NEW LIBERTY GOLD MINE, BEA MOUNTAIN MINING LICENCE SOUTHERN BLOCK, LIBERIA, WEST AFRICA

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

Prepared For

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UK4936

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NEW LIBERTY GOLD MINE, BEA MOUNTAIN MINING LICENCE SOUTHERN BLOCK, LIBERIA, WEST AFRICA

1 SUMMARY

1.1 Introduction

This Technical Report on the New Liberty Gold Mine (New Liberty, NLGM 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 Avesoro Resources Inc. (Avesoro). Bea Mountain Mining Corporation (BMMC or the Company), which is a 100% owned subsidiary of Avesoro, has a 100% interest in the Bea-MDA.

The Project is an operating gold mine. Construction activities started in late 2012/early 2013, pre-stripping mining activities commenced in October 2014 and the first gold pour occurred in May 2015.

This report has an effective date of 01 November 2017, has been compiled by SRK, 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 is based on the Life of Mine (LoM) plan currently in place.

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 BMMC 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. BMMC 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 (the 2013 Technical Report).

Since this time, the Company has continued to conduct further evaluation work at New Liberty, including grade control drilling to produce a better geological understanding of the deposit at a mining scale.

Prior to issue of this report, the most recent Technical Report produced on the Project was issued on 25 March 2015 entitled ‘New Liberty Gold Project, Bea Mountain Mining Licence, Southern Block, Liberia, West Africa, Definitive Project Plan’ (the 2015 Technical Report) which reflected the status of the Project at that time.

1.3 Geology

The mineralisation targeted by BMMC 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 hosts 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 10m to 15m, while typically extending hundreds of metres both along strike and down dip.

1.4 Mineral Resources

The most up to date Mineral Resource estimate for the Project has been derived by SRK and is presented in Table 1-1 below. This is reported with an effective date of 31 July 2017 and has taken into account mining depletion up to this point. The estimate is reported according to CIM Standards and at a 0.8g/t Au cut-off for open pit material and 2.0g/t Au for underground material. The Resource Statement has been split to show both remaining in-situ open pit and underground resources and also ore stockpiles as at 31 July 2017. The ore stockpiles have been classified as Indicated Resources as while the stockpiles are surveyed and reconciled with truck counts for tonnage, the material is not sampled (subsequent to excavation) and the grade is based on theoretical block model grades.

The independent qualified person, as defined by Canadian Securities Administrators National Instrument 43-101, for this mineral resource estimate is Dr Mike Armitage BSc, MIMMM, C.Eng, C.Geol, SRK Consulting (UK) Limited.

Table 1-1: SRK Mineral Resource Statement as at 31 July 2017 for the New Liberty Deposit prepared in accordance with the CIM Code

Category	Cut-off	Tonnes Mt	Au Grade g/t	Au Koz
In-Situ				
Measured	0.8 g/t (OP)	0.1	3.6	15
Indicated	0.8 g/t (OP)	8.5	3.3	890
	2.0 g/t (UG)	0.6	3.3	65
Measured and Indicated	0.8 g/t (OP)	8.6	3.3	905
	2.0 g/t (UG)	0.6	3.3	65
Inferred	0.8 g/t (OP)	3.6	2.8	325
	2.0 g/t (UG)	2.8	3.3	295
Sub-total Measured		0.1	3.6	15
Sub-total Indicated		9.1	3.3	955
Sub-total Measured and Indicated		9.2	3.3	970
Sub-total Inferred		6.4	3.0	620
Stockpiles				
Indicated	Oxide and Fresh Ore	0.2	1.5	10
Indicated	Sub-Grade Ore	0.2	0.8	5
Sub-total Indicated		0.4	1.1	15
Total				
Total Measured		0.1	3.6	15
Total Indicated		9.5	3.2	970
Total Measured and Indicated		9.6	3.2	985
Total Inferred		6.4	3.0	620

1. The marginal cut-off grade used for resource reporting is 0.8g/t Au for Open Pit and 2.0g/t Au for Underground Mining.
2. All figures are rounded to reflect the relative accuracy of the estimate.
3. Mineral Resources are report inclusive of those converted to Mineral Reserves
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

1.5 Mineral Reserves

The Mineral Reserve statement for the NLGM project is presented in Table 1-2. Dr Armitage is also the independent qualified person, as defined by Canadian Securities Administrators National Instrument 43-101, for this Mineral Reserve estimate.

The Project base case economic analysis presented in Section 22 shows that the NLGM project life-of-mine plan founded on the Mineral Reserve Estimate in Table 1-2 provides a positive net present value of the net cash flow and a positive rate of return, confirming that the Mineral Reserves are economically viable and that economic extraction can be justified.

Table 1-2: NLGM Mineral Reserve Statement, Effective 31 July 2017

	Quantity	Au Grade	Au Contained
Category	**(Mt)**	**(g/t)**	**(koz)**
Proven	**0.2**	**3.03**	**15**
In-Pit	0.2	3.03	15
Probable	**7.2**	**3.03**	**702**
In-Pit	7.0	3.09	690
Stockpiles	0.2	1.40	11
Total Proven & Probable	**7.4**	**3.03**	**717**

Notes:
1. Mineral Reserves are included in the Mineral Resource Estimate dated 31 July 2017.
2. Mineral Reserves are reported at a cut-off grade of 0.85g/t Au within an engineered pit design. The cut-off grade is considered appropriate for a selling price of USD1,300/oz, processing cost of USD20/t, G&A cost of USD7/t, royalty of 3%, selling costs of USD3.5/oz and processing recovery averaging 91.2%.
3. Includes ore loss and dilution as reported from a regularised block model at 5 m x 2.5 m x 5 m, which has an average ore loss and dilution of 3.3% and 13.5%, respectively.

1.6 Mining Plan

SRK has developed a life of mine plan to estimate ore loss and dilution, pit optimisation, mine design, mine schedule, equipment and labour requirements and capital and operating costs. The findings of the study are summarised below:

- The updated mine designs based on the USD1,300/oz optimised shell result in 7.1 Mt of RoM at 3.08 g/t Au with 117.5 Mt of waste at a cut-off of 0.85 g/t Au.
- Average ore loss and dilution values are 3.3% and 13.5%, respectively within the pit design. Significant improvements are expected given the introduction of a new grade control programme to reduce ore loss and dilution from the current levels.
- The mine schedule produces 1.64 Mtpa of mill feed, totalling 7.4 Mt at an average grade of 3.03 g/t Au. The average strip ratio is 16.5 with 117.5 Mt of waste. Total material movement will average 3,905 kt/month in 2018 (totalling 46.9 Mt).
- The mine schedule is aggressive with up to 8 benches mined per year. Mine production quantities will need to triple by January 2018 and quadruple by March 2018 from current production levels.
- It is predicted there will be a number of periods when there is insufficient RoM Fresh material available on the stockpile to mitigate any shortfalls. Should any shortfalls arise, additional material will can only be sourced from the RoM Oxide stockpile which has lower grades and recovery.
- One 12 m^3 backhoe and up to six 6 m^3 backhoes will continue to be used with 90 t haul trucks supported by 40 t ADTs. Up to 16 90 t haul trucks will need to be leased from February 2018 to support the mine plan.
- Significant improvements in availability and productivity of the excavators and trucks is required to meet the mine plan. The availabilities and productivities of the excavators and trucks should continue to be monitored to ensure the increases expected are realised. The mine plan will be significantly impacted should these improvements not be achieved.
- Significant cost savings are expected from the historical 2017 costs compared to the forecast. Regular monitoring of the mining costs in comparison to the forecast will be important to ensure the improvements are realised.
- A maximum of 892 personnel are required at peak material movement (2018), with 714 in mine operations, 157 in mine maintenance and 21 in technical services.

1.7 Mineral Processing

The process plant was commissioned during 2015 and initial operation was problematical for a variety of reasons. These issues can be summarised as availability of planned ore, oversize feed to the ball mill resulting in inefficient grinding and excessive stone discharge from the ball mill trommel, poor grinding ball quality, ball mill liner and grate material problems, under-utilisation of the regrind mill, excessive wear in the grinding circuit, gravity circuit feed screen capacity problems, lower than expected gravity circuit gold extraction due to insufficient gravity concentrator capacity, oxygen plant operational problems, oxygen sparging issues in pre-oxidation and CIL tanks, poor CIL leach extraction, poor carbon management in CIL circuit, cyanide detoxification circuit performance issues and availability of reagents and maintenance spares.

A number of plant modifications have been implemented to address these issues together with further changes planned for late 2017. These changes should reduce plant downtime, enhance plant throughput and improve plant performance to the expected levels in the feasibility study.

The main plant modifications installed or planned are:

- Better management of the ROM stockpiles and controlled blending of oxide ore to a maximum of 10% by mass resulting in more consistent feed, in terms of gold grade and ore characteristics, to the plant;
- Introduction of a tertiary crusher to reduce the ball mill feed size to nominally 100% minus 12 mm, 80% minus 8 mm. The nominal capacity of the modified crushing circuit will be 280 tph, operating 7 days per week and up to 18 hours per day;
- Installation of a new mill liner and grate system from an established supplier;
- Use of higher specification, smaller diameter grinding balls;
- Improvement in the overall power utilisation of the ball mill to allow increased mill throughput up to 200 tph whilst achieving the required grind size of 80% passing 50µm;
- Installation of all metal cyclone feed pumps to improve pump wear issues;
- Installation of a larger gravity circuit feed screen;
- Installation of a second gravity concentrator;
- Recommissioning of the regrind Vertimill® to increase the available grinding circuit power to allow higher throughputs whilst still achieving a CIL feed of 80% passing 50µm;
- Installation of two additional packaged oxygen plants to operate in parallel with the existing units;
- Increased gravity gold recovery and better control of grinding circuit performance to improve overall CIL gold extraction in the originally installed CIL tankage;
- Improved carbon management in the CIL circuit to improve gold adsorption efficiency and reduce soluble gold losses to tailings;
- Improved reagent addition systems to the cyanide detoxification and arsenic precipitation circuits;
- An increased number of operators and targeted cyanide detoxification and arsenic precipitation circuit performance management to maintain more consistent operation and acceptable tailings discharge levels of CN_{WAD} and soluble arsenic to the TSF; and
- Better management and improved availability of operating spares for the plant, increasing overall circuit utilisation.

These plant modifications should result in improved plant operating hours and plant metallurgical performance and BMMC is assuming the following plant parameters going forward:

- Throughput: c.1.7 Mtpa, 200 tph at 80% passing 50µm.
- Plant operating time: 93% of total time.
- Gold recovery will be dependent on feed grade and is forecast to vary from 89% at a feed grade of 2.0g/t up to 93% at feed grades of 4.0g/t and higher.

1.8 Infrastructure

The construction of the Project infrastructure is now essentially complete and the infrastructure is adequate to support the ongoing operations at the Project.

The diversion dams and cutting for the Marvoe Creek Diversion are complete and are appropriately diverting the surface water from the watercourse away from the open pit and infrastructure.

1.9 Tailings Storage Facility

The current Tailings Storage Facility (TSF) arrangement has been in operation since July 2015. As of the beginning of August 2017, the TSF has been operated as a self-raising facility, in which deposited tailings material will be reworked to form the main embankment itself.

The configuration of TSF was significantly altered during 2016. This was required due to periodic uncontrolled release of supernatant to the environment which did meet compliance limits (between December 2016 and June 2017). A temporary TSF configuration was constructed to ensure that discharge of excess supernatant to the environment met acceptable discharge limits. This involved segregation of the TSF into a series of compartments or cells, designed to promote a tortuous flow path for supernatant before discharge via the penstock to environment. This, combined with plant modifications, has ensured that discharge water quality has improved and is reported to now be within acceptable limits.

NewFields was commissioned by BMMC during October 2016 to prepare an alternative TSF design, which would allow safe storage of water on the facility and controlled release of supernatant to the environment. This new design involves conversion of the TSF to a water retaining, downstream raised facility. In addition, a water retaining dam is to be constructed to the east of the TSF, which will divert inflows of fresh water from the upstream catchment during storm events. This fresh water will be routed via the existing penstock arrangement and safely discharged downstream.

Overall, SRK considers the design of proposed TSF modifications to be a workable solution, assuming that the critical structures can be constructed timeously with the tailings rate of rise in the current facility.

1.10 Environmental Studies, Permitting and Social or Community Impact

NLGM is situated in the north-western portion of Liberia within the Gola Konneh District of Grand Cape Mount County. The climate is equatorial, the average annual rainfall is in the order of 3,400 mm and most rain falls between May and November.

Dense tall rainforest surrounds the mine site. Prior to mining, the mine site had been somewhat disturbed by past artisanal mining, prospecting, logging and bush meat hunting.

The area around the mine is sparsely populated. Two settlements, Kinjor and Larjor, comprising 325 households, have had to be relocated to make way for mining. The resettlement process is nearing completion and the affected households are in temporary accommodation. There are three settlements downstream of the mine, approximately 5 km, 11 km and 12 km downstream of the mine site and TSF.

The livelihoods of people living in Kinjor and Larjor were largely based on artisanal mining. The livelihoods of other villages in the area around the mine are largely based on subsistence agriculture and fishing from streams and rivers.

BMMC has the primary agreements and approvals required to operate, which include a Mineral Development Agreement (MDA), a mining licence, an environmental permit and a discharge permit. BMMC also has a number of secondary approvals and officially approved environment and social management plans. There are hundreds of compliance obligations in the approval documents and management plans. BMMC recognises that it needs to review these and agree revisions to unrealistic or poorly worded obligations.

There are some elements of an environmental, social, health and safety (ESHS) management system in place at NLGM, but the management system is not fully fledged. A more systematic approach to ESHS management has been taken in response to the cyanide incident at the mine in late 2015/ early 2016. Lessons learned and actions taken in response to this incident should be transferred to the ESHS management system as a whole.

BMMC manages the mineral processing operation, the tailings detoxification plant and the TSF operations such that cyanide and arsenic compliance criteria in the watercourses downstream of the mine are not exceeded.

After the mineral processing operation was first commissioned in 2015, there was a suite of challenges that resulted in failure to meet cyanide compliance criteria downstream of the mine and fish deaths in the downstream watercourses were observed. The problems have been addressed and impact studies by independent specialists contracted by the Company have confirmed that the river ecosystem has largely recovered and that people living downstream of the mine have not been adversely affected.

The mine’s monitoring data demonstrates compliance with relevant cyanide and arsenic criteria at the environmental compliance points from May 2016 to July 2017. There are internal check points for cyanide and arsenic in water on the mine. These include the tailings prior to discharge to the TSF, the penstock on the TSF and the point of release of supernatant from the TSF to the engineered wetland below the TSF. Data from the internal check points suggest that the cyanide detoxification and the arsenic removal processes interfere with each other. When the cyanide detoxification performance is highly effective, the performance of the arsenic removal process is not optimal. This does not result in non-conformance with environmental compliance criteria but can result in internal check point values being exceeded. BMMC is investigating this issue with the aim of optimising the performance of both detoxification processes.

BMMC has an extensive water monitoring programme, but interpretation of the data for parameters other than cyanide and arsenic has been not been thorough to date.

Several pollution control measures still have to be implemented by the mine.

The mine does have a commitment to develop and implement a biodiversity offset programme in its environmental permit. Biodiversity investigations and monitoring required for this are ongoing. Recent studies have confirmed that there could be critical habitat affected by the mine. A population of Isomacrolobium (Anthonotha) explicans could have been affected by waste rock dump development and it is noted that the critically endangered African slender-snouted crocodile (Mecistops cataphractus) is likely to occur in the rivers downstream of the TSF.

Full execution of the relocation action plan (RAP) was delayed by the Ebola epidemic (2014 to mid-2015) and a period of financial instability experienced by the mine (mid-2015 to mid-2016).

The stalled resettlement of Kinjor and Larjor is addressed in a memorandum of understanding agreed with the affected households. BMMC has committed to fully complete the resettlement by Q4 2018, with interim commitments to complete 200 household units by end of 2017 and implement a rolling plan of occupation commencing in Q1 2018.

The mine's stakeholder engagement needs improvement. Community engagement and grievance management has up until recently centred on a resettlement working group. The approach to stakeholder and community relations is currently being restructured and managed by a recently appointed Community Relations Manager and a revised stakeholder engagement plan will be finalised in November 2017.

BMMC has set up a number of cooperatives and community based initiatives as alternative livelihood activities. Reportedly BMMC is in the process of developing a comprehensive livelihood restoration plan that will be operational by the end of 2017. The intention is for this plan to include a range of additional initiatives including start-up of women's rotating credit schemes, and extension of modular brick making, following the RAP house unit construction, to a cooperative.

A closure plan was produced for the mine in 2013. The closure cost estimate based on the 2013 closure plan is USD10.0 million. SRK recommends that the closure plan and cost estimate are updated.

1.11 Economic Analysis

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

The financial model reflects post-finance, post-tax cashflows in real USD terms, allows for working capital and is based on the latest LoM plan commencing 1 October 2017.

A net present value (NPV) has been calculated for the expected cash flows from 1 October 2017 (i.e. excluding all capital costs (sunk costs), revenues and operating costs prior to this) through the application of Discounted Cash Flow (DCF) techniques to post-financing post-tax cash flows derived from the inputs and assumptions presented in this report. All figures are presented in Q3 2017 real USD terms.

A flat gold price of USD1,300/oz has been assumed along with a government royalty of 3% of net revenue. 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, however, it is noted that no corporation tax becomes payable under the current set of assumptions.

A summary of cash flow modelling is presented below in Table 1-3. In summary, this indicates a post-tax and post-financing NPV at a 5% discount rate of some USD179M.

Table 1-3: Cash Flow Model Summary

Description	Units	Project Totals/Averages
Recovered gold	koz	642
Mill processing life	Years	4.5
Net smelter revenue (after royalty)	USDM	808
Operating costs (including working capital)	USDM	(415)
Net operating cash flow	USDM	393
Capital, sustaining capital and closure costs	USDM	(53)
Net post-tax cash flow	USDM	340
Debt financing cash flows	USDM	(142)
Post-tax, post-financing cash flow	USDM	198
Post-tax, post-financing NPV (5%)[1]	USDM	179
Operating cash cost per ounce[1]	USD/oz	659
All-in sustaining cash cost[1]	USD/oz	749

1 Net present value ("NPV"), operating cash costs and all-in sustaining costs ("AISC") per ounce of gold produced are non-IFRS financial measures. These non-IFRS financial measures do not have any standardised meaning. Accordingly, these financial measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with International Financial Reporting Standards ("IFRS"). Operating cash costs and all-in-sustaining cash costs are a common financial performance measure in the mining industry but have no standard definition under IFRS. Operating cash costs are reflective of the cost of production and include a net-smelter royalty of 3%. AISC include operating cash costs, corporate costs, sustaining capital expenditure, sustaining exploration expenditure and capitalised stripping costs.

1.12 Conclusions

SRK's conclusion is that the Project remains technically feasible and economically viable. The Project has faced a number of challenges since its inception, both technical and financial, however, many of these issues have now been resolved or there are on-going plans in place for project improvements, following a recent change of ownership and investment by the new owners.

Compared with historical physical performance achieved to date, BMMC is forecasting increases in both mine and plant production on an annual basis. While these increases are achievable in theory with the equipment planned, if these increases are not achieved in practice then the unit operating costs will be higher than currently assumed and the resulting Project NPV would be lower than presented herein.

Similarly, compared with historical operating costs achieved to date, BMMC is forecasting savings to be made going forward and a corresponding reduction in unit costs. These cost savings are at an early stage of implementation and require confirmation in practice. SRK is confident that if the cost savings are made then the Project NPV presented in this report will be realistic, however, if the changes are not realised then the NPV could be considerably lower.

2 INTRODUCTION

This Technical Report on the New Liberty Gold Mine (New Liberty, NLGM 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 Avesoro Resources Inc. (Avesoro). Bea Mountain Mining Corporation (BMMC or the Company), which is a 100% owned subsidiary of Avesoro, has a 100% interest in the Bea-MDA.

The Project was the subject of a Feasibility Study completed by BMMC which was reported in October 2012. Subsequent to this, additional work was 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 this time, the Company has continued to conduct further evaluation work at New Liberty, including grade control drilling which has helped to produce a better geological understanding of the deposit.

Furthermore, construction commenced in late 2012/early 2013, pre-stripping mining activities commenced in October 2014 and the first gold pour occurred in May 2015 and the Project has been in operation since this time.

Prior to issue of this report, the most recent Technical Report produced on the Project was issued on 25 March 2015 and was entitled ‘New Liberty Gold Project, Bea Mountain Mining Licence, Southern Block, Liberia, West Africa, Definitive Project Plan’ (the 2015 Technical Report), and reflected the status of the Project at that time.

This updated 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). It has been compiled by SRK and describes the current status of the Project, 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 derived by BMMC and reviewed by SRK which reflects the current Life of Mine (LoM) plan.

SRK is part of an international group (the SRK Group), which comprises over 1,400 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 this Technical Report are Dr Mike Armitage BSc, MIMMM, C.Eng, C.Geol; Dr David Pattinson CEng, MIMMM, BSc and Jane Joughin Pr.Sci.Nat, all of SRK. 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, 18-19 and 21-27; Dr David Pattinson for Sections 13 and 17 and Jane Joughin for Section 20.

All QPs have visited the site on a number of occasions as follows:

- Dr Mike Armitage: 20-23 November 2012, 7-10 July 2015 and 8-11 November 2016
- Dr David Pattinson: 7-10 July 2015, 1-5 December 2015, 2-5 February 2016, 4-11 May 2016, and 8-11 November 2016
- Jane Joughin: 7-10 July 2015, 19-23 April 2016 and 8-11 November 2016

SRK's opinion, effective as of 01 November 2017, is based on information provided to SRK by BMMC 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 Avesoro and neither SRK nor any affiliate of SRK has acted as advisor to Avesoro 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. SRK has not, however, 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 BMMC 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 also relied on copies of a document provided by BMMC that confirm the terms of the Licence.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 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.

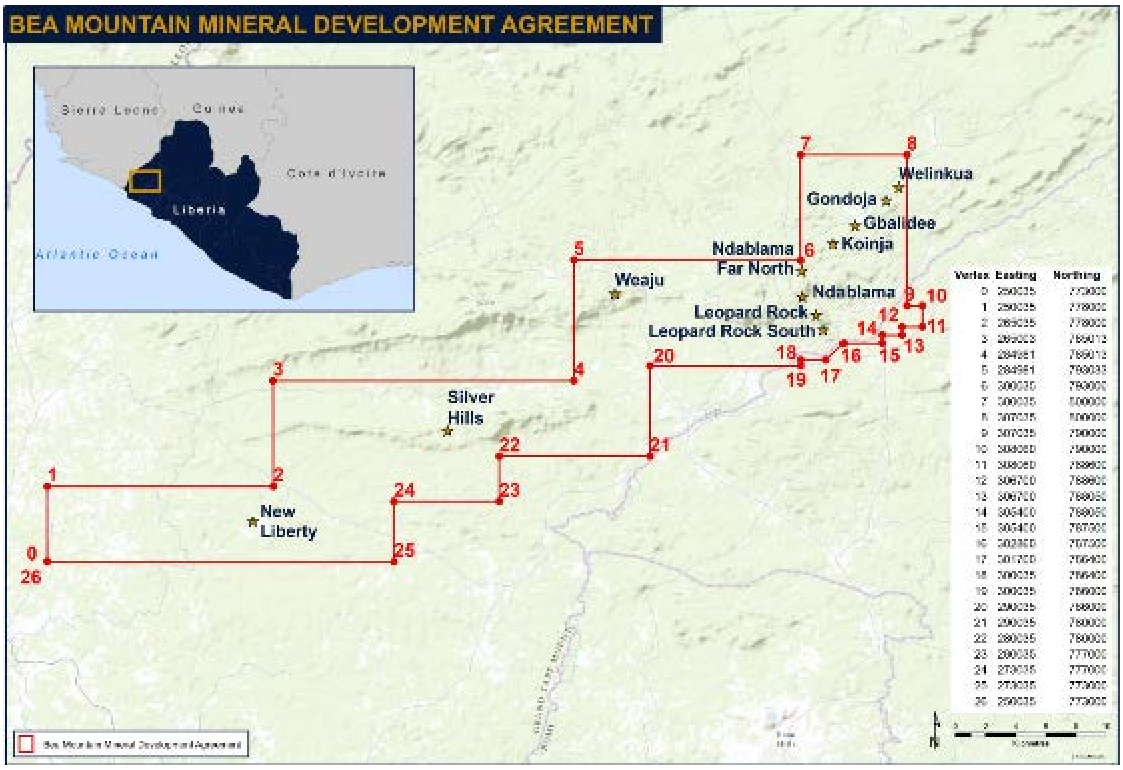
The Bea-MDA property is situated 90 km north-west of the capital in Grand Cape Mount County, in the north-western portion of Liberia, approximately longitude 11º west, 7º north, as shown in Figure 4-1. The Project is situated within the Bea-MDA property, the UTM coordinates of which are shown in Table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

Source: *BMMC, 2017*

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

4.2 Property Description

The Bea-MDA property covers an area of 478km² 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 BMMC but which are not discussed in this report. The Bea-MDA property, which is covered by a Class A mining licence, was reduced in size from a prior exploration lease which covered a total of 1,000 km².



Source: *BMMC, 2017*

Figure 4-2: Class A Mining Licence Limits

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

Vertex	Easting	Northing
0	250,035	773,000
1	250,035	778,000
2	265,035	778,000
3	265,003	785,013
4	284,981	785,013
5	284,981	793,033
6	300,035	793,000
7	300,035	800,000
8	307,035	800,000
9	307,035	790,000
10	308,060	790,000
11	308,060	788,600
12	306,700	788,600
13	306,700	788,050
14	305,400	788,050
15	305,400	787,500
16	302,860	787,500
17	301,700	786,400
18	300,035	786,400
19	300,035	786,000
20	290,035	786,000
21	290,035	780,000
22	280,035	780,000
23	280,035	777,000
24	273,035	777,000
25	273,035	773,000
26	250,035	773,000

4.3 Ownership

BMMC has a 100% interest in the current Bea-MDA, which was signed with the Liberian Government in November 2001. BMMC was previously a wholly owned subsidiary of African Aura Mining Inc. (African Aura), formerly called Mano River Resources Inc. 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 Mining Inc (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.

During 2016, following a period of financial difficulty, Aureus was the subject of a change of control following three equity investments from MNG Gold recapitalising the business, During December 2016, Aureus was renamed Avesoro Resources Inc. (Avesoro), whilst MNG Gold was renamed to Avesoro Holdings Jersey Ltd. 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	BMMC	Approval received
22 April 1998	BMMC	Bea-MDA defined as 1000 km²
28 November 2001	BMMC	Bea-MDA reduction to 457 km² came into effect
29 July 2009	BMMC	Granted a Class A Mining Licence
19 September 2013	BMMC	Restated and Amended Mineral Development Agreement

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 BMMC which was subsequently approved by the government on 22 November 1996. In April 1998, in anticipation of a new Mining Code, BMMC replaced the existing licence and assignment, and entered into a specially-negotiated Exploration Agreement. Upon ratification of the new Mining Code in 2000, BMMC, 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 BMMC’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, BMMC 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 BMMC the right to free access to public land and will assist BMMC in cases where access to private lands is necessary. Prior to the commencement of exploitation and production BMMC 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 BMMC is required to submit detailed plans "for the protection, correction and restoration of the water, land and the atmosphere".

BMMC was granted a Class A Mining Licence (the Licence) on July 29, 2009. The annual licence fee, based on the production area of 457 km^2 ("the Production Area"), amounts to USD0.80 per acre, which equates to USD90,146 per annum (1 km^2 = 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. This licence area was added to on the south eastern corner of the property, increasing the area to 478km^2. This change was made on 11 May 2015. The Licence allows BMMC 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.

BMMC was granted all the normal operating licences and permits for the mining operation, including licences associated with explosive storage and use, abstraction and discharge of water and construction.

4.5 Environmental Management

Prior to the commencement of mining operations in late 2012/early 2013, to the extent known, the area had only limited artisanal workings, and no historical environmental issues.

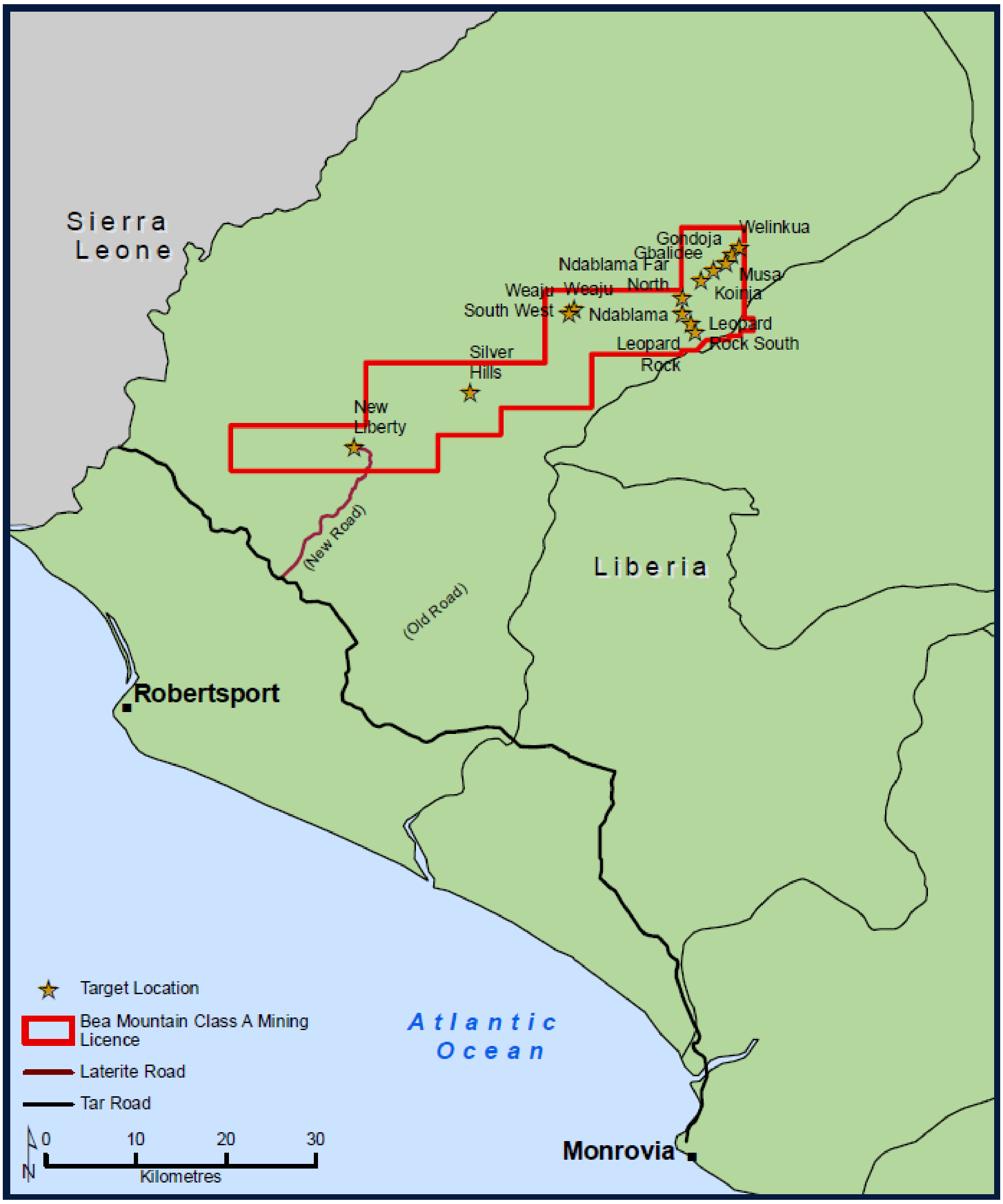
Baseline data collection for the ESIA was initiated in the fourth quarter of 2010 and was conducted during both the wet and dry seasons. The ESIA, as per Liberian legislation, included a Public Participation Process (PPP). An Environmental Impact Statement (EIS) was submitted to the Environmental Protection 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. BMMC 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 Danielstown and a further laterite section of 20 km to the Project. BMMC has upgraded the laterite section of road and installed five new culvert-type bridges to facilitate access to site. Secondary roads on the licence, built by BMMC, provide access across the property. The sandy nature of the roads allows all year round access, including during the height of the rainy season. During 2017 BMMC has constructed a laterite capped airstrip at the Project site which is capable of accepting small fixed wing aircraft. A helipad is also located on site.



Source: *BMMC, 2017*

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 50m above mean sea level (amsl) to a maximum of 600m amsl with the majority of the licence area being composed of gently undulating plains which reach less than 200m 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 30m to 40m above the forest floor, with thick undergrowth common. The (primary rain forest is mainly in the mountainous area while the 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 peak rainfall occurring between mid-July and August. During the winter months of November to March, dry dust-laden Harmattan winds blow from the north and east. The average annual rainfall at the site is some 3,500mm with over 4,000mm falling along the coastal belt but declining 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 mining activities have been able to continue throughout the rainy seasons.

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 10,600km of road networks throughout the country, of which 650km 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 490km 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 mine site uses 2,560-1,024 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 palm oil 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 occurred within the Bea-MDA property highlighted 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 and development activities across the licence is show in Figure 6-1.

Prior to completion of the Feasibility Study on the Project in 2012, 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.

Following completion of the Feasibility Study, and as reported in the 2015 Technical Report, AMC derived Mineral Resource and Reserve estimates as presented in Table 6-3 and Table 6-4 respectively. These were reported with effective dates of 1 October 2012 and 20 May 2013 respectively and were reported before any mining had commenced and are therefore un-depleted.

Table 6-3: AMC Mineral Resource Estimate (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

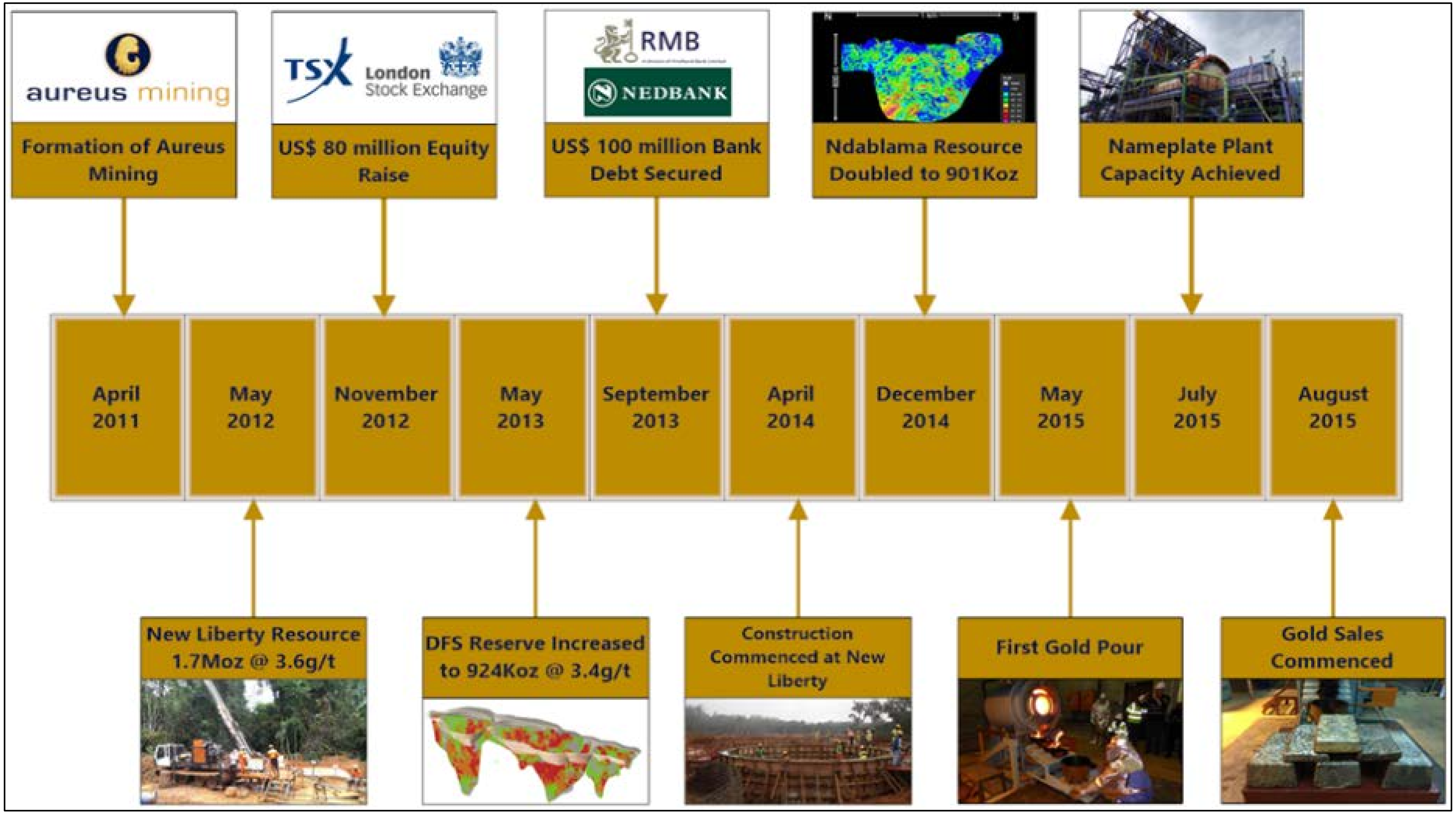
Table 6-4: AMC Mineral Reserve Estimate (as at 20 May 2013)

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 reviewed the work completed to produce the above Mineral Resource and Reserve estimates as presented in the 2015 Technical Report and considered these to have been appropriately derived and to reflect the information available and the mine plan available at the time.

This Technical Report has an effective date of 01 November 2017 and presents updated Mineral Resource and Reserve estimates as derived by SRK, with effective dates of 31 July 2017 and therefore take into account mining depletion up to this time.



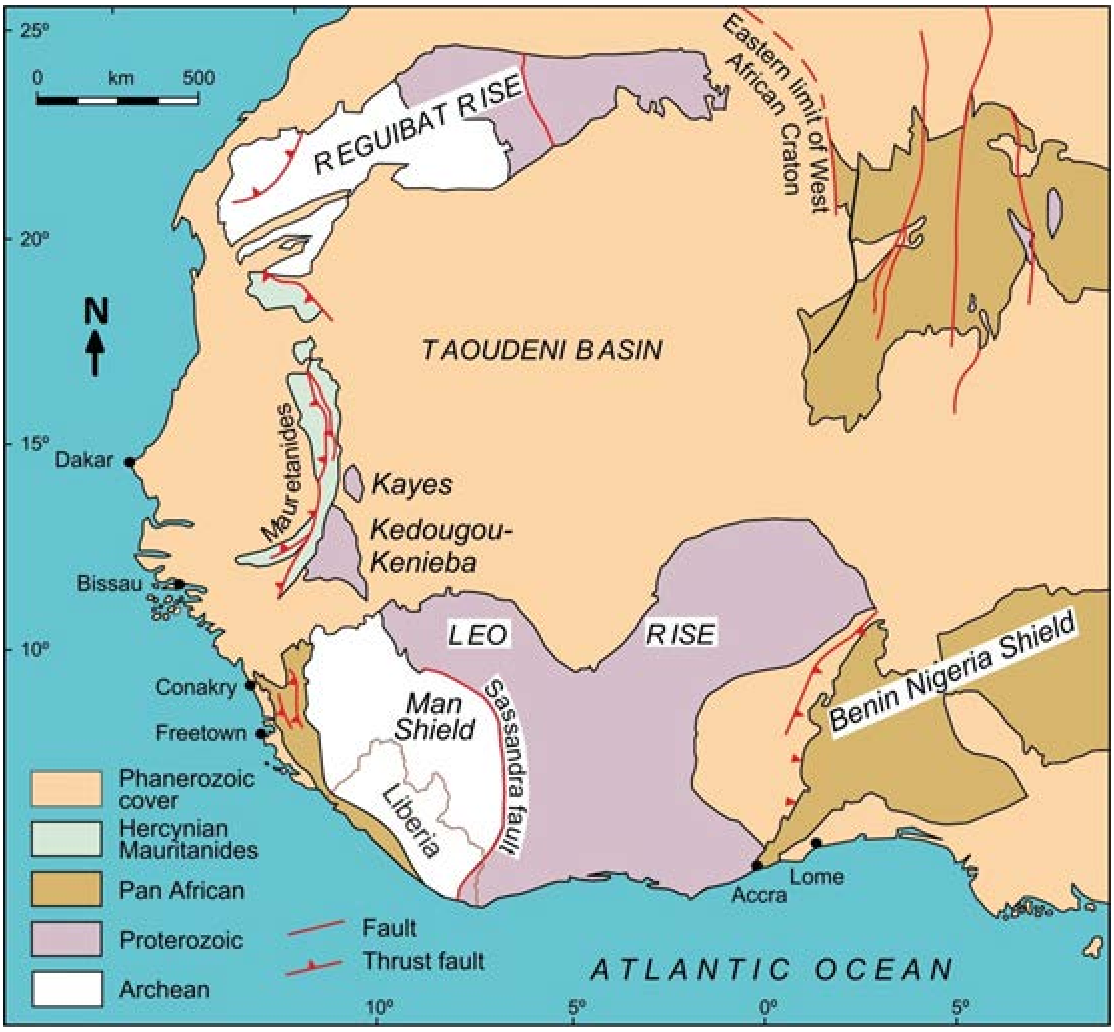
Source: BMMC, 2017

Figure 6-1: History of Development at the New Liberty Project

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 billion years ago (1.7Ga). 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).

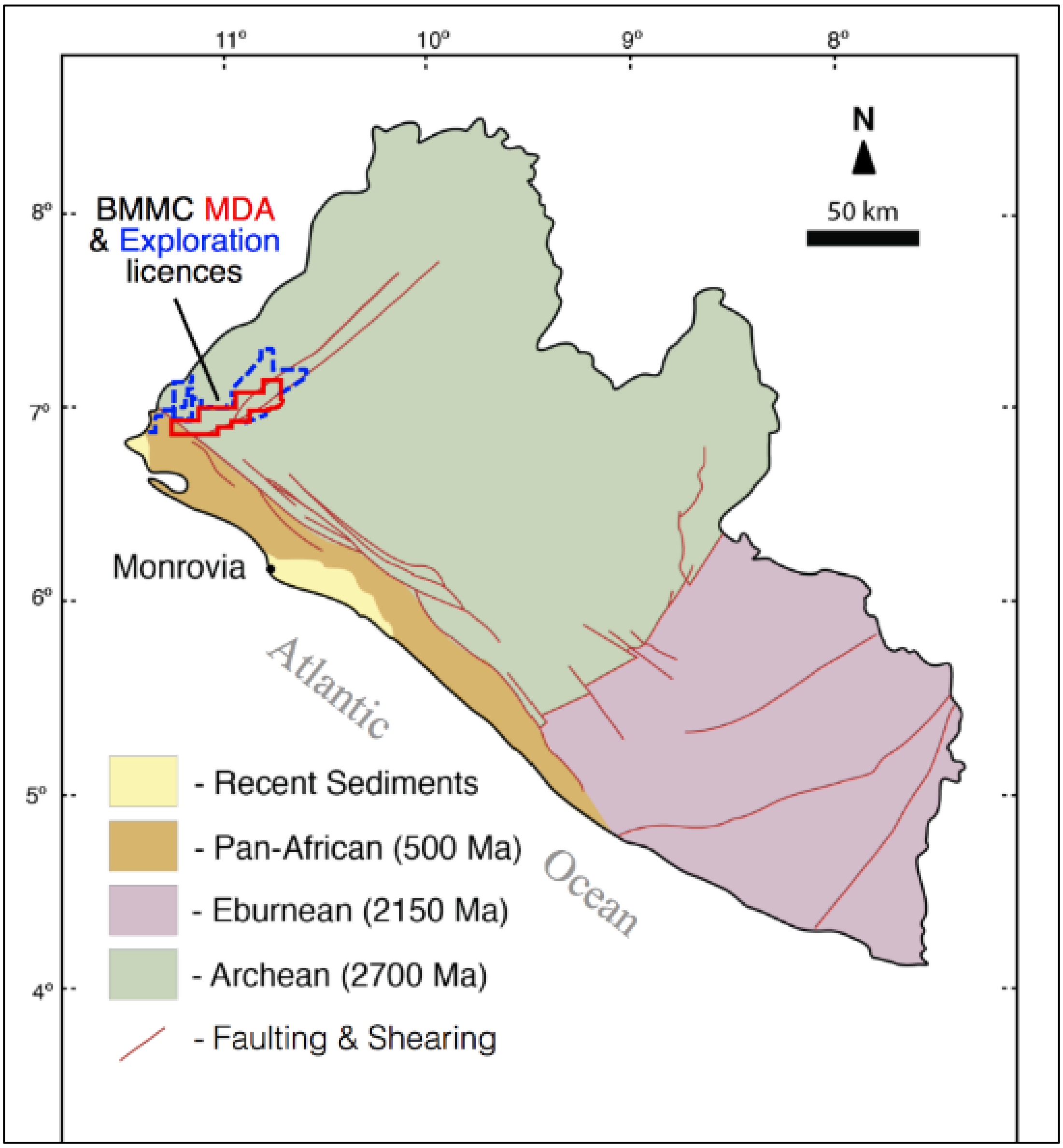


Modified from: *Milési et al. 1992*

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.



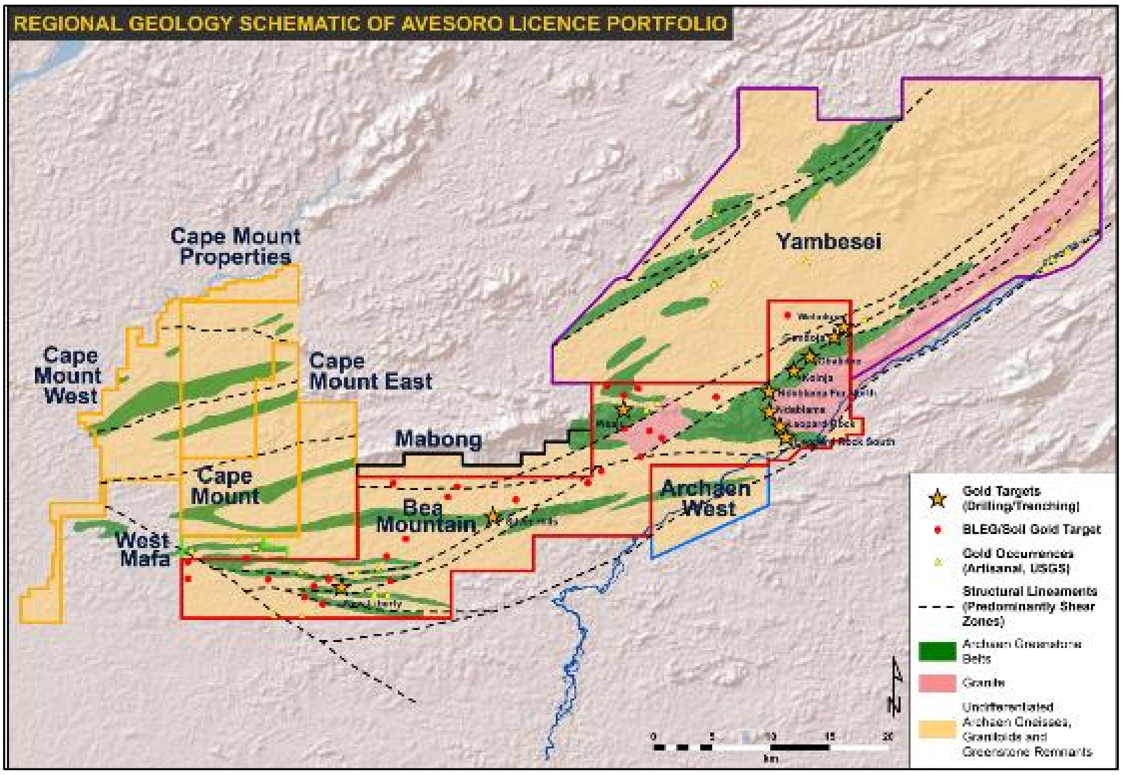
Source: *Hurley et al., BMMC 2017*

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.



Source: BMMC 2017

Figure 7-3: General Geology of the Bea-MDA and surrounding properties

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 (summarised 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 an example cross section through the New Liberty deposit.

Source: *BMMC, 2017*

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

Source: *BMMC, 2017*

Figure 7-5: Almandine Garnet Porphyroblasts in HWC

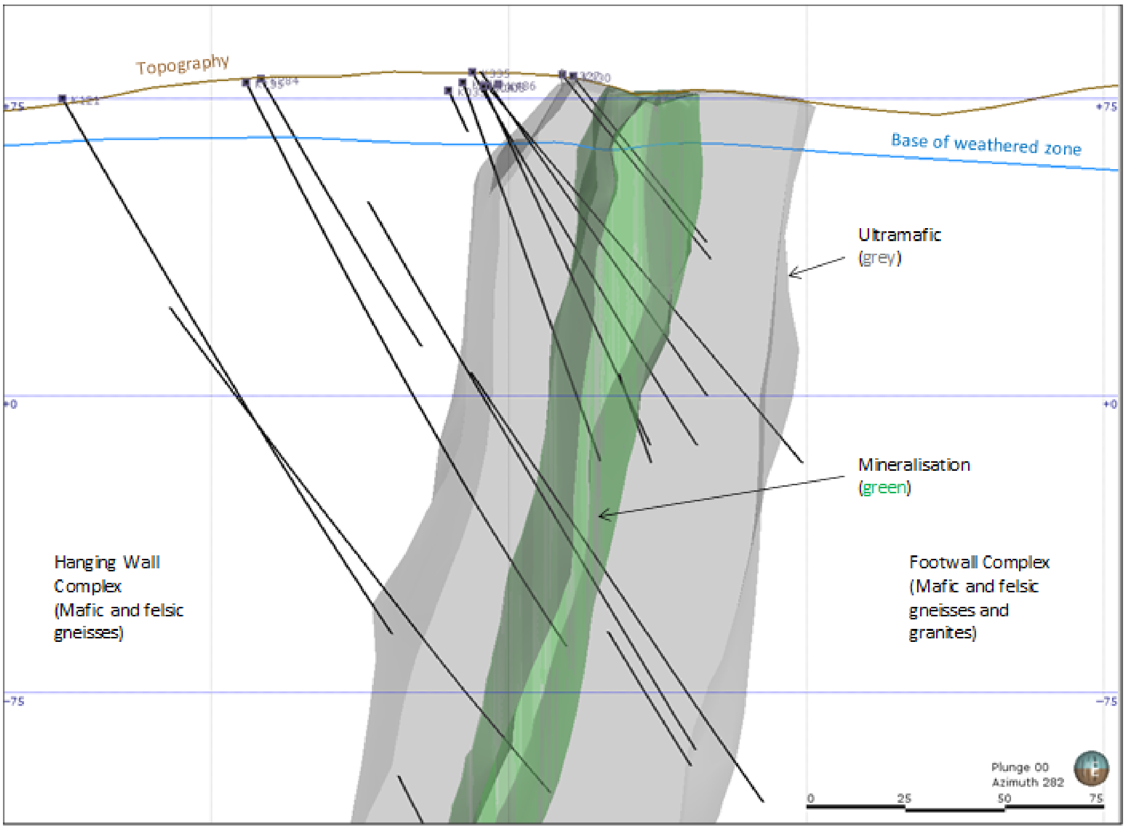


Figure 7-6: Example Cross Section through the New Liberty deposit (looking west)

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.

7.4 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.

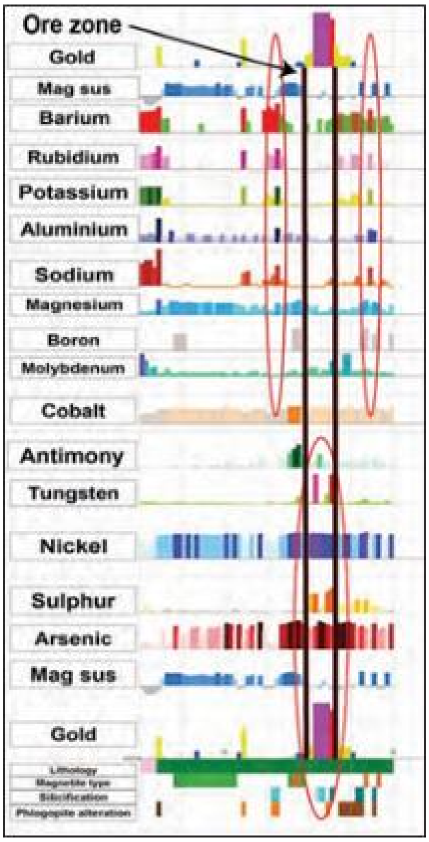
7.5 Alteration

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 mineralised 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 and it has a low sulphide content with sulphides forming between 0.1 and 1 % of the mineralised zones.

Relationships have been established between magnetite depletion, silicification, phlogopite alteration and gold mineralisation.

Figure 7-7 shows the geochemical associations both in the mineralised zone and margins to these. 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.



Source: *BMMC, 2013*

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

7.6 Mineralisation

The vast majority of the mineralisation at the Project is hosted within the altered parts of the sheared ultramafic rocks. Pyrrhotite, gersdorffite and arsenopyrite are the main sulphides with occasional pyrite and rare chalcopyrite or pentlandite. 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 10m, and is nearly continuous along 2km 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.

7.7 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, 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, gersdorffite, arsenopyrite, coarse grained pyrite, chalcopyrite, sphalerite and minor pentlandite are the principal sulphides.

In Figure 7-8, pyrrhotite, arsenopyrite and pyrite are shown in cut and uncut ultramafic core, with the bulk of the sulphides aligned to the dominant cleavage.

Source: *BMMC, 2013*

Figure 7-8: Mineralisation in Core

7.8 Summary of Field Character of the Mineralisation

The gold mineralisation at the Project is sometimes associated with sulphides, hosted in metamorphosed ultrabasic rocks (which is locally intruded by tourmaline-bearing granites and quartz breccias 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 sheared and is locally 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.

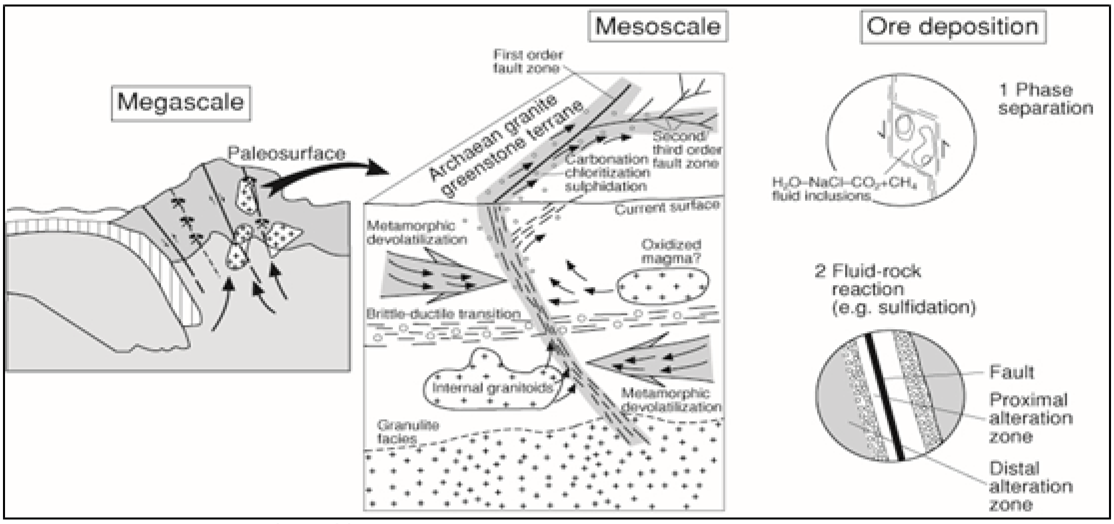
8 DEPOSIT TYPES

The mineralisation being targeted by BMMC 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 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 8km-15km depth. Structures are typically formed at, or close to, contacts between rock types of contrasting competencies, and mineralisation is often localised 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 BMMC's mineral exploration programme in Liberia are shear zone-hosted gold systems, sometimes 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 BMMC 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 was also 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

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

9.2.5 Trenching

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

9.2.6 Pitting

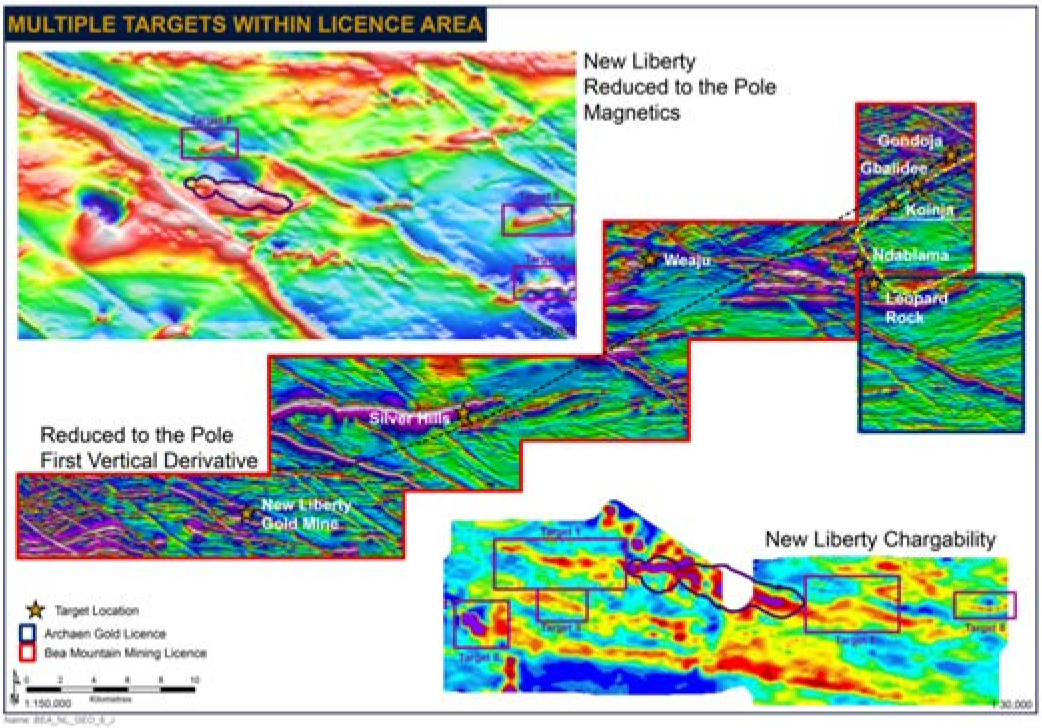
Pits were staked by geologists, typically in lines that perpendicularly intersect the strikes of structures or anomalies. They were square, 1.2m by 1.2m, and excavated to bedrock for a maximum depth of 4m and were surveyed and logged, with samples taken as continuous channels perpendicular to regolith and lithological boundaries.

9.2.7 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 BMMC. Sufficient overlap between the old and new survey and matching line spacing enabled the surveys to be merged together. The survey parameters of both are summarised 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



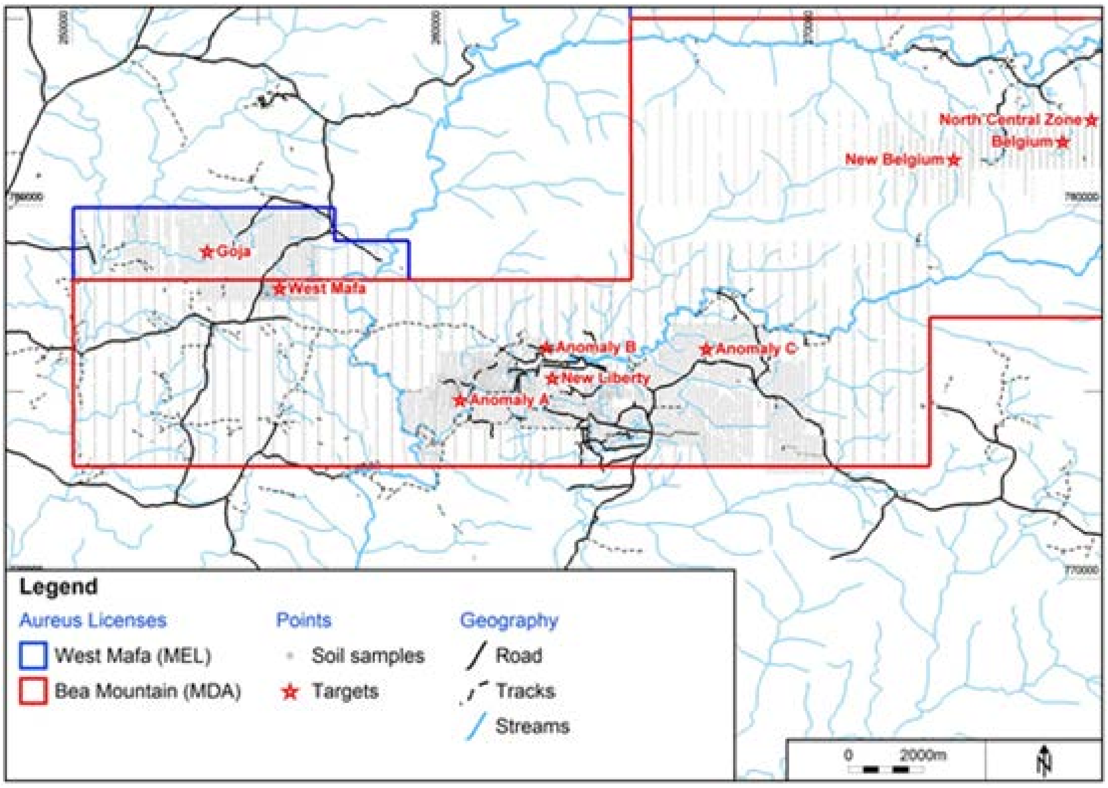
Source: *BMMC, 2012*

Figure 9-1: New Liberty Geophysics Interpretation

9.3 Regional Exploration

9.3.1 Soil Geochemistry

Geochemical soil sampling in 1999 on a 100m by 20m grid over 1km each side of the known mineralisation detected a strong anomaly over 200m 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: BMMC, *2015*

Figure 9-2: Soil Sampling Coverage over the New Liberty Area showing targets identified

9.3.2 Trenching

Following an encouraging channel sample programme of artisanal workings (Figure 9-3), which yielded intersections including 19.95m at 4.06g/t Au in the west and 13.1m at 4.56g/t Au in the centre of the system, trenches T1–T12 were excavated in 1997, each 3m deep trench aligned approximately perpendicular to the east-west strike of the mineralisation. This covered an along-strike extent of 1,800m (Figure 9-3). During 1998, trenches T13–T24 were completed at intervals of 100m along the geological strike and 20m–80m long to depths ranging from 2.0m to 4.0m 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.

