store on site. This will ensure that total stockholding on site will be equal to or exceed the typical sourcing lead times for relevant imported product.

**Table 21-5: Reagent Cost Estimate**

| Reagents | | Cost |
|---|---|---|
| **Process plant and gold room reagents** | **USD /t** | **5.52** |

*Liners and Grinding Media*

Crusher Liners

The maintenance cost for the jaw crusher has been based on replacing one liner set per year, while the cone crusher costs have been based on replacing two liner sets per year.

Mill Liners

The liner life for the ball mill was estimated using the abrasion index test results, mill dimensions and anticipated grinding media load. Unit costs (for full sets) were given by various mill suppliers in their tenders submitted for the project.

**Table 21-6: Liner Cost Estimate**

| Liners | | Cost |
|---|---|---|
| Crushing and Milling | USD/t | 0.76 |
| **Total Liners** | **USD/t** | **0.76** |

Grinding Media

The ball mill grinding media wear rates were determined by the DRA in-house milling consultant, while the Vertimill media consumption figure has been estimated based on a consumption figure of 0.05kg/kWh as provided by Metso.

**Table 21-7: Grinding Media Cost Estimate**

| Grinding Media | | Cost |
|---|---|---|
| Ball Mill | g/t | 510.0 |
| VertiMill | g/t | 334.7 |
| **Grinding Media** | **USD/t** | **1.53** |

*Diesel Consumption and Costs*

The expected nominal diesel consumption rates are based on diesel consumption figures from vendors for plant equipment that is diesel driven namely the elution heating system, carbon regeneration kiln and gold room furnace.

**Table 21-8: Diesel Cost Estimate**

| | | Cost |
|---|---|---|
| **Diesel** | USD/t | 0.34 |
| **Total** | **USD/t** | **0.34** |

*Mechanical Maintenance*

An estimate for the plant maintenance costs was determined by summation of the vendor specified operational spares required for large critical equipment and includes:

**Table 21-9: Maintenance Cost Estimate**

| Equipment/Description | Cost |
|---|---:|
| Spares - Crushing, Milling, CIL | 173,338 |
| Mechanical Stores - Crushing, Milling, CIL | 54,412 |
| Pipes and Fittings | 42,036 |
| Steel Plate & Sections | 41,674 |
| Conveyor Belt & Spares | 22,348 |
| Cyclone Spares | 19,697 |
| Agitator Spares | 16,801 |
| Apron Feeder Spare | 14,939 |
| O/H Cranes Crawl | 10,718 |
| Tower Crane | 10,321 |
| Ball Mill Spares | 9,520 |
| Regen Kilns | 13,116 |
| Other consumables | 15,467 |
| Vibrating Screens | 10,606 |
| Screens | 7,007 |
| Interstage Screens | 6,701 |

*Overall Plant Operating Cost*

The overall plant operating cost estimate is shown in Table 21-10 below. The breakdown shows all the costs discussed above and includes costs for the laboratory and other miscellaneous items.

**Table 21-10: Overall Processing Operating Cost Estimate Summary**

| Variable Cost | | Cost |
|---|---|---|
| Reagents | USD/t | 5.52 |
| Labour | USD /t | 2.01 |
| Grinding Balls | USD /t | 1.53 |
| Mill Liners | USD /t | 0.35 |
| Crusher Liners | USD /t | 0.41 |
| Power Costs (incl Verti-Mill & G&A) | USD /t | 11.33 |
| G&A Power usage | USD /t | (1.72) |
| Elution Costs | USD /t | 0.34 |
| Laboratory / Assay Costs | USD /t | 0.42 |
| Freight Costs | USD /t | 0.60 |
| Maintenance (Eng Costs) | USD /t | 1.00 |
| Milling Screens & Cyclones | USD /t | 0.06 |
| Detox & Gold Room | USD /t | 0.22 |
| TSF operating | USD /t | 0.43 |
| Aachen & Oxygen Contracts | USD /t | 0.48 |
| Lubes and maintenance | USD /t | 0.29 |
| Various | USD /t | 0.31 |
| **Total Estimated Operating Cost (Excl G&A) cost** | **USD /t** | **23.58** |

An allowance of USD0.10 / tonne mined has been included in the mining cost to cover ROM re-handle costs.

## 21.3 Mining Operating Costs

### 21.3.1 Introduction

The following section provides a description of the operating costs for the New Liberty mining operations. It should be noted that the following are not included in this estimate:

- All VAT, import duties and/or any other statutory taxation, levies and/ or national and local institutions
- Contributions to social programmes
- All owner’s budget costs, head office, administration charges, payroll etc.
- Contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements
- Final product transport, marketing and sales agreement costs.

The basis for the estimated operating cost is given in each of the sections to follow. All costs are provided in United States dollars.

### 21.3.2 Labour

The labour costs have been based on the organogram for the mining department, as presented in Table 21-11. The cost for labour has been determined based on a labour cost model derived for the Project using information from the owner’s team.

**Table 21-11: New Liberty Mining Operations Labour Requirements**

| Personnel Complement Breakdown | Section | Responsibility | Grade | No. | Local/ Expat |
|---|---|---|---|---|---|
| **Management** | | | | | |
| Project Manager, Mining (D/S) | Management | Mining | 2 | 1 | Expat |
| Project Manager, Mining (N/S) | Management | Mining - Night Shift | 3 | 1 | Expat |
| Manager, Technical Services | Management | Technical Services | 2 | 1 | Expat |
| Manager, Mine Planning and Survey | Management | Mine Planning & Survey | 2 | 1 | Expat |
| Manager, Grade Control | Management | Grade Control | 3 | 1 | Expat |
| Manager, SHEQ | Management | SHEQ | 3a | 1 | Local |
| | | **Total** | | **6** | |
| **Administration** | | | | | |
| Clerk, Mining | Administration | Administration | C | 1 | Local |
| Data Base Administrator | Administration | Data integrity | 3 | 1 | Expat |
| Training Officer | Operating | Training | 5 | 1 | Expat |
| Officer, Safety | Operating | Safety | B | 1 | Local |
| | | **Total** | | **4** | |
| **Mining Supervision** | | | | | |
| Foreman, Mining | Supervision | Mining | 5 | 4 | Expat |
| Engineer, Drill & Blast | Supervision | Drilling & Blasting | 5 | 1 | Expat |
| Foreman, Drill and Blast | Supervision | Drilling & Blasting | 5 | 2 | Expat |
| | | **Total** | | **7** | |
| **Mining Operations** | | | | | |
| Pit Technician | Operating | Mining operating | B | 132 | Local |
| Crew, Dewatering | Operating | Dewatering | C | 12 | Local |
| Pit Technician, Drill & Blast | Operating | Drilling & Blasting | B | 10 | Local |
| Mining Engineer | Operating | Mining operating | 4a | 1 | Local |
| | | **Total** | | **155** | |
| | | **Day Shift Mining** | | **77** | |
| | | **Total per shift** | | **78** | |
| **Services** | | | | | |
| Surveyor, Senior | Technical Services | Survey | 5 | 1 | Expat |
| Surveyor | Technical Services | Survey | 5 | 1 | Expat |
| Surveyor Technicians | Technical Services | Survey | A | 4 | Local |
| Learner Survey | Technical Services | Survey | B | 3 | Local |
| Grade Control Supervisor | Technical Services | Grade Control | A | 4 | Local |
| Pit Technicians, Grade Control | Technical Services | Grade Control | B | 8 | Local |
| Grade Control Spotters | Technical Services | Grade Control | C | 6 | Local |
| Mine Geologist (Intern) | Technical Services | Geology | 4a | 2 | Local |
| Geology Supervisor | Technical Services | Geology | 5a | 1 | Local |
| Senior Mine Geologist | Technical Services | Geology | 4 | 1 | Expat |
| | | **Total Technical Services** | | **31** | |
| | | ***Total Staff Complement*** | | ***203*** | |
| | | **Expat Complement** | | **17** | |
| | | **Local Complement** | | **186** | |

The total cost for labour includes all production and Technical services staff directly associated with the mining operations, and covers a complement of 203 people. The cost shown is a cost to company inclusive of medical contributions and insurances, as provided by the Company. The mining organogram is presented in Table 21-11, where the number of employees and proposed grading band is shown.

