AMC Consultants (UK) Limited
Registered in England and Wales
Company No 3688365

Level 7, Nicholsons House
Nicholsons Walk, Maidenhead
Berkshire SL6 1LD
UNITED KINGDOM
T +44 1628 778 256
F +44 1628 638 956
E amcmaidenhead@amcconsultants.com

NEW LIBERTY GOLD PROJECT, LIBERIA, WEST AFRICA

UPDATED TECHNICAL REPORT

AUREUS MINING INC.

Prepared by

Christopher G Arnold MAusIMM CP(Geo) of AMC Consultants (UK) Limited

Martin W Staples FAusIMM of AMC Consultants (UK) Limited

Robin M Welsh Pr Eng MSAIEE of DRA Mineral Projects

Glenn Beziudenhout FSAIMM of DRA Mineral Projects

Graham Trusler of Digby Wells Environmental (Pty) Ltd.

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AMC 413003

DATE AND SIGNATURE PAGE

This report has been prepared and signed for by the following "qualified persons" (within the meaning of National Instrument 43-101). The effective date of this report is July 3, 2013.

Signed the 3rd July, 2013

Christopher Garstang Arnold
Principal Geologist, AMC Consultants (UK) Limited

Martin W Staples
Director, AMC Consultants (UK) Limited

Graham Trusler
Chief Executive Officer, Digby Wells & Associates (Pty) Ltd.

Robin Welsh
Senior Project Manager, DRA Mineral Projects

Glenn Bezuidenhout
Process Director, DRA Mineral Projects

1 SUMMARY

Introduction

Aureus Mining Inc. (Aureus) is assessing and evaluating the technical and economic viability of the New Liberty Gold Project (the Project or the New Liberty Project) in the Republic of Liberia (Liberia), West Africa.

This technical report (Technical Report or Report) on the Project within the Bea Mountain Mineral Development Agreement (Bea-MDA) property in Liberia has been prepared by AMC Consultants (UK) Limited (AMC) of Maidenhead, UK, for Aureus. It has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgement on the CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR). This report restates the estimate of mineral resources, and reports an updated ore reserve statement and economic assessment.

This Technical Report is a summary of the Definitive Feasibility Study (DFS), which Aureus prepared with assistance from AMC, DRA Mineral Projects (Pty) Ltd (DRA), MDS Ltd (MDS), RPS Aquaterra (RPS), Golder Associates Ghana Ltd (Golder) and Digby Wells (Pty) Ltd (Digby Wells), combined with further work undertaken during the optimization phase of the project.

For the purpose of this report, the following work has been performed by external consultants:

- AMC Consultants (UK) Limited – Mineral Resource Estimate, Mineral Reserve Estimate, geotechnical and hydrogeological evaluation, mine design and mining operations.
- DRA – review of the seven phases of metallurgical test work conducted over the last two years, plus the process optimization phase test work, design of the Marvoe Creek Diversion Channel (MCDC) and Tailings Storage Facility (TSF), and the design of the process flowsheet and the gold process plant.
- Digby Wells – Environmental Impact Assessment, Relocation Action Plan (RAP) and Community Development Plan (CDP).

The Project

The Project is located within the Bea-MDA property, in Grand Cape Mount County in the north-western portion of the Republic of Liberia, approximately 90 km north-west of the capital, Monrovia. From the capital there is approximately 80 km of excellent paved road to the town of Danieltown and then a small 20 km section of laterite road to the Project site, which has recently been regraded by Aureus. Road access is all year round.

The property occupies a lowland area of tropical forest with thick undergrowth, cut by two prominent east-west ridges of resistant rock units (the Bea Mountain and Tokani ranges). Elevations at the Project area range between 40 m and 80 m above mean sea level.

The Republic of Liberia is situated on the coast along the south-west corner of West Africa, bordered by Sierra Leone, Guinea and Cote d'Ivoire. Since the end of the civil war in 2003, Liberia has experienced a period of reform and reconstruction under President Ellen Johnson Sirleaf.

Ownership

The project is owned by Bea Mountain Mining Corporation (BEA), a fully owned operating subsidiary of Aureus Mining Inc.

The Bea-MDA property covers an area of 457 km² and the agreement has an initial and renewable term of 25 years. In July 2009 BEA was granted a Class A Mining Licence for the whole area, subject to an annual licence fee of US$ 90,146. Under the terms of the Bea-MDA the Republic of Liberia is entitled to receive, free of charge, an equity interest on BEA's operations equal to 10% of its authorized and outstanding share capital without dilution (i.e. a 10% "carried interest"). There is also a 3% royalty, calculated on a production basis, payable to the Republic of Liberia in the Bea-MDA licence area.

BEA has a 100% interest in the Bea-MDA, which was originally signed with the Liberian Government in November 2001. To the best of AMC's knowledge, the area has only limited surface artisanal workings and no historical environmental issues.

Geology and Mineralization

Liberia is geologically traversed by the Man Shield, which in turn lies within the West African Craton, and hosts rocks dating from 3.0-2.5 Ga. The Project is located within an area characterized by Archean-age greenstone belts (metamorphosed mafic and ultramafic rocks, bounded by granitic gneiss).

The Project is underlain by three main stratigraphic units, consisting of a footwall and hanging wall gneiss and banded migmatite suite, between which is a belt of greenschist-amphibolite facies metamorphosed ultramafic rocks. These south-dipping ultramafic units, within a zone of high ductile shear strain, represent the remains of schist belt, and are hosts to the gold mineralization. Gold mineralization occurs in zones of variable thickness and is nearly continuous along 1.8 km of strike length. The Project deposits consist of high-grade gold mineralization, including intersections in excess of 5 g/t Au, contained within lower grade (0.5 to 1.0 g/t Au) material. This zone is marked by magnetite destruction, weak silicification, and the presence of pyrrhotite and arsenopyrite.

The primary targets of Aureus' mineral exploration programme in Liberia are shear zone-hosted gold. In addition to the New Liberty mineralization, there are six other identified localities on the Bea-MDA property which are currently undergoing exploration. These are Silver Hills, Weaju, Ndablama, Gondoja, Gbaladee and Koinja, all of which have similar geological characteristics to the New Liberty project. Ndablama, Gondoja, Gbaladee and Koinja are located within a regional structural corridor. Drilling to date has been carried out on Ndablama, Weaju, Gondoja and Gbaladee.

Project Status

Subsequent to the previously announced Feasibility Study in October 2012, Aureus has undertaken a number of studies with the objective of optimizing the project economics and improving the understanding of the gold deposit ahead of implementation.

The optimization of the project has resulted in the relocation of the processing plant and the tailings storage facility to the south of the open pit, revision to the grinding circuit design, refinement of the geotechnical parameters and waste dump design, and a more detailed water management plan.

AMC understands that Aureus intends to continue with construction and development activities at the project site following the release of this Technical Report and finalization of the financing process.

Exploration and Data Management

There has been no material additional exploration drilling at the New Liberty project since the Feasibility Study announcement in October 2012.

Exploration activity at the Project, which dates from reconnaissance site visits in 1997, has been prolonged as a result of stoppages associated with instability during the civil conflict in Liberia. Initial trench channel sampling led to further trenching, followed by soil geochemical surveys and later to drilling. Subsequent surveys included satellite imagery, aerial photography and airborne and ground geophysics.

Drilling has since been conducted in seven campaigns, either by contractors or with owner drill rigs. A total of 438 diamond drillholes have been completed for 65,187 m, the majority of which have been utilized in the mineral resource estimate.

Drilling at the Project has been carried out with holes typically inclined at -60°, resulting in true widths generally representing 75% of intersection lengths. Core recovery is typically over 90%. Downhole surveys are carried out on the majority the holes, though intermittently in the early campaigns. Full re-surveys of drillhole collar coordinates were carried out in 2010 and 2011.

QA/QC procedures have been varied and have evolved across the drilling campaigns. During the first campaign, half core samples were despatched for assay to the SGS laboratory in Abidjan, Ivory Coast. Sample pulp check assaying was conducted through the OMAC laboratory in Ireland (OMAC). However, no standard or blank sampling was undertaken and neither were any standard QA/QC procedures implemented.

During subsequent drilling campaigns, increasing levels of QA/QC were introduced, and samples were despatched to a preparation facility in Monrovia (operated by OMAC). Sample pulps were submitted to OMAC's laboratory in Ireland. ALS Chemex in Vancouver and, in 2011, SGS Canada Inc. in Toronto, have been used as umpire laboratories. All laboratories are accredited and participate in inter-laboratory proficiency test and certification programmes.

The progressive introduction of QA/QC procedures has seen the implementation of field duplicates, blank samples, standards and laboratory repeats, as well as specific programmes of re-assaying and umpire laboratory assaying. Analyses by AMC of the results from these procedures have exposed some areas of concern relating mainly to precision and low bias. AMC is of the opinion that the magnitudes of the QA/QC concerns are not sufficient to compromise the use of the gold assays for Mineral Resource estimation, but recommends ongoing work be undertaken to raise the level of confidence in sample and assay quality. AMC is aware of recent changes and corrections made by Aureus that are in keeping with some of these recommendations.

In general, the drilling and associated data has been collected in a diligent fashion and AMC regards the corresponding products of this work, including survey, geological and geochemical data, to be suitable for inclusion in the mineral resource estimation studies.

The geological data is managed by site geologists using a series of worksheets as data entry into an MS Access database. AMC considers this method of data management to be adequate for the size of the project database and the associated field working conditions.

Metallurgical Testing

Metallurgical consultants MDS Ltd (MDS), a subsidiary of the DRA group, completed a Feasibility Study on New Liberty in conjunction with DRA Mineral Projects (DRA) in the last quarter of 2012. An optimization phase was initiated by DRA in the last quarter of 2012, including additional test work phases on composite and variability samples with the following objectives:

- Evaluate the possibility of reducing CIL residence time from 48 hours to 24 hours by including a high shear pre-oxidation step to the flowsheet.
- Determination of the optimum cyanide addition requirements for the CIL circuit.
- Determination of the optimum reagent addition requirements for the cyanide destruction process.
- Verify that an overall gold recovery of 93.0% was achievable on variability samples.

The test work programme was undertaken by the Perth, Western Australia based ALS Laboratories (‘ALS’).

The findings of the optimization phase were as follows:

- The CIL circuit residence time could be reduced to 24 hours without negatively impacting on gold recovery by including a high shear pre-oxidation stage prior to leaching.
- The overall gold recovery was found to be strongly dependant on mill grind. Test work on the composite sample indicated that gold recovery could be improved by 2.8% when the grind was increased from 80% passing 75µm to 80% passing 42µm. 80% passing 45 µm was selected as the optimum target mill grind.

- The target mill grind of 80% passing 45 µm was achieved with the inclusion of a regrind milling stage, using a VertiMill with steel grinding media.
- Test work on the variability samples indicated cyanide addition requirements in the range of 0.50 kg/t – 1.92 kg/t with an average addition requirement of 0.65 kg/t. This was based on an initial cyanide addition of 0.5kg/t and further addition to maintain a solution concentration of 100 ppm cyanide.
- Test work indicated that the optimal pH control regime was to target a pH of 11 prior to the pre-oxidation stage and control the pH at 10 in the CIL circuit. Based on the optimized pH control regime lime addition requirements were found to be in the range of 0.88 kg/t – 2.13 kg/t with an average requirement of 1.48 kg/t.
- The SO_2/Air cyanide destruction process produced a tailings stream with less than 50 ppm CNwad at a CNwad: SO_2 ratio of 5:1. This translated to a Sodium Metabisulphate (SMBS) consumption of 0.77 kg/t for a CNwad level of 80 ppm in the CIL effluent stream.

The results of the optimization phase test work were used in conjunction with the test work results from Mintek test work phases 1, 2 and 7 to generate a correlation between head grade, grind and overall recovery. The test results from the Mintek test work phases 3, 4 and 5 did not include CIL testing. The Mintek test work phase 6 testing was not accepted by MDS and these results were not considered to be representative. This resulted in MDS requiring the additional test work phase 7 on the composites from phase 6 to verify leach recoveries. The correlation for a target grind size of 80% passing 45µm was then used to determine the expected plant recovery for full scale plant operations based on the mine plan that was developed in the October Feasibility Study. This recovery estimate is presented in Table 1.1 below:

Table 1.1 Plant Recovery

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

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

Mineral Resource Estimation

The Mineral Resource estimate reported in October 2012 remains the current estimate.

Resource estimation of the New Liberty gold deposit has been based on interpretations using integrated geological and grade information recorded from diamond drill core logging and assaying. The final drilling database available for use in the primary evaluation was received on 9 December 2011 and represents drilling up to and including drillhole K375, whereas an updated database, received on 4 April 2012, represents drilling up to and including drillhole K438.

The database was subject to a series of standard validation procedures, and up to 20% of the data was randomly selected for validation against source information. Not all requested source data was available for checking; however, no significant errors were detected in the checks undertaken.

Geological interpretations, with reference to surface mapping and drill logs, have been restricted to defining boundaries enclosing the known extents of the ultramafic unit. The mineralization has been correlated and interpreted as a number of distinctive planar units, within strike sectors named as; Larjor, Latiff, Kinjor and Marvoe. As additional data has become available, through the various drilling campaigns, the external limits of mineralized zones have been adjusted and refined, and currently constitute three laterally-extensive zones and two minor zones.

The interpreted mineralized zones all present as strongly planar shapes, aligned sub-parallel to the “silicified metamorphosed ultrabasic suite” unit (SMUS), and generally striking slightly south of east and dipping between 60° and 80° to the south. The characterization of mineralized intersections within drillholes was based primarily on assay information, with boundaries judged according to the clearest evidence of zone-to-background contrasts, for grades exceeding 0.3 to 0.5 g/t Au.

The interpreted geological and mineralization shapes were formed into solid triangulated wireframes, which were in turn filled with model cells based on parent cell XYZ dimensions of 10 m x 5 m x 10 m, with associated sub-celling to better represent the zone geometries. The model was coded to reflect stratigraphic units and individual mineralized zones, and to distinguish between weathered and unweathered material. A detailed digital terrain model of surface topography, generated from the December 2012 LiDAR survey, was used to constrain the upper bounds of the model.

Samples were coded by mineralization zone and weathering in a manner consistent with the cell model. Statistical analysis of gold grades was conducted on one metre composites from subsets extracted from the coded sample file (e.g. by mineralization zone). The results of the analysis demonstrated that the mineralized zones have a broad range of mean grades but a correspondingly relatively narrow set of coefficients-of-variation values. All zones exhibit some bimodality in their distributions, and this is attributed largely to grade variability both in the form of short scale changes within zone intersections and grade ‘shoots’ observed in the strike-dip planes.

The statistical analysis results were used, in conjunction with a spatial assessment of high grade samples, to determine individual high grade capping values for each mineralized zone, which were applied during grade estimation.

Variographic analysis was conducted on all mineralized zones with sufficient composites (four of five zones).

Gold grade estimation was conducted from one metre composites, using ordinary kriging in those fresh material zones for which variogram parameters were generated. Inverse distance squared weighting was applied to the remaining constrained zone, and to the background mineralization and the weathered zone.

The model was populated through grade interpolation into parent cells, under hard-bounded zonal control, using mostly 35 m x 60 m x 10 m (strike / dip / cross plane) search ellipsoids, aligned according to the local planar orientation of each zone. Cells not estimated in the first pass were estimated by subsequent passes with expanded search ellipses.

Procedures for classifying the estimated mineral resources were undertaken within the context of NI43-101.

Estimated tonnages and grades have been classified with consideration of the following criteria:

- Quality and reliability of raw data (sampling, assaying, surveying).
- Confidence in the geological interpretation.
- Number, spacing and orientation of intercepts through mineralized zones.
- Knowledge of grade continuities gained from observations and geostatistical analyses.
- The likelihood of defined material meeting economic mining constraints over a range of reasonable future scenarios, and expectations of relatively low selectivity of mining.

In general the drill spacing is more closely spaced near surface, progressively reducing in density with depth as the drilling tracks each mineralized zone down dip. On drill spacing alone, therefore, the level of confidence in the resource declines with depth. Consequently all candidate material for Measured or Indicated classification is located relatively nearer to surface.

All material has been reported at a 1.0 g/t Au cut-off (Table 1.2) as this value corresponds to likely open pit cut-off ranges. All of the Measured and Indicated material is within potentially open-pittable depths.

Table 1.2 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. Canadian Institute of Mining, Metallurgy and Petroleum (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.

Mineral Reserve Estimation

AMC carried out pit optimizations using the revised geotechnical, mining and processing cost parameters. AMC subsequently developed pit designs for the Project pits based on the measured and indicated mineral resources.

The mineral reserves, including appropriate allowances for dilution and ore loss, were estimated within the pit designs at an economic breakeven cut-off grade of 0.8 g/t. The estimate is summarized in Table 1.3.

Table 1.3 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.

Mining Operations

From the pit designs and Mineral Reserves, a life-of-mine (LOM) schedule was prepared using the following parameters:

- Process plant nominal ore throughput of 1.1 million tonnes per annum.
- Total material movement peaking at an average rate of 70,000 tonnes per day.
- Material grading between 0.65 g/t and 0.8 g/t will be stockpiled to areas of the waste dumps designated as mineralized waste stockpiles (this material is not processed and is not included in the study base case, however, it may be treated at the end of the mine life if economically viable).
- The capital and operating costs for the operation were estimated on the basis of the mine being operated by a mining contractor and power being generated on site using diesel generators. The mining equipment and associated infrastructure has been evaluated and chosen based on the mining and treatment schedule and detailed discussions with the mining contractor.

The Project will comprise an open pit mining operation extracting ore at a nominal rate of 1.1 Mtpa with an operating life of eight-and-a-half years. The open pit will comprise two adjacent and interconnecting pits.

The mine designs were completed by AMC based on pit optimizations carried out using Whittle-4X and limited to the Measured and Indicated Mineral Resources. Staged pits were designed within the ultimate pit limits to provide flexibilty for production scheduling.

The ultimate pit design is divided into two pits; these two pits are joined at surface. The Larjor zone to the west is separated from the Latiff zone by a poorly mineralized area which forms a saddle between the pits that comes to within 30 meteres of the original surface. The Latiff, Kinjor and Marvoe zones coalesce into a single pit. For the purpose of developing a mining schedule, the mine was divided into six-phases, Larjor, a starter pit within Latiff-Kinjor, a cut back to the Latiff-Kinjor pit, the final Latiff-Kinjor pit, a starter pit at Marvoe, and the final Marvoe pit.

The LOM schedule sees the operation produce a total of 8.5 million tonnes of plant feed at an average mined grade of 3.4 g/t Au, with an associated waste production of 131 Mt over an 8.5 year mining period. The annualized mining production can be summarized as shown in Table 1.4.

Table 1.4 Mining and Treatment Schedule

		Total	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Ore	**Mt**	8.5	0.3	0.7	1.1	1.1	1.2	1.3	1.1	1.3	0.3
Grade	**g/t**	3.4	3.8	2.7	3.5	3.2	4.0	3.8	3.3	3.1	2.4
Contained Gold	**000 oz**	924	40.1	64.4	126.3	116.3	154.0	154.4	114.4	126.6	27.2
Waste	**Mt**	131	2.1	22.1	24.4	24.5	24.4	19.2	9.4	5.1	0.3
Total Material	**Mt**	140	2.4	22.9	25.6	25.6	25.6	20.5	10.5	6.3	0.6
Strip Ratio	**t w:o**	15.5	6.4	29.9	21.5	21.6	20.3	15.3	8.7	4.0	0.8
Ore Milled	**Mt**	8.5	-	1.0	1.1	1.1	1.1	1.1	1.1	1.1	0.8
Grade	**g/t**	3.4	-	3.1	3.6	3.2	4.0	4.0	3.5	3.3	2.0
Contained Gold	**000 oz**	924	-	101.1	127.0	115.2	142.1	142.2	126.3	118.2	51.6
Recovery	**%**	93%		93%	93%	93%	93%	93%	93%	93%	93%
Gold Produced	**000 oz**	859	-	94.0	118.2	107.1	132.1	132.2	117.5	110.0	48.0

A pre-strip period of 6 months with relatively low material movement rates was scheduled to establish the pits and to build an ore stockpile ahead of plant commissioning. The schedule steps up to an average material movement rate of 70,000 tpd from the end of the second quarter of Year 1 (Period 15).

In this schedule, processing commences at the start of year 1, treating the small initial ore stockpile and pit ore production. Open-pit material movement achieves a steady state annual production of 1.1 Mt ore and 25.6 Mt total movement by the end of Year 1. This steady state production at this rate continues until halfway through Year 5. The total movement declines after Year 5 as the strip ratio declines from around 25:1 to around 10:1 towards the end of the mine-life.