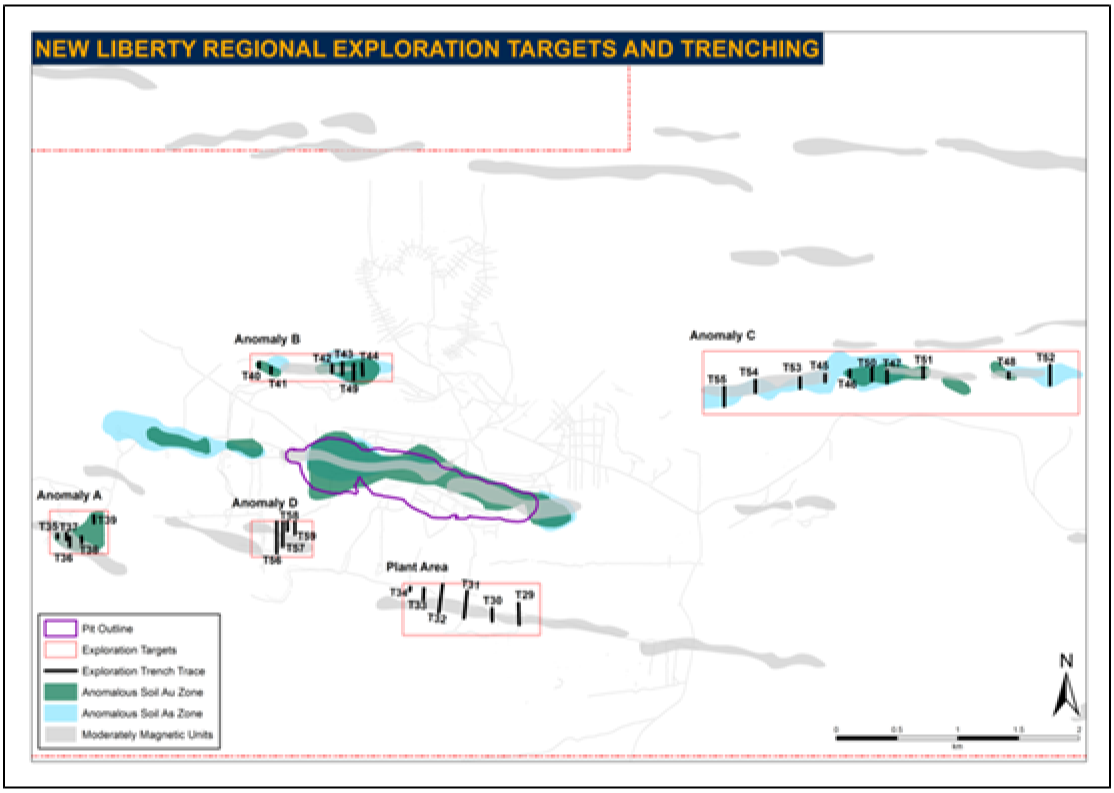
All trenches were geologically mapped and channel sampled (metre-length samples). All samples were despatched to the SGS Laboratory in Monrovia for analysis for gold, and results were assessed as they were received.

Source: *BMMC, 2012*

Figure 9-3: Artisanal Workings in Larjor

Source: *BMMC, 2012*

Figure 9-4: Exploration Trench



Source: *BMMC, 2013*

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

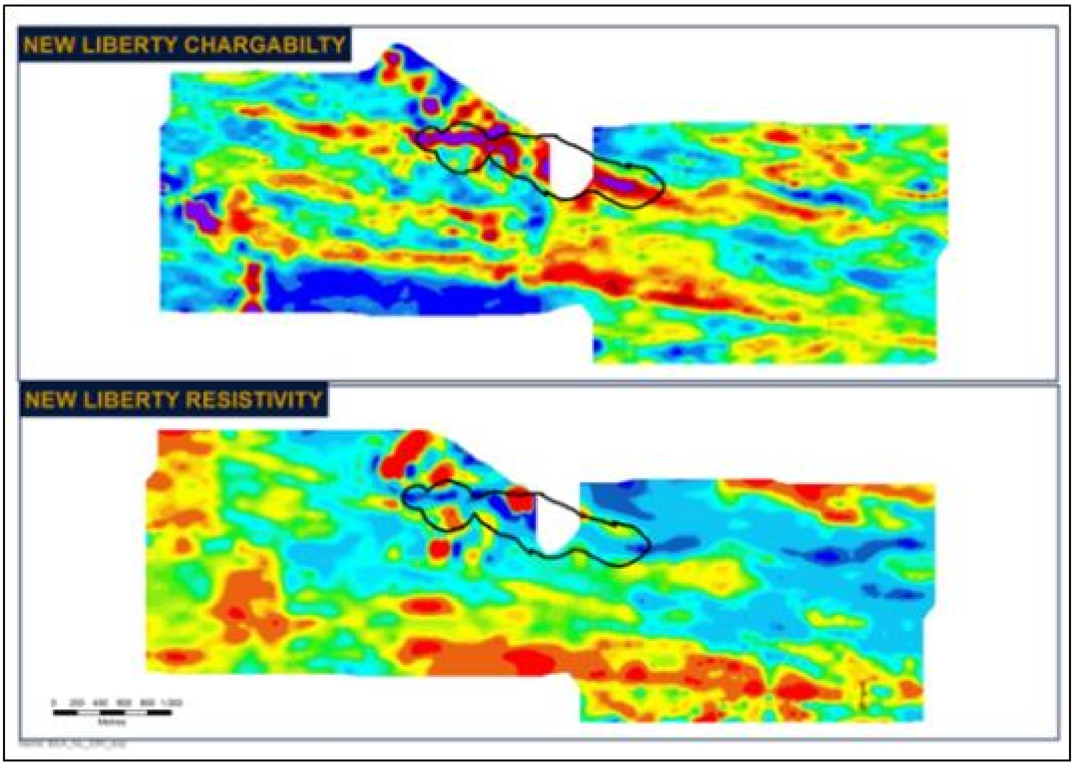
9.3.3 Pitting

Following up from mapping and a revaluation of soil sampling, 1 x 1 m wide pits were dug over several near mine targets during the 2015/2016 field season, to a depth of 3-4 m. These were then geologically mapped and sampled from pit floor to surface at intervals accordant with regolith and lithology. Samples were dispatched to ALS laboratory in South Africa for gold analysis, and results were assessed as they were received.

Information gathered from pitting was used to enhance the geological interpretation of near mine targets, including regolith and structure.

9.3.4 Geophysics

Following from the airborne survey, ground magnetic, induced polarisation (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 15km^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.

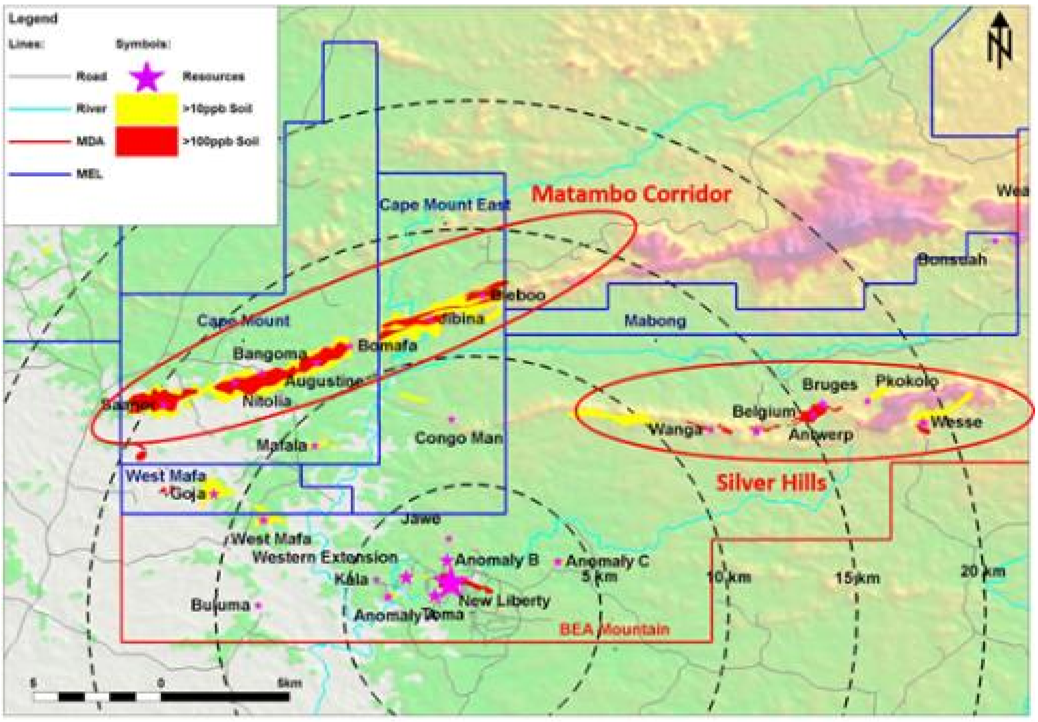


Source: *BMMC, 2012*

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: *BMMC, 2017*

Figure 9-7: Further 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 13km north-east of the Project. Soil sampling, trenching, pitting, and detailed mapping results have highlighted a zone potentially 3 km long within a 15km soil corridor. Channel samples have shown narrow, but high grade mineralisation but this has not been drill tested.

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

This section of the report outlines the drilling and in-pit channel sampling that has been completed on the Project and data made available to SRK for Mineral Resource estimation (MRE) purposes. The drillhole database made available to SRK comprised all sample data for the Project completed up to 18 January 2016. Since then, additions to the database have been made and for the purposes of Mineral Resource estimation these have been limited to grade control drilling at the Marvoe deposit area to reflect the Company's current focus on mining within this area. The files supplied had an effective cut-off date of 04 August 2017.

The drillhole database provided to SRK comprises a total of 1,036 holes totalling some 115,984m of drilling and 25 channels for 1,574 m of sampling. In comparison to the previous MRE reported by AMC in October 2012, the database utilised for the current MRE includes an additional 924 drillholes (53,131m) which largely relates to infill grade control drilling that is taking place ahead of mining as this progresses.

10.2 Exploration Drilling

10.2.1 Introduction

Diamond drilling at the Project was conducted periodically between 1999 and 2012 (Table 10-1). The total number of meters drilled in the exploration phase was 67,998m which was completed in 7 campaigns.

Table 10-1: Summary of Diamond Drilling Campaigns

Campaign	Hole Numbers	Year	Number of Holes	Meters
1	1 – 19	1999 - 2000	19	1,949
2	20 – 26	2000	7	792
3	27 – 61	2005	35	3,027
4	62 – 114	2006	53	5,069
5	115 – 130	2008	16	4,487
6	131 – 184	2009 - 2010	54	14,556
7	185 – 441	2011 - 2012	248	38,118
Total			**432**	**67,998**

*Drilling totals exclude all hydrogeological, geotechnical, metallurgical and sterilisation holes completed at the Project

The drilling was carried out in part by contractors and in part by BMMC. Campaigns 1-5 were completed 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 BMMC-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 and Figure 10-2 highlight the position of exploration drillhole collars and drilling orientation in relation to mineralisation wireframes.

The core sizes drilled varied over time as well as within holes, typically HQ (63mm) or NQ (47mm) but also ranging from AQ/DT48 (27mm) to HW/T6116 (90mm). The quarter core from the first 27 diamond drillholes and half core for the remaining holes are stored on site. Figure 10-3 shows a view of the core storage facilities at the time of the Campaign 6 drilling.

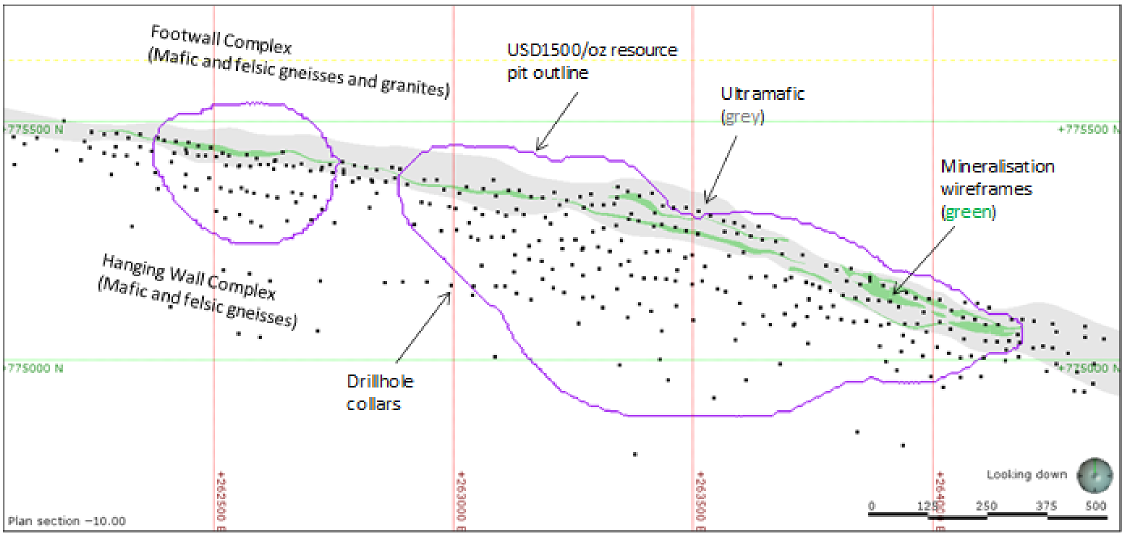


Figure 10-1: Location of Diamond Drillhole Collars

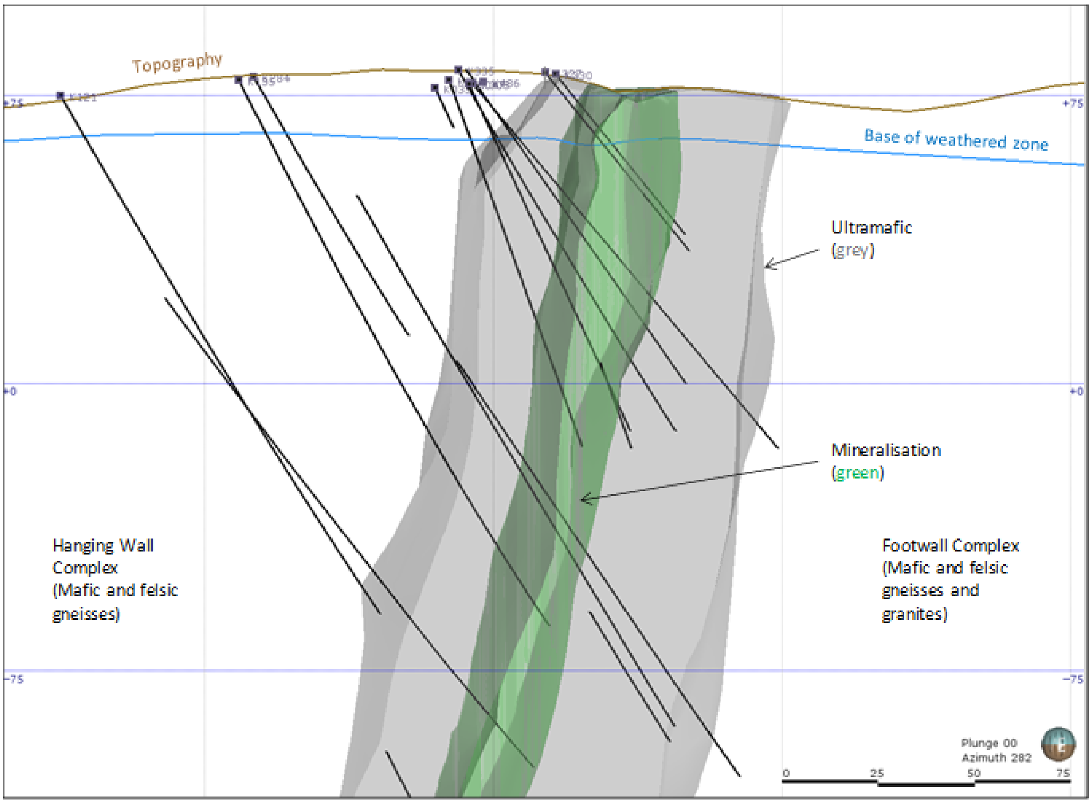


Figure 10-2: Example cross section through the New Liberty deposit (looking West)

Source: *BMMC, 2012*

Figure 10-3: Core Shed

10.2.2 Drill Programme Campaigns

Campaign 1 comprised 19 holes drilled at HQ (68mm), with the exception of hole K16, which was started at HQ and reduced to NQ (48mm). The holes were drilled on 50m centres and intersected mineralisation at depths ranging from 20m to 30m 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 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 Zone. K20 and K23 were drilled in the central part of the Larjor ore body and intersected mineralisation at some 50m and 100m below surface respectively. K21 and K22 were drilled on the Marvoe Zone near hole K10.

The third diamond core drilling campaign, designed to close along-strike inter-hole distances to a maximum of 25m 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 80m below surface. The programme also aimed at further evaluating the eastern extremity of the Marvoe 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 300m below surface elevation while two (both in Larjor) investigated and demonstrated that the Larjor zone mineralisation persists to -600m level.

In 2009 (Campaign 6) a 10,730m 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. 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, 248 diamond drill holes were drilled for a total of 38,118m. 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.

10.2.3 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 9.2.1) 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 Mineral Resource estimation.

Additional resurveying and validation of accessible pre-2011 collars was conducted in 2011 and all additional collars associated with the 2011 campaign were surveyed with the Leica DGPS survey procedures described in Section 9.2.1.

10.2.4 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 50m), the results of which demonstrate minor downhole azimuth and dip deviations (less than 5° deviation over 100m), 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 109m, do not have downhole survey records. For the 2008 programme, multiple downhole surveys were conducted, but intervals between readings were relatively wide, typically between 50m and 100m. All holes drilled during the 2009/2010 and 2011/2012 campaigns were surveyed at short intervals (10m and 5m respectively) and constitute the best records of drillhole deviations for the Project. During the 2011/2012 campaign, initially 5m intervals were used (up to and including K331 and K336), with the remainder at 10m interval

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

10.2.5 Acoustic Televiewer (ATV) Probe

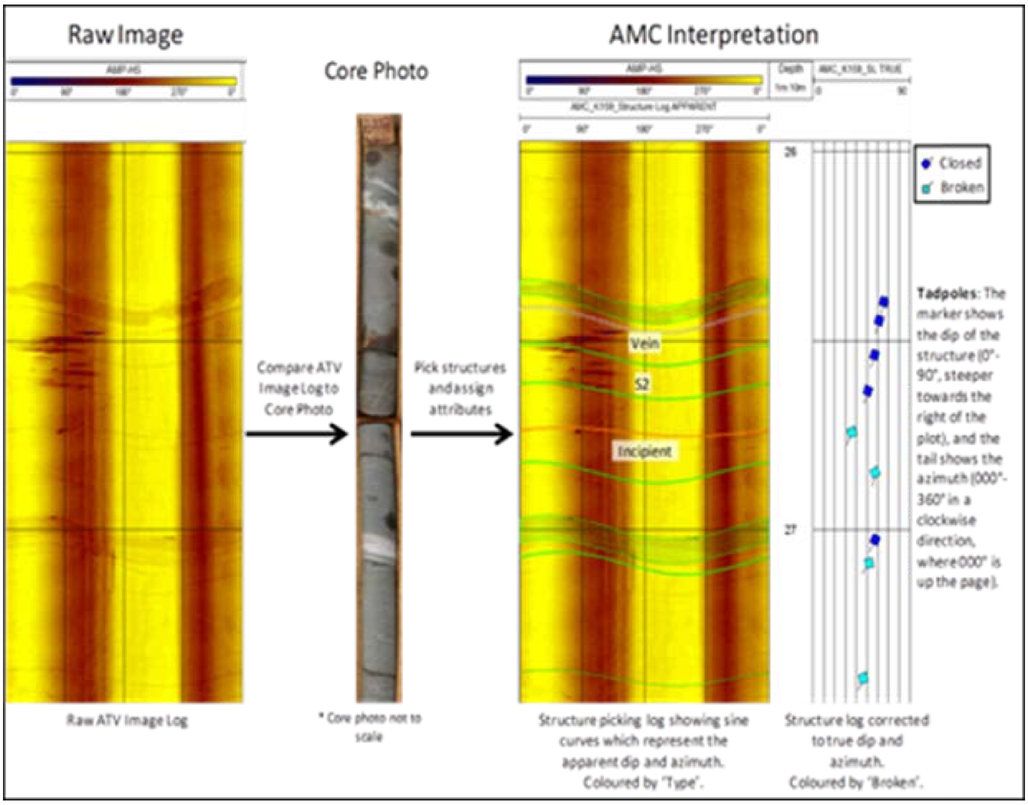
In order to obtain additional high quality geotechnical baseline information from existing inclined diamond boreholes, Lim Logging were commissioned to undertake ATV surveys. A total of 19 holes were surveyed (Table 10-2), between 2 and 15 of March 2013, for a combined total of 1,783 metres.

The data gathered by the ATV probe was processed on-site, generating an orientated acoustic image of the borehole wall (Figure 10-4) 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.

Table 10-2: 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



Source: *BMMC, 2015*

Figure 10-4: Acoustic Image and Interpretation of ATV Survey

10.2.6 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 7-5 shows good core recovery in spite of the tendency for mineralised rock competencies to be lower than in adjacent un-mineralised intervals.

Source: *BMMC, 2015*

Figure 10-5: Drill Core Showing Recovery

10.3 Sterilisation Drilling

Some 6,810m of sterilisation drilling has now been completed within the Project area. The 2013 drilling phase consisted of 12 diamond drill holes beneath the plant site, waste dump footprint and the tailings storage facility. The details are shown in the table below, which also lists the details of previous sterilisation drilling undertaken on alternative sites proposed in earlier studies on the Project (Table 10-3).

Table 10-3: New Liberty Sterilisation Drilling

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

10.4 Grade Control Drilling

10.4.1 Introduction

Reverse Circulation (RC) Grade Control drilling was undertaken during 2014 and 2015 by Ore Search Drilling and from 2016 to date this has been undertaken in-house by the Company. Grade Control drilling is used by the Company to update short term grade control models for short term mine planning purposes.

To date, the drilling has been undertaken in three phases, the first of which focused on bringing the drill hole spacing in the upper levels of the Larjor pit profile down to some 12m by 12m (see Figure 10-6). The second phase was undertaken in the Kinjor area of the main pit, 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. The third and most recent phase of grade control drilling has been completed in the Marvoe area, which has resulted in a sample coverage of approximately 10m by 10m.

A summary of the grade control drilling completed is provided in Table 10-4, with the positions of the grade control drillhole collars (and prior diamond hole collars) available for the current MRE illustrated in Figure 10-6.

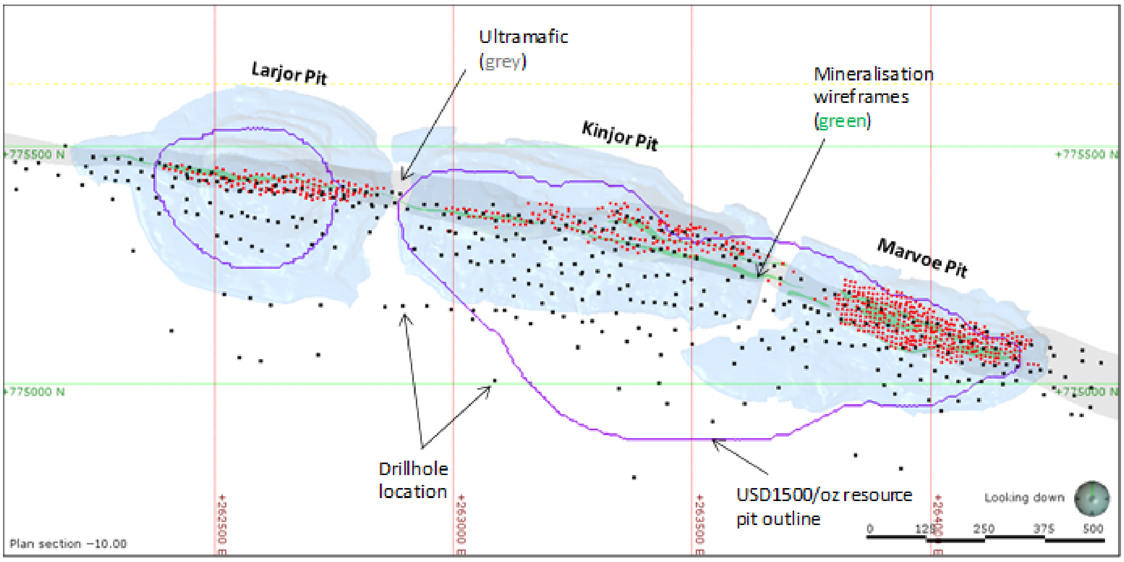


Figure 10-6: Location of grade control collars (red) completed up to 04 August 2017

Table 10-4: Summary of Grade Control Drilling as at 04 August 2017

Drilling Type	Count	Total length (m)
Grade Control	723	29,251

10.4.2 Survey and Orientation

Grade control collars were surveyed using a Trimble differential GPS, with downhole surveys typically completed for holes greater than 40m in length at 10m increments.

Drilling was conducted on a grid, with holes generally drilled on an azimuth between 005° to 010° azimuth (magnetic) and inclined at between minus 45° and 60° to intersect the south-dipping zones. Figure 10-7 illustrates the grade control drilling orientation in relation to exploration drilling and mineralisation wireframes.

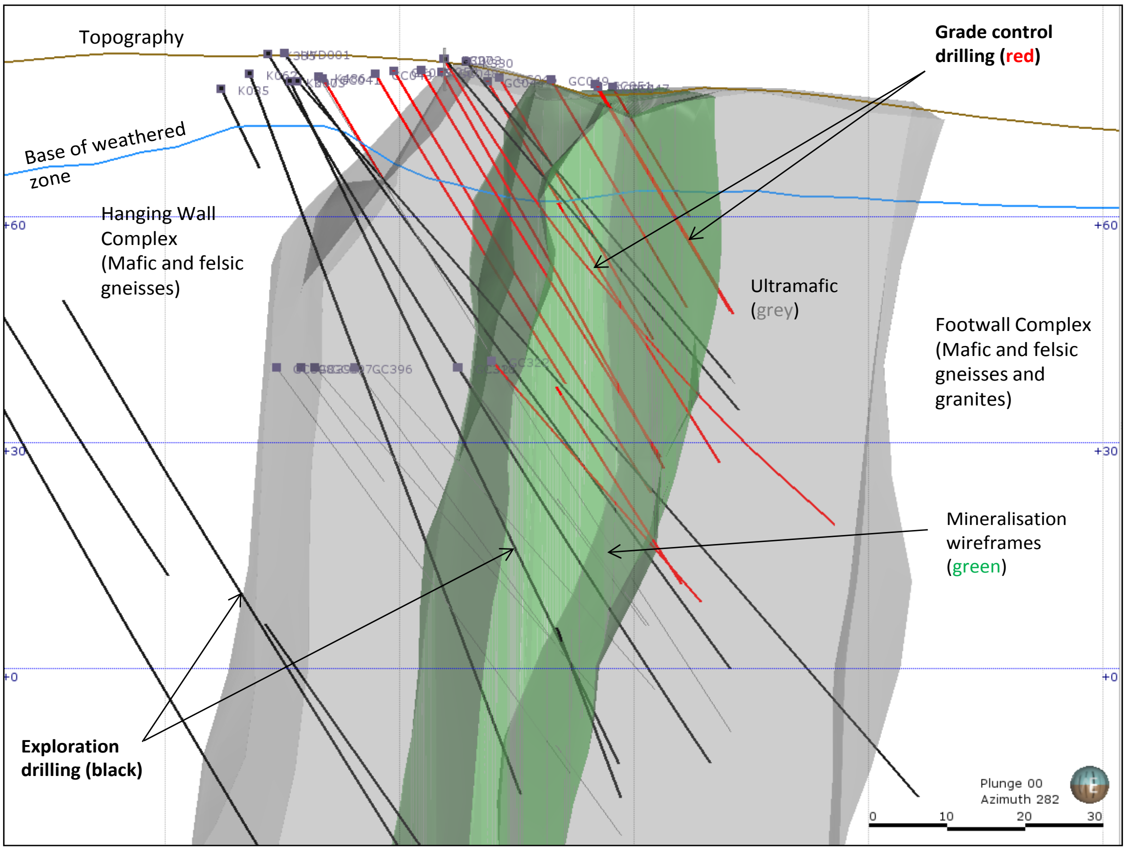


Figure 10-7: Example cross section through the New Liberty deposit showing Grade control and exploration drilling looking west

10.4.3 Drilling Procedure

Reverse Circulation (RC) Grade Control drilling was undertaken during 2014 and 2015 by Ore Search Drilling using a tracked EDM 2000 with Auxiliary Booster. Since January 2016, Grade Control drilling has been undertaken using a Sandvik DR560 DTH drill owned by BMMC with convertible RC top drive. A 140mm hammer is used for sampling 1m intervals to a maximum vertical depth of 30m. Bulk samples were then reduced using a rotational cone splitter. Samples collected using the Pozitif Drilling rig since November 2016 were split using a riffle splitter. This enables 30RL of benches to be drilled out at any one time with 5m of over drill. If additional information is required beyond 30m depth, then an external contractor is utilised.

Collar positions are marked out by surveyors, with marker pegs used to align the rig to the correct azimuth without the need of a compass to avoid issues from magnetic interference. An inclinometer with a spirit bubble is used to set the mast at the correct inclination. During drilling, sampling is completed at 1m intervals with each sample recorded using Hole ID, sample ID, start and end depth. Samples are weighed, logged in terms of moisture content, split using a riffle splitter and then geologically logged. The cyclone is inspected and cleaned typically every 3-5 (3m) drill runs, with cleaning of the sample splitter completed using compressed air between each 1m sample.

Batches of samples are subsequently transported to the core shed for insertion of QA/QC materials and laboratory dispatch processing.

10.4.4 In Pit Channel Sampling Programmes

In order to further define the mineralisation outlines in the pit floor (in conjunction with RC Grade control methods), a series of 1 metre channel samples were collected from shallow trenches dug at 10m intervals perpendicular to the strike of the mineralisation across the floor of the pit, as illustrated in Figure 10-8.

Channels were marked out by a surveyor and then excavated to a depth of approximately 0.5m using a hydraulic excavator (CAT 330). The one metre channel samples are bagged in pre-labelled plastic bag and transported to the ALS on-site laboratory for preparation and analysis, using the same methodologies used for the grade control samples as described in Section 11.3.3.

A summary of the channel sampling completed by the Company is provided in Table 10-5 with the position of the channel samples (and drillhole collars) available for the current MRE at the Marvoe deposit area illustrated in Figure 10-9.

Table 10-5: Summary of Channel Sampling as at 04 August 2017

Drilling Type	Count	Total length (m)
Channel Sampling	25	1,574

Figure 10-8: Channel sampling across the floor of the Marvoe open pit to add to grade control information

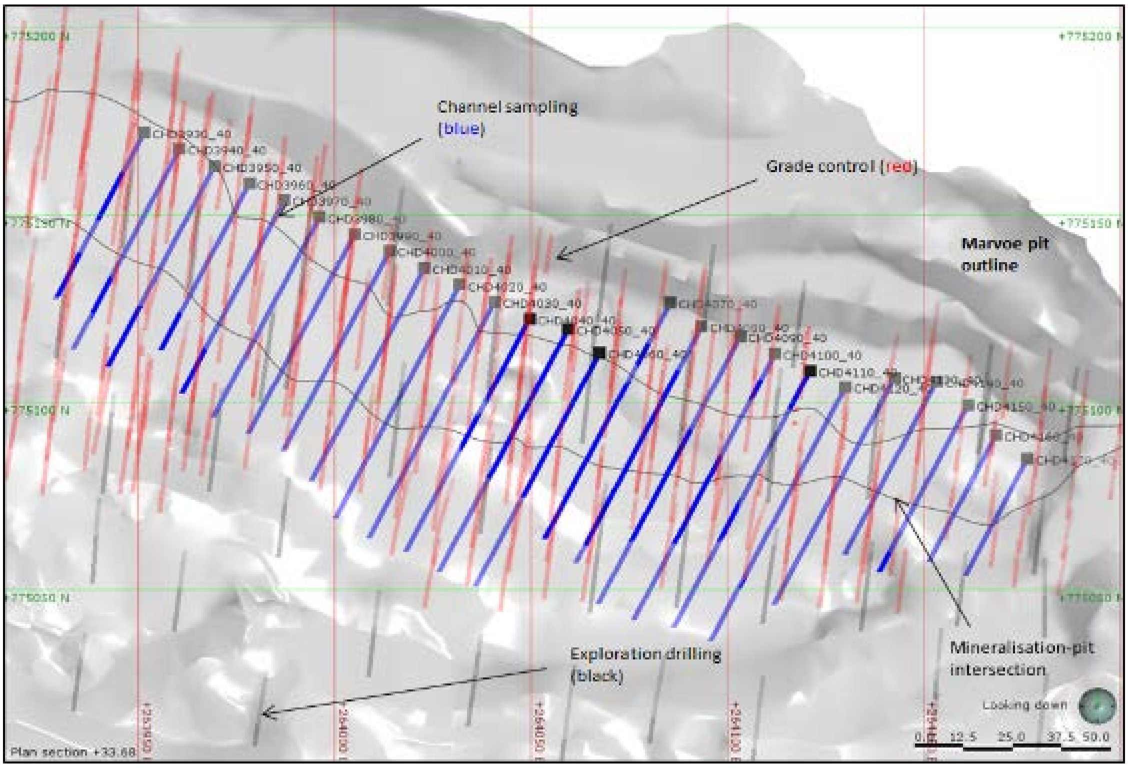
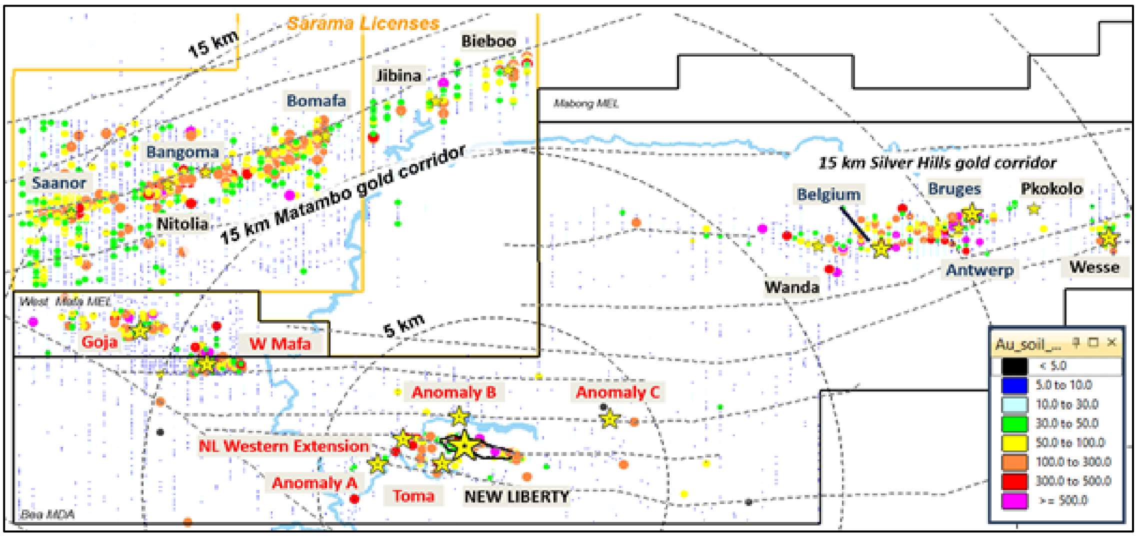


Figure 10-9: Channel Sampling completed at Marvoe open pit

10.5 Drilling Near the Project

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



Source: *BMMC,2017*

Figure 10-10: Drill Targets Near to the Project

10.6 SRK Comments

In SRK's opinion, with the exception of a few inconsistencies in the earlier drilling campaigns, the drilling procedures at the New Liberty Project generally conform to industry best practices and the resultant drilling pattern is sufficiently dense to interpret the geometry and boundaries of the gold mineralisation with an appropriate level of confidence.

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 (in the database provided to SRK) 51,137 drill core samples, 23,099 grade control samples and 1,579 channel samples have now been collected and submitted for gold assay from the Project.

11.2 Soils and Trenches

Soil samples have been collected from 0.5m 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 10cm 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 BMMC geologists.

11.3 Diamond Drillhole Samples

Diamond-drilling activity at the Project was also supervised by BMMC geologists. Core and core blocks were placed in core boxes by the driller. Upon reception in the core shed on site, core was cleaned or washed (if required) and core blocks checked by BMMC staff. The core was then photographed, wet and dry, in a frame to ensure a constant angle to and distance from the photographer. Magnetic susceptibility readings were taken every metre. For unconsolidated core this was 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, K265, K266, K289, K293, K303, K306, K317, K325, K327, K329, K339, K340, K349, K365, K371 and K464 to K495. Otherwise, only sulphides were recorded before the core was cut. For oriented core, additional point data was collected, as defined by depth and alpha and beta angles of fabrics.

Geological logging used a from-to format to record depths, rock codes and brief descriptions of the lithological units and angles of contacts. Sample intervals were 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 was chosen to be at 90° to the predominant structure so that each cut half of the core was a mirror image.

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

Early exploration samples were 2.0 m in length (holes K1-K18). For holes K21-K27, the 2m sampling interval over suspected mineralised zones (rich in arsenopyrite and pyrrhotite) was maintained but sampling adjacent to the mineralised zone was extended to 4m. Subsequently, from K27 to K40, 1m samples were introduced for target intersections, retaining 2m intervals over suspected weakly mineralised material. Thereafter, the adopted norm was to sample boreholes uniformly at 1m 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 were taken at 2m intervals within the same lithology and on every lithological break. This was carried out by weighing samples in air and water with a balance. Porous samples were first wrapped in plastic. For drillholes K1-130, measurements were carried out on half core, i.e. post-sampling, but for subsequent holes whole core was used. Measurements were recorded using a balance with top and under-slung measuring capabilities with detection limit of +/-1 gm.

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

For unconsolidated material, density was 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

A total of some 14,044 bulk density measurements were supplied by the Company. The raw density data was initially coded using the modelled base of weathering surface and mineralisation wireframes, with the descriptive statistics per domain provided in Table 11-2 and Table 11-3.

Table 11-2: Summary of density per mineralisation and weathering domain

Group	Description	Field	Zone	Sample No.	Mean	Max	Min
0	Host rock	DENSITY	Weathered	391	1.6	3.1	0.9
			Fresh	11,996	2.9	3.8	1.0
100	Mineralisation		Weathered	79	1.7	3.0	1.0
			Fresh	1,578	3.0	3.7	1.1

Table 11-3: Summary of density per weathering domain

Group	Description	Field	Zone	Sample No.	Mean	Max	Min
All	Host rock and Mineralisation	DENSITY	Oxide	470	1.64	3.11	0.93
			Fresh	13,574	2.91	3.84	0.99

The density data has been interpolated in to the Mineral Resource block model using an Inverse Distance Weighting Squared (IDW) algorithm, with a value of 1.65 g/cm^3 applied to all the oxidised material (which is the average of the raw sample and declustered mean) given its comparatively sparsely sampled nature and limited contribution to overall deposit metal (1%).

During statistical and visual analysis of the density sample database, SRK noted limited overall differences between the mineralised zones and background host rock and therefore density interpolation and assessment of average density was completed using a combined dataset. The parameters used for IDW interpolation of density in to the block model are presented in Section 14.

11.3.2 Sample Security

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

11.3.3 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 were 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 Certified Reference Material (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 (1kg 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 1kg 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 BMMC.

The flow chart in Figure 11-1 summarises sample collection, sample preparation, assaying and QA/QC procedures adopted during the 2009/2010 and 2011/2012 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).

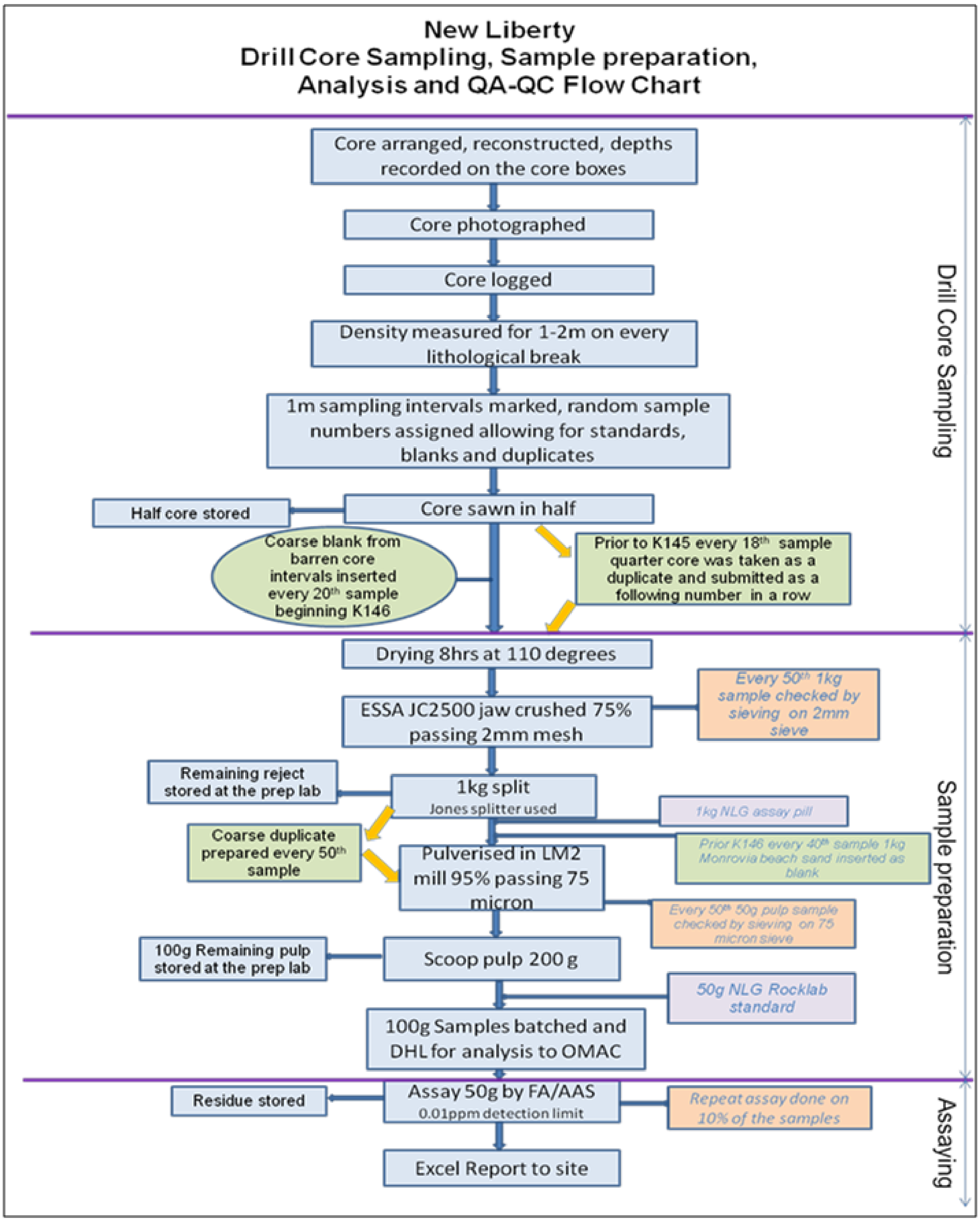
On arrival of the prepared pulps at the laboratory, samples were checked against the submission sheet, logged into LIMS, and homogenized to prevent segregation that might have occurred in transit. Large consignments of samples (>300) were split into smaller sub-batches of 200 samples for convenience 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 2mm using terminator jaw crusher. 1kg crushed material splits were taken using a riffle splitter and milled using a LM2 mill to 90% passing 100 microns. 50g 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.



Source: BMMC, 2015

Figure 11-1: Sample Preparation and QA/QC Flow Chart

2014-2017 Grade Control Drilling

Pre-generated sample ID's and Standard QAQC inserts were developed on 1m intervals for all holes. Samples were taken for every interval (not just those perceived to be in/and around the mineralised zone), but only those intercepting the Mineral Resource wireframe limits plus an additional 5m wide buffer zone where dispatched to the laboratory for both Au and As analysis. Standards, field duplicates and blank samples were also taken within the mineralised zone to monitor analytical performance at the laboratory.

Full 1 metre RC samples were collected from the rig via a cyclone and were riffled on site to a split totalling about 2.5-3.5 kg which was bagged in pre-labelled plastic bags. These were transported to a clean sorting area, sequenced, had standards inserted and were then batched and sent for analysis.

Prior to October 2015, laboratory samples were sent for sample preparation and analysis for gold to SGS in Monrovia and ALS Johannesburg. Samples were dried using an electric drying oven, crushed using a Boyd crusher to 85% passing 2mm sieve and pulverised with a Labtech Essa LM2 mill to a size of 90% passing 75 microns with a 50 g split sent for analysis. All samples were analysed for gold by fire assay with AAS finish.

Since then, all sample preparation and analysis for grade control drilling has been completed at the ALS on-site laboratory (ALS NLGM). At the on-site facility, samples are dried at a temperature of 105°C in an electric drying oven, crushed using a terminator crusher to a size of 75% passing 2mm sieve and pulverised to a size of 85% passing 75 microns using a Labtech Essa LM2 mill. A 50 g split is subsequently analysed for gold by fire assay with AAS finish. The on-site laboratory is managed by ALS.

Grade Control Bulk Samples

Bulk samples were split on site using a Jones Riffle splitter down to a 2.5kg laboratory sample. These 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 11-4 details the statistics recorded for the bulk recoveries for the phases of drilling completed within Larjor and Kinjor pit areas.

Table 11-4: 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 rock once holes had been collared off improved significantly as expected.

11.4 SRK Comments

In SRK's opinion, the sampling preparation, security and analytical procedures used by the Company are consistent with generally accepted industry standard practices and have facilitated the production of data of sufficient quality to support the Mineral Resource estimate presented in this report.

12 DATA VERIFICATION

12.1 Verifications by SRK

SRK has completed several visits to the Project between 2012 and 2016, and most recently from 8-11 November 2016. During the visits SRK has reviewed drill core for selected holes to confirm both geological and assay values stored in the database, discussed geological and structural interpretations and witnessed the extent of the exploration and mining completed to date.

12.1.1 Verification of Sample Database

SRK completed a phase of data validation on the digital sample database supplied by the Company which included, but was not limited to the following:

- A search for sample overlaps, duplicate or absent samples, anomalous assay and survey results. No material issues were noted in the final sample database;
- The exclusions of the following historic drillholes from the database that did not pass all aspects of SRK's validation procedures:
  - K070 – Low confidence in collar location, based on position of the mineralised zone;
  - K004 – Low confidence in position of mineralised intercepts in hangingwall;
  - K084, K082, K080 – Non-sampled holes within significantly mineralised zone (poor fit);
  - K192 – Low confidence in collar location and downhole survey;
  - K023 – Low confidence in downhole survey, superseded by more recent drillholes.
- The exclusion of the Marvoe in-pit channel samples given concerns over the confidence in the survey data based on apparent visual offset (in the order of 2-8m) from the expected position of the mineralised zone when compared with drilling information, particularly towards the east of the deposit. Excluded channel samples comprise CHD3930_40 to CHD4170_40.
- The exclusion of the '0 g/t Au' assay results in the grade control drilling at Marvoe which have been confirmed by NLGM to represent zones of poor sample recovery in the weathered material close to surface; this represents <1% of the sample database.
- A search for non-sampled drillhole intervals within the mineralised zones. SRK noted the presence of a significant number of absent sample intervals for gold within the mineralised zones (some 12% of the database), which largely relate to planned grade control (GC) holes or poor recovery in the drilling close to surface. SRK has reviewed the absent interval data on a case by case basis and inserted a low grade value (0.1 g/t Au) only in instances where the intercept has been interpreted as non-mineralised and therefore 'not sampled' (1% of the sample database), with sample data noted as missing (or lost) ignored during the sample compositing process.

12.2 Verifications by the Company and its Consultants

BMMC completes routine data verification as part of its on-going drilling programmes. Checks completed include validation for all tabulated data, including collar and down-hole survey, sampling information, assay and lithology interval data, with validation of sample results from the latest phase of drilling using standards, field duplicates and blank samples inserted routinely into each batch submitted to the laboratory.

As part of previous Mineral Resource estimates for the Project, during 2010 and 2012, AMC completed a phase of checks between database-entered data against the original sources (hard copies). With the exception of a limited number of typographical errors and inconsistencies in downhole survey records for the 2000-2006 drilling campaign, which have been subsequently been rectified, no material data entry errors were reported.

12.3 Assay QAQC

12.3.1 Introduction

For the discussion below, the drilling campaigns have been combined into periods, since little QA/QC work was carried out during in the early campaigns. The QA/QC results relating to latest phase of drilling are presented in detail, with previous programs presented in summary form.

12.3.2 Period 1999-2000

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. Sample-to-sample comparison between the original and check assays were poor, however, given the observations by Lakefield Research (Lakefield Research, 1999) which showed the presence of abundant free gold, ACA Howe noted that a strong nugget affect could be expected to influence the correlations.

12.3.3 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 were 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 were 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 was 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 this 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.

12.3.4 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. Pulp blanks recorded one outlier that most likely indicates a misclassification of a standard.

Standards (CRM)

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

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

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

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 recognised 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

At the start of the 2009/2010 campaign, field (quarter) 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 that this practice be ceased and increase the number of crush 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 duplicates 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.

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 10g/t Au were excluded.

The laboratory repeat assay results showed an overall precision of 12.5% which was considered high, with the expected precision for laboratory repeats expected to be well below 10%. The poor precision could be attributed to inherent high nugget effect or poor preparation procedures.

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.

Umpire Laboratory Check Assays

BMMC selected ALS Chemex as an umpire laboratory, and for the programme a new set of standards was purchased from Rocklabs. ALS Chemex used a 30g fire assay method compared to the 50g 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 analysis showed a precision of 19.3% which, in context of sample pulps re-prepared from coarse rejects, was considered to be within acceptable limits. In addition, the OMAC results show some negative bias (1.05%) relative to the ALS Chemex values.

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, albeit less of a bias than in the original OMAC results.

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

12.3.5 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 0.606g/t Au to 4.107g/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 these 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 was probably not significant and it is notable that the umpire laboratory returned assays consistent with those from the primary laboratory.

In addition, a suite of 1,116 samples were submitted to an umpire laboratory, SGS, for verification against the original OMAC 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 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.

12.3.6 Period 2014-2017

Since the start of the Grade Control (GC) drilling from 2014, CRM samples have been routinely inserted in to the analytical sample stream. Blank and field duplicate samples are also inserted, with the exception of the following drilling periods:

- No blank samples were used for the QAQC programme for drillholes completed prior to GC218 (relating to 35% of the GC samples inside mineralisation wireframes);
- No field duplicate samples were inserted between drillholes GC215 and GCM103 (relating to 25% of the GC samples inside mineralisation wireframes).

Despite the incomplete QAQC support for the drilling periods listed above, these drillholes are interspersed with those that are supported by QAQC data, they are visually comparable with adjacent intersections with QAQC and also show comparable sample distributions and mean grades.

Standards

Since the start of the Grade Control drilling at New Liberty, the Company has introduced 14 different externally certified standard materials (CRMs) into the analysis sample stream at an overall insertion rate of approximately 8%. The selected CRMs were sourced from Geostats Pty Ltd (Geostats) in Australia and range in gold values from 0.22g/t Au to 8.66g/t Au, as illustrated in Table 12-1.

Table 12-1: Summary of Certified Reference Material for gold submitted by the Company in sample submissions

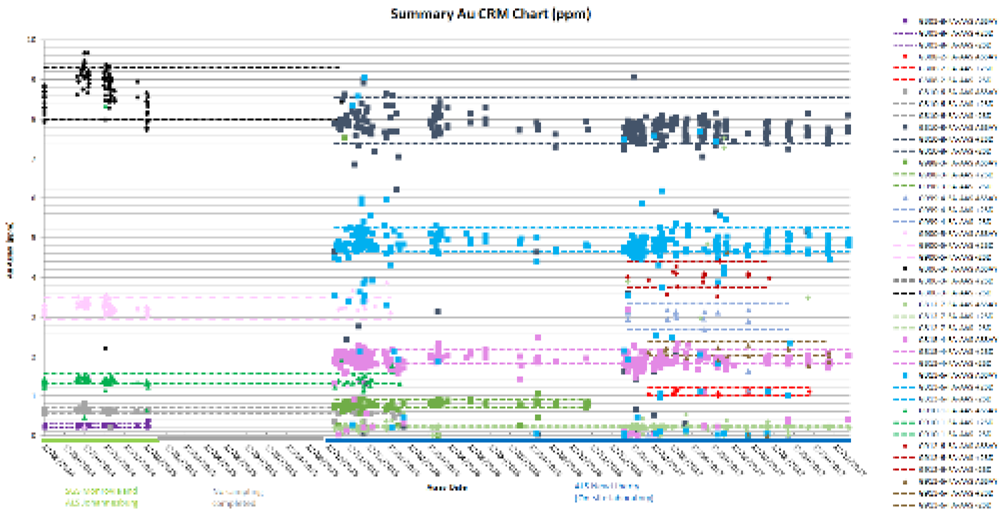
Standard Material	Gold; Au (ppm)		
	Certified Value	SD	Company
G-303-8	0.26	0.03	Geostats PTY LTD
G-306-3	8.66	0.33	Geostats PTY LTD
G-310-6	0.65	0.04	Geostats PTY LTD
G310-8	7.97	0.29	Geostats PTY LTD
G312-7	0.22	0.01	Geostats PTY LTD
G313-4	2.00	0.08	Geostats PTY LTD
G313-6	4.94	0.15	Geostats PTY LTD
G-900-5	3.21	0.13	Geostats PTY LTD
G-910-1	1.43	0.06	Geostats PTY LTD
G998-3	0.81	0.05	Geostats PTY LTD

SRK has reviewed the CRM results for gold and notes that whilst the analytical accuracy in general is considered acceptable (on average within +/-1% of the certified value), there is evidence of CRM swapping and in a small number of cases potential mislabelling of non-QA/QC samples. This has occurred most notably since the change in assay laboratory from SGS Monrovia and ALS Johannesburg to the on-site ALS facility (ALS NLGM) during October 2015 (relating to some 65% of sample data inside mineralisation wireframes).

The change in laboratory also coincides with a drop in precision in the CRM results, which is highlighted as scatter either side of the +/-2SD limits on the individual CRM charts as illustrated in Figure 12-1. There does not however appear to be a consistent bias towards lower or higher grade.

SRK consider that the (10m) close-spaced nature of the grade control drilling combined with the use of multiple samples to inform block estimates sufficiently smooths the scatter and potential sample swaps in the CRM results, resulting in no overall material impact on the interpolated block model gold grades.

Notwithstanding this, to maximise confidence in the grade control database, SRK has recommended further investigating (and rectifying) the CRM swaps and reduction in analytical precision noted since the start of the ALS NLGM laboratory and re-submitting 5-10% of sample pulps analysed at ALS NLGM to an umpire laboratory to further verify analytical performance.



* Four anomalous high-grade CRM results ranging between 12-16 g/t Au are excluded from this chart to help illustrate the results for the grade range of interest for the New Liberty model.

Figure 12-1: QAQC Standard Summary Charts for gold from submission of New Liberty Grade Control Samples

Blanks

Since grade control drillhole GC218, a certified blank sourced from Geostats (GLG912-2, 0.00254g/t Au) has been inserted in to the sample stream, which results in an overall insertion rate for the program of 1.2%. SRK has reviewed the results from the blank sample analysis, and (excluding a few high grade anomalies) has determined that in general there is little evidence for significant contamination at the preparation facility. However, SRK note that 8% of the results are >0.1 ppm, which suggests that there may be some low-level contamination effecting the sample results, which SRK has recommended should be monitored by the Company during on-going grade control programmes. The blank sample analysis chart is presented in Figure 12-3.

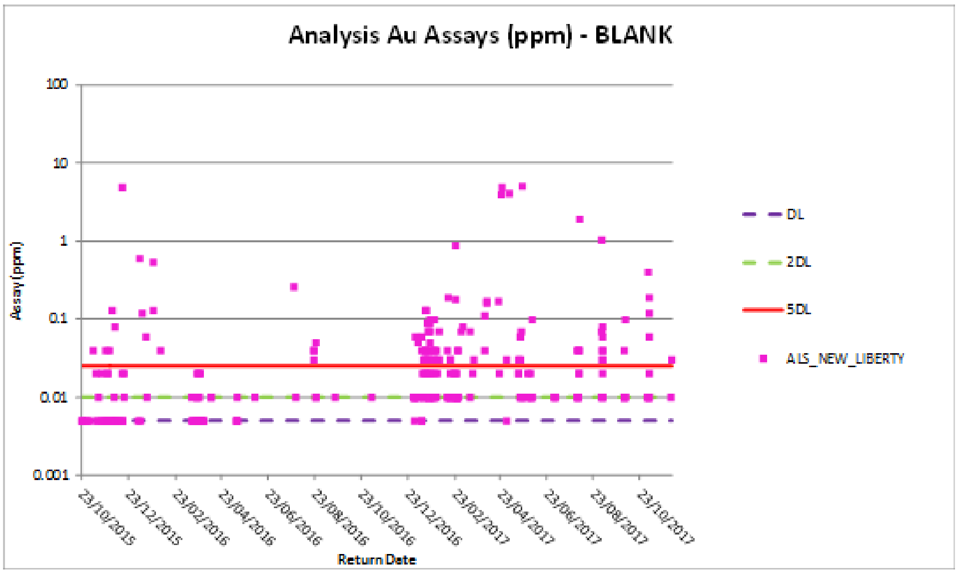


Figure 12-2: QAQC Blank Summary Chart for gold from submission of New Liberty Grade Control Samples

Duplicates

Reverse circulation (RC) field duplicates were inserted into the routine sample stream at a rate of approximately 5%. Excluding a small number of anomalies, the results for gold show a reasonable correlation to the original assays with correlation coefficient in excess of 0.9. There is however a general scatter of duplicate data either side of the X=Y line (with no obvious overall bias) which SRK consideres to be a reflection of the geological variability and resultant inhomogeneous distribution of the mineralisation in the grade control samples.

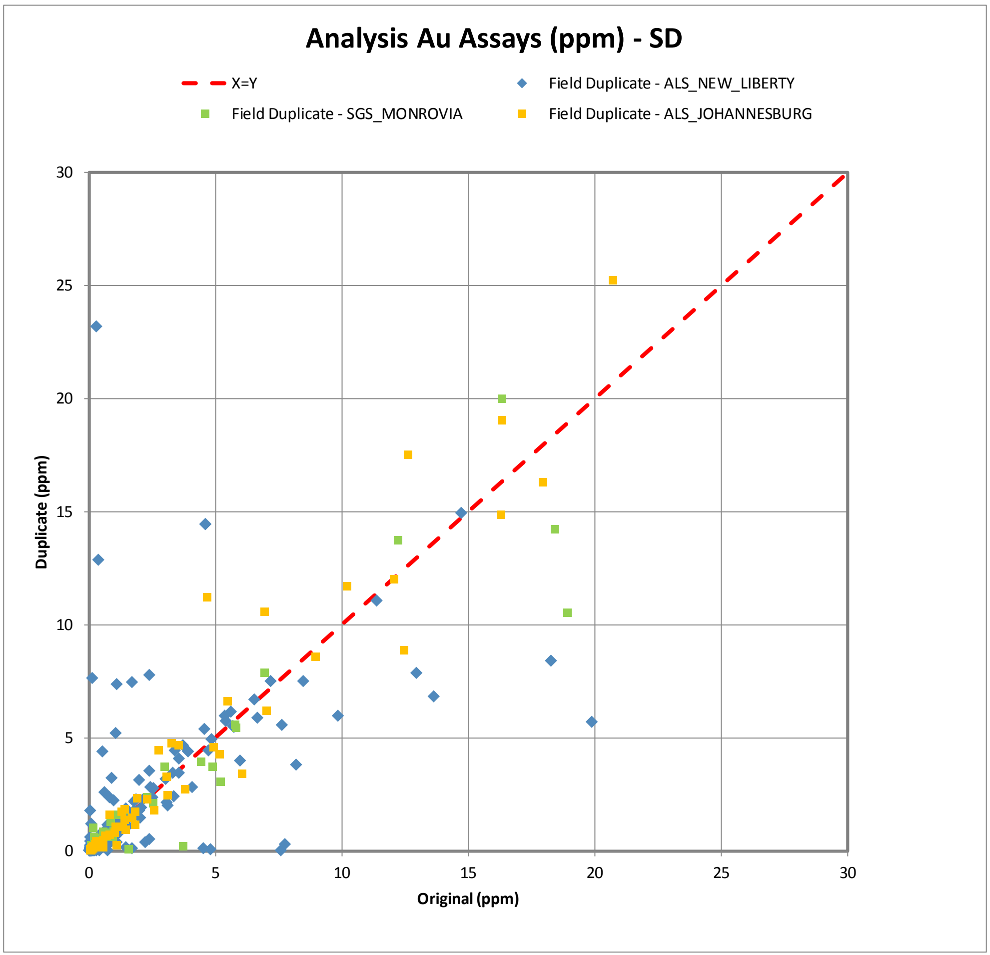


Figure 12-3: QAQC Field Duplicate Summary Chart from submission of New Liberty Grade Control Samples

12.3.7 SRK Comments

SRK has reviewed the data collection methodologies during its site visits, and has undertaken a review of the assay and geology database during the Mineral Resource estimation process (Section 14).