**Table 21-12: New Liberty Mining Labour Requirements**

| Employee Group | Complement | Annual Cost (USD) |
|---|---|---|
| Management | 6 | 1,071,802 |
| Administration | 2 | 172,838 |
| Supervision | 7 | 586,600 |
| Operating | 157 | 1,787,290 |
| Technical Services | 31 | 655,855 |

### 21.3.3 HME

The Heavy Mining Equipment (HME) costs have been based on the mining plan requirements for the mining department, as presented in Table 21-13 below. The cost for HME and associated support equipment, as well as MonuRent support staff costs has been determined based on the number of machines required to execute the mine plan. This cost has been derived for the operation using information from the owner's team and the management of MonuRent who will be renting the equipment to Aureus.

The fuel consumption was determined by taking the equipment usage hours at the hourly burn rate per machine and the number of machines scheduled for use to arrive at the fuel volume requirement. A diesel price of USD0.80 per litre was used through to the end of 2017 then USD0.98 through the remainder of the life of mine to arrive at the cost for fuel.

**Table 21-13: New Liberty HME and associated costs**

| Description | | Unit Cost |
|---|---|---|
| HME | USD/t | 1.18 |
| Support Equipment | USD /t | 0.10 |
| Support Staff Costs | USD /t | 0.05 |
| **Total Hire cost** | **USD /t** | **1.34** |
| | | |
| Fuel | USD /t | 0.33 |
| **Total HME usage cost** | **USD /t** | **1.67** |

### 21.3.4 Explosives, Consumables and ROM Re-Handle

The explosives, consumables and ROM re-handle costs have been based on the mining plan requirements for the mining department, as presented in Table 21-14 below. The cost for explosives and blast consumables is calculated from a drilling program based on lengths of holes drilled and the type of explosives for the conditions.

The ROM re-handle costs have been derived by dedicating two pieces of equipment to the task of re-handling on the ROM. Again, the fuel consumption was determined by taking the equipment usage hours at the hourly burn rate for the selected machines.

**Table 21-14: New Liberty Explosives, Consumables and ROM Re-Handle costs**

| Description | | Unit Cost |
|---|---|---|
| Explosives | USD/t | 0.27 |
| Blast Consumables | USD /t | 0.04 |
| Drill Spares (rods; bits & Access) | USD /t | 0.02 |
| GET | USD /t | 0.02 |
| Tyres | USD /t | 0.05 |
| **Total Consumables** | **USD /t** | **0.41** |
| ROM Re-Handle | USD /t | 0.10 |
| **Total** | **USD /t** | **0.51** |

### 21.3.5 Mining - Other Departmental Costs

The mining support costs expected over the mine life and covered here include:

- Survey: survey equipment and software
- Re-licensing costs
- Mining Operations: rentals of equipment not scheduled under the MonuRent contract for items such as dewatering pumps, dewatering drilling, light plants and stemming crushing
- Geology and Geotechnical: various consultants
- Grade Control: RC grade control drilling and the resultant assay costs

**Table 21-15: New Liberty Mining – Other Departmental costs**

| Description | | Unit Cost |
|---|---|---|
| Mine Survey | USD/t | 0.01 |
| Mining Operations | USD /t | 0.05 |
| Geology | USD /t | 0.01 |
| Geotech | USD /t | 0.01 |
| Grade Control | USD /t | 0.03 |
| **Total** | **USD t** | **0.11** |

### 21.3.6 Mining – Total Unit Cost

The total mining cost estimate is shown in Table 21-16 below.

**Table 21-16: New Liberty Mining – Total Operating costs**

| Variable Costs | | Unit Cost |
|---|---|---|
| HME | USD /t | 1.18 |
| Support Equipment | USD /t | 0.10 |
| Support Staff Costs | USD /t | 0.05 |
| Fuel | USD /t | 0.33 |
| Explosives | USD /t | 0.27 |
| Consumables | USD /t | 0.14 |
| ROM Re-Handle | USD /t | 0.10 |
| Departmental Costs | USD /t | 0.11 |
| Labour | USD /t | 0.25 |
| **TOTAL Mining Costs** | **USD /t** | **2.53** |

## 21.4 General and Administration Operating Costs

General and Administrative overhead (G&A) costs required to directly support the New Liberty operations have been estimated as part of the budgeting process by Aureus. These costs include, amongst others, overhead labour, accommodation and messing, camp management and security, social and environmental programmes and studies, power costs, professional services to support the mine operations and general administrative overheads. The life of mine average cost estimate for G&A costs is USD6.85 / tonne of ore processed.

## 21.5 Capital Cost Estimate

### 21.5.1 Introduction

The capital cost estimate has been developed based on the value of orders placed and final or interim payments made on contracts, with a forecast cost to complete estimated by the Aureus Owners Team based on inputs from the DRA engineering and construction teams. It should be noted that this is the total capital cost for the construction of the Project inclusive of that already spent.

**Table 21-17: Capital Cost Estimates**

| | Incurred to 31 December 2014 USD million | Forecast to complete USD million | Total capital cost estimate USD million |
|---|---|---|---|
| Earthworks and Civils | 41.7 | 4.3 | 46.0 |
| Structural, Mechanical, Pipework and Plating – Supply & Install | 16.7 | 4.6 | 21.3 |
| Electrical Instrumentation – Supply & Install | 5.7 | 1.9 | 7.6 |
| Mechanical Equipment | 13.8 | 0 | 13.8 |
| Infrastructure | 1.6 | 0.1 | 1.7 |
| Camp Management | 1.8 | 1.9 | 3.7 |
| Transport | 5.5 | 1.7 | 7.2 |
| Indirects and Engineering – includes EPCM | 22.7 | 6.6 | 29.3 |
| Mining | 5.5 | 20.8 | 26.3 |
| Owner's costs – includes contingency | 8.4 | 6.9 | 15.3 |
| **TOTAL** | **123.4** | **48.8** | **172.2** |

Deferred capital costs will be incurred following the commencement of production. Sustaining capital includes the mine closure costs of USD7.8 million. The mine closure costs cover environmental aspects at the mine and process plant sites. The diesel generators and fuel farm are covered by lease agreements over the LOM. The deferred capital costs over the LOM are summarised in Table 21-18.

**Table 21-18: Deferred Capital Cost Estimates**

| | USD million |
|---|---|
| Sustaining capital and mine closure | 14.5 |
| Diesel Generators and fuel farm over LOM | 18.2 |

### 21.5.2 Earthworks

The earthworks estimate can be divided into three main sections:

*Marvoe Creek Diversion Channel (MCDC)*

This area is complete and costs are based on actual values.