The Larjor pit is completed at the end of Year 4, and waste is backfill into the Larjor pit from this period until the end of the operation in Year 8.

The waste material will be placed in a waste dumps wrapping around the pit for the first four years of production, after which the majority of waste material will be backfilled into Larjor pit.

A mining contractor will be used for all earthmoving activities and mining operations will use a conventional truck-and-shovel method. Aureus has appointed been through a pre-selection process and is now working with a reputable West-African experienced mining contractor as an external contractor to perform the mining operations at the Project, with the company's own mining personnel providing a managerial and technical services function to the operation.

The mining contractor will supply all of the capital mining fleet requirements, including pumps for pit de-watering, and will be responsible for the site infrastructure associated with mining operations. Mining will be based on conventional drill-and-blast, load-and-haul techniques with both ore and waste rock being blasted and then loaded into 100 t trucks with hydraulic excavators. It is envisaged that 50% of the ore will report directly to the crusher tip and 50% to the ROM ore pad. Waste will either be dumped on the closest available waste dump or backfilled into mined out areas in the pit. The ROM pad stockpile area will be constructed adjacent to the process plant crusher station with ore stockpiled being sorted under a stockpile grade control management scheme and fed to the plant by means of a front end loader as required.

The mine will adopt a three crew system for all operating personnel, one shift day shift, one shift night shift and a roster off shift.

The open pit dewatering strategy is based upon a study undertaken by Aquaterra in June 2013. The total average monthly inflows have been estimated for the life of the mine. The water management plan for the open pit will be two pronged, firstly managing surface runoff water and then pumping from inside the pit with an adequately sized pumping infrastructure.

Production Schedule

The gold production schedule is shown in Table 1.4.

Recover Methods - Process Plant Design

The design criteria for the process plant have been based on the metallurgical test work and industry design principles. The New Liberty process plant is designed to treat 1.1 million tonnes per annum.

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

The process flowsheet is typical of CIL plants in the gold mining industry. It is considered to be low risk and historically has been proven a successful processing route for ore bodies in Africa.

Infrastructure

The current site infrastructure consists of an exploration mine camp, core storage and accommodation facilities. The proposed infrastructure required for the development of the Project, which will support construction, mining and production, includes the following:

- Mining infrastructure and general infrastructure
  - Mining equipment workshops, fuel storage and explosives storage.
  - Processing plant, administrative facilities, security, assay laboratory and medical facilities.
  - Water services and waste control.
  - Camp accommodation and facilities.
  - Security
  - Communications
  - Access roads
- Power supply and distribution.
- Process tailings management – TSF
- Marvoe Creek Diversion Channel
- Waste rock dump (WRD)

Infrastructure, including materials and consumable goods, will be transported by truck to the Project site from either the port of Monrovia or Roberts International Airport in Monrovia. Currently there is approximately 80 km of paved road to the town of Danieltown and then 20 km of laterite road to the Project site. Some upgrade work has recently been conducted on the road between Danieltown and the Project site to facilitate the transportation of infrastructure to site - this work was carried out by an external contractor, and primarily focused upon the construction of three culvert type bridges, road widening and the installation of drainage gullies and culverts.

During the construction phase of the Project, it is estimated that the workforce will peak at 600 employees. For the production phase of the Project it is estimated that there will be an average of 300 workers employed.

Tailings Storage Facility

Under the management of DRA, Epoch Resources (Pty) Ltd (Epoch) were appointed to undertake an optimization study and detailed design of the Project's Tailings Storage Facilty (TSF).

The brief included a review of the Golder's Feasibility Study TSF option, evaluations of the LiDAR terrain model for alternative sites, development of a proposed design philosophy and compliation of a bill of quantities and capital cost for the selected TSF option.

There are no Liberian guidelines related to the design, operation and closure of tailings storage facilities and water diversion systems. In the absence of local guidelines, internationally recognized publications and industry standards have been used to develop site specific design criteria.

The milling process is expected to generate a total of approximately 9.4 Mt of tailings over the life-of-mine. The dry density of the deposited tailiings has been estimated at 1.36 t/m^3, requiring storage of a total of 6.9M m^3 of tailings.

A previously considered, but discounted site to the south of the open pit was selected by Epoch, and the main advantages offered over the Feasibility Study site are:

- Valley dam TSF with a single embankment / starter wall of significantly less earth fill volumes
- Close proximity to and downslope from the process plant requiring less pumping effort and slurry piping and lower pumping costs
- Design mitigates environmental risks and no longer requires the tailings pipeline to cross the Marvoe Creek

Marvoe Creek Diversion Channel

The Marvoe Creek is the primary drainage feature in the Project area. It is fed by numerous small tributaries and is itself a tributary of the Mafa River, which lies 5 km south-west of the Project. The creek diagonally bisects the Project site and the alignment is such that it passes through the proposed Open Pit and WRD site. As a result, a permanent diversion channel is planned to route Marvoe Creek around the Open Pit and the WRD. The drainage area of the creek upstream of the proposed diversion is approximately 109 km^2. Where is passes though the project site, the Marvoe Creek is approximately 30 m wide with a mild slope.

Following the Feasibility Study Design and under the management of DRA, Epoch was commissioned to conduct a review and optimization study of the MCDC.

The chosen Epoch design philosophy of the diversion channel is to utilise the existing natural environment by diverting by way of dams and excavations the existing Marove Creek into an adjacent natural channel. The design has two objectives, to safely and cost effectively divert the Marvoe Creek around the proposed pit and secondly to minimize impacts on the local environment.

The general arrangement of the MCDC consists of two flood control dams with a cutting connecting the two dams, a by-wash spillway, diversion channel, and flood control berms.

Environmental and Social

The primary permit/licence required for the development of the Project is an Environmental Permit issued by the Minister of Environment. This was granted for the Project in October 2012 and is valid for three years subject to an annual renewal by the Liberian EPA.

An Environmental and Social Impact Assessment (ESIA) for the project was undertaken from Q4 2010 to Q2 2012 to establish baseline data for the local environmental and social situation existing prior to the development of the Project and to determine the likely positive and negative impacts of the Project.

The study area for the ESIA consists of the footprint of the proposed Project (approximately 8 km^2); upstream and downstream areas; local topography; the directly-affected villages of Kinjor and Larjor; neighbouring villages within a radius of approximately 5 km of the site; and the 22 km gravel road via Danieltown in the South that serves as the Project's main access road. In addition, consideration was given to the wider geographical context where applicable.

The EIS details the findings of the environmental and social baseline studies and specialist studies conducted during the ESIA; it also presents the Environmental Management Plan (EMP). The format for the Environmental Impact Assessment is taken from and aligned with the Liberian EPA "Environmental Impact Assessment Procedural Guidelines" (2006) and commitments are in line with international standards and practices.

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

Subsequent to the completion of the ESIA, a mine optimization study was conducted to identify better positions for the plant, tailings dam and mine village. The newly-identified positions are all within the area permitted for mining and are therefore covered by the ESIA. These changes however, necessitated modification to the areas subject to detailed baseline studies, which are currently underway and will be finalized by the end of June 2013. The findings of these studies will be submitted as an addendum to the approved ESIA.

Resettlement Action Plan (RAP)

Development of the Project will involve relocation and resettlement of two villages (Kinjor and Larjor) encompassing 325 property owners and their households, as well as the relocation of some households along the access road. Relocation of the latter is primarily motivated by potential safety impacts associated with increased traffic volumes caused by the Project.

Aureus contracted the services of independent consultants Digby Wells and Associates (Pty) (Digby Wells) to develop the RAP to address the resettlement impacts associated with the project. International best practice for resettlement related to private sector projects is guided by the IFC's Performance Standards on Social and Environmental Sustainability, and particularly defined by the IFC's Performance Standard 5: Land Acquisition and Involuntary Resettlement. The RAP was approved by the Liberian EPA during April 2013 and the relevant owners' compensation packages approved.

Digby Wells also developed a Community Development Plan (CDP) for the resettlement affected households and communities. This plan was completed in December 2012 and was subsequently approved by the Liberian EPA in January 2013.

The development and operation of the Project will have both positive and negative impacts on the socio-economic structure of the Project area and its environments, as well as impacts at a District and National level.

Negative impacts relate to the disruption of the local social dynamics and increased pressure on local infrastructure and resources, mostly due to the influx of people to the area. The positive impacts will relate mainly to the economic and infrastructure advantages which will have immediate and long-term benefits for the socio-economic environment. This will be achieved in various ways at National, District and Local levels through the payment of taxes and royalties, increased employment opportunities, training, purchase of goods manufactured and supplied in Liberia, cash compensation for farms, commercial opportunities and an improvement in local infrastructure by the establishment of the Resettlement site/town and upgrading of the local roads. The development of the Project will bring much-needed investment and development opportunities with consequent positive impacts on employment and the affected communities.

Capital and Operating Cost Estimates

The combined study team, including those listed in Section 1.1, developed estimates of the capital and operating costs to an accuracy level of ±10%.

Capital Cost

The capital cost estimate is based upon an engineering, procurement and construction management (EPCM) approach where the owner assumes the builder's risk.

The capital cost for the Project is summarized in Table 1.5 and Table 1.6.

Table 1.5 Initial Capital Cost Estimates

Category	US$ million	%
Processing plant	56.0	41%
Infrastructure – earthworks and buildings	26.0	19%
Indirect costs – EPCM fee, pre-production costs, consumable and spares	27.2	20%
Initial mining pre-strip	6.3	5%
Tailings dam construction	7.2	5%
Marvoe Creek diversion	6.0	4%
Village relocation	3.5	3%
G&A and owner costs	3.8	3%
TOTAL	**136.0**	**100%**
Contingency	13.6	10.0%

Deferred capital expenditure will be incurred following the commencement of production.

Table 1.6 Deferred Capital Cost Estimates

Category	US$ M
Sustaining capital and mine closure	5.7
Diesel generators, fuel farm and mining fleet over LOM	77.5
Mine strip – phases 2, 3 and 4	18.0

The mine closure costs cover environmental aspects at the mine and process plant sites. Mining operations will be undertaken on a contract basis. The diesel generators, fuel farm, and mining fleet equipment are covered by lease agreements over the LOM.

Operating Costs

The operating costs were estimated as summarized in Table 1.7.

Table 1.7 Operating Cost Estimates

Item	US $/tonne processed	US $/oz produced
Mining (excluding deferred strip)	38.71	383
Processing	22.57	223
General and Administration	6.25	62
Total	67.53	668

The annual operating costs are summarized in Table 1.8.

Table 1.8 Annual Operating Cost Estimates

US$ M	Total	2015	2016	2017	2018	2019	2020	2021	2022
Mining Costs	328.8	43.0	57.6	58.7	61.5	47.7	27.6	11.5	21.3
Processing Costs	191.8	23.0	25.1	25.1	25.1	25.1	25.1	25.1	18.4
General & Administrative Expenses	53.1	6.6	6.9	6.9	6.9	6.9	6.90	6.9	5.1
Total Operating Costs	**573.7**	**72.6**	**89.6**	**90.7**	**93.40**	**79.7**	**59.5**	**43.6**	**44.8**

Economic Evaluation

An economic evaluation has been conducted on the project using a cash flow model. Based upon the mining and processing schedules described and the costs estimated, an economic evaluation was carried out by Aureus using a flat gold price of US$1,400.

A government royalty rate of 3% of net revenue was included in the analysis.

The key outcomes of the economic evaluation for 100% of the Project pre-financing costs are summarized in Table 1.9.

Table 1.9 Economic Evaluation

Gold price	US $/oz	1,300 flat	1,400 flat	1,500 flat
Gross revenue	US$ M	1,116	1,202	1,288
Net smelter revenue	US$ M	1,080	1,163	1,247
Net operating cash flow	US$ M	506	589	673
Net pre-tax cash flow	US$ M	269	353	436
Pre-tax NPV (5%)	US$ M	166	230	293
Pre-tax IRR	%	23	29	34
Post-tax NPV (5%)	US$ M	119	165	210
Post-tax IRR	%	19	24	28
Payback	years	3.9	3.4	2.9

Taxes and Royalties

Based on the base case of a flat gold price of US$1,400 per oz the government of Liberia will receive corporate tax revenues of US$87 million and gold royalties of US$36 million.

This is based on a royalty rate of 3% for Government of Liberia and a corporate tax rate of 30%.

Payback

Payback for the project in respect of the initial capital expenditure of US$136 million is 3.4 years.

Sensitivities

The economic model was used to test the Project's robustness to changes in the key value driver assumptions. Sensitivity analysis, as summarized in Table 1.10, indicates that the Project is most sensitive to gold price (or ore grade) fluctuations, followed by operating cost fluctuations, and then capital cost fluctuations.

Table 1.10 Project Sensitivities

Sensitivity	NPV 5% Discount Rate	Variance to Base Case
Gold Price	US$M	%
+10%	319	+39
-10%	141	-39
Initial Capital Costs		
+10%	218	-5
Operating Costs		
+10%	184	-20
Grade		
-10%	141	-39

Further Optimization Work Being Conducted

Studies continue to test the groundwater levels and flow rates, and to develop a groundwater model.

Interpretations and Conclusions

The detailed evaluation of the Project in respect of the exploration, test work and study work that has been undertaken for the purposes of this Technical Report indicates that the exploitation of the Project is operationally and economically viable.

Mineral Resources

The various phases of exploration at the Project have in general been conducted in a comprehensive and systematic manner, with good quality field professional and technical input. The quality of some work has varied over time and by technical area, and a number of recommendations for improvement made in 2010 were adopted during the 2011/2012 drilling campaign.

Diamond core drilling over a series of campaigns has consistently intersected gold mineralization in a broad zone representing a predominantly southerly-dipping schist belt remnant. Within this stratigraphic interval, the defined concentrations of higher grade mineralization, extending from known surface occurrences to drill intersections more than 500 m below surface, have been intersected in what has become an increasingly predictable distribution with each campaign.

The 2011/2012 infill drilling campaign verified the general character of the mineralization, and provided increased levels of confidence in local interpretations and grade estimates across the upper portions of the mineralized zones, leading to an expansion of indicated classified material and the inclusion of some areas as measured category.

The 2011/2012 drilling focused on infill objectives; however, some extension drilling, along strike and below the Latiff Zone was only partially successful in extending the mineralization beyond the previously defined limits. With depth the Latiff Zone was shown to continue, but in the central and western extensions, identified zone thicknesses and grades are poor. The Latiff Zone has now been shown to correlate with the Larjor Zone in the west and the Kinjor Zone in the east, but it is only in the deeper portions of the Latiff-Kinjor interface that economically significant grade intersections have been found.

The additional drilling data used for the April 2012 updated mineral resource estimate includes an intersection in drillhole K427 which is of significantly greater width and grade relative to surrounding intersections. The location and general character of this intersection are consistent with the assigned interpreted mineralized zone; hence AMC has no basis for concluding that the intersection is other than a valid representation of the mineralization.

AMC cautions, however, that the combined effects of the marked grade and thickness of drillhole K427 have been shown, through sensitivity estimates, to significantly impact on the local estimates of gold metal. AMC strongly recommends that further drill data be collected to clarify the lateral extents of the interpreted mineralization and the economic impacts associated with this single intersection.

Aureus continues to explore the Project for the purposes of both resource definition and extension; however the substantial drilling conducted during the 2011/2012 campaign has largely fulfilled the requirements of resource definition for the current level of evaluation.

Economic Evaluation

The Project has a pre-tax IRR of 29% and a payback period of 3.4 years based on a flat gold price of US$ 1,400 / oz. The initial capital cost for the Project is US$136 million.

On the basis of this evaluation, the Project is viable at the gold price of US$1,400/oz used for the evaluation. The sensitivities conducted on the economic model confirm that the project is robust.

The gold price selected by Aureus of $1,400 /oz was based on the spot price during the compilation of the DFS and taking a view of industry gold price forecasts.

On this basis, AMC concurs with the decision by Aureus to proceed with the development of the mine.

Recommendations

Resources and Reserves

The lateral extent of the interpreted mineralization around the strongly mineralized drillhole, K427, should be tested with at least three drillholes in the immediate vicinity.

Sample QA/QC management and tracking procedures should continue to be a focus of exploration activity, and periodic reviews by independent specialists should be conducted.

As part of standard good practice, an audit by an independent sampling and assaying specialist of the full chain of sample preparation activities should be undertaken, including sample preparation procedures at SGS Monrovia.

In anticipation of further expansion of the quantity of exploration and resource definition data, all aspects of resource data management should be reviewed.

There is scope to increase current reserves through the drilling of inferred resources on hanging wall lenses within the pit as well as drilling of inferred resource blocks just below the bottom of the current optimized pit.

The mining reserve was estimated using budget mining costs provided to Aureus by mining contractors with experience in the region. Aureus has now selected their preferred contractor and are working closely with them to develop the final contract mining costs for the project. These costs should be compared with those used for the optimizations and analysis in this report.

Open-Pit Mining Contract

Aureus have selected a preferred mining contractor and contract discussions are in an advanced stage.

CONTENTS

TABLES

FIGURES

2 INTRODUCTION

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

Aureus completed a Feasibility Study which was reported in October 2012. Subsequent to this, additional work has been carried out with a view to optimizing the project; this report reflects that additional work.

This report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), “Standards of Disclosure for Mineral Projects”, of the Canadian Securities Administrators (CSA) for lodgement on the CSA’s “System for Electronic Document Analysis and Retrieval” (SEDAR). This report restates the estimate of mineral resources, and reports an updated ore reserve statement and economic assessment.

This Technical Report has been prepared by Christopher Arnold, MAusIMM (CP), Martin W Staples, FAusIMM, Robin M Welsh Pr Eng MSAIEE, Glenn Bezuidenhout, and Graham Trusler, all of whom meet the requirements of a Qualified Person, and are independent as defined in NI 43-101.

Chris Arnold, Principal Geologist, visited the Project site between 1 and 3 December 2009 and 1 and 5 October 2011, and during the visits he conducted ground inspections of exploration activities, including drilling, sampling, core logging, core cutting and site data management. The sample preparation facility, located in Monrovia, was inspected during the second site trip but the primary laboratory in Loughrea (Ireland) has not been visited. The Silver Hills, Weaju, Ndablama and Gondoja exploration prospects, which are also located within the Bea-MDA property, were not visited by Chris Arnold and are described in this report in terms of property information but are not encapsulated within the purpose of the report.

Martin Staples, Principal Mining Engineer, visited the site 19–25 November 2012 and several other AMC staff members who have worked on the Project with Martin have visited site. Marnie Pascoe, MAusIMM (CP), Principal Geotechnical Engineer, visited the New Liberty site in February 2011 at which time she inspected diamond drill cores and held discussions with the site-based geologists. Martin Tucker, Senior Mining Consultant has visited site on several occasions through 2011 and 2012 and Tom Styles, Geotechnical Engineer visited site in 2011.

Robin Welsh visited the site from 21–22 May 2012, and 21–25 January 2013, and 5–8 March 2013, and 14–17 May 2013 and 11–14 June 2013. Gelnn Bezuidenhout visited the site between 21–22 November 2012. Graham Trusler visited the site 20–21 March 2013 and 13–17 May 2013.

3 RELIANCE ON OTHER EXPERTS

With respect to the Mineral Development Agreement between The Republic of Liberia and Bea Mountain Mining Corporation (Section 4 of this report), AMC has relied on copies of documents provided by Aureus that confirm the terms of the Agreement.

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

Most of the factual text for this Technical Report covering Items 4–12 was prepared by Aureus and provided to AMC for review. Aureus also supplied supporting technical documents which AMC has used to verify this data where practical.

AMC has compiled this report based on sections prepared by the individuals listed in Table 3.1. The corresponding sections are recorded in the table, and each author has consented to publication.

Table 3.1 Qualified Persons Responsible for Report

Qualified Person	Position	Employer	Independent of Issuer?	Professional Designation	Sections of Report
Christopher G Arnold	Principal Geologist	AMC Consultants (UK) Limited	Yes	MAusIMM (CP)Geo	Sections 4-12, 14 and 23
Martin W Staples	Principal Mining Engineer	AMC Consultants (UK) Limited	Yes	FAusIMM	Sections 1,2,3,15,16,19,22,24,25,26,27 and parts of Section 18 and Section 21
Glenn Bezuidenhout	Process Director	DRA Mineral Projects	Yes	FSAIMM	Section 13 , parts of Sections 17, 21, and 25
Robin M Welsh	Senior Project Manager	DRA Mineral Projects	Yes	Pr Eng MSAIEE	Parts of Sections 17, 18, and 21.
Graham Trusler	Chief Executive Officer	Digby Wells Environmental	Yes	Pr.Sci.Nat	Section 20

4 PROPERTY DESCRIPTION AND LOCATION

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

4.1 Location

The Bea-MDA property is situated 90 km north-west of the capital in Grand Cape Mount County, in the north-western portion of Liberia, approximately longitude 11° west, 7° 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.