SRK's assessment of the available QAQC data indicates in general a steady improvement in standard 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 some data subsets where procedures were less comprehensive.

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 were exacerbated by the low proportion of routine QA/QC samples within mineralised material and the distance between the Project site and the laboratory.

SRK notes that the apparent low bias in the exploration drilling has only been partially explained or resolved, however, acceptable accuracy of the CRM results from the close-spaced grade control drilling (combined with use of multiple samples for grade estimation, which acts to smooth the slight issues relating to precision and sample-swaps) has increased the confidence in block grade estimates ahead of mining.

To maximise confidence in the grade control database, SRK has recommended further investigating (and rectifying) the CRM swaps and reduction in analytical precision noted since the commissioning of the ALS NLGM laboratory and re-submitting 5-10% of sample pulps analysed at ALS NLGM to an umpire laboratory to further verify analytical performance.

Certainly SRK considers that improved performance would be required (based on the CRM results noted above) at ALS NLGM should the Company consider using this laboratory for analysing future exploration drilling (in to less well sampled areas of the model).

In summary, and notwithstanding the comments above in relation to ALS NLGM, SRK is confident that the assay data provided by the Company is of sufficiently high quality, and has been subjected to a sufficiently high level of checking, to support the Mineral Resource estimates presented in this report at the confidence levels that have been assigned.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Background

This section highlights the previous testwork conclusions, details the historical plant performance and outlines the projected plant performance due to the circuit modifications. Details of the plant and the modifications to this are given in Section 17.

The details of metallurgical test undertaken during the exploration stage and as input to the initial feasibility studies are documented within the previous Technical Report on Updated Mineral Resources and Mineral Reserves dated 22 October 2012.

The details of the metallurgical test work undertaken by ALS Laboratories (ALS), Perth, Australia, and completed as part of the optimisation phase of the Feasibility Study are documented within the previous 2015 Technical Report.

The process plant was commissioned during 2015 and the initial operation was problematical for a variety of reasons. These issues can be summarised as availability of planned ore, high proportion of oxide material in the ore feed, oversize feed to the ball mill resulting in inefficient grinding and excessive stone discharge from the ball mill trommel, poor grinding ball quality, ball mill liner and grate material problems, under-utilisation of the Vertimill, excessive wear in the grinding circuit, gravity circuit feed screen capacity problems, lower than expected gravity circuit gold extraction due to insufficient gravity concentrator capacity, oxygen plant operational problems; oxygen sparging issues in pre-oxidation and CIL tanks, poor CIL leach extraction, poor carbon management in the CIL circuit, cyanide detoxification circuit performance issues, and availability of reagents and maintenance spares.

A number of plant modifications have been implemented to address these issues and further changes are planned for implementation in late 2017. These changes should reduce plant downtime, enhance plant throughput and improve plant performance to beyond the levels assumed in the Feasibility Study.

13.2 Test Work Samples

The Feasibility Study metallurgical optimisation test work programme was performed on both composite and variability samples.

The bulk composite sample used in the ALS metallurgical testwork programme was comprised of pre-crushed core samples collected from the western and central portions of the deposit from 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.

The distribution of the optimisation phase metallurgical composite test sample drill holes are presented in Figure 13-1 below (as circled in red).

In addition to the test work conducted on the master composite sample, further variability test work was conducted on samples using the optimised flowsheet and reagent consumptions. A total of eleven (11) variability samples were used. The ore variability samples represented various spatial locations distributed throughout the deposit with gold grades ranging from between 4.26 g/t and 10.44 g/t. The spatial distribution of the variability samples is shown (circled in red) in Figure 13-2.

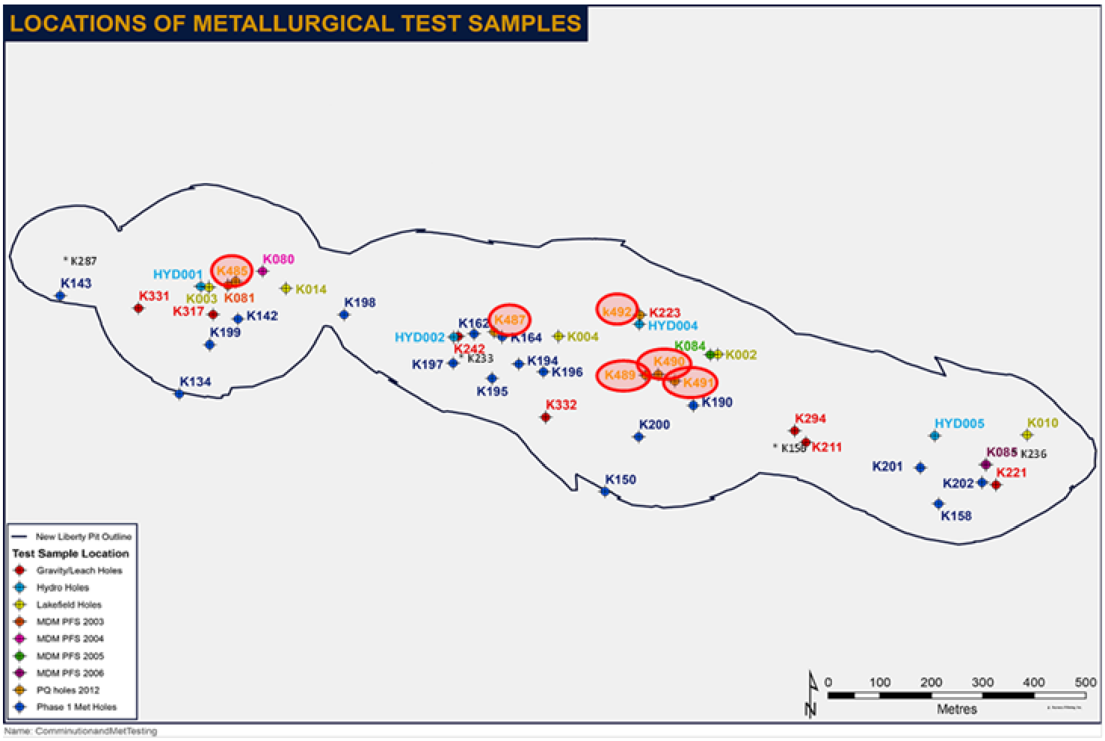


Figure 13-1: Optimisation Phase Distribution of Composite Test Sample Drill Holes

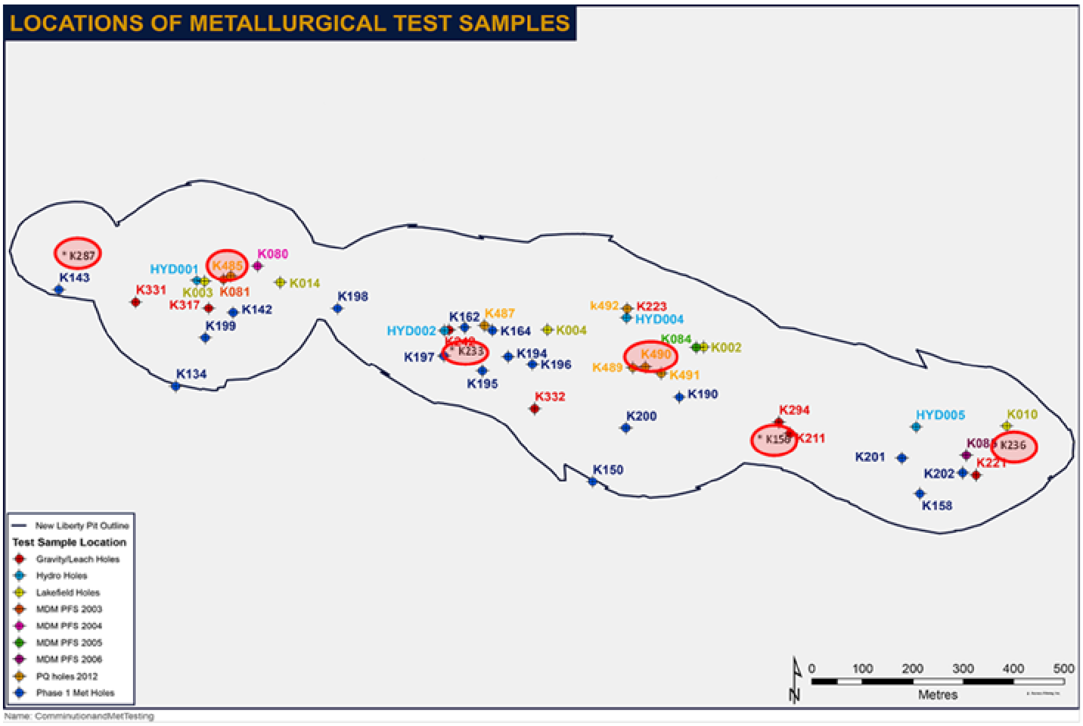


Figure 13-2: Optimisation Phase Distribution of Variability Test Sample Drillholes

13.3 Leach Optimisation Test Work

13.3.1 Leach residence time

Gold leach extraction versus residence time results for the ALS master composite for different grind sizes are shown in Figure 13-3. The gold leach extraction versus residence time results for the ALS variability tests (which were conducted at a target grind of 80% passing 50 µm) are presented in Figure 13-4.

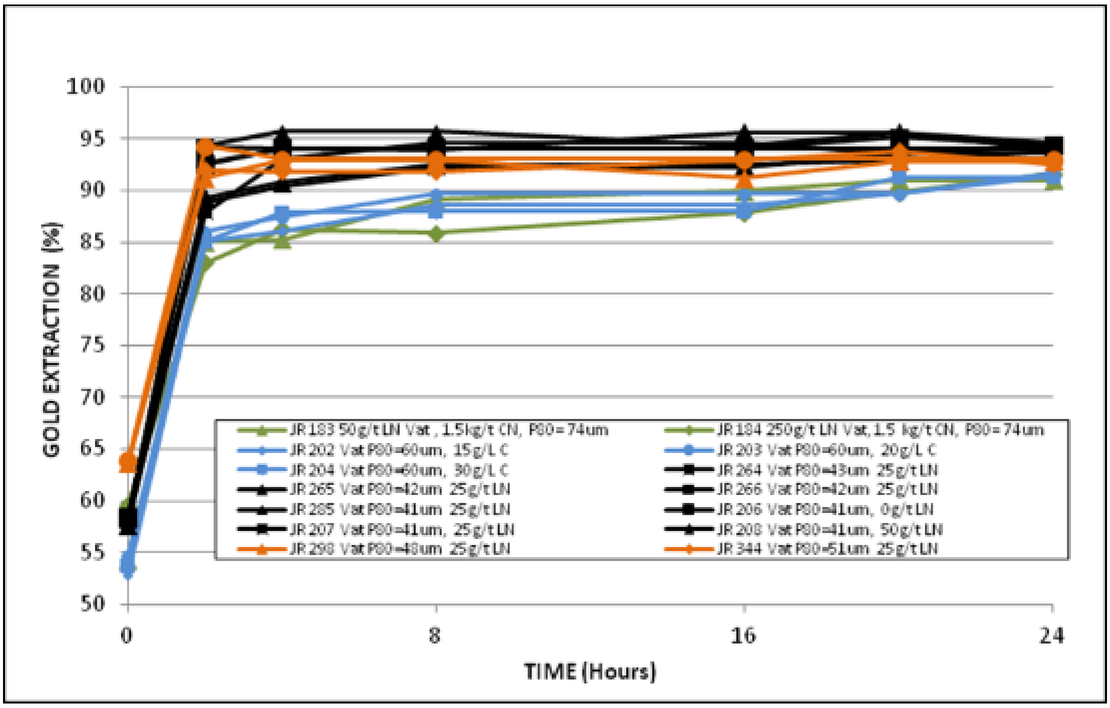


Figure 13-3: Effect of Target Grind Size on Gold Extraction for the New Liberty Master Composite Sample

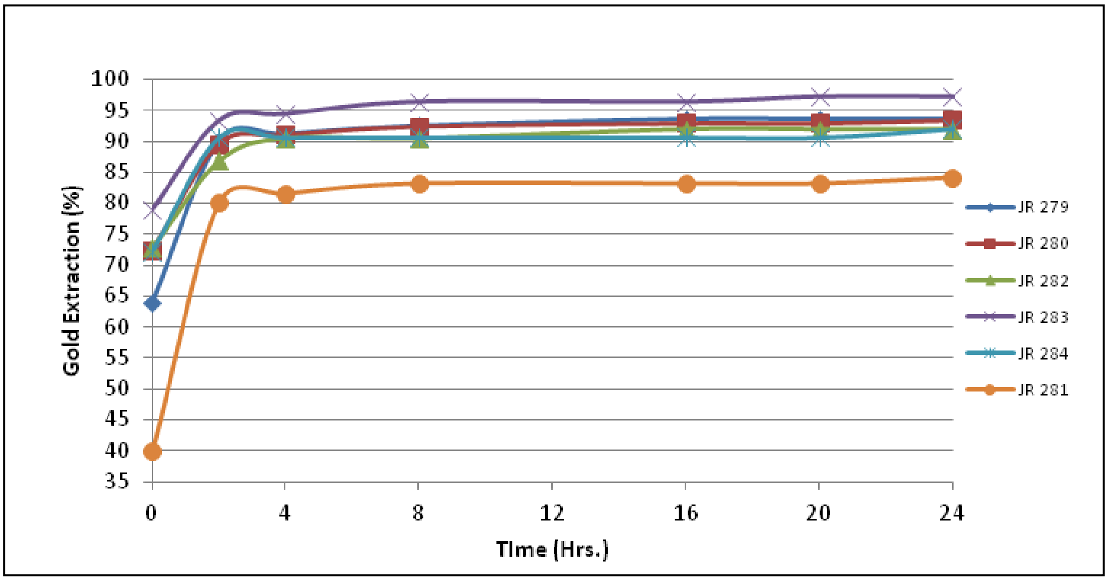


Figure 13-4: Gold Recovery for New Liberty Variability Tests Conducted at a Target Grind Size of 80% Passing 50 µm

Both graphs demonstrate that the leaching is completed relatively quickly, over 80% extraction in the first 4 hours and is essentially complete between 8 and 16 hours depending on the grind size. As the residence time of the existing six tank CIL circuit is over 21 hours at 146 tph and 19 hours at 200 tph the leach extraction should not be an issue with an optimised circuit.

13.3.2 Evaluation of Preg-Robbing

Leaching preg-robbing tests using master composite gravity tailings, performed as part of the leach optimisation testwork, demonstrated that preg-robbing was not an issue.

13.3.3 Effect of high-shear, pre-treatment with oxygen in comparison to the feasibility flowsheet performance.

Comparative leach testwork on composite samples ground to 80% passing 75 µm showed that the best leach extraction of 91.7% was achieved with 4 hours of high shear pre-oxidation followed by 24 hours of CIL. Leach tails grades were found to be between 0.35g/t and 0.40g/t. The high shear oxygenation resulted in dissolved oxygen levels of around 16 ppm.

13.3.4 Optimisation of Cyanide Addition

Leach kinetic tests performed on the composite sample, ground to 80% passing 75 µm, pre-treated with 4-hours high shear pre-oxidation, at cyanide addition of 1.5 kg/t, 1.0 kg/t and 0.5 kg/t demonstrated that similar recoveries and kinetics could be achieved at the lower cyanide addition of 0.5kg/t compared to tests with 1.0 kg/t and 1.5 kg/t. The leach residue grades were in the range of 0.40g/t to 0.43g/t.

13.3.5 Lead Nitrate Addition

The effect of lead nitrate addition on leach kinetics was evaluated and an addition rate of up to 25g/t was identified. Higher dosages did not provide an improvement in recovery or leach kinetics.

The optimised lime consumption was found to be in the range of 0.88kg/t to 2.13 kg/t with an average consumption of 1.48kg/t.

13.3.6 Determination of Optimum Grind

Grind-leach optimisation testwork to investigate the effects of finer grinding indicated that overall gold extraction increased by up to 2.8% when the fineness of grind increased from 80% passing 75 µm to 80% passing 42µm. The following results were achieved:

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

13.4 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 identified in the grind-leach optimisation work.

Bond Work Index

The Bond Work Index for the master composite was determined as 18.8 kWh/t at a test aperture of 106 µm.

Levin and IsaMill Testing – Vertimill energy requirement

Fine grinding testwork was undertaken to determine the specific energy requirements for a regrind application. Levin and IsaMill testing procedures were used. Based on the results from both test methods and the interpretation by Metso for the application of a Vertimill type regrind mill, the energy requirement was determined as 6.7 kWh/t of regrind feed to achieve the target grind of 80% passing 50 µm, utilising 12.7 mm diameter media.

13.5 Evaluation of Leach Feed Density

13.5.1 Introduction

45% solids by mass was identified as the maximum solids concentration in the leach feed.

13.5.2 Diagnostic Leach Tests

Multi-stage sequential diagnostic gold leach test work was conducted on two subsamples of the mater composite sample. The testing confirmed that over 90% of the gold is free milling and recoverable by a combination of gravity and cyanidation and that up to 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.

13.6 Variability Test Work

13.6.1 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.

The following optimised leach conditions were used:

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

13.6.2 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 are presented above in Figure 13-4.

The actual grind for the variability samples varied from 80% passing 47µm to 80% passing 71µm.The overall gold recovery for the six variability composite samples ranged from 83.3% to 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% to 79.0%, while the CIL recovery ranged from 70.5% to 86.7%. The leach residue grades achieved ranged from 0.22 g/t to 0.71 g/t.

Checks on the particle size distribution achieved in the sample preparation for the gravity-leach fine grinding testwork showed that size distribution curves for samples 80% passing 50 microns were similar to that normally seen in regrind applications and thus the results can be evaluated in the context of a full scale primary grinding and Vertimill regrind application.

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

For the gravity-leach tests where the regrind size was 80% passing <20 µm, the overall gold recovery for the variability composite samples ranged from 93.5% to 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% to 81.4%, while the CIL recovery ranged from 80.0% to 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.

For gravity-leach tests at a finer grind of 80% passing 20 µm, it was apparent that the size distributions 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, the recoveries at this finer grind are probably overstated for a full-scale application and are not considered reliable.

13.7 Selection of Mill Grind at 80% Passing 45 µm

While not specifically tested, the target regrind of 80% passing 45 µm was selected as the PSD. From the regrinding testwork this demonstrated that the regrind mill would produce a slightly finer output. As this would be slightly finer than the testing at 80% passing 50 µm, 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 and may be very slightly enhanced.

13.8 Gravity Recoverable Gold Testwork

As part of the optimisation phase, Extended Gravity Recoverable Gold (eGRG) testing was carried out by Consep in Australia.

The overall Gravity Recoverable Gold (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 and 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 operation and estimated by Consep to be between 38% and 46% with an expected plant GRG recovery of 41%.

It is also noted that test work on composite samples at ALS showed gravity gold recoveries ranging from 51% to 63%, and on variability samples from 38% to 81%.

13.9 Cyanide Destruction and Arsenic Precipitation Test Work

13.9.1 SO_2/Air Cyanide Destruction Test Work

Cyanide destruction testwork was conducted on the product of bulk leach tests from the master composite at a cyanide addition of 0.5 to 1.5 kg/t to investigate the SO_2/Air process. At 1.5 kg/t cyanide addition, the CN_{WAD} level in the leach effluent stream was around 163.8 ppm and the test performed showed that an SO_2:CN ratio of 4:1 was not sufficient to reduce CN_{WAD} levels in the cyanide destruction product stream to below 50 ppm. Additional SO_2 would be required.

Subsequent testwork based on the optimised leach conditions indicated the leach effluent stream is expected to have a CN_{WAD} level of 50 to 100 ppm. Based on the results of this test work, the SO_2/Air process will be able to produce a cyanide destruction effluent stream CN_{WAD} level of less than 50 ppm, as per the requirements of the international cyanide code of practice.

This was used as the basis for the cyanide detoxification process included in the final plant design.

13.10 Arsenic Precipitation Tests

Arsenic leaching and precipitation testwork was performed on the composite sample under different conditions to assess the long term stability arsenic precipitates in slurry as would report to the TSF. The testwork indicated that stable arsenic precipitates could be produced from slurry samples with an arsenic in solids content around 1,200ppm 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 detoxification. The flowsheet, 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.

The kinetic column test returned an arsenic in solution value of 0.034ppm after 8 weeks and less than the target value of 0.1ppm after 23 weeks. Further results have not been reported.

Based on the kinetic column test results, the process plant design was updated to allow for a 1,000m^3 tank in which cyanide destruction and arsenic leaching take place. The additional 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.

13.11 Metallurgical Recovery and Historical Performance

13.11.1 Introduction

During the Feasibility Study stage, the gold recovery testwork results at different grind sizes was evaluated statistically. Historical Mintek testwork and the ALS testwork performed as part of the Feasibility Study were used. In order to provide an estimate of the expected recovery for full scale continuous plant operations, the bench scale laboratory recoveries were discounted in order to account for process inefficiency and solution gold losses due to:

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

13.11.2 Derivation of a Correlation between Grade, Recovery and Mill Grind

Based on this evaluation, the following feed grade-recovery correlations at each target grind size were developed:

- Final Residue (P_{80} 75µm) = 0.217 x (Head Grade)$^{0.35}$
- Final Residue (P_{80} 60µm) = 0.187 x (Head Grade)$^{0.35}$
- Final Residue (P_{80} 50µm) = 0.167 x (Head Grade)$^{0.35}$
- Final Residue (P_{80} 42µm) = 0.151 x (Head Grade)$^{0.35}$
- Final Residue (P_{80}<20µm) = 0.107 x (Head Grade)$^{0.35}$

This modelled grade–recovery relationship is graphically presented in Figure 13-5.

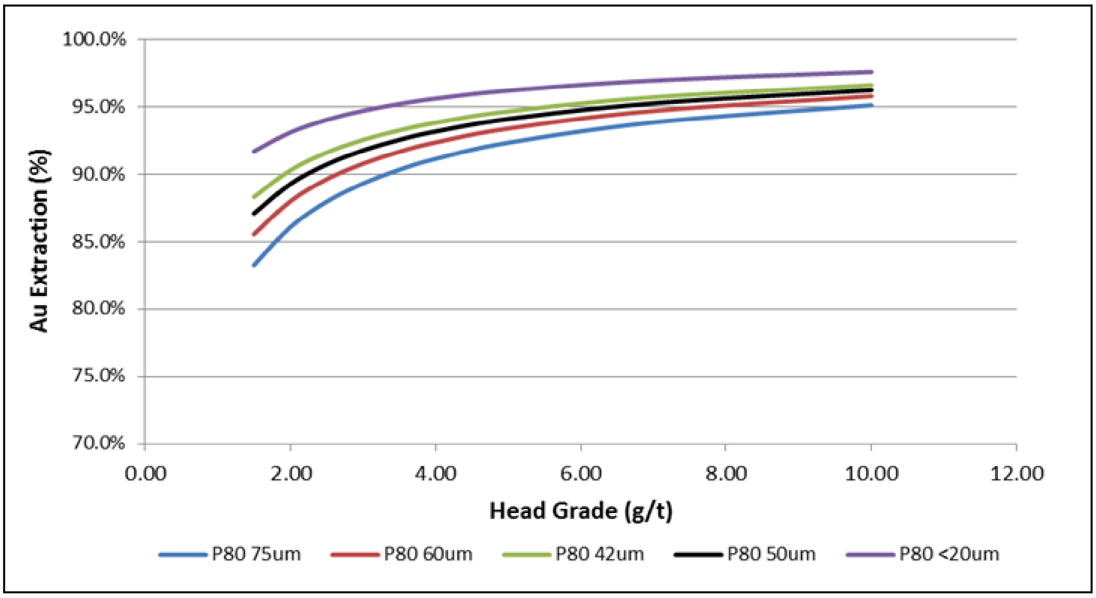


Figure 13-5: Model Predicted Grade Recovery Curve at Each Target Grind Size

The grade recovery relationship as predicted by the model as derived above was found to be in good agreements with the Mintek (Phase 1, 2 and 7) and the ALS test work.

A Monte Carlo analysis was performed on the recoveries as determined from these grade recovery models (at a 90% confidence recovery range) and confirmed that they were acceptable.

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.

13.11.3 Historical Plant performance

Since start-up in 2015, the plant performance has been below the anticipated Feasibility Study figures. A summary of plant performance data is given in Table 13-1.

Table 13-1: Plant performance data

	Feed Rate (tph)	Feed Grade (g/t)	Tailings Grade (g/t)	Gold Recovery (%)	Plant Utilisation (%)
Target 2016	146	3.38	0.30	91.0%	92%
H1 2016	143	2.86	0.51	85.0%	57%
Q3 2016 Oct	136	2.83	0.59	79.0%	74%
Oct-16	134	2.04	0.40	85.7%	88%
Nov-16	136	1.82	0.30	86.6%	96%
Dec-16	130	2.43	0.26	90.6%	89%
Target 2017	148	2.55	0.36	91.0%	92%
Jan-17	143	1.62	0.23	85.0% (1)	93%
Feb-17	136	2.05	0.22	90.0%	89%
Mar-17	141	2.24	0.24	90.8%	95%
Apr-17	142	1.82	0.23	90.0%	84%
May-17	144	1.93	0.27	84.0%(2)	87%
Jun-17	164	2.19	0.25	89.3%	97%
Jul-17	164	2.62	0.26	90.2%	69%(3)

(1) Mainly partly oxidised low grade ore fed to the plant in the month due to shortage of fresh ore from the pit, hence the lower recovery, which was expected.

(2) Low recovery due to treatment of some ore from Eastern Marvoe and high proportion of oxide material in the feed Historical testwork indicates that gold extraction from this material is approximately 83%.

(3) No ore for 9 days resulted in low plant utilisation in July 2017.

It is noted that the plant feed grades have been consistently below the target RoM grade.

In recent months, the tailings grade has been below plan, which is an indication that the overall leach residence time is sufficient.

The general improvement in the hourly plant throughput, the overall plant utilisation and the CIL tailings grade during the final quarter of 2016 and 2017 to date, compared to the first 9 months of 2016, is evident. Over this period the operating parameters of the CIL circuit, such as carbon inventory, carbon concentration profile across the six CIL tanks, and the CIL solution tailings grade, have been more consistent indicating that the plant operation is more stable than previously.

13.11.4 Planned Plant performance

Plant modifications identified in Section 17 of this report should result in improved plant operating hours and plant metallurgical performance. BMMC is assuming the following plant parameters going forward:

- Throughput: c.1.7 Mtpa, 200 tph at 80% passing 50 µm.
- Plant operating time: 93% of total time.
- Gold recovery will be dependent on feed grade as shown in Table 13-2.

Table 13-2: Target Gold Recovery Vs Feed Grade

Feed grade g/t	Target gold recovery %
1.5	85.5%
2.0	89.0%
2.5	91.0%
3.0	92.5%
4.0	93.0%

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource statement presented here has been prepared by SRK and is based on some 115,984 m of drilling for a total of 1,306 drillholes and 25 channels for 1,574 m of sampling. Further description of the data utilised to produce this statement is presented above in Chapter 10. This Mineral Resource model used to report this statement also forms the basis of the Mineral Reserve and mine plan described in subsequent sections of this report. The effective date of the Mineral Resource statement is 31 July 2017 and it excludes material already mined by this time.

14.2 Resource Estimation Procedures

The resource estimation methodology involved the following procedures:

- database compilation and verification;
- construction of wireframe geological models and definition of resource domains;
- data conditioning (compositing and capping review) for statistical analysis, geostatistical analysis;
- variography, block modelling and grade interpolation;
- resource classification and validation;
- assessment of "reasonable prospects for economic extraction" and selection of appropriate reporting cut-off grades; and
- preparation of the Mineral Resource Statement.

14.3 Resource Database

SRK was supplied with a Microsoft Excel Database from the Company. The database comprises all sample data for the Project completed up to 18 January 2016. Since then, additions to the sample database utilised by SRK have been limited to the Marvoe deposit area to reflect the Company's current focus on mining within this area. The files supplied had an effective cut-off date of 04 August 2017. The database has been reviewed by SRK and imported into Datamine to complete the Mineral Resource Estimate. SRK is satisfied with the quality of the database for use in the construction of the geological block model and associated Mineral Resource Estimate.

14.4 Statistical Analysis – Raw Data

An initial global statistical analysis was undertaken on the raw drill data. Summary statistics, incremental and log histograms were calculated. The skewed log normal distributions for gold are shown in Figure 14-1, with the separate populations noted in the gold assays relating to background host rock and low and high grade zones.

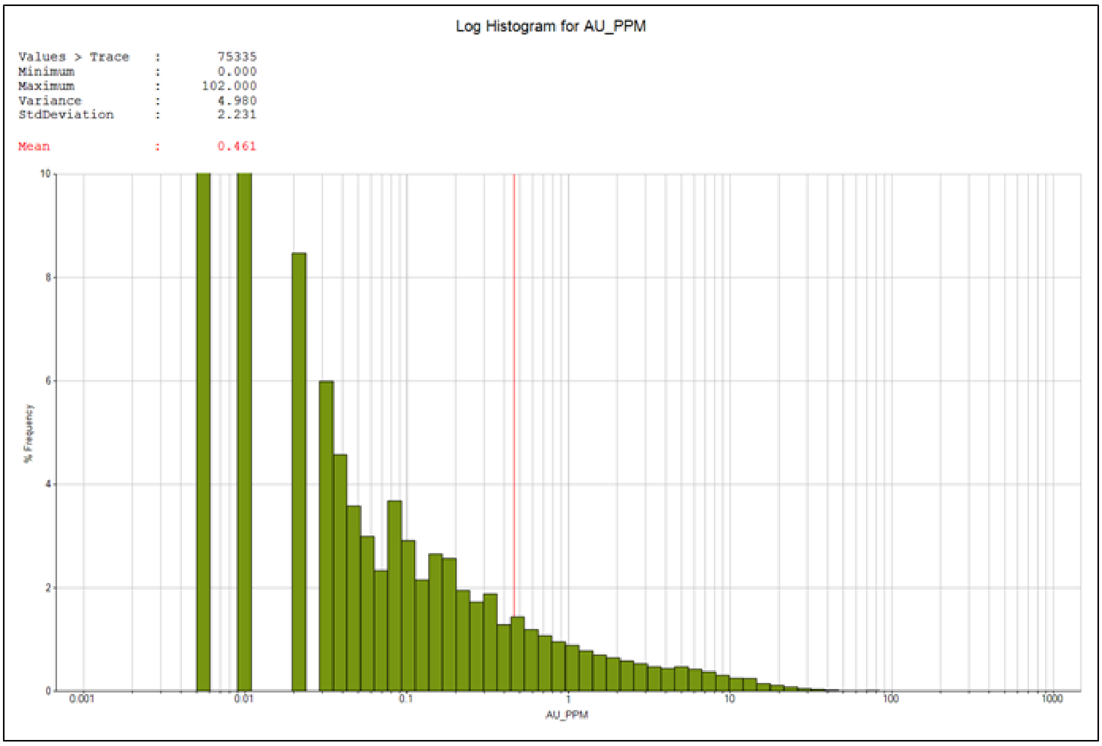


Figure 14-1: Log Histogram of Length Weighted Project Gold Assays

14.5 3D Modelling

For this update, based on available drilling data, including infill grade control drilling and initial geological interpretations provided by the Company, SRK has modelled the following geological units for the deposit:

- Silicified Metamorphosed Ultrabasic Suite (SMUS) zone;
- Base of Weathering;
- Gold Mineralised Structures.

14.5.1 Geological Wireframes

SMUS zone

The SMUS zone is the host rock to the gold mineralisation and has been modelled primarily based on lithological logging. The contacts of the SMUS with the hangingwall and footwall complexes have been used as a guide to constrain the lateral extents of the mineralisation wireframes. SRK created a 3D solid wireframe from selected lithological intervals using the Leapfrog Geo Software ("Leapfrog").

Base of Weathering Contact

A base of weathering contact has been modelled as a surface based on geological logging, with resultant model zones defined as 'weathered' or 'fresh'. The vertical thickness of the weathered zone typically ranges between 5m and 15 m.

14.5.2 Mineralisation Wireframes

Gold Mineralised Structures

The mineralisation domains developed by SRK have been defined primarily based on elevated gold grade and visual assessments of geological and grade continuity. Selected mineralised intervals were typically above 0.3 g/t Au, which provides a visually appropriate boundary between the mineralised zones and background host rock, with lower grade samples incorporated where necessary to honour geological continuity. SRK created 3D solid wireframes from selected sample intervals using hangingwall and footwall surfaces in Leapfrog, with KZONE numerical codes (1-7) used to differentiate between spatially separate mineralised corridors.

At the Marvoe deposit area, observations during mining to date have highlighted the presence of discrete high and low grade zonation within the main modelled mineralised corridor, with higher grade zones (and the highest gold grades, which are typically located in the domain hangingwall) interpreted to relate to areas of increased shearing. SRK has modelled the high grade zones using Leapfrog grade shell interpolations (constrained within the mineralised corridor), based on a range of cut-off grades and structural orientations to assess the associated continuity. Grade shell interpolations were run based on 4m composite samples to help define contiguous grade zones without being overly influenced by high grade variability between 1m sample results.

The most visually representative scenario for the higher grade zones at Marvoe was selected at a 0.5 g/t Au cut-off. The lower grade zones within the mineralised corridor (<0.5 g/t Au) were excluded from the updated model to reflect experience at Marvoe to date whereby the grade continuity of any higher grade mineralisation within these lower grade zones is typically considered too poor from a practical mining perspective.

Statistical Analysis

Modelled domains were checked to ensure they formed appropriate sample populations for grade estimation with respect to gold grade, with the presence of any bimodal populations noted to ensure appropriate representation during block grade interpolation. An example of the bimodal raw sample population for gold for the KZONE1 domain, which is a reflection of the presence of plunging high grade shoots within a (lower grade) mineralised zone, is illustrated in Figure 14-2. Furthermore, the presence and variable orientation of the grade shoots were investigated using isotropic grade shell interpolations in Leapfrog, as shown in Figure 14-3.

Visual and statistical assessments of gold grade data above and below the base of weathering suggested limited differences in the sample populations and therefore SRK has treated the oxidised and fresh material as a single domain for grade interpolation purposes.

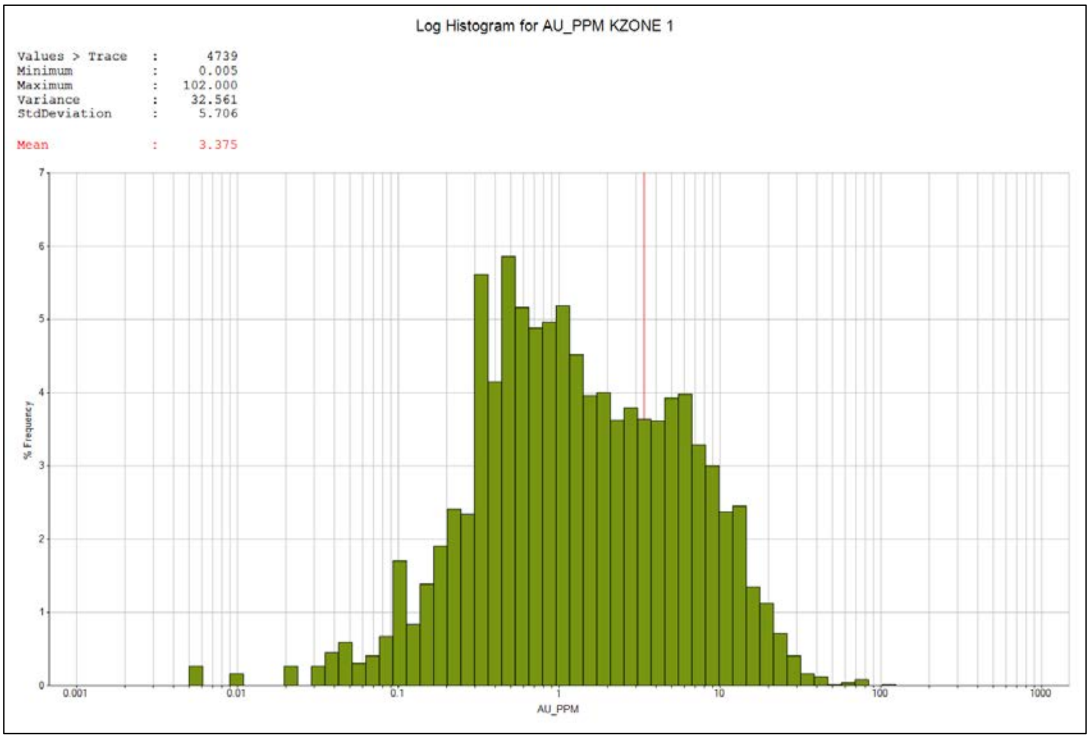


Figure 14-2: Log histogram plot for gold for mineralisation domain KZONE1

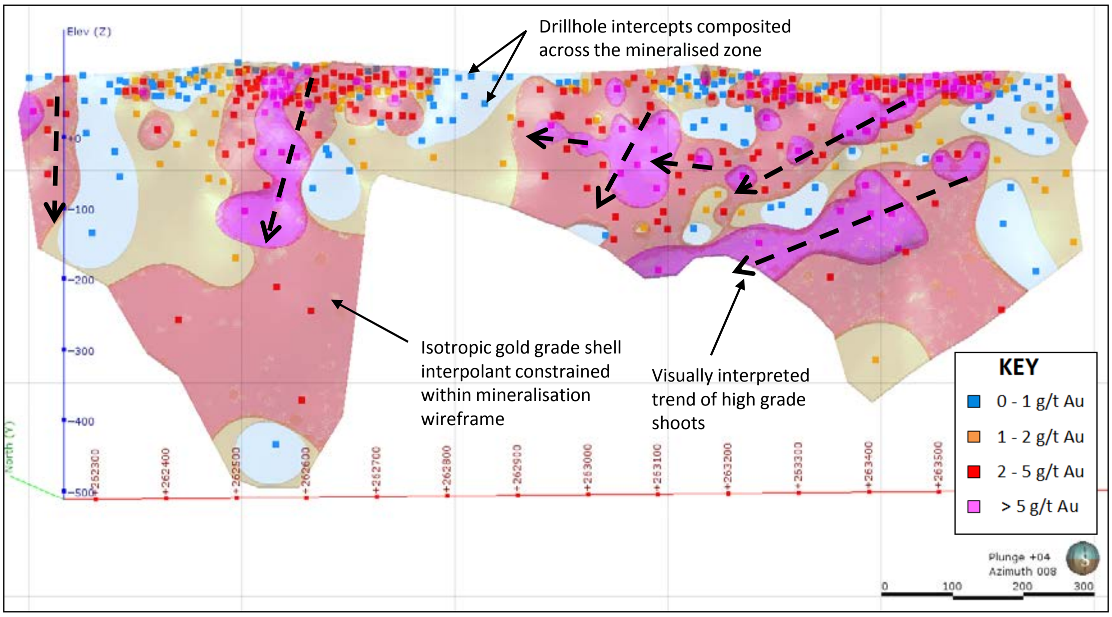


Figure 14-3: Assessment of high grade shoot orientation for mineralisation domain KZONE1 (looking north)

14.5.3 Mineralisation Model Coding

A summary of the mineralisation domains versus statistical (GROUP) and estimation (KZONE) zone code and modelled wireframe name for New Liberty is provided in Table 14-1.

Figure 14-4, Figure 14-5 and Figure 14-6 provide example 3D illustrations of the modelled mineralisation wireframes, which have been reviewed by the Company and have been deemed acceptable for use in the MRE.

The mineralisation modelled comprises several separate zones which are each geologically continuous along strike for between 200m and 1.5km, have dip extents of up to 550 m and an average thickness normally between 4m and 10m, reaching over 20m in certain areas.

Table 14-1: Summary of Mineralisation Zones at the New Liberty Project

GROUP	KZONE	Wireframe	Deposit Area	Description
100	1, 3-7	Gold mineralised structures (k1_tr, k3_tr - k7_tr)	New Liberty (Larjor, Kinjor and Marvoe)	Gold mineralised corridors hosted within the Silicified Metamorphosed Ultrabasic Suite (SMUS); selected mineralised intervals were typically above 0.3 g/t Au
100	2	High-grade gold mineralised structure (k2_hg_tr)	New Liberty (Marvoe)	High-grade gold zones hosted within the main mineralised corridor at Marvoe, hosted by the Silicified Metamorphosed Ultrabasic Suite (SMUS); selected mineralised intervals were typically above 0.5 g/t Au

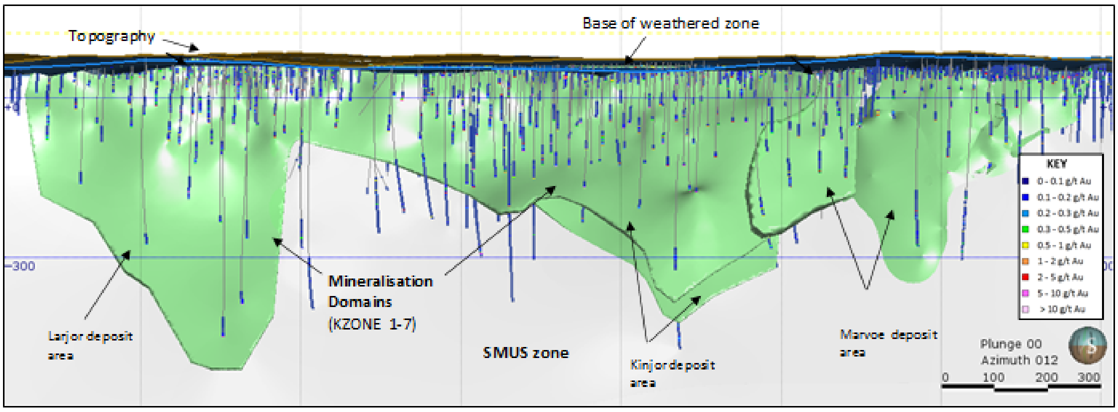


Figure 14-4: New Liberty Mineralisation Model: Long Section, looking north

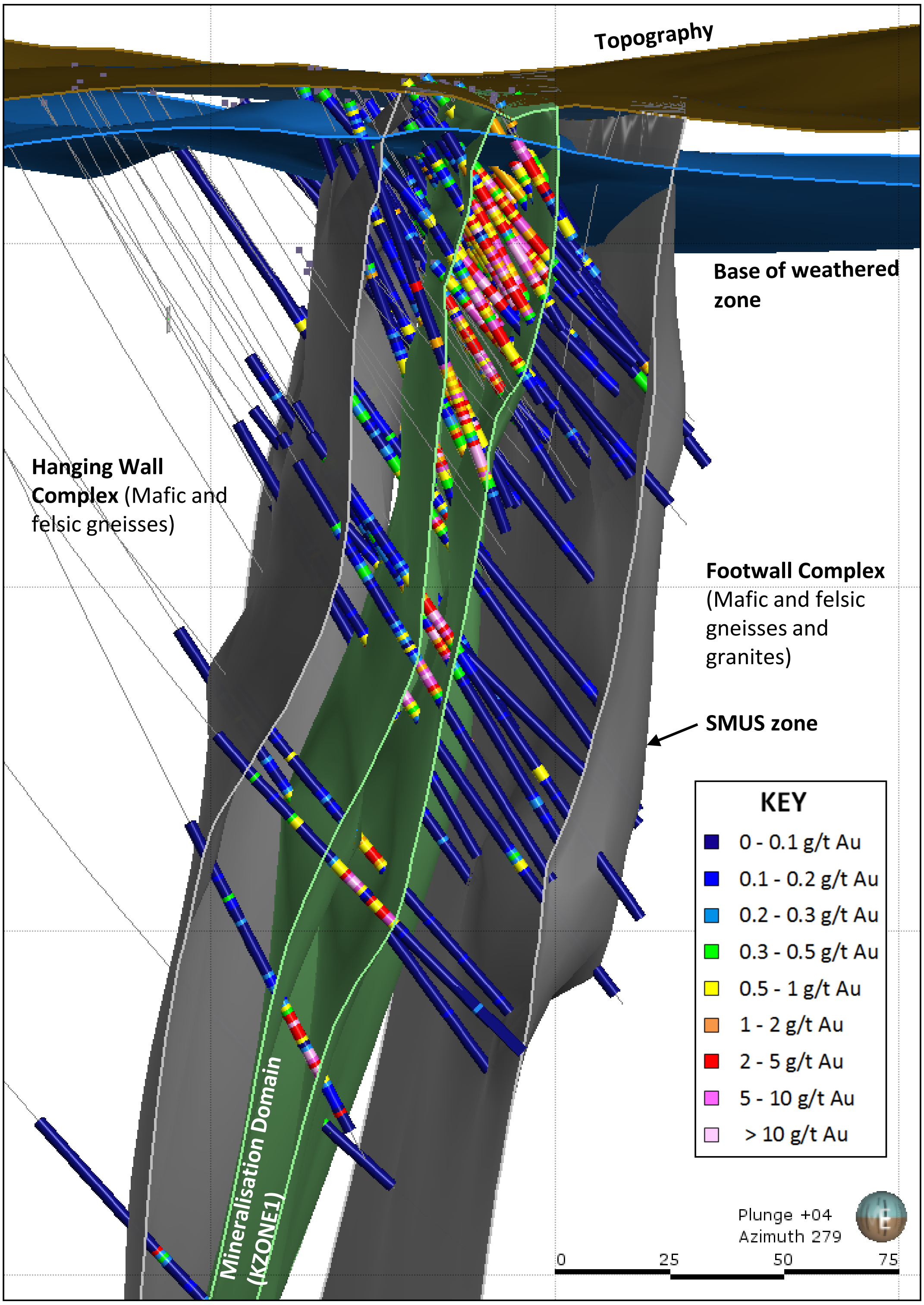


Figure 14-5: New Liberty Mineralisation Model (KZONE1): Cross Section, looking west

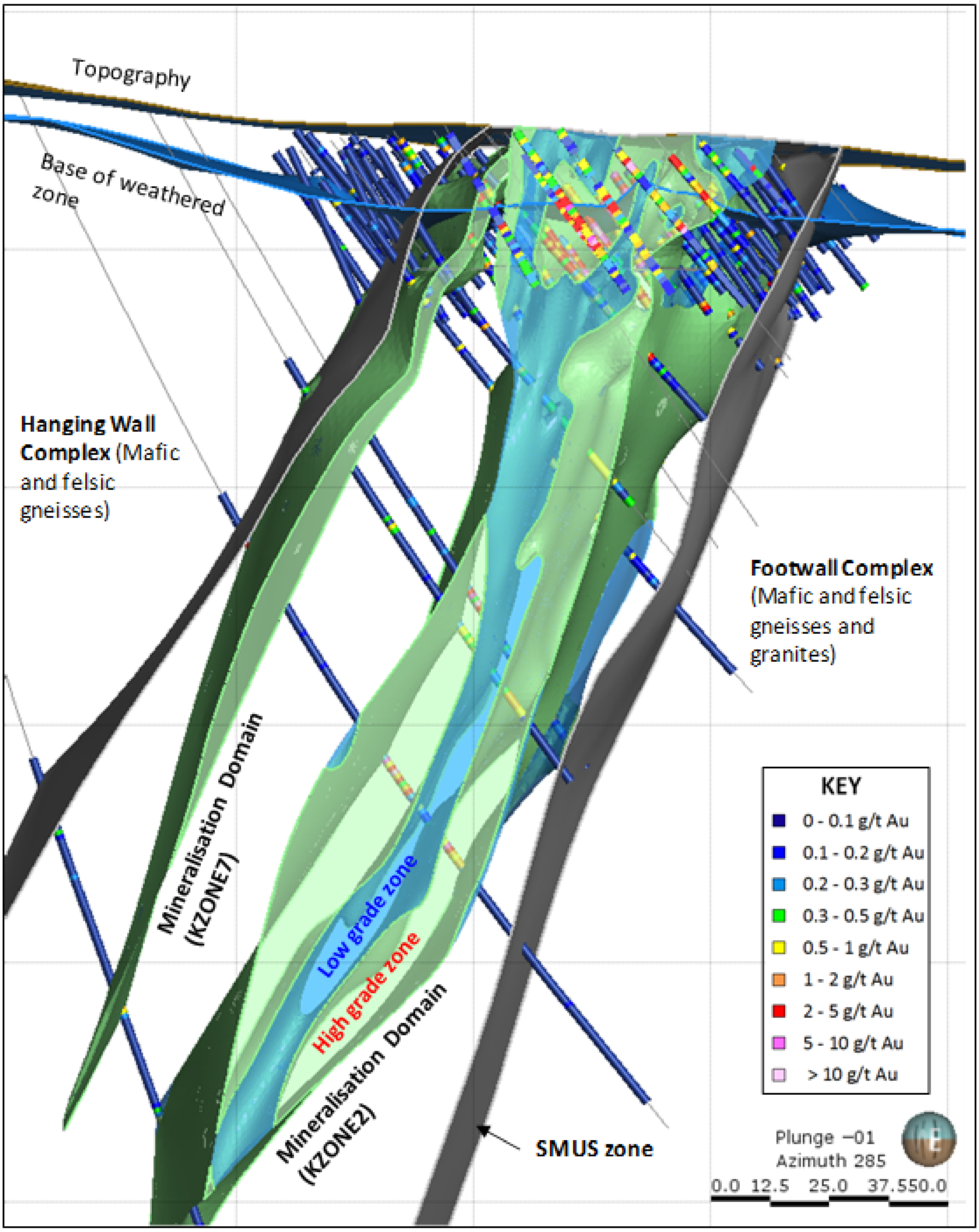


Figure 14-6: New Liberty Mineralisation Model at Marvoe (KZONE2): Cross Section, looking west

14.6 Compositing

The gold grade data shows that there are higher and lower grade areas within the deposit both along strike in the form of plunging grade shoots and as zonation from hangingwall to footwall, particularly within the thicker zones of mineralisation. Given this, SRK has created 1m composites to ensure sufficient resolution during block grade estimation whilst honouring the mean sample length.

14.7 Evaluation of Outliers

High grade capping is undertaken where very high grade data is considered to be unrepresentative of the main population and could bias the interpolation procedure.

SRK has completed the analysis based on log histograms (in context of a visual assessment for sample support) which can be used to distinguish the grades at which samples have significant impacts on the local estimation and whose affect is considered extreme.

Log histogram plots were created for each domain as illustrated for domain KZONE1 in Figure 14-7 by way of an example.

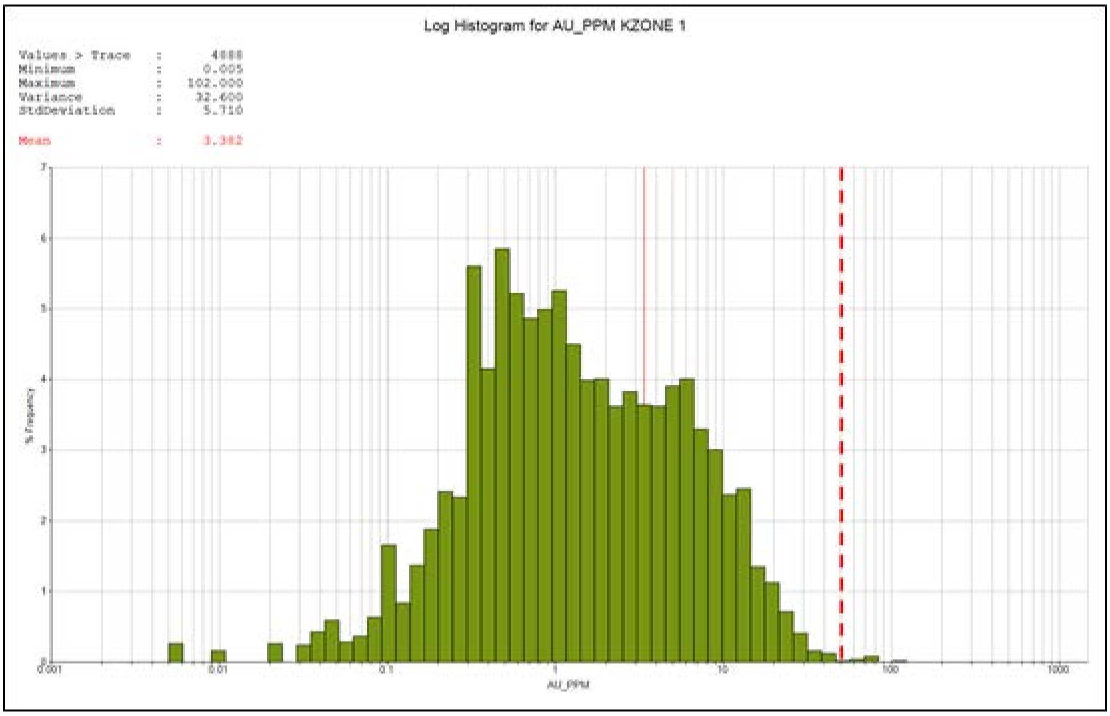


Figure 14-7: Log Histogram and Log Probability Plot for gold for the KZONE1 domain showing selected grade cap

Table 14-2 shows the selected capping limits and a comparison of the mean grades within each domain based on the grade capping applied. Prior to capping, high-grade samples have been visually checked to see whether they form separate populations.

The global reduction in the mean grade in capped zones is typically in the order of 1-3% which SRK deems to be within acceptable margins. SRK note a slightly higher percentage difference in the means for mineralisation domain KZONE 7, which is as a result of the pre-capped mean being skewed by a small number of comparatively high grade sample outliers.

Table 14-2: Comparison of Mean Composite Grades (Raw Composite versus Capped)

KZONE	FIELD	NSAMP	MIN	MAX	MEAN	CAP	VAR	STDDEV	COV	% DIFF	ABS MEAN DIFF
1	AU_PPM	4888	0.01	102.00	3.4	50.00	32.6	5.7	1.7	-1%	-0.04
	AUCAP	4888	0.01	50.0	3.3		27.7	5.3	1.6		
2	AU_PPM	3147	0.01	52.00	2.3	-	16.3	4.0	1.8	0%	0.00
	AUCAP	3147	0.01	52.0	2.3		16.3	4.0	1.8		
3	AU_PPM	1368	0.00	93.40	2.3	50.00	33.8	5.8	2.5	-3%	-0.07
	AUCAP	1368	0.00	50.0	2.2		25.4	5.0	2.3		
4	AU_PPM	66	0.04	8.12	1.1	-	3.1	1.8	1.6	0%	0.00
	AUCAP	66	0.04	8.1	1.1		3.1	1.8	1.6		
5	AU_PPM	101	0.04	10.71	0.7	-	1.6	1.3	1.9	0%	0.00
	AUCAP	101	0.04	10.7	0.7		1.6	1.3	1.9		
6	AU_PPM	334	0.01	53.76	5.1	-	66.2	8.1	1.6	0%	0.00
	AUCAP	334	0.01	53.8	5.1		66.2	8.1	1.6		
7	AU_PPM	657	0.01	86.40	1.5	20.00	27.4	5.2	3.4	-16%	-0.24
	AUCAP	657	0.01	20.0	1.3		6.6	2.6	2.0		

14.8 Geostatistical Analysis

Variography is the study of the spatial variability of an attribute, in this case gold grade. The Snowdon Supervisor Software ("Supervisor") was used for geostatistical analysis and the data has been analysed using a pairwise relative variogram in order to define variogram models of sufficient clarity. In completing the analysis for the mineralisation domains, variograms were modelled in the along-strike, down-dip and across-strike orientations, with a short-lag variogram calculated to characterise the nugget effect.

SRK treated the mineralisation domains KZONE's 1 and 3-7 as a single zone for variography to reflect their comparable geological characteristics and mineralisation styles. The variogram for the Marvoe high grade mineralisation domain KZONE 2 was modelled separately and calculated using a relatively narrow search cone in attempt to reduce the influence of sample data from the highest grades in the hangingwall spatially impacting on the assessment of grade continuity at the lower grade footwall and vice versa.

All modelled variances were re-scaled for each mineralised zone to match the total variance ('VAR') for that zone.

The pairwise relative variograms for the combined mineralisation domain for KZONE's 1 and 3-7 for gold is shown in Figure 14-8, with modelled parameters summarised in Table 14-3.

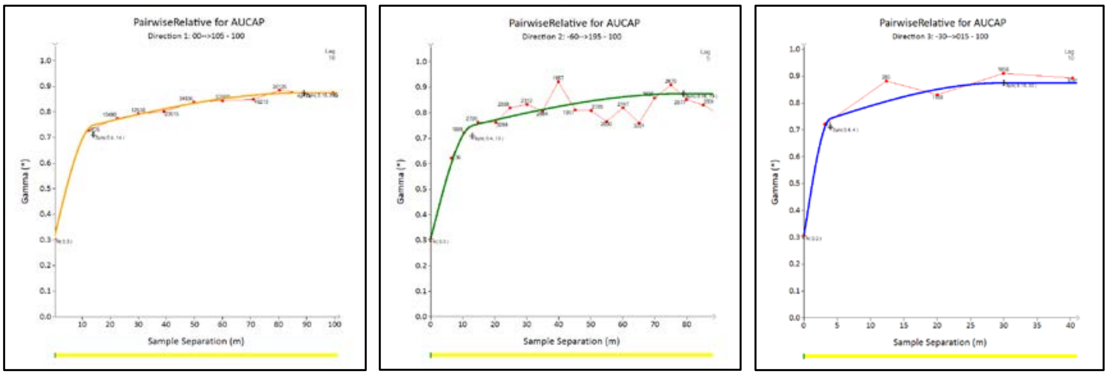


Figure 14-8: Summary of modelled semi-variogram parameters for the New Liberty Mineralisation domain (GROUP 100)

Table 14-3: Summary of semi-variogram parameters*

Variogram Parameter	AUCAP-GROUP100
Co	0.30
C1	0.40
A1 – Along Strike (m)	14
A1 – Down Dip (m)	13
A1 – Across Strike (m)	4
C2	0.16
A2 – Along Strike (m)	90
A2 – Down Dip (m)	80
A2 – Across Strike (m)	30
C3	0.00
A3 – Along Strike (m)	0
A3 – Down Dip (m)	0
A3 – Across Strike (m)	0
Nugget Effect (%)	35%

*Variogram structures are subsequently re-scaled to the total sample variance per estimation KZONE

14.9 Block Model and Grade Interpolation

A block model prototype was created for New Liberty based on UTM 29N coordinate system. Block model parameters were chosen to reflect the average spacing of the grade control drilling (along strike and on section) and to appropriately reflect the grade variability both along strike and from hangingwall to footwall.

To improve the geometric representation of the geological model, sub-blocking was allowed along the boundaries to a minimum of 1x1x1 m (x, y, and z). A summary of the block model parameters is given in Table 14-4.

Table 14-4: Details of Block Model Dimensions for the New Liberty Geological Model

Model	Dimension	Origin (UTM)	Block Size	Number of Blocks	Min Sub-blocking (m)
New Liberty (KZONE 1, 3-7)	X	261650.0	5	639	1
	Y	774380.0	5	357	1
	Z	-560.0	5	140	1
New Liberty (Marvoe KZONE 2)	X	261650.0	5	639	1
	Y	774382.5	2.5	712	1
	Z	-560.0	5	140	1

The Marvoe mineralisation domain KZONE 2 has a reduced block dimension (2.5m) in the across strike (y) axis when compared with the other domains at the project which use 5m; this narrower block size helps to reflect the more notable gradation in gold grade within this structure from hangingwall to footwall. After grade estimation, all model domains were re-blocked to 5.0x2.5x5.0 m using the block model framework shown for KZONE 2.

SRK notes that the selected block size is considered small when reviewed against the average drillhole spacing in the less well drilled areas of the deposit. However, SRK has accounted for this during grade interpolation by ensuring the use of sufficiently expanded search ellipses and elevated sample numbers for block grade estimates in areas away from the close spaced drilling.

14.10 Final Estimation Parameters

Ordinary Kriging (“OK”) was used for the gold grade interpolation. Search ellipses were orientated to follow the trend of each domain using Datamine’s Dynamic Anisotropy and domain boundaries have been treated as hard boundaries during the estimation process.

Based on visual assessments, SRK consider the orientation of these to be variable both on a deposit scale and within each mineralised domain. Therefore, SRK has used a relatively small search ellipse with equal dimension both along strike and down dip to allow block grade variability on a local scale without forcing a pre-determined plunge orientation.

At the Marvoe deposit area, given the observations from new close-spaced sampling for discrete high and low grade zonation, SRK has applied a high grade search restriction distance during grade interpolation to appropriately control the impact of the highest grades. This approach honours the local presence of grade shoots, without resulting in overly large volumes of high grade in areas of relatively limited sample coverage at depth.

The selected estimation parameters have been verified based on the results of a quantitative Kriging Neighbourhood Analysis ("QKNA"), and are presented in Table 14-5. High grade search restriction distances are presented in Table 14-6 and were selected based on grade control data observations, initial variogram ranges and histogram analysis.