*Tailings Storage Facility (TSF)*

Epoch Resources was appointed to design the tailings facility and produce a bill of quantities. This area is partially complete and final costs are based on a combination of actuals and forecasts.

*Plant Terrace*

This area is complete and costs are based on actual values.

### 21.5.3 Civil Works

The civil works can be divided into two sections:

*Infrastructure*

With the exception of the TSF Penstock line, the civil works for infrastructure are essentially complete. Costs are based on actual cost for complete areas and forecasts for the remainder.

*Processing Plant Civils*

Civil works are essentially complete in the plant with costs being based on actual costs incurred.

### 21.5.4 Building Works

Building works are complete and costs are based on actual cost.

*Structural Steelwork Supply and Erection*

The supply contract for structural steel was placed with Group 5. This contract is essentially complete and costs are based on a preliminary final account. The costs for the installation portion of the contract are based on the actual amounts of the interim certificates, and a forecast based on work still to be completed and an allowance based on the EPCM contractors experience in construction.

*Plate work and Lining*

The supply contract for plate work and lining was placed with Group 5. This contract is essentially complete and costs are based on a preliminary final account. The costs for the installation portion of the contract are based on the actual amounts of the interim certificates, and a forecast based on work still to be completed and an allowance based on the EPCM contractors experience in construction.

*Mechanical Equipment*

All equipment costs are based on actual order values with provisions for commissioning on site by vendors as appropriate.

*Piping and Valves*

The piping supply and install contract was placed on G5. Supply is essentially complete and the draft final certificate has been used for the supply costs.

*Electrical and Instrumentation*

An 11kV, 10.8MW, diesel driven Build, Own, Operate and Transfer (BOOT) power station will be operated on a deferred terms agreement.

Electrical and instrumentation equipment supply costs are based on orders placed.

The Group 5 interim certificates for both the supply and install contracts have been used for the costs estimate, with the EPCM contractors forecast for work still to be completed.

### 21.5.5 Transportation

Transport costs are based on actual costs to date, and a forecast for incomplete deliveries.

### 21.5.6 Project and External Services

*EPCM*

The EPCM costs are based on DRA's interim invoices and a forecast based on manpower still required to complete construction and commissioning, with an allowance for disbursements still to be made.

*Mining*

Mining costs include pre-production stripping and mining costs for approximately 10 million tonnes of material and the related mining infrastructure requirements.

*Consumables and Spares*

The costs for the first fill and for consumables have been calculated by Aureus based on DRA's (EPCM Contractor) estimate of the quantities required. Spare parts costs have been included in the equipment orders where the costs reflect as such.

*Village Relocation*

The village relocation costs have been based on the requirement to relocate approximately 325 households (approximately 1,800 individuals). This is based on actual costs to date and forecast costs to complete.

*Preliminary and General Costs (P&Gs)*

P&G costs include all contractors' overheads such as contractual requirements (safety, sureties, insurance, etc.), the site establishment and the removal thereof, and company and head office overheads. They also cover supervision, travel to and from the site, contractor supplied temporary facilities, offices and lay-down areas, tools and contractors' equipment. P&Gs costs are included in the relevant contracts.

# 22 ECONOMIC ANALYSIS

## 22.1 Economic Model

Aureus has developed a financial model based on the Mineral Reserves only in order to evaluate the economics of the Project. SRK confirms that the inputs to the financial model have been appropriately derived from, and reflect the investigations of the various studies, as commented on in the previous sections of this report.

### 22.1.1 General Assumptions

The financial model reflects pre-finance cashflows, allows for working capital and is based on a detailed analysis of gold processing throughput as detailed in Section 16. The financial model is based on the following key assumptions:

- Currency base is the USD in real Q4-2104 terms.
- A discount rate of 5%.
- A flat gold price of USD 1,300/oz across all periods.
- Royalty is calculated as 3% of net revenue. The financial model does not account for the Republic of Liberia's retention of a free of charge equity interest in Bea's operations equal to 10% of its authorized issued and outstanding share capital without dilution (i.e. a 10% "carried interest").
- The financial model includes an estimated USD58m of historical capital expenditure (sunk costs) incurred prior to the start of project execution and mine construction.
- Cashflow forecasts are calculated on a quarterly basis.

All mining and processing tonnages and grade are as shown in Table 16-1 Gold recovery has been assumed to be 93% after a 3 month ramp up starting at 89% and in 1% increments per month.

### 22.1.2 Project Economics

A net present value (NPV) has been calculated for the expected cash flows from commencement of commercial production (i.e. from 1 July 2015 and excluding all initial Project capital costs reflected in Table 21-17) through the application of Discounted Cash Flow (DCF) techniques to pre-financing cash flows derived from the inputs and assumptions presented in this and previous sections of the report. All figures are presented in Q4 2014 real USD terms.

For the base case analysis a flat gold price of USD 1,300/oz has been used.

A government royalty of 3% of net revenue has been assumed. The financial model is reported on the basis of 100% of the Project, with no consideration of the free carried interest. The model assumes a corporation tax rate of 25% which is taken from the restated and amended Mineral Development Agreement.

A summary of cash flow modelling is presented below in Table 22-1 with the annualised pre-tax cash flow model shown in Table 22-3.

**Table 22-1: Cash Flow Modelling Summary**

| Description | Units | Project Totals/Averages |
|---|---|---|
| Recovered gold | koz | 858 |
| Mill processing life | Years | 8 |
| Net smelter revenue (after royalty) | USD m | 1,079 |
| Operating costs | USD m | 594 |
| Net operating cash flow | USD m | 486 |
| Initial capital costs | USD m | 172 |
| Net post-tax cash flow from start of production | USD m | 401 |
| Post-tax NPV (5%)* | USD m | 328 |
| Average cash cost per ounce | USD/oz | 692 |
| Internal Rate of Return (IRR)+ | (%) | 21% |
| Payback Period+ | (years) | 3.7 |

*present value of expected cash flows from commencement of commercial production before debt servicing and repayment

+IRR and Payback Period include all capital cost cash flow from the start of construction. 72% of capital cost has been incurred as at December 31, 2014.

As a sensitivity the economic assessment was repeated assuming a flat gold price of USD1,200/oz and at USD1,400/oz. This is compared with the initial analysis in Table 22-2.