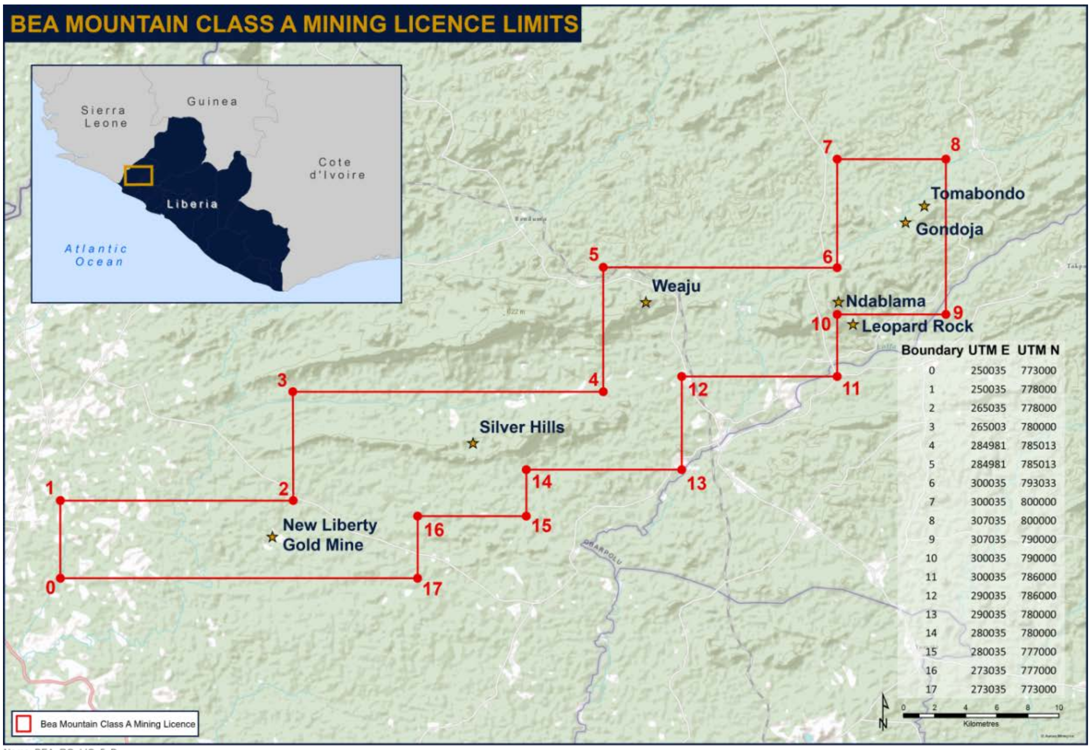
Figure 4.1 Location of the Bea-MDA Property in Liberia

Source: *Aureus, 2012*

4.2 Property Description

The Bea-MDA property covers an area of 457 km² with boundaries described by cadastral and cartographic survey in maps at the Ministry of Lands, Mines and Energy Republic of Liberia. The Project is show in Figure 4.2, along with the other targets which are currently the subject of exploration. The Bea-MDA property, which is covered by a Class A mining licence, has been reduced from a prior exploration lease which covered a total of 1,000 km².

Figure 4.2 Class A Mining Licence Limits



Name: BEA_RG_LIC_5_B

Source: *Aureus, 2012*

Table 4.1 WGS84 UTM Zone 29N Vertices of the Class A Mining Licence

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

4.3 Ownership

BEA has a 100% interest in the current Bea-MDA, which was signed with the Liberian Government in November 2001. BEA is a wholly owned subsidiary of Aureus. BEA 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 and US$10.6 million cash (the "Transferred Assets") to Aureus and African Aura was renamed Afferro Mining Inc.

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

Table 4.2 summarizes the ownership history.

Table 4.2 Ownership History

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

4.4 Title

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

Under the terms of the Bea-MDA, there is a 3% royalty to the Republic of Liberia calculated on a production basis. In addition, the Republic of Liberia is entitled to receive, free of charge, an equity interest on BEA's operations equal to 10% of its authorized and outstanding share capital without dilution (i.e. a 10% "carried interest"). African Aura through its subsidiary was required to pay the Republic of Liberia US$0.08

per acre per year as a rental fee for the Exploration Licence. Due to the civil unrest in the country, the Ministry of Land, Mines, and Energy suspended the exploration period as from July 2002 until 4 January 2005.

During the initial term of the Bea-MDA, BEA was required to make minimum exploration expenditures of US$1.40 per acre per year. Excess expenditures in a given year can be credited against succeeding years work requirements. The Bea-MDA provides BEA the right to free access to public land and will assist BEA in cases where access to private lands is necessary. Prior to the commencement of exploitation and production BEA is required to provide an Environmental Impact Statement to the Minister, detailing any adverse effects operations may have on the environment and review plans to mitigate such effects. From time to time BEA is required to submit detailed plans “for the protection, correction and restoration of the water, land and the atmosphere”.

BEA was granted a Class A Mining Licence (the Licence) on July 29, 2009. Annual licence fees for the Licence, based on the production area of 457 km² (“the Production Area”), amounts to US$0.80 per acre, which equates to US$90,146 per annum (1 km² = 247.1 acres). The Licence for the Production Area selected by the operator of the Project shall remain valid and effective for the unexpired portion of the Bea-MDA and any extensions thereof. The Licence allows BEA to commercially exploit minerals found in the Production Area and all other activities incidental thereto, including the design, construction, installation, fabrication, operation, maintenance and repair of infrastructure, facilities and equipment and the mining, excavation, extraction, recovery, handling, beneficiation, processing, milling, stockpiling, transportation, export and sale of minerals.

In order to maintain the Licence, BEA is required to demonstrate proven mineral reserves.

BEA will need to apply for and acquire normal operating licences and permits for the mining operation, including licences associated with explosive storage and use, abstraction and discharge of water and construction.

4.5 Environmental

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

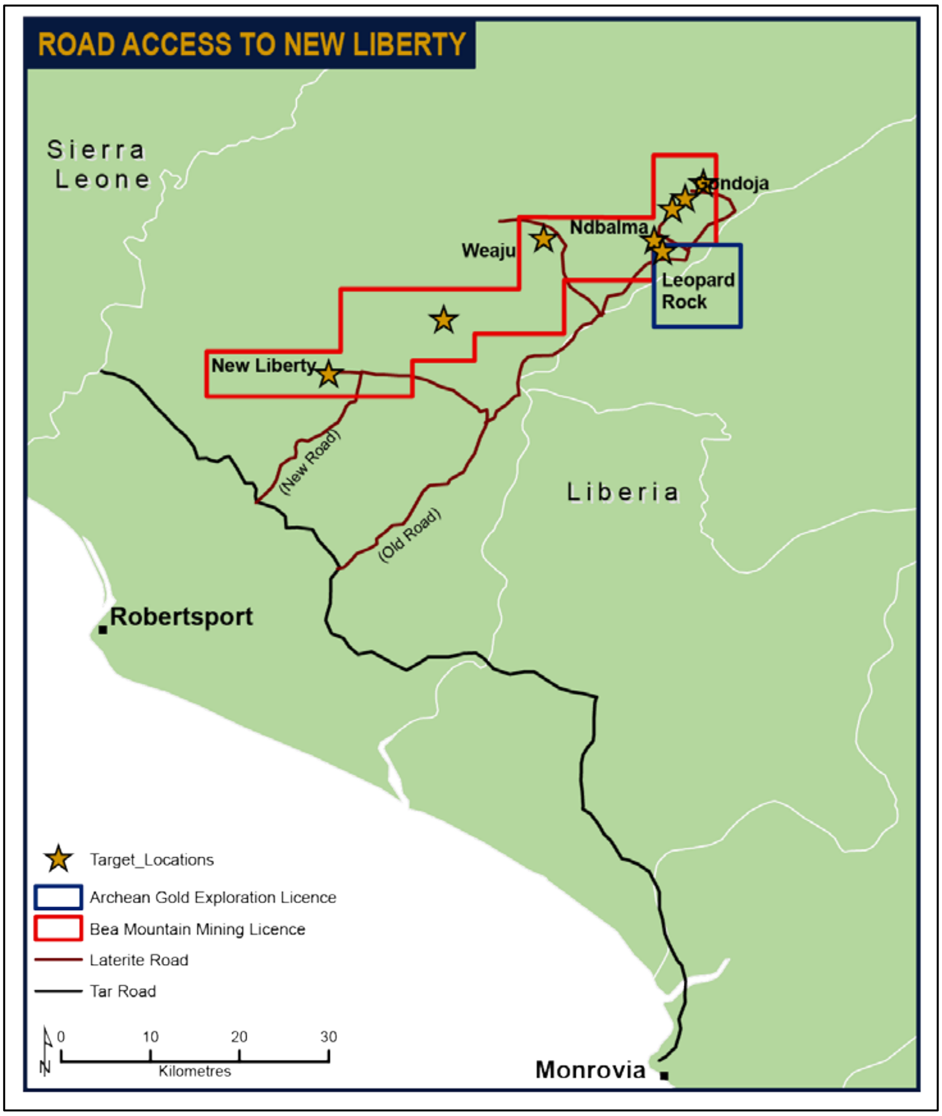
Baseline data collection for the ESIA was initiated in the fourth quarter of 2010 and was conducted during both the wet and dry seasons. The ESIA, as per Liberian legislation, included a Public Participation Process (“PPP”). An Environmental Impact Statement (EIS) was submitted to the Environmental Protection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

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

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

Figure 5.1 Road Access to the Project



Source: *Aureus, 2012*

5.2 Physiography

Within the Bea-MDA property are both primary and secondary forest, as well as some grassland and farmland. The topography ranges from around 50 m above sea level to a maximum of 600 m with the majority of the licence area being composed of gently undulating plains at less than 200 m height, with two prominent east-west ridges of resistant rock units (the Bea Mountain and Tokani ranges). Vegetation consists of tropical trees attaining heights of 30 m to 40 m above the forest floor, with thick undergrowth common (primary rain forest is mainly in the mountainous areas). The Project consists of gently undulating terrain at around 70 m above sea level, 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 year-round with heavy rainfall from May to October and a short interlude in mid-July to August. During the winter months of November to March, dry dust-laden Harmattan winds blow inland. Average annual rainfall along the coastal belt is over 4,000 mm and declines to 1,300 mm at the forest-savannah boundary in the north (Bongers, F et al, 1999). Temperatures range from the low 20 ºC's during the rainy season to warm (low 30 ºC's) during the dry season. Exploration has been able to continue throughout the rainy season.

5.4 Infrastructure

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

The Liberian Electricity Corporation currently supplies 10MW in Monrovia, with private generators meeting the remaining requirement. The Freeport 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. It can accommodate third-generation container ships.

Liberia has approximately 10,600 km of road networks throughout the country, of which 650 km are paved highway. Some of the dirt roads in the interior of the country were constructed in the 1990s, chiefly by Asian timber companies. These roads were well built and maintained at the time. Access to the Project is addressed in Section 5.1.

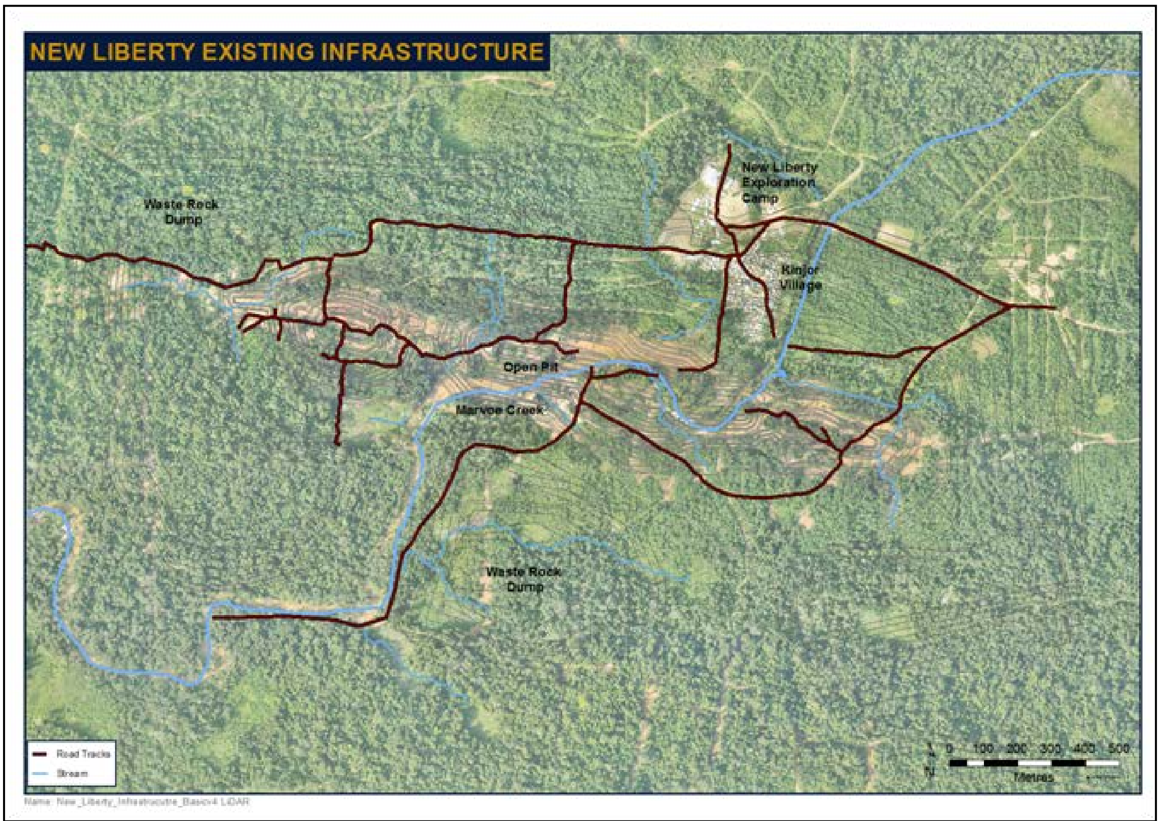
The 490 km of rail line in Liberia was primarily constructed to haul iron-ore from interior mining areas to ports. Much of the Bong Mine rail is still usable, while ArcelorMittal has renovated the Nimba railway to the port of Buchanan. Buchanan lies well to the east of the New Liberty Project (250 km) and consequently has no impact thereon.

Private satellite internet service is available in Monrovia and in some smaller urban centres. The Aureus camps at the New Liberty and the Ndablama and Weaju sites have 2560-1024 kbps and 512-512 kbps Vsat VOIP facilities. Cellular phone coverage in

Liberia is good within the major urban areas and is widespread throughout much of the country.

The Project area is split by the Marvoe Creek, a small perennial creek, which, during the dry season is the focus of artisanal alluvial gold workings. The Marvoe Creek would require diversion to allow full scale open cast mining, with raw water being extracted from a diversion trench and the overflow being diverted to join the Marvoe Creek south of mine workings. There is adequate space on the BEA leases or in the surrounding area for the processing plant, tailings storage facility and waste dumps. Figure 5.2 shows infrastructure at the Project.

Figure 5.2 Current Infrastructure at the Project



Source: *Aureus, 2013*

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

5.5 Local Resources

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

Within the Property, there are several local, small-scale alluvial diamond and gold operations on the Mabong, the Lofa and the Yambassei rivers. The New Liberty, Gondoja North, Weaju and Ndablama projects all have small scale artisanal mining communities.

6 HISTORY

Numerous artisanal gold mining sites that occur within the Bea-MDA property highlight the potential for local, 'source' gold mineralization. At the Project, to the extent known, there are only limited artisanal workings, with the majority of miners seeming only interested in alluvial gold. In cases where workings encounter bedrock or solid quartz, the pit is abandoned.

Work carried out by Golden Limbo had consisted of desktop studies, 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 mineralization were identified through geological mapping, supported by soil and stream geochemical sampling programmes. An overview of exploration activities across the licence is show in Figure 6.1.

Two previous resource estimates were undertaken 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 mineralized 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. JORC definitions were used for Mineral Resources.
2. Cross-section method employed.
3. No cut-off used, as mineralized zone taken.

The LQS estimate was produced in support of a study by MDM Engineering Group Limited (MDM) and was based on significantly more holes than the early estimate.

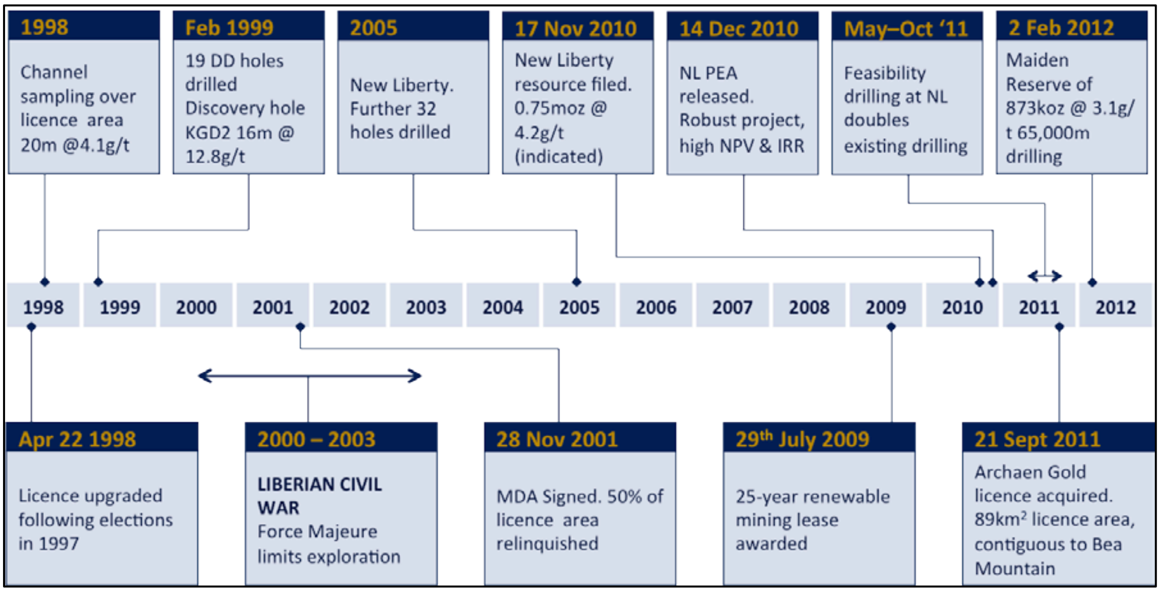
Table 6.2 LQS 2006 Mineral Resource Estimate

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. CIM definitions were used for Mineral Resources.
2. A cut-off of 1.0 g/t Au is applied for all zones.

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

Figure 6.1 History of Exploration at Bea Mountain Property

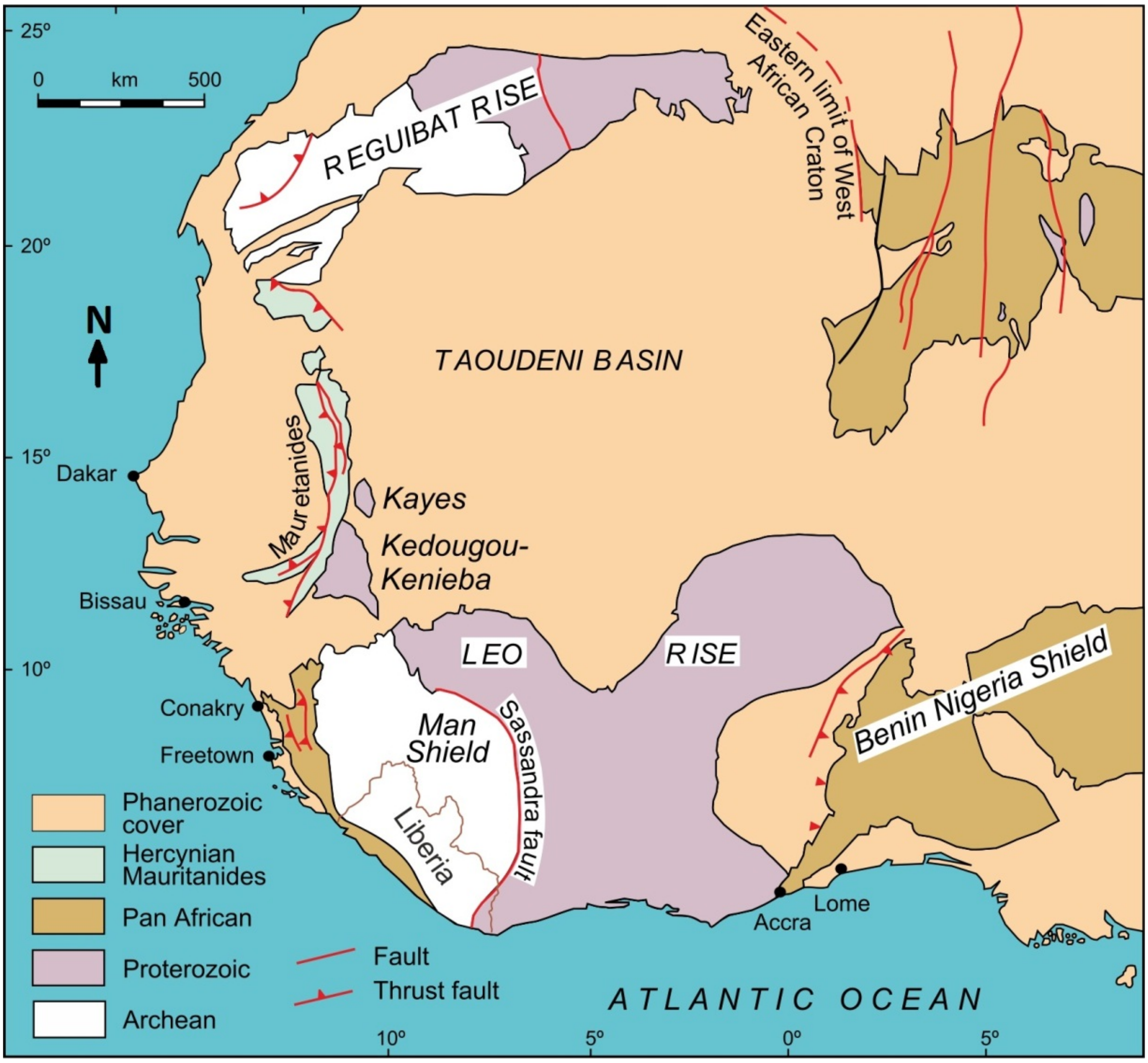


7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

Liberia is situated within the West African Craton, which has remained stable since about 1.7 Ga. The 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 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).