Table 14-5: Summary of Final Estimation Parameters for New Liberty

Estimation Parameters			Description
KZONE	1, 3 - 7	2	Kriging zones for estimation
FIELD	AUCAP	AUCAP	Field for interpolation
SREFNUM	1	2	Search reference number
SMETHOD	2	2	Search volume shape (2 = ellipse)
SDIST1	30	30	Search distance 1 (dip)
SDIST2	30	30	Search distance 2 (strike)
SDIST3	10	5	Search distance 3 (across strike)
SANGLE1	Dynamic	Dynamic	Search angle 1 (dip direction)
SANGLE2	Dynamic	Dynamic	Search angle 2 (dip)
SANGLE3	0	0	Search angle 3 (plunge)
SAXIS1	3	3	Search axis 1 (z)
SAXIS2	1	1	Search axis 2 (x)
SAXIS3	3	3	Search axis 3 (z)
MINNUM1	15	6	Minimum sample number (SVOL1)
MAXNUM1	30	36	Maximum sample number (SVOL1)
SVOLFAC2	2	2	Search distance expansion (SVOL2)
MINNUM2	15	6	Minimum sample number (SVOL2)
MAXNUM2	40	36	Maximum sample number (SVOL2)
SVOLFAC3	5	5	Search distance expansion (SVOL3)
MINNUM3	2	3	Minimum sample number (SVOL3)
MAXNUM3	40	36	Maximum sample number (SVOL3)
MAXKEY	5	3	Maximum number of samples per drillhole
SANGL1_F	TRDIPDIR	TRDIPDIR	Dynamic Anisotropy
SANGL2_F	TRDIP	TRDIP	Dynamic Anisotropy

Table 14-6: High Grade Search Restriction Distances

Deposit Area	KZONE	Restriction Parameters		
		Restriction Grade Au g/t	Search distance beyond which restriction gold grade applied is applied (m)	Comment
Larjor	1	-	-	Becomes Kinjor where X > 262890
Marvoe	2	20	40x15m (plunge (-45°) x across plunge)	Applied at depth where Z < 0m RL
Kinjor	3	-	-	-
Kinjor	4	-	-	-
Kinjor	5	-	-	-
Marvoe	6	10	40x20m (dip (90°) x strike)	Applied to expanded search volumes
Marvoe	7	10	40x20m (dip (90°) x strike)	Applied to expanded search volumes

Inverse distance weighting squared ("IDW^2") was used for the interpolation of density in the fresh rock (as discussed in Section 11.3.1) and for verification of the OK estimates for gold. The interpolation parameters used for density are presented in Table 14-7.

Table 14-7: Summary of Estimation Parameters for Density

Estimation Parameters		Description
KZONE	Fresh Rock	Kriging zones for estimation
FIELD	DENSITY	Field for interpolation
SREFNUM	2	Search reference number
SMETHOD	2	Search volume shape (2 = ellipse)
SDIST1	65	Search distance 1 (dip)
SDIST2	65	Search distance 2 (strike)
SDIST3	65	Search distance 3 (across strike)
SANGLE1	0	Search angle 1 (dip direction)
SANGLE2	0	Search angle 2 (dip)
SANGLE3	0	Search angle 3 (plunge)
SAXIS1	3	Search axis 1 (z)
SAXIS2	1	Search axis 2 (x)
SAXIS3	3	Search axis 3 (z)
MINNUM1	50	Minimum sample number (SVOL1)
MAXNUM1	150	Maximum sample number (SVOL1)
SVOLFAC2	2	Search distance expansion (SVOL2)
MINNUM2	50	Minimum sample number (SVOL2)
MAXNUM2	150	Maximum sample number (SVOL2)
SVOLFAC3	3	Search distance expansion (SVOL3)
MINNUM3	25	Minimum sample number (SVOL3)
MAXNUM3	150	Maximum sample number (SVOL3)
MAXKEY	-	Maximum number of samples per drillhole
SANGL1_F	0	Dynamic Anisotropy ("0" = not used)
SANGL2_F	0	Dynamic Anisotropy ("0" = not used)

14.11 Model Validation and Sensitivity

14.11.1 Sensitivity Analysis

The grade interpolation itself was performed in Datamine, based on optimum parameters verified through a QKNA exercise. The exercise was based on varying kriging parameters for gold (namely number of samples and search ellipse size) to reflect a number of different scenarios. This focused on the KZONE1 domain given its significant contribution to metal (60%) in the geological model and typical representation of the deposit as whole in terms of drillhole spacing and gold grade distribution.

Whilst SRK noted a degree of sensitivity in the mean block grade to a change in the estimation parameters (notably in relation to search ellipse size), block grades (visually) better reflected the overall grade distributions shown by sample composites by restricting the search ellipse dimension and maximum number of composites per drillhole to within reasonable limits. The final parameters were selected to ensure that the plunging high grade shoots and grade zonation from hangingwall to footwall within the deposit were appropriately reflected in block grade estimates.

14.11.2 Block Model Validation

SRK has validated the block model using the following techniques:

- visual inspection of block grades in comparison with drillhole data;
- sectional validation of the mean samples grades in comparison to the mean model grades; and
- comparison of block model statistics.

Visual Validation

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection has been undertaken in plan, section and 3D, comparing the sample grades with the block grades, which demonstrates in general good comparison between local block estimates and nearby samples.

Within some of the less well drilled areas of the model, SRK note a relatively high level of smoothing between the high and low grade block estimates, which is due to the relatively elevated nugget variance (35%) and high variability in grade between adjacent drillhole intercepts. Within these areas, whilst the higher and lower grade drillholes intercepts are appropriately honoured in relative terms by higher and lower grade patches in the block model, SRK has restricted the resource classification to reflect the need for further infill drilling.

Figure 14-9 to Figure 14-12 provide examples of the visual validation checks completed, highlight the overall block grades corresponding with composite sample grades and illustrate the presence of plunging grade shoots and grade zonation from hangingwall to footwall in the block grade estimates.

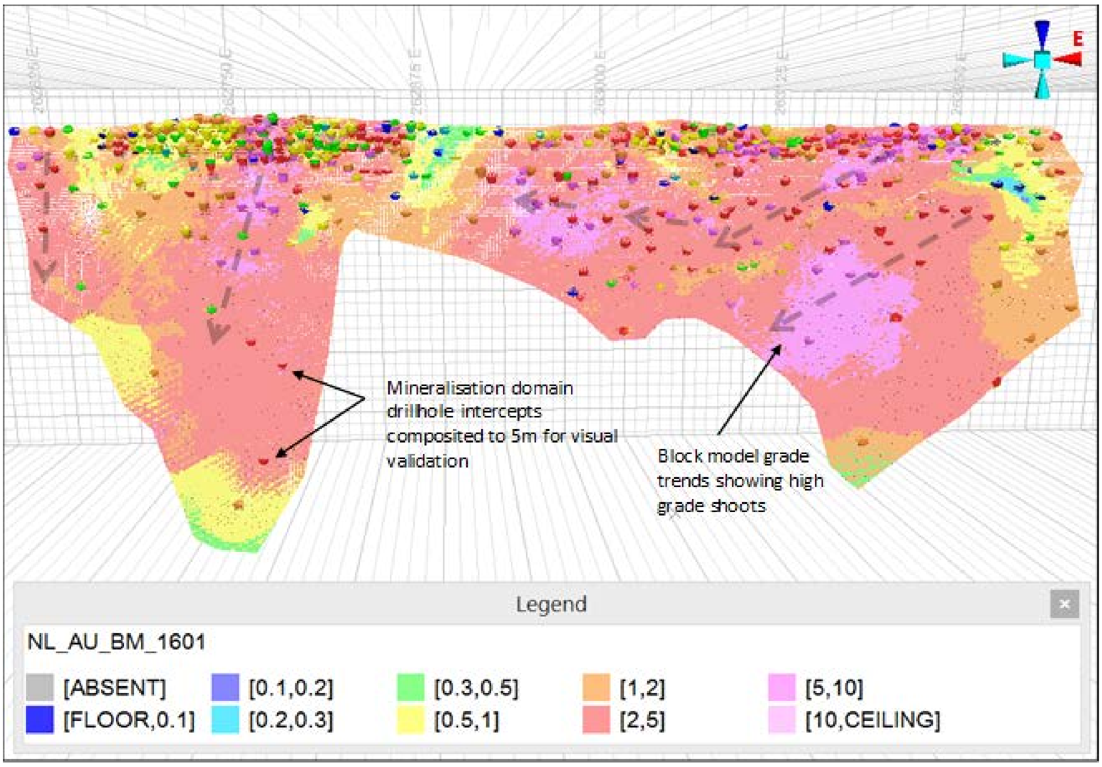


Figure 14-9: 3D Block Model Gold Grade Distribution, looking North: KZONE1

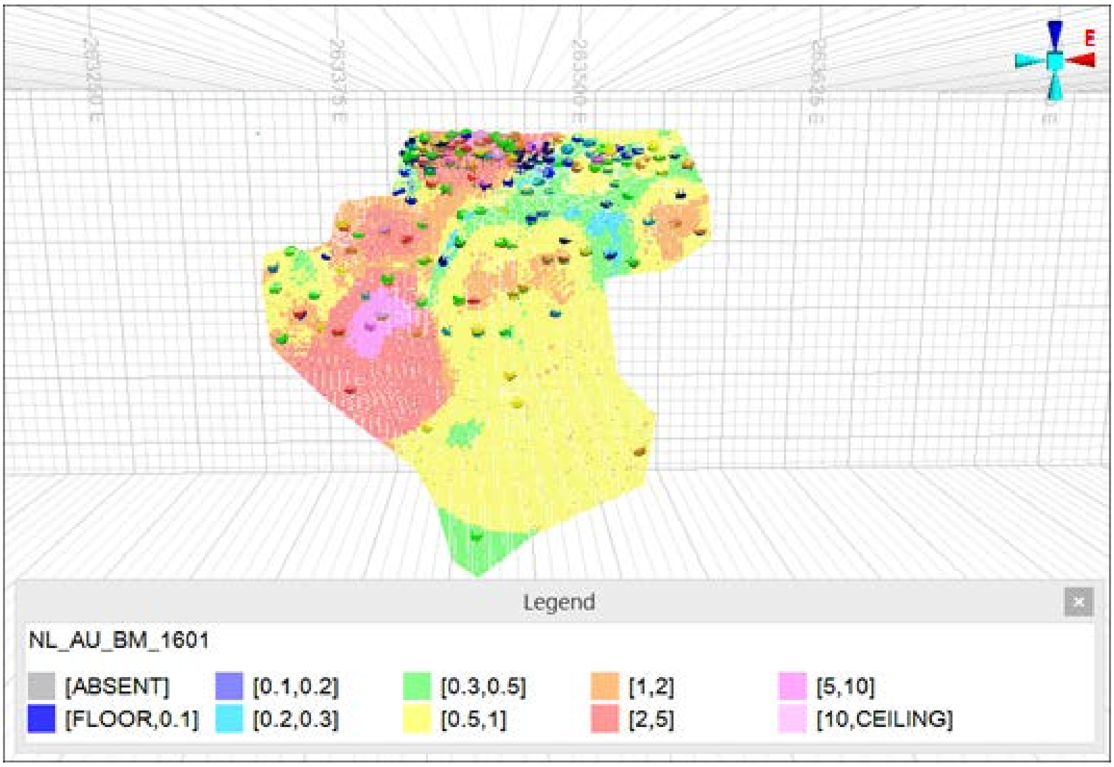
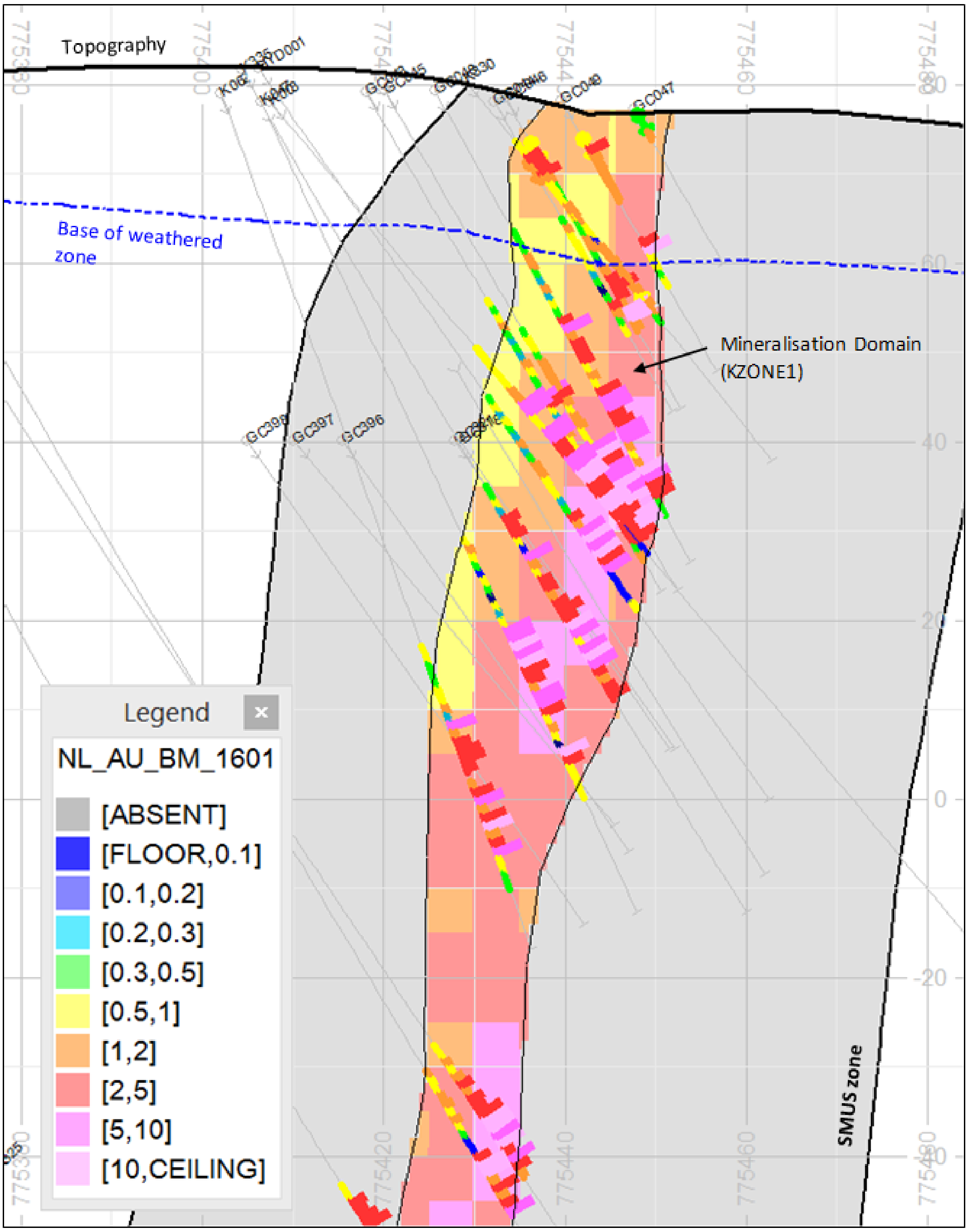


Figure 14-10: 3D Block Model Gold Grade Distribution, looking North: KZONE3



Note: A clipping width of 40m is applied to the cross-section, hence certain mineralised drillhole intervals may appear offset from the block model slice. All mineralised intervals shown are included in the geological model and grade interpolation

Figure 14-11: Block Model Gold Grade Distribution, looking West: KZONE1 cross-section

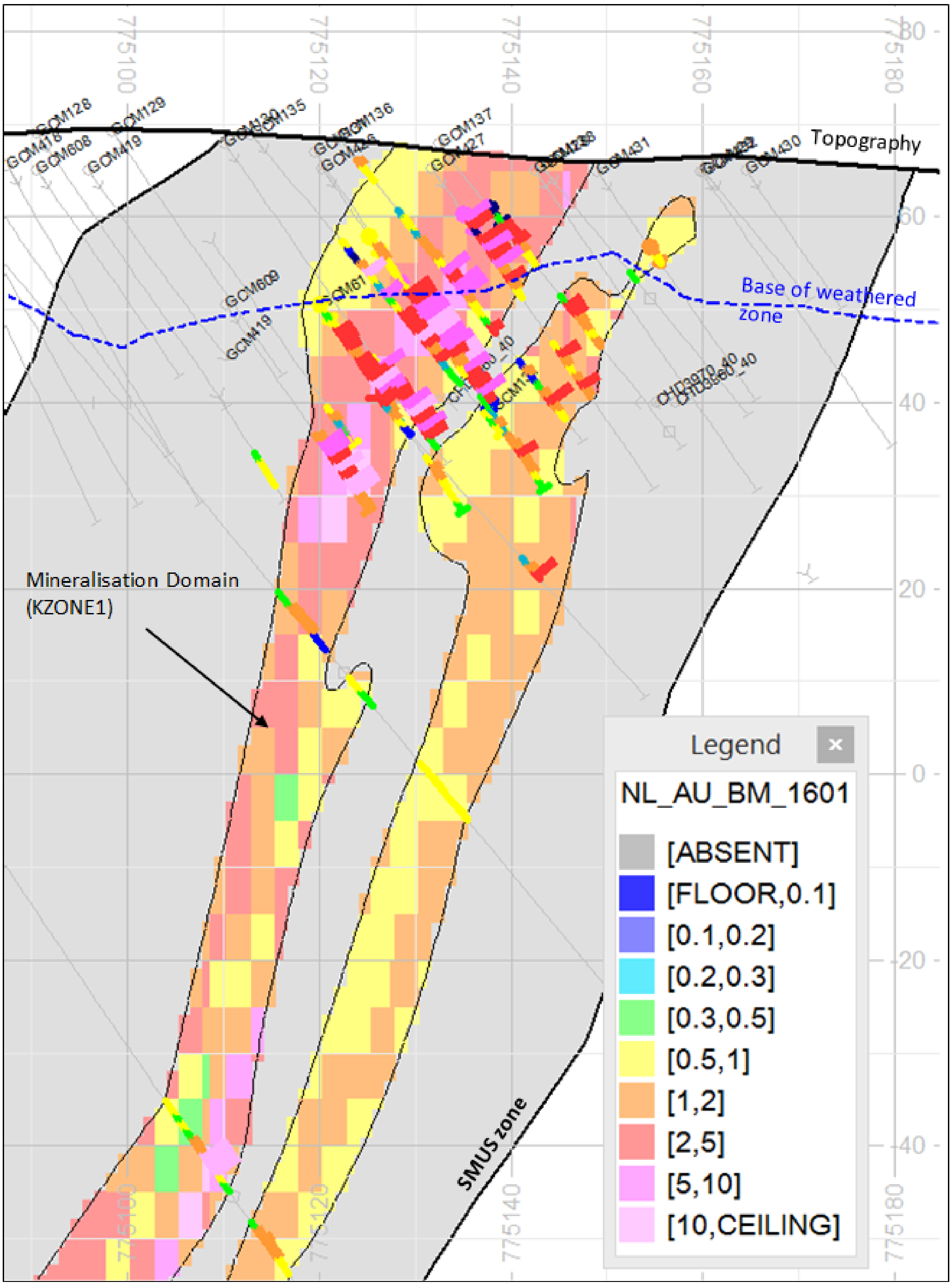


Figure 14-12: Block Model Gold Grade Distribution, looking West: KZONE2 cross-section

Sectional Validation

As part of the validation process, the input composite samples are compared to the block model grades within a series of coordinates (based on the principle directions). The results of which are then displayed on charts to check for visual discrepancies between grades. Figure 14-13 shows the results for the gold grades for the domain KZONE1 based on section lines cut along x-coordinates by way of example.

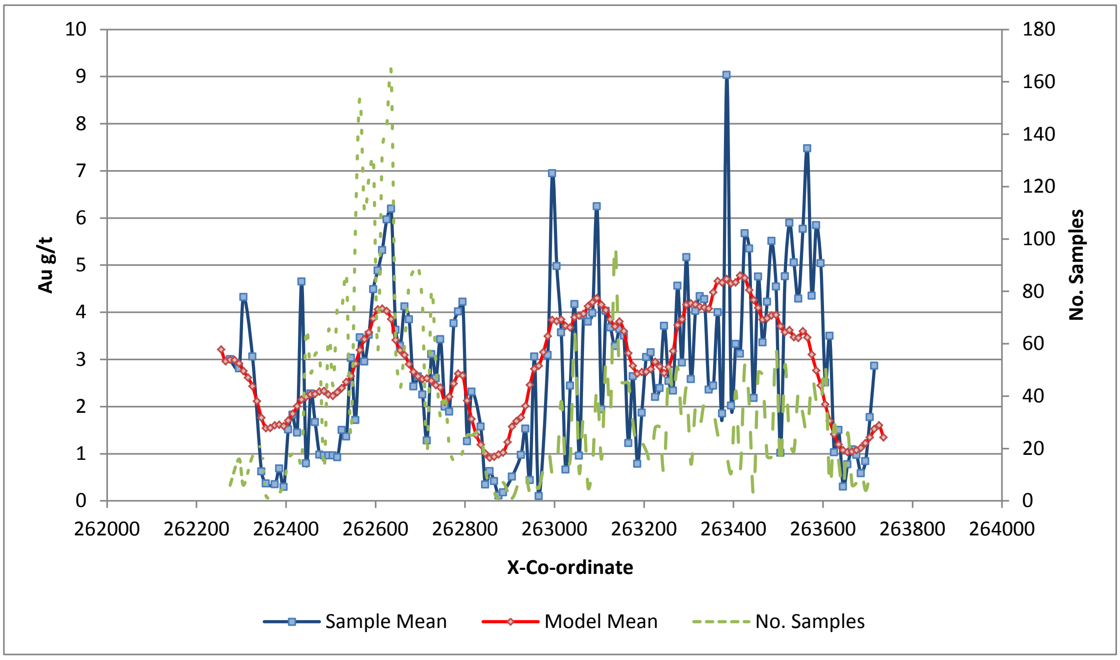


Figure 14-13: Validation Plot (Easting) showing Block Model Estimates versus Sample Mean (10m Intervals) for domain KZONE1 for gold

The resultant plots show a reasonable correlation between the block model grades and the composite grades, with the block model showing a typically smoothed profile of the composite grades as expected. SRK notes that in less densely sampled areas, minor grade discrepancies do exist on a local scale. Overall, however, SRK is confident that the interpolated grades reflect the available input sample data and the estimate shows no sign of material bias.

Statistical Validation

The block estimates have been compared to the mean of the composite samples (Table 14-8) which indicate the overall percentage difference in the mean grades for mineralisation domains KZONE 1, 2, 4 and 5 typically vary between 0% – 10%, which SRK deems to be within acceptable levels.

SRK notes a slightly higher percentage difference in the means for domains KZONE 3, 6 and 7, but these have irregular sample coverage and as a result, the sample mean is skewed by relatively few high / low grade samples.

Based on the visual, sectional and statistical validation results, SRK considers the grades estimated in the block model to be reasonable and free from material error or bias.

Table 14-8: Summary Block Statistics for Ordinary Kriging and Inverse Distance Weighting Estimation Methods

KZONE	Field	Estimation Method	Block Estimate Mean (ppm)	Composite Mean (ppm)	% Difference	Absolute Difference (ppm)
1	AU	OK	3.18	3.34	-5.0%	-0.17
		IDW	3.34	3.34	0.0%	0.00
2	AU	OK	1.95	2.26	-13.9%	-0.31
		IDW	2.02	2.26	-10.7%	-0.24
3	AU	OK	1.62	2.24	-27.6%	-0.62
		IDW	1.63	2.24	-27.2%	-0.61
4	AU	OK	1.05	1.10	-4.1%	-0.04
		IDW	1.16	1.10	6.0%	0.07
5	AU	OK	0.61	0.66	-7.0%	-0.05
		IDW	0.63	0.66	-4.2%	-0.03
6	AU	OK	3.25	5.07	-36.0%	-1.82
		IDW	3.22	5.07	-36.4%	-1.85
7	AU	OK	1.69	1.28	32.2%	0.41
		IDW	1.83	1.28	42.8%	0.55

14.12 Mineral Resource Classification

Block model quantities and grade estimates for the New Liberty deposit were classified according to the CIM Code.

Mineral Resource classification is typically a subjective concept, industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar resource classification.

Data quality, geological confidence, sample spacing and the interpreted continuity of grades controlled by the deposit has allowed SRK to classify the block model in the Measured, Indicated and Inferred Mineral Resource categories. The following guidelines apply to SRK's classification:

Measured

Measured Mineral Resources comprise the blocks in domains which have close spaced (10-15m) grade control drilling that show visually predictable grade continuity and high geological confidence. Additional in-pit structural investigation to support the interpreted orientation and nature of the contacts of the higher grade zones at Marvoe and Kinjor North is required prior to reporting the resource in these areas with 'Measured' confidence.

Indicated

Indicated Mineral Resources comprise the blocks which have a good level of geological confidence and which are situated within relatively well drilled areas of the model and typically between 25-50 m beyond these areas.

Inferred

Inferred Mineral Resources are in domains that display reasonable geological confidence and where blocks are typically within 70-100m of sample data. These areas require infill drilling to improve the quality of the geological interpretation and local block grade estimates to a level suitable for mine planning.

An example of SRK's Mineral Resource classification for the New Liberty deposit is shown in Figure 14-14.

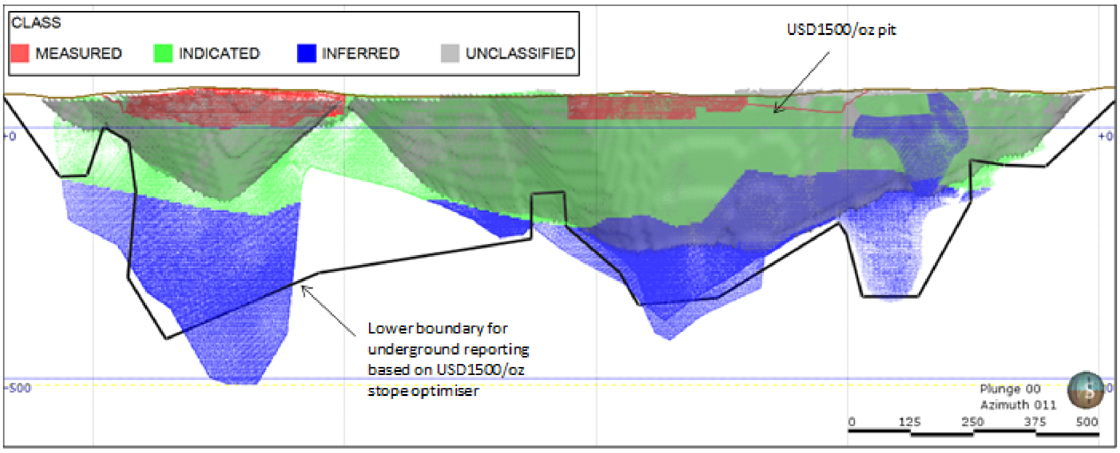


Figure 14-14: SRK's Classification Scheme for the New Liberty Project, looking north

14.13 Mineral Resource Statement

The CIM Code defines a Mineral Resource as:

A "concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries.

Reporting and Cut-off Derivation

SRK considers that the New Liberty Project shows potential for having reasonable prospects for economic extraction with respect to both open-pit and underground mining methods. The Mineral Resource has therefore been subject to a pit optimisation study to derive a depth constraint to which an open pit operation could be considered viable and then SRK has applied a cut-off grade of 0.8 g/t Au to the material falling within this pit outline. A cut-off grade of 2g/t Au was determined for material with the potential to be mined underground, with lower-grade material within thinner (and less contiguous) zones of mineralisation removed using Deswick's Mining Stope Optimizer (SO) as a spatial guide.

A summary of the parameters used to derive the reporting constraints and cut-off grades for reporting of the Mineral Resource are shown in Table 14-9.

SRK has used a gold price of USD1,500/oz based on typical long term consensus forecasts and to include some upside to reflect the requirement for "reasonable prospects" for eventual extraction.

Table 14-9: Summary of key assumptions for Conceptual Open Pit Optimisation and cut-off grade calculation

Parameter	Value	Unit	Comment
Gold Price	1,500	USD/oz	
Mining Cost	1.85	USD/t rock	
Incremental Mining Cost	0.04	USD/t/10m	
Reference Level	60	Z Elevation	
Processing Cost	20.00	USD/tore	
General and Administrative	7.00	USD/tore	
Refining/transport/marketing/other	3.5	USD/oz	
Mining Dilution	12	%	
Gold Process Recovery	=if(au>=4,0.93,0.0026*au^3-0.0386*au^2+0.178*au+0.6661)	%	Based on regression with gold grade
Royalty	3.0	%	

The Resource Statement for the New Liberty deposit is shown in Table 14-10. Note that this is the position as at 31 July 2017 and so it excludes material already mined by this time. The Resource Statement has been split to show both remaining in-situ open pit and underground resources and also ore stockpiles as at 31 July 2017. The ore stockpiles have been classified as Indicated Resources as while the stockpiles are surveyed and reconciled with truck counts for tonnage, the material is not sampled (subsequent to excavation) and the grade is based on theoretical block model grades.

Table 14-10: SRK Mineral Resource Statement as at 31 July 2017 for the New Liberty Deposit prepared in accordance with the CIM Code

Category	Cut-off	Tonnes Mt	Au Grade g/t	Au Koz
In-Situ				
Measured	0.8 g/t (OP)	0.1	3.6	15
Indicated	0.8 g/t (OP)	8.5	3.3	890
	2.0 g/t (UG)	0.6	3.3	65
Measured and Indicated	0.8 g/t (OP)	8.6	3.3	905
	2.0 g/t (UG)	0.6	3.3	65
Inferred	0.8 g/t (OP)	3.6	2.8	325
	2.0 g/t (UG)	2.8	3.3	295
Sub-total Measured		0.1	3.6	15
Sub-total Indicated		9.1	3.3	955
Sub-total Measured and Indicated		9.2	3.3	970
Sub-total Inferred		6.4	3.0	620
Stockpiles				
Indicated	Oxide and Fresh Ore	0.2	1.5	10
Indicated	Sub-Grade Ore	0.2	0.8	5
Sub-total Indicated		0.4	1.1	15
Total				
Total Measured		0.1	3.6	15
Total Indicated		9.5	3.2	970
Total Measured and Indicated		9.6	3.2	985
Total Inferred		6.4	3.0	620

1. The marginal cut-off grade used for resource reporting is 0.8/t Au for Open Pit and 2.0g/t Au for Underground Mining.
2. All figures are rounded to reflect the relative accuracy of the estimate.
3, Mineral Resources are reported inclusive of those converted to Mineral Reserves.
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

14.14 Grade Sensitivity Analysis

The results of grade sensitivity analysis completed for New Liberty are shown in 14-11 and Table 14-12. This is to show the continuity of the grade estimates at various cut-off increments and the sensitivity of the Mineral Resource to changes in cut-off. The tonnages and grades in these tables, however, should not be interpreted as Mineral Resources.

Table 14-11: Gradations for In-Situ Open Pit Material at New Liberty at various Au g/t Cut-off Grades

Grade - Tonnage Table, New Liberty Open Pit						
Cut-off Grade	Measured and Indicated			Inferred		
	Quantity	Gold		Quantity	Gold	
Gold (g/t)	(Mt)	Grade (g/t)	Metal (koz)	(Mt)	Grade (g/t)	Metal (koz)
0.20	9.6	3.00	925	3.7	2.8	325
0.40	9.4	3.04	925	3.7	2.8	325
0.80	8.6	3.28	905	3.6	2.8	325
1.00	7.9	3.48	885	3.5	2.9	325
2.00	5.8	4.24	790	2.6	3.3	275
3.00	4.1	4.94	655	1.2	4.3	165
4.00	2.7	5.73	495	0.6	5.2	100
5.00	1.6	6.59	335	0.3	5.8	60

Table 14-12: Gradations for In-Situ Underground Material at New Liberty at various Au g/t Cut-off Grades

Grade - Tonnage Table, New Liberty Underground						
Cut-off Grade	Measured and Indicated			Inferred		
	Quantity	Gold		Quantity	Gold	
Gold (g/t)	(Mt)	Grade (g/t)	Metal (koz)	(Mt)	Grade (g/t)	Metal (koz)
0.20	1.4	1.98	90	4.2	2.6	350
0.40	1.4	2.00	90	4.2	2.6	345
0.80	1.1	2.31	85	3.7	2.8	340
1.00	0.9	2.64	80	3.5	3.0	330
2.00	0.6	3.33	65	2.8	3.3	295
3.00	0.3	4.19	40	1.6	3.9	200
4.00	0.1	5.07	20	0.6	4.7	95
5.00	0.1	5.97	10	0.1	5.7	25

14.15 Comparison to Previous Mineral Resource Estimates

In terms of in-situ resources, in comparison to the previous AMC October 2012 Mineral Resource estimate for the Project (as reported in the 2015 Technical Report), which was reported at a cut-off grade of 1 g/t gold and unconstrained at depth, this updated Mineral Resource estimate (which is reported at separate cut-off's for open pit and underground, as shown in Table 14-10 above) represents a 15% reduction in metal content within the Measured and Indicated from 1,143koz to 970koz. This reduction is primarily due to depletion from mining and the use of an open pit depth constraint (the USD1,500/oz optimised pit shell as noted above) which limits the depth to which material between 0.8-2 g/t Au is reported. Since the start of operations and to the end of July 2017, some 1.8Mt of ore has been processed with an average grade of 2.5g/t and containing 145koz of gold.

Within the Inferred classified material, SRK notes a small increase in the metal content for the Project from 593koz to 620koz due to the addition of tonnes within extensions to the mineralisation wireframes mainly at depth, based on an updated assessment of the grade and geological continuity.

In addition to the changes noted above, SRK considers that other key changes in the Mineral Resource result from a combination of the following factors:

- Reporting of an underground Mineral Resource beneath the material considered viable for open pit mining. This required an increase in the cut-off grade from 1g/t to 2g/t Au and use of Deswick's SO to guide exclusion of lower-grade material within thinner and less continuous zones of mineralisation;
- Greater detail applied to the grade interpolation strategy to allow good local variability within the well drilled areas with appropriate level of grade smoothing within the less well drilled areas at depth;
- Infill grade control drilling defining a wider mineralised zone close to surface in the Larjor and Kinjor areas and better constraining the volume and distribution of some of the high grade zones defined in the previous model;
- Updated geological interpretation at Marvoe (based on mining observations to date) to reflect hard boundary zonation between areas of higher and lower grade;
- Reduction to the cut-off grade used to report the open pit portion of the Mineral Resource (from 1 to 0.8 g/t Au), which is mainly due to the lower costs used in the latest optimisation and mine planning work;
- The increased upper limit for the high-grade cap value from 35g/t Au to 50g/t Au; and
- Removal of the hard domain boundary between oxidised and fresh material for gold grade interpolation.

14.16 Exploration Potential

SRK notes that the mineralisation and high grade shoots remain open at depth at Kinjor South and Marvoe and Larjor (as illustrated in Figure 14-15), where there is potential for increasing the tonnage in the reported underground Mineral Resource at New Liberty with additional drilling and modelling.

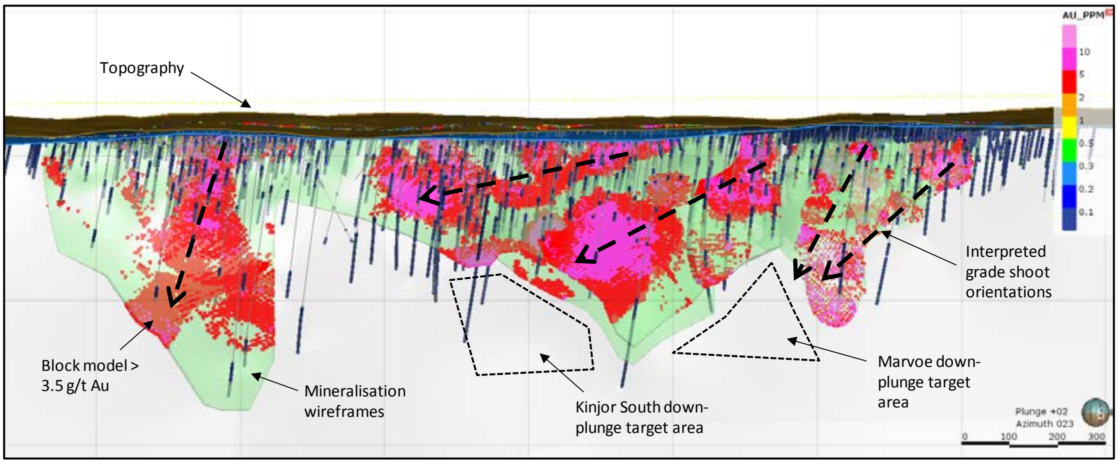


Figure 14-15: New Liberty down-plunge Exploration Targets

The economic valuation presented in Section 22 is based on the Mineral Reserves which fall within the current designed pit which the Company is planning to exploit in the Life of Mine plan presented. SRK believes that there is potential for further exploration to extend the Mineral Reserve by infill drilling the Inferred Mineral Resource within the current pit design; through infill drilling of the Inferred Mineral Resource lying beneath this and through drilling extensions to the Mineral Resource at depth.

Notably:

- Some 0.1Mt of Inferred Mineral Resource with a mean grade of 2.7g/t Au has been estimated to be present within the designed pit. This has been treated as “waste” in the valuation presented in Section 22.
- A further 3.5Mt of Inferred Mineral Resources with a mean grade of 2.8g/t Au has been delineated below the current design pit but within the open pit reporting limit using the USD1,500/oz optimised Measured, Indicated and Inferred (MII) pit shell.
- Some 2.8Mt of Inferred Resources with a mean grade of 3.3g/t Au has been reported as underground resources and has the potential to be exploited by underground mining.
- In addition to the above, plunging high grade shoots delineated at Kinjor South, Marvoe and Larjor remain open at depth and so there is potential for increasing the underground Mineral Resource in these areas through further drilling.

The Company agrees with the above comments and has planned an initial 14,000m drilling programme targeted to infill the 3.5Mt of the Inferred Mineral Resource lying below the current designed pit and within the USD1,500/oz optimised MII pit shell as noted above SRK has reviewed this drilling plan which has been costed at USD1.5M and agrees that this exploration is justified and if successful has potential to extend the envisaged mine life. Of this Inferred Resource, some 3.0Mt of with a mean grade of 2.8g/t has been delineated below the current design pit but within a USD1,300/oz optimised MII pit shell, which demonstrates that a significant proportion of the total Inferred Mineral Resource below the current design pit would have potential to extend the envisaged mine life through extensions of the current design pit, subject to this being upgraded to the Indicated classification.

The Company then intends to undertake a second drill programme which will test for extension of the deeper underground potential of high grade shoots below the USD1,500/oz optimised MII pit shell.

Finally, the Company has identified continuations and parallel bands of the prospective ultramafic host rock elsewhere within the Licence Boundary (Figure 10-10), highlighting the potential to find for additional zones of gold mineralisation.

14.17 Concluding Remarks

The New Liberty deposit is now an open pit mining operation which is at an advanced stage of drilling and geological understanding. Recent grade control infill drilling has added further geological confidence to the local scale geometry of the mineralisation and grade distributions close to surface.

The geological interpretation used to generate the Mineral Resource presented herein is generally considered to be robust; however, there are areas of lower geological confidence which may be subject to further revision in the future. SRK considers the exploration data accumulated by the Company is generally reliable and suitable for the purpose of this Mineral Resource estimate.

14.18 Recommendations

SRK considers there to be good potential to improve confidence in the reported Mineral Resource at New Liberty with additional drilling, in-pit geological investigation and further modelling work.

In relation to exploration drilling and sampling, SRK would recommend the following:

- Targeted infill drilling to add geological confidence to convert the Inferred Resources to Indicated and convert more of the Indicated to Measured Resources (as noted above);
- Additional exploration drilling at depth, specifically around the down-dip continuation of the grade shoots at Kinjor South, Marvoe and Larjor, where there is potential for increasing the tonnage in the reported underground Mineral Resource;
- Additional exploration within the surrounding permit area where there is good potential to find further gold mineralisation.

In relation to grade control drilling, whilst SRK has a high overall confidence in the block tonnage and grade estimates in the geologically well-constrained, well-drilled parts of the mineralisation domains, sufficient for Measured Mineral Resources, SRK would recommend the further investigation (and rectification) of the CRM swaps and reduction in analytical precision noted since the start of the ALS NLGM laboratory and the re-submission of 5-10% of sample pulps analysed at ALS NLGM with the Geostats CRMs to an umpire laboratory to further verify analytical performance;

In relation to geological fieldwork, SRK recommends in-pit mapping as part of a structural study to help improve understanding of the geological controls on (and 3D structural framework for) the higher grade mineralised zones at Marvoe, Kinjor North and Larjor. This exercise should be completed with a targeted structural re-assessment of the drillcore, with a focus on the orientation of the zones of increased shearing which are currently interpreted to host the higher grade zones.

15 MINERAL RESERVE ESTIMATES

15.1 Approach

The Mineral Reserves for the New Liberty deposit have been reported using the CIM Code. The Mineral Reserves are part of the Mineral Resources as stated in Section 14. The CIM Code defines a Mineral Reserve as:

"the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined."

The ore loss and dilution have been estimated using a regularised model as described in Section 16.2.3. Average values are 3.3% ore loss and 13.5% dilution for the deposit within the pit designs.

An optimisation process was used to define the optimal pit limits based on the following parameters (described further in Section 16.4.2):

- Au selling prices: USD1,300/oz;
- Base mining cost of USD1.85/t with an incremental cost of USD0.04/t/10m;
- Processing cost of USD20/t milled and general and administrative ("G&A") cost of USD7/t milled;
- The processing recovery is variable dependent on Au grade (see Table 16-2), averaging 91.2%;
- Royalty of 3% with additional selling costs of USD3.5/oz; and
- Overall slope angles of 42° to 48° in the fresh material and 38° in the oxide.

The marginal economic cut-off grade is 0.85 g/t Au which was used for the Mineral Reserve estimation.

The Mineral Reserves occur within the engineered pit designs with an average strip ratio of 16.5 (t_{waste} : t_{ore}).

15.2 Mineral Reserve Statement

The Mineral Reserve statement for the NLGM project is presented in Table 15-1. The independent qualified person, as defined by Canadian Securities Administrators National Instrument 43-101 for mineral reserve estimates, is Dr Mike Armitage BSc, MIMMM, C.Eng, C.Geol, SRK Consulting (UK) Limited.

The Project base case economic analysis presented in Section 22 shows that the NLGM project life-of-mine plan founded on the Mineral Reserve Estimate in Table 15-1 provides a positive present value of the net cash flow and a positive rate of return, confirming that the Mineral Reserves are economically viable and that economic extraction can be justified.

Table 15-1: NLGM Mineral Reserve Statement, Effective 31 July 2017

Category	Quantity (Mt)	Au Grade (g/t)	Au Contained (koz)
Proven	**0.2**	**3.03**	**15**
In-Pit	0.2	3.03	15
Probable	**7.2**	**3.03**	**702**
In-Pit	7.0	3.09	690
Stockpiles	0.2	1.40	11
Total Proven & Probable	**7.4**	**3.03**	**717**

Notes:

1. Mineral Reserves are included in the Mineral Resource Estimate dated Jul, 31, 2017.
2. Mineral Reserves are reported at a cut-off grade of 0.85g/t Au within an engineered pit design. The cut-off grade is considered appropriate for a selling price of USD1,300/oz, processing cost of USD20/t, G&A cost of USD7/t, royalty of 3%, selling costs of USD3.5/oz and processing recovery averaging 91.2%.
3. Includes ore loss and dilution as reported from a regularised block model at 5 m x 2.5 m x 5 m, which has an average ore loss and dilution of 3.3% and 13.5%, respectively.

16 MINING METHODS

16.1 Introduction

A life of mine plan was undertaken following the completion of the updated Resource Model noted above in Chapter 14, which included the following:

- Development of a mining model for estimating ore loss and dilution on a local basis;
- Pit optimisation to determine the optimal pit shape;
- Mine design of the selected pit shell;
- Life of mine schedule to meet the mill feed targets;
- Equipment and labour estimate; and
- Mine capital and operating cost forecasts.

The geotechnical assessment is based on the original study completed during the FS.

16.2 Mining Model

16.2.1 Approach

A mining model was used to estimate ore loss and dilution on a local level. A regularisation approach was undertaken in Vulcan to a selective mining unit (“SMU”).

16.2.2 Reconciliation

The reconciliation of the historically milled material compared to the updated Resource Model is shown in Figure 16-1. The ore loss and dilution is shown for the previous Resource Model and the new model from June 2017. Dilution values were quite high in the earlier periods, which could be attributed to a number of issues, including inadequately estimated tonnages and grades on the stockpiles, poor blasting and mining practices and inadequate grade control techniques.

A new grade control approach was adopted during June 2017, along with a blast monitoring system, which lowered ore loss and dilution values in July 2017 as compared to the new model. BMMC plans to continue with the blast monitoring system and the updated grade control techniques in order to maintain lower ore loss and dilution values.

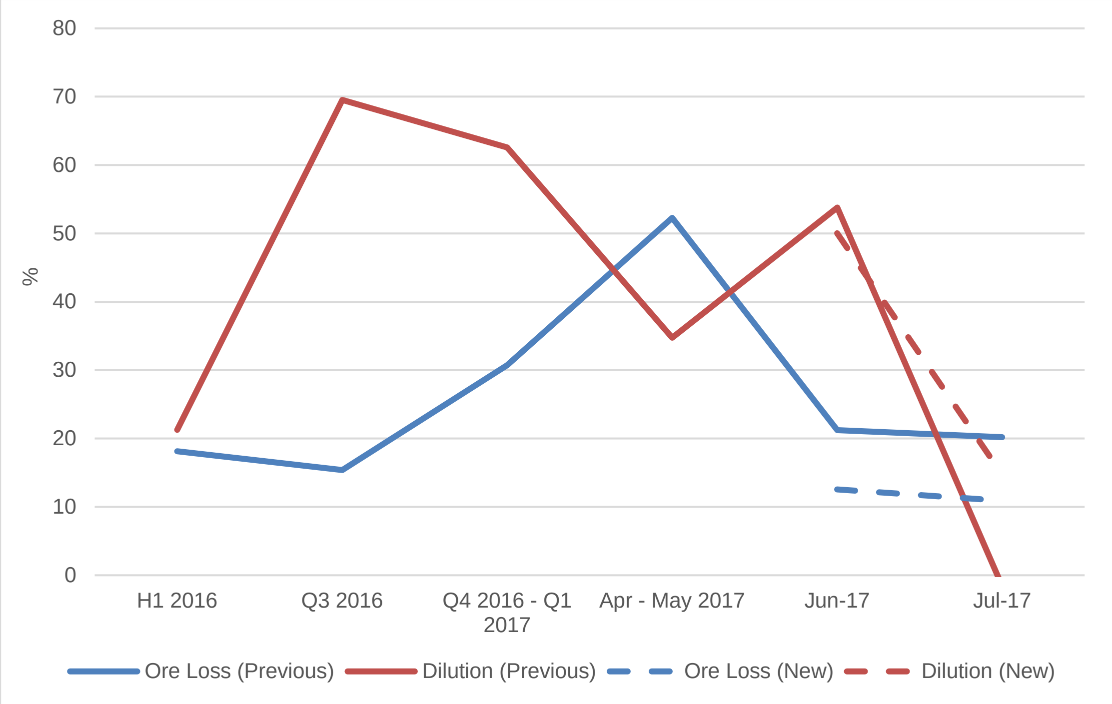


Figure 16-1: Mill Feed Reconciliation to the Resource Model (Previous and New)

16.2.3 Regularisation

The resource model was regularised to a block size or 5 m x 2.5 m x 5 m, which is deemed appropriate for the equipment, grade control system and blasting practices. The results within the final design are shown in Table 16-1, reported by deposit at a cut-off grade of 0.85 g.t Au.

Table 16-1: Mining Model Regularisation Results

Regularisation Results	Units	Resource Model	Mining Model
Total Model	(kt)	6,486	7,119
Au Grade	(g/t Au)	3.50	3.08
Au Contained	(koz Au)	730	706
Ore Loss	(%)	-	3.3
Dilution	(%)	-	13.5
Larjor	(kt)	1,073	1,145
Au Grade	(g/t Au)	3.24	2.97
Au Contained	(koz Au)	112	109
Ore Loss	(%)	-	2.4
Dilution	(%)	-	9.2
Kinjor	(kt)	3,153	3,591
Au Grade	(g/t Au)	3.77	3.19
Au Contained	(koz Au)	382	368
Ore Loss	(%)	-	3.6
Dilution	(%)	-	18.1
Marvoe	(kt)	2,259	2,383
Au Grade	(g/t Au)	3.25	2.98
Au Contained	(koz Au)	236	228
Ore Loss	(%)	-	3.4
Dilution	(%)	-	9.2

16.3 Geotechnical Assessment

AMC developed a 3D structural model in 2012 in collaboration with BMMC personnel. Geotechnical domains were defined, 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).

AMC developed an RQD database to compare lithologies and pit areas, showing consistency across the pit areas, with a greater amount of poor ground in the low magnetite ultramafic, and the granitioids.

The variation in discontinuity condition ratings was analysed across the geotechnical domains, and a revised discontinuity strength was developed for each domain. Subsequent 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. BMMC has indicated that the 8.5 m berms incorporated in the designs are sufficient to contain most failures, however, SRK recommends that the geotechnical assessment is updated based on the most recent designs and the current operations.

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. A slope depressurisation programme has not been undertaken at this stage but BMMC has indicated that this will be done for the final pit faces. SRK however recommends that a hydrogeological testing programme is undertaken as a slope depressurisation programme may need to be undertaken prior to the pit being developed to the final faces.

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°); and

- Marvoe northern walls.

AMC visited the mine between 21-26 November 2016 and reviewed the pit faces and the systems in place to manage the geotechnical aspects of the operation. Its report confirms the original findings and parameters used in the pit design but also highlighted a number of areas for improvement, which included:

- Updating of the structural model using pit mapping/photogrammetry and drillhole information so that interpretations can be provided to guide the mine planning team.
- Managing faces and safety berms for isolation of falling material, ensuring design compliance of batter faces and berm widths. Drilling patterns to take into consideration flatter foliations on footwall boundary thereby not undercutting the rock structures. Pre-splitting trials to be undertaken and trim blasting to continue. Depressurisation drilling to be undertaken in wet areas.
- Provision of more training in ground control hazard awareness, working below working levels, working near faces, and implement some additional procedures.
- Nomination of a responsible person and a dedicated person to work on geotechnical and water issues.

16.4 Pit Optimisation

16.4.1 Approach

A pit optimisation has been undertaken using the regularised block model with Datamine's NPV Scheduler ("NPVS") software. The pit optimisation parameters are based on the historical performance at the current operation and the budget forecast.

16.4.2 Optimisation Parameters

The pit optimisation parameters used in the study are shown in Table 16-2. The geotechnical parameters have been based on those given in Section 16.3 and have been adjusted based on the expected ramp locations.

The mining model was used in the optimisation, to account for local ore loss and dilution, no additional ore loss and dilution factors have been added.

Only Measured and Indicated Mineral Resources have been used in the optimisation process as plant feed.

An end of July 2017 surface was used as the starting surface in the optimisation process.

Table 16-2: Pit Optimisation Parameters

Parameters	Units	Value
Production		
Production Rate - Ore	(ktpa)	1,680
Geotechnical - Overall Slope Angles		
Larjor North	(°)	42
Larjor South	(°)	43
Kinjor North	(°)	48
Kinjor South	(°)	47
Marvoe North	(°)	48
Marvoe South	(°)	48
Weathered All	(°)	38
Processing		
Recovery	(%)	if(au>=4,0.93,0.0026*au^3-0.0386*au^2+0.178*au+0.6661)
Operating Costs		
Mining Costs		
Mining Costs	(USD/t_{rock})	1.85
Incremental Mining Cost	(USD/t/10m)	0.04
Reference Level	(Z Elevation)	60
LoM Average	(USD/t_{rock})	2.20
Processing	(USD/t_{ore})	20.00
G&A	(USD/t_{ore})	7.00
Royalty	(%)	3.0
Selling Cost	(USD/oz)	3.5
Metal Price		
Gold	(USD/oz)	1,300
Other		
Discount Rate	(%)	6
Cut-Off Grade		
Marginal Diluted	(USD/t_{ore})	27.00
	(g/t Au)	0.85

16.4.3 Optimisation Results

The pit shell sensitivity to metal price is shown in Figure 16-2 where run of mine (RoM) tonnages are reported above a 0.85 g/t Au cut-off. Reported discounted cashflows (“DCF”) in the figure represent a reporting metric from the optimisation shell:

- Best – assumes a mining scenario which progresses the reference revenue factor shell through each incremental pit shell as an independent phase; and
- Worst – assumes a mining scenario which progresses the reference revenue factor shell without any incremental phases, bench by bench.

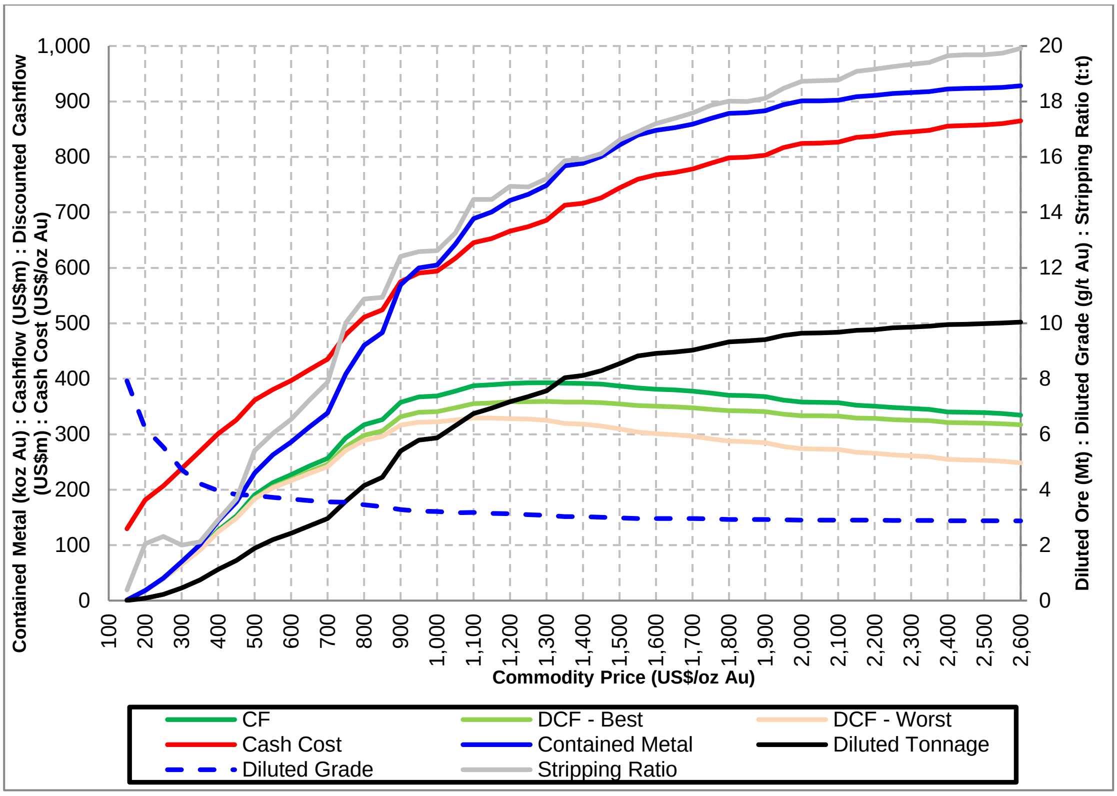


Figure 16-2: Pit Shell Sensitivity to Metal Price

Figure 16-3 to Figure 16-6 show the plan view and section of the pit shell sensitivity.

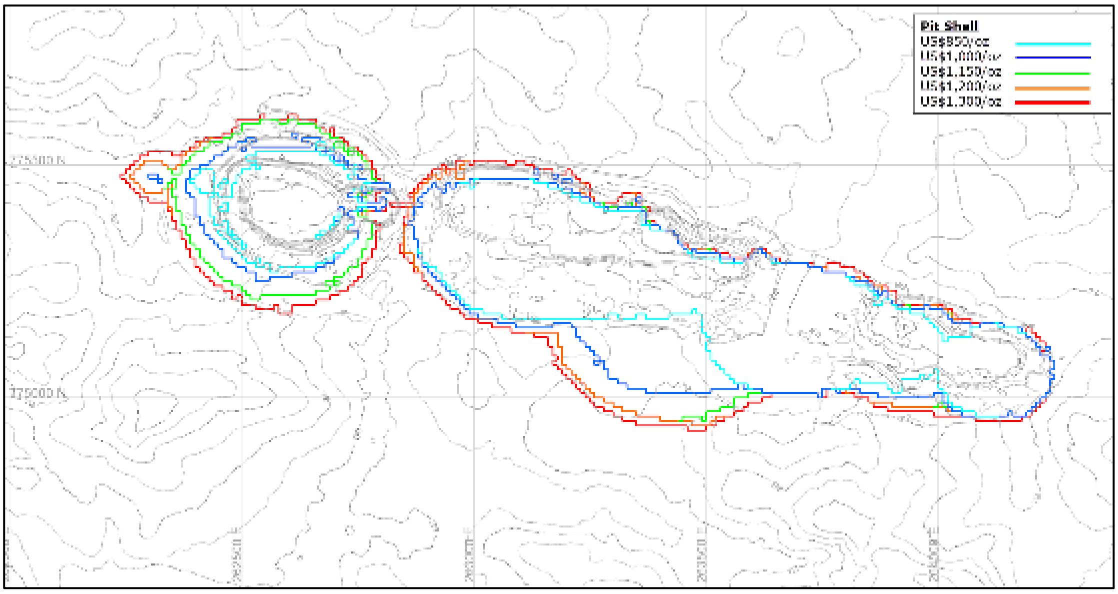


Figure 16-3: Plan View - Pit Shell Size Sensitivity to Metal Price

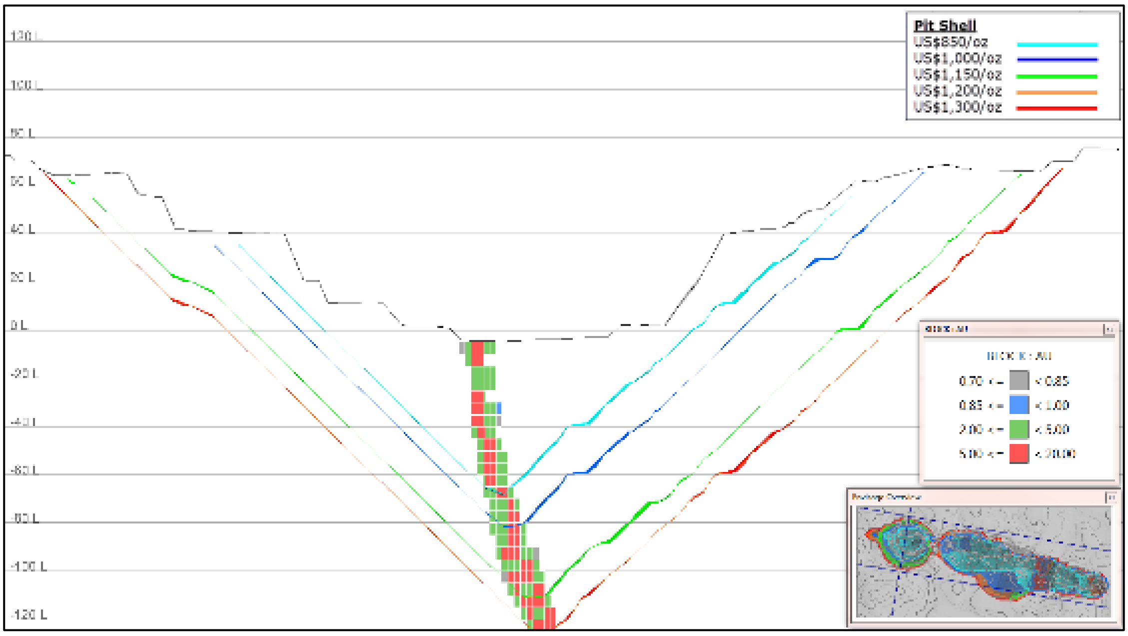


Figure 16-4: Larjor Section View - Pit Shell Size Sensitivity to Metal Price

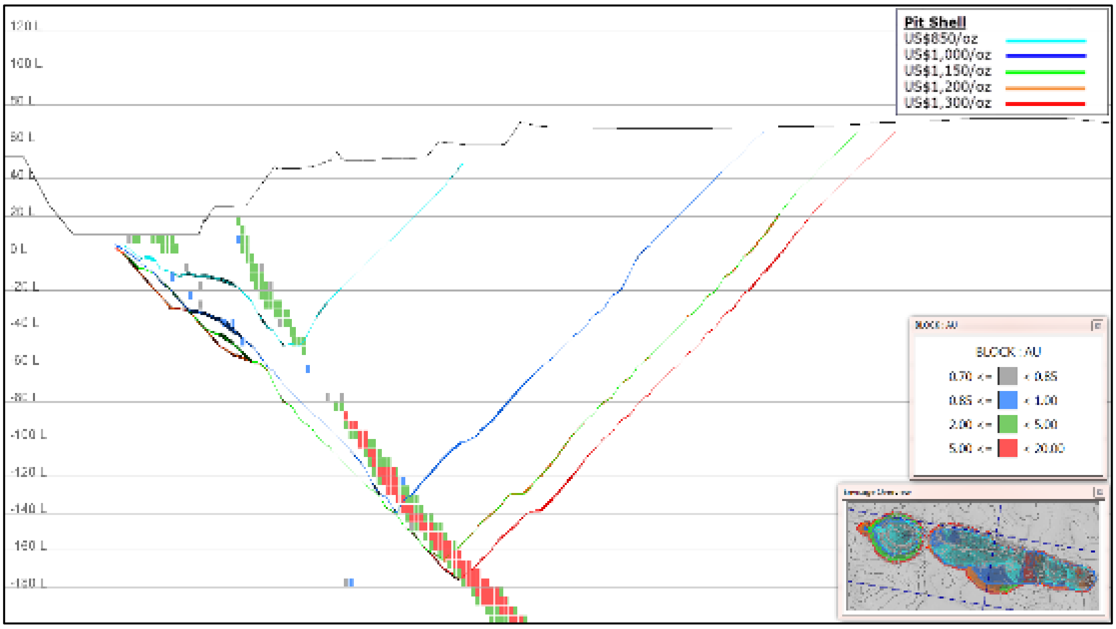


Figure 16-5: Kinjor Section View - Pit Shell Size Sensitivity to Metal Price

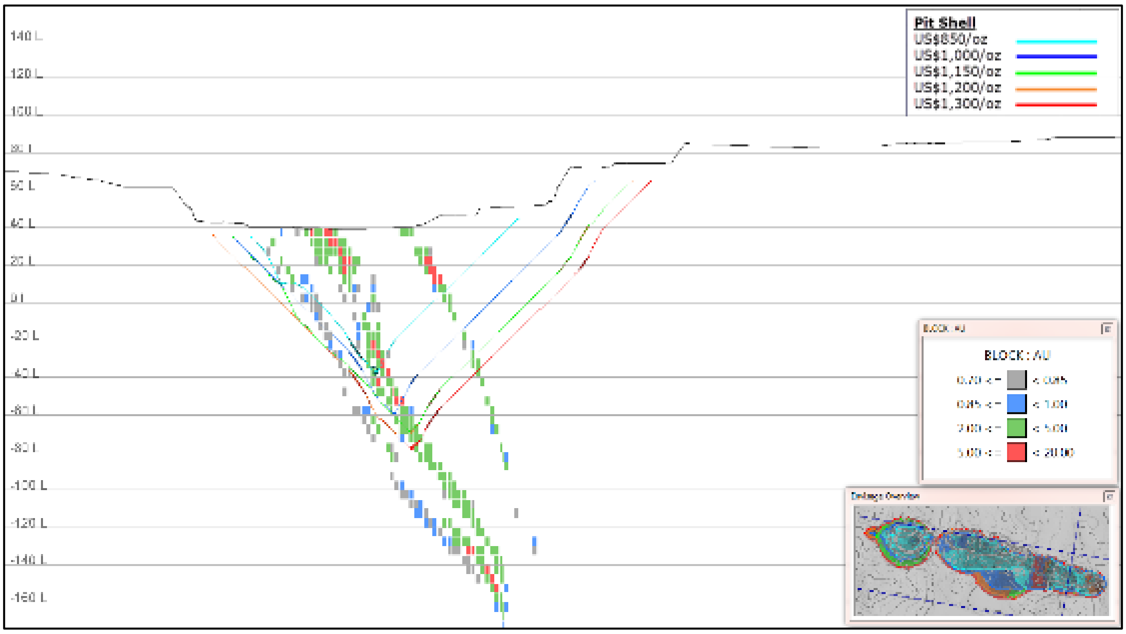


Figure 16-6: Marvoe Section View - Pit Shell Size Sensitivity to Metal Price

16.4.4 Selected Pit Shell

The USD1,300/oz pit shell was selected to carry forward to the mine design in order to maximise resource recovery. The results of the USD1,300/oz pit shell are shown in Table 16-3 reported at a 0.85 g/t Au cut-off grade.