**Table 22-2: Gold Price Sensitivity**

| Gold price | USD/oz | 1,200 flat | 1,300 flat | 1,400 flat |
|---|---|---|---|---|
| Gross revenue | USD m | 1,030 | 1,116 | 1,202 |
| Net smelter revenue (after royalty) | USD m | 996 | 1,079 | 1,163 |
| Net operating cash flow | USD m | 402 | 486 | 569 |
| Post-tax NPV (5%)* | USD m | 274 | 328 | 378 |

*present value of expected cash flows from commencement of commercial production before debt servicing and repayment

**Table 22-3: Annualized Pre-Tax Cash Flows**

| | Units | Totals | 2014 | 2015 | 2016 | 2017 | 2018 | 2019 | 2020 | 2021 | 2022 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore mined | Kt | 8,494 | 10 | 856 | 1,156 | 1,032 | 1,342 | 1,142 | 1,021 | 1,114 | 820 |
| Waste mined | Kt | 131,606 | 968 | 18,474 | 24,793 | 25,185 | 24,321 | 21,047 | 9,419 | 6,099 | 1,300 |
| Strip ratio | X | 15.5 | 96.8 | 21.6 | 21.4 | 24.4 | 18.1 | 18.4 | 9.2 | 5.5 | 1.6 |
| Ore processed | Kt | 8,494 | - | 545 | 1,245 | 1,140 | 1,140 | 1,140 | 1,140 | 1,140 | 1,004 |
| Grade | g/t | 3.4 | - | 3.8 | 3.1 | 3.3 | 3.5 | 4.2 | 3.4 | 3.8 | 2.3 |
| Gold contained | Koz | 924 | - | 66 | 123 | 119 | 127 | 154 | 123 | 138 | 73 |
| Gold recovered | Koz | 858 | - | 61 | 115 | 111 | 118 | 143 | 114 | 128 | 68 |
| | | | | | | | | | | | |
| **Revenue** | | | | | | | | | | | |
| | | | | | | | | | | | |
| Gold price | USD/ oz | 1,300 | - | 1,300 | 1,300 | 1,300 | 1,300 | 1,300 | 1,300 | 1,300 | 1,300 |
| Gross revenue | USD'000 | 1,115,805 | - | 78,910 | 149,238 | 144,367 | 152,969 | 186,325 | 148,519 | 166,915 | 88,562 |
| Freight & refining | USD'000 | (3,004) | - | (212) | (402) | (389) | (412) | (502) | (400) | (449) | (238) |
| Royalty | USD'000 | (33,384) | - | (2,361) | (4,465) | (4,319) | (4,577) | (5,575) | (4,444) | (4,994) | (2,650) |
| Net revenue | USD'000 | 1,079,417 | - | 76,337 | 144,371 | 139,659 | 147,981 | 180,248 | 143,676 | 161,472 | 85,673 |
| | | | | | | | | | | | |
| **Operating Costs** | | | | | | | | | | | |
| Total operating costs | USD'000 | (593,795) | - | (37,160) | (92,645) | (95,280) | (104,757) | (101,585) | (69,241) | (58,330) | (34,795) |
| | | | | | | | | | | | |
| **Capital costs** | | | | | | | | | | | |
| Sustaining and deferred capital costs | USD'000 | (32,691) | - | (2,963) | (5,821) | (5,121) | (5,121) | (5,121) | (4,769) | (1,889) | (1,889) |
| | | | | | | | | | | | |
| **Pre-tax cash flow** | **USD'000** | **452,931** | **-** | **36,214** | **45,906** | **39,258** | **38,103** | **73,542** | **69,666** | **101,253** | **48,990** |

### 22.1.3 Taxes and Royalties

Based on the base case of a gold price of USD1,300 per oz the government of Liberia will receive corporate tax revenues of USD53 million and gold royalties of USD 33 million over the LOM of the Project.

### 22.1.4 Project Sensitivities

Table 22-4 illustrates an analysis of the Project NPV sensitivity to variations in gold price, operating cost and capital cost estimates used in the base case.

**Table 22-4: Project Sensitivities**

| Sensitivity | NPV* 5% Discount Rate | Variance to Base Case |
|---|---|---|
| | USD m | % |
| **Gold Price** | | |
| +10% (USD1,430/oz) | 393 | +20 |
| -10% (USD1,170/oz) | 258 | -21 |
| **Operating Costs** | | |
| +10% | 289 | -12 |
| **Grade** | | |
| -10% | 259 | -21 |

*present value of expected cash flows from commencement of commercial production before debt servicing and repayment

The financial model sensitivities indicate that the Project is economically robust and when ranked shows that the Project is most sensitive to gold price or grade

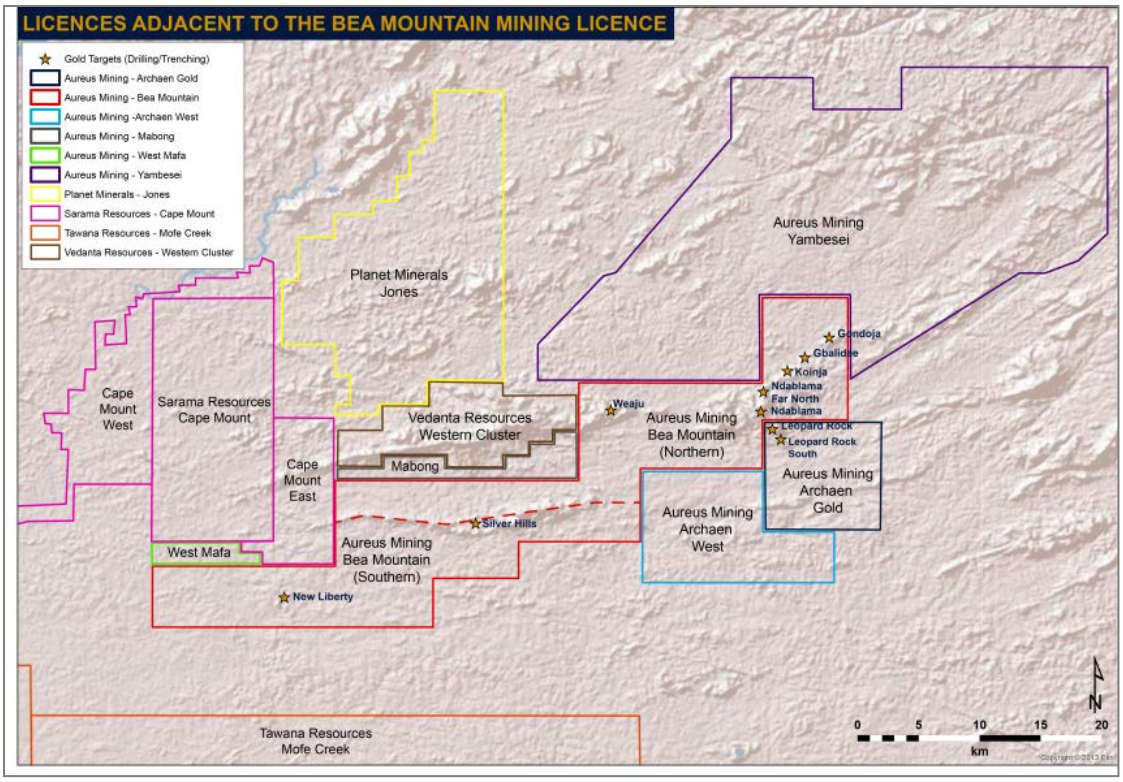
SRK has verified that the financial model inputs reflect accurately the technical and financial costs reported in the study.

SRK has reviewed the basis of the technical assumptions applied to the economic assessment, together with the operating and capital cost estimates and they are considered appropriate to support the definitive feasibility study.

# 23 ADJACENT PROPERTIES

## 23.1 Overview

The most recent Mineral Land Holding map update was published April 2011 by the Ministry of Lands Mines and Energy. Since the publication of this government data, Aureus has acquired an exploration licence known as Archaen Gold (89 km$^2$) from Archaen Gold Ltd, as announced on 21 September 2011. Additionally, as reported on 19 November 2013, Aureus has been granted four new exploration licences, contiguous to the Bea Mountain Mining licence by the Ministry of Lands, Mines and Energy. The four new exploration licences are referred to as Yambesei (759 km$^2$), Archaen West (112.6 km$^2$), Mabong (36.6 km$^2$) and West Mafa (15.6 km$^2$). In all cases, the company has 100% ownership, and these acquisitions bring the company's contiguous land holdings to an area of 1,470 km$^2$, this and various exploration / mining company websites are compiled in Figure 23-1.