Figure 7.1 Regional Geological Setting

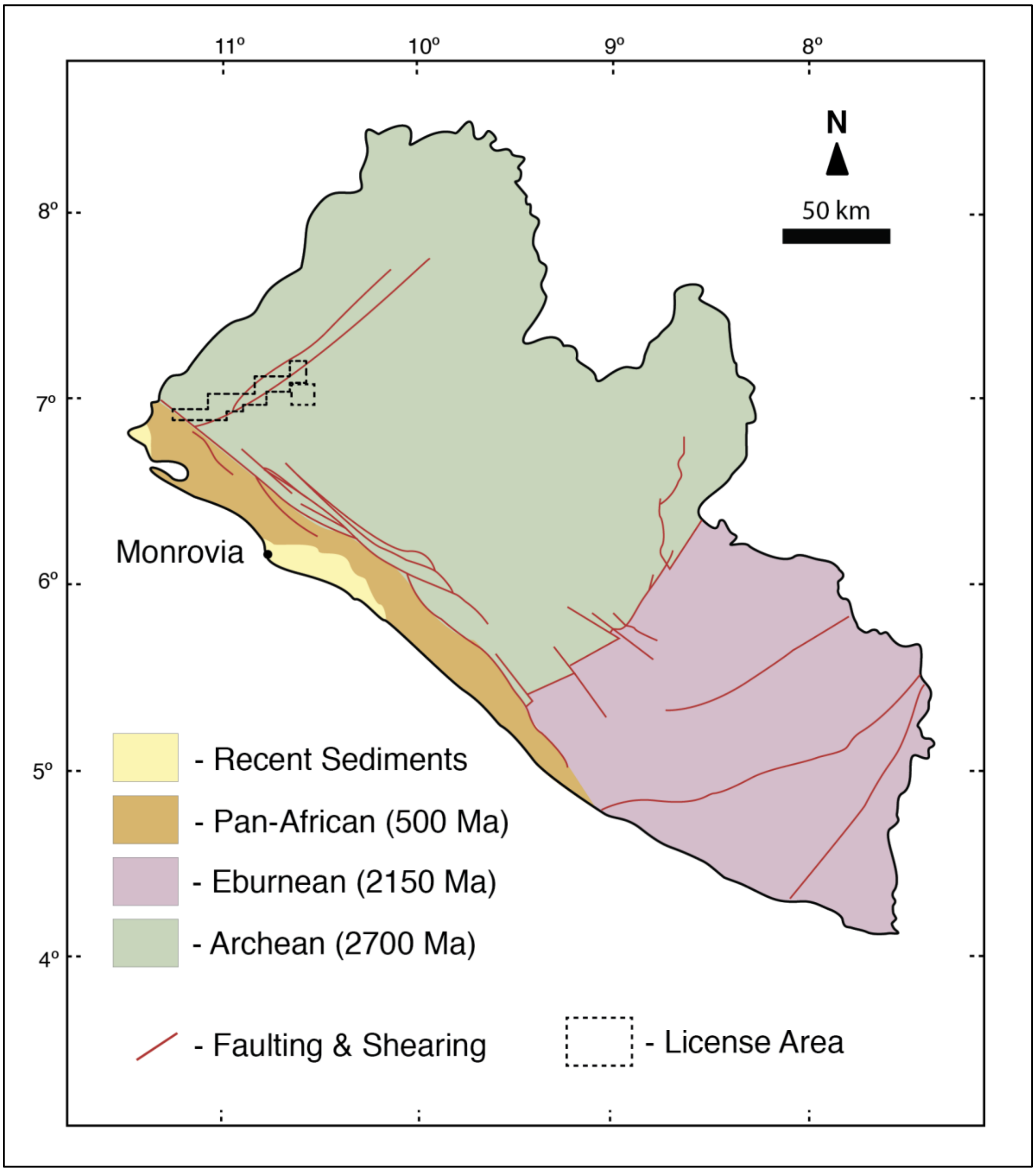


Modified from: *Milési et al. 1992*

To the east of the 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 Archean 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 Archean rocks have been subjected to deformation and shearing, with the principal structures acting as conduits for mineralizing fluids.

An Archean 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 Archean 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 mineralization.

Figure 7.2 Age Province Map of Liberia



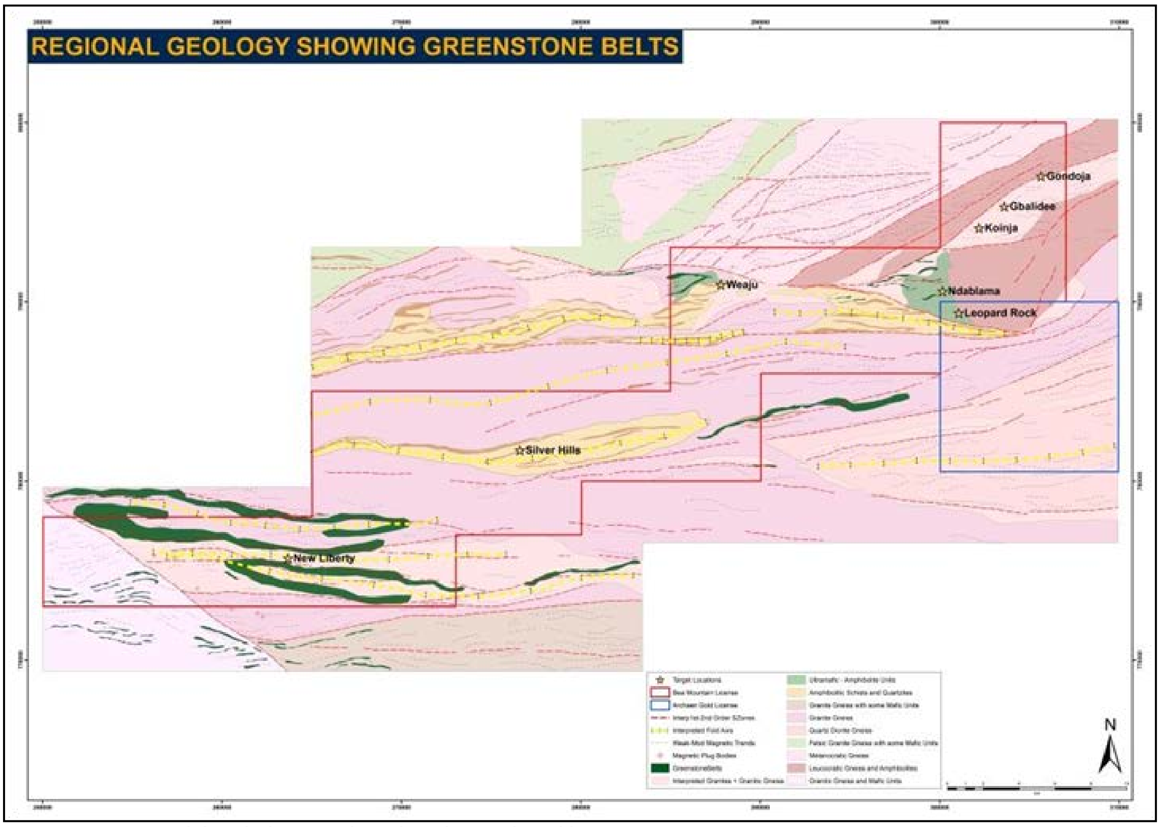
Source: *Hurley et al.*

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 Archean 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.

Figure 7.3 General Geology of the Bea-MDA Property Geology



Modified from: *Tysdal and Thorman (1983) and reproduced by Aureus 2013*

The Bea-MDA property contains several known areas of gold mineralization, typical of Upper Archean 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 Archean orogenic gold deposits described by Hagemann and Cassidy (2000), Richards and Tosdal (2001) Goldfarb, Groves and Gardoll (2001), Roberts et al (1998).

7.3 Geology of the Project

7.3.1 Stratigraphy

The Project is underlain by three main stratigraphic units (summarized in Table 7.1), which are further subdivided into minor zones of varying mineralogical assemblages. The geology is dominated by tremolite-chlorite-actinolite-talc ± magnetite rich meta-ultramafics, sometimes with phlogopite, and flanked by migmatitic gneisses.

Table 7.1 Simplified Stratigraphic Succession

Main Stratigraphic Zones	**Lithologies**
Hanging Wall Complex (HWC)	Mafic and felsic gneisses
Silicified Metamorphosed Ultrabasics (SMUS)	Ultramafic schist which hosts the mineralization. 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 gold mineralization, and generally contains quartz, chlorite and amphibole, and a host of mafic minerals, including talc.

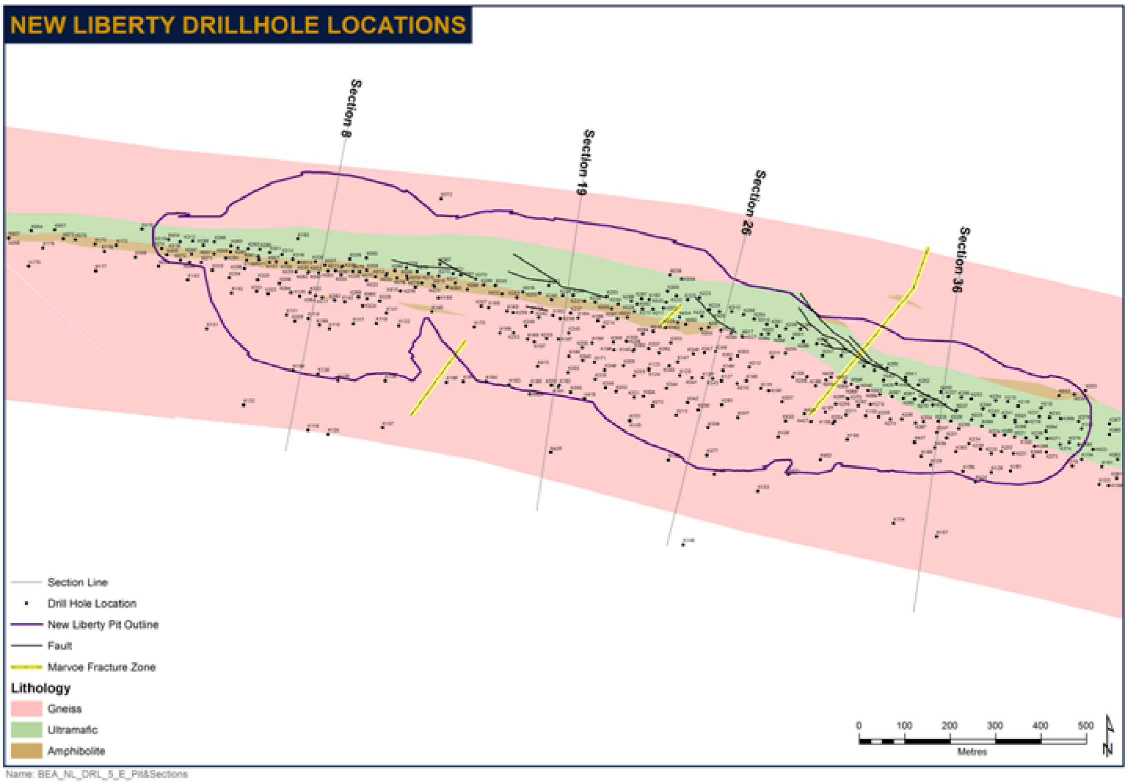
At the contact separating the HWC and FWC from the SMUS are transitional rocks, named here as garnet phlogopite ± actinolite gneiss (GNgp), which have a strong schistosity and coarse grain size (Figure 7.5). GNgp is also found within the ultramafic sequence.

Figure 7.6 shows a simplified plan view of the Project geology and Figure 7.7 a suite of cross-sections.

Figure 7.4 Hanging Wall Gneiss Complex (HWC)

Figure 7.5 Almandine Garnet Porphyroblasts in HWC

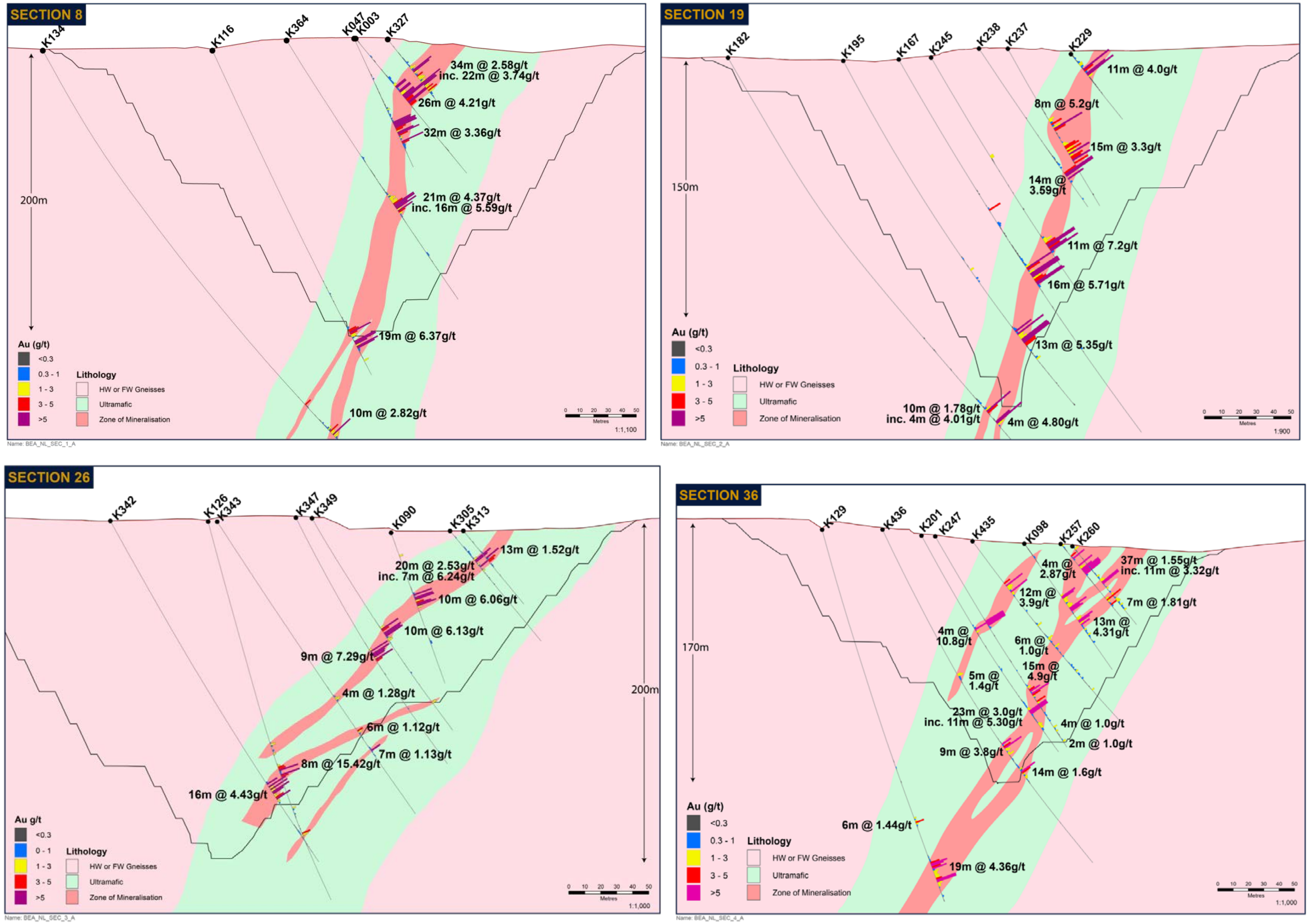
Figure 7.6 Project Geology



Source: *Aureus, 2013*

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 mineralization are not known at this stage.

Figure 7.7 Schematic South–North Cross-sections: New Liberty Geology



7.3.2 Structure

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

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

7.4 Alteration

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

7.5 Geology of Other Main Targets

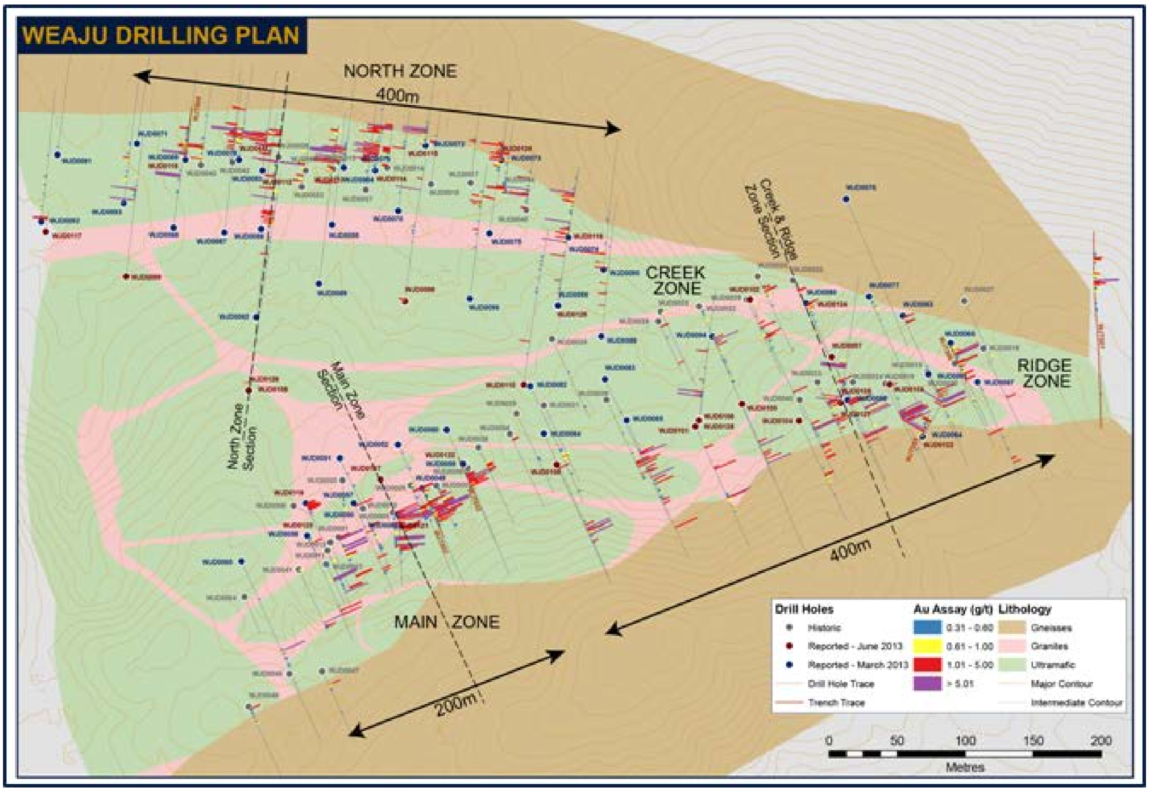
7.5.1 Weaju

The Weaju deposit is located at the eastern end of the Bea Mountain ridge (Figure 7.3 above) – the ridge is a prominent topographic feature comprising banded iron formation, ultramafic and mafic schists, meta-quartzites, paragneisses and intrusive granitoids.