Table 16-3: Selected Pit Shell

Optimisation Results	Units	1300 USD/oz
Quantities		
Total Rock	(Mt)	121.9
Mineral Inventory	(Mt)	7.5
	(g/t Au)	3.08
	(koz Au)	747
Waste	(Mt)	114.3
Stripping Ratio	(t:t)	15.2
Operating Expenditures		
Mining	(USD/t_{mined})	2.14
	(USD/t_{ore})	34.54
	(USD/oz)	382
Processing + G&A	(USD/t_{ore})	27.00
	(USD/oz)	299
Au Selling Cost	(USD/oz)	3.5
Total Cash Cost	(USD/oz)	684
Product		
Recovered Metal	(koz Au)	681
Economic Summary		
Metal Price	(USD/oz)	1,300
Revenue	(USDm)	885
Mining Costs	(USDm)	260
Processing Costs	(USDm)	203
Selling Costs	(USDm)	29
Cashflow	(USDm)	393
Discount Rate	(%)	6.0
Mill Rate	(Mtpa)	1.68
DCF - Best Case	(USDm)	340
DCF - Worst Case	(USDm)	326
DCF - Average	(USDm)	333
Project Life	(years)	4.5
DCF: Discounted cashflow		

16.5 Mine Design

16.5.1 Approach

The engineered final pit designs have been completed in order to verify the technical feasibility of the optimised pit shells. The engineered ultimate pit design is guided by the selected USD1,300/oz pit shell. Cutbacks have been designed in order to delay stripping requirements and access higher value material earlier in the mine life.

The waste rock dumps (“WRD”) designs have been engineered based on the waste inventory within the designed pits.

16.5.2 Pit Design

The pit design parameters are shown in Table 16-4. The road parameters have been based on Komatsu HD785 and Caterpillar (“CAT”) 777 haul trucks.

Table 16-4: Pit Design Parameters

Parameter	Unit	Value
Oxide		
Face Angle	(°)	45
Berm Width	(m)	5
Bench Height	(m)	10
Fresh		
Face Angle		
North Wall	(°)	70
South Wall	(°)	75
Berm Width	(m)	8.5
Bench Height	(m)	20
Roads		
Dual Lane Road Width	(m)	25
Single Lane Road Width	(m)	16
Ramp Gradient	(%)	10

The staged pit designs are shown in Figure 16-7 to Figure 16-9.

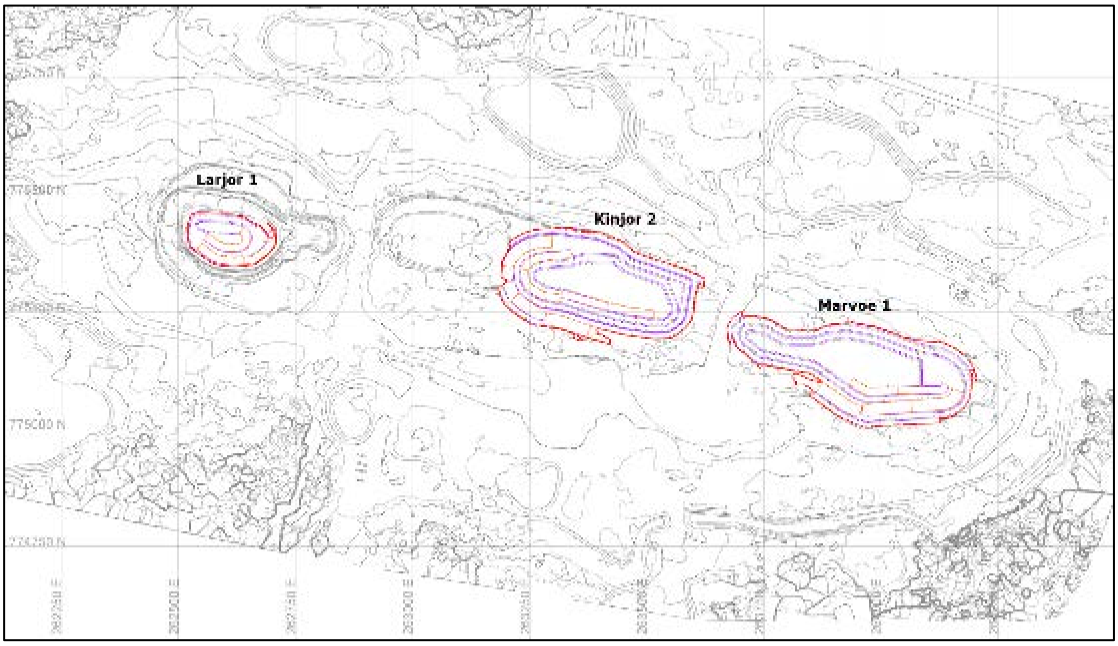


Figure 16-7: Pit Design – Stage 1, 2 & 3: Larjor 1, Marvoe 1 & Kinjor 2

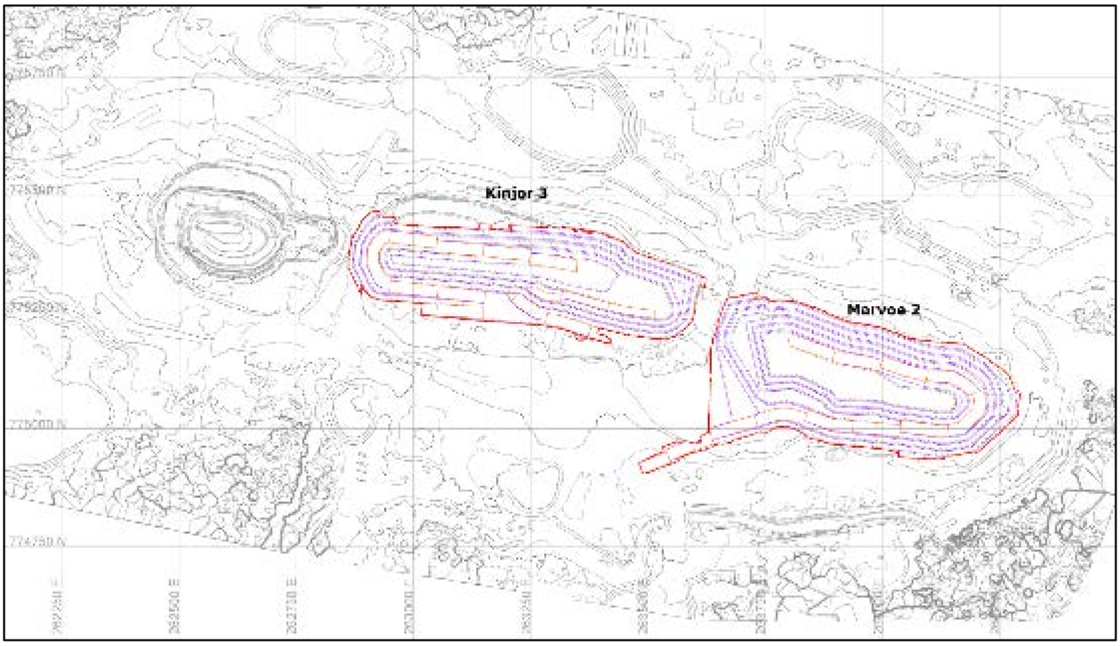


Figure 16-8: Pit Design – Stage 4 & 5: Kinjor 3 & Marvoe 2

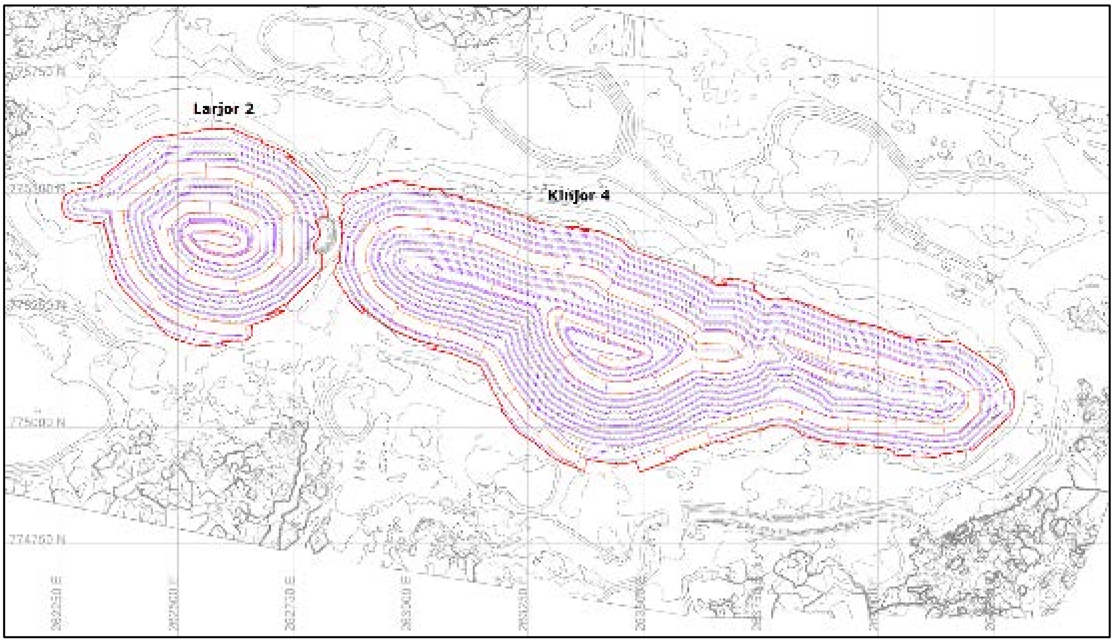


Figure 16-9: Pit Design – Stage 6 & 7: Kinjor 4 & Larjor 2

The pit design inventory is shown in Table 16-5. The average ore loss and dilution within the pit design is estimated at 3.3% and 13.5%, respectively at a cut-off grade of 0.85 g/t Au. The Inferred material has been treated as waste and represents 134 kt of the waste fresh.

Table 16-5: Pit Design Inventory

	Units	Total	Stage 1	Stage 2	Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
Total	**(kt)**	**124,626**	**873**	**5,189**	**5,741**	**6,710**	**14,976**	**67,312**	**23,823**
Total Waste	**(kt)**	**117,507**	**742**	**4,501**	**5,169**	**6,280**	**13,990**	**64,015**	**22,810**
Waste Fresh	(kt)	**113,248**	742	4,418	5,160	5,987	13,331	61,839	21,772
Waste Oxide	(kt)	**4,259**	0	83	9	294	659	2,176	1,038
Total RoM	**(kt)**	**7,118**	**131**	**688**	**572**	**430**	**986**	**3,298**	**1,014**
RoM Fresh	(kt)	**7,107**	131	684	572	430	986	3,298	1,006
RoM Oxide	(kt)	**12**	-	3	0		0	0	8
Au Grade	**(g/t)**	**3.08**	**3.73**	**3.01**	**2.92**	**3.21**	**2.65**	**3.28**	**2.87**
RoM Fresh	(g/t)	**3.09**	3.73	3.01	2.92	3.21	2.65	3.28	2.88
RoM Oxide	(g/t)	**1.90**	-	2.52	2.87	-	1.06	1.63	1.64
RoM: >=0.85 g/t Au									

Figure 16-10 to Figure 16-14 and Table 16-6 shows the comparison of the final pit design to the selected optimised shell. SRK notes that the differences are within acceptable tolerances.

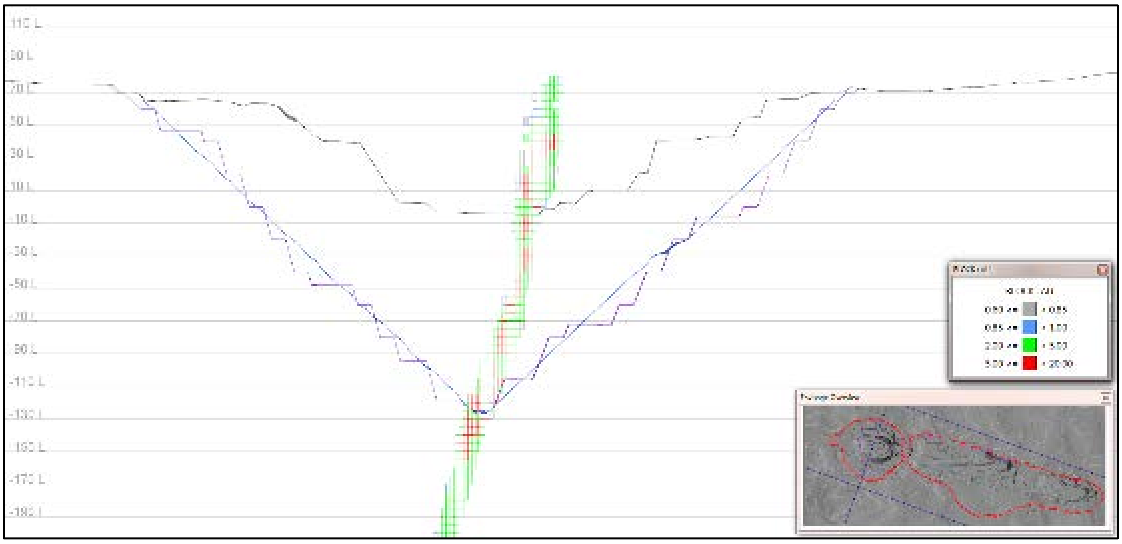


Figure 16-10: Comparison of Pit Design to Optimised Shell – Larjor

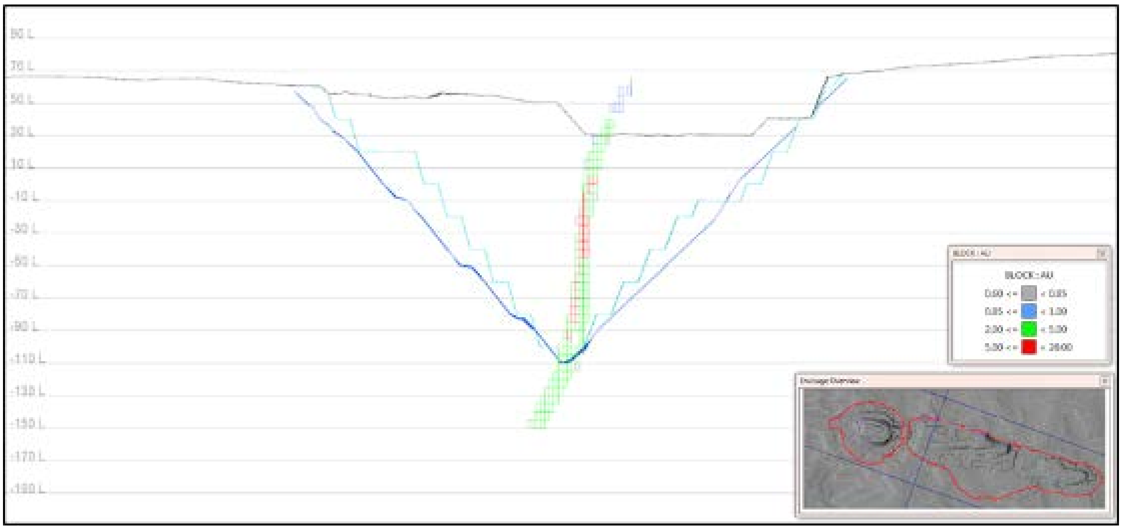


Figure 16-11: Comparison of Pit Design to Optimised Shell – Kinjor West

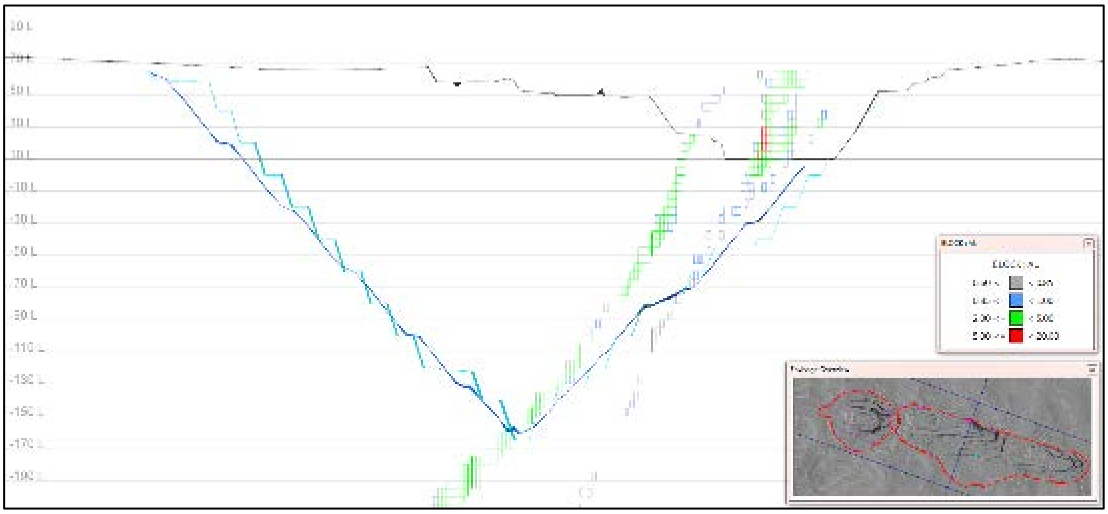


Figure 16-12: Comparison of Pit Design to Optimised Shell – Kinjor Central

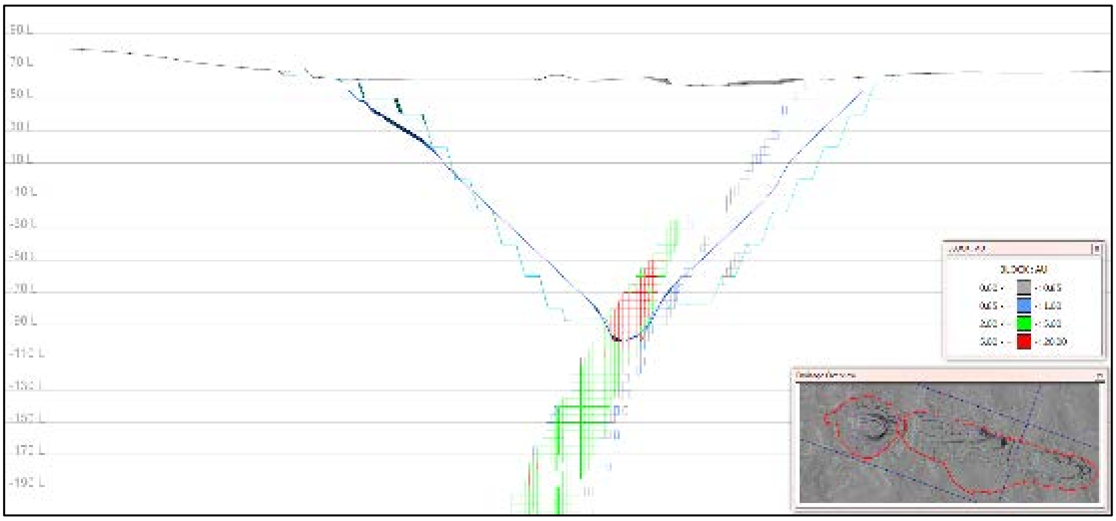


Figure 16-13: Comparison of Pit Design to Optimised Shell – Kinjor East

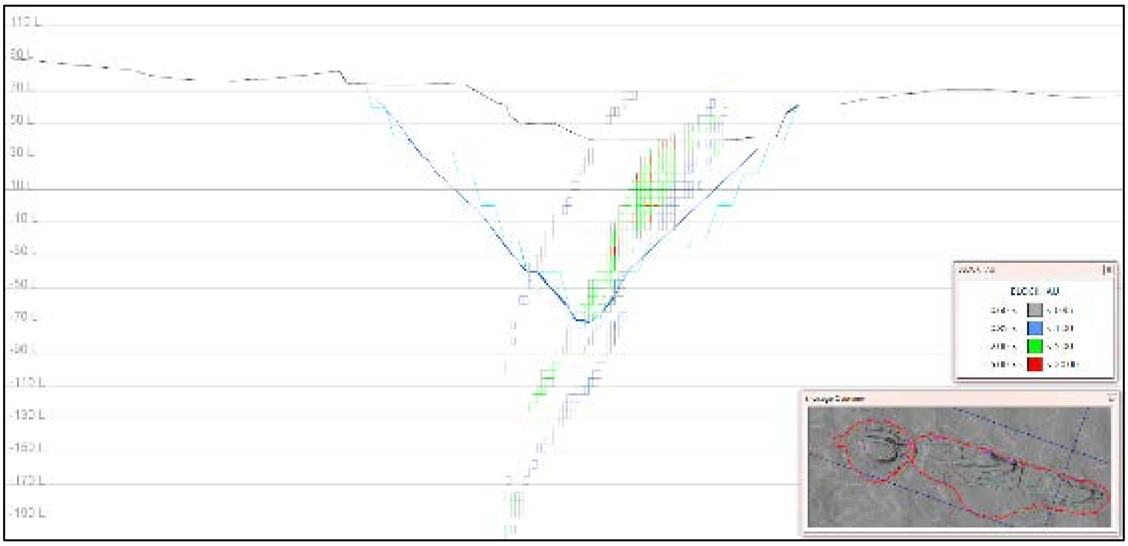


Figure 16-14: Comparison of Pit Design to Optimised Shell – Marvoe

Table 16-6: Comparison of Pit Design to Optimised Shell

Comparison	Units	Design	Shell	Difference	(%)
Total	(Mt)	124.6	121.8	2.8	2.3
Waste	(Mt)	117.5	114.4	3.1	2.7
RoM	(Mt)	7.1	7.3	-0.2	-2.5
	(g/t)	3.08	3.09	-0.01	-0.18
	(koz)	706	730	-24	-3
Strip Ratio	(t:t)	16.5	15.6	0.9	5.8
RoM: >=0.85 g/t Au					

16.5.3 Waste Dump Design

The WRD design parameters from the 2012 FS are shown in Table 16-7. Some modifications have been undertaken to the designs, in order to increase capacity, allow sufficient space for the air strip and the addition of backfill areas in the pit void. The design parameters have remained largely unchanged, however, the backfill areas have larger lift heights to facilitate dumping from existing levels, along with larger berms widths. The overall slope angles in the backfill areas remain at 18°.

Table 16-7: Waste Dump Design Parameters

Parameter	Units	Value
Overall Slope Angle	(°)	18
Lift Height	(m)	15
Berm Width	(m)	25
Batter Angle	(°)	33

In the 2012 FS, AMC stated that the 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.

The external WRD designs are shown in Figure 16-15 and the backfill dump design is shown in Figure 16-16. The backfill dumps include some extensions to the external dumps as additional space becomes available. The backfill will start once the Marvoe pits have been completed.

Some 600,000 bcm of fresh waste will also be required for tailings embankment construction from December 2017 to March 2018.

The waste dump capacities are shown in Table 16-8. Swell factors of 30% have been used for fresh waste and 20% swell factors for oxide waste.

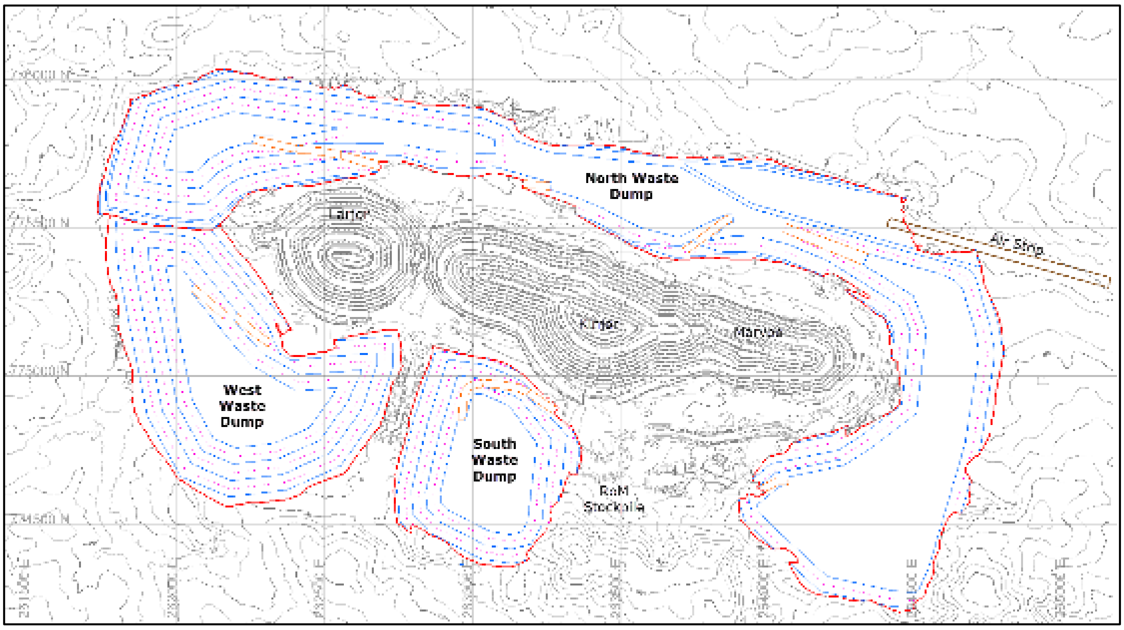


Figure 16-15: External Waste Dump Designs

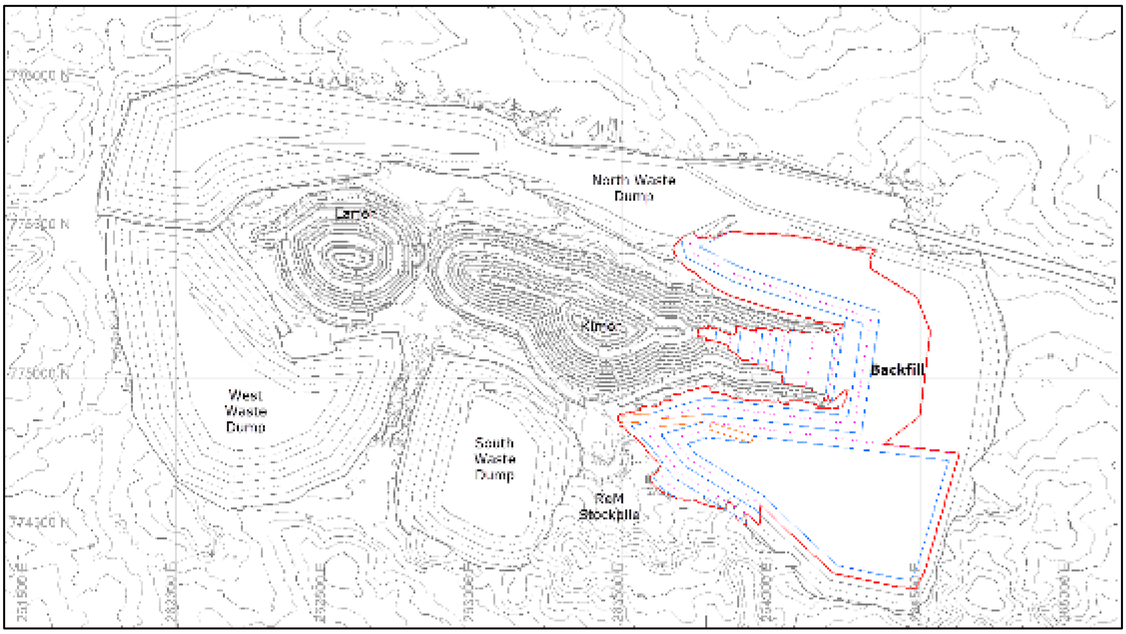


Figure 16-16: Backfill Dump Designs

Table 16-8: Waste Dump Design Capacity

Waste Dump	Pit Inventory (M bcm)	Pit Inventory (M lcm)	Design (M lcm)
North Waste Dump			19.6
West Waste Dump			13.3
South Waste Dump			9.4
Backfill			14.2
Tailings Embankment			0.6
Total	**41.6**	**53.9**	**57.0**

16.6 Mine Production Schedule

16.6.1 Approach

A life of mine plan was undertaken based on the following assumptions:

- Start date of 01 August 2017;
- Stockpile balance as of 01 August 2017 as shown in Table 16-9;
- Plant feed target of 3,945 tpd in 2017 and 4,603 tpd from January 2018;
- Oxide material will only be fed at the end of the mine life at a ratio of 1:7 with marginal material;
- Maximum vertical advance rate of 8 benches per year; and
- The limited of material movement quantities to ensure a maximum number of mining faces depending on the number of active stages.

Table 16-9: Stockpile Balance as of Aug. 1, 2017

Stockpile Balance	Quantity (kt)	Au (g/t)
RoM Fresh	97.8	1.25
RoM Oxide	151.6	1.50
Marginal Fresh	211.5	0.75
RoM: >=0.85 g/t Au		
Marginal: 0.70 to 0.85 g/t Au. Not used as plant feed in the mine schedule.		

16.6.2 Material Movement

The total material movement is shown monthly in Figure 16-17 and quarterly in Table 16-10. The material movement by stage is shown in Figure 16-18.

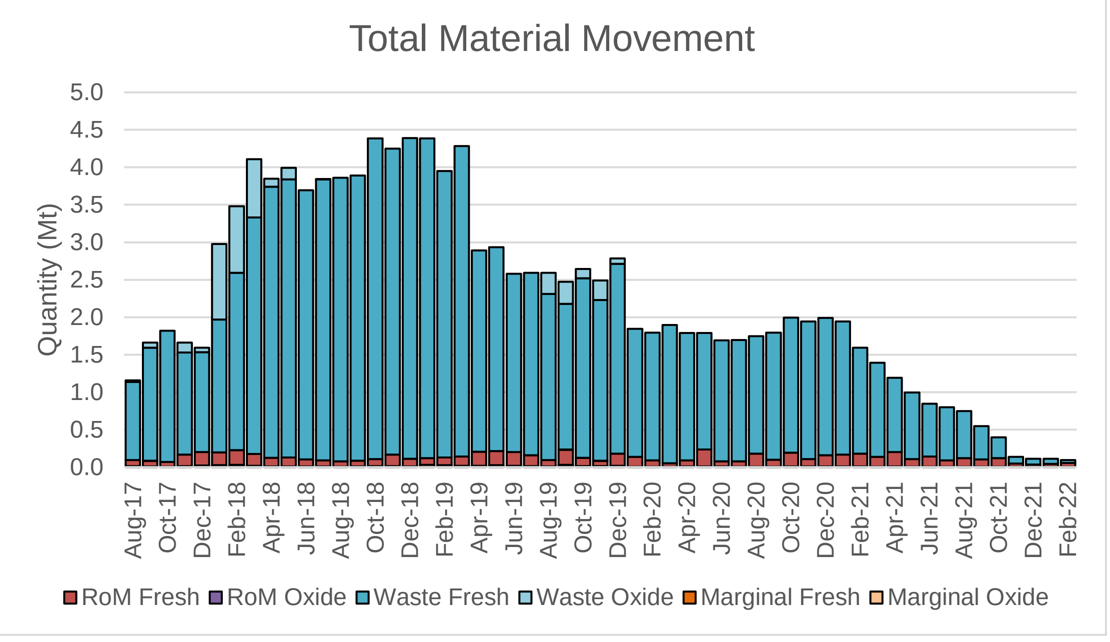


Figure 16-17: Mine Schedule: Total Material Movement by Material Type

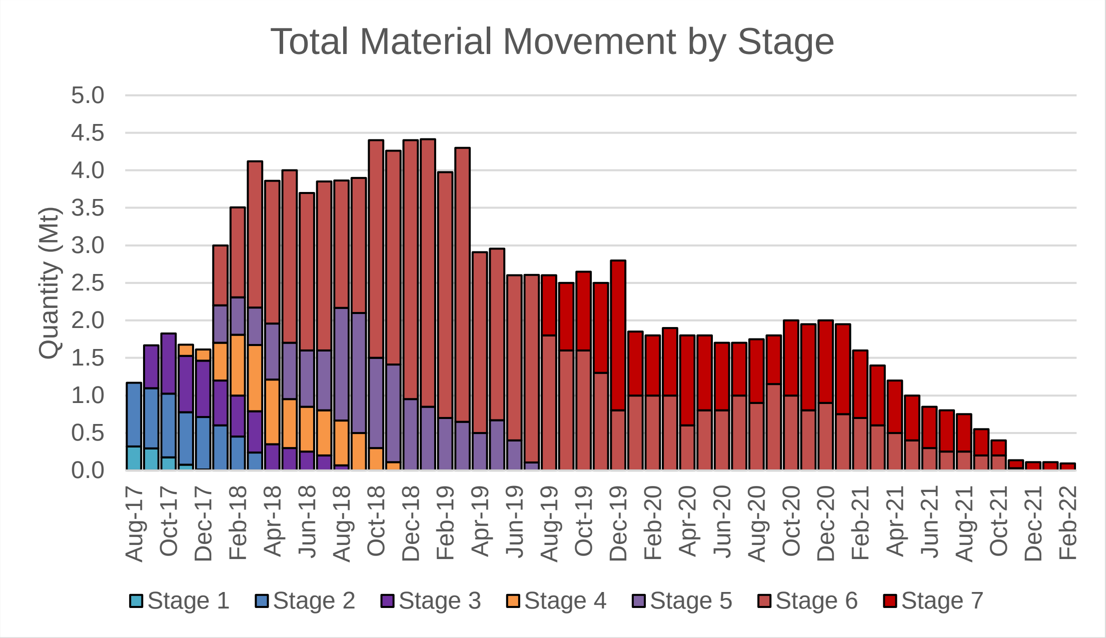


Figure 16-18: Mine Schedule: Total Material Movement by Stage

SRK notes that the production schedule is aggressive, with up to 8 benches mined per year and mine production targets significantly in excess of what has been achieved so far.

Actual production data up to September 2017 is shown in Figure 16-19. Production in September is already behind plan and the additional tonnage (645 kt) will need to be made up in the coming months to meet the forecast. The production data shows that mine production will need to triple by January 2018 and quadruple by March 2018 from current levels. Additional equipment and improved management practices are currently underway to ensure that these production levels are achievable; however, SRK notes that there is a risk to the plan should these improvements be delayed or not be achieved.

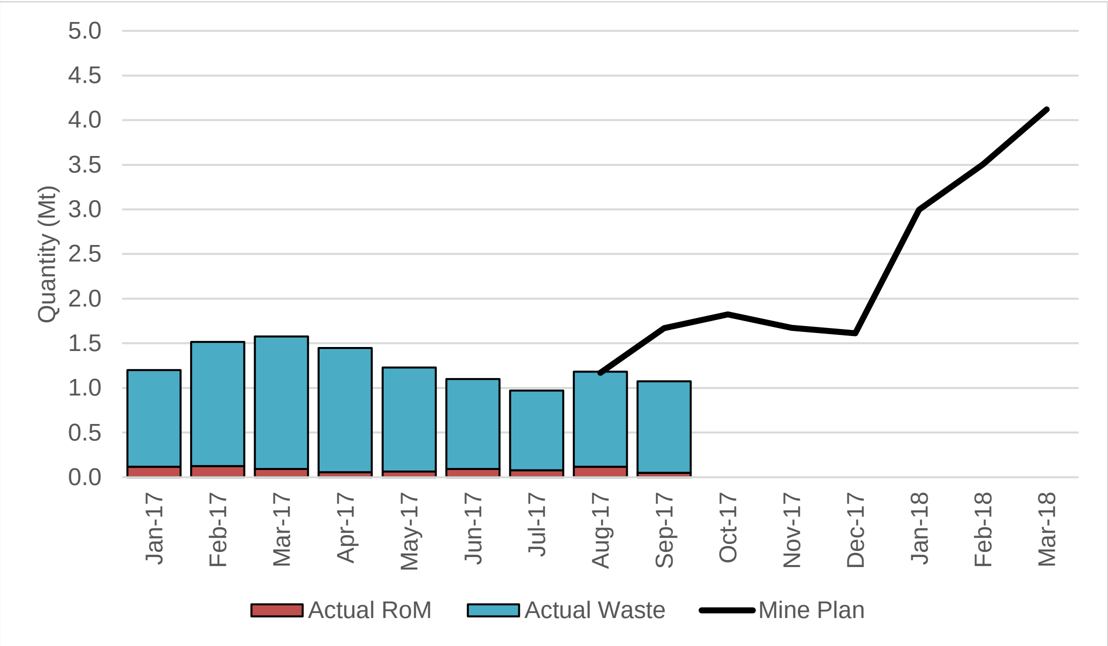


Figure 16-19: Mine Production: Actuals vs. Mine Plan

16.6.3 Plant Feed Schedule

The plant feed schedule is shown in Figure 16-20 with the gold grades shown in Figure 16-21. There is insufficient RoM Fresh feed material at the end of the mine life to meet the plant feed targets because this has been restricted to reasonable bench advance rates.

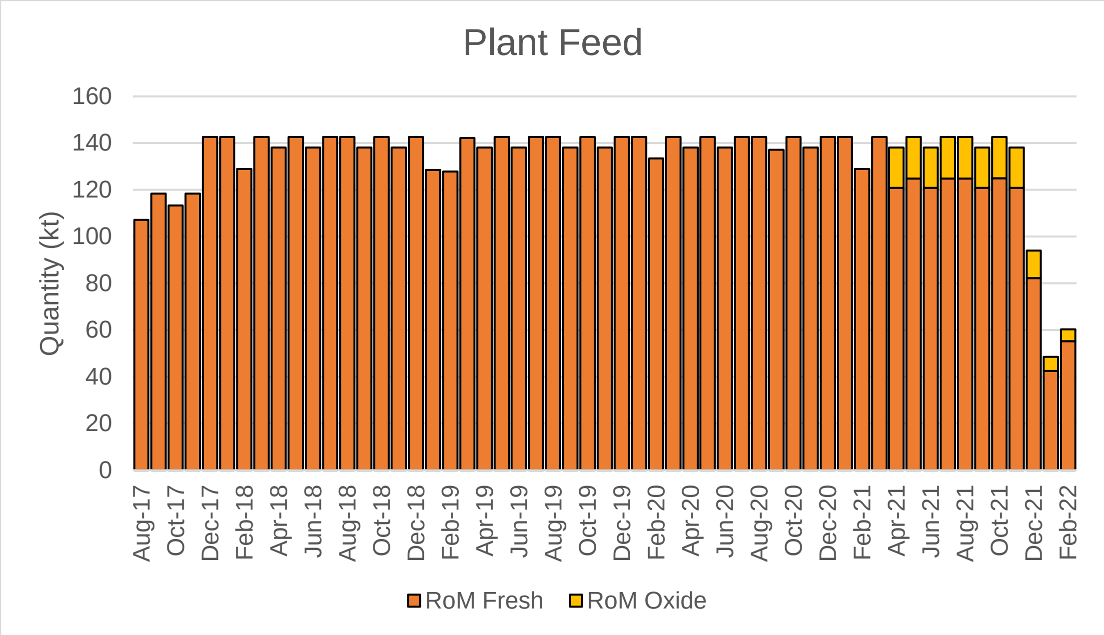


Figure 16-20: Plant Feed Schedule by Material Type

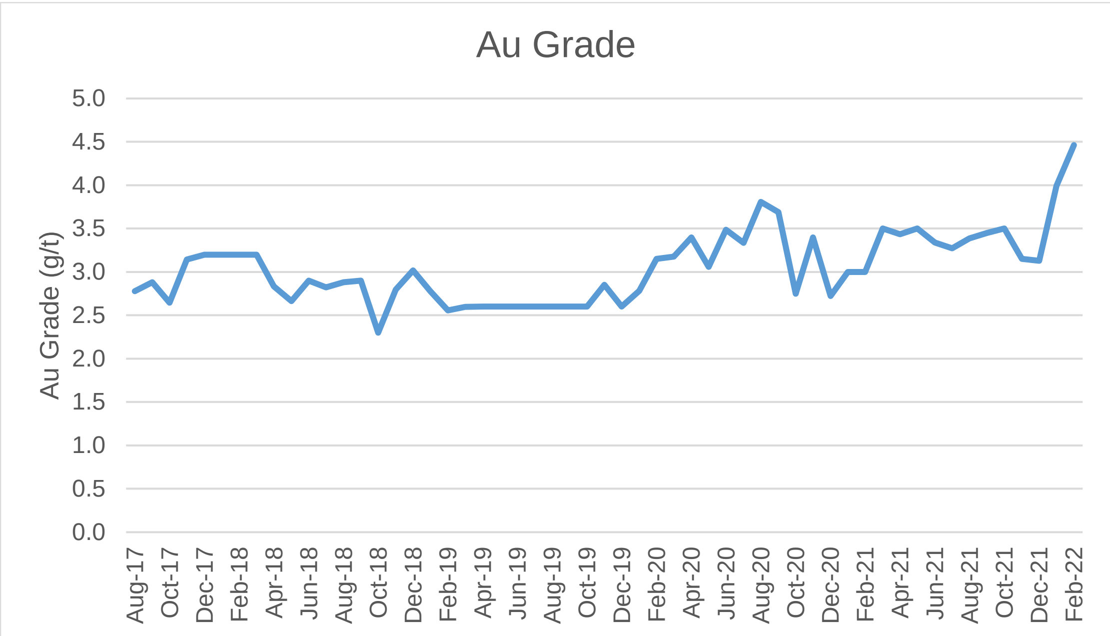


Figure 16-21: Plant Feed Grades

Table 16-10: Quarterly Mine Schedule

Mine Schedule	Units	Total	2017 Q3	2017 Q4	2018 Q1	2018 Q2	2018 Q3	2018 Q4	2019 Q1	2019 Q2	2019 Q3	2019 Q4	2020 Q1	2020 Q2	2020 Q3	2020 Q4	2021 Q1	2021 Q2	2021 Q3	2021 Q4	2022 Q1
Total Ex-Pit	**(kt)**	**124,626**	**2,839**	**5,111**	**10,624**	**11,559**	**11,616**	**13,061**	**12,691**	**8,467**	**7,705**	**7,950**	**5,550**	**5,300**	**5,250**	**5,950**	**4,950**	**3,050**	**2,100**	**648**	**203**
Waste	**(kt)**	**117,507**	**2,661**	**4,679**	**10,029**	**11,207**	**11,363**	**12,677**	**12,302**	**7,848**	**7,221**	**7,560**	**5,268**	**4,898**	**4,894**	**5,494**	**4,468**	**2,598**	**1,787**	**448**	**106**
Fresh	(kt)	**113,115**	2,568	4,481	7,344	10,947	11,362	12,649	12,271	7,824	6,641	7,101	5,268	4,898	4,886	5,467	4,468	2,598	1,787	448	106
Oxide	(kt)	**4,259**	93	191	2,677	260	0				580	458									
Inferred	(kt)	**134**	0	7	9		0	28	30	24				0	9	27	0	0	0		
RoM	**(kt)**	**7,118**	**178**	**433**	**595**	**353**	**253**	**384**	**389**	**619**	**484**	**390**	**282**	**402**	**356**	**456**	**482**	**452**	**313**	**200**	**98**
Fresh	(kt)	**7,107**	174	433	595	352	253	384	389	619	481	386	282	402	356	456	482	452	313	200	98
Oxide	(kt)	**12**	4		0	0					3	5									
Au Grade	**(g/t)**	**3.08**	**3.29**	**3.22**	**2.93**	**2.70**	**3.19**	**2.79**	**2.60**	**2.72**	**2.57**	**2.74**	**3.23**	**3.42**	**3.47**	**3.05**	**3.16**	**3.35**	**3.72**	**4.15**	**4.56**
Fresh	(g/t)	**3.09**	3.30	3.22	2.93	2.70	3.19	2.79	2.60	2.72	2.58	2.75	3.23	3.42	3.47	3.05	3.16	3.35	3.72	4.15	4.56
Oxide	(g/t)	**1.90**	2.52		1.33	0.99					1.48	1.75									
Plant Feed	**(kt)**	**7,368**	**225**	**374**	**414**	**419**	**423**	**423**	**398**	**419**	**423**	**423**	**419**	**419**	**423**	**423**	**414**	**419**	**423**	**375**	**109**
Fresh	(kt)	**7,205**	225	374	414	419	423	423	398	419	423	423	419	419	423	423	414	366	371	328	98
Oxide	(kt)	**163**																52	53	47	11
Au Grade	**(g/t)**	**3.03**	**2.83**	**3.01**	**3.20**	**2.80**	**2.87**	**2.70**	**2.64**	**2.60**	**2.60**	**2.68**	**3.03**	**3.31**	**3.61**	**2.95**	**3.17**	**3.43**	**3.37**	**3.28**	**4.25**
Fresh	(g/t)	**3.06**	2.83	3.01	3.20	2.80	2.87	2.70	2.64	2.60	2.60	2.68	3.03	3.31	3.61	2.95	3.17	3.70	3.63	3.53	4.56
Oxide	(g/t)	**1.53**																1.53	1.53	1.53	1.53

16.6.4 Stockpiling

The stockpile balance is shown in Figure 16-22. The RoM Oxide material is only fed at the end of the mine life in a 1:7 ratio with RoM Fresh. The RoM Fresh stockpile balance drops to extremely low levels in Q3 2017 and then again in Q3 2018 until Q1 2019 and again in 2020. The mine schedule is aggressive and therefore there are no opportunities to stockpile additional material to mitigate any changes. While the plant could be fed with RoM Oxide during these periods, should a shortfall arise the RoM Oxide material has lower grades and recoveries.

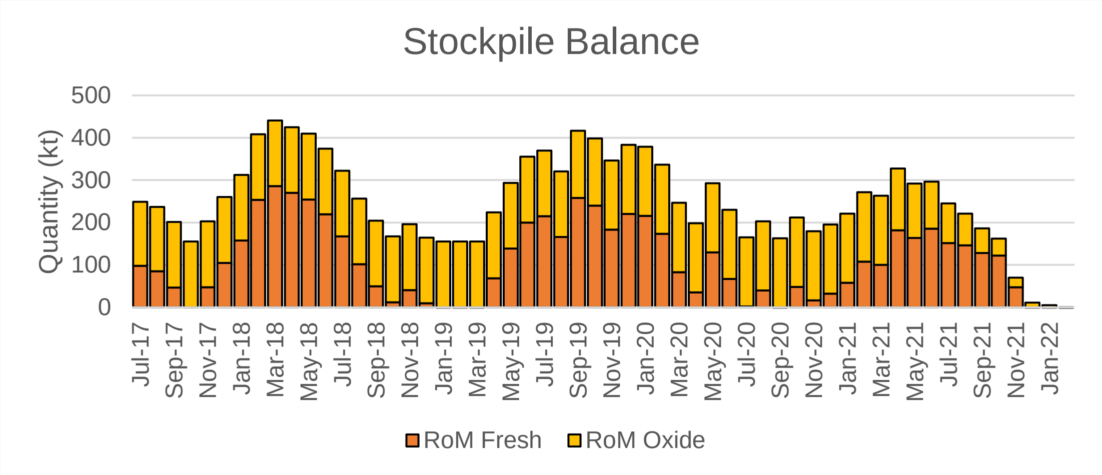


Figure 16-22: Stockpile Balance

16.7 Operating Strategy

16.7.1 Grade Control

The grade control strategy on site is characterised as follows:

- A series of channel sample lines are excavated across the mineralised zone on the cleaned (un-blasted) pit floor of every 5m bench. These channels are aligned perpendicular to the strike and are spaced at 10m intervals along the length of the mineralised zone. Channel samples are collected at 1m intervals.
- Channel sample results are plotted based on survey pick-ups and different grade ore blocks are defined. The results are used to define the blast and dig blocks for the bench.
- Waste zones adjacent to the higher grade zones and along strike from the mineralisation are blocked out for resampling ('RSP') whereby this material (after blasting) is transported to a resampling area where each truckload is sampled with four samples.
- When the RSP sample results are returned, the four results for each pile are reviewed and an 'average' grade is assigned to the pile. This material is then added to the ROM feed at that grade or discarded to the waste dump.
- Reverse circulation ("RC") drilling may be used when the conditions allow, to drill areas prior to mining in order to inform a grade control model which is updated onsite based on the results;
- The grade control drilling uses inclined holes (minus 60°) at a spacing of approximately 10m by 10m;
- No blasthole sampling is undertaken.

16.7.2 Drill & Blast

It has been assumed that all oxide material will be free-dig, while 100% of the fresh material will require drilling and blasting. It has been assumed the mineralised areas and some of the waste will be drilled with the Small Drill (currently Sandvik 1500i), while the remaining waste will be drilled with the Large Drill (ex. Atlas Copco D65) from March 2018. The drilling parameters are shown in Table 16-11.

Table 16-11: Drilling Parameters

Drilling Parameters	Units	Waste	Waste	RoM
Drill		Large Drill	Small Drill	Small Drill
Bench Height	(m)	10.0	10.0	5.0
Hole Diameter	(mm)	203	140	127
Subdrill	(m)	1.0	1.0	0.5
Spacing	(m)	6.7	5.0	3.5
Burden	(m)	5.8	4.5	3.0
Penetration Rate	(m/hr)	34.8	34.8	34.8
Drill time per Hole	(min.)	26.6	26.6	12.9
Productivity per meter	(m/doh)	24.8	24.8	25.5
Productivity per tonne	(t/doh)	2,507	1,467	718
	(Mtpa)	11.0	6.4	3.1

The final pit walls will require pre-split drilling with the Small Drill at a spacing of 1.5 m.

The blasting parameters are shown in Table 16-12.

Table 16-12: Blasting Parameters

Blasting Parameters	Units	Waste	Waste	RoM
Drill		Large Drill	Small Drill	Small Drill
Explosive Density	(t/m^3)	1.30	1.30	1.30
Charge Height	(m)	6.5	8.0	3.0
Powder Factor	(kg/m^3)	0.71	0.71	0.94
Powder Factor	(kg/t)	0.25	0.25	0.32

16.7.3 Load & Haul

The truck and shovel operation consists of a 12 m^3 backhoe (PC2000) and 6 m^3 backhoes (PC1250) along with owner-operated 90 t haul trucks (Komatsu 785-7) and leased Caterpillar (“CAT”) 777B 90 t haul trucks. Some articulated trucks (“ADT”) are used for development work and to provide extra capacity to the production fleet. A front-end loader is used at the crusher.

The loading productivities have been estimated based on time trials undertaken at New Liberty and are shown in Table 16-13.

The PC2000 productivities have been significantly lower than predicted, averaging 753 t/op. hr. Mining in the Oxide material has decreased the productivities as expected, however, availability of the PC2000 is expected to improve in order to meet the planned productivities going forward. The average productivity for the PC1250s since January 2017 is 445 t/op. hr. Again, starting the Marvoe pit in the Oxide has led to lower productivities, and given that these are lower than expected this will need to improve to meet the mine plan.

Table 16-13: Loading Productivities

Loading Parameter	Units	Waste Fresh	Waste & RoM Fresh	Waste & RoM Fresh	Waste Oxide	Waste & RoM Oxide	RoM Fresh Reclaimed	RoM Oxide Reclaimed
Loading								
Loading Unit		PC2000	PC1250	PC1250	PC2000	PC1250	Primary Wheel Loader	Primary Wheel Loader
Bucket Size	(m^3)	12.0	6.7	6.7	12.0	6.7	6.4	6.4
Loading Spot Time	(min.)	1.30	1.20	1.20	1.30	1.20	0.70	0.70
Loading Cycle Time	(min.)	0.52	0.50	0.50	0.52	0.50	0.60	0.60
First Bucket Dump	(min.)	0.25	0.20	0.20	0.25	0.20	0.10	0.10
Haulage								
Truck		Haul Trucks 90t	Haul Trucks 90t	ADTs 40t	Haul Trucks 90t	Haul Trucks 90t		
Capacity	(t)	90	90	39.5	90	90		
Capacity	(m^3)	80	80	30	80	80		
Dump & Spot Time	(min.)	1.2	1.2	1.2	1.2	1.2		
Loading Parameters								
Bucket Fill Factor	(%)	90	90	90	90	90	100	100
In-Situ Density	(t/bcm)	2.89	2.89	2.89	1.77	1.77	2.94	1.84
Swell Factor	(lcm/bcm)	1.3	1.3	1.3	1.2	1.2	1.3	1.2
Loose Density	(t/lcm)	2.22	2.22	2.22	1.47	1.47	2.27	1.53
Moisture Factor	(%)	5	5	5	5	5	5	5
Passes	(#)	4	7	3	6	10	1	1
Loaded Quantity	(t)	90.0	90.0	39.5	90.0	90.0	14.5	9.8
Loading Productivity								
Total Loading Cycle Time	(min.)	3.11	4.4	2.4	4.15	5.9	0.8	0.8
Loader Productivity	(t/doh)	1,654	1,169	940	1,239	872	1,036	699
	(t/op. hr)	939-1,139	664-805	534-648	704-853	495-600	588-713	397-481

doh: direct operating hour
op. hr: machine operating hour

DeswikCAD's Landform and Haulage tool has been used to model and estimate the waste deposition sequence. Wireframe solids have been used to represent the available dumping volume for the waste dumps and backfill areas.

A haulage network consisting of strings was used to represent in-pit haulage, pit ramps, ex-pit haulage and on-dump haulage. This network was used to estimate haulage distances and travel times between mining solids, RoM stockpiles and dump solids. The parameters used in the haulage estimate are shown in Table 16-14.

Table 16-14: Haulage Estimate Parameters

Deswik Truck Type File	Rolling Resistance (%)	Maximum Gradient (%)
Komatsu 785	3.0	10

Theoretical period by period travel times and haulage distances were estimated based on the haul profile and the retard and rimpull curves for the specified truck type.

SRK has applied a 90% factor to de-rate the estimated theoretical travel times resulting from the haulage simulation.

The haulage cycle times and fleet average haulage productivities per operating hour are shown in Figure 16-23.

The haul profiles are characterised by increasing waste hauls as the pits get deeper and available waste space gets further away. The dips highlight when backfilling space becomes available. The RoM haulage profiles are generally increasing as the pits get deeper.

The average productivity for the 90 t haul trucks was 223 t/op. hr since January 2017, which will need to increase to an average of 283 t/op. hr for the remainder of 2017.

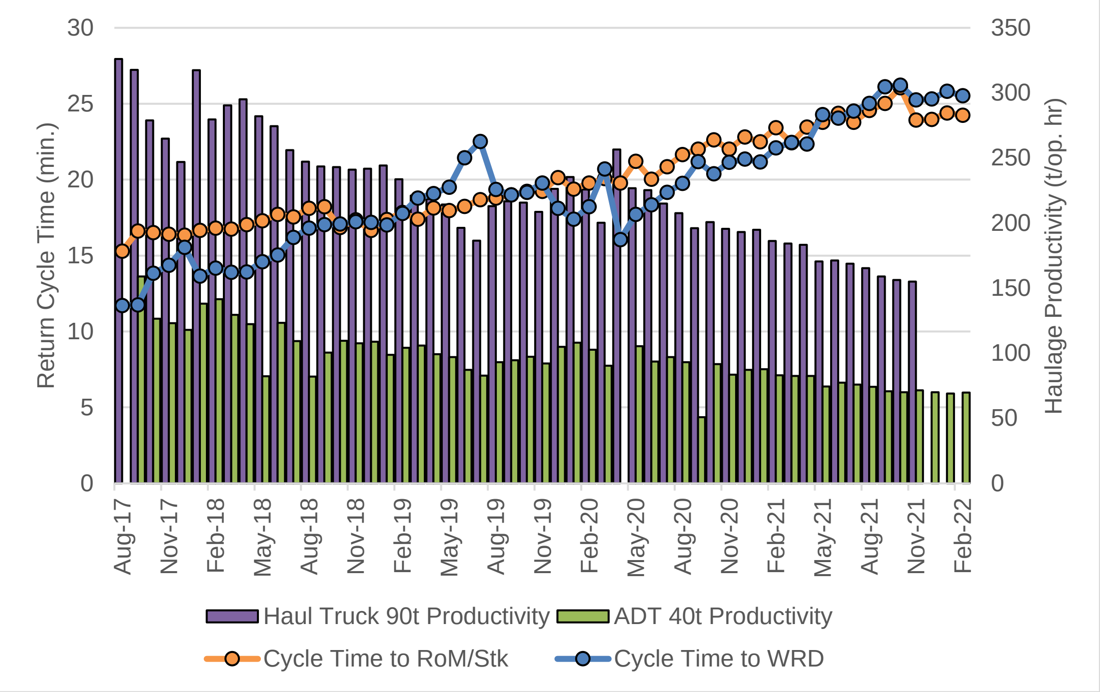


Figure 16-23: Haulage Cycle Times & Productivities

16.7.4 Equipment Operating Time

The equipment list used to develop the cost estimates in this study is shown in Table 16-15, which is based on the current equipment on site and expected purchases.

Table 16-15: Mining Equipment List

Equipment	Make	Model	Description
PC2000	Komatsu	PC2000	12 m^3 excavator
PC1250	Komatsu	PC1250SP-8R	6.7 m^3 excavator
Backhoe	Caterpillar	374DL ME	3 m^3 excavator
Haul Trucks 90t	Komatsu	HD785-7	91 t haul truck
Haul Trucks 90t Lease	Caterpillar	777B	91t haul truck
ADTs 40t	Caterpillar	740B	40 t articulated truck
Water Truck	Komatsu	HD465-7 WC	
Primary Rockbreaker	Komatsu	PC350-8	
Primary Track Dozer	Caterpillar	D9R	
Secondary Track Dozer	Komatsu	D275A	
Wheel Dozer	Caterpillar		
Small Drill	Sandvik	D1500i	Production drill 102-152mm
Large Drill	Atlas Copco*	D65	Production drill 110-203mm
Grade Control Drill	Sandvik	DR560RC	Grade Control Drill
Primary Grader	Komatsu	GD825A-2	
Secondary Grader	Komatsu	GD655-5	
Primary Wheel Loader	Caterpillar	992	
Secondary Wheel Loader	Caterpillar	988	
Crane Truck	Iveco	320E34	
Medium Service Truck	Astra	HD9	6x6 with P9000 module
Emulsion Truck			
Tire Handler	Komatsu	WAS740-5	
Welder	Lincoln	Air Vantage 500	Mobile welder
Air Compressor	Atlas Copco	XAS 97DD	Mobile compressor
Primary Pump	Sykes	HH160i	
Secondary Pump	Pioneer	PP66S	
Lighting Plant	Technogen	MT9810	
Light Vehicle	Toyota	Hilux	
Mine Bus			

* Equipment not yet purchased, the make and model stated are provided as a reference only.