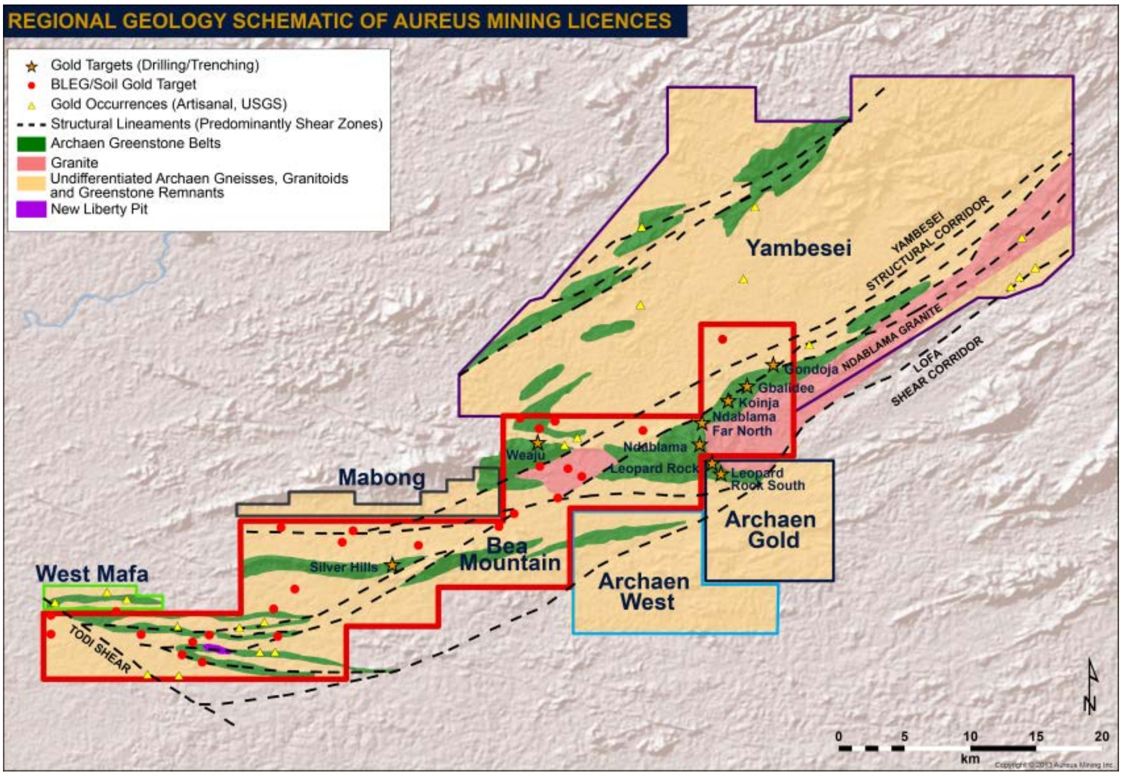

Source: *Aureus, 2013*

**Figure 23-1: Properties adjacent to the Bea-MDA Mountain mining licence**

For the purposes of this report, the Bea Mountain Mining licence (457 km$^2$) has been split along a ridge running east to west through the Silver Hills project area. This ridge separates the northern and southern portions of the Bea Mountain Mining Area. The New Liberty Project is located in the southern portion of the licence and is described in this Technical Report. Information relating to the Ndablama and Weaju Projects, situated in the Northern portion of the licence area is available within the technical report dated 1 December 2014, entitled "Nedablama and Weaju Gold Projects, Bea Mountain Mining Licence Northern Block, Liberia, West Africa", and available on SEDAR.

Aureus' licence portfolio hosts multiple greenstone belts and associated shear structures which to date have been the principal hosts to the gold mineralization systems discovered in Liberia. At the time of this report, and with the exception of the Archaean Gold licence, a desktop review of existing data and regional exploration activities has shown in excess of 50 gold occurrences and gold geochemical anomalies have been outlined on the Company's ground holdings. This is detailed in Figure 23-2. Gold mineralization is associated with the primary shear systems or in subordinate structures related to these major breaks.

A regional BLEG campaign has been carried out to delineate prospective zones with 277 samples collected to date. A soil sampling programme (800 m by 100 m) was also undertaken along the Yambesei structural corridor to check possible extension of the Gondoja gold corridor to the east, with 615 soil samples collected.




Source: *Aureus, 2013*

**Figure 23-2: Geological interpretation of Aureus mining licence package**

# 24 OTHER RELEVANT DATA AND INFORMATION

## 24.1 Project Implementation

The overall Project implementation schedule is based on the assumption that first gold will be produced in May 2015. The Project execution schedule reflects the work required, from detailed engineering at the planning stage, to the construction phase and then onto the commissioning and production phases.

The remaining major Project milestones include the following:

- Completion of construction activities;
- First gold pour in May 2015;
- Further plant optimisation and final commissioning during June and July 2015; and
- Steady state plant production from August 2015.

Aureus has appointed DRA Projects (Pty) Ltd as an independent engineering company to execute the Project on an Engineering, Procurement and Construction Management (EPCM) basis.

## 24.2 Execution Strategy and Owner's Team

### 24.2.1 Project Manager

The execution of the construction of the Project has been led by Aureus General Manager ("GM") of Construction and Operations. The EPCM contractor and its project manager report directly to the GM Construction and Operations.

### 24.2.2 Owner's Team

The Owner's Team has been created from a combination of Aureus' existing employees and recent new appointees. The specialist technical, operational and project execution skill sets required for the Owner's Team include:

- Project engineering and mining engineering – monitoring and approving EPCM contractor's design and engineering work;
- Electrical, construction, earthworks and procurement – monitoring and approving the EPCM contractor's work;
- Geology, Mining engineering and Survey;
- Environmental and health and safety;
- Community relations;
- Government liaisons for permitting and legislative matters; and
- Back office support, including, but not limited to, accounting and legal matters.

## 24.3 EPCM

### 24.3.1 Engineering

The EPCM Contractor is responsible for the following engineering aspects of the Project execution:

- Project specifications;
- Design criteria;
- Plant and infrastructure designs;
- Data sheets;
- Drawings;
- Technical reports; and
- Engineering schedules.

### 24.3.2 Procurement

The EPCM Contractor is also responsible for the following procurement aspects of the Project execution:

- Generation of enquiry documents;
- Obtaining and control of quotations;

- Preparation of commercial adjudications;
- Bid clarification and negotiations;
- Preparation of orders and contract documentation and any modifications;
- Expediting;
- Delivery;
- Quality assurance and inspections;
- Freight forwarding control; and
- Preparation of control documents and schedules.

### 24.3.3 Construction Management

The EPCM Contractor is responsible for the following construction aspects of the Project execution:

- Performing required scope of work in accordance with timetable and budget; and
- Adherence to requisite quality and workmanship standards, including occupational health, safety and environmental practices.

## 24.4 Commissioning

The commissioning approach will be based on four basic phases as described in a typical enquiry document:

- Pre-commissioning
- Cold commissioning
- Hot commissioning
- Training and operational assistance period.