Weaju consists of the same lithologies as the Project, with the addition of a magnetite tourmaline granite. The area is comprised of a major synform which plunges to the south west. The ultramafic body is surrounded by mafic and granitic gneisses and intruded by a prominent network of granites and pegmatites. Individual veins and patches of granite may, in places, reach up to 50 m in width (Figure 7.8).

Mineralization-related alteration includes silicification and a phlogopite-tourmaline-magnetite-carbonate assemblage, together with pyrrhotite, arsenopyrite, pyrite, chalcopyrite and niccolite. Gold mineralization at Weaju is found in tremolite talc chlorite schists adjacent to low magnetite content tourmaline granites. Free gold has also been observed within core samples. The mineralization is found with a strong westerly plunge.

Figure 7.8 Weaju Deposit



Source: *Aureus, 2013*

7.5.2 Gondoja, Koinja and Gbalidee

Gondoja (Figure 7.9), Koinja and Gbalidee lie within the north-east–south-west-trending structural corridor of the Property. The three targets represent separate en-echelon shear dilatational structures trending north-east, south-west, with Gondoja in the north, Gbalidee in the middle and Koinja furthest to the south-west. They all are represented by soil anomalies and have associated artisanal workings.

The area is geologically underlain by an approximately 200 m wide mafic body consisting of marginal amphibolite and a central zone of ultramafic rock. A series of quartz carbonate veins intrude mainly into the ultramafic unit, although one prominent quartz vein cuts through amphibolite close to the northern margin of the body. The quartz veins are closely related to granitized schists or greisens. Two moderate-sized (circa 100 m long) quartz veins, which intersect amphibolite, are mineralized with sphalerite, galena, scheelite, and gold with minor chalcopyrite. The geological setting of gold mineralization appears to be very similar to that at the New Liberty and at Weaju Projects. The principal differences are the veins and a polymetallic (Au-W ± Zn-Pb-Cu) metal assemblage at Gondoja. The association between the mineral assemblage and granitoid intrusion is comparable to intrusion-related gold mineralization systems.

Figure 7.9 Gondoja Trench and Drillhole Locations

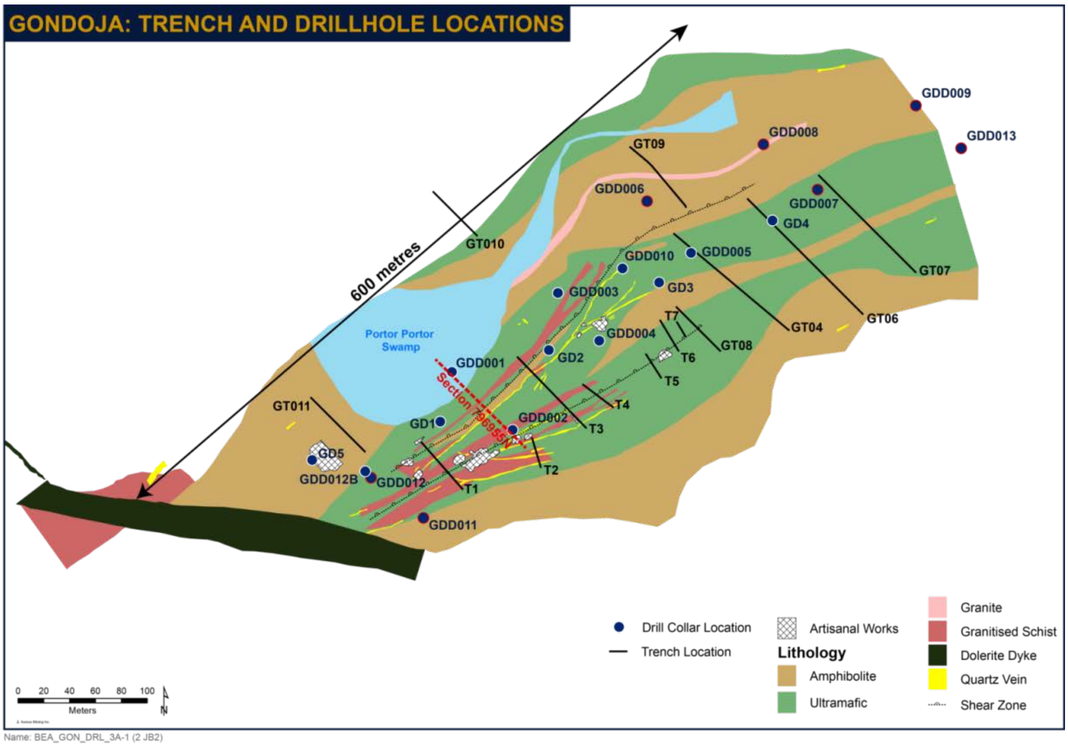
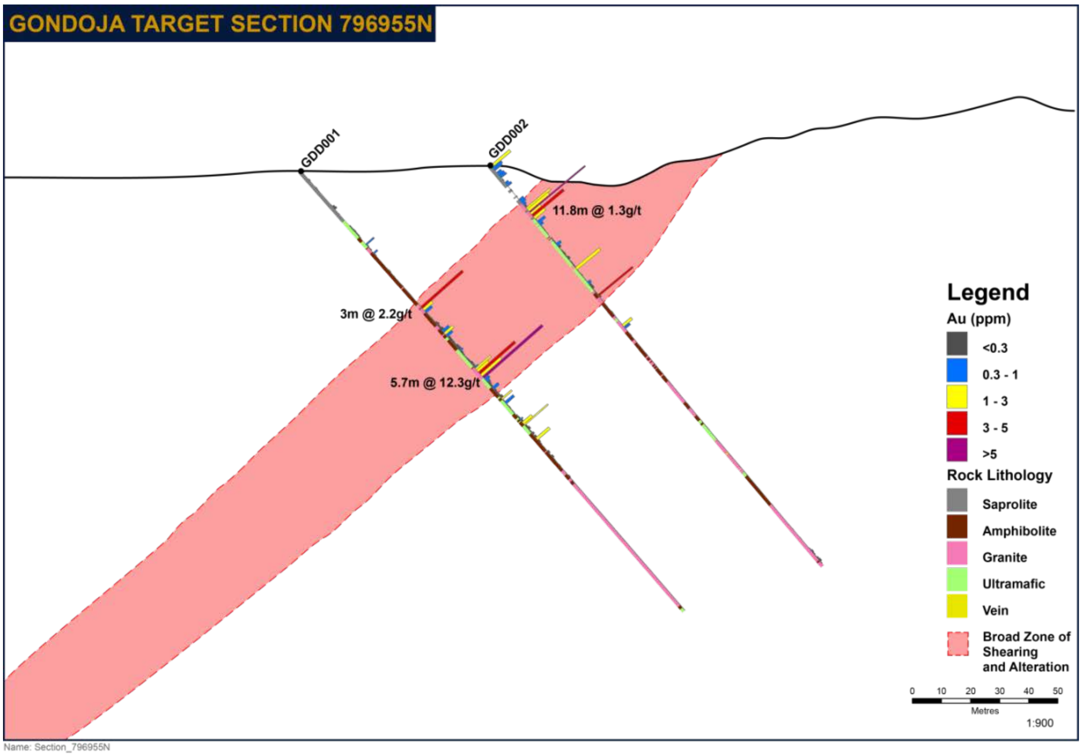


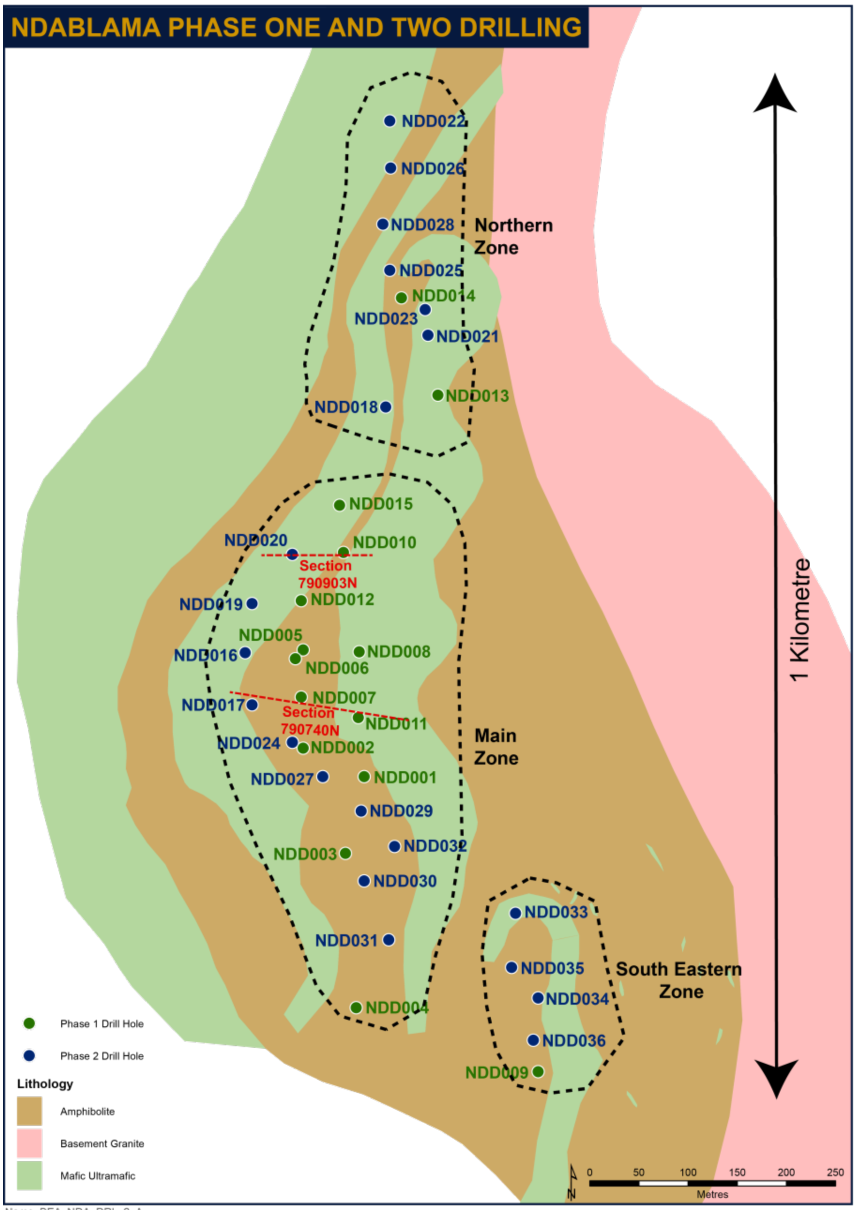
Figure 7.10 Gondoja Sections



7.5.3 Ndablama

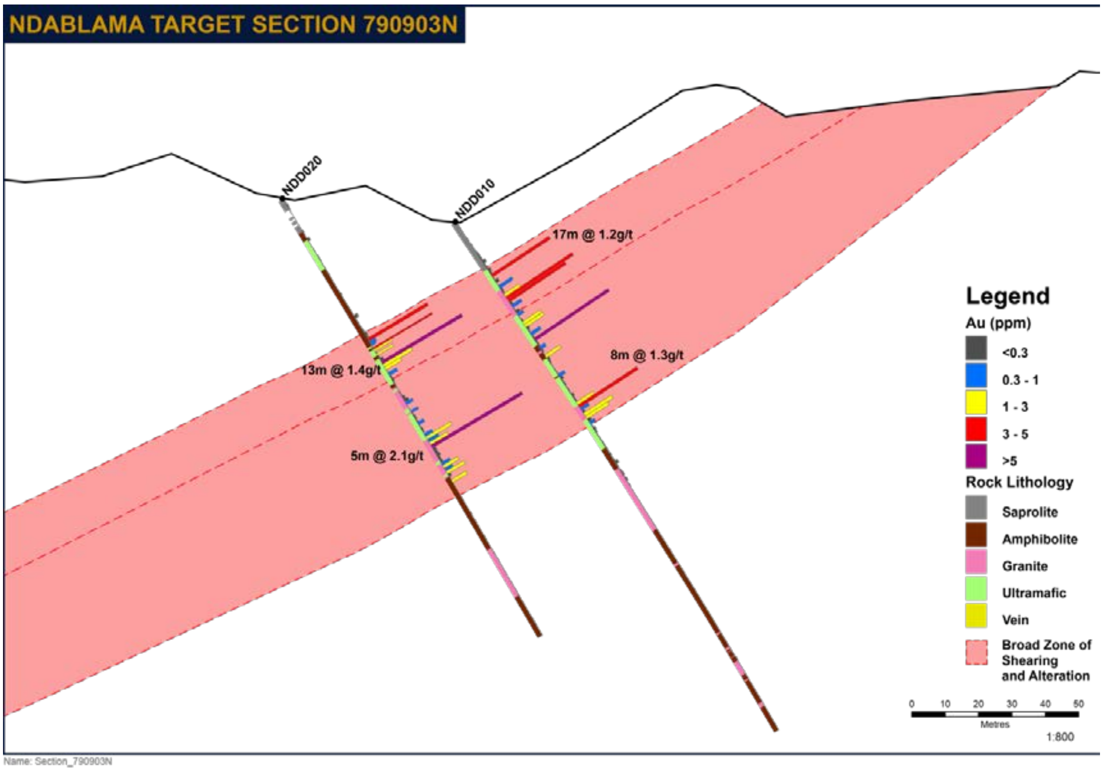
The Ndablama gold target is located approximately 40 km ENE of the Project and forms part of the Gondoja Hills, which are underlain by Archean amphibolitic gneisses and ultramafic rocks. The mineralized system strikes in a northerly direction and dips westwards at shallow angles ranging between 40° and 50°. The gold mineralization is associated with disseminated pyrite and pyrhotite, located within sheared and altered ultramafic and mafic rocks. Drillhole intercepts often occur at the sheared contact zones between the two rock types. The ultramafic and mafic rocks have been intruded by granitic dykes (Figure 7.11).

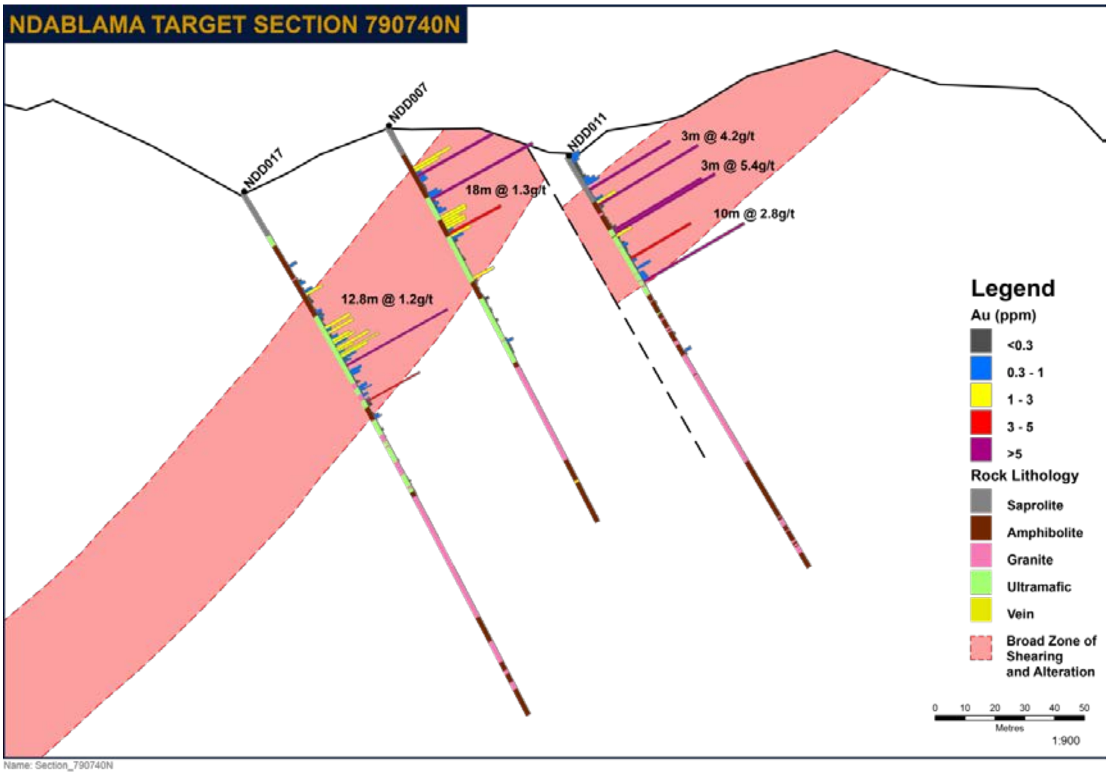
Figure 7.11 Geological Map of Ndablama



Source: *Aureus, 2012*

Figure 7.12 Ndablama Example Cross-sections





Source: *Aureus, 2012*

7.6 Mineralization

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

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

In Figure 7.13, pyrrhotite, arsenopyrite and pyrite are shown in cut and uncut ultramafic core, with the bulk of the sulphides aligned to the dominant cleavage.

Figure 7.13 Mineralization in Core

7.8 Summary of Field Character of the Mineralization

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

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

8 DEPOSIT TYPES

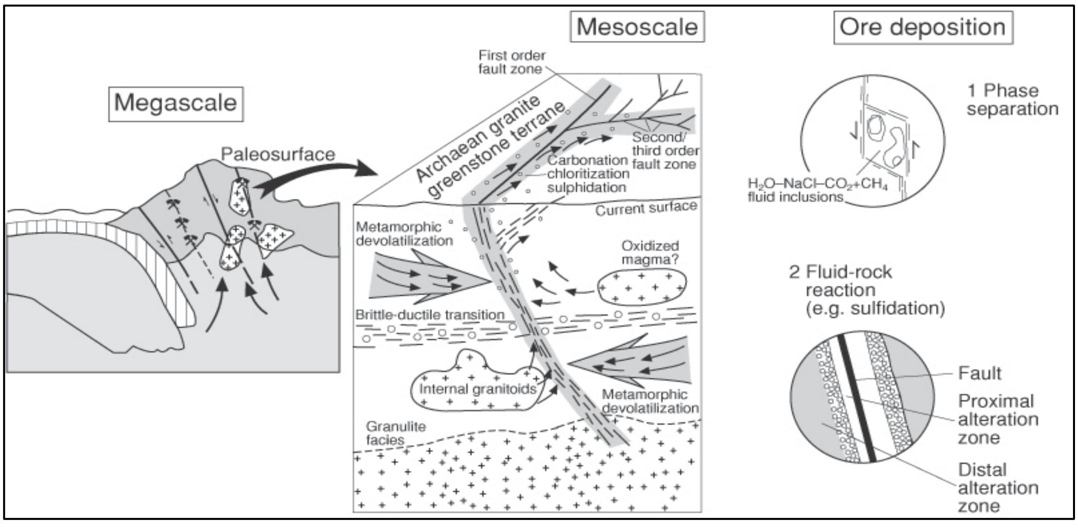
The Property hosts a typical Upper Archean to Lower Proterozoic style of metallogeny, characteristic of greenstone-hosted lode gold mineralization, where deposits are often referred to as orogenic, and characterized by the presence of gold-quartz veins and disseminated mineralization.

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

Mineralization in Archean orogenic deposits are associated with characteristic alteration styles (quartz-carbonate-sericite-biotite-sulphides) and often enriched in ‘lodes’ that plunge steeply. Gold deposits may occur in a variety of host rocks, which include granite, meta-volcanic rock (greenstones) and include mafic and ultramafic rock units and associated volcaniclastic, banded iron-formations and siliciclastic sediments, as observed within the Bea-MDA licence area. The schematic diagram (Figure 8.1) depicts a typical orogenic lode system with analogous geological settings for the deposit styles found on the Property.

The primary targets of Aureus’ mineral exploration programme in Liberia are shear zone-hosted gold systems, whether associated with quartz, granite veins, breccia zones or granitic bodies. A structural control to mineralization is eminent with areas of multiple structures intersecting. Gold mineralization 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.

Figure 8.1 Schematic of Orogenic Gold Systems



Modified from: *Hageman and Cassidy 2001*

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, some of which rank as world class, 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 at the Bea-MDA property follows a systematic process of reconnaissance work, grab-sampling followed by soil geochemistry, mapping, trench sampling and eventually drilling. Airborne and ground geophysics have also been conducted in situations where appropriate.

9.2 Methodologies

9.2.1 Coordinates, Datum, Grid Control and Topographic Surveys

At the Project, geological and geographical information was first set out on a local grid using a baseline at 285° magnetic, which parallels the strike of the mineralization. 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

Company geologists map lithology, alteration, mineralization and structures using outcrop mapping. Outcrop is limited mostly to artisanal pits and trenches; therefore maps are progressively updated as more data from trenches and drilling becomes available.