The mining equipment operating time has been developed based on inputs from site, based on mechanical losses, operating standby and operational delays. The mining equipment operating time is shown in Table 16-16.

In comparison to actual availability, the shovels have been underperforming with an average availability of 63% since Jan. 2017 for the PC1250 and 60% for the PC2000. The mine plan only requires up to 4 loading units for the remainder of 2017 and there are currently 5 units on site, with three additional purchases expected before the end of the year. SRK expects that should availability levels not increase; the additional excavators can be used instead. Availability of the excavators should continue to be monitored and the mine plan adjusted should availabilities not improve.

The haul trucks have generally been performing well since January 2017, with some decreases in June to August. These dips were resolved by the use of leased trucks during these periods.

The use of availability for the excavators has generally matched the cyclical seasons, which are planned in the operating time model. The haul trucks have seen an increase in use of availability from February 2017 which indicates an improving trend.

Table 16-16: Mining Equipment Operating Time

Equipment	Calendar Time (hr/yr)	Availability (%)	Use of Availability (%)	Operating Efficiency (%)	Effective Utilisation (%)	Direct Operating Time (hr/yr)
PC2000	8,760	90	80	90	65	5,707
PC1250	8,760	90	80	90	65	5,707
Backhoe	8,760	85	58	90	44	3,864
Haul Trucks 90t	8,760	90	80	90	65	5,707
Haul Trucks 90t Lease	8,760	70	75	90	47	4,130
ADTs 40t	8,760	85	82	90	62	5,473
Water Truck	8,760	85	70	90	54	4,702
Primary Rockbreaker	8,760	85	70	90	54	4,702
Primary Track Dozer	8,760	85	70	90	54	4,702
Secondary Track Dozer	8,760	85	70	90	54	4,702
Wheel Dozer	8,760	85	45	90	35	3,027
Small Drill	8,760	80	78	80	50	4,379
Large Drill	8,760	80	78	80	50	4,372
Grade Control Drill	8,760	81	64	80	41	3,628
Primary Grader	8,760	85	64	90	49	4,283
Secondary Grader	8,760	85	64	90	49	4,283
Primary Wheel Loader	8,760	85	58	90	44	3,864
Secondary Wheel Loader	8,760	85	58	90	44	3,864
Crane Truck	8,760	85	45	90	35	3,027
Medium Service Truck	8,760	85	45	90	35	3,027
Emulsion Truck	8,760	85	20	90	15	1,351
Tire Handler	8,760	85	45	90	35	3,027
Welder	8,760	85	45	90	35	3,027
Air Compressor	8,760	85	45	90	35	3,027
Primary Pump	8,760	85	48	90	37	3,239
Secondary Pump	8,760	85	48	90	37	3,239
Lighting Plant	8,760	85	45	90	35	3,027
Light Vehicle	8,760	85	45	90	35	3,027
Mine Bus	8,760	85	45	90	35	3,027

16.7.5 Stockpile Strategy

There are currently three stockpiles as shown in Table 16-9, RoM Fresh, RoM Oxide and Marginal Fresh. The Marginal Fresh material is not fed to the plant and therefore will remain in the stockpile as waste.

All material is stockpiled prior to being fed to the plant. The Primary Wheel Loader (CAT 992) is used to feed the crusher directly.

16.7.6 Open-Pit Dewatering

Dewatering was a challenge during the height of the 2017 wet season and production was affected by heavy rains during Q3 2017. BMMC reports that this was the wettest quarter (July to September 2017) on record at New Liberty (2,253 mm vs 1,917 mm total rainfall). Dewatering was affected as a result of insufficient pumping capacity to deal with this peak. Three additional Global Pumps were procured to increase the pumping capacity. The three Sykes pumps worked in the pits from July to September, but there was often one on breakdown. Mining activity was focused in Larjor and Marvoe and all the available Global and Sykes pumps were installed in these pits. The challenge with insufficient pumping capacity for this peak rain period resulted in some flooding in the Kinjor Pit (which was not active). The mining activity was focused on the Larjor and Marvoe Pits in the quarter as they provided the best source of ore supply to the ROM Pad.

Additional measures have been put in place in order to prepare for next wet season. These include:

- 4 Global High Performance High Head pumps with maximum head of 227m have been ordered to increase the pumping capacity of the mine.
- 8-inch pipes are being used for pit dewatering instead of 6-inch pipes as previously used.
- The pits have been re-designed to provide enough room for stage pumping. The final Pushback of Larjor pit has been designed to have two stage pumping. There will be three stage pumping in Kinjor-Marvoe final pushback.
- Additional drains have been prepared to reinforce the diversion of surface run-off water away from the pits.

16.7.7 Mine Infrastructure

The mining infrastructure is detailed in Section 18 as part of the general site infrastructure requirements of the Project.

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.

16.8 Mine Equipment & Labour Requirements

16.8.1 Drilling

The drilling fleet requirements over the life of mine are shown monthly in Figure 16-24.

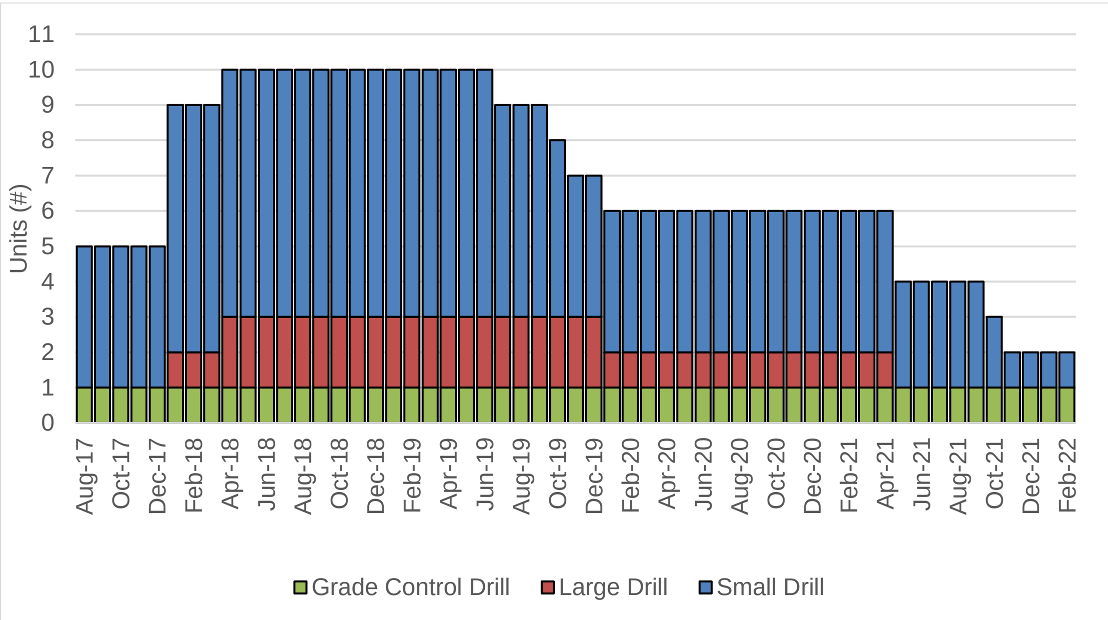


Figure 16-24: Drilling Fleet Requirements

16.8.2 Loading

The loading fleet requirements are shown in Figure 16-25. A single PC2000 is used until Q2 2021. The smaller PC1250 fleet will require up to 6 units from Q1 2018 until the end of Q1 2019. A Primary Wheel Loader is always required at the crusher.

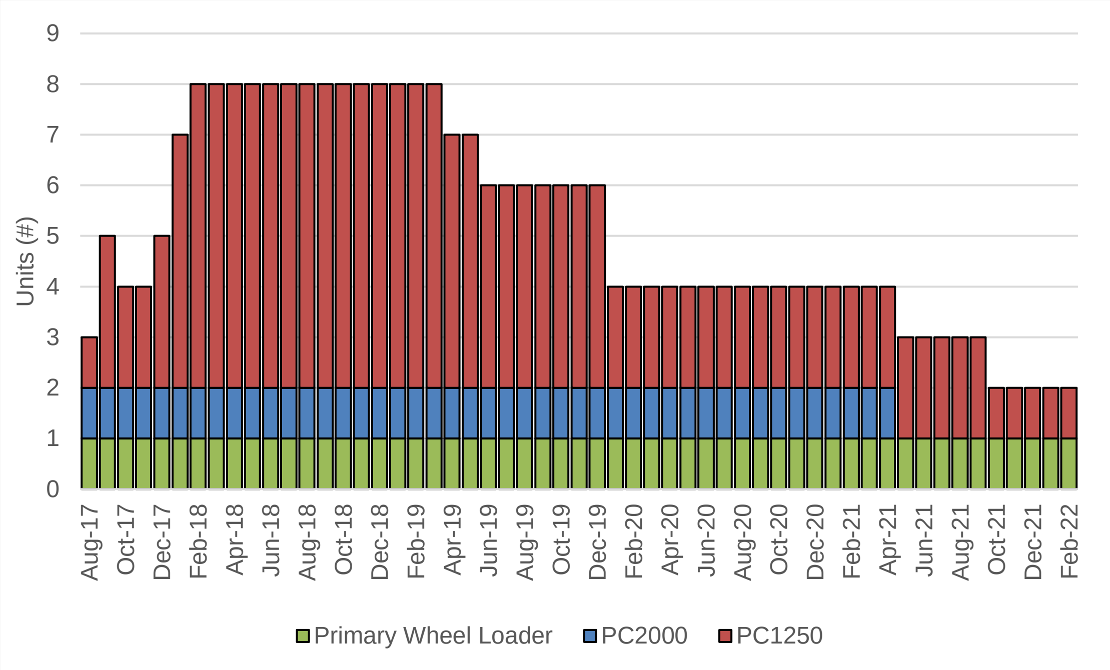


Figure 16-25: Loading Fleet Requirements

There will be limited space available in the pits with up to 7 loading units in 2018 to 2019. Appropriate and effective management and operating approaches will therefore be required to ensure the additional loading units within the pits does not impact on production targets. BMMC is confident that production will not be impacted, however, SRK highlights this risk should management and operations not improve prior to Q1 2018. SRK has also recommended that larger loading units are investigated to help limit the number of units in a mining stage.

16.8.3 Hauling

The hauling fleet requirements are shown in Figure 16-26. A maximum of 21 Komatsu 785 trucks will be supported by five 40 t articulated trucks (“ADTs”). Starting in February 2018 CAT 777B (90 t) trucks will be leased, up to 16 trucks in March 2019.

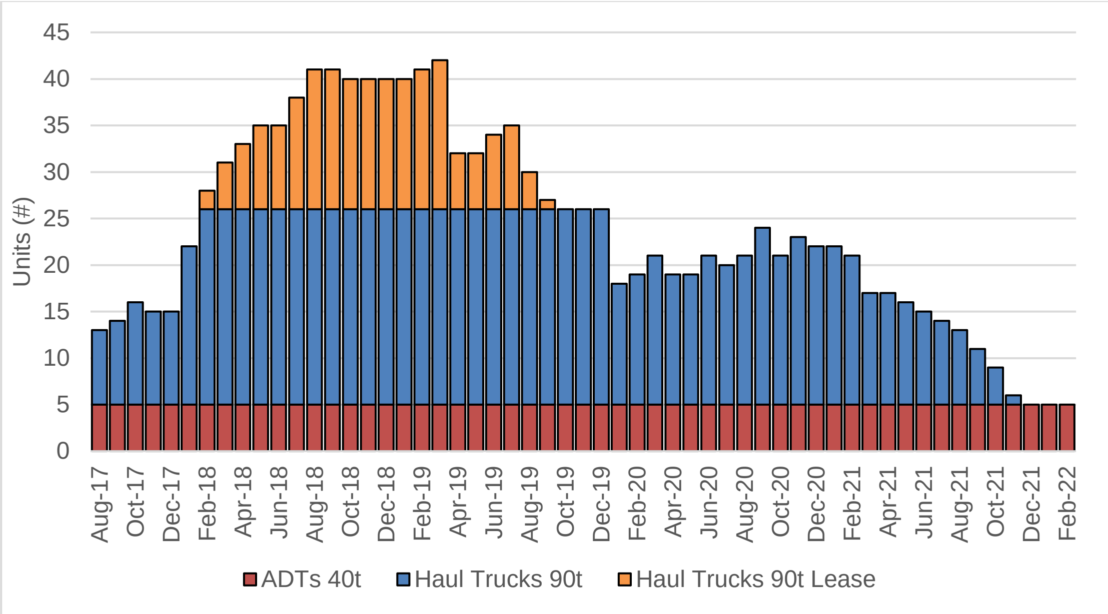


Figure 16-26: Loading Fleet Requirements

16.8.4 Ancillary Equipment

The mobile equipment ancillary fleet requirements have been based on material movement rates, labour requirements, total load and haul units, the number of working areas and manufacturer recommendations. The ancillary fleet is planned to support the load and haul operation and perform general pit and dump support activities. These activities include support for dust suppression, tailings embankment construction, surface water diversion construction, haul road maintenance, in-pit maintenance of tracked equipment, lighting of working areas and personnel transport. The equipment requirements are shown by period in Table 16-17.

Table 16-17: Mine Equipment Requirements

Equipment	Units	Maximum	Q3 2017	Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	Q1 2019	Q2 2019	Q3 2019	Q4 2019	Q1 2020	Q2 2020	Q3 2020	Q4 2020	Q1 2021	Q2 2021	Q3 2021	Q4 2021	Q1 2022
PC2000	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
PC1250	(#)	**6**	3	3	6	6	6	6	6	5	4	4	2	2	2	2	2	2	2	1	1
Backhoe	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Trucks 90t	(#)	**21**	9	11	21	21	21	21	21	21	21	21	16	16	19	18	17	12	9	4	-
Haul Trucks 90t Lease	(#)	**16**	-	-	5	9	15	14	16	8	9	-	-	-	-	-	-	-	-	-	-
ADTs 40t	(#)	**5**	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Water Truck	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Primary Rockbreaker	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Primary Track Dozer	(#)	**4**	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3	3	2
Secondary Track Dozer	(#)	**2**	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Wheel Dozer	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Small Drill	(#)	**7**	4	4	7	7	7	7	7	7	6	5	4	4	4	4	4	4	3	2	1
Large Drill	(#)	**2**	-	-	1	2	2	2	2	2	2	2	1	1	1	1	1	1	-	-	-
Grade Control Drill	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Primary Grader	(#)	**2**	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Secondary Grader	(#)	**1**	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Primary Wheel Loader	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Secondary Wheel Loader	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Crane Truck	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Medium Service Truck	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Emulsion Truck	(#)	**2**	1	1	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1
Tire Handler	(#)	**2**	1	1	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1
Welder	(#)	**2**	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Air Compressor	(#)	**2**	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Primary Pump	(#)	**7**	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Secondary Pump	(#)	**1**	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant	(#)	**32**	17	20	30	32	32	32	32	30	28	26	20	20	20	20	20	19	15	12	11
Light Vehicle	(#)	**16**	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	14	14	14	9
Mine Bus	(#)	**3**	2	2	3	3	3	3	2	2	2	2	2	2	2	2	2	2	1	1	1

16.8.5 Labour Requirements

The annual labour requirements are shown annually in Table 16-18 and monthly in Figure 16-27.

Table 16-18: Mine Labour Requirements

Labour Roles	Units	2017	2018	2019	2020	2021	2022
Labour Requirements	**(#)**	**609**	**938**	**814**	**533**	**521**	**256**
Mine Operations	**(#)**	**500**	**760**	**636**	**404**	**396**	**183**
Expat	**(#)**	**44**	**44**	**44**	**22**	**22**	**9**
Manager	(#)	2	2	2	2	2	2
Superintendent	(#)	4	4	4	4	4	1
Supervisor	(#)	13	13	13	13	13	5
Administrator	(#)	1	1	1	1	1	1
Operator	(#)	24	24	24	2	2	0
Local	**(#)**	**456**	**716**	**592**	**382**	**374**	**174**
Supervisor	(#)	17	17	17	17	17	13
Technicians	(#)	35	67	67	37	37	9
Operator	(#)	404	632	508	328	320	152
Mine Maintenance	**(#)**	**93**	**157**	**157**	**113**	**109**	**62**
Expat	**(#)**	**1**	**1**	**1**	**1**	**1**	**1**
Supervisor	(#)	1	1	1	1	1	1
Local	**(#)**	**92**	**156**	**156**	**112**	**108**	**61**
Superintendent	(#)	1	1	1	1	1	1
Supervisor	(#)	3	3	3	3	3	2
Planner	(#)	2	2	2	2	2	1
Administrator	(#)	2	2	2	2	2	1
Workers	(#)	84	148	148	104	100	56
Technical Services	**(#)**	**16**	**21**	**21**	**16**	**16**	**11**
Expat	**(#)**	**11**	**13**	**13**	**11**	**11**	**8**
Manager	(#)	4	4	4	4	4	2
Supervisor	(#)	3	3	3	3	3	2
Engineer	(#)	2	4	4	2	2	2
Geologist	(#)	1	1	1	1	1	1
Technicians	(#)	1	1	1	1	1	1
Local	**(#)**	**5**	**8**	**8**	**5**	**5**	**3**
Geologist	(#)	2	3	3	2	2	1
Operator	(#)	2	3	3	2	2	1
Administrator	(#)	1	2	2	1	1	1

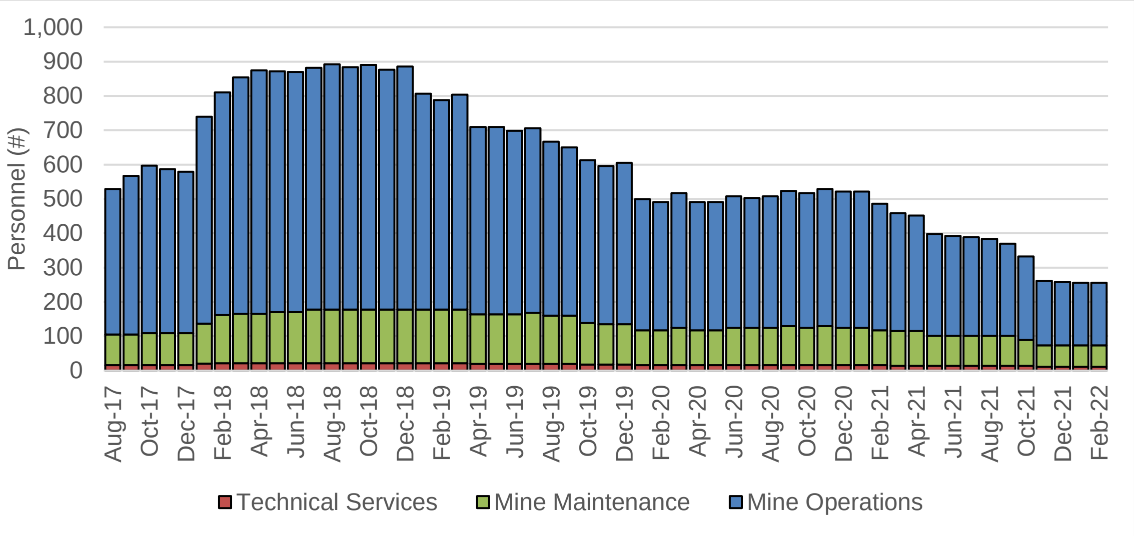


Figure 16-27: Mine Labour Requirements

16.9 Conclusions

The conclusions from the life of mine study are summarised below:

- The updated mine designs based on the USD1,300/oz optimised shell result in 7.1 Mt of RoM at 3.08 g/t Au with 117.5 Mt of waste at a cut-off of 0.85 g/t Au.
- Average ore loss and dilution values are 3.3% and 13.5%, respectively within the pit design.
- Significant improvements are expected with the new grade control programme to reduce current levels of ore loss and dilution.
- The mine schedule produces 1.64 ktpa of mill feed, totalling 7.4 Mt at an average grade of 3.03 g/t Au. The average strip ratio is 16.5 with 117.5 Mt of waste. Total material movement will average 3,905 kt/month in 2018 (totalling 46.9 Mt).
- The mine schedule is aggressive with up to 8 benches mined per year.
- Mine production quantities will need to triple by January 2018 and quadruple by March 2018 from current production levels.
- There are a number of periods when there is insufficient RoM Fresh material available on the stockpile to mitigate any shortfalls. Should any shortfalls arise, additional material will be able to be sourced from the RoM Oxide stockpile but this is lower grade and will have lower recoveries.
- One 12 m^3 backhoe and up to six 6 m^3 backhoes will continue to be used with 90 t haul trucks supported by 40 t ADTs. Up to 16 90 t haul trucks will need to be leased from February 2018 to support the mine plan.
- Significant improvements in availability and productivity of the excavators and trucks is required to meet the mine plan.
- A maximum of 892 personnel are required at peak material movement (2018), with 714 in mine operations, 157 in mine maintenance and 21 in technical services. The personnel requirements in 2018 are significantly higher than current levels (approximately 529). BMMC will need to recruit sufficient qualified personnel in order to meet the mine plan.

17 RECOVERY METHODS

17.1 Plant Design Criteria

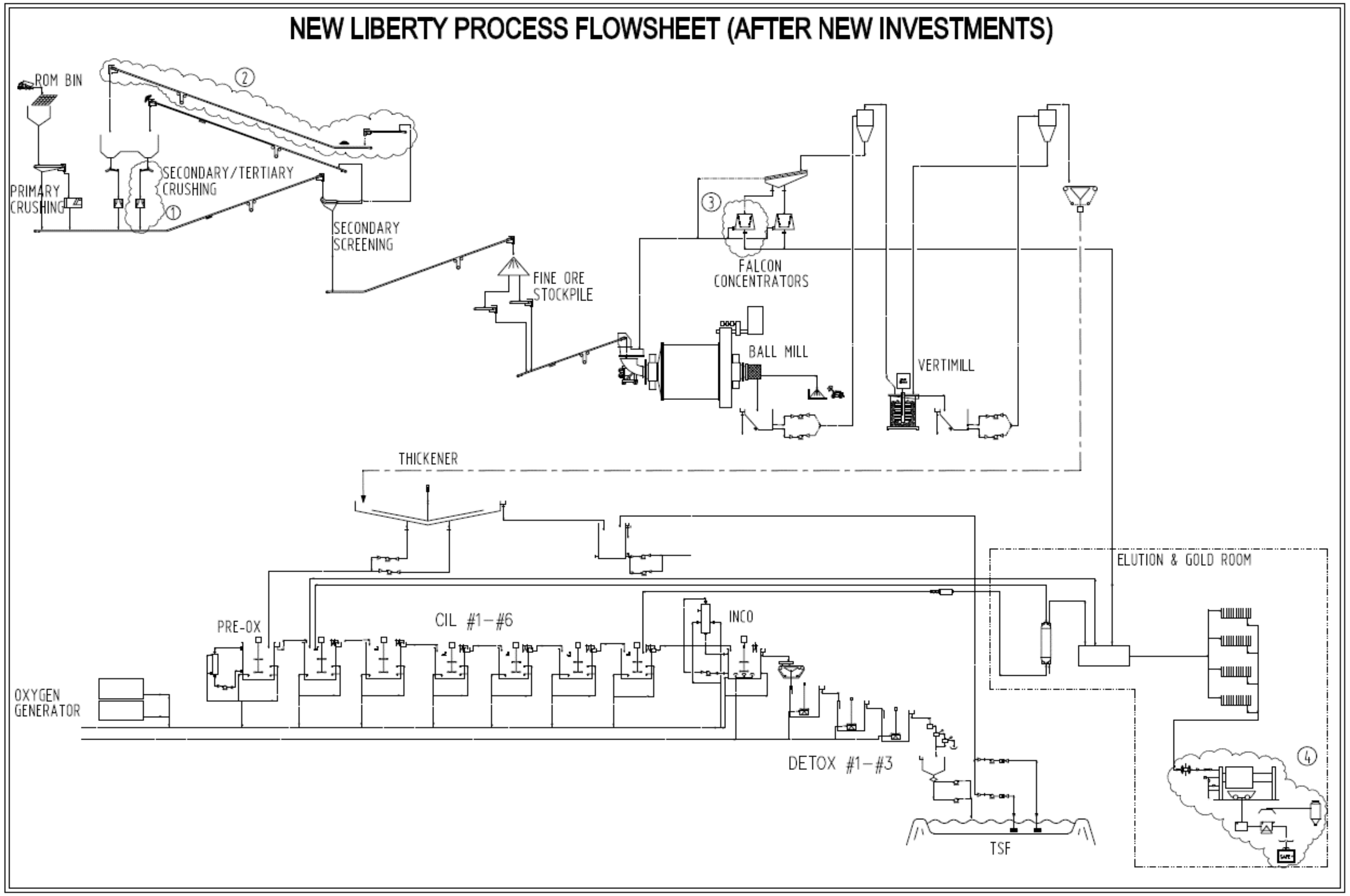
17.1.1 Introduction

Metallurgical testwork results and industry norms were used to define the process design criteria for the New Liberty Plant. The original process plant was designed, built and commissioned in July 2015 by DRA, an international engineering company, to treat 1.1 million tonnes per annum of ore, corresponding to a mill feed of 146 t/hour.

Since start-up there have been a number of operational issues that have affected plant performance. In late 2016 following the change of project ownership, the operational issues were evaluated and measures identified to improve performance. The planned modifications to the plant, some already implemented and some planned for late 2017, have been designed to increase the throughput in stages up to 200 t/hour, while achieving the original design gold recovery of 91 to 93%.

The process flowsheet is an industry-standard arrangement consisting of crushing, ore stockpiling, grinding and classification, gravity gold recovery, thickening and gold extraction by cyanidation in a Carbon-in Leach (CIL) circuit. Gold is recovered from the activated carbon by acid washing, elution and electowinning, followed by smelting to produce gold doré. The CIL tailings undergo cyanide detoxification followed by arsenic precipitation from tailings solution prior to disposal in the tailings dam.

The flowsheet is shown below in Figure 17-1 while an aerial photo of the plant layout is show in Figure 17-2.



Source: BMMC, 2017

Figure 17-1: Process Flow Diagram

1. Additional cone crusher to increase size reduction. Planned implementation by end 2017

2. Conveyor for the transfer of finer fraction off the second deck (oversize) of the screen of the new cone crusher. Planned implementation by end 2017.

3. Additional Falcon concentrator to increase proportion of gravity recovery. Implementation Q4 2017.

4. New induction furnace to replace existing diesel fired furnace, increasing load from 25kg to 250kg. Planned implementation by end 2017.

Source: BMMC, 2017

Figure 17-2: Plant Layout aerial photograph

The main operating criteria for the plant at different feed rates are given in Table 17-1.

Table 17-1: Process Plant Criteria

Parameter	Unit	Mill feed rate		
		146 tph (original)	175 tph	200 tph
Annual throughput	dmt/year	1,100,000	1,320,000	1,510,000
Crushing stages	#	2	3	3
Crushing circuit product size	80%-mm	15	18	8
Grinding circuit product size	80% -µm	45	approx. 75	75
Grinding/CIL operating days	#	350	350	350
Grinding/CIL operating hours	h/day	24	24	24
Grinding/CIL % operating time	%	90	93	93
Gravity circuit concentrators	#	1	1	2
Gravity circuit gold recovery	% of feed	~ 30% (actual)	~30%	up to 50%
CIL feed flowrate (45% w/w)	m^3/hour	229	277	316
CIL residence time	hours	26.2	21.7	19.0
CIL tailings assay	g/t	0.245 to 0.28	0.245 to 0.28	0.245 to 0.28
Gold recovery -2.0 g/t feed	%	-	89.0	89.0
Gold recovery -2.5 g/t feed	%	-	91.0	91.0
Gold recovery -3.0 g/t feed	%	-	92.5	92.5
Gold recovery -4.0 g/t feed	%	91 to 93	93.0	93.0
Elution circuit batch size	tC/batch	5	4	4

17.2 Ore Characteristics

Gold mineralisation occurs in zones of variable thickness and is nearly continuous along 1.8 km of strike length. The ore is free milling and contains a significant proportion of gravity recoverable gold.

The gold grade of the plant feed varies according to the mine plan, which over the LoM is between 2.3 and 4.5 g/t but which is typically between 2.5 and 3.0g/t up to the end of 2019 and increasing to typically between 3.0 and 3.5 from 2020.

The grinding circuit Bond Ball mill Work Index (BWi) varies between 14.0 to 22.1 kWh/t.

The amount of oxide ore is variable and will be limited by ROM blending to around 10% of the overall plant feed to avoid detrimental effects on cyanidation. Furthermore, this blending of oxide ore has only been assumed in the mine plan towards the end of the mine life during 2021 and 2022.

17.3 Operating Schedule

The crushing circuit operates 7 days per week, and up to 18 hours per day.

The grinding and cyanidation plant is designed to operate 350 days per year, 24 hours/day.

The elution circuit can process up to 7 batches of 4.5t of carbon per week under normal operating circumstances with the gravity circuit operating processes 7 batches per week (one batch per day).

17.4 Process Plant Design and Modifications

17.4.1 Ore Receipt and Crushing

The original crushing circuit was designed as a two-stage crushing plant producing a minus 18 mm feed for the ball mill. The plant will be modified in late 2017 to incorporate a third stage of crushing to reduce the product top size to nominally 80% minus 8 mm. This sized feed will be more suited to direct ball mill feed and will allow smaller (70mm) balls to be used in the mill resulting in more efficient grinding producing a finer mill discharge, less oversize stone discharging from the ball mill trommel, and a higher mill throughput.

ROM ore is delivered by truck on to the ROM stockpiles adjacent to the primary crusher or directly to the primary crusher. The ROM piles allow some degree of blending in terms of grade and ore type, to maintain a consistent feed to the circuit. Oxide ore containing clays is problematical in terms of materials handling in the crushing circuit and in the thickening and leach circuit.

ROM ore is treated in a primary crushing circuit comprising of a ROM bin fitted with a 500 mm static grizzly, a variable speed apron feeder and a primary jaw crusher operating in open circuit. Oversize ore is removed from the grizzly by an excavator. The original design incorporated a 700 mm static grizzly which resulted in blockages in the feed to the crusher. A dust suppression system is installed.

The primary crusher product and apron feeder dribblings gravitate onto the jaw crusher product conveyor.

Secondary crusher product is combined with primary crusher product on the jaw crusher product conveyor which feeds the crushing circuit sizing screen. The primary crusher product conveyor is fitted with a weightometer, positioned before the recycling point of the secondary crusher product.

Circuit screen oversize is weighed and conveyed to the secondary crushing circuit comprising of a bin, a vibrating feeder and a secondary cone crusher operating in closed circuit to produce a crushed product stream with a P_{100} of 22 mm. A dust suppression system is installed. Circuit screen undersize is conveyed to a 3,500 t mill feed stockpile via the stockpile feed conveyor. Crushed ore is sampled automatically.

The original design allowed for the installation of an additional secondary cone crusher. This has not been incorporated but the space will be used for the installation of a tertiary cone crusher as noted above. The sizing screen will be replaced with a multideck screen to feed both crushers. The revised circuit product will be nominally 100% minus 12 mm, 80% minus 8 mm.

The nominal capacity of the modified crushing circuit will be 280 tph operating 7 days per week and up to 18 hours per day.

17.4.2 Milling

The original grinding circuit, incorporating a ball mill together with a single Metso Vertimill® circuit operating in closed circuit with two stages of hydrocyclone classifiers, was designed to treat relatively coarse crushed ore at 100% minus 22 mm at a design feed rate of 146 t/h dry solids producing a final product with a nominal P_{80} of 45 µm and P_{60} of 25 µm. Initial operation of the circuit was problematical and the circuit did not consistently meet the design operating throughput and product specification and encountered high operating costs. A number of modifications and upgrades have been implemented to increase the availability of the circuit, improve the ball mill liner and grate life, reduce wear issues around the circuit, reduce steel ball consumption, increase the utilisation of the installed grinding power, reduce coarse stone discharge from the ball mill trommel and increase the throughput of the circuit while achieving the targeted grind size.

The modified crushing and grinding circuit is planned to operate at an increased throughput of 200 t/hour and a target grind size of 80% minus 75 µm, and minus 50 µm if needed.

The coarse top size of the crushed ore has been problematical and has resulted in excessive coarse stones exiting from the discharge end of the mill. The coarse feed solids are not ideal for a ball mill feed and necessitated the use of a relatively large, 90 mm diameter, steel ball. The quality of these balls was poor resulting in reduced grinding efficiency and resulted in a coarse mill discharge and high circuit circulating loads contributing to high wear. In addition, the use of large diameter poor quality balls probably also contributed to the damage of the mill liners and grate. The introduction of the tertiary crusher in to the crushing circuit will reduce the top size of the mill feed, reducing the amount of coarse solids exiting the mill and will allowing smaller 70 mm diameter balls to be used which in turn will reduce the issues with the mill liners and grate and should result in a finer mill discharge, reduced coarse stone discharge from the mill and much reduced wear in the grinding circuit as a whole.

Excessive breakage of the grinding balls has been an issue and an alternative, better quality, ball is now being used and ball consumption should also reduce from 240t per month to 70t per month (currently) with forged balls and further reduced to 30t per month with high chrome balls.

The ball mill is fed from the mill feed stockpile using variable speed belt feeders and into the milling circuit by a conveyor fitted with a weightometer.

The 17.5ft x 22ft EGL ball mill incorporates a 3,500 kW motor and operates in closed circuit with hydrocyclones. The mill discharge incorporates a grate and trommel screen to remove coarse stone and mill ball scats. 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 ±73-78% solids by mass, by the addition of process water to the mill inlet.

The mill product gravitates to the mill discharge sump where it is diluted to 50%-55% solids by mass before being pumped to the primary hydrocyclone classification cluster. The typical overflow product is 80% passing 75 µm at an estimated 38-40% solids by mass.

The cyclone underflow is routed to the gravity concentrator feed screen via a feed box, with any excess re-cycled directly to the mill feed.

The cyclone overflow is gravity fed to the secondary cyclone feed sump where it is joined by the product from the regrind mill before being diluted and pumped to the second stage hydrocyclone classifier cluster to produce an overflow product of 80% passing 47 µm.

The secondary hydrocyclone underflow is fed to the Vertimill® (VM 1500). This mill incorporates a 1,119 kW motor. The Vertimill® is required to achieve the relatively fine grind for leaching. The mill was bypassed during initial operation but has recently been reinstated as per the original design as part of the overall grinding circuit. The Vertimill® product will join the primary classifier overflow for combined secondary hydrocyclone classification.

The secondary hydrocyclone overflow is gravity fed to the 21m diameter pre-leach Hi-rate thickener via a linear trash screen and a primary cross cut and secondary vezin sampling system. Trash screen oversize, typically mica, is collected, drained and disposed of to waste. This material is detrimental to the CIL circuit performance causing blockages of the interstage screens.

Spillage in the grinding area is collected and pumped to the mill discharge sump.

17.4.3 Gravity Concentration

The ore contains significant gravity recoverable gold (GRG) and the original design indicated that a recovery of up to 60% gold was possible by gravity. Following start-up and during the initial operation, the amount of gold recovered by the gravity circuit was below design, around 30%. The gravity screen was replaced as it was too small. A second concentrator has been installed in the circuit to increase the amount of cyclone underflow treated and the expectation is that the extraction of GRG will increase significantly.

The gravity concentrator feed is pre-screened on a vibrating screen to remove oversize material not suited for the concentrator. The screen sprays are used to dilute the feed to 60%-65% solids by mass while also increasing screening efficiency. Screen oversize returns to the mill, with the underflow gravitating to the gravity concentrator. Concentrator tailings gravitate to the ball mill feed while the concentrate reports to a batch dissolution reactor. The entire unit is fenced for security reasons.

A Gemini table has been installed in the gold room as an alternative to the original intensive cyanidation circuit, which is no longer used. Gravity concentrate is collected and transferred to the gold room for further table concentration. The gold rich table concentrate is smelted separately. Table tailings are returned to the grinding circuit.

17.4.4 CIL Feed Thickening

Secondary hydrocyclone overflow gravitates to the 21m diameter Hi-rate pre-leach thickener 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. Any spillage in the thickener area will be pumped back to the thickener feed box. Flocculant is added to the thickener.

Thickener overflow is recirculated as process water.

17.4.5 Pre-Oxidation, Pre-Leach and CIL

The CIL circuit comprise eight tanks: a pre-oxidation tank, six Carbon-in-Leach (CIL) tanks and a cyanide detoxification (INCO) tank). All tanks are the same dimensions with a nominal volume of 1,000 m^3 each. Each tank is stepped by 600 mm differential height to allow gravity flow from tank to tank and each tank can be bypassed via a standard launder and knife-gate system.

Metallurgical testwork indicated that a pre-oxidation step was required to reduced cyanide consumption. 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 an external, pump fed, oxygen high shear reactor will increase the dissolved oxygen content of the slurry. The high shear reactor feed pump is fed from the pre-oxidation tank and the oxygenated slurry discharges in to the top of the tank.

Lead nitrate solution is added to the tank to aid the process together with milk of lime for pH control/adjustment.

The CIL circuit consists of 6 × 1000 m^3 tanks connected 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 to effect 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 tonnes per day. The CIL circuit will be operated to achieve a desired gold grade of less than 0.15 – 0.25 g/t in the solid tailings (dependant on mill feed grade) and a target solution gold tenor of 0.005 ppm.

The nominal cyanide addition to the CIL circuit is expected to be 0.65 to 0.8 kg/t feed.

The original design for the circuit was 145 t/hour at a solids concentration of 45% by mass equivalent to a pulp flow of 229 m3/hour. This equals to a leach residence time of 26.2 hours in the six CIL tanks. At 200 t/hour and 45% by mass solids the leach residence time will be 19.0 hours and leaching should be completed by CIL tank 4.

The slurry from the 6th CIL tank gravitates to the cyanide detoxification (INCO) tank.

Loaded carbon at 1,000 to 1,600 g/t (dependent on feed grade and GRG recovery) from the first or second CIL tank is pumped to the loaded carbon vibrating screen where it is washed with water. The screen underflow gravitates back to the CIL circuit. The screen oversize (washed loaded carbon) gravitates to the elution circuit acid-wash tank.

A spillage pump is installed in the CIL bund. CIL spillage is recycled to the pre-oxidation tank.

Hydrogen cyanide gas detection is installed in the area above the first two CIL tanks and the detox tank. Slurry samples are taken automatically from the CIL circuit and the tailings and analysed in automated cyanide analysers. Cyanide addition is controlled automatically via a flow meter and control valve. pH is controlled automatically to pH 10.5 by milk of lime addition via a control valve.

A maintenance bay with a screen frame and washing facilities is supplied for the cleaning of the inter-stage screens. A large tower crane (3.2 tonnes capacity), used during construction, is available for maintenance.

The pre-oxidation tank, the CIL tanks and the detox tank are installed in a concrete bunded area sized to hold up to 110% of a CIL tank volume.

A safety shower is installed in the CIL bund and on top of the CIL tanks.

17.4.6 Acid Wash and Elution

The elution circuit processes one ±4.5 tonne batch per day of gold-loaded 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. Under normal circumstances eluted carbon will contain 100 to 150 g/t gold.

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. The IBCs are stored in the chemical storage shed. An IBC container is moved to the elution area as required and hydrochloric acid at 33% w/w is pumped using a peristaltic pump to the dilute acid make-up tank 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 located directly above the elution column. Dilute hydrochloric acid is pumped 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. 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 3 bar 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 3 bar 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 3 bar 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. The water from this step is the solution for the next soak and elution cycle.

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. This water is also used in the next soak and elution cycle.

Carbon Regeneration

On completion of the cold rinse cycle, the carbon within the column is transported to the carbon regeneration kiln feed sieve-bend by pressurising the column with water and pressure-educted from the column to the carbon regeneration kiln dewatering feed sieve-bend via pipeline. Dewatered wet carbon from the sieve-bend gravitates to the kiln feed hopper which discharges into the kiln feeder. The water drained from the sieve bend and any additional drainage water from the kiln hopper or feeder reports to the carbon quench tank. The carbon is regenerated in the rotary kiln at 650 to 700C from where it discharges via a submerged pipe into the carbon quench tank. The regenerated carbon is pumped back to the final CIL tank via the carbon sizing screen as part of the daily carbon transfer sequence and screen oversize discharges in to CIL tank 6 or CIL tank 5. Fine carbon in the sizing screen underflow is piped to the fine carbon collection pond located close to the CIL plant bund.

Make-up carbon is delivered in bulk bags and is added via the sump pump to the carbon quench tank as required. Any carbon fines are removed on the carbon sizing screen above the CIL tanks.

Elution Area Spillage Handling and Services

The elution column is located in a discrete concrete bund and any spillage is collected in the spillage sump and pumped to the first CIL tank.

The acid wash column is located in a separate acid tank bund and any spillage is pumped to the spent acid tank.

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.

17.4.7 Electrowinning and Gold Room

The electrowinning circuits process pregnant solution from the CIL elution circuit to recover gold for downstream smelting.

Two pregnant solution tanks feed the electrowinning cells for the CIL elution circuit, together with the electrowinning barren solution tank, used to collect and recycle the barren electrowinning solutions back to the plant. These are situated in a high security, concrete-bunded area immediately adjacent to the gold room.

High-security gold room processing comprises equipment for tabling high grade concentrate if necessary, electrowinning of gold from the pregnant solutions, followed by filtration and 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 personnel movement in-out of the gold room area. Gold room ingress and egress are controlled and monitored via a proximity card and turnstile system.

Of the four identical electrowinning (EW) cells, two are dedicated to the CIL circuit, and one is dedicated to the gravity circuit. The fourth cell is operated as a common standby unit. 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.

The loaded cathodes are manually hoisted from the EW cells and taken to the cathode wash table where the gold sludge is removed from them by high pressure water blasting, with the sludge reporting to the cathode wash sludge settling tank which also receives loosened sludge from the EW cell drains. Dewatered sludge is dried in a drying oven, prior to direct-smelting with flux in a furnace 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. The original oil fired furnace has recently been replaced with an electric induction unit.

Gold room spillage accumulates in a dedicated sump within the area and is pumped via a gold trap to the cathode wash sludge collection tank.

A safety shower is located in the gold room area.

17.4.8 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 50ppm CN_{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 CN_{WAD} (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 and 6.0 as opposed to the conventional pH of 8.0 and 9.0.

The addition of SMBS in this process has a dual purpose, firstly taking part in the cyanide detoxification reaction, and secondly lowering the pH (due to the formation of sulphurous acid) to such an extent favouring the arsenic 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 air is injected at the bottom of the tank using spargers. The slurry from the detox tank flows by gravity, via a carbon safety linear screen to the arsenic precipitation circuit for further treatment before being pumped to the tailings dam.

Reagent addition problems to both the cyanide detoxification tank and the arsenic leaching and precipitation circuits resulted in inconsistent CN_{WAD} and soluble arsenic tailings discharges to the TSF. These reagent pumping and delivery systems have been upgraded and modified to ensure better reagent delivery and addition control to maintain the discharges within acceptable limits. It is noted that further optimisation work will be required in this area.

17.4.9 Arsenic Precipitation

Detoxified tailings and arsenic leaching as described above gravitate to the first of three agitated tanks which make up the Arsenic precipitation and conditioning circuit.

The arsenic precipitation circuit has 3 x 260m^3 tanks. The first two tanks allow enough residence time for the leached arsenic to precipitate, whilst the purpose of the third tank is to allow for any pH corrections to be made before being pumped to the tailings storage facility (TSF). 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 and 6. As noted above the pH of the slurry discharging from the detoxification tank 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 chloride, which is used as an oxidant, is also added to this tank as a solution.

The slurry from the second arsenic precipitation tank gravitates to the final conditioning tank where the final pH correction to pH 6.0 is made by the addition of milk of lime, as required by legislation, before it is pumped to the final TSF. This tank is also used for additional residence time for the precipitation of the As^{+5} ion.

The final slurry gravitates to the final tailings pump box via a primary cross-cut/secondary vezin sampler system.

Plant tailings are pumped to the tailings storage facility via a 280 mm diameter HDPE pipeline.

A spillage pump and safety shower are planned to be installed in the detox/tailings disposal area.

17.4.10 Reagents

Caustic Soda Make-up and Storage

Sodium hydroxide or caustic soda (NaOH) is delivered to the plant in 1,000 kg bulk bags in “pearl” form. The sodium hydroxide system includes a mixing and a holding tank. The reagent is mixed with fresh water to a 20% w/v solution strength and is pumped to the required points of use (cyanide make-up, intensive leach reactor, strip solution make-up tank and electrowinning) using fixed speed helical screw pumps (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.

Sodium Cyanide Make-up and Storage

Sodium cyanide (NaCN) is delivered to the plant in 1,000 kg bulk bags contained in wooden boxes. The bags are lifted by the dedicated 2 t cyanide bag hoist and delivered as required to the sealed and ventilated cyanide bag-splitter cabin located above the mechanically agitated sodium cyanide mixing tank. The pH of the water in the cyanide mixing tank is adjusted to pH 11 before cyanide mixing starts in order to prevent the formation of HCN gas. 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 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 as required.

The safety showers in this area are equipped with a high-flow switch which will alarm when the shower is in use and alert the control room operator to investigate the cause of activation.

A hydrogen cyanide gas monitor and alarm is installed. The cyanide store and mixing/storage tanks are located in a secure area.

Copper Sulphate Make-up and Storage

Copper sulphate ($CuSO_4$) is delivered to the plant in 1,000 kg bulk bags. The copper sulphate system includes a mixing and a holding tank. The reagent is mixed with fresh water to a 20% w/v solution strength and is pumped to the cyanide detoxification circuit by variable speed dosing pumps.

Copper Sulphate make-up spillage is pumped by the common reagent spillage pump to the detoxification circuit.

SMBS Make-up and Storage

Sodium meta bi-sulphite (SMBS) is delivered to the plant in 1,250 kg bulk bags. The SMBS system includes a mixing and a holding tank. The reagent is mixed with fresh water to a 20% w/v solution strength and is pumped to the cyanide detoxification circuit and the arsenic precipitation tanks by variable speed dosing pumps.

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 to the detoxification circuit.

Lead Nitrate Make-up and Storage

Lead Nitrate ($PbNO_3$) is delivered to the plant in 1,000 kg bulk bags. The lead nitrate system includes a mixing and a holding tank. The reagent is mixed with fresh water to a 20% w/v solution strength and is pumped to the CIL circuit via a variable speed dosing pump.

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 to the detoxification circuit.

Ferric Chloride (40% purity) Make-up and Storage

Ferric chloride ($FeCl_3$) is delivered to the plant in 1,400 kg IBC barrels. The ferric chlroide system includes a mixing and a holding tank. The reagent is mixed to solution and is pumped to the detoxification and arsenic precipitation circuits via variable speed dosing pumps.

A safety shower supplied with potable water is strategically located within the area.

Ferric chloride make-up spillage is pumped by the common reagent spillage pump to the detoxification circuit.

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 and is mixed automatically with fresh water in an automated mixing package to a 0.25% solution strength. Stock flocculant solution is aged in a mechanically agitated flocculant transfer tank from where it is pumped to the pre-leach thickener using a variable speed helical screw pumps.

Flocculant area make-up area spillage is pumped by the spillage pump to the detoxification circuit feed.

Hydrated Lime Make-up and Distribution

Hydrated lime ($Ca(OH)_2$) is delivered in 1,000 kg bulk bags which are transported to the plant by road in containers. The bulk bags are lifted by the reagent area overhead crane and the contents are discharged into the feed hopper located in a sealed cabin. The feed hopper is equipped with a vibrating system and variable speed rotary feeder.

The rotary feeder meters the hydrated lime into the agitated milk of lime mixing tank to a 20% w/v slurry. The milk of lime slurry is pumped to the mill feed and the cyanide detoxification circuit via a ring-main. Additions can be made to the pre-leach thickener and CIL tank No. 1.

Lime make-up area spillage is pumped by the common reagent spillage pump to the detoxification circuit. A safety shower provided with potable water is strategically located within the lime mixing area.

Table 17-2: Reagent consumption

Reagent	Unit	Value
Hydrated lime consumption as 100% $Ca(OH)_2$ equivalent CIL including detox consumption	kg/t	1.5
Sodium cyanide consumption - CIL	kg/t	0.65
Sodium cyanide consumption - elution	kg/Batch	100
Sodium hydroxide consumption - elution	kg/Batch	150
HCl consumption at 33% strength (nominal)	m^3/batch	0.829
SMBS consumption (Nominal)	kg/t	1.75
$CuSO_4$ consumption (Nominal)	g/t	625
Lead nitrate consumption - CIL	g/t	25
Ferric chloride consumption	m^3/t	0.64
Total flocculant consumption	g/t	30 – 40
Activated carbon consumption	g/t	25

17.4.11 Water

Process Water

Return water from the tailings storage facility return water pond is pumped to the plant process water tank where it is joined by the overflow from the pre-leach thickener. Process water is supplied to the plant by two dedicated process water pumps (one operating and one 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, which receives make-up water from the river water pumping system. The clean water tank provides for plant clean water and gland service water requirements with two dedicated clean water pumps (one operating and one 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. Potable water is supplied to the plant via a dedicated potable water supply pump.

Fire Water

Fire water is supplied from the bottom section of the clean water tank to the plant via a dedicated vendor package fire water pumping system incorporating a jockey pump, an electric supply pump and an emergency diesel pump in case of power failure.

17.4.12 Plant Services

High Pressure Air Services

Compressed air at nominally 750 kPa is supplied by two (operating and standby) compressors and delivered to the plant via a ring main system. The plant air receiver is designed to hold 8 m^3 of air at 750 kPa.

Filtered and dried instrument air is taken from the compressed air system. The compressed air purification system consists of high efficiency filters and a dryer installed between the filters. The filters remove contaminants such as water, oil and solid particles from the compressed air stream. The instrument air dryer removes moisture prior to the instrument air receivers. The instrument air receiver is designed to hold 5 m^3 of air at 750 kPa

Oxygen Plant

Oxygen gas is supplied to the Pre-Oxidation and detoxification high shear reactors and the Pre-Oxidation and CIL tank spargers. The oxygen purity is 90% v/v. Two package Pressure Swing Adsorption (PSA) oxygen plants were supplied as part of the original project but proved unreliable having a detrimental effect on leaching and detoxification. Two additional packaged oxygen plants have been installed to operate in parallel with the existing units to improve the reliability of oxygen supply to ensure leaching and detox performance.

The total installed oxygen supply capacity is 6,500m^3 per day at 650 kPa.

Low Pressure Air Services

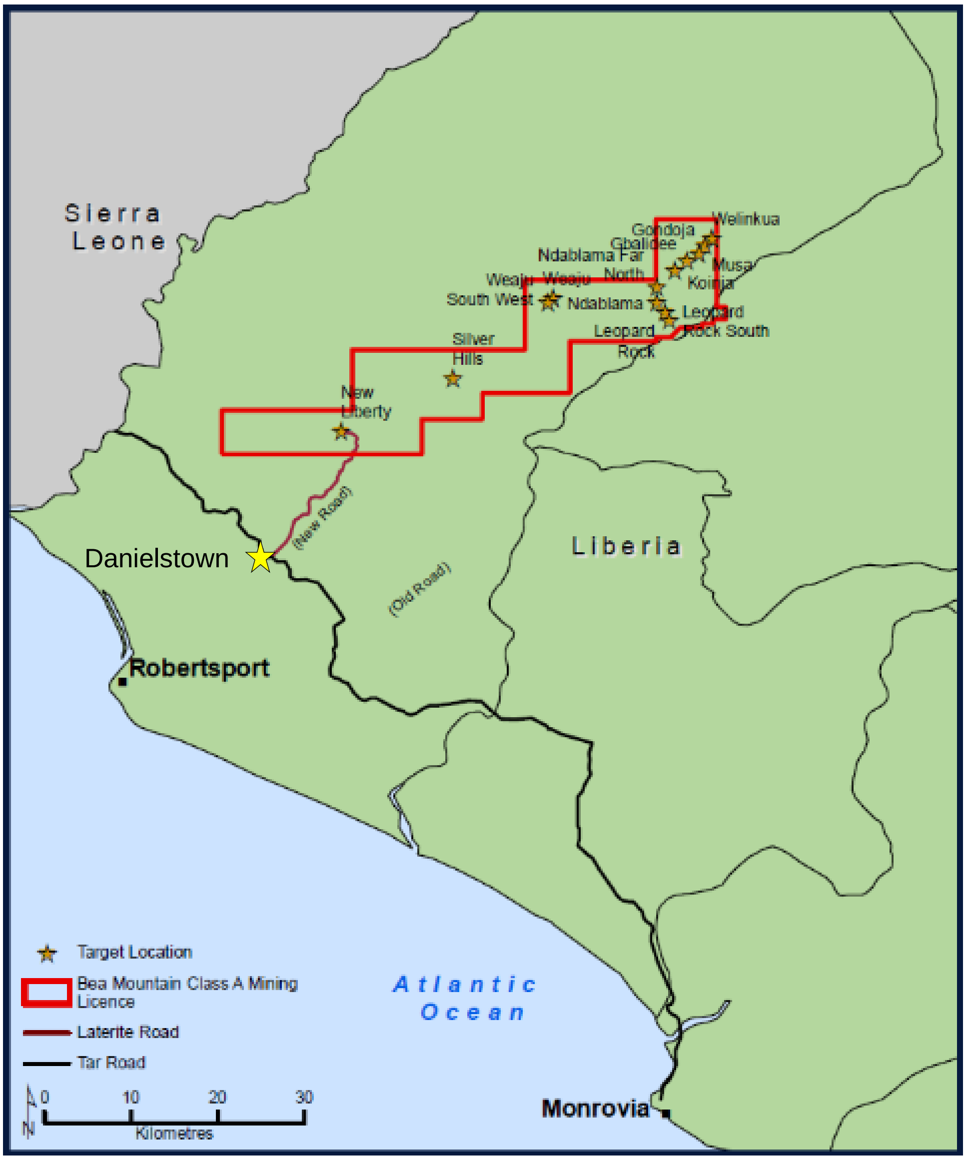
A low pressure air circuit is comprised of two low pressure air blowers which supply low pressure air to the cyanide detoxification and arsenic precipitation circuits at a minimum pressure of 171 kPa.

18 PROJECT INFRASTRUCTURE

18.1 Introduction and Access

18.1.1 Access Roads

The Project is located approximately 100km north–north-west of the Liberian capital, Monrovia and is accessed by an existing 80km long bituminous road between Monrovia and Danielstown and a 20km long laterite road to the project site. The current bituminous and laterite roads to the Project site allows for easy access for larger cargo and was used successfully during the construction of the Project. This is shown in Figure 18-1.



Source: *BMMC, 2017*

Figure 18-1: Project Location and Site Access Roads

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.

18.1.2 Road Upgrades

As the primary access route, BMMC has widened and re-graded the laterite road between Danielstown and New Liberty, and has made improvements to road drainage and upgraded and installed concrete culvert type bridges. Secondary roads on the Bea-MDA licence area, built by BMMC, provide access across the property. Due to the laterite nature of the roads, access is possible all year round, including during the height of the rainy season.

18.1.3 Danielstown Diversion

BMMC has constructed a road diversion around the Danielstown village, which was completed in early 2017.

18.1.4 Air-Strip

An air strip has recently been constructed at the site. Liberian air regulations were followed in the design of the airstrip and the Liberian Civil Aviation Authority visited the site to inspect this. The air strip is suitable for light aircraft such as a “Cessna Caravan”. The Security Manager controls the incoming and outgoing flight arrangements both from the security office and the airfield. Security personnel are responsible for area inspections prior to flight arrivals and control access to the plane while it is on the ground. The on-site ambulance and other security vehicles attend the scheduled arrivals and departures.

18.2 Site Infrastructure - Current Status

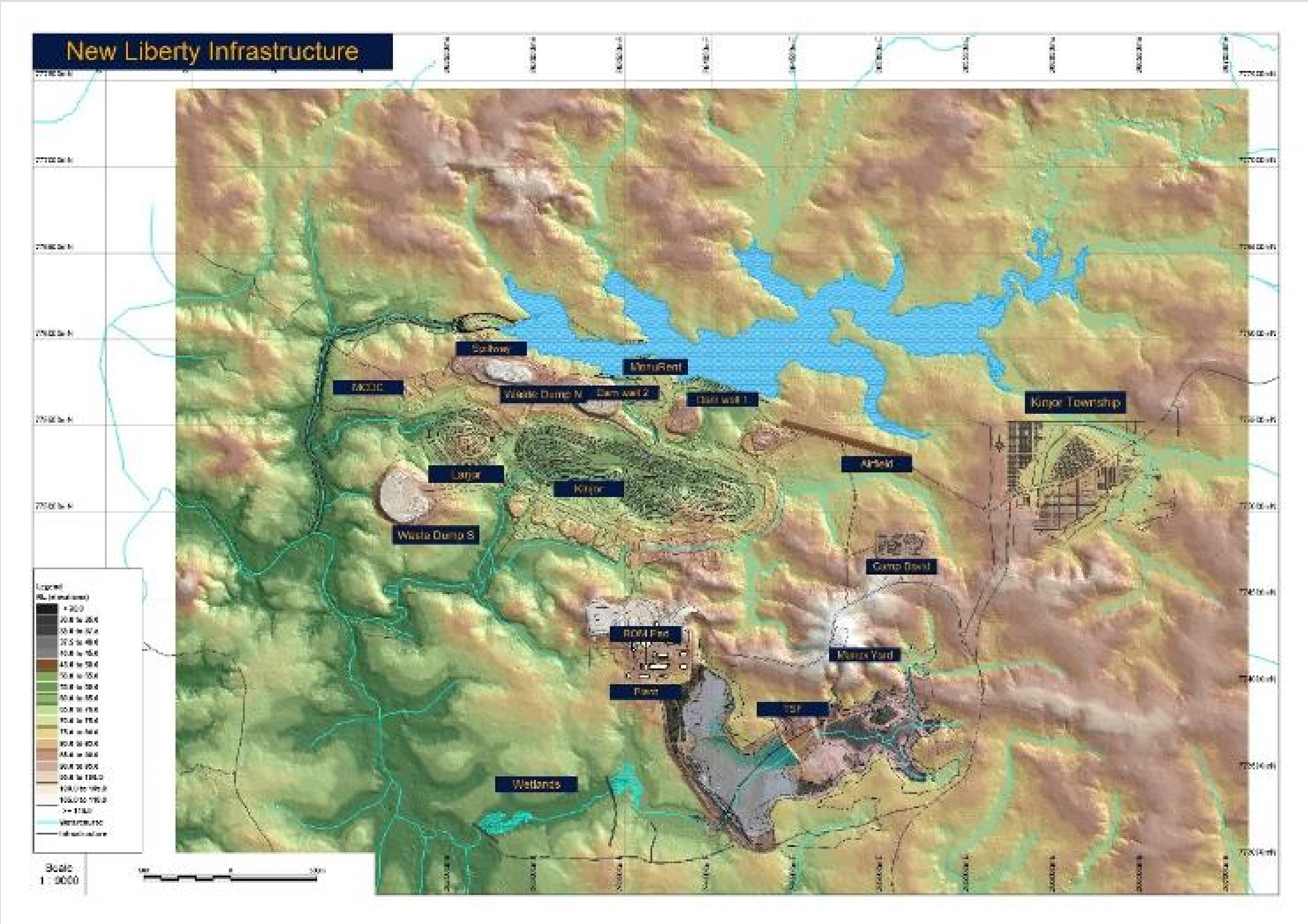
The infrastructure currently in place at NLGM to support the mining, plant and TSF operations can be summarised as follows:

- Mining Support Infrastructure and Services;
- Process Plant Support Infrastructure;
- Laboratory;
- Accommodation Facilities; and
- Power supply and distribution.

Prior to the construction of the Project and commencement of operations, the original infrastructure consisted of an exploration camp comprising offices, staff accommodation, messing facilities and core storage facilities. A temporary equipment workshop was added to these facilities on commencement of mining. These facilities are no longer in use as they were within the 500m blast radius of the open pit operations. Equipment workshop activities are now carried out in the new facility (see below) and while some equipment may still be left in this area, this will be gradually moved and no work is being undertaken there.

18.3 Project Layout

A project layout, supplied by BMMC, is presented in Figure 18-2. This shows the position of the open pit, process plant, TSF, MCDC and water storage dams and other general site infrastructure relative to each other and the surrounding topography.



Source: ***BMMC, 2017***

Figure 18-2: General Infrastructure Layout

18.4 Support Infrastructure

18.4.1 Introduction

Support infrastructure is in place and comprises the following:

- Security Services;
- On-Site Roads/bulk earthworks;
- Mining Office;
- Mining Equipment Workshop;
- Fuel Storage and Dispensing;
- Explosives Storage;
- Communications;
- Laboratory Services;
- Medical Services; and
- Processing Support Buildings.

18.4.2 Security

The plant site is enclosed within a security fence. Access to the plant area is via gates located on access roads to the site. Additional fencing is provided for further safety and security within process plant areas, such as the power plant, fuel storage, gold room area, transformers and substations. CCTV cameras are installed at strategic locations in the plant for surveillance purposes. The cameras are integrated with the plant's overall network, which are the responsibility of the security manager.

Security operations are managed by a primary Security Contractor ("Blackpool Security") who oversee two Liberian security contractors (186 staff in total) and a Liberian Government Emergency Response Unit (15 staff) as well as its own staff.

A new contract with Black Pool Risk Management for provision of security services was signed on 1 September 2017 and covers a two-year period.

18.4.3 On-Site Roads/Bulk Earthworks

Site access roads are in place with a 40 km/h speed limit. This limit is enforced by the site security team. Access roads comprise unbound pavement construction with a surface wearing course of crushed waste rock. All bulk earthworks for road and platforms for buildings are complete.