## 24.5 Project Schedule

The Project schedule is summarised in Table 24-1. In SRK's opinion, while there remains a risk that construction may take longer than planned to complete, particularly given that the Ebola situation has not yet been fully resolved, and while there is further work to be done, the Project currently remains on course to start producing gold in May 2015.

## Table 24-1: Project Schedule

| New Liberty Gold Commissioning Schedule | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2012 | | | 2013 | | | | | | | | | | | | 2014 | | | | | | | | | | | | 2015 | | | | | | | |
| | OCT | NOV | DEC | JAN | FEB | MAR | APR | MAY | JUN | JUL | AUG | SEP | OCT | NOV | DEC | JAN | FEB | MAR | APR | MAY | JUN | JUL | AUG | SEP | OCT | NOV | DEC | JAN | FEB | MAR | APR | MAY | JUN | JUL | AUG |
| MOBILISE DRA TEAM | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| PROJECT OPTIMISATIOPN | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| DESIGN & ENGINEERING | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| PROCUREMENT | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| FABRICATE/MANUFACTURE | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| EARTHWORKS & CIVILS (Incl TSF and MCDC) | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| SMPP & EI | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| PRE STRIP MINING | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| COMMISSIONING | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| FIRST GOLD POUR | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| HANDOVER | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| GOLD PRODUCTION | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |

# 25 INTERPRETATION AND CONCLUSIONS

## 25.1 Mineral Resources and Reserves

The exploration work undertaken by Aureus and previous owners on the Project to date has delineated a significant gold deposit.

Notably, the most up to date Mineral Resource reported according to CIM Standards comprises 0.65Mt with a mean grade of 4.77g/t Au which has been classed as Measured, 9.15Mt with a mean grade of 3.55g/t Au which has been classed as Indicated and 5.73Mt with a mean grade of 3.2 g/t Au which has been classed as Inferred.

In addition, the most recent technical work commented upon in this report, notably the new mining plan and updated processing assumptions as well as updated estimates of capital and operating costs, have confirmed the technical feasibility and economic viability of the Project and supported the Mineral Reserve statement which according to CIM Standards comprises a Proven Mineral Reserve of 0.7Mt with a mean grade of 4.4 g/t Au and a Probable Mineral Reserve of 7.8Mt with a mean grade of 3.3 g/t Au.

## 25.2 Mining Plan

The Mineral Reserve is contained within an open pit which will be mined using conventional drill-and-blast, load and haul mining techniques.

A new mine plan has recently been developed by Aureus to extract the Mineral Reserve which compensates for the delay in the commencement of processing operations and improves the Project's economics through the development of two starter pits (maximising operational face lengths and reduced haulage distances for waste).

SRK considers the updated mining plan to be a robust plan in terms of the production assumed and to appropriately reflect work completed by Aureus. It incorporates additional information obtained and compensates for the current delayed status of construction, which has resulted from factors largely beyond the Company's control.

The revised mine plan assumes more gold is produced than previously reported plan over the early periods. This is achieved through the mining of more ore tonnes at a higher gold grade and processing more tonnage at a higher grade by selectively feeding from ROM stockpiles, thereby producing more ounces earlier in the mines life. This shortens the LOM by approximately four months and improves the Projects economics.

In addition the results of the recently completed close spaced drilling have suggested that if anything the estimates of tonnes and grade used as the basis of the plan may be slightly conservative.

It is this updated mining plan that forms the basis of the economic analysis presented in this Technical Report.

## 25.3 Mineral Processing and Metallurgical Testwork

Subsequent to the completion of the Feasibility Study metallurgical test work in 2013, Aureus employed DRA to scope and manage a test work programme investigating the mobility of Arsenic during the cyanide leach and Detox processes within the New Liberty Process Plant. Under the governance and supervision of Digby Wells the Arsenic mobility on the Tailings Storage Facility (TSF) was also investigated and the geochemical test work was scoped by them. DRA facilitated the metallurgical preparation of these samples for testing at ALS Environmental.

This test work confirmed that a process which includes an $SO_2$/Air detox step in combination with 2.5kg/t ferric chloride addition on CIL tailings results in arsenic leaching with subsequent precipitation of a stable arsenate compound. A solids sample containing 1200ppm arsenic was treated using the optimum conditions (JR 1256) and has achieved an arsenic in solution value of 0.005ppm after 23 weeks in a kinetic column test.

Based on the kinetic column test results for test JR1256, the New Liberty process plant design was updated to allow for 1000m$^3$ tank in which cyanide destruction and arsenic leaching will take place. The additional leach tank is the same volume as the CIL tanks. In addition three 260m$^3$ tanks were added to treat the detox/arsenic leach product stream. These tanks allow for additional ferric sulphate addition and pH correction of tailings.

A large portion of the arsenic optimisation test work was conducted using ferric chloride reagent as a source of ferric ion. In test JR1256 the option of using ferric sulphate as opposed to ferric chloride was investigated and no difference in metallurgical response has been noted between the two reagents after 23 weeks in a kinetic column test. The plant operating cost estimate has been based on the use of ferric sulphate as a source of ferric ion.

It is these updated metallurgical assumptions that inform the economic analysis presented in this Technical Report.

## 25.4 Project Infrastructure

The construction of the Project infrastructure is now well advanced.

The diversion dams and cuttings for the Marvoe Creek Diversion are essentially complete.

The wall of the TSF is 75% complete and work is progressing well on the penstock line, with current forecasts to completion in line with the planned commissioning dates.

The civil and earthworks for the process plant are complete, and steel, plate work and piping well advanced, with Electrical and instrumentation following behind on schedule for planned commissioning dates.

## 25.5 Environmental Management

An Environmental and Social Impact Assessment (ESIA) was undertaken from Q4 2010 to Q2 2012 to investigate the local environmental and social situation existing prior to the development of the Project, and to determine the likely positive and negative impacts of associated with its development.

The ESIA was completed using accepted international standards (notably those of the World Bank and World Health Organisation), best practice principles and techniques, and the Liberian EPA ESIA Procedural Guidelines (2006), and the Environmental Permit for the Project was granted by the Liberian Environmental Protection Agency in October 2012.

Subsequent to the completion of the ESIA and the granting of the Projects Environmental Permit, Aureus embarked upon an optimization process which resulted in the relocation of various mining associated infrastructure. The revised infrastructure locations remain within the area permitted for mining and as a result of this, Digby Wells were engaged to revise and update the ESIA. Following a review of these documents by the IFC prior to their investment in the Company in 2014, an addendum to the updated ESIA was also produced and submitted to the EPA during March 2014.

The results of the impact assessment indicate that the management and mitigation of environmental and social impacts associated with the project are amenable to standard technical solutions. No issue has been identified that presents a technical challenge beyond that which is regularly encountered and resolved by comparable mining operations elsewhere in Africa.

Environmental management of the Project will be an evolving process over the life of the mine. In particular, the environmental management and mitigation measures and the monitoring programme outlined in this EIS will be updated annually for continual improvement to occur and for management practices to remain current and aligned with Liberian legislation and industry good practice.

In July 2014, following the investment in the Company by the IFC, a Critical Habitat Assessment and Terrestrial Ecological Biodiversity survey was carried out by a team of international specialists at the New Liberty Project site. The report concluded that the overall study area comprises Natural Vegetation, with a smaller percentage of Modified Habitat, most of which was restricted to the mine footprint area. In total, 4.9% of the total Natural Habitat will be directly lost due to mine activities.