9.2.3 Regional Stream and Outcrop Sampling

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

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

9.2.4 Soil Geochemistry

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

9.2.5 Trenching

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

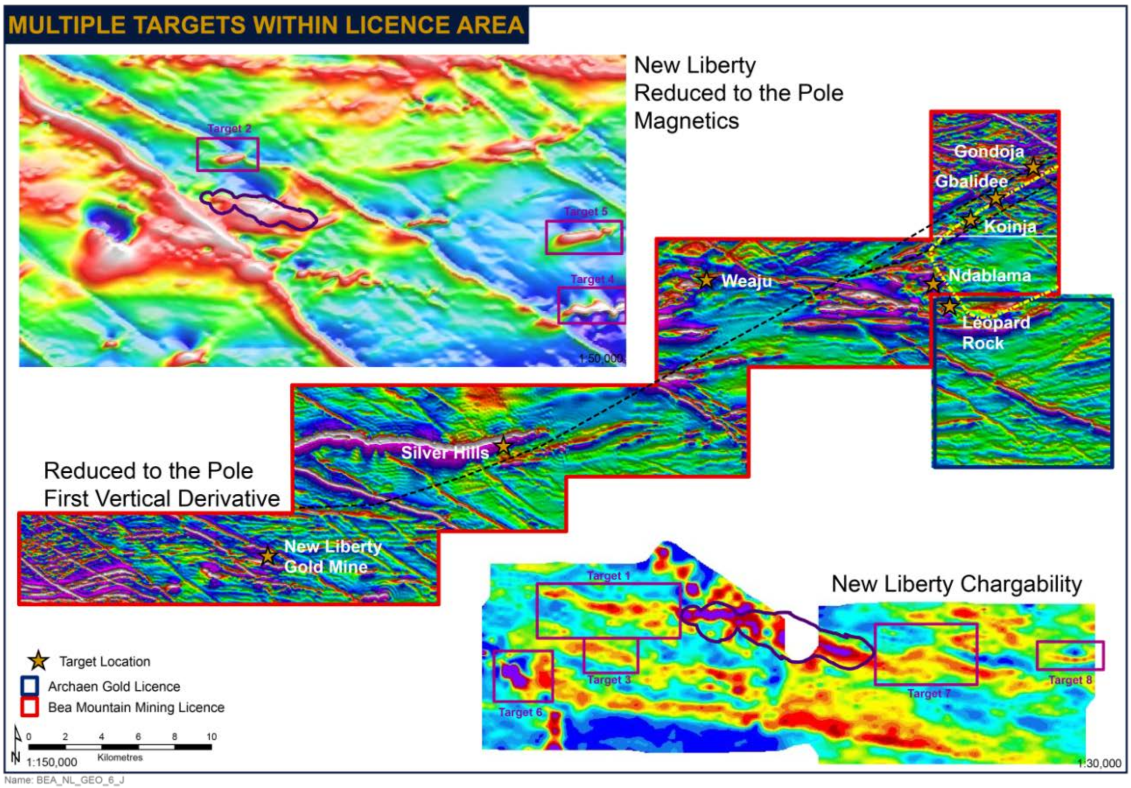
9.2.6 Geophysics

In May 2006, a high resolution helicopter-borne, combined magnetic gradient and gamma-ray spectrometer survey was conducted over the south-west and north-east sections of the licence area by New Resolution Geophysics (NRG). This was then complimented by a further survey, carried out by Geotech Airborne Limited in 2012 which covered the remainder of the Bea-MDA property, and the adjacent Archean licence which is also owned by Aureus. Sufficient overlap between the old and new survey and matching line spacing enable the surveys to be merged together. The survey parameters of both are summarized in Table 9.1. The datasets were merged by Geotech Airborne analysts and data quality control was undertaken by an independent consultant geophysicist. The radiometric spectrometry enables the demarcation of different lithology types, and the magnetics show both structure magnetic bodies, such as the ultramafic host rock at the New Liberty deposit (Figure 9.1).

Table 9.1 Comparisons of 2006 and 2012 Airborne Geophysical Surveys

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

Figure 9.1 New Liberty Geophysics Interpretation



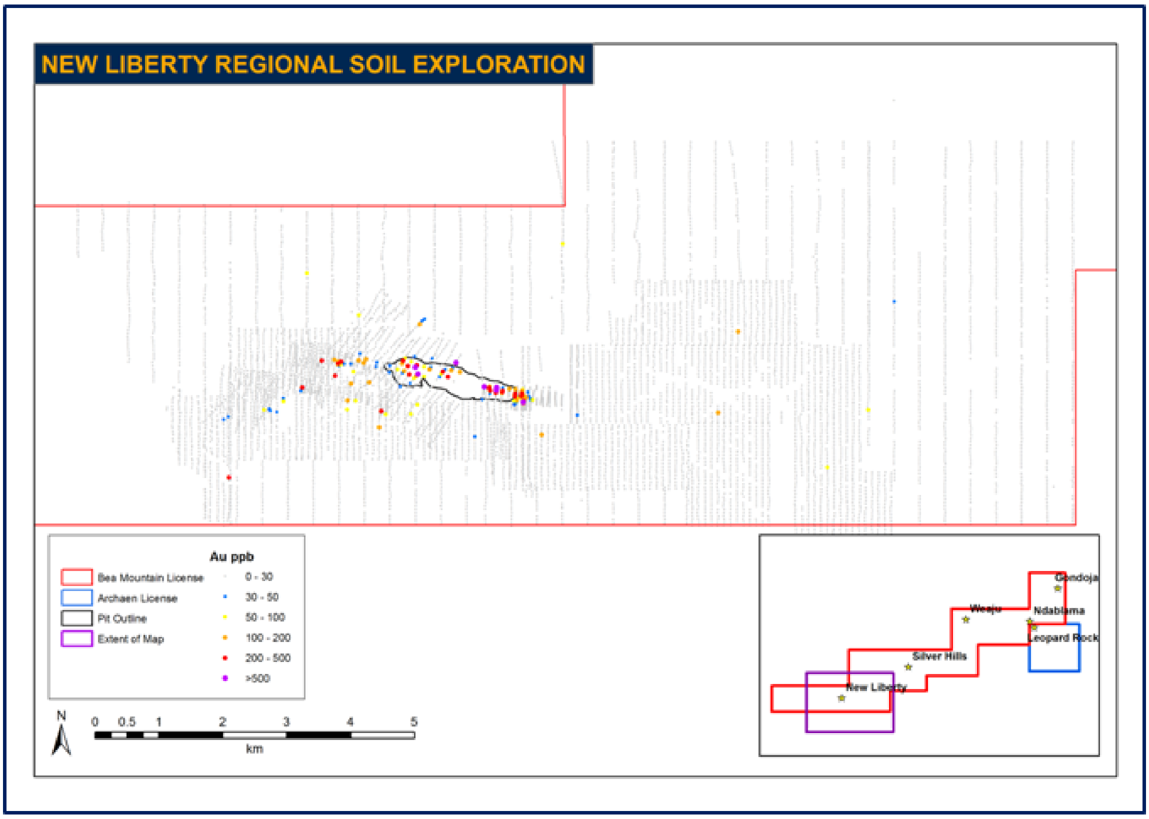
9.3 Regional Exploration Activity

9.3.1 New Liberty Exploration

9.3.1.1 Soil

Geochemical soil sampling in 1999 on a 100 m by 20 m grid over 1 km each side of the known mineralization detected a strong anomaly over 200 m to the west and east. Further along-strike soil sampling in 2011 and 2012 extended the areas surveyed to the east and west, in conjunction with geophysics and exploration. Sampling, covering 34.9 km^2 around the Project, was conducted on a 100 m × 50 m grid close to the deposit, and 500 m × 50 m further away (Figure 9.2).

Figure 9.2 Soil Sampling Coverage over the New Liberty Area



Source: *Aureus, 2013*

9.3.1.2 Trenching

Following encouraging channel samples from artisanal workings (Figure 9.3), including 19.95 m at 4.06 g/t Au in the west and 13.1 m at 4.56 g/t Au in the centre of the system, trenches T1–T12 were excavated in 1997, each 3 m deep trench aligned approximately perpendicular to the east-west strike of the mineralization, This covered an along-strike extent of 1,800 m (see trench example Figure 9.3). During 1998, trenches T13–T24 were completed at intervals of 100 m along the geological strike and 20 m–80 m long to depths ranging from 2.0 m to 4.0 m into saprolitic material (Figure 9.4). Later trenching (T27 and T28) was used for outcrop demarcation to assist in the positioning of borehole collars in poorly exposed terrain beyond the ultramafics and mineralization.

Further to this, during the 2012/2013 field season, a total of 29 trenches were dug across four key sites at New Liberty (totaling 3,241 metres, Figure 9.6). The trenches targeted anomalies represented by elevated soil gold and arsenic values coincident with geophysics anomalies.

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

Figure 9.3 Artisanal Workings

Figure 9.4 Exploration Trench

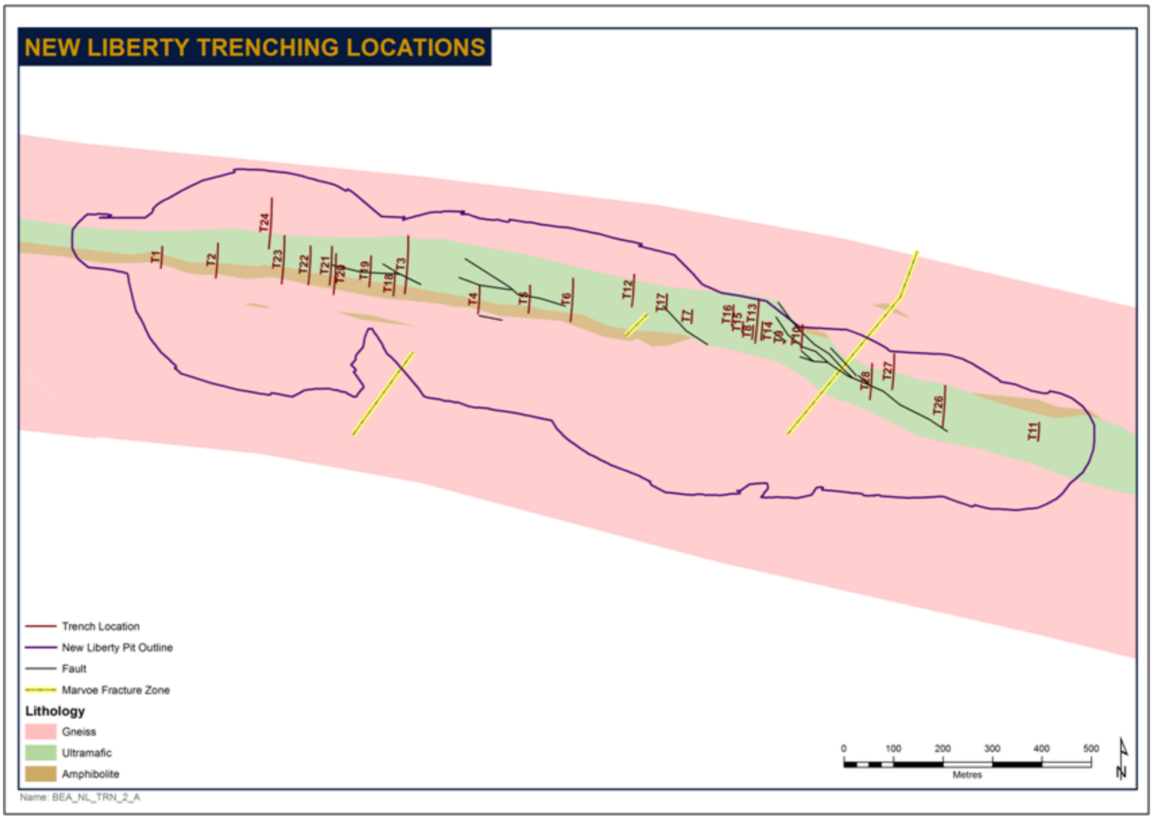
Source: *Aureus, 2012*

The trench channel sampling results, having interval values greater than 1.0 g/t Au, are shown in Table 9.2. Figure 9.5 presents the locations of the trenches.

Table 9.2 Summary of Trench Results

Trench Number	From (m)	To (m)	Au g/t
T2	9.0	20.0	11.0 @ 2.6
T3	22.0	23.0	1.0 @ 2.3
T5	29.0	31.0	2.0 @ 1.6
T5	41.0	42.0	1.0 @ 1.1
T5	43.0	44.0	1.0 @ 1.3
T6	26.0	35.0	9.0 @ 4.7
T6	41.0	45.0	4.0 @ 3.0
T7	3.0	4.0	1.0 @ 2.7
and	8.0	11.0	3.0 @ 1.8
and	16.0	28.0	12.0 @ 5.8
T8	0.0	2.0	2.0 @ 1.8
and	15.0	17.0	2.0 @ 1.4
T11	11.0	13.0	2.0 @ 1.8
and	15.0	18.0	3.0 @ 1.6
and	21.0	23.0	2.0 @ 2.7
T12	6.0	8.0	2.0 @ 3.4
and	40.0	42.0	2.0 @ 37.8
T13	16.0	20.0	4.0 @ 3.8
and	28.0	30.0	2.0 @ 2.2
T14	0.0	14.0	14.0 @ 1.5
T15	10.0	12.0	2.0 @ 1.6
T25	40.0	42.0	2.0 @ 1.4
T26	22.0	32.0	10.0 @ 8.6

Figure 9.5 Trench Locations

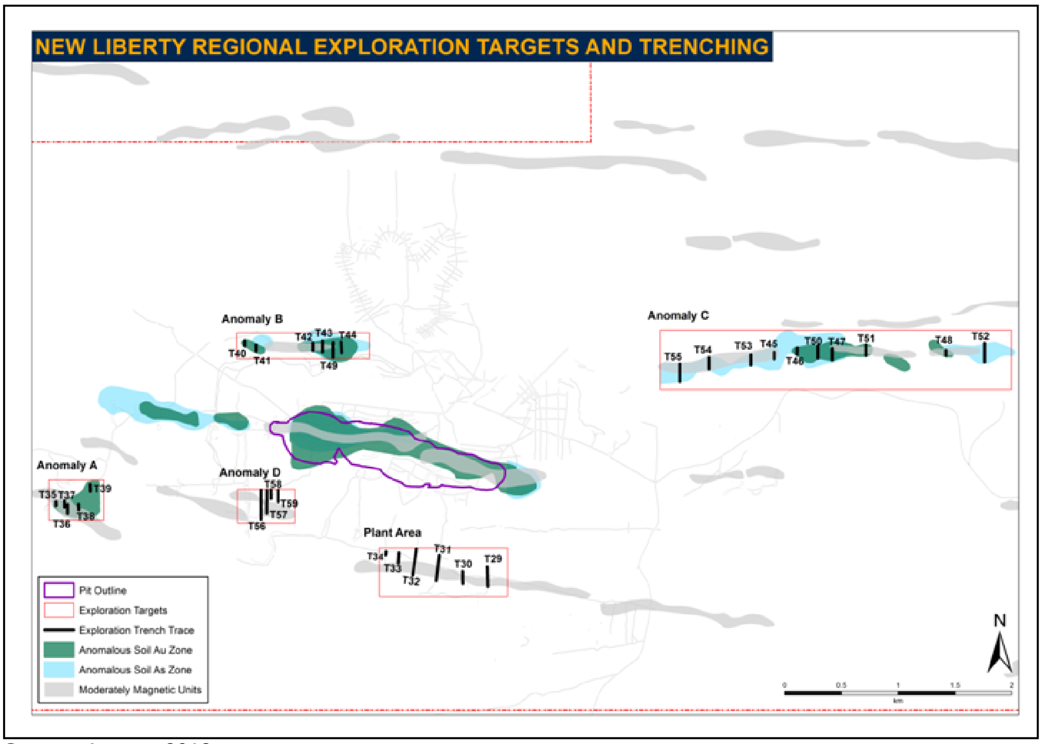


Source: *Aureus, 2013*

9.3.1.3 Geophysics

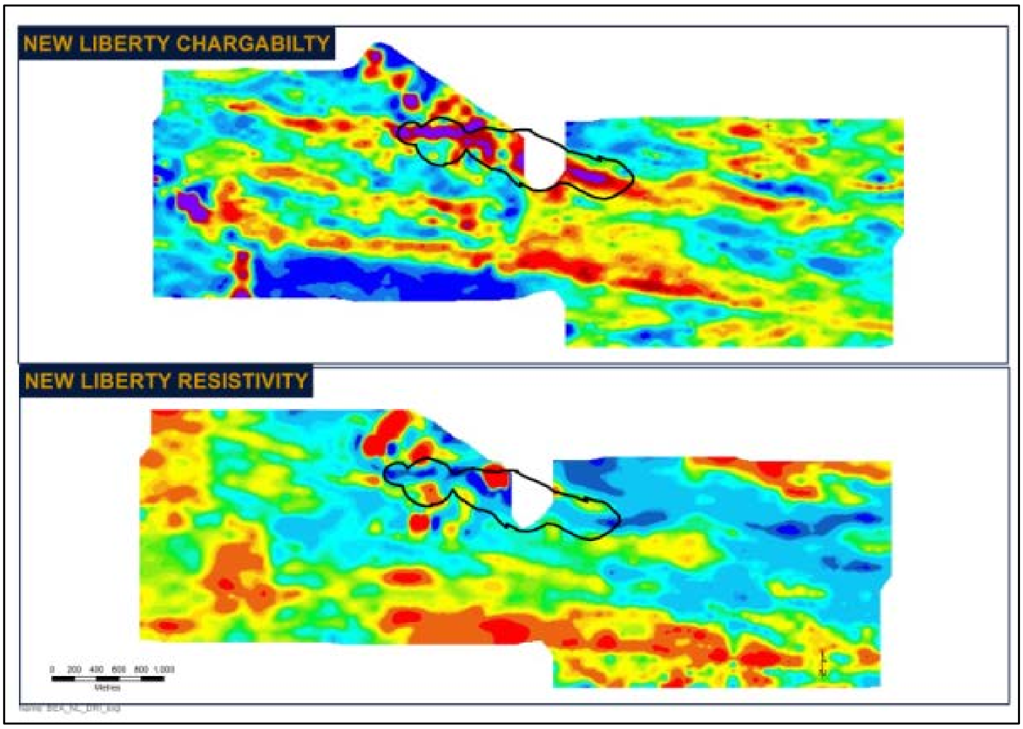
Following from the airborne survey, ground magnetic, induced polarization (IP) dipole-dipole lines and gradient array surveys have been undertaken by international geophysics survey company, Fugro, in 2011 and 2012. Initially, the areas of known mineralization were surveyed to gain an understanding of the signature of mineralization, with areas outside then used to extrapolate to other features. Further investigation is based on the airborne magnetic data, and along-strike from the mineralization. Fifty-two line kilometres of survey were completed for the ground magnetics and a further 15 km^2 for the IP grid and dipole-dipole. The IP detected a low resistivity corridor thought to represent a continuation of the mineralization within the ultramafic unit (Figure 9.7). Regionally, a further 1.8 km^2 has been completed to the south of the Ndablama target.