18.4.4 Mining Offices and Canteen

The mining offices are located close to the RoM pad and processing plant. The offices consist of a primary office building of prefabricated construction and four converted container offices. This gives a total of 27 desks distributed across 11 office spaces. The following departments are based in the offices: Mining, Mine Planning, Drill & Blast, Mine Management, Survey, Geology, Grade Control, Mine Supervision, Training, Health & Safety. There is a mess building for the mining staff comprising a cement block construction with dimension of 11 x 22m.

18.4.5 Mining Equipment Workshop

The permanent Heavy Mining Equipment workshop (referred to as the “HME Workshop”) is nearing completion. Prior to this, a temporary workshop was utilised, which was located adjacent to the old exploration camp (between MCDC Dam Walls 1 and 2) as discussed above.

The HME workshop is located adjacent to the RoM pad. The HME workshop compound houses two workshop buildings for mining equipment (Figure 18-3), washing pad, tyre change area, tyre storage and repair sheds, welding bay and stress sheds for machining, compressed air etc., as well as a parking areas and office/ablutions. The two workshop buildings are equipped with portal cranes. The HME office is of prefabricated construction and the portal frame workshops of pre-engineered steel construction.

Source BMMC, March 2017

Figure 18-3: HME workshop under construction (March 2017)

18.4.6 Fuel Storage Area

Diesel is required to operate the power generators which provide the power to the processing plant and infrastructure as well as for the mining operations. The average monthly consumption of diesel for the Project as a whole is currently approximately 1.9 million litres.

The fuel storage and the dispensing facility for both diesels and lubricants supplies all Project operations. The facility has a fire suppressant system on the Modular Pump House and 100kg dry and foam chemical extinguishers for all other areas. The facility was constructed by an external fuel supply contractor (“Aminata”) and contains the following key infrastructure:

- 12 x P75 double skinned tanks (Main Fuel Farm - 869,400lts), 2 x P69 tanks (Workshop Fuel Farm - 134,240lts) – Grand Total = 1,000,364 lts / 1,000.4m^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;
- Firefighting equipment;
- Connection to the day tank at the power plant;
- Contractor's offices; and
- Security fencing and entrance check point.

The fuel farm and lubricant storage area was previously managed by Aminata (the fuel supplier) and this is now managed by BMMC staff who are responsible for the following:

- Operations:
  - On-site offloading of fuels and lubricants
  - Handling of used diesel and oil and general management of hydrocarbons
- Filtration:
  - Filtration solution implementation
  - Filters and filtration equipment maintenance
- Technical support
- Site and risk management and daily control of usage

Supply of fuel to the site remains the responsibility of Aminata in line with the agreement signed between BMMC and Aminata which commenced on 1 January 2015 and covers a period of 8 years.

The fuel in the storage facility will be able to run the processing plant and camp for approximately 16 days before refilling is needed. A fleet of trucks is used by the external contractor to deliver product from the port at Monrovia to the Project site.

Figure 18-4: View of the completed fuel farm and Power Plant (April 2016)

18.4.7 Explosives Storage

Bulk explosives are supplied and stored by external contractors Manex and CGGC. CGGC is an in-country Chinese supplier, and BMMC indicates it is moving towards CGGC taking over total explosive supply responsibility for the Project. CGGC has a production facility on site and has supplied explosives to the project over the last 12 months.

A contract with Manex contract was signed on 25 February 2015 and has a duration of five years. It covers the supply of explosives and the charging of the holes. Manex has teamed up with Maxam in Ghana to supply emulsion and accessories to site and these are only used when the stock is available. The construction of the Manex emulsion facility is not complete and no further construction work will be undertaken until a number of outstanding contractual matters have been resolved.

A contract with CGGC for the supply of bulk emulsion and accessories was signed on 28 September 2016 and this is valid for a period of 3 years.

The Explosives Magazine is 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 500m radius of the explosives storage magazine. The Explosives Magazine is used by the explosives contractors Manex and CGGC to store detonators and boosters. The area is securely fenced and guarded and provision has been made for adequate lighting at night.

18.4.8 Communications

The project has well developed communications systems. The property is covered by two mobile phone providers, Lonestar and Orange (previously Cellcom), who have mobile communications masts located on the property. The Orange mast is 25m high and located next to the Mining Office. The Lonestar mast is near to the mine entrance. The Company also engages a two-way radio system for pit and plant operatives.

A satellite link is in place for all internet access, comprising a 25W booster and Marlink internet service provider with a 10Mbps download, 2Mbps upload.

18.4.9 Assay Laboratory

The assay laboratory is in the form of containerised units supplied to the Project and managed by an independent third-party laboratory service provider ("ALS Global"). The fully furnished and fitted containers are under a large shed structure and supplied with full services. A contract with ALS was signed in May 2015 for a 5-year term.

This laboratory conducts all of the onsite test work for samples from the processing plant, grade control and environmental teams and is located adjacent to but outside from the entrance of Camp David.

The laboratory offers analytical services for solid and liquid samples from the mine and plant and environmental samples. The laboratory can process 6,000 solid samples per month and is currently operating around 60% capacity. The shift samples from the mill take priority and are targeted for a 12-hour turnaround. Other samples are on a 24-hour turnaround.

The laboratory has environmental and limited metallurgical testwork capability with a wet chemistry section for water sample analyses and bottle roll testing equipment and sieving equipment is available for metallurgical testwork.

There is also a small containerised metallurgical test laboratory in the plant which is used for leach tests, cyanide tests and other plant related testwork. This is used by the plant personnel for real-time tests.

18.4.10 Medical Facilities

An equipped medical facility is provided within the footprint of the accommodation camp (known as "Camp David"), which allows for the treatment of any injuries as well as treatment of sick personnel. It is understood the medical team consists of an ex-pat doctor, three ex-pat paramedics and two local nurses. There is a fully equipped ambulance stationed at the clinic.

18.4.11 Processing Plant Support Buildings

The following plant buildings have been constructed and are in use:

- Security Office;
- Plant Change Houses – male and female;
- Plant Control Room;
- Process Plant Equipment Workshop and Offices;
- Plant administration building;

- Plant Store; and
- Reagents Building

The plant equipment and building itself are discussed in the processing section (Section 17).

Change Houses

A change house and ablutions are provided for 50 males and 10 females.

Plant Administration Buildings and Gatehouse

The processing plant area has its own fence-line and entrance gate with a security guard building. The Administration building is located at the entrance gate to the plant area and has offices and meeting rooms.

Plant Control Room

A dedicated plant control room is located in a double container arrangement. The top container houses the control room and the bottom container houses one of the MCC units. The control room houses the SCADA system and provides operators with an elevated view of the entire plant.

Process Plant Equipment Workshop

A workshop with an area of 480m^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 has bays for servicing light vehicles. The workshop has 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 are provided in the form of a modular building situated next to the workshop.

Plant Store Building

A store with an area of 480m^2 has been constructed adjacent to the process plant. The store consists of a steel framed building.

18.5 Site Services

Sewage Treatment

Sewage from Camp David is processed in a treatment plant at the camp where wastes are treated and discharged. Sewage from ablutions around the site are treated in individual sceptic tanks (e.g. plant change-house).

Waste Management

A waste management facility (landfill) is located within the TSF area and receives municipal and degradable wastes from the project. Scrap metal is stockpiled near the plant site and a contractor has been identified to remove the scrap iron. All waste oils and greases, filters and oily waste & rags are sent to Edgail Services (an EPA registered oil recycler) in Monrovia. Worn tyres are currently stored on site.

Water Services

The Project is located in a net water surplus climate. To minimise the volume of non-contact surface rainfall run-off reporting to the TSF or the open pits, water diversion channels and ditches have been constructed.

Raw Water Supply Dam

The design of this facility (i.e. the water stored upstream of MCDC diversion dam structures 1 and 2 as discussed below) is based on meeting or exceeding agreed design criteria which comply with World Bank and other international standards.

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 is 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 TSF is insufficient to meet the requirements of the plant.

Potable Water

A water treatment plant is installed at Camp David to ensure potable water is available in areas such as the change houses and plant administration building. Raw water is 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. In respect of potable water for human consumption, bottled water is provided and dedicated potable water supply boreholes have also been drilled. The water from the water plant has been tested and is free of coliforms and fit for 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 fire water system consists of a fire water distribution system with hydrants strategically positioned around the process plant.

18.6 Accommodation Facilities

The permanent accommodation camp (“Camp David”) was initially constructed to house up to approximately 500 individuals during the construction phase and was scaled back for the production phase. The camp includes the following infrastructure:

- Kitchen constructed using modular construction materials
- Camp dining room
- Laundry
- Potable water plant
- Sewage disposal plant;
- Guest accommodation (10 rooms) with a large meeting room;
- Communal TV room;
- Recreation area and gym;
- Administration offices for the Catering Contractor;

- Gate House with search rooms and toilet; and
- Medical facility.

Until recently the total camp capacity was 250 persons, however, BMMC has recently upgraded the camp to accommodate 320 persons with the addition of the following:

- Two new dormitories of 12 rooms each.
- 8 rooms for senior managers.
- One 19 room building for foremen and mid-level supervision (accommodating 38 people)
- Two VIP rooms (under construction)
- A new kitchen has been commissioned to serve meals to 250 people (the original kitchen still serves meals to the junior local staff).
- A new water treatment plant has been installed and commissioned and is in use.
- The sewage plant treats the water in accordance with South African DWEA General Limits for the release of treated water into the environment.

18.7 Power Supply and Distribution

18.7.1 Power Supply

An external contractor currently provides the power generating capability at the Project site which is 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, BMMC is responsible for generating its own power.

An external contractor (Jozi Power) provides an 11 kV, 10.8 MW, diesel driven, build, own, operate and transfer (BOOT) power station at the Project. The generators (6 operating and 2 standby) are housed in 12 m shipping containers.

The power plant has been designed to be self-sufficient and has 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 is run via cables to the plant main 11kV substation. Synchronisation is 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 is via the bulk diesel storage facility located adjacent to the power plant fence line. The diesel is free-issued to the power plant contractor. A diesel day tank is included 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

Key Info	
Prime power output per genset	1,965 kW (m)
Installed units	6 No.
Installed capacity	10,800 kW (e) at 11kV busbar
Average demand	7,505 kW (e)
Guaranteed power MD	7,200 kW (e) (99.5% availability)
Fuel consumption	204.2 g/kWh (at 11kV)
Fuel consumption	0.234 L/kWh
Maximum continuous running capacity of 4 gensets (normal operation)	7,200 kW
Continuous average load capability of 5 gensets (75% of 9,000 kW)	8,100 kW
Capacity of 3 remaining gensets if 1 genset trips during normal genset operations (3 x 110%)	5,940 kW
Guaranteed power output for 99.5% of the time	7,200 kW

18.7.2 Power Cost

At the recently agreed fuel supply cost of USD0.75/litre, the current fixed fees and fuel consumption for power generation (average of 921,000 litres per month), the cost of power to the operation is USD0.255/kWh. Future changes in fuel price or consumption will affect this power cost; an isolated 25% increase in fuel price equates to a broadly corresponding increase in power cost.

18.7.3 Power Distribution

Power from the power plant is transferred at 11,000V, 50Hz via individual feeders from each of the generators to the plant main 11kV substation.

Medium voltage electrical power is distributed throughout the main processing facility via 11,000V XLPE cable. Low voltage electrical power distribution is distributed to loads (motors, distribution panels, light fittings etc.) via 1000/600V PVC cable, which generally run above ground on cable ladder, or buried where the use of cable ladder is not appropriate.

The main electrical power consumer is the 11,000V 3.5MW ball mill and 1.1MW Vertimill motors, which is 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 are of the wound rotor type and utilise 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 are rated for 380V (3 phase) and 220V (single phase), 50Hz.

Power to the plant infrastructure and the New Liberty accommodation camp is supplied from the plant 11,000V substation via overhead line.

Pit dewatering and raw water intake pumps are 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 has been retained and acts as the emergency backup generator.

The specification and selection of electrical equipment has been in accordance with South African Standards (SANS Standards).

18.7.4 Future Changes to Power Supply

There are currently two contracts with Jozi Power: one for the supply of power and one for the deferred purchase of the equipment. Both contracts were signed on 06 February 2014 and are valid for a period of 6 years. BMMC, however, aims to bring the power supply service in-house in order to reduce operating costs and as such Jozi Power has recently been given notice of termination to take effect on 31 December 2017. A termination payment is payable for early termination of the contract.

BMMC is in the process of making plans for bringing the power supply in-house and this includes:

- Purchasing new gensets (identical to the existing Jozi Power units) which will need transporting to site and connecting to the site system following demobilisation of the existing gensets; and
- Recruiting an in-house maintenance team consisting of 2 supervisors, 3 mechanics, 3 electricians and 10 local mechanics.

Furthermore, BMMC has estimated that an additional 600kW of load is required for the increased plant throughput that is planned and this is within capacity of the existing set up.

SRK notes that the power supply to the Project via Jozi Power has operated well to date and the key driver by BMMC for the change is to reduce operating costs. SRK considers that changing the power supply set up will bring a slightly elevated risk to the Project for a period of time, particularly as new gensets will be acquired given the need to demobilise the existing gensets and install and commission new ones. However, by utilising spare gensets and changing them out one by one, this should minimise downtime during this changeover and other critical electrical work can be able to be accommodated around planned maintenance shuts. Further, BMMC will need to ensure that appropriately experienced and qualified personnel are recruited to ensure there is a continued smooth operation of power supply to the Project be it with the existing gensets or newly acquired ones. Finally, the additional load for the increased plant throughput will need to be confirmed and monitored during operations and BMMC will need to evaluate the performance of the power plant for this increase in load and take any corrective actions as necessary.

18.8 Summary of Planned / On-Going Capital Works

The following works are still on-going on site and BMMC has made for this as part of its sustaining capital budget:

- Improvements at the Accommodation Camp;
- HME workshop construction.

18.9 Tailings Storage Facility

18.9.1 Design Overview

The current Tailings Storage Facility (TSF) arrangement has been in operation since July 2015. The TSF consists of the following key components:

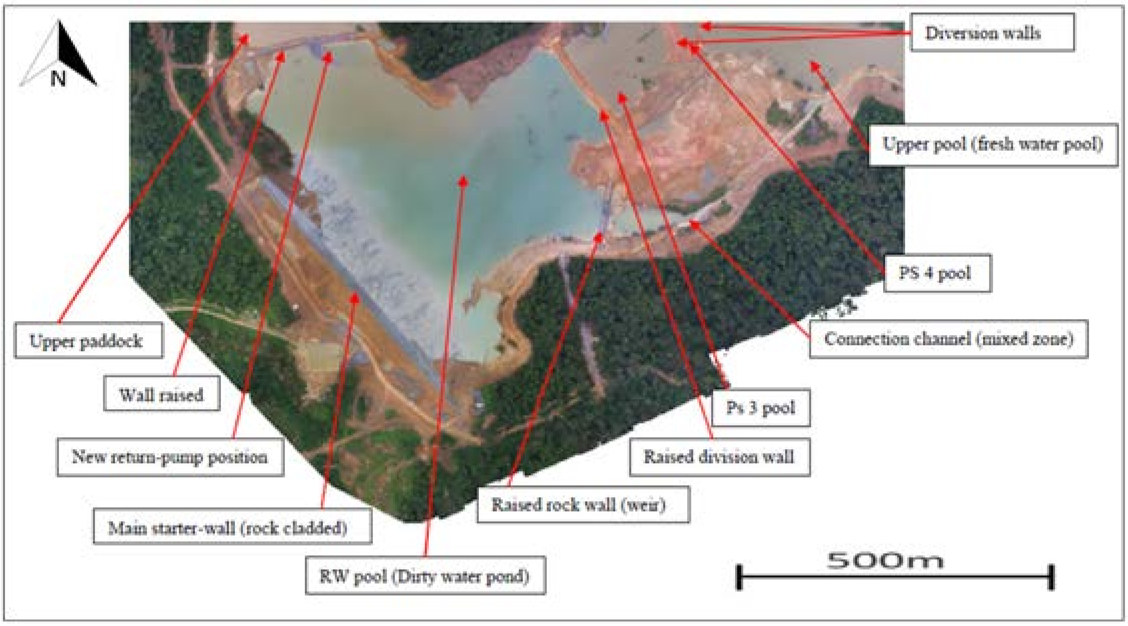
- A valley storage impoundment structure which utilises the natural topography of the river valley, consisting of a single embankment to the southwest of the facility.
- A starter embankment constructed from imported fill material, won from a borrow source located within the TSF footprint. The upstream face has a 1V:2H slope, with the downstream slope having a 1V:3H slope. The starter embankment has been constructed to specifications and the internal drainage systems are functioning as intended.
- A concrete penstock decant structure to transport supernatant fluids to the return water dam. The concrete penstock consists of a single 900ND class 150D reinforced concrete pipeline, which extends through the central area. A series of vertical penstock sections have been installed, which are placed progressively away from the main embankment. This will ensure operability is maintained as the tailings level increases.
- A seepage cut-off trench located beneath the upstream side of the main embankment. This has been installed to a total depth of 5.0m, or the depth to competent rock. This consists of compacted clay material, sourced locally.
- A vertical chimney drain (or curtain drain) within the main embankment to control the phreatic surface within the starter embankment during the operations phase.
- A 315mm OD HDPE tailings discharge pipeline, with discharge valves installed along the crest of the main embankment.
- A return water dam located to the downstream (southwest) side of the main embankment. Outflow from the decant penstock is discharged into this structure. A return water pipeline transported fluids back to the process plant. This has since been modified.

As of the beginning of August 2017, the TSF has been operated as a self-raising facility, in which deposited tailings material will be reworked to form the main embankment itself. Using the so called ‘day-wall’ method of deposition, coarse tailings are collected in a series of cells adjacent to the main embankment for use as construction material. Fine tailings plus supernatant drains towards the central pond location. Due to lower than forecast production rates to date, no upstream raises have been constructed to date, however, a drained beach above water (BAW) zone has been developed against the main embankment.

Based upon the last TSF review report supplied by BMMC (April 2017), the operating freeboard at that time was 2.7m, which exceeds the minimum freeboard of 1.0m and is therefore within acceptable limits. This report notes an ‘extremely high rate of rise’, which is a function of the reduced footprint area of the TSF available for tailings storage (described in more detail below). The report states that an average Rate of Rise (RoR) of 3.21 to 3.55 m/y has been calculated. Using the lower bound RoR value, this would indicate that there is a maximum of approximately six months of storage capacity available in the current TSF arrangement (i.e. November 2017) until the minimum freeboard marker is breached.

18.9.2 Current Status

The configuration of TSF was significantly altered during 2016. This was required due to the periodic uncontrolled release of supernatant to the environment between December 2015 and June 2016 which did not meet compliance limits. and was caused as a result of a defective cyanide destruction circuit in the processing plant. A temporary TSF configuration was constructed to ensure that discharge of excess supernatant to the environment met acceptable discharge limits (see in Figure 18-5 below). This involved segregation of the TSF into a series of compartments or cells, designed to promote a tortuous flow path for supernatant before discharge via the penstock to environment. This, combined with plant modifications, has ensured that discharge water quality has improved and is reported to now be within acceptable limits. Downstream water quality monitoring records suggest that all discharges to environment are now compliant (refer to Section 20.5).



Source: *BMMC, 2016*

Figure 18-5: Aerial photograph overview of the TSF (temporary configuration 2016/2017)

To ensure that a well-developed BAW and pond offset from the main embankment are maintained, BMMC has implemented a specific tailings deposition strategy involving rapid rotation between the discharge spigots (15 minute intervals) to steepen the beach along all sections of the main embankment. This appears to have been successful throughout the wet season and the beach slope angle is estimated to be between 0.25 and 0.5%.

The tailings distribution pipeline and spigot system has been extended along the southern flank of the TSF, which will allow tailings distribution to occur over a wider area. Following completion of the penstock extension (as detailed below), it is now possible to utiliser wider areas of the TSF for tailings storage. The southern pipeline section is due to was commissioned during August 2017.

18.9.3 Proposed Alternative Arrangement

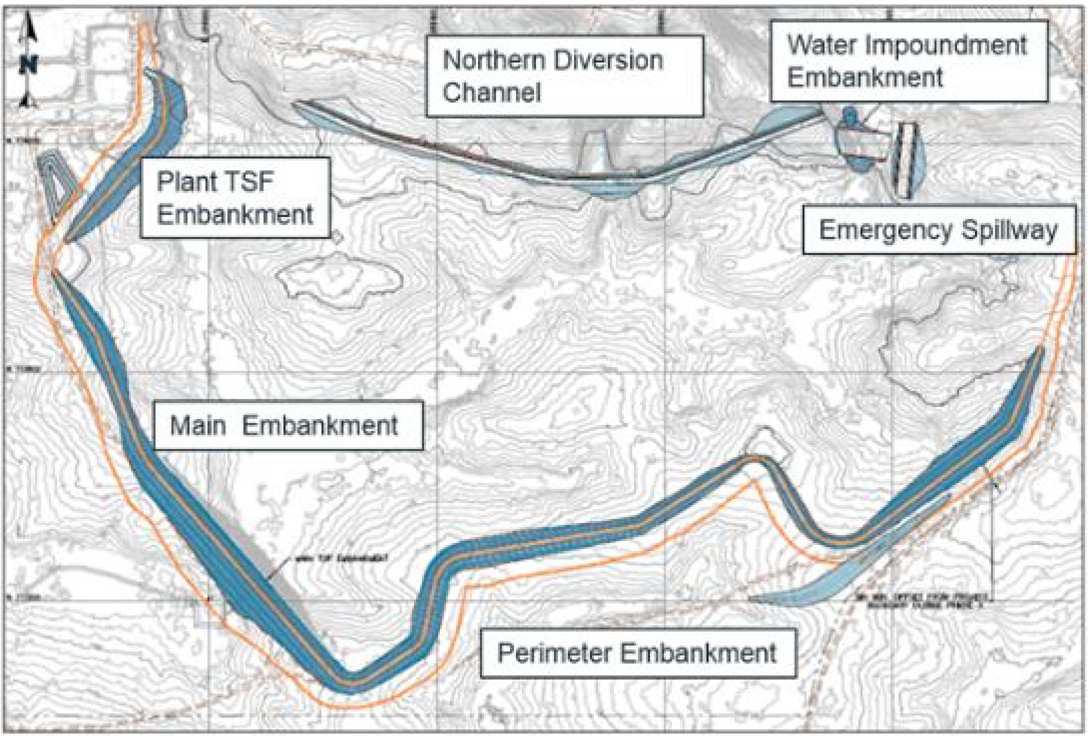
NewFields was commissioned by BMMC during October 2016 to prepare an alternative TSF design, which would allow safe storage of water on the facility and controlled release of supernatant to the environment. This new design involves conversion of the TSF to a water retaining, downstream raised facility. In addition, a water retaining dam is to be constructed to the east of the TSF, which will divert inflows of fresh water from the upstream catchment during storm events. This fresh water will be routed via the existing penstock arrangement and safely discharged downstream.

The NewFields design concept can be summarised as follows (Figure 18-6):

- Main Embankment - To be constructed using screened waste rock from the open pit mine. The upstream face of the main embankment will be fully lined with 2mm double side texture HDPE geomembrane. This will be underlain by a non-woven geotextile and granular filter zones. The embankment shall be raised twice, using the downstream construction method. The embankment will extend around the south side of the TSF to allow deposition from this area (this embankment section is known as the 'Perimeter Embankment').
- Water Embankment - Consists of a homogenous earth fill embankment, constructed to elevation 80.5 (the elevation of the existing road to the east of the facility). The upstream and downstream slopes of the embankment will be fully lined with HDPE. A gravity flow pipe (1m^3/s flow capacity) connects the water pond to existing Penstock 4, to convey fresh water runoff directly to the downstream side of the TSF. The Water Embankment shall be protected with and emergency spillway, which will be sized to accommodate anticipated flows associated with storm events.
- Stormwater diversion channels running along the north side of the TSF and a portion of the south side. These have been designed to minimise inflows of freshwater into the facility.
- Two barge mounted return water pumps which will be retained to ensure excess water is pumped back to the processing plant for re-use. The design includes provision to pump excess water via Penstock 5, for mixing with the freshwater discharge gravity flow pipe (to be used for emergency release events only).

Issued for Construction (IFC) level engineering drawings and technical specifications for all materials have been produced for the above design by NewFields. SRK considers these sufficiently detailed for construction.

Based upon a construction schedule provided by BMMC (July 2017), construction of the penstock extension to the Water Embankment (extension between Penstock 4-6) is essentially complete. All remaining earthworks are currently postponed until late October 2017 (i.e. the beginning of the next dry season).



Source: *BMMC, 2017*

Figure 18-6: Proposed Alternative TSF General Arrangement

SRK Comments

- NewFields completed a geotechnical field investigation to build upon existing data obtained from the previous Knight Piesold (KP) investigations completed during 2013 and before commencement of operations. Confirmatory boreholes, trial pits and Standard Penetration Testing (SPT) was completed beneath new embankment areas not previously considered in the original design. Whilst no report was issued to document these works, BMMC has indicated that this information was used to inform the design of the new arrangement.
- Construction Scheduling – Based on the construction schedule provided, the main embankment will be constructed and fully lined by early March 2018. The perimeter embankment construction activities are due to be completed at the beginning of May 2018. BMMC has confirmed that 5 ADT trucks will be released from the mine fleet between October 2017 and March 2018 to transport rockfill material from the open pit for embankment construction. Based on the estimated volumes of material required for construction (as provided in material take offs) SRK considers the construction schedule to be feasible.
- BMMC has confirmed that NewFields will be retained throughout the 2017/2018 dry season to supervise QA/QC testing and reporting during embankment construction activities. NewFields will be responsible for preparation of daily construction reports.

- As noted above, due to continued deposition within the main compartment of the TSF only, the rate of rise significantly exceeds that assumed in the NewFields design. Volumetric analysis completed by SRK using a DTM of the main compartment indicates that the available volume for tailings storage in this sector is reaching capacity (assuming 1.0m freeboard allowance) as of November 2017. This could potentially lead to a shortfall in tailings storage capacity until such time as the main embankment is commissioned in March 2018. To alleviate this issue, BMMC is in the process of installing additional tails delivery pipelines to transport tailings further east, to expand the area available for tailings deposition into the Upper Pool Area.
- SRK notes that the forecast tailings production rate is some 120kt per month up to and including November 2017and increasing to 140ktpm from December 2017 onwards. NewFields capacity calculations are based on an average deposition rate of 110kpta. SRK has recommended that the volumetric checks and capacity calculations are updated to provide an accurate estimate of the anticipated storage capacity and to highlight if any shortfall exists going forward.
- Whilst BMMC appear to be taking reasonable measures to maximise the remaining capacity of the TSF (by extending tailings distribution pipeline and spigots around the southern flank), the tailings deposition strategy and volumetric should be updated to minimise the risk of plant downtime as the proposed embankment raises are constructed. Should the shortfall in overall capacity be confirmed and alternative deposition strategy may have to be implemented (for a temporary period).
- The total capital cost allocated to the TSF modifications is USD1.5M.

Overall, SRK considers the design of proposed TSF modifications to be a workable solution, assuming that the critical structures (particularly in the Main Embankment and Perimeter Embankment) can be constructed timeously with the tailings rate of rise in the current facility. Notwithstanding this, as discussed above, there is potential for a temporary period of shortfall in capacity for tailings storage until such time as the tailings delivery pipeline can be extended into the Upper Pool Area. This will allow tailings to be distributed over a wider footprint area as the Main Embankment construction work proceeds in late 2017/2018.

18.10 Marvoe Creek Diversion

The Marvoe Creek Diversion Channel (MCDC) consists of a river diversion channel which routes surface water flows around the north and west side of the open pit area. Two water retaining embankments (Embankment or Dam 1 and 2) have been constructed to divert flows from away from the open pit (Figure 18-7). The channel sections around the northern side of the pit have been constructed with erosion protection features (gabions, rockfill protection etc) as has the spillway section. Notwithstanding minor erosion due to high wet season flows, these features appear to be functioning as designed.

Source: *BMMC, 2017*

Figure 18-7: MCDC Overview

SRK inspected both Marvoe Creek Diversion Channel (MCDC) diversion embankments during its last site visit in November 2016. The following observations were noted at this time:

- An eroded area at the downstream toe of Embankment 1 has been cleared and covered with geofabric material. Whilst continued seepage was noted through the lower sections, installation of this feature will reduce the likelihood of fines migration through the main embankment. It is not clear whether seepage is through the foundations of the TSF or if this is a sign that the internal drainage systems are functioning correctly. SRK has recommended that this embankment is shored up with waste rock as a precautionary measure, to ensure that the integrity of the embankment in maintained.
- At Embankment 2, no seepage was noted, however, it is noted that the pressure relief valve which extends through the embankment is exposed and could be prone to damage as the waste rock dump (WRD) expands across this zone. SRK recommends that a layer of sacrificial fill (laterite soil) is placed around this feature and that care is taken during placement of waste rock on the downstream site of Embankment 2.

Recent drone survey footage provided by BMMC (June 2017) indicates that there are large areas of ponded water downstream of both MCDC embankments, which has collected over the wet season period. This is a function of partial blockage of a culvert beneath a haul road and possibly blockage of the perimeter diversion ditch. BMMC has acknowledge this issue and has set up three temporary water pumps to pump water back into the MCDC throughout the wet season period. SRK notes that continued ponding of water could impact on the slope stability of the adjacent waste rock dumps in these areas and should be prevented from occurring.

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 (Section 22) BMMC has used a flat gold price of USD1,300/oz.

There is currently no gold refining capability in Liberia. As such, the gold doré produced at the New Liberty operation is air freighted from site to refineries in Europe.

19.2 Contracts

There are currently 11 key contracts in place for the supply of goods and services to the Project, as summarised below in Table 19-1. These are discussed further below.

Table 19-1: NLGM Contracts

	Provider	Product or Service
1	Aminata and Sons	Diesel fuel supply
2	Jozi Power/ Liberia Power	Generators and power supply
3	CGGC	Explosives
4	Manex	Explosives
5	Black Pool	Security services (international)
6	Sodjatt Guard Service	Security services (local)
7	Excops	Security services (local)
8	African Accommodation Providers	Catering
9	ALS Liberia	Laboratory services
10	Peridot	Mining equipment rental
11	BIA Equipment	Supply of spare parts

19.2.1 Fuel Supply

As already commented, the fuel supply agreement between BMMC and Aminata and Sons (Aminta) was signed on 1 January 2015 with a period of 8 years. This agreement was amended on 1 August 2017 to reduce overhead costs by enabling BMMC to manage and maintain the fuel and lubricant storage depot at the New Liberty mine site. BMMC staff are now responsible for the following, with Aminta retaining responsibility for the supply and delivery of fuel to site:

- Operations:
  - 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
- Site and risk management and daily control of usage

19.2.2 Power Generation

As already commented, the external contractor (Jozi Power) provides an 11 kV, 10.8 MW, diesel driven, build, own, operate and transfer (BOOT) power station at the Project. The generators are housed in 12 m shipping containers.

There are two contracts with Jozi Power: one for the supply of power and one for the deferred purchase of the equipment. Both contracts were signed on 06 February 2014 and both are valid for a period of 6 years, although the contractor has recently been given notice of termination to take effect on 31 December 2017. The aim is to bring this service in-house in order to reduce the on-going operating costs of power generation.

A provision has been made by BMMC to purchase identical diesel driven generators which are due to be commissioned on site in December 2017, prior to the decommissioning and removal of the Jozi provided generator sets.

19.2.3 Explosives

There are two contracts in place, one with CGGC and the other with Manex (which has very little business activity on site).

CGGC

An explosives supply contract with CGGC (an in-country Chinese explosives supplier) for the supply of bulk emulsion and blasting accessories was signed on 28 September 2016 and this is valid for a period of 3 years. The explosive is supplied to the blasthole, and an MMU truck has been provided by the contractor to facilitate this.

CGGC honours its supply responsibility for emulsion and blasting accessories from a production facility in Bong county and the facility on the site. They have supplied explosives to New Liberty for a year.

Manex

The original explosives supply contract was signed with Manex, at the time the only registered supplier in Liberia who had the potential to meet the demand.

The Manex contract was signed on 25th February 2015 with a duration of five years. It covered the supply of explosives and the charging of the holes. Manex have teamed up with Maxam in Ghana to supply emulsion and accessories to site under licence and these are only used when the stock is available.

Manex has a partially completed production facility on site which has yet to be commissioned. Supply of explosives from Manex have been sporadic and the primary supplier on site is CGGC.

19.2.4 Security

Black Pool

Security operations are managed by a primary Security Contractor ("Black Pool Security") who oversee two Liberian security contractors (SOGUSS and ExCops with 186 staff in total) and the Liberian Government Emergency Response (ERU) Unit (15 staff) as well as their own staff.

Following the expiry of the contract with Black Pool Risk Management on 1 September 2017, a new two-year contract was signed covering their new agreed duties. This new wide-ranging security, fire-fighting and ad-hoc emergency services contract between Black Pool and BMMC was signed on 1 September 2017 and it has a two-year term.

This contract covers Risk consultancy, security management, gold room and plant security, fire-fighting and training, ad-hoc emergency contingency services and aircraft facilitation. In addition, they will develop and implement security procedures, undertake security, risk and threat assessments, facilitate national police liaison, undertake mentorship, undertake physical CCTV monitoring, investigate incidents, gather intelligence and report on performance.

Black Pool supply seven personnel for these duties and supervise the Liberian security contractors.

Sodjatt Guard Service (SOGUSS)

The SOGUSS contract covers the rendering of guard services for BMMC sites at New Liberty, Ndablama, Weaju (and previously at the Aureus office In Monrovia). It is responsible for providing uniformed guards on a 24 hour per day basis.

With 209 uniformed guards provided, SOGUSS is the main provider of manned security across all of BMMC's properties.

The contract was signed on 2 January 2016 with a period of 12 months. This contract needs to be renewed, however, SOGUSS continues to provide services to the site.

ExCops

The contract with ExCops (another manned guard provider) is not been completed yet. ExCops have 14 people deployed at New Liberty, working side-by-side with the SOGUSS.

19.2.5 Catering

Catering contractor Africa Accommodation Providers Inc (AAP), a Liberian company are contracted to supply Liberian food service for BMMC employees within the operating site of the New Liberty Mine. Daily meals are provided from the catering operation at Camp David and charges are based on a rate per meal. The provider feeds all the local junior staff on the mine. The contract between AAP and BMMC was signed on 19 April 2017 and expires on 30 April 2018.

19.2.6 Laboratory

As already commented, the on-site laboratory is run by an independent third-party laboratory service provider (“ALS Global”). The ALS contract was signed in May 2015 for a 5-year term. While the contract is not physical date, this was emailed out as signed by BMMC on 8 May 2015.

ALS provide BMMC with the laboratory buildings and equipment, technical operations management and staffing and will undertake all laboratory analyses and tests, manage consumables, ensure that methods and procedures are appropriate, ensure quality systems are in line with industry standards and conduct routine quality assurance programmes.

BMMC provides all the utility services, security meals and accommodation, emergency and first aid treatment cover.

The contract entails the payment of a fixed charge and a variable charge based on the number and type of procedures undertaken.

19.2.7 Mining Equipment Rental

The contract with Peridot focuses on the hire of mining equipment to the BMMC operation. The reason for entering into the agreement was to boost the mining fleet and accelerate the waste mining rate.

The Peridot contract was signed on 7 December 2016 for a period of 3 years. The contractor currently provides the following equipment to site at the following monthly lease rates:

- Komatsu Excavator PC1250 USD30,000/mth
- Cat 992C Wheel loader USD22,500/mth
- Cat 777 dump truck USD11,000/mth

19.2.8 Spare Parts

There are two agreements with Equipments & Services BIA s.a (BIA), one for the supply of spare parts and the other for the on-site support.

The contract for the supply of spares focuses on the exclusive supply of Komatsu and Sandvik parts and components to the BMMC mining operation, the provision of information reports on open orders, optimising the stock levels on site, building up a delivery planning of consumables, providing advice regarding safety stock, using special prices and discounts for spares and performing follow-up site visits.

The effective date of the contract is 1 April 2015 which had the initial period of 2 years and has been automatically extended for a further 2 years.

The contract covering the on-site support has the same start date and period as that for the spare parts supply. This contract sets clear guidelines for the contractor on-site support team for Komatsu and Sandvik machines.

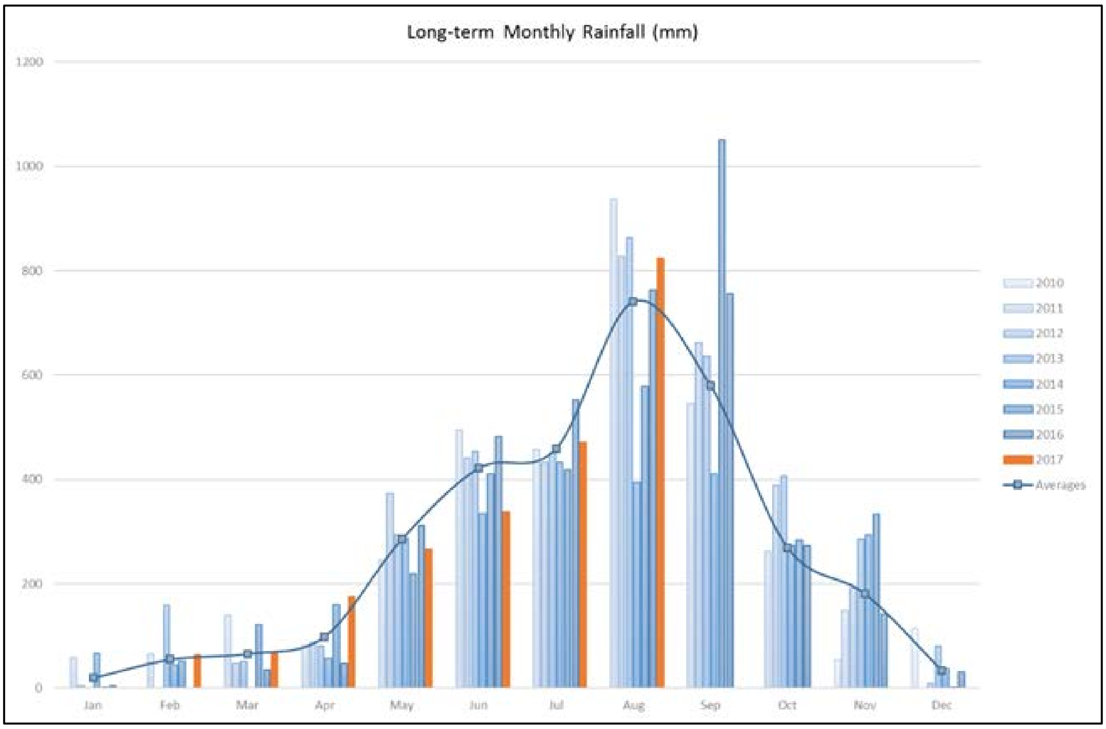
20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The information presented in this section is based on a series of environmental and social impact assessment (ESIA) reports produced for BMMC (Golders 2012 and Digby Wells Environmental 2013a, 2014), government approval documents and the mine's environmental and social (E&S) management plans, monthly and annual reports, monitoring data and audit reports. SRK environment and social specialists have visited the mine site prior to the construction phase (June 2013), at the end of the construction phase/at the time of plant commissioning (July 2015) and during operations (April and November 2016).

20.2 Environmental and Social Setting

NLGM is situated in the north-western portion of Liberia within the Gola Konneh District of Grand Cape Mount County. The climate is equatorial with a wet season extending from May to November. Rainfall that has been recorded at the mine site in the last seven years and is shown in Figure 20-1. The average annual rainfall recorded at the site during the period November 2010 to August 2017 is 3,387 mm, which is less than what is typically recorded along the coastal belt, which is over 4,000mm. The temperatures on site are generally within the range 20ºC to 35ºC and are generally lower in the wet season than in the dry season.

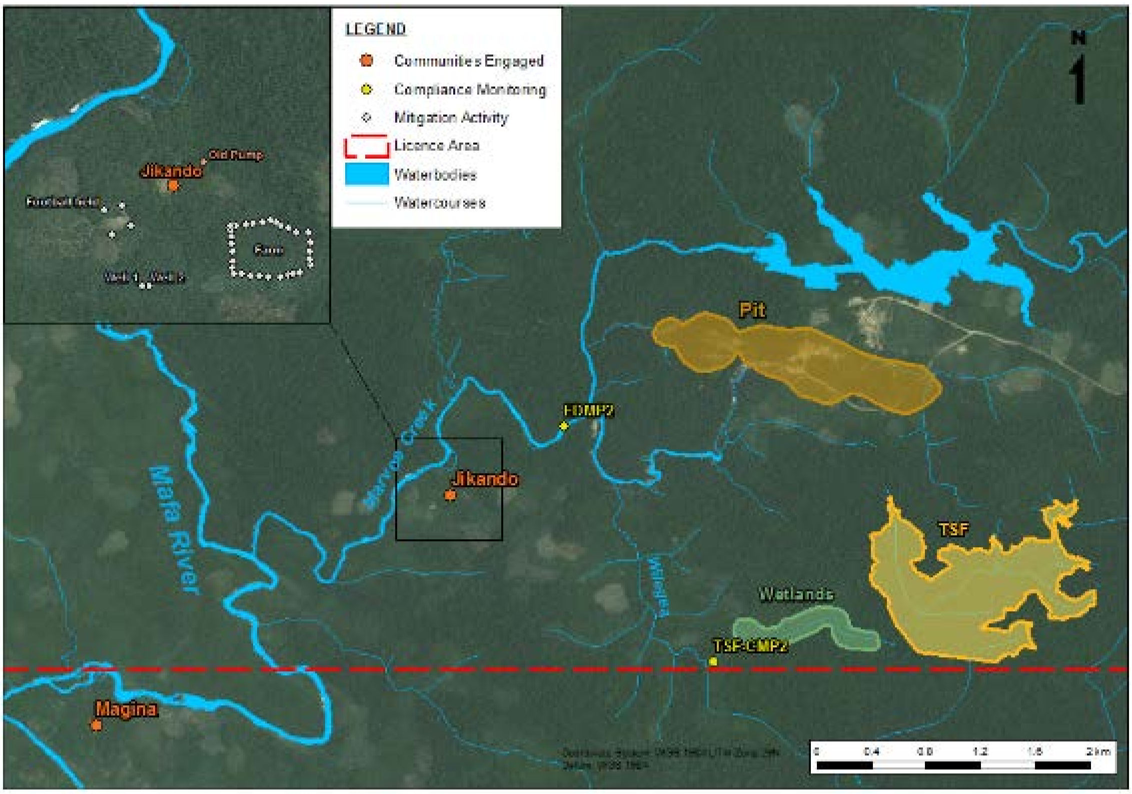


Source: BMMC, 2017

Figure 20-1: Monthly rainfall recorded at NLGM over the period November 2010 to August 2017

The mine is 40km from the coast and the topography is gently undulating with occasional small hills. The mine site is in the catchments of the Marvoe Creek and the Wilagea Creek, a tributary of Marvoe Creek, within the Mafa River basin. All the named rivers are perennial. The Marvoe Creek has been diverted around the mine site. The TSF is in the catchment of the Wilagea Creek. The confluence of the Wilagea Creek and the diverted Marvoe Creek is about 4.5km downstream of the TSF. The confluence of the Marvoe Creek and the Mafa River is about 8 km downstream of the TSF. The Mafa River flows into the Atlantic Ocean.

Figure 20-2 below illustrates the watercourses downstream of the mine and environmental compliance points for water quality monitoring. It also shows the first two of three villages downstream of the mine.



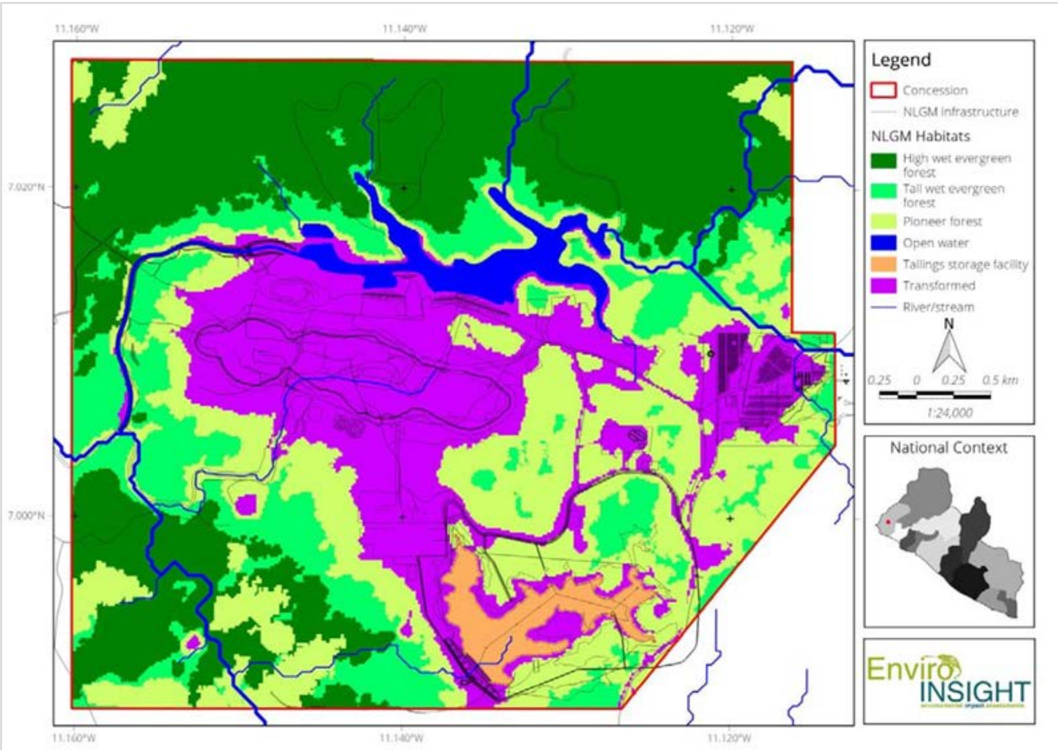
Source: BMMC (2017)

Figure 20-2: Watercourses downstream of the mine

Baseline water quality studies undertaken prior to development of the mine found that the quality of water in watercourses draining the mine site was generally good, but levels of aluminium, iron and arsenic were elevated in some samples. Historical artisanal mining was found to have had a measurable impact on aquatic habitats and ecological water quality, particularly the Marvoe Creek.

Dense tall rainforest surrounds the mine site. Prior to mining, the mine site had been somewhat disturbed by past artisanal mining, prospecting, logging and bush meat hunting. Figure 20-3 presents a map of habitats in the BMMC concession that was produced in 2016.

Several biodiversity studies have been completed in the vicinity of the mine. By 2015, these studies had recorded 264 plant species, which included 82 upper Guinea endemic species and two species endemic to Liberia. The diversity of fish and amphibian species was found to be high. Three of the 17 fish species recorded are near threatened or vulnerable species. In February 2017, Digby Wells completed an aquatic survey around the mine site. As a result of this study, 22 species of fish and a number of aquatic invertebrates were identified. Of the 28 amphibian species recorded, more than half are Upper Guinea endemics. Reptiles identified included the vulnerable African Dwarf Crocodile (*Osteolaemus tetraspis*). Numerous bird species (139) were recorded, including vulnerable hornbill, parrot and greenbul species (specifically *Ceratogymna elata*, *Psittacus timneh* and *Criniger olivaceus*). Thirty mammal species were recorded including bats, rats, squirrels, antelope and primate species.



Source: BMMC (2017) Produced by Enviro Insight for BMMC. Note that there is one habitat that is not mapped. This is Riverine Forest and Swamp Forest, which is associated with the rivers and streams.

Figure 20-3: Habitat types in the BMMC Concession

The area around the mine is sparsely populated. Only six settlements were recorded within a 5 km radius of the mine site during social baseline studies in 2011 and 2012. Of these, Jawajei, Ganganma and Weagea are not directly affected by the mine activities, Kinjor and Larjor were situated in the mine foot print and had to be relocated and Jikando is situated about 5 km downstream of the mine site and TSF (Figure 20-2). Further downstream of the TSF, near the Mafa River are the settlements Magina (also referred to as "Malina" or "Madina"), and Koma (also referred to as "Koma Djacin" and "Kohnma"), about 11km and 12km downstream respectively.

The nearest major towns to the NLGM project site are Sinje (approximately 40 km) and Daniels Town (22 km). Sinje is the nearest settlement with government medical facilities and education facilities beyond primary level.

Kinjor and Larjor have been relocated to a settlement site while permanent replacement structures are being completed. The two settlements comprised 322 households plus associated facilities including business structures, community facilities, economic trees, agricultural fields, graves and sacred sites. The resettlement action plan (RAP) for the resettlement was completed in 2013 and amended in 2014 both in response to changes in the mine plan and a need for temporary resettlement to accommodate the construction schedule. During the RAP preparation, it was agreed that the affected households would be relocated to one site approximately 4.5km from Kinjor, initially referred to as the “Leilema site”, now known as “New Kinjor”. Full execution of the RAP was delayed by the Ebola epidemic (2014 to mid-2015) and a period of financial instability experienced by the mine (mid-2015 to mid-2016). This has resulted in the project affected households remaining in the temporary accommodation, constructed in 2014, while construction of the permanent dwellings is completed. Of the 322 household structures, 100 have been completed, with another 100 underway. Completion of the permanent dwellings is planned for 2018. It has been agreed that each affected household will keep both the temporary and the permanent dwellings. The status of the resettlement process is discussed further in Section 20.6.2.

The civil wars in Liberia (1990-2003) caused population displacement, disrupted infrastructure and services and exacerbated poverty in the country. These effects were evident in the project study area at the time the 2011 – 2012 ESIA was undertaken. The ESIA recorded that the small settlements in the vicinity of the mine site were engaged in livelihood activities such as agriculture, artisanal mining and work associated with mining prospects. The livelihoods of people living in Kinjor and Larjor were largely based on artisanal mining. Artisanal miners are thought to have entered the area in the 1960s, and residents of Kinjor described the settlement as being established in 1970. Larjor and Kinjor increased in size following establishment of BMMC’s mine camp in 1998, attracting families with potential opportunities for formal and informal employment, and with ongoing artisanal mining prospects.

The livelihoods of other villages in the area around the mine are largely based on subsistence agriculture and fishing from streams and rivers. Unlike the agricultural villages, very few of the households in Kinjor and Larjor had access to farming land and so they purchased food from surrounding farmers or from Monrovia.

The ESIA recorded that structures within the project area reflected a dependence on natural, local resources and the lack of access to more durable, man-made construction materials. Energy needs were serviced by natural available resources, primarily by wood or charcoal. Villages did not have piped water and mostly reliant on streams and rivers for water supply. By 2012, BMMC had installed water pumps in Kinjor, Jikando and Jawajei villages. In 2016, BMMC installed water pumps in Magina and Koma too.

Malaria is the most common illness affecting local communities and other common illnesses are respiratory illnesses, typhoid and diarrhoeal diseases. Food shortages are experienced during the wet season. Many villages lack latrines and practice open defaecation.

Grand Cape Mount County has only one paved major road, leading from Monrovia to Bo Waterside. Access to the mine site is via this road and then via a 20 km laterite road between Daniel’s Town and the mine site. This laterite road was upgraded so that large items of equipment could be brought to the site in the construction phase.

About one third of Grand Cape Mount County's population is literate and the ratio of literate females to males is low. Education levels are largely dependent on the proximity of households to schools.

20.3 Permits and Approvals

The primary approvals for the NLGM take the form of:

- A Mineral Development Agreement (MDA) between the Government of Liberia and BMMC, dated 18 September 2013 and which replaced an earlier MDA dated 28 November 2001;
- A Class A Mining Licence granted to BMMC on 29 July 2009;
- An environmental permit granted to BMMC by the Liberian Environmental Protection Agency (EPA) on 4 November 2012, which was renewed as required in December 2015 and will need to be renewed in December 2018; and
- A discharge permit granted to BMMC by EPA for discharges from the TSF, which is renewed on an annual basis and was recently renewed on 26 June 2017.

All of the above approvals, with the exception of the Mining Licence, contain environmental and social conditions.

In addition to the above approvals, BMMC has approvals for exploration, development of a landfill waste site, importation and handling of cyanide (valid until 29 May 2018) and is renewing the radioactive sources permit.

A number of the environment and social management plans produced for NLGM have been formally approved by government and are referred to in the MDA, environmental permit and/or discharge permit. These include:

- A Resettlement Action Plan (RAP), formally approved by the EPA on 25 March 2013;
- A Community Development Plan (CDP), approved by the EPA on 25 March 2013;
- The mine's environmental and social management plan (ESMP) within the ESIA report approved by the EPA (assumed to be the 2013 ESIA report, which was the last ESIA report submitted to the EPA for approval), which is referred to in the environmental permit; and
- The cyanide management plan, which focuses on cyanide in water released by the mine, referred to in discharge permit.

The MDA also refers to a project linkages plan, which is also referred to in the MDA as a plan for procurement of local goods and services. The status of this plan is discussed below in Sections 20.6.4.

The International Finance Corporation (IFC) had an equity investment in the NLGM project over the period 2014 to 2016 and required extensive review and revision of the mine's ESMP prior to making this investment. A series of environment and social management plans were produced for NLGM, by consultants, under the auspices of the IFC. A problem with the management plans is that there are voluminous and add overwhelmingly to the mine's compliance obligations.

NLGM has numerous compliance obligations in the above mentioned approvals and management plans. In 2015, a substantial effort was made to bring the commitments into a register but there were gaps and inconsistencies in the register and for various reasons discussed in the next section there was little progress in updating the register until recently.

Many of NLGM compliance obligations are unachievable and in SRK's opinion, BMMC should review its compliance obligations, beginning with those that are legally binding – including conditions in the MDA, environmental permit and discharge permit. Revisions to those obligations that are unrealistic or poorly worded need to be proposed and agreed with relevant regulatory authorities.

The RAP is out of date but it is accompanied by an updated Memoranda of Understanding (MOU) (Section 20.6.2). The CDP, although formally approved by the EPA, is an advisory document with many recommendations but no clear commitments. BMMC's commitments to community development need to be more clearly framed and aligned with the government expectations implied in the MDA, as well as the need to restore the livelihoods of people who have been displaced by development of the mine.

The ESMP referred to in the environmental permit needs thorough revision so that it is relevant to the mining operation and focused on key environment and social risks. The environment and social management plans produced subsequently (most of which have not been formally submitted to government) need to be rationalised so that they are less bulky and easier to comprehend and implement.

BMMC is aware of the above shortcomings and intends to address these, in consultation with relevant regulatory authorities.

20.4 Environmental, Social, Health and Safety Management System

There are some elements of an environmental, social, health and safety (ESHS) management system in place at NLGM, but the management system is not fully fledged. Hindrances to the establishment of the management system include the Ebola crisis (2014 to mid-2015), a shortage of finances experienced by the mine (mid-2015 to mid-2016) and staff vacancies for key ESHS management positions (in Q1 and Q2 2017). Other challenges have included the need for high investment in training of local contractors on site in ESHS management, coupled with a high turnover of contractors.

Vacancies for new environmental and safety supervisors have recently been filled.

Elements of the management system that need attention are:

- Evidence of top level management commitment to ESHS management including adequate human and financial resources for ESHS management and management reviews of the system's adequacy, suitability and effectiveness;
- Tracking of compliance with compliance obligations (Section 20.3);
- Review and updating of a ESHS risk register and a focus on key risks (the risks identified in Section 20.5);
- Refinements to monitoring programmes, particularly water quality monitoring data interpretation (Section 20.5.2);

- Better documentation of the safety elements of the ESHS management system, including hazard identification, risk assessment and incident investigation;
- Regular internal audits; and
- Better integration of the environmental, social and safety elements of the system.
- A more systematic approach to ESHS management has been taken in response to the cyanide incident at the mine (Section 20.5.1). Lessons learned and actions taken in response to this incident should be transferred to the ESHS management system as a whole. Specifically, with respect to cyanide and arsenic in discharges from the TSF, there is good evidence of most elements of an ESHS management system being in place. Most importantly, it is clear that there is top management commitment ensuring there is compliance with relevant compliance criteria. Organisational roles, responsibilities and authorities are well defined, staff are aware of compliance obligations, monitoring and evaluation of compliance is undertaken at the necessary frequency, and there is clear evidence of management review and corrective actions being undertaken in response to nonconformities.

20.5 Key Environmental Issues

20.5.1 Compliance with Cyanide and Arsenic Criteria in Water Downstream of the TSF

BMMC manages the mineral processing operation, the tailings detoxification plant and the TSF operations such that cyanide and arsenic compliance criteria in the watercourses downstream of the mine are not exceeded.

The tailings detoxification plant is equipped with two units to destroy cyanide and promote arsenic precipitation from tailings. Arsenic removal is required because the ore is rich in arsenic and arsenic is liberated during the mineral processing operation. Arsenic is removed by means of an iron co-precipitation process. Cyanide is destroyed by an INCO cyanide destruction unit.

After the mineral processing operation was first commissioned, there was a suite of challenges that resulted in failure to meet cyanide compliance criteria downstream of the mine and fish deaths in the downstream watercourses were observed. The problems have been addressed and impact studies by independent specialists contracted by the Company have confirmed that the river ecosystem has largely recovered and that people living downstream of the mine have not been adversely affected.

Among the measures put in place to ensure compliance with cyanide and arsenic criteria for downstream watercourses are:

- Improvements to the control of the leaching circuit and detoxification plant;
- Improvements to the design of the TSF so that the water volume and residence time of the TSF was increased so there is more time for breakdown of residual cyanide and precipitation of arsenic on the TSF;
- Improvements in the process water management so to optimise the return of water from the TSF to the process;
- Modification of the natural wetland below the TSF to include permeable reactive barriers (gabion baskets with iron and charcoal) and increase the reed bed density to slow down transit of water through the wetland so that arsenic, suspended solids and cyanide levels

in the water flowing into the Wilagea Creek are as low as possible;

- Improvements to the water monitoring including upgrading of equipment and procedures in the analytical laboratory on the mine site and establishment of a database that facilitates interpretation of the monitoring data; and
- Improvement to the ESHS management system including top management commitment ensuring there is compliance with relevant compliance criteria, daily review of management review of monitoring data and corrective action taken in response to nonconformities.

Use of the watercourses downstream of the mine is limited, but there are three villages between 6 km and 14 km downstream of the TSF (Jikando, Magina and Koma). BMMC has drilled boreholes to supply drinking water to these villages and has equipped these with hand pumps. BMMC has also provided the villages with supplementary food in the form of fish and beans and intends to continue with this support until December 2017 as independent ecological monitoring has confirmed that the fish abundance recovery in the downstream rivers is complete. Social and health specialists have warned BMMC that the supply of food to the communities is not sustainable and a dependence on this food has already developed. Mitigation measures will be required to ensure the health of the communities is maintained when there is a decision to withdraw the support.

The mine's monitoring data demonstrates compliance with relevant cyanide and arsenic criteria at the environmental compliance points from May 2016 to present. The environmental compliance points defined in the mine's cyanide and arsenic management plans are below the TSF above the confluence with the Wilagea Creek (CMP2) and in the Marvoe Creek below the confluence with the Wilagea Creek (EDMP2), as shown in Figure 20-2. The mine is also compliant with the discharge criteria specified in its discharge permit, which apply to the point of release of tailings from the TSF (referred to as TSF-R by the mine) to the engineered wetland downstream of the TSF, which is more than 1 km above the Wilagea Creek confluence.

There are internal check points for cyanide and arsenic in water on the mine. These include the tailings prior to discharge to the TSF and the penstock on the TSF. Data from the internal check points suggest that the cyanide detoxification and the arsenic removal processes interfere with each other. When the cyanide detoxification performance is highly effective, the performance of the arsenic removal process is not optimal. This does not result in non-conformance with environmental compliance criteria but can result in internal check point values being exceeded. BMMC is investigating this issue with the aim of optimising the performance of both detoxification processes.

The cyanide levels in the tailings discharged to the TSF between May and October 2017 have been elevated on occasions, with weakly acid dissociable (WAD) cyanide levels exceeding 50 mg/l on 11 occasions. The incidents can be attributed to some problems with the oxygen distribution balance between pre-ox, leach and detox; blockages in the reagent tanks; and issues with the acid pumps. These problems have been attended to, respectively, by means of flow meters and regulators on oxygen delivery points, clearing of blockages, and replacement of acid pumps. BMMC reports that the average WAD cyanide levels on the TSF remain well below environmental permit and discharge permit conditions.

20.5.2 Interpretation of Water Quality Impacts and Implementation of Pollution Control Measures

BMMC has an extensive water monitoring programme, but interpretation of the data for parameters other than cyanide and arsenic has been lacking. The last detailed review of monitoring data was undertaken in early 2015 by Aquaterra and focused on 2014 data.

Water samples are taken upstream and downstream of the mine, on the mine site and in the resettlement village. They are taken from streams and boreholes, on a monthly basis and are sent to ALS in Prague for analysis. The range of parameters determined is wide.

BMMC's environmental permit requires comparison of water monitoring results with baseline water quality, but BMMC has not formally defined what constitutes baseline water quality in its management plans and procedures. The data available in the ESIA reports is limited. There is some data for a few monitoring points in the Marvoe Creek for five months (over the period September 2011 to March 2012).

BMMC is entering all monitoring data into a database to facilitate comprehensive interpretation of water quality results. SRK has seen evidence that cyanide and arsenic monitoring data is entered into the database and interpreted on an ongoing basis, but has not seen evidence this is the case for other water quality parameters.

Sediment studies recently undertaken in watercourses downstream of the mine indicate that there is moderate natural metal enrichment of sediments in these watercourses. There is no evidence that the mine is contributing to the load of metals in the sediments, but there is evidence of metal enrichment in sediments on the mine site – specifically in the relic channel of the Marvoe Creek (the creek has been diverted around the mine workings). This area of the creek is immediately downstream of the artisanal washing sites and was heavily silted as a result of this activity. Arsenic, chromium and nickel enrichment were recorded in this location. This needs to be monitored and action may need to be taken to ensure this does not impact on downstream watercourses in future.