## 25.6 Resettlement Action Plan

Development of the Project has required the resettlement of two relatively small villages, Kinjor and Larjor, which are located within the proposed mine pit.

The Project Resettlement Action Plan (RAP) to address the above resettlement impacts was granted by the Liberian Environmental Protection Agency in March 2013, and the relocation of the local communities was completed in September 2014 to a temporary area within the RAP site. Construction at the RAP is nearing completion for the permanent houses within the village, and once complete, these will be handed over to the local community.

At the time of writing this report, the area of the old Kinjor village had been cleared, allowing for the commencement of mining operations and grade control drilling.

## 25.7 Community Development Plan

The CDP was completed in December 2012 and was approved by the Liberian EPA in January 2013.

Based on the development opportunities assessed, a number of potential CDP projects were identified, which include vegetable production, cashew nut production and coconut production as well as other further agricultural activities. A number of these initiatives have been started on site, including an agricultural cooperative producing vegetables and a woodworking and brickmaking cooperative producing construction materials. Where feasible the CDP will be expanded to incorporate the community development aspects of the Bea- MDA.

The development and operation of the project will have both positive and negative impacts on the socio-economic structure of the project area and its environments. The development of the project will bring much needed investment and development opportunities with consequent impacts on the employment and the affected communities.

## 25.8 Concluding Remarks

There remain some risks to the Project. Notably, given that the Ebola situation has not yet been fully resolved, there may be some further construction delays which could impact on revenues and capital costs. There is also a risk that some the assumed reduction in mining costs may not be achieved.

Notwithstanding this, SRK has concluded that the Project is both technically feasible and economically viable and, while there remains work to be done, the Project currently remains on course to start producing gold in May 2015.

In SRK's opinion, the strong and experienced team on site has enabled Aureus to negotiate what has been a very difficult period and has minimised the impact of factors which have been outside of the Company's control and that assuming this team is retained, the risks identified above should be minimised.

# 26 RECOMMENDATIONS

The principal conclusion arising from this review of the Project is that the construction of the Project should continue and that Aureus should focus on this in order to ensure that the Project is developed as envisaged and in the planned timeframe.

Given the above, SRK is not making any further recommendations for more technical work at this stage.

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# CERTIFICATES AND LETTERS OF CONSENT

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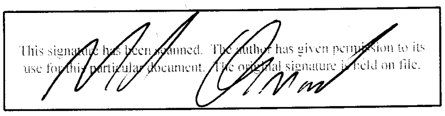
To accompany the technical report entitled “New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan” (the “Technical Report”) for Aureus Mining Inc. with the effective date March 25th 2015.

I, Dr Mike Armitage, BSc, MIMMM, CEng, residing at Maesaeson House, Peterston-Super-Ely, Vale of Glamorgan CF5 6NE, Wales, UK, do hereby certify that.

1. I am Group Chairman and Corporate Consultant (Mining Geology) with the firm of SRK Consulting (UK) Ltd (“SRK”) with an office at 5th Floor, Churchill House, Churchill Way, Cardiff, UK;
2. I am a graduate from the University of Wales, College Cardiff with an BSc. Honours Degree in Mineral Exploitation, (Specializing in Mining Geology) awarded in 1983 and also have a PhD from Bristol University in Structural and Resource Geology awarded in 1993. I have practised my profession continuously since 1983.
3. I am a Professional Member of the Institute of Materials, Minerals and Mining and I am a Chartered Engineer and a Fellow of the Geological Society of London.
4. I personally visited the project area between November 20th and November 23rd 2012 during which time I visited all localities and exposures within the licence area relevant to the most up to date Mineral Resource Estimate and made first hand observation of the drill core and sampling facilities.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of National Instrument 43-101;
6. I am one of the authors of this Technical Report and take responsibility for Sections 1 to 12, 14 to 16, 19 and 21 to 27;
7. As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

8. I have had no prior involvement with the subject property;
9. I have read National Instrument 43-101 and confirm that this Technical Report has been prepared in compliance therewith;
10. SRK was retained by Aureus Mining Inc. to prepare the Technical Report. The Technical Report is based on a site visit, a review of project files and discussions with Aureus Mining Inc. personnel;
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the New Liberty project or securities of Aureus Mining Inc.;
12. That, as of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public of extracts from the Technical Report.
14. I confirm that I have read the news release dated February 9th 2015 in which the findings of the Technical Report have been disclosed publically and have no reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the press release dated February 9th 2015 contains any misrepresentations of the information contained in the Technical Report.




Dr Mike Armitage, *FGS, CGeol, MIMMM, CEng*
Group Chairman & Corporate Consultant
(Mining Geology), SRK (UK) Ltd.
Cardiff, UK, March 25th 2015

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.co.uk
Tel: + 44 (0) 2920 348 150
Fax: + 44 (0) 2920 348 199

Project number: UK4936,
Cardiff, Wales, 25th March 2015

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Prince Edward Island Securities Office
Government of Newfoundland and Labrador
Government of Yukon
Government of Northwest Territories
Government of Nunavut

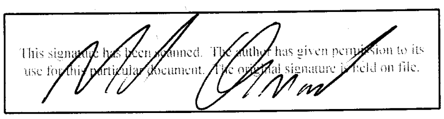
**CONSENT OF QUALIFIED PERSON**

I, Dr Mike Armitage, have been responsible for preparing or supervising the preparation of Sections 1 to 12, 14 to 16, 19 and 21 to 27of the technical report entitled “New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan” (the “Technical Report”) and dated 25th March 2015 for Aureus Mining Inc. (the “Company”).

I further consent to the public filing of the Technical Report and to extracts from, or a summary of, the Technical Report in the news release of the Company dated 9th February 2015 (the “Release”).

I also certify that I have read the Release that the Technical Report supports and, that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated this 25th day of March 2015.




**Dr Mike Armitage, *BSc, MIMMM, FGS, CEng***
**Group Chairman & Corporate Consultant**
**(Mining Geology), SRK (UK) Ltd.**

# CERTIFICATE OF QUALIFIED PERSON

Robin Mark Welsh
DRA Mineral Projects
Suite 502
80 Strand Street
Cape Town 8000
South Africa

Telephone: +27 21 285 0119
Email: rob.welsh@DRAglobal.com

This certificate applies to the technical report titled "New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan" (the "Technical Report") for Aureus Mining Inc. with the effective date 25 March 2015.

I, Robin Mark Welsh, do hereby certify that:

I am a Senior Project Manager for DRA Mineral Projects of Suite 502, 80 Strand Street, Cape Town, 8000, South Africa.

2. I graduated with a BSc Electrical Engineering from the University of Natal, South Africa in 1990.

3. I have been a registered Professional Engineer with the Engineering Council of South Africa since 1999 (Pr.Eng), (Registration number 990118) and I have been a member of the South African Institute of Electrical Engineers since 1993.

4. I have practiced continuously as an Electrical Engineer and Project Manager since 1990, and have been involved in mineral processing and mining projects for a period of 25 years.

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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 18 and parts of Section 25 of the Technical Report.

7. I visited the property on 21–22 May 2012, 21–25 January 2013, 5–8 March 2013, 14–17 May 2013, 11–14 June 2013, 10–15 March 2014 and 7-12 June 2014.