Figure 9.6 Trench Coverage Around the New Liberty Project



Source: *Aureus, 2013*

Figure 9.7 IP Corridor at New Liberty

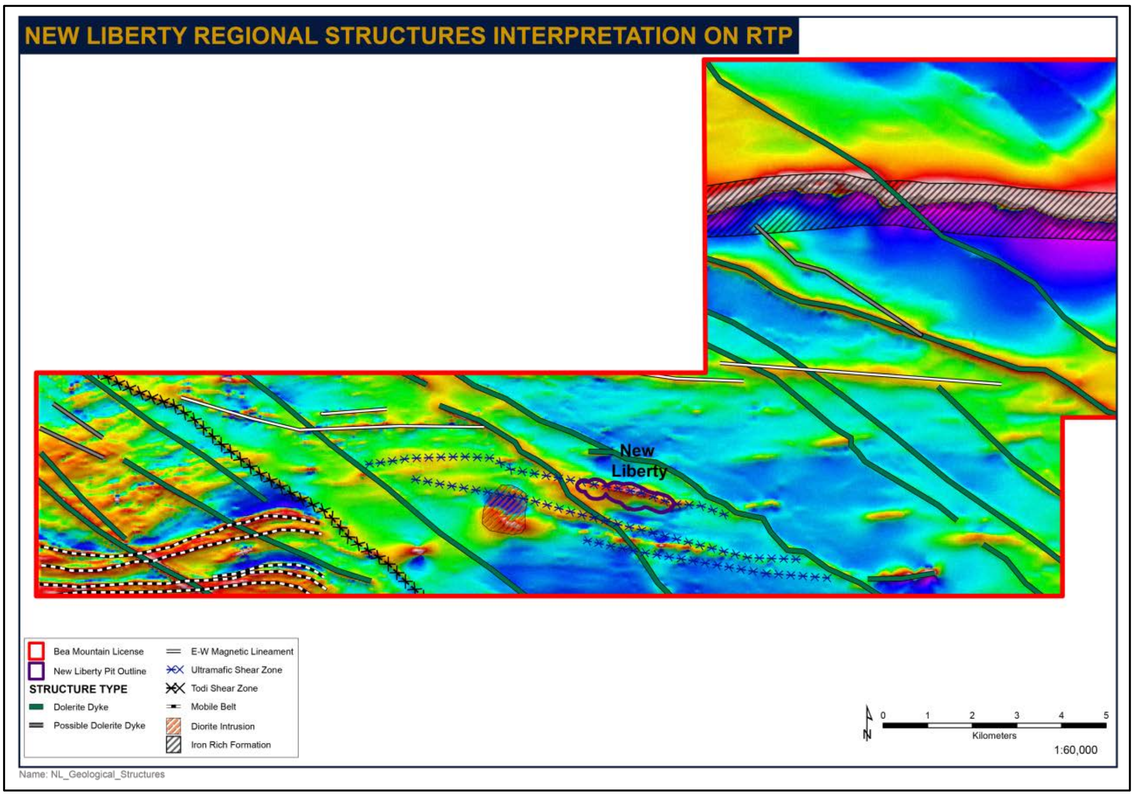


Source: *Aureus, 2012*

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 for further investigation (Figure 9.8). These are undergoing investigation with soil sampling, outcrop mapping and surveys to delineate potential targets for drilling.

Figure 9.8 Aerial Magnetics Targets



Source: *Aureus, 2012*

9.5 Other Targets in the Bea-MDA Property

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

Descriptions of exploration activities and results from these other targets have been provided by Aureus, but the associated data has not been reviewed by AMC, and the localities have not been visited. The information has been included here in the context disclosing other activities on the Bea-MDA property, but these are unrelated to the purpose of the report.

9.5.1 Silver Hills

Silver Hills is situated approximately 14 km north-east of the Project. Soil sampling results have highlighted a zone 80 m long and 30 m wide, which was followed up by the excavation of three trenches (Table 9.3).

Table 9.3 Silver Hills Trench Results

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

9.5.2 Weaju

The Weaju deposit is situated 30 km east-north-east of the Project, at the eastern end of the Bea Mountain ridge.

Mapping, supplemented by later drilling, indicates that mineralization is located within a sheared ultramafic host unit, folded into a synform bounded by gneisses and granite basement.

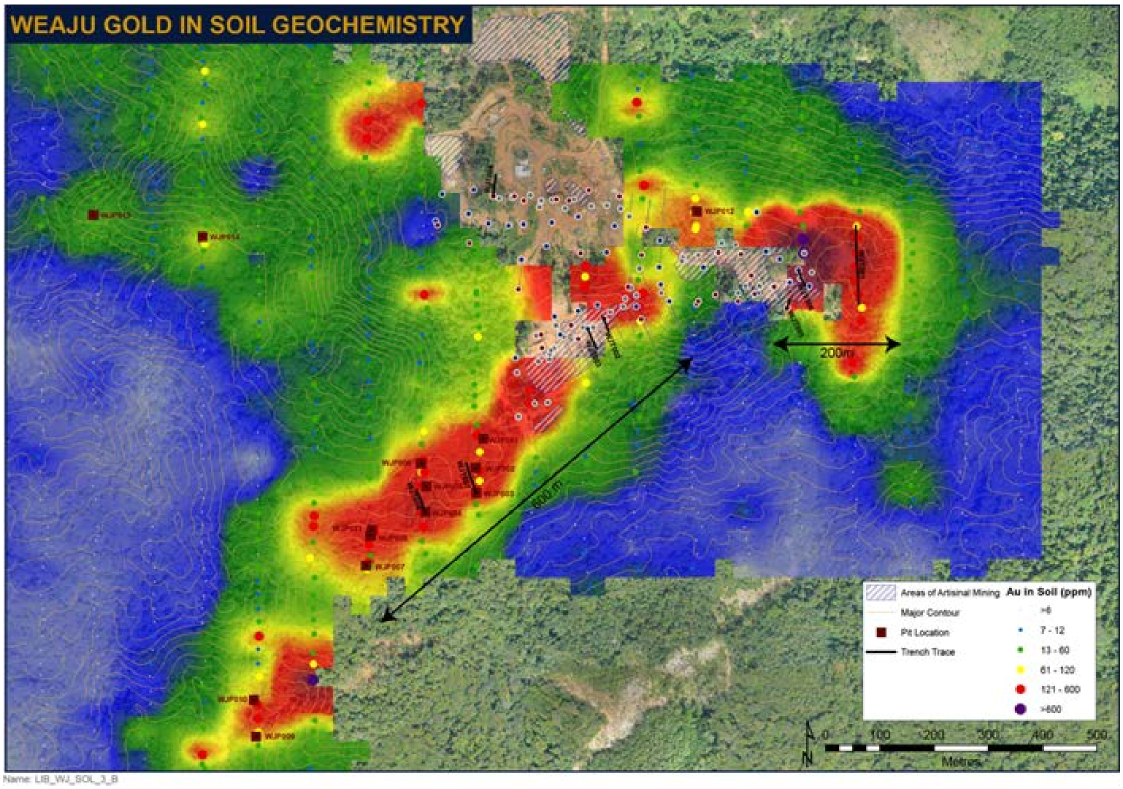
Soil geochemical data correlates well with the results from diamond drilling, and trenching and pitting, showing that the mineralization is found on both limbs with 1.5 km of anomalism on the southern limb (Figure 9.9). Further investigation by 12 trenches was undertaken, and during 2013 an additional 8 trenches were completed, showing that mineralization is hosted on two limbs of a fold.

Gondoja

The Gondoja area has been described in literature of the 1970s. It has been the subject of small-scale artisanal mining; and repeated visits in the period 1970-1980.

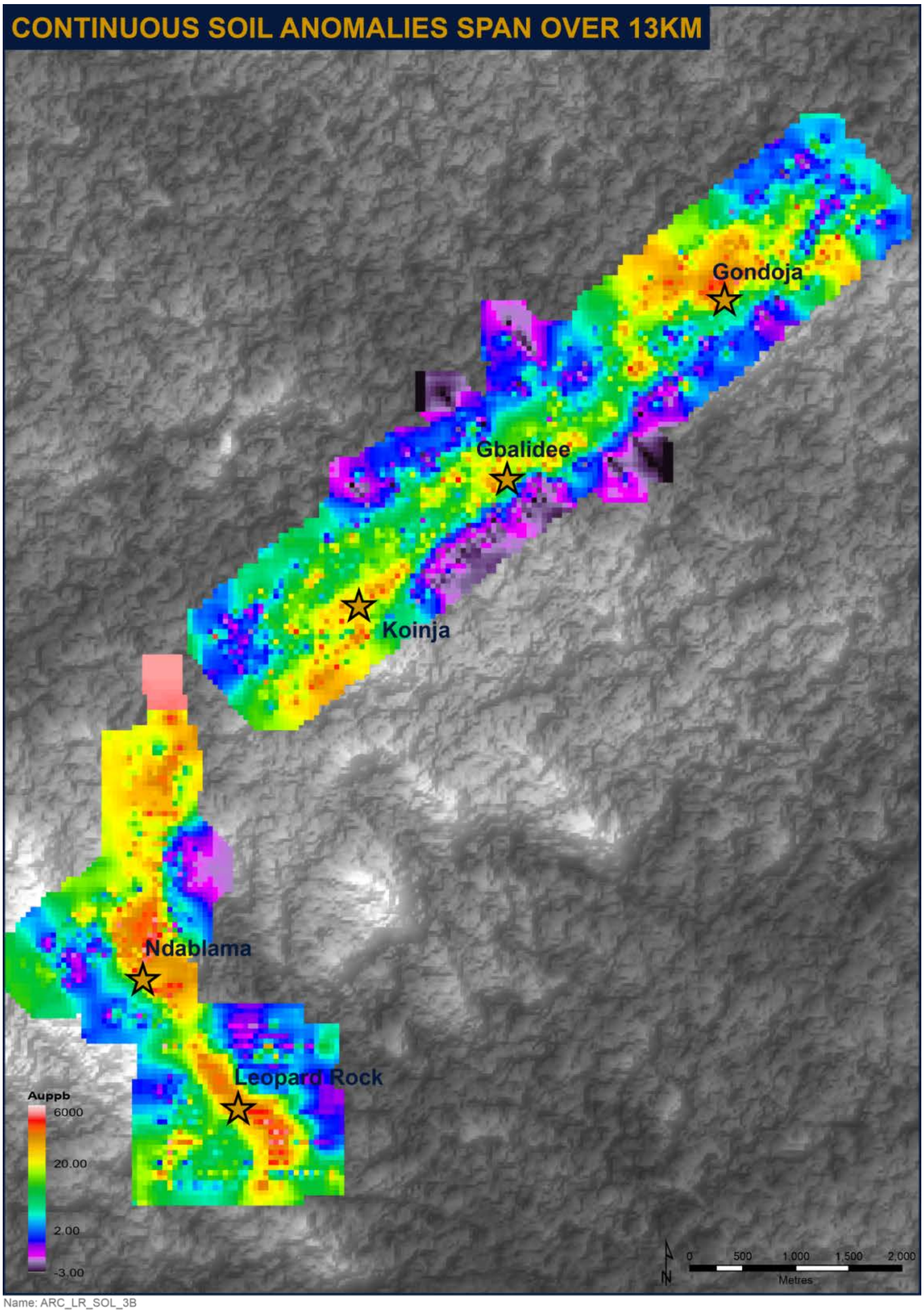
In addition to work conducted by ACA Howe in 2000 (Figure 9.10), quartz veins in a mafic body have been described and verified by fieldwork.

Figure 9.9 Weaju Soil Geochemistry



Source: *Aureus, 2013*

Figure 9.10 Regional Soils over the Gondoja Ndablama Targets



Source: *Aureus, 2012*

The mineralization is defined by a strong soil anomaly which is centred on an ultramafic unit cut by quartz veins and granitoid material. The mineralized footprint is 300 m long and 50 m-100 m wide. Eleven trenches have been completed at the site, the results of which are summarized in Table 9.4

Table 9.4 Trenches at Gondoja North

Trench ID	From (m)	To (m)	Intersection Length (m)	Au Grade(g/t)
GT04	37	40	3	0.4
and	47	62	15	1.7
including	50	62	12	2.0
and	63	89	26	3.5
including	63	67	4	19.2
GT08	11	49	38	1.2
T-7	-	-	15	4.2
GT10	4	9	5	0.7
and	25	27	2	0.3
T-4	-	-	30	2.2
T-3	-	-	64	1.0
T-2	-	-	20	1.9
T-1	-	-	50	1.4
GT11	8	14	6	1.2

Assay grade data is un-cut.
NSV -T5, T6, GT06, GT07 and GT09

9.5.3 Gbaldee and Koinja

Gbaldee is a set of soil anomalies, the largest being 500 m by 300 m, which runs north-east south-west connecting Ndablama and Gondoja (Figure 9.8). Further to the south another soil anomaly; Koinja has been identified.

9.5.4 Ndablama

A soil sampling survey detected a 1.2 km long, north-south-trending zone of gold enrichment, up to 100 m wide, that remains open along strike (Figure 9.9). Follow-up trenching along a 400 m long southern section of the anomaly (Trenches 1–13) was completed in 2010 (Table 9.5). This has been followed up by detailed mapping and the competition of 63 trenches totalling 3,967 m over 1.6 km of strike.

Table 9.5 Ndablama Trenching Significant Intercepts

Trench Values are Reported from North to South.				
Trench	**From (m)**	**To (m)**	**Length (m)**	**Mean Au g/t**
NT23	36	46	10	0.5
NT30	15	17	2	9.4
NT29	115	148	33	0.8
And	122	134	12	1.3
NT62	30	39	9	2.1
including	34	38	4	3.9
and	42	46	4	0.8
NT42	0	3	3	0.7
and	22	37	15	0.6
NT17	20	42	22	0.6
including	25	34	9	0.8
and	68	150	82	2.1
including	74	80	6	3.5
including	101	105	4	5.3
including	132	147	15	4.8
NT43	76	87	11	3.8
and	90	100	10	0.7
and	116	124	8	1.3
and	148	154	6	0.6
NT21	12	20	8	1.0
and	38	45	7	1.1
NT16	60	67	7	1.0
and	20	42	22	0.6
NT13	0	12	12	2.3
and	34	89	55	2.2
and	54	59	5	1.7
and	88	90	2	4.6
and	113	121	8	1.2
NT8	35	40	5	1.7
NT7	0	24	24	1.8
including	14	22	8	3
NT3	0	44	44	1.0
including	36	43	7	1.8
NT1	0	70	70	1.4
including	4	12	8	5.2
NT2	16	86	70	1.1
including	77	86	10	1.6
NT48	38	45	7	2.3
NT9	56	88	32	1.1
NT11	40	42	2	4.8
NT34	2	6	4	0.5
NT36	0	3	3	1.5
and	60	70	10	2.0
and	84	101	17	0.6
NT32	25	30	5	0.7
and	113	120	7	0.6
and	137	147	10	7.8
NT18	6	10	4	1.3
and	15	18	3	3.2
NT10	36	55	19	1.5

Assay grade data is un-cut.
The following trenches fell within the three zones but yielded NSV:
Trenches: NT031, NT033, NT035, NT037-041, NT044, NT051 and NT053-NT055, NT057-NT061 and NT063 are unmineralized.

9.5.5 Regional Targeting

As part of an ongoing exploration programme a geochemical and structural study of known areas of mineralization 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 Project Drill Programme

Diamond drilling at the Project was conducted periodically between 1999 and 2012 (Table 10.1). The total meterage drilled in the exploration of the Project is 65,187 m in seven campaigns.

Table 10.1 Summary of Drill Campaigns

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

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

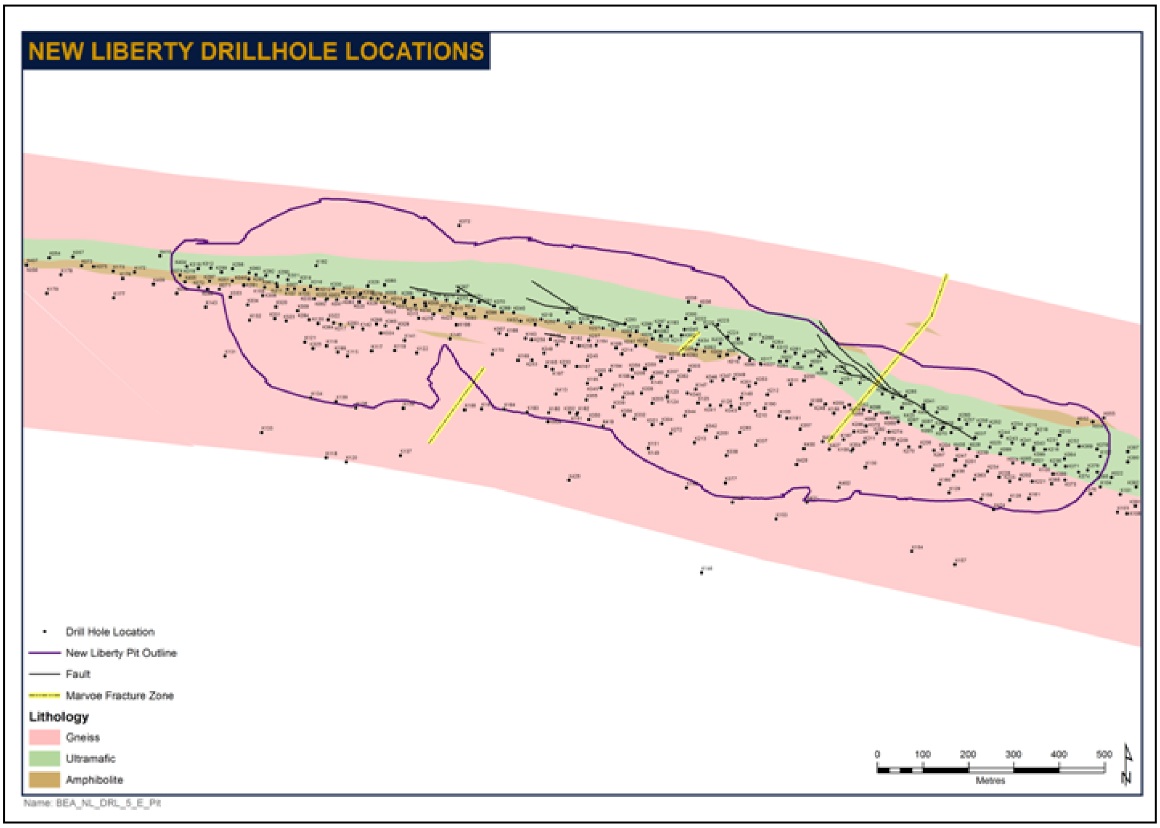
Drilling was conducted on a grid, with holes generally drilled on a 015° azimuth (magnetic) and inclined at between minus 45° and 70° to intersect the south-dipping zones. During drilling campaigns 6 and 7 a grid pattern was used. At times exceptions to the default bearing were introduced because of inaccessibility due to swampy conditions or the distance to the target depth exceeded the capability of the Onram 100 rigs. This occurred in the case of six boreholes, K10, K32, K34, K55 in the Marvoe zone and K36 and K38 in the Kinjor zone. In each case the back bearing of 195° was used. Figure 10.1 is a plan view illustration of drillhole locations in the four mineralized zones.

The core sizes drilled varied over time as well as within holes, as shown in Table 10.2. The quarter core from the first 27 diamond drillholes and half core for the remaining holes are stored on site. Figure 10.2 shows a view of the core storage facilities at the time of the Campaign 6 drilling.

Table 10.2 Drill Metres by Campaign

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

Figure 10.1 Location of Zones and Drilling



Source: *Aureus, 2013*

Figure 10.2 Core Sheds

10.2 Drill Programme Campaigns

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

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

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

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

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

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

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

The drilling programme was flexible and dynamic, allowing changes to be implemented during the programme based on feedback from site, assay results received and to account for practical issues such as positioning of drill pads (Figure 10.3). One outcome of this was the discovery of the Latiff Zone from wildcat borehole K144 in the gap between the Larjor and Kinjor zones, which led to the revised drilling across the gap.

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

Figure 10.3 Diamond Core Drill Rig

Figure 10.4 Drill Core Showing Recovery

10.3 Collar Coordinates

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

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

10.4 Downhole Surveys

Downhole surveying practices varied through the different drilling campaigns, and the database shows that surveys were not conducted in 96 of the 375 holes.

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

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

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

10.4.1 Acoustic Televiewer (ATV) Probe

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

The data gathered by the ATV probe was processed on-site, generating an orientated acoustic image of the borehole wall (Figure 10.5) and provided to AMC for interpretation. The spatial orientation of each structure was determined by the amplitude of the sinusoidal curve in relation to the inclination of the borehole.