- Several key pollution control measures have not been fully implemented by the mine yet. These measures do need to be implemented to ensure there are no impacts on water quality in the future. These measures include:
- Bunded concrete work surfaces for vehicle maintenance and associated drainage infrastructure and oil traps – these are still being installed;
- The waste sorting facility – this was partially constructed and is now being moved;
- The incinerator for burning the hazardous and medical waste on site that can be safely burned – this will be installed in 2017;

- Sedimentation basins downstream of the waste rock dumps – work has started in 2017 and will continue with a focus on the dumps with the highest potential for erosion; and
- Implementation of special measures to identify potentially acid generating waste rock and encapsulate this in the core of the waste rock dumps (only a small percentage of the rock is acid generating).

20.5.3 Biodiversity Impacts

The mine's environmental permit contains the following condition (Condition 7):

> *"Design, and operate a biodiversity offset program based on the no net loss principle that demonstrates compensation for environmental damage equivalent to or better than the loss of habitat ecosystems services and structure given in ESIA."*

Biodiversity studies were undertaken at the mine site to provide input the ESIA for the development and a draft biodiversity management plan was produced in 2014. This plan does not cover offsets. Further work has been undertaken to produce a comprehensive biodiversity management plan for the mine. SRK understands this work includes the following tasks:

- Higher resolution habitat mapping to define the different types of forest in the area;
- Surveys to confirm the absence/ presence of critical habitat triggers and to provide more precise information about populations in the area;
- More comprehensive assessment of indirect effects such as forest degradation associated with population influx;
- Advice for the closure plan restoration strategy, including more clarity on vegetation types and objectives;
- Preparation of action plans to achieve net gains in critical habitat biodiversity values; and
- Development of procedures for preventing the spread of alien invasive species on the site.

BMMC has provided SRK with a copy of a specialist report that address the first two tasks. It is based on the work of Enviro Insight (West Africa Division) and The Biodiversity Company ('Enviro Insight'), two companies registered in South Africa, with recognised experience working in West Africa. The one report is accompanied by a second report by these two consultancies that recommends an extensive biodiversity monitoring programme in response to the findings to date.

Enviro Insight reviewed previous biodiversity studies that had been undertaken for NLGM under the supervision of Golders in 2012 and Digby Wells Environmental in 2014. The study also covered a recent aquatic ecology study undertaken by Digby Wells Environmental in 2016. The review of these documents was coupled with review of international literature on critically endangered species and endangered species that could have distributions that overlap with the BMMC concession. The findings of the study are summarised below. The studies also produced the habitat map presented in earlier Figure 20-3).

- A tall tree found within the BMMC concession (in 2014) is considered to trigger the IFC Criterion 1 for critical habitat as a "flagship species". The species is *Isomacrolobium (Anthonotha) explicans* and the population identified in 2014, which is believed to comprise 25% of the known individuals of this species, appears to have been disturbed by waste rock dump development. The consultants have advised BMMC that the potential loss of this population will be regarded as a major impact on the global population of Isomacrolobium (Anthonotha) explicans and the status of the NLGM population must be re-evaluated and monitored as part of the agreed monitoring programme.
- No IUCN critically endangered or endangered bird species have known distribution ranges that overlap with the concession or have been identified within the concession.
- Four IUCN critically endangered or endangered mammal species have distributions that overlap with concessions, but have not been identified within the concession and are not expected to occur there. Four vulnerable mammal species could occur in the concession, but have not been recorded there to date.
- Habitat for amphibian and reptile species of conservation concern in the region of the project is not present in the concession. However, tadpoles of the IUCN endangered Sierra Leone water frog (Odontobatrachus natator) and Ivory Coast Frog (Amnirana occidentalis) may occasionally be present within river and streams in the concession. Also, the critically endangered African slender-snouted crocodile (Mecistops cataphractus) is likely to occur in the rivers downstream of the TSF.
- Although numerous weeds have been recorded within and surrounding the concession, most of them are native and regarded as important pioneer species that stabilise soil in cleared areas and drive the succession of pioneer forest into early secondary forest. Of all the weeds recorded to date within the concession, only three are regarded as being alien and invasive. These are Chromolaena odorata, Croton hirtus and sensitive plant (Mimosa pudica).

Based on the finding to date Enviro Insight have recommended an extensive biodiversity monitoring programme that includes:

- Water quality monitoring, including continuous monitoring of certain parameters using automated real-time logging probes linked via satellite to a web-accessible user interface or downloaded manually on a daily basis from probes, archived in a database and immediately inspected/evaluated;
- Aquatic assessment;
- Vegetation change monitoring;
- Sediment analysis;
- Roadkill monitoring;
- Diatom, macro-invertebrate and fish monitoring;
- Forest mammal community monitoring; and
- Alien and invasive fauna monitoring.

20.6 Social Commitments

The RAP was originally structured around four key elements linked to the legal framework and in accordance with the IFC Performance Standard 5. These were:

- 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.

BMMC faced a challenging start up with the construction phase being completed during the Ebola outbreak and a series of subsequent cash flow issues during initial operation phase. These challenges have contributed to the social management plans and associated data management systems not being implemented, the RAP village not being completed within the agreed time frame. Whilst a number of the social obligations and commitments to the host community have been met, there are a few that remain outstanding. These issues and their current status are expanded on in this section.

20.6.1 Stakeholder engagement planning and management

A stakeholder engagement plan was prepared in 2013 and issued in 2014. The plan while seemingly comprehensive at the start of the project was designed to guide stakeholder engagement across the life of the mine. It did not, however, account for a sustained population increase.

A degree of population influx was anticipated during the Project development and construction, through encroachment and job seekers and an outline influx management plan was prepared. This plan was not implemented because of the Ebola outbreak and associated social restrictions resulting in no control of population influx or engagement with project affected people (PAP). It was estimated that the local population increased from 2,000 (NLGM RAP 2012) to more than 8,000 (SRK 2015), reducing to around 4,500 according to a 2015 population census (University of Liberia 2015).

Community engagement and grievance management has up until recently centred on a resettlement working group (RWG). The RWG, initially established to facilitate effective community relations during the resettlement process, has assumed a wider remit. It has representation from the resettler community as well as the influx community with reported in fighting and conflict (BMMC Monthly reports 2015,2016). These and other community conflicts recorded in the BMMC monthly reports can in part be attributed to the lack of a robust stakeholder engagement plan. BMMC indicates this is being addressed.

The approach to stakeholder and community relations is currently being restructured and managed by a recently appointed Community Relations Manager and a revised stakeholder engagement plan will be finalised in November 2017. The restructuring process includes alternatives to the RWG, such as re-establishing the Resettlement Committee (RC) and operationalising its “Organisational Structure” (NLGM RAP, October 2013); introduction of the BMMC Newsletter” to enhance consistent communication with stakeholders; more regular engagement with stakeholders, and the establishment of community grievance boxes to raise grievances and a more robust grievance mechanism. The expectation is that these measures will reduce the potential risk of further social conflict associated with ineffective community engagement.

Engagement with the Government of Liberia and related ministerial departments and agencies is managed from Monrovia by the BMMC Country Manager. There is reportedly a constructive relationship between the Project and the Government evidenced by the President visiting the Project in 2015, the ‘cyanide incident’ in 2016 and support received to resolve a series of community conflict issues in 2016 and 2017, and the collaboration with the Ministry of Health to assume operation of the Kinjor Clinic constructed by BMMC in April 2017.

20.6.2 Implementation of the resettlement process

There were 322 households identified in the RAP cut off census as being physically displaced by the Project. Between this date and the planned resettlement date there was an encroachment of a further 265 households. To distinguish between the two, the Project Affected People (PAPs) original dwellings were marked with yellow paint and the ‘encroachers’ dwellings marked with blue paint. The name has stuck and these two entwined groups are commonly referred to as ‘blue paints’ and ‘yellow paints’.

Because of the Ebola epidemic, building works were delayed and a temporary relocation was agreed with the yellow paint PAPs. They were supported with materials and labour costs to construct temporary housing. BMMC put in infrastructure comprising 15 boreholes with hand pumps and 96 communal toilets for the use of the yellow paint temporary settlement. This works out at a ratio of 3.5 households per toilet and 21.5 households per well.

Further to BMMC’s obligations to resettle the directly affected households, it was agreed with the PAPs to reallocate community land for the ‘encroachers’ (blue paints) and to provide some building materials for them to construct dwellings.

At the end of 2016, the resettlement process had stalled, with approximately half of the permanent replacement dwellings completed but none being inhabited. A memorandum of understanding (MOU) was drawn up between the ‘yellow paints’ and BMMC in the first quarter of 2016 to agree completion of the permanent dwellings by December 2017. This MOU has subsequently been amended with an anticipated completion date of Q4 2018 and a commitment to complete 200 household units by end of 2017. BMMC is planning a rolling plan of occupation commencing in Q1 2018.

The construction of a health clinic for New Kinjor was completed in Q1 2017. This was a priority RAP social infrastructure commitment. SRK understands that agreement has recently been reached with the Ministry of Health regarding its staffing and operation.

20.6.3 Livelihood restoration and community development

At the time of the RAP, livelihoods were defined as being dependent on the natural resource base, with men and women engaged in a combination of subsistence agriculture and artisanal mining. Compensation payments have been made related to structures not being replaced, crops and trees lost.

BMMC has set up a number of cooperatives and community based initiatives as alternative livelihood activities. The "Inclusive Community Empowerment" wood cooperative is providing wood products for BMMC's construction work. The Kinjor Villagers are supplying the sand for the mine's construction works and the making of modular brick for RAP house unit construction, and re-engagement of local construction workers to build the RAP settlement, including low-skill employment by the mine. A tailoring initiative is still functional, albeit on a limited scale. Agricultural extension support is also being explored for mechanised farming.

Currently, a competitive local bid for supporting the development of agriculture cooperatives is being implemented to restore livelihoods displaced by the Project. Notwithstanding these initiatives, there have been a few isolated reported incidences of artisanal mining activities, albeit now illegal and regarded as trespassing. Local police have been involved with offenders being arrested and cautioned.

Reportedly BMMC is in the process of developing a comprehensive livelihood restoration plan that will be operational by the close of 2017. The intention is for this plan to include a range of additional initiatives including start-up of women's rotating credit schemes, and extension of modular brick making following the RAP house unit construction to a cooperative.

20.6.4 Social obligations

In accordance with the MDA a project linkages plan was developed and with it a number of initiatives to support local content and feed into the local regional and national economy. Currently, the mine provides a number of supply linkages for Liberian firms, which include Africa Accommodation Providers, Cape Mount Construction Company, SOGUSS, operating out of Monrovia, and employing people from the Kinjor Community.

BMMC set up a number of cooperative and community based initiatives, many of which have ceased to operate and have been replaced by Liberian firms, operating out of Monrovia, employing people from the Kinjor community. The exception is a wood cooperative and a tailoring initiative that are both still functional, albeit on a limited scale.

20.7 Provision for Closure

A closure plan was produced for the mine in 2013 by Digby Wells Environmental (2013b). The plan was presented in the ESMP within the 2013 ESIA and has, therefore, effectively been approved by the EPA.

The closure cost estimate based on the 2013 closure plan is USD10.0 million.

SRK's comments on the 2013 closure plan are listed below.

- The closure plan is highly conceptual. In terms of good international industry practice, it is appropriate to start developing a detailed plan for closure of the mine now.

- The closure design involves construction of a closure spillway and cover of the entire TSF with a 700 mm saprolite layer and a 300mm layer of topsoil. The capping method is appropriate for this type of facility.
- Regarding the transfer of assets to the government, the plan states “an agreement with the Liberian Government has been reached, whereby liability for all permanent and immovable mine infrastructure, and its future maintenance, will be formally transferred to the Liberian government upon cessation of the mining operation”. Section 29.1 of the 2013 MDA references transfer of immovable assets but does not include comment on liability and maintenance of the assets. There is a risk demolition may still be required and additional costs will be incurred.
- Modelling of post-closure water quality impacts from the TSF is a data gap and will need to be evaluated to inform closure planning and closure costing in future plan revisions. The exclusion of a provision for post-closure water treatment represents a risk of insufficient financial provisioning for closure.

SRK recommends that the closure plan is updated and the closure cost estimate is revised.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following section summarises the capital and operating costs assumed in the current LoM plan for the Project. The LoM plan assumes continuation of a conventional open-pit mining operation, a three-stage crushing process, ball and secondary milling, a CIL circuit followed by AARL elution. The plant design parameters are based on the treatment of 1.1 million tonnes per annum of ore, however, BMMC is assuming that following modifications to the plant as discussed in Section 13 and 17, the plant will be capable of processing some 1.7 million tonnes per annum of ore. The mining rate is also assumed to increase accordingly.

The forecast capital and operating costs presented below form the basis of the economic analysis presented in Section 22.

21.2 Operating Cost Estimate

21.2.1 Accuracy and Basis of Estimate

The forecast operating cost estimate has been completed by BMMC and is based on historical costs achieved during the period January to July 2017 as modified for projections going forward for August to December 2017 and beyond. All labour, materials, consumables and utilities have been included in the estimate.

The base date of this operating estimate is Q3 2017.

21.2.2 Definition 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 costs are 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
- Administration

Variable costs can be defined as costs that are only incurred during production. These costs are based on rates per tonnage and the total costs are incrementally incurred as production rates increase.

Variable costs typically include the following:

- Reagents
- Maintenance Spares
- Diesel
- Tyres

- Oil
- Consumables
- Power
- Explosives

21.2.3 Mining Costs

Introduction

The following section provides a description of the operating costs for the mining operations. It should be noted, however, that the following are not included in the mining cost estimate and are included elsewhere in the financial model:

- 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; and
- Final product transport, marketing and sales agreement costs.

The cost estimate is in USD and has been developed by SRK based on NLGM historical performance, input from BMMC, SRK's internal cost database and the 2017-2018 Infomine cost database[1].

Total Mining Operating Costs

The forecast total mining operating cost estimate is shown in Table 21-2. It is noted that the 2017 forecast figures presented in this table represent August to December 2017 (as the current LoM plan described in Chapter 16 commences from 1 August 2017), however, the economic analysis presented below Chapter 21 commences from 1 October 2017 and so any differences with the information presented in Section 22 are due to this timing difference only. The large increase in unit operating costs in Year 2022 is due to the re-handling of stockpiled material, with limited ex-pit material. A contingency of 10% was added to the forecast mining costs in 2017 and 5% thereafter. The contingency is to account for unforeseen costs, such as additional freight and duty costs, additional airfare and accommodation costs. These costs are expected to decline significantly from 2018.

[1] Infomine, 2017-2018. Equipment Cost Calculator. [online] Available at: <http://costs.infomine.com/> [Accessed July, 2017].

Table 21-1: Mining – Forecast Total Operating costs

Operating Costs	Units	Total	2017[1]	2018	2019	2020	2021	2022
Total Material Moved Expit	(kt)	**124,896**	7,950	47,131	36,813	22,050	10,748	203
Total Operating Costs	**(USDk)**	**231,217**	**17,203**	**74,691**	**65,247**	**43,801**	**28,138**	**2,137**
Loading	(USDk)	**13,234**	927	4,930	3,856	2,364	1,132	25
Hauling	(USDk)	**38,263**	1,736	13,709	12,528	6,340	3,815	135
Drilling	(USDk)	**11,433**	712	4,026	3,345	2,121	1,195	34
Ancillary	(USDk)	**31,256**	2,539	7,438	7,402	7,323	5,732	821
Fuel	(USDk)	**41,264**	2,979	13,136	11,708	7,955	5,113	373
Labour	(USDk)	**32,025**	3,443	9,076	7,982	6,082	4,908	535
Grade Control	(USDk)	**2,498**	159	943	736	441	215	4
Explosives	(USDk)	**49,489**	3,143	17,876	14,583	9,089	4,689	109
Miscellaneous	(USDk)	**11,755**	1,564	3,557	3,107	2,086	1,340	102
Unit Operating Costs	**(USD/t)**	**1.85**	**2.16**	**1.58**	**1.77**	**1.99**	**2.62**	**10.51**
Loading	(USD/t)	**0.11**	0.12	0.10	0.10	0.11	0.11	0.12
Hauling	(USD/t)	**0.31**	0.22	0.29	0.34	0.29	0.35	0.66
Drilling	(USD/t)	**0.09**	0.09	0.09	0.09	0.10	0.11	0.17
Ancillary	(USD/t)	**0.25**	0.32	0.16	0.20	0.33	0.53	4.03
Fuel	(USD/t)	**0.33**	0.37	0.28	0.32	0.36	0.48	1.84
Labour	(USD/t)	**0.26**	0.43	0.19	0.22	0.28	0.46	2.63
Grade Control	(USD/t)	**0.02**	0.02	0.02	0.02	0.02	0.02	0.02
Explosives	(USD/t)	**0.40**	0.40	0.38	0.40	0.41	0.44	0.54
Miscellaneous	(USD/t)	**0.09**	0.20	0.08	0.08	0.09	0.12	0.50

1 August to December 2017

Comparison to Historical Operating Costs

A comparison of the 2017 forecast mining costs to the historical 2017 mining costs are shown in Table 21-2.

Note that the loading and hauling costs were historically reported as a combined cost. The forecast costs for load and haul appear to be in line with 2017 historical costs. BMMC expects reductions from the costs achieved from January to June 2017 actual costs when compared to the forecast for the remainder of 2017 and further cost reductions beyond this. Specifically:

- Drilling and Explosives: A reduction is expected based on revised drill and blast parameters.
- Fuel: A reduction is expected as newer and more fuel efficient equipment replaces older equipment.
- Labour: A reduction is expected as contractors are phased out. Further reductions are expected in 2019 as ex-pat labour is replaced by locals.
- Grade control: The new programme is expected to reduce costs significantly.
- Miscellaneous: Unexpected costs, such as last minute freight and contractors are expected to reduce by following all mining operations now being brought in-house.

These cost savings represent a significant decrease from historical costs. If these reductions are not realised the costs realised will be higher than presented here and will negatively impact on the Project NPV.

Table 21-2: Comparison to Historical Mining Operating Costs (2017)

Operating Costs	Units	Jan-Jul Actual	Aug-Dec Forecast
Total Material Moved Expit	(kt)	9,044	7,950
Total Operating Costs	**(USDk)**	**21,409**	**17,203**
Loading	(USDk)	3,122	927
Hauling	(USDk)		1,736
Drilling	(USDk)	1,225	712
Ancillary	(USDk)	1,130	2,539
Fuel	(USDk)	4,034	2,979
Labour	(USDk)	4,634	3,443
Grade Control	(USDk)	818	159
Explosives	(USDk)	4,165	3,143
Miscellaneous	(USDk)	2,280	1,564
Unit Operating Costs	**(USD/t)**	**2.37**	**2.16**
Loading	(USD/t)	0.35	0.12
Hauling	(USD/t)		0.22
Drilling	(USD/t)	0.14	0.09
Ancillary	(USD/t)	0.12	0.32
Fuel	(USD/t)	0.45	0.37
Labour	(USD/t)	0.51	0.43
Grade Control	(USD/t)	0.09	0.02
Explosives	(USD/t)	0.46	0.40
Miscellaneous	(USD/t)	0.25	0.20

21.2.4 Processing Costs

This section provides a description of the operating costs for the New Liberty processing facility. The following are examples of items excluded from this estimate and are included elsewhere in the financial model:

- All, import duties and/or any other statutory taxation, levies and/ or national and local institutions.
- Contributions to social programmes.
- 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.

Overall Plant Operating Cost

The overall plant operating cost estimate is shown in Table 21-3 below. For the purposes of the economic analysis presented in Chapter 22, a forecast assumption of USD20/t processed has been assumed for the total processing operating cost which is lower than has been achieved historically. This is due to the reduction in fixed costs and bringing procurement activities in house which were previously outsourced at a margin as well as increasing the plant throughput which results in a reduction in unit costs.

If these reductions are not realised the costs will be higher than presented here and will negatively impact on the Project NPV.

Table 21-3: Processing – Total Operating costs

Processing Cost	Actual Q1-Q3 2017 unit cost USD /t	Forecast average annual cost USD '000	Forecast unit cost USD/ t
Reagents	3.04	4,340	2.61
Labour	1.34	1,911	1.15
Grinding balls	0.97	1,378	0.83
Mill liners	0.06	81	0.05
Crusher liners	0.03	45	0.03
Power costs (incl verti-mill)	9.73	13,887	8.34
Elution costs	0.84	1,205	0.72
Laboratory / Assay costs	0.63	902	0.54
Freight costs	0.68	974	0.59
Maintenance	0.28	398	0.24
Milling screens & cyclones	0.01	9	0.01
Detox & gold room	4.53	6,473	3.89
TSF operating	0.40	565	0.34
Aachen & oxygen	0.12	173	0.10
Lubes and maintenance	0.01	11	0.01
Various	3.65	1,130	0.67
Total Estimated Processing Cost	**26.32**	**33,483**	**20.11**

21.2.5 General and Administration Operating Costs

General and Administrative overhead (G&A) costs required to directly support the operation have been estimated as part of the budgeting process by BMMC and based on actual data as adjusted for anticipated changes going forward. These costs include, amongst others; overhead labour, accommodation and messing, camp management and security, social and environmental programmes and studies, ancillary power costs, professional services to support the mine operations and general administrative overheads. For the purposes of the economic analysis presented in Chapter 22, a forecast assumption of USD7/t processed has been assumed for the total G&A operating cost. This is lower than the historic costs achieved predominantly due to the planned increase in the plant throughput which results in a reduction in unit costs due to the largely fixed nature of G&A costs.

If these reductions are not realised the costs will be higher than presented here and will negatively impact on the Project NPV.

Table 21-4: G&A – Total Operating costs

G&A Cost	Actual Q3 2017 unit cost USD /t	Forecast average annual cost USD '000	Forecast unit cost USD / t
Labour	2.89	3,389	2.04
Travel & accommodation	1.15	1,350	0.81
Camp, catering, power	0.83	972	0.58
Transport (incl. fuel)	0.31	364	0.22
SHEQ & clinic	0.96	1,125	0.68
Other – site	3.83	4,489	2.70
Total Estimated G&A Cost	**9.98**	**11,688**	**7.02**

21.3 Capital Cost Estimate

A breakdown of the sustaining capital cost required for the remainder of the Life of Mine is summarised in Table 21-5. Sustaining capital includes the mine closure costs of USD10million. The mine closure costs cover environmental aspects at the mine and process plant sites.

Certain capital equipment is to be procured through equipment finance facilities that defer the payment terms across the life of mine. As such the capital cost outlay is not incurred in the period that the equipment is purchased and the scheduling of these payments can be seen in Table 22-1 in the following section.

Table 21-5: Deferred Capital Cost Estimates

Capital, Sustaining Capital and Closure Cost	USD M
Mining capital and sustaining capital	26.9
Other capital and sustaining capital	15.6
Closure costs	10.0
Total capital, sustaining capital and closure costs	**52.5**

22 ECONOMIC ANALYSIS

22.1 Economic Model

BMMC 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 and current Project status, as commented on in the previous sections of this report.

22.1.1 General Assumptions

The financial model reflects post-finance and post-tax cashflows, allows for working capital and is based on the current LoM plan as discussed in this report. The financial model is based on the following key assumptions:

- Currency base is the USD in real Q3 2017 terms.
- Start date of 1 October 2017.
- 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. Based on the current assumptions, the Republic of Liberia's retention of a free of charge equity interest in BMMC's operations equal to 10% of its authorised issued and outstanding share capital without dilution (i.e. a 10% "carried interest") has no impact on the financial model.
- The financial model does not include all initial capital costs to construct the project as these are treated as sunk costs and nor does it include any revenues and operating costs incurred prior to the start of financial model start date noted above.
- Cashflow forecasts are calculated on a quarterly basis.

All mining and processing tonnages and grades are shown in Table 22-2. A recovery algorithm has been incorporated to the model which varies the recovery according to a varying feed grade. The LoM average recovery is 92%. Operating and capital cost assumptions are as summarised above in Chapter 21 are shown in Table 22-2. In summary, over the LoM the average unit cost per tonne of total material mined is forecast to be USD1.85/t mined, while the average LoM processing cost is forecast to be USD20/t processed and the average LoM G&A cost is forecast to be USD7/t processed. An assumption of USD3.5/oz sold is included to cover freight and refining costs.

Certain future capital costs (for mining equipment and replacement gensets) are intended to be purchased through finance arrangements, thereby deferring the payments over the LoM. The scheduling of the capital cost payments are shown below in Chapter Table 22-2.

22.1.2 Project Economics

A net present value (NPV) has been calculated for the expected cash flows from the commencement date of the model (1 October 2017) through the application of Discounted Cash Flow (DCF) techniques to post-financing and post-tax cash flows derived from the inputs and assumptions presented in this and previous sections of the report. All figures are presented in Q3 2017 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. Based on the current assumptions the Republic of Liberia's retention of a free of charge equity interest in BMMC's operations equal to 10% of its authorised issued and outstanding share capital without dilution (i.e. a 10% "carried interest") has no impact on the financial model. The model assumes a corporation tax rate of 25% which is taken from the restated and amended Mineral Development Agreement, however, it is noted that no corporation tax becomes due over the LoM using the assumptions presented herein for the base case.

Repayment of debt principal and associated finance costs are included in the model which total approximately USD142M over the LoM (based on current LIBOR rates).

A summary of cash flow modelling is presented below in Table 22-1 with the pre-finance and post-tax cash flow model shown in Table 22-2. In summary, this indicates a post-tax and post-financing NPV at a 5% discount rate of USD179M.

Table 22-1: Cash Flow Modelling Summary

Description	Units	Project Totals/Averages
Recovered gold	koz	642
Mill processing life	Years	4.5
Net smelter revenue (after royalty)	USDM	808
Operating costs (including working capital)	USDM	(415)
Net operating cash flow	USDM	393
Capital, sustaining capital and closure costs	USDM	(53)
Net post-tax cash flow	USDM	340
Debt financing cash flows	USDM	(142)
Post-tax, post-financing cash flow	USDM	198
Post-tax, post-financing NPV (5%)[1]	USDM	179
Operating cash cost[1]	USD/oz	659
All-in sustaining cash cost[1]	USD/oz	749

1 Net present value ("NPV"), operating cash costs and all-in sustaining costs ("AISC") per ounce of gold produced are non-IFRS financial measures. These non-IFRS financial measures do not have any standardised meaning. Accordingly, these financial measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with International Financial Reporting Standards ("IFRS"). Operating cash costs and all-in-sustaining cash costs are a common financial performance measure in the mining industry but have no standard definition under IFRS. Operating cash costs are reflective of the cost of production and include a net-smelter royalty of 3%. AISC include operating cash costs, corporate costs, sustaining capital expenditure, sustaining exploration expenditure and capitalised stripping costs.

Table 22-2: Project Cash Flows

Description	Unit	Total	Q4 2017	2018	2019	2020	2021	Q1 2022
Ore mined	kt	6,941	433	1,585	1,883	1,496	1,447	98
Waste mined	kt	114,846	4,679	45,276	34,930	20,554	9,301	106
Strip ratio	x	16.5	10.8	28.6	18.6	13.7	6.4	1.1
Ore processed	kt	7,142	374	1,680	1,664	1,684	1,631	109
Grade	g/t	3.0	3.0	2.9	2.6	3.2	3.3	4.3
Gold contained	koz	697	36	156	141	175	174	15
Gold Recovered	koz	642	33	144	129	162	161	14
Gross revenues	USDM	835.0	43.5	186.9	167.4	210.0	209.3	18.0
Royalty and other sales costs	USDM	(27.2)	(1.4)	(6.1)	(5.5)	(6.8)	(6.8)	(0.6)
Net Revenues	USDM	807.8	42.1	180.8	161.9	203.2	202.4	17.4
Mining costs	USDM	(225.1)	(11.1)	(74.7)	(65.3)	(43.8)	(28.1)	(2.1)
Processing costs	USDM	(142.8)	(7.5)	(33.6)	(33.3)	(33.7)	(32.6)	(2.2)
G&A and other expenses	USDM	(50.0)	(2.6)	(11.8)	(11.6)	(11.8)	(11.4)	(0.8)
Movement in provisions & working capital	USDM	3.1	0.4	5.5	(3.2)	(1.3)	1.8	0.0
Operating Cash Flows	USDM	393.0	21.3	66.2	48.5	112.6	132.0	12.3
Capital, sustaining capital and closure costs	USDM	(52.5)	(1.0)	(13.4)	(10.0)	(9.5)	(7.9)	(10.6)
Taxation	USDM	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Net post-tax cash flow	**USDM**	**340.5**	**20.3**	**52.8**	**38.5**	**103.0**	**124.1**	**1.7**

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 gold royalties of USD 25 million over the LoM of the Project. No corporation tax is payable over the LoM under the current set of assumptions.

22.1.4 Project Sensitivities

A sensitivity analysis was performed on the post-financing, post-tax cash flows by varying key variables (gold price and operating costs) to +/- 10% of the base case at a range of discount rates (0% to 10%). These results are summarised in Table 22-3.

Table 22-3: Project Sensitivities

Gold Price (USD / oz)	Post-finance Post-tax NPV (USDM)		
	0%	**5%**	**10%**
1,200	136	123	112
1,300	198	179	162
1,400	246	223	202
Opex change (%)	**Post-finance Post-tax NPV (USDM)**		
	0%	**5%**	**10%**
-10%	230	208	189
0%	198	179	162
+10%	153	138	125

22.1.5 SRK Comments

SRK has verified that the financial model inputs reflect accurately the LoM plan and financial costs reported by the Company.

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 for the current stage of the Project and based on historical performance to date.

Notwithstanding this, SRK notes that compared with historical physical performance achieved to date, BMMC is forecasting increases in both mine and plant production on an annual basis. While these increases are achievable in theory with the equipment planned, if these increases are not achieved in practice then the unit operating costs will be higher than currently assumed and the resulting Project NPV would be lower than presented herein.

Compared with historical operating costs achieved to date, BMMC is forecasting savings to be made going forward and a corresponding reduction in unit costs. These cost savings are at an early stage of implementation and require confirmation in practice. SRK is confident that if the cost savings are made then the Project NPV presented in this report will be realistic, however, if the changes are not realised then the NPV could be considerably lower.

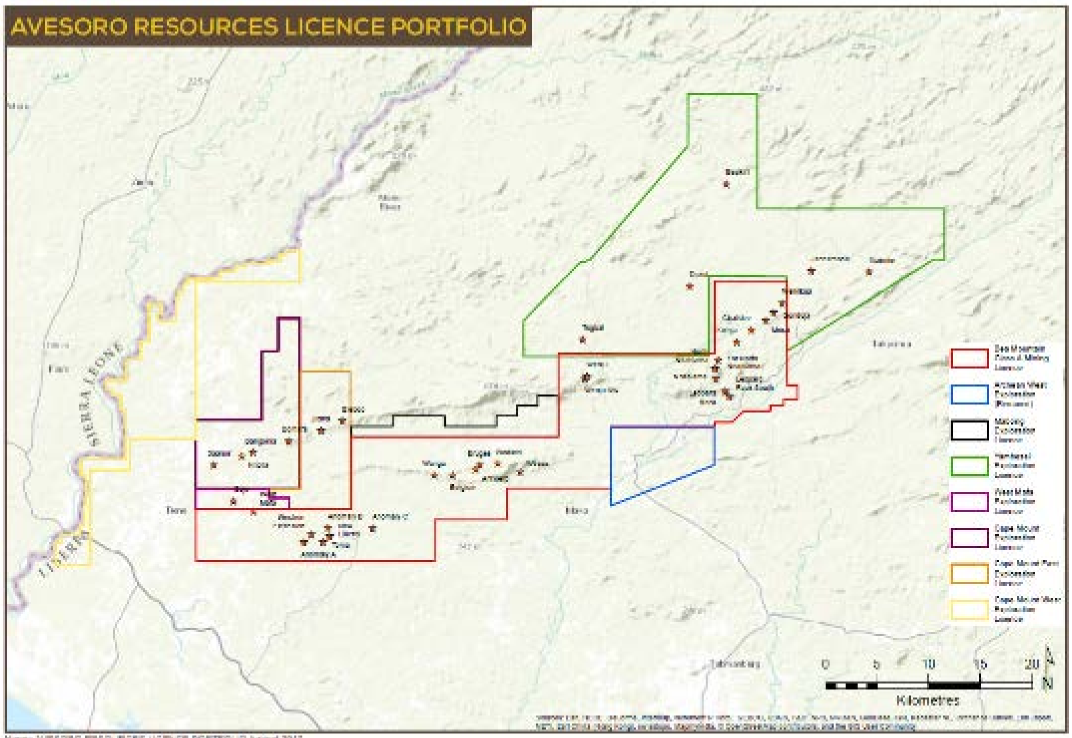
23 ADJACENT PROPERTIES

23.1 Overview

The most recent Mineral Land Holding map update was published in February 2017 by the Ministry of Lands Mines and Energy. In addition to the Bea-MDA, BMMC acquired an exploration license known as Archaen Gold (89 km^2) from Archaen Gold Ltd, as announced on 21 September 2011. After incorporating 21 km^2 of the license into the Bea-MDA license (reported 11 May 2015), the Archaen Gold license was subsequently removed from BMMC's holdings.

Additionally, as reported on 19 November 2013, BMMC was been granted four new exploration licenses, contiguous to the Bea Mountain Mining license by the Ministry of Lands, Mines and Energy. The four exploration licenses 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). Following the acquisition of three exploration licenses from Sarama Investments Limited on 6 January 2016, the Yambesei and Archaean West licenses were reduced to 473 km^2 and 55.7 km^2 respectively.

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 (Figure 23-1).



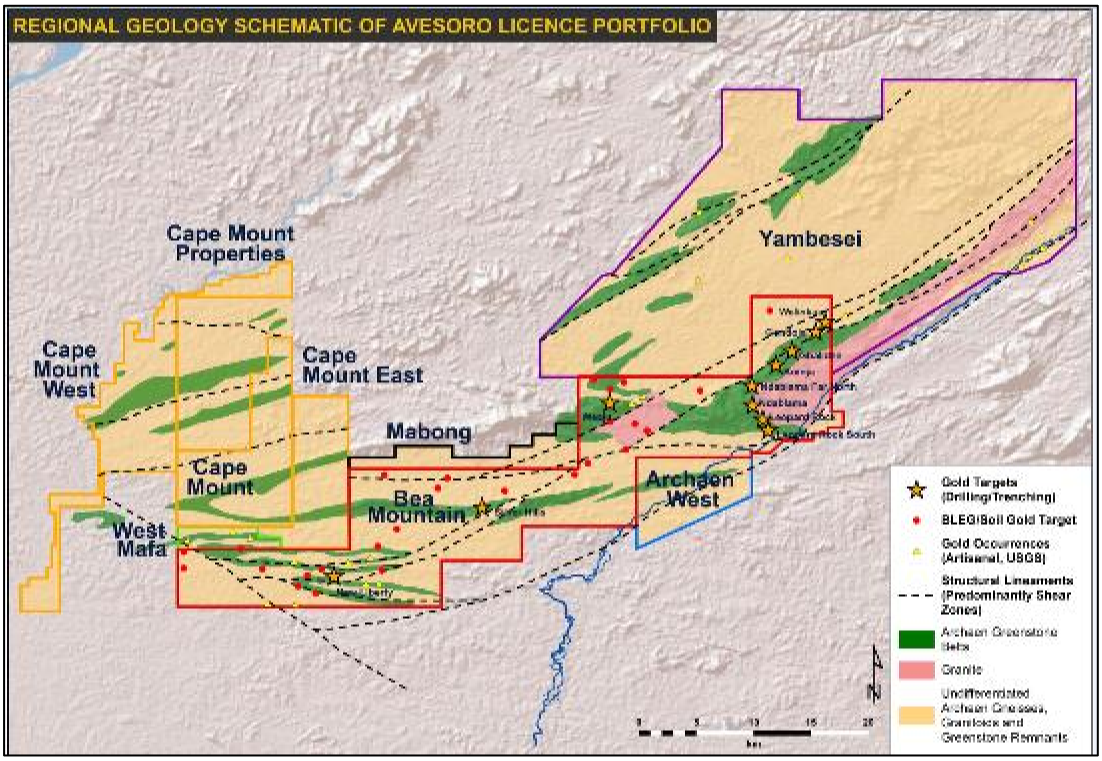
Source: *BMMC, 2017*

Figure 23-1: Properties adjacent to the Bea-MDA Mountain mining license

The New Liberty Project is located in the southern portion of the Bea-MDA license 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 "Ndablama and Weaju Gold Projects, Bea Mountain Mining License Northern Block, Liberia, West Africa", and available on SEDAR.

BMMC's license portfolio hosts multiple greenstone belts and associated shear structures which to date have been the principal hosts to the gold mineralisation systems discovered in Liberia. At the time of this report, 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 mineralisation 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 349 samples collected to date, including 72 in the Archaean West license. Soil sampling programmes were also undertaken in the Yambesei license, including the extension of the Yambesei structural corridor to check possible extension of the Gondoja gold corridor to the east. with 615 soil samples collected. Some 3,000 soil samples have been taken from the Yambesei license to date, with 327 soil samples from the Mabong license.



Source: *BMMC, 2017*

Figure 23-2: Geological interpretation of BMMC mining license package

24 OTHER RELEVANT DATA AND INFORMATION

No other information is considered necessary.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resources and Reserves

The New Liberty deposit is now an open pit mining operation which is at an advanced stage of drilling and geological understanding. Recent grade control infill drilling has added further geological confidence to the local scale geometry of the mineralisation and grade distributions close to surface.

The geological interpretation used to generate the Mineral Resource presented herein is generally considered to be robust albeit that there are areas of lower geological confidence which may be subject to further revision in the future. SRK considers the exploration data accumulated by the Company is generally reliable and suitable for the purpose of this Mineral Resource estimate.

Notwithstanding this, the economic valuation is based on the Mineral Reserves which fall within the current designed pit which the Company is planning to exploit in the Life of Mine plan presented. SRK believes that there is potential for further exploration to extend the Mineral Reserve by infill drilling the Inferred Mineral Resource within the current pit design, through infill drilling of the Inferred Mineral Resource lying beneath this and through drilling extensions to the Mineral Resource at depth.

Notably:

- Some 0.1Mt of Inferred Mineral Resource with a mean grade of 2.7g/t Au have been estimated to be present within the designed pit. This has been treated as “waste” in the valuation presented in Section 22.
- A further 3.5Mt of Inferred Mineral Resources with a mean grade of 2.8g/t Au have been delineated below the current design pit but within the open pit reporting limit using the USD1,500/oz optimised ‘MII’ (Measured, Indicated and Inferred) pit shell.
- Some 2.8Mt of Inferred Resources with a mean grade of 3.3g/t Au has been reported as underground resources and has the potential to be exploited by underground mining.
- In addition to the above, plunging high grade shoots delineated at Kinjor South, Marvoe and Larjor remain open at depth and so there is potential for increasing the underground Mineral Resource in these areas through further drilling.

The Company agrees with the above comments and has planned an initial 14,000m drilling programme targeted to infill the 3.5Mt of the Inferred Mineral Resource lying below the current designed pit and within the USD1,500/oz optimised MII pit shell as noted above SRK has reviewed this drilling plan which has been costed at USD1.5M and agrees that this exploration is justified and if successful has potential to extend the envisaged mine life. Of this Inferred Resource, some 3.0Mt of with a mean grade of 2.8g/t has been delineated below the current design pit but within the USD1,300/oz optimised MII pit shell, which demonstrates that a significant proportion of the total Inferred Mineral Resource below the current design pit would have potential to extend the envisaged mine life through extensions of the current design pit, subject to this being upgraded to the Indicated classification.

The Company then intends to undertake a second drill programme which will test for extension of the deeper underground potential of high grade shoots below the USD1,500/oz optimised pit shell.

25.2 Mining Plan

SRK has undertaken a life of mine plan including the development of a mining model to estimate ore loss and dilution, pit optimisation, mine design, mine schedule, equipment and labour requirements and cost estimation. Based on the results of the study, SRK can conclude the following:

- The updated mine designs based on the USD1,300/oz optimised shell result in 7.1 Mt of RoM at 3.08 g/t Au with 117.5 Mt of waste at a cut-off of 0.85 g/t Au.
- Average ore loss and dilution values are 3.3% and 13.5%, respectively within the pit design. Significant improvements are expected with the new grade control programme to reduce current levels of ore loss and dilution. Any increases in loss and dilution will impact the tonnage and grade in the mine plan.
- The mine schedule produces 1.68 ktpa of mill feed, totalling 7.4 Mt at an average grade of 3.03 g/t Au. The average strip ratio is 16.5 with 117.5 Mt of waste. Total material movement will average 3,905 kt/month in 2018 (totalling 46.9 Mt).
- The mine schedule is aggressive with up to 8 benches mined per year. Mine production quantities will need to triple by January 2018 and quadruple by March 2018 from current production levels. Improvements are required on the management and operational level in order to achieve these results. The mine plan and plant feed will be impacted if these improvements are not realised.
- There are significant periods when there is insufficient RoM Fresh material available on the stockpile to mitigate any shortfalls. Should any shortfalls arise, additional material will can only be sourced from the RoM Oxide stockpile which has lower grades and recovery.
- One 12 m^3 backhoe and up to six 6 m^3 backhoes will continue to be used with 90 t haul trucks supported by 40 t ADTs. Up to 16 90 t haul trucks will need to be leased from February 2018 to support the mine plan. Significant improvements in availability and productivity of the excavators and trucks is required to meet the mine plan.
- A maximum of 892 personnel are required at peak material movement (2018), with 714 in mine operations, 157 in mine maintenance and 21 in technical services. The personnel requirements in 2018 are significantly higher than current levels (approximately 529). BMMC will need to recruit sufficient qualified personnel in order to meet the mine plan.

25.3 Mineral Processing

A number of design issues in the original plant have been addressed by the new owners and modifications have been implemented or planned that will improve the overall operating time of the plant and improve the metallurgical performance of the CIL circuit to the levels expected in the feasibility study.

The modified circuit and operating parameters should allow plant feed rates up to 200 tph to be achieved.

In addition, the reduced crushing circuit product size of nominally 80% passing 12 mm, together with the improved power utilisation in the grinding circuit and the reintroduction of the Vertimill, should produce an overall CIL feed size of 80% passing 50 µm at the increased throughput whilst achieving acceptable gold extraction in the originally installed CIL tankage.

An increased number of operators in the cyanide detoxification and arsenic circuit and targeted performance management should result in more consistent operation and achieve acceptable tailings discharge levels of CN_{WAD} and soluble arsenic into the TSF. Further optimisation work will be required.

25.4 Infrastructure

The construction of the Project infrastructure is now essentially complete and the infrastructure is adequate to support the ongoing operations at the Project.

The diversion dams and cutting for the Marvoe Creek Diversion are complete, functioning as designed and are appropriately diverting the surface water from the watercourse away from the open pit and site infrastructure.

25.5 Tailings Storage Facility

The current Tailings Storage Facility (TSF) arrangement has been in operation since July 2015. As of the beginning of August 2017, the TSF has been operated as a self-raising facility, in which deposited tailings material will be reworked to form the main embankment itself.

The configuration of TSF was significantly altered during 2016. This was required due to periodic uncontrolled release of supernatant to the environment which did meet compliance limits (between December 2015 and June 2016). A temporary TSF configuration was constructed to ensure that discharge of excess supernatant to the environment met acceptable discharge limits. This involved segregation of the TSF into a series of compartments or cells, designed to promote a tortuous flow path for supernatant and dilution with the cleaner water in the upper reaches of the TSF, before discharge via the penstock to environment. This, combined with plant modifications, has ensured that discharge water quality has improved and is reported to now be within acceptable limits.

NewFields was commissioned by BMMC during October 2016 to prepare an alternative TSF design, which would allow safe storage of water on the facility and controlled release of supernatant to the environment. This new design involves conversion of the TSF to a water retaining, downstream raised facility. In addition, a water retaining dam is to be constructed to the east of the TSF, which will divert inflows of fresh water from the upstream catchment during storm events. This fresh water will be routed via the existing penstock arrangement and safely discharged downstream.

Overall, SRK considers the design of proposed TSF modifications to be a workable solution, assuming that the critical structures can be constructed timeously with the tailings rate of rise in the current facility.

25.6 Environmental and Social Management

BMMC has the primary agreements and approvals required to operate, which a Mineral Development Agreement (MDA), a mining licence, an environmental permit and a discharge permit. BMMC also has a number of secondary approvals and officially approved environment and social management plans. There are numerous compliance obligations in the approval documents approvals and management plans. BMMC recognises that it needs to review these and agree revisions to unrealistic or poorly worded obligations.

There are some elements of an environmental, social, health and safety (ESHS) management system in place at NLGM, but the management system is not fully fledged. A more systematic approach to ESHS management has been taken in response to the cyanide incident at the mine in late 2015/ early 2016. Lessons learned and actions taken in response to this incident should be transferred to the ESHS management system as a whole.

BMMC manages the mineral processing operation, the tailings detoxification plant and the TSF operations such that cyanide and arsenic compliance criteria in the watercourses downstream of the mine are not exceeded.

After the mineral processing operation was first commissioned in 2015, there was a suite of challenges that resulted in failure to meet cyanide compliance criteria downstream of the mine and fish deaths in the downstream watercourses were observed. The problems have been addressed and impact studies by independent specialists contracted by the Company have confirmed that the river ecosystem has largely recovered and that people living downstream of the mine have not been adversely affected.

The mine’s monitoring data demonstrates compliance with relevant cyanide and arsenic criteria at the environmental compliance points from May 2016 to July 2017. There are internal check points for cyanide and arsenic in water on the mine. These include the tailings prior to discharge to the TSF, the penstock on the TSF and the point of release of supernatant from the TSF to the engineered wetland below the TSF. Data from the internal check points suggest that the cyanide detoxification and the arsenic removal processes interfere with each other. When the cyanide detoxification performance is highly effective, the performance of the arsenic removal process is not optimal. This does not result in non-conformance with environmental compliance criteria but can result in internal check point values being exceeded. BMMC is investigating this issue with the aim of optimising the performance of both detoxification processes.

BMMC has an extensive water monitoring programme, but interpretation of the data for parameters other than cyanide and arsenic has been not been thorough to date.

Several pollution control measures still have to be implemented by the mine.

The mine does have a commitment to develop and implement a biodiversity offset programme in its environmental permit. Biodiversity investigations and monitoring required for this are on-going. Recent studies have confirmed that there could be critical habitat affected by the mine. A population of *Isomacrolobium (Anthonotha) explicans* could have been affected by waste rock dump development and it is noted that the critically endangered African slender-snouted crocodile (*Mecistops cataphractus*) is likely to occur in the rivers downstream of the TSF.

Full execution of the relocation action plan (RAP) was delayed by the Ebola epidemic (2014 to mid-2015) and a period of financial instability experienced by the mine (mid-2015 to mid-2016).

The stalled resettlement of Kinjor and Larjor is addressed in a memorandum of understanding agreed with the affected households. BMMC has committed to fully complete the resettlement by Q4 2018, with interim commitments to complete 200 household units by end of 2017 and to implementation of a rolling plan of occupation commencing in Q1 2018.

The mine's stakeholder engagement needs improvement. Community engagement and grievance management has up until recently centred on a resettlement working group. The approach to stakeholder and community relations is currently being restructured and managed by a recently appointed Community Relations Manager and a revised stakeholder engagement plan will be finalised in November 2017.

BMMC has set up a number of cooperatives and community based initiatives as alternative livelihood activities. Reportedly BMMC is in the process of developing a comprehensive livelihood restoration plan that will be operational by the end of 2017. The intention is for this plan to include a range of additional initiatives including start-up of women's rotating credit schemes, and extension of modular brick making following the RAP house unit construction to a cooperative.

A closure plan was produced for the mine in 2013. The closure cost estimate based on the 2013 closure plan is USD10.0 million. SRK recommends that the closure plan and cost estimate are updated.

25.7 Economic Analysis

Compared with historical physical performance achieved to date, BMMC is forecasting increases in both mine and plant production on an annual basis. While these increases are achievable in theory with the equipment planned, if these increases are not achieved in practice then the unit operating costs will be higher than currently assumed and the resulting Project NPV would be lower than presented herein.

Compared with historical operating costs achieved to date, BMMC is forecasting savings to be made going forward and a corresponding reduction in unit costs. These cost savings are at an early stage of implementation and require confirmation in practice. SRK is confident that if the cost savings are made then the Project NPV presented in this report will be realistic, however, if the changes are not realised then the NPV could be considerably lower.

26 RECOMMENDATIONS

26.1 Drilling, Sampling and Mineral Resource

SRK considers there to be good potential to improve confidence in the reported Mineral Resource at New Liberty with additional drilling, in-pit geological investigation and further modelling work.

In relation to exploration drilling and sampling, SRK has recommended the following:

- Targeted infill drilling to add geological confidence to convert the Inferred Resources to Indicated and convert more of the Indicated to Measured Resources;
- Additional exploration drilling at depth, specifically around the down-dip continuation of the grade shoots at Larjor, Kinjor South and Marvoe, where there is potential for increasing the tonnage in the reported underground Mineral Resource; and
- Additional exploration within the surrounding permit area where there is good potential to find further gold mineralisation.

In relation to grade control drilling, whilst SRK has a high overall confidence in the block tonnage and grade estimates in the geologically well-constrained, well-drilled parts of the mineralisation domains, sufficient for Measured Mineral Resources, SRK has recommended further investigations into the CRM swaps and the reduction in analytical precision noted since the start of the ALS NLGM laboratory and the re-submission of 5-10% of sample pulps analysed at ALS NLGM with the Geostats CRMs to an umpire laboratory to further verify analytical performance;

In relation to geological fieldwork, SRK recommends in-pit mapping as part of a structural study to help improve understanding of the geological controls on (and 3D structural framework for) the higher grade mineralised zones at Marvoe, Kinjor North and Larjor. This exercise should be completed with a targeted structural re-assessment of the drillcore, with a focus on the orientation of the zones of increased shearing which are currently interpreted to host the higher grade zones.

It will be important to continue to monitor production grades going forward and SRK recommends that detailed mine to mill reconciliation analyses are undertaken on a regular basis (monthly) to assess whether any modifications need to be made to the underlying block models and productions.

26.2 Mining

BMMC has indicated that the 8.5 m berms incorporated in the designs are sufficient to contain most failures, however, SRK recommends that the geotechnical assessment is updated based on the most recent designs and the current operations.

SRK further recommends that a hydrogeological testing programme is undertaken as a slope depressurisation programme may need to be undertaken prior to the pit being developed to the final faces.

The grade control programme currently in use on site is still fairly new and will require constant reconciliation to validate the ore loss and dilution estimates. Any significant deviations from those estimates should be investigated and the life of mine plan should be updated accordingly.

The availabilities and productivities of the excavators and trucks should continue to be monitored to ensure the increases expected are realised. The mine plan will be significantly impacted should these improvements not be achieved.

BMMC should investigate the use of larger shovels in the waste in order to limit the number of loading units in each mining stage to ensure productivity levels can be achieved.

Significant cost savings are expected from the historical 2017 costs compared to the forecast. Regular monitoring of the mining costs in comparison to the forecast is important to ensure the improvements are realised.

26.3 Mineral Processing

Further optimisation work will be required on the cyanide detoxification and arsenic circuit to ensure this consistently achieves the targeted performance and tailings discharge levels are maintained at acceptable levels.

SRK recommends that the plant throughput and recovery performance be monitored going forward and modify the forecast assumptions accordingly if material variances to plan are observed on a frequent basis. Specifically, NLGM should continue to monitor the effect of the plant modifications at the increased plant throughput in terms of gravity gold recovery, CIL solids 80% passing feed size, CIL feed grade, CIL % solids, CIL residence time, CIL tailings grade and CIL tailings soluble losses.

26.4 Infrastructure

SRK recommends that appropriate plans are put in place for the anticipated changeover from Jozi Power to in-house arrangements for power supply, to minimise any downtime and ensure a smooth changeover and continued power supply to the operation.

26.5 Tailings Storage Facility

SRK notes that the forecast tailings production rate is some 120kt up to November 2017, increasing to 140ktpm from December 2017. NewFields capacity calculations are based on an average deposition rate of 110kpta. SRK has therefore recommended that the volumetric checks and capacity calculations are updated to provide an accurate estimate of the anticipated storage capacity and to highlight if any shortfall exists going forward.

Whilst BMMC appear to be taking reasonable measures to maximise the remaining capacity of the TSF (by extending tailings distribution pipeline and spigots around the southern flank), the tailings deposition strategy and volumetric should be updated to minimise the risk of plant downtime as the proposed embankment raises are constructed. Should the shortfall in overall capacity be confirmed and alternative deposition strategy may have to be implemented (for a temporary period).

26.6 Marvoe Creek Diversion

SRK has recommended that Embankment 1 is shored up with waste rock as a precautionary measure, to ensure that the integrity of the embankment in maintained.

At Embankment 2, no seepage was noted, however, it is noted that the pressure relief valve which extends through the embankment is exposed and could be prone to damage as the waste rock dump (WRD) expands across this zone. SRK recommends that a layer of sacrificial fill (laterite soil) is placed around the pressure relief valve of Embankment 2 and that care is taken during placement of waste rock on the downstream site of Embankment 2.

26.7 Environmental and Social Management

SRK's recommendations with regards to Environmental and Social Management to the Company are that it should:

- Review compliance obligations, beginning with those that are legally binding – including conditions in the MDA, environmental permit and discharge permit and officially approved management plans. Propose changes to obligations that are unrealistic or poorly worded and agree these with regulatory authorities.
- Continue with the establishment of the ESHS management system and ensure there is integration of the environmental, social and safety elements of the system.
- Continue with review and monitoring aimed at optimising the performance of cyanide detoxification and the arsenic removal processes.
- Process and review all water quality monitoring data and checks that all parameters for compliance with relevant criteria.
- Implement the outstanding pollution control measures.
- Complete, reach agreement on and implement a biodiversity offset plan as required by the environmental permit.
- Complete the resettlement process and ensure all commitments made in the recent memorandum of understanding agreement with affected households are met.
- Complete and implement the livelihood restoration plan in consultation with affected households.
- Continue with improvements to the company's approach to stakeholder and community relations.
- Update the closure plan and revise the closure cost estimate.

26.8 Economic Analysis

SRK recommends the actual operating costs are continually monitored and compared against budget forecasts and the LoM plan and economic analysis be updated on a regular basis should significant variances be noted.

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CERTIFICATES AND LETTERS OF CONSENT

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To accompany the technical report entitled "New Liberty Gold Mine, Bea Mountain Mining Licence Southern Block, Liberia, West Africa" (the "Technical Report") for Avesoro Resources Inc. (Avesoro) with the effective date 01 November 2017.

I, Dr Mike Armitage, BSc, MIMMM, CEng, 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 a Chartered Engineer and a Fellow of the Geological Society of London and a Chartered Geologist.
4. I visited the project area between 20-23 November 2012, 7-10 July 2015 and 8-11 November 2016.
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, 18 to 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. My involvement in the subject property has been in a review role other than my role in the production of the Mineral Resource and Reserve estimates presented in this report.

9. I have read National Instrument 43-101 and confirm that the portions of this Technical Report that I am responsible for have been prepared in compliance therewith.

10. That, as of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions of this Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dr Mike Armitage, *FGS, CGeol, MIMMM, CEng*
Group Chairman & Corporate Consultant
(Mining Geology), SRK (UK) Ltd.
Cardiff, UK, 21 November 2017

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Project number: UK4936,
Cardiff, Wales, 21 November 2017

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, 18 to 19 and 21 to 27 of the technical report entitled “New Liberty Gold Mine, Bea Mountain Mining Licence Southern Block, Liberia, West Africa” (the “Technical Report”) and dated 21 November 2017 for Avesoro Resources Inc. (Avesoro).

I consent to the public filing of the Technical Report and to the use of extracts from, or a summary of, the Technical Report in the news release of the Company dated 11 October 2017 (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 21 day of November 2017.

Dr Mike Armitage*, BSc, MIMMM, FGS, CEng*
Group Chairman & Corporate Consultant
(Mining Geology), SRK (UK) Ltd.

srk consulting

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CERTIFICATE OF QUALIFIED PERSON

Dr David Pattinson
SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom

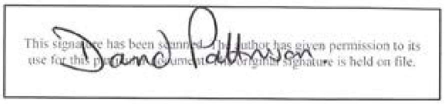
Tel: + 44 (0) 2920 348 150
E-mail: dpattinson@srk.co.uk

To accompany the technical report entitled "New Liberty Gold Mine, Bea Mountain Mining Licence Southern Block, Liberia, West Africa" (the "Technical Report") for Avesoro Resources Inc. (Avesoro) with the effective date 01 November 2017.

I, Dr David Pattinson, BSc, MIMMM, CEng, do hereby certify that.

1. I am a Corporate Consultant (Metallurgy &Mineral Processing) 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 Birmingham University with a BSc. Honours Degree in Mineral Engineering awarded in 1978 and also have a PhD from Birmingham University awarded in 1982. I have practised my profession continuously since 1981.
3. I am a Professional Member of the Institute of Materials, Minerals and Mining and I am a Chartered Engineer.
4. I personally visited the project area on a number of occasions between 7-10 July 2015, 1-5 December 2015, 2-5 February 2016, 4-11 May 2016, and 8-11 November 2016, during which time I visited the mine site and the processing plant.
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 13 and17.
7. As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101.
8. My involvement in the subject property has been in a review role as presented in this report.
9. I have read National Instrument 43-101 and confirm that the portions of this Technical Report that I am responsible for have been prepared in compliance therewith.

10. That, as of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portions of this Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.



Dr David Pattinson, BSc, *MIMMM, CEng*
Corporate Consultant
(Metallurgy & Mineral Processing), SRK (UK) Ltd.
Cardiff, UK, 21 November 2017

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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 David Pattinson, have been responsible for preparing or supervising the preparation of Sections 13 and 17 of the technical report entitled "New Liberty Gold Mine, Bea Mountain Mining Licence Southern Block, Liberia, West Africa" (the "Technical Report") and dated 21 November 2017 for Avesoro Resources Inc. (Avesoro).

I consent to the public filing of the Technical Report and to the use of extracts from, or a summary of, the Technical Report in the news release of the Company dated 11 October 2017 (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 21 day of November 2017.

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

Dr David Pattinson*, BSc, MIMMM, FGS, CEng*
Corporate Consultant
(Metallurgy & Mineral Processing), SRK (UK) Ltd.

CERTIFICATE OF QUALIFIED PERSON

Jane Joughin
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17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom

Tel: + 44 (0) 2920 348 150
E-mail: jjoughin@srk.co.uk

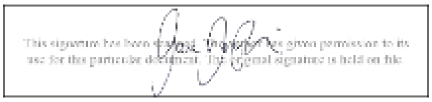
To accompany the technical report entitled "New Liberty Gold Mine, Bea Mountain Mining Licence Southern Block, Liberia, West Africa" (the "Technical Report") for Avesoro Resources Inc. (Avesoro) with the effective date 01 November 2017.

I, Jane Joughin, Pr.Sci.Nat, do hereby certify that.

1. I am a Corporate Consultant (Environmental & Social Management) with the firm of SRK Consulting (UK) Ltd ("SRK") with an office at 5th Floor, Churchill House, Churchill Way, Cardiff, UK.
2. I graduated with a BSc Honours degree (Life Sciences) in 1986 from the University of Natal, South Africa, and with an MSc (Life Sciences) in 1989 from the University of the Witwatersrand, South Africa.
3. I have practiced my profession in the field of environmental and social management in the mining industry continuously since 1991.
4. I am a Registered Professional Environmental Scientist, registered with the South African Council of Natural Scientific Professions (Registration Number: 400057/94).
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 visited the New Liberty Mine site on 7 to 10 July 2015, 19 to 23 April 2016 and 8 to 11 November 2016.
7. I am responsible for Section 20 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. My involvement in the subject property has been in a review role as presented in this report.

10. I have read National Instrument 43-101 and confirm that the portion of this Technical Report that I am responsible for has been prepared in compliance therewith;

11. That, as of the effective date of this Technical Report, to the best of my knowledge, information and belief, the portion of this Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.



Jane Joughin, Pr.Sci.Nat
Corporate Consultant
(Environmental & Social Management), SRK (UK) Ltd.
Cardiff, UK, 21 November 2017

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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, Jane Joughin, have been responsible for preparing or supervising the preparation of Section 20 of the technical report entitled “New Liberty Gold Mine, Bea Mountain Mining Licence Southern Block, Liberia, West Africa” (the “Technical Report”) and dated 21 November 2017 for Avesoro Resources Inc. (Avesoro).

I consent to the public filing of the Technical Report and to the use of extracts from, or a summary of, the Technical Report in the news release of the Company dated 11 October 2017 (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 section of the Technical Report for which I am responsible.

Dated this 21 day of November 2017.

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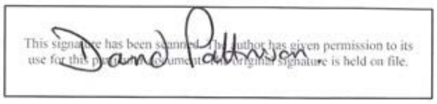
Jane Joughin, *Pr.Sci.Nat*

Corporate Consultant
(Environmental & Social Management), SRK (UK) Ltd.

SIGNATURE PAGE

Report Effective Date: 01 November 2017

Dr Mike Armitage, *BSc, MIMMM, FGS, CEng, CGeol*
Corporate Consultant & Chairman
(Mining Geology),
SRK Consulting (UK) Limited
21 November 2017



Dr David Pattinson, BSc, MIMMM, FGS, CEng
Corporate Consultant
(Metallurgy & Mineral Processing),
SRK Consulting (UK) Limited
21 November 2017

Jane Joughin, Pr.Sci.Nat
Corporate Consultant
(Environmental & Social Management),
SRK Consulting (UK) Limited
21 November 2017