8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as a Senior Project Manager for the preparation of the work which forms part of the Technical Report.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1 and Section 18 and parts of Section 25 of the Technical Report have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Section 18 and parts of Section 25 of the Technical Report and its supporting documentation contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 25 March 2015

Robin Mark Welsh
BSc Engineering, Pr. Eng., MSAIEE
Senior Project Manager
DRA Mineral Projects

March 25 2015

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Prince Edward Island Securities Office
Government of Newfoundland and Labrador
Government of Yukon
Government of Northwest Territories
Government of Nunavut

# CONSENT OF QUALIFIED PERSON

Dear Sirs/Mesdames,

I, Robin Mark Welsh, have been responsible for preparing or supervising the preparation of Section 18 and parts of Section 25 of the technical report entitled "New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan" and dated 25 March 2015 (the "Technical Report") for Aureus Mining Inc. (the "Company").

I consent to the public filing of the Technical Report and to extracts from, or a summary of, the Technical Report in the news release of the Company dated 9 February 2015 (the "Release").

I also certify that I have read the Release that the Technical Report supports and, that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Signed the 25 March 2015

Robin Mark Welsh
BSc Engineering, Pr. Eng., MSAIEE
Senior Project Manager
DRA Mineral Projects

# CERTIFICATE OF QUALIFIED PERSON

Glenn Bezuidenhout
DRA Mineral Projects
3 Inyanga Close
Sunninghill 2157
Johannesburg
South Africa

Telephone: +27 11 202 8686
Email: glenn.bezuidenhout@DRAglobal.com

This certificate applies to the technical report titled "New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan" (the "Technical Report") for Aureus Mining Inc. with the effective date 25 March 2015.

I, Glenn Bezuidenhout, do hereby certify that:

1. I am a Process Director for DRA Mineral Projects of 3 Inyanga Close, Sunninghill 2157, Johannesburg, South Africa.

2. I graduated with a National Diploma in Extractive Metallurgy from the Witwatersrand Technicon South Africa in 1979.

3. I have been a Fellow of the Southern African Institute of Mining and Metallurgy since 2012 (FSAIMM) (Membership number 705704).

4. I have practiced continuously as a Process Engineer since 1992, and have been involved in mineral processing and mining projects for a period of 23 years.

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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 13, Section 17 and parts of Section 25 of the Technical Report.

7. I visited the property on 21-22 November 2012.

8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as a Process Director for the preparation of the work which forms part of the Technical Report.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1 and Section 13, Section 17 and parts of Section 25 of the Technical Report have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Section 13, Section 17 and parts of Section 25 of the Technical Report and its supporting documentation contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 25 March 2015

Glenn Bezuidenhout
NDT Ex. Met, FSAIMM
Process Director
DRA Mineral Projects

March 25 2015

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Prince Edward Island Securities Office
Government of Newfoundland and Labrador
Government of Yukon
Government of Northwest Territories
Government of Nunavut

## CONSENT OF QUALIFIED PERSON

Dear Sirs/Mesdames,

I, Glenn Bezuidenhout, have been responsible for preparing or supervising the preparation of Section 13, Section 17 and parts of Section 25 of the technical report entitled "New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan" and dated 25 March 2015 (the "Technical Report") for Aureus Mining Inc. (the "Company").

I consent to the public filing of the Technical Report and to extracts from, or a summary of, the Technical Report in the news release of the Company dated 9 February 2015 (the "Release").

I also certify that I have read the Release that the Technical Report supports and, that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Signed the 25 March 2015

Glenn Bezuidenhout
NDT Ex. Met, FSAIMM
Process Director
DRA Mineral Projects

# CERTIFICATE OF QUALIFIED PERSON

Graham Trusler
Digby Wells Environmental
Fern Isle, Section 10
359 Pretoria Avenue
Private Bag X10046
Randburg 2125
South Africa

Telephone: +27 11 789 9495
Email: graham.trusler@digbywells.com

This certificate applies to the technical report titled "New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan" (the "Technical Report") for Aureus Mining Inc. with the effective date 25 March 2015.

I, Graham Trusler, do hereby certify that:

1. I am the Chief Executive Officer of Digby Wells Environmental, Fern Isle, Section 10, 359 Pretoria Avenue, Randburg 2125, South Africa.

2. I graduated with a B.Sc Chemical Engineering (University of Natal, 1986), M.Sc Engineering (University of Natal, 1988) and B. Comm (University of South Africa, 1994). I completed the Integrated Environmental Management course (I.E.M.) (University of Cape Town, 1993).

3. I am a Professional Engineer registered with the Engineering Council of South Africa, (Pr.Eng) (Registration Number 920088).

4. I have practiced my profession continuously since 1990 and have been involved in Environmental and Social Impact Assessments for 25 years.

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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 20 of the Technical Report.

7. I visited the property on 20-21 March 2013, 13-17 May 2013 and 10-14 February 2014.

8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as Chief Executive Officer of Digby Wells Environmental on technical matters, the results of which form part of the Technical Report.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and Section 20 of the Technical Report has been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Section 20 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 25 March 2015

Graham Trusler
M.Sc (Eng.), Pr.Eng.
Chief Executive Officer
Digby Wells Environmental

DIGBY WELLS
ENVIRONMENTAL

March 25, 2015

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Prince Edward Island Securities Office
Government of Newfoundland and Labrador
Government of Yukon
Government of Northwest Territories
Government of Nunavut

**CONSENT OF QUALIFIED PERSON**

Dear Sirs / Mesdames:

I, Graham Trusler, have been responsible for preparing or supervising the preparation of Section 20 of the technical report entitled "New Liberty Gold Project, Bea Mountain Mining Licence Southern Block, Liberia, West Africa, Definitive Project Plan" and dated 25 March 2015 (the "Technical Report") for Aureus Mining Inc. (the "Company").

I consent to the public filing of the Technical Report and to extracts from, or a summary of, the Technical Report in the news release of the Company dated 9 February 2015 (the "Release").

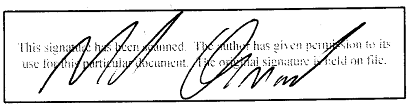
I also certify that I have read the Release that the Technical Report supports and, that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Signed the 25 March 2015

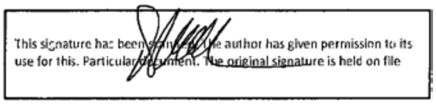
Graham Trusler
M.Sc (Eng.),Pr.Eng.
Chief Executive Officer
Digby Wells Environmental

**SIGNATURE PAGE**

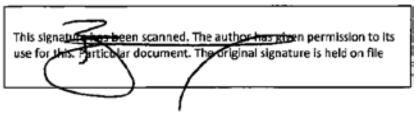
Report Effective Date: 25th March 2015



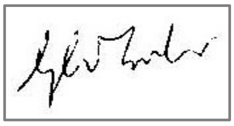

Dr Mike Armitage, *BSc, MIMMM, FGS, CEng*
Corporate Consultant & Chairman (Mining Geology),
SRK Consulting (UK) Limited
25th March 2015




Robin Mark Welsh
BSs Engineering, Pr. Eng, MSAIEE
Senior Project Manager
DRA Mineral Projects
25th March 2015




Glenn Bezuidenhout
NDT Ex. Met, FSAIMM
Process Director
DRA Mineral Projects
25th March 2015



Graham Trusler
M.Sc (Eng.), Pr. Eng.
Chief Executive Officer
Digby Wells Environmental
25th March 2015