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

Figure 10.5 Accoustic Image and Interpretation of ATV Survey

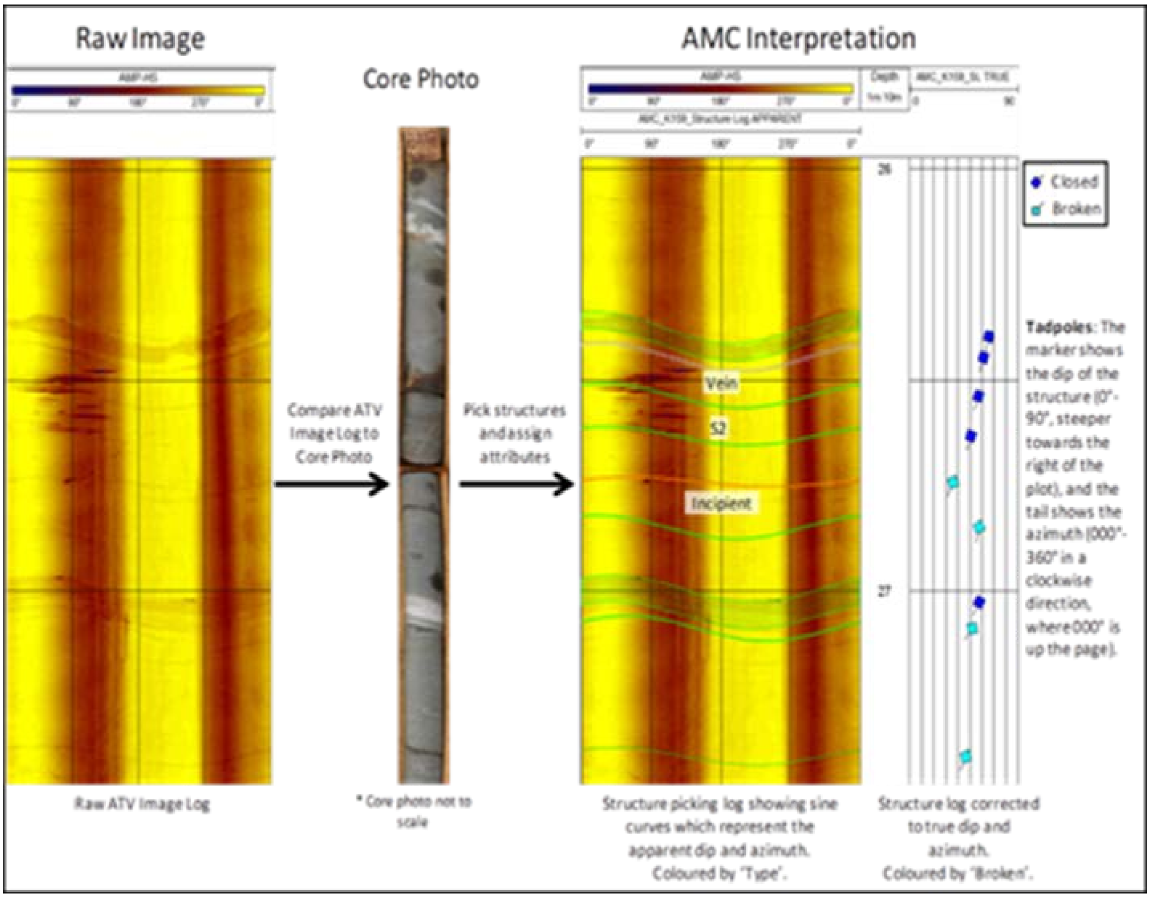


Table 10.3 details holes logged during the ATV Probe Survey:

Table 10.3 Holes Logged Using the ATV Probe

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

10.5 Core Recovery

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

Figure 10.4 shows good core recovery in spite of the tendency for mineralized rock competencies to be lower than in adjacent unmineralized intervals.

10.6 Significant Drill Intersections

Drilling is designed on cross-sectional fences and typically inclined at an average of 60°. Mineralized zone true widths would in general constitute about 75% of intersection lengths. In Table 10.4 the grades and widths of mineralized intersections are shown by zone. These intersections are consistent with those used in the resource estimation described in Section 14, but exclude intersections either with a grade of less than 1.0 g/t Au or a length of less than 2.0 m.

Table 10.4 Significant Intersections at New Liberty

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
Larjor	262275	K072	62.0	65.0	3.0	5.83
	262300	K046	91.0	97.0	6.0	2.99
	262300	K143	163.0	169.0	6.0	4.05
	262300	K287	46.0	53.0	7.0	3.83
	262325	K071	70.0	79.0	9.0	3.33
	262325	K299	4.0	12.0	8.0	2.76
	262425	K108	43.0	49.0	6.0	2.18
	262425	K292	14.0	19.0	5.0	1.34
	262425	K306	68.0	81.0	13.0	1.05
	262450	K290	4.0	14.0	10.0	1.26
	262450	K320	97.0	117.0	20.0	2.76
	262450	K331	146.0	172.0	26.0	1.95
	262475	K007	24.0	34.0	10.0	1.14
	262475	K107	32.0	45.0	13.0	1.05
	262475	K133	440.0	456.0	16.0	2.57
	262475	K301	4.0	18.0	14.0	1.96
	262475	K323	131.0	149.0	18.0	1.27
	262500	K309	111.0	128.0	17.0	1.40
	262500	K314	7.5	23.0	15.5	1.12
	262525	K012	12.0	28.0	16.0	1.26
	262525	K035	69.0	91.0	22.0	1.05
	262525	K121	182.0	195.0	13.0	1.26
	262525	K284	127.0	150.0	23.0	1.82
	262525	K316	12.0	32.0	20.0	1.98
	262550	K062	71.0	93.0	22.0	3.07
	262550	K134	331.0	347.0	16.0	1.88
	262550	K135	140.0	161.0	21.0	3.32
	262550	K325	203.0	212.0	9.0	1.50
	262550	K335	58.0	82.0	24.0	4.18
	262575	K003	36.0	62.0	26.0	4.21
	262575	K047	54.0	86.0	32.0	3.36
	262575	K116	218.0	237.0	19.0	6.37
	262575	K322	129.0	148.0	19.0	3.00
	262575	K364	129.0	147.0	18.0	4.90
	262600	K063	61.0	90.0	29.0	8.84
	262600	K139	358.0	364.0	6.0	2.44
	262600	K199	199.0	221.0	22.0	3.60
	262600	K317	146.0	155.0	9.0	5.81
	262600	K327	21.0	50.0	29.0	2.84

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
	262625	K013	20.0	46.0	26.0	6.17
	262625	K020	56.1	74.0	17.9	4.51
	262625	K081	55.0	73.0	18.0	4.29
	262625	K115	209.0	226.0	17.0	6.61
	262625	K120	517.0	525.0	8.0	2.53
	262625	K283	143.0	161.0	18.0	6.36
	262625	K336	26.0	61.0	35.0	5.59
	262650	K076	84.0	96.0	12.0	4.94
	262650	K136	378.0	389.0	11.0	4.57
	262650	K142	145.0	160.0	15.0	7.05
	262675	K008	26.0	48.0	22.0	3.91
	262675	K266	128.0	133.0	5.0	2.33
	262675	K326	35.0	63.0	28.0	3.38
	262700	K077	67.0	72.0	5.0	3.37
	262725	K014	22.0	34.0	12.0	7.47
	262725	K033	60.0	64.0	4.0	6.32
	262725	K141	154.0	159.0	5.0	1.56
	262725	K329	124.0	130.0	6.0	1.38
	262750	K078	63.0	71.0	8.0	3.70
	262750	K288	21.0	33.0	12.0	3.62
	262775	K009	36.0	44.0	8.0	1.60
	262775	K030	66.0	70.0	4.0	2.08
	262775	K279	7.0	18.0	11.0	4.36
	262800	K079	67.0	70.0	3.0	3.43
	262800	K276	78.0	80.0	2.0	2.39
	262800	K278	25.0	31.0	6.0	3.34
	262825	K015	46.0	48.0	2.0	2.84
	262825	K140	138.0	142.0	4.0	1.02
	262825	K275	30.0	36.0	6.0	2.43
Latiff	262925	K170	143.0	149.0	6.0	1.29
	262950	K168	73.0	81.0	8.0	3.06
	262950	K307	66.0	77.0	11.0	1.53
	263000	K163	80.0	88.0	8.0	8.53
	263000	K169	161.0	174.0	13.0	3.49
	263000	K258	69.0	81.0	12.0	3.00
	263025	K019	20.0	25.0	5.0	1.17
	263025	K253	129.0	141.0	12.0	4.98
	263025	K263	29.0	37.0	8.0	4.46
	263050	K144	68.0	80.0	12.0	3.76
	263050	K165	165.0	191.0	26.0	4.47

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
	263050	K249	112.0	125.0	13.0	6.80
	263075	K197	149.0	164.0	15.0	4.90
	263075	K242	88.0	98.0	10.0	4.31
	263075	K250	16.0	19.0	3.0	4.16
	263075	K408	285.0	290.0	5.0	3.80
	263075	K413	230.0	244.0	14.0	3.98
	263100	K162	80.0	93.0	13.0	6.81
	263100	K233	150.0	173.0	23.0	5.55
	263100	K240	28.0	34.0	6.0	12.95
	263125	K167	139.0	155.0	16.0	5.71
	263125	K229	6.0	17.0	11.0	4.05
	263125	K235	4.0	14.0	10.0	1.74
	263125	K237	47.0	77.0	30.0	3.07
	263125	K238	83.0	89.0	6.0	6.60
	263125	K352	254.0	268.0	14.0	1.23
	263125	K426	344.0	356.0	12.0	6.38
	263150	K164	74.0	88.0	14.0	7.20
	263150	K182	252.0	266.0	14.0	2.58
	263150	K195	183.0	196.0	13.0	5.35
	263150	K245	122.0	133.0	11.0	7.17
	263150	K350	256.0	269.0	13.0	3.15
	263150	K350	276.0	288.0	12.0	3.22
	263150	K355	226.0	240.0	14.0	4.58
	263175	K205	158.0	172.0	14.0	2.74
	263175	K227	22.0	36.0	14.0	1.19
	263175	K345	189.0	198.0	9.0	3.12
	263200	K171	163.0	171.0	8.0	5.35
	263200	K194	136.0	146.0	10.0	2.82
	263200	K339	222.0	228.0	6.0	2.11
	263200	K419	292.0	296.0	4.0	6.45
	263225	K005	50.0	54.0	4.0	2.53
	263225	K348	197.0	205.0	8.0	1.69
	263225	K356	227.0	231.0	4.0	4.33
	263225	K358	123.0	135.0	12.0	3.94
Kinjor	263250	K196	144.0	153.0	9.0	4.13
	263250	K230	21.0	23.0	2.0	1.10
	263250	K308	168.0	178.0	10.0	6.74
	263250	K332	223.0	228.0	5.0	1.28
	263250	K332	273.0	282.0	9.0	1.19
	263275	K145	130.0	136.0	6.0	3.26

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
	263275	K145	154.0	156.0	2.0	1.84
	263275	K151	260.0	265.0	5.0	5.76
	263275	K360	129.0	134.0	5.0	3.66
	263300	K094	67.0	75.0	8.0	6.94
	263300	K203	188.0	196.0	8.0	2.82
	263300	K203	227.0	229.0	2.0	1.18
	263300	K215	19.0	27.0	8.0	4.51
	263300	K304	255.0	260.0	5.0	9.68
	263300	K321	269.0	274.0	5.0	13.73
	263325	K016	56.0	62.0	6.0	4.35
	263325	K124	172.0	180.0	8.0	4.89
	263325	K124	193.0	199.0	6.0	2.00
	263325	K207	118.0	124.0	6.0	2.69
	263325	K207	133.0	141.0	8.0	3.49
	263325	K217	16.0	22.0	6.0	6.09
	263325	K302	0.0	16.0	16.0	2.61
	263350	K045	41.0	44.0	3.0	2.44
	263350	K093	29.0	42.0	13.0	3.50
	263350	K150	312.0	329.0	17.0	4.72
	263350	K272	283.0	285.0	2.0	1.00
	263350	K300	20.0	23.0	3.0	2.37
	263350	K303	128.0	133.0	5.0	2.60
	263350	K340	145.0	152.0	7.0	2.87
	263350	K362	126.0	134.0	8.0	5.08
	263350	K362	149.0	162.0	13.0	4.22
	263375	K092	25.0	29.0	4.0	2.05
	263375	K125	155.0	157.0	2.0	1.24
	263375	K147	130.0	138.0	8.0	2.74
	263375	K213	224.0	235.0	11.0	15.84
	263375	K222	31.0	36.0	5.0	2.66
	263375	K434	20.0	22.0	2.0	2.93
	263375	K434	66.0	85.0	19.0	1.88
	263400	K220	17.0	22.0	5.0	4.46
	263400	K342	191.0	207.0	16.0	4.48
	263400	K346	102.0	108.0	6.0	5.82
	263425	K006	28.0	34.0	6.0	6.25
	263425	K006	68.0	84.0	16.0	2.01
	263425	K091	29.0	39.0	10.0	5.63
	263425	K126	159.0	169.0	10.0	12.43
	263425	K126	204.0	206.0	2.0	3.72

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
	263425	K146	468.0	476.0	8.0	1.12
	263425	K200	213.0	217.0	4.0	5.18
	263425	K223	19.0	34.0	15.0	5.88
	263425	K347	95.0	104.0	9.0	7.27
	263425	K433	0.0	11.0	11.0	3.29
	263425	K433	43.0	67.0	24.0	3.82
	263450	K224	21.0	46.0	25.0	2.40
	263450	K305	17.0	33.0	16.0	2.92
	263450	K338	229.0	235.0	6.0	7.50
	263450	K343	132.0	136.0	4.0	1.34
	263450	K343	158.0	164.0	6.0	1.12
	263450	K349	81.0	91.0	10.0	6.02
	263450	K377	288.0	295.0	7.0	5.85
	263475	K018	38.0	46.0	8.0	4.26
	263475	K090	40.0	50.0	10.0	6.06
	263475	K127	137.0	150.0	13.0	2.78
	263475	K148	115.0	120.0	5.0	1.17
	263475	K152	290.0	296.0	6.0	4.29
	263475	K152	368.0	370.0	2.0	1.18
	263475	K285	177.0	182.0	5.0	3.35
	263500	K017	38.0	46.0	8.0	8.32
	263500	K027	44.0	52.0	8.0	3.96
	263500	K210	158.0	163.0	5.0	2.11
	263500	K313	22.0	30.0	8.0	2.26
	263500	K351	91.0	99.0	8.0	3.86
	263500	K353	82.0	87.0	5.0	1.76
	263525	K190	120.0	129.0	9.0	5.36
	263525	K264	0.0	9.0	9.0	5.33
	263525	K337	199.0	210.0	11.0	3.68
	263550	K155	117.0	124.0	7.0	2.89
	263575	K002	24.0	40.0	16.0	8.22
	263575	K191	118.0	129.0	11.0	5.80
	263575	K261	17.0	24.0	7.0	4.79
	263575	K310	12.0	22.0	10.0	6.29
	263600	K153	338.0	367.0	29.0	3.48
	263600	K153	380.0	387.0	7.0	3.45
	263600	K259	20.0	29.0	9.0	1.54
	263600	K261	28.0	30.0	2.0	3.50
	263600	K357	132.0	139.0	7.0	1.90
	263600	K428	185.0	215.0	30.0	1.53

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
	263625	K430	147.0	160.0	13.0	6.14
	263650	K248	116.0	121.0	5.0	3.25
	263650	K269	13.0	19.0	6.0	1.69
	263650	K429	143.0	153.0	10.0	3.53
	263650	K429	164.0	174.0	10.0	1.07
	263650	K431	275.0	323.0	48.0	2.28
	263675	K427	158.0	182.0	24.0	12.04
	263675	K427	184.0	189.0	5.0	1.02
Marvoe	263700	K059	76.0	79.0	3.0	1.05
	263700	K187	136.0	150.0	14.0	2.84
	263700	K187	159.0	163.0	4.0	1.19
	263700	K251	34.0	39.0	5.0	1.07
	263700	K402	245.0	269.0	24.0	1.76
	263700	K402	276.0	279.0	3.0	1.72
	263700	K438	70.0	72.0	2.0	3.02
	263725	K156	156.0	170.0	14.0	4.13
	263725	K294	117.0	124.0	7.0	7.74
	263725	K354	147.0	156.0	9.0	6.49
	263750	K052	47.0	61.0	14.0	6.81
	263750	K068	62.0	70.0	8.0	5.83
	263750	K096	54.0	62.0	8.0	16.37
	263750	K211	159.0	170.0	11.0	12.68
	263750	K211	197.0	208.0	11.0	1.03
	263750	K286	161.0	174.0	13.0	2.12
	263750	K375	96.0	100.0	4.0	2.67
	263775	K049	47.0	57.0	10.0	10.05
	263775	K069	60.0	74.0	14.0	6.44
	263775	K282	139.0	159.0	20.0	1.62
	263825	K066	85.0	93.0	8.0	2.07
	263850	K425	39.0	61.0	22.0	1.45
	263875	K067	22.0	24.0	2.0	1.40
	263875	K154	411.0	437.0	26.0	1.38
	263875	K206	103.0	136.0	33.0	1.83
	263900	K095	11.0	24.0	13.0	7.44
	263900	K095	52.0	82.0	30.0	1.02
	263900	K160	140.0	142.0	2.0	2.96
	263900	K204	61.0	63.0	2.0	1.15
	263900	K267	99.0	138.0	39.0	3.72
	263900	K361	4.0	7.0	3.0	2.65
	263900	K437	154.0	184.0	30.0	1.10

Zone	Section (Northing)	BHID	From (m)	To (m)	Length (m)	Au (g/t)
	263925	K039	38.0	72.0	34.0	1.26
	263925	K098	41.0	75.0	34.0	2.69
	263925	K129	224.0	246.0	22.0	3.82
	263925	K204	102.0	145.0	43.0	1.33
	263925	K361	42.0	91.0	49.0	2.89
	263950	K247	64.0	70.0	6.0	35.48
	263950	K247	114.0	139.0	25.0	3.05
	263950	K260	25.0	54.0	29.0	1.89
	263950	K435	34.0	45.0	11.0	4.22
	263950	K436	106.0	111.0	5.0	1.43
	263950	K436	159.0	190.0	31.0	1.98
	263975	K201	69.0	73.0	4.0	10.88
	263975	K201	127.0	150.0	23.0	2.96
	263975	K257	10.0	53.0	43.0	2.49
	264000	K256	6.0	38.0	32.0	2.68
	264000	K363	117.0	125.0	8.0	4.10
	264025	K158	171.0	230.0	59.0	1.77
	264025	K252	0.0	15.0	15.0	1.42
	264050	K244	8.0	34.0	26.0	2.29
	264075	K024	56.0	92.0	36.0	3.16
	264075	K225	111.0	166.0	55.0	1.13
	264100	K202	95.0	151.0	56.0	2.36
	264100	K241	38.0	46.0	8.0	1.72
	264100	K399	22.0	42.0	20.0	3.61
	264125	K021	53.0	74.8	21.8	2.72
	264125	K216	21.0	25.0	4.0	3.09
	264125	K221	98.0	142.0	44.0	3.02
	264150	K100	50.0	71.0	21.0	1.51
	264150	K236	45.0	76.0	31.0	2.76
	264150	K366	65.0	93.0	28.0	3.54
	264150	K368	92.0	116.0	24.0	1.39
	264175	K232	2.0	15.0	13.0	3.74
	264200	K369	23.0	31.0	8.0	1.16

10.6.1 Sterilization Drilling

Within the Project area, 6,810 m of sterilization drilling has been completed. .The 2013 drilling phase consisted of 12 diamond drill holes beneath the new plant site, the revised waste dump footprint and the new tailings dam. The details are shown in the table below, which also lists the details of the previous sterilization drilling (Table 10.5).

Table 10.5 New Liberty Sterilization Drilling

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

10.7 Drilling at Other Targets

Outside of the Project, drilling has been carried out on three of the four targets discussed in Section 9, namely the Ndablama, Weaju and Gondoja targets.

10.7.1 Ndablama

During 2011 and 2012 a 36 holed diamond core drilling programme, totalling 6,012 m was completed. At Ndablama, drilling and trench results have outlined multiple gold intercepts associated with three mineralized zones referred to as the North, Main and SE zones. In each zone the mineralized system strikes in a northerly direction and dips westwards at shallow angles ranging between 40° and 50° degrees. The diamond drilling cores demonstrate that the gold mineralization is associated with disseminated pyrite and pyrhotite, located within sheared and altered ultramafic and mafic rocks. Intercepts often occur at the sheared contact zones between the two rock types. Selected drill intersections are summarized in Table 10.6.

Table 10.6 Ndablama Drilling Intersections

Borehole ID	From (m)	To (m)	Intersection Length (m)	Au g/t	Mineralized Zone
NDD026	26	31	5	0.5	Northern
and	36	38.7	2.7	0.7	Northern
NDD028	27	29	2	1.4	Northern
NDD025	16	35	19	1.1	Northern
including	18	22	4	3.5	Northern
and	39	45	6	1.8	Northern
and	48	53	5	0.7	Northern
and	60	69	9	0.7	Northern
NDD14	4	12	8	1.1	Northern
NDD023	0	2	2	0.7	Northern
NDD021	0.8	3	2.2	0.6	Northern
NDD018	35	38.7	3.7	2.0	Northern
and	44	46.5	2.5	1.4	Northern
NDD10	26	43	17	1.2	Main
and	65	73	8	1.3	Main
NDD020	49	62	13	1.4	Main
including	57	60	3	2.9	Main
and	82.2	87.2	5	2.1	Main
and	92.2	99.2	7	0.7	Main
NDD12	24	38	14	2.9	Main
NDD019	51	61	9	0.8	Main
and	66	71	5	0.8	Main
NDD016	39	42	3	0.5	Main
and	49.5	60.5	11	7.1	Main
NDD06	28	54	26	0.5	Main
and	64	69	5	2.6	Main
and	90	99	9	1.6	Main
NDD08	0	3	3	1.1	Main
NDD07	16	22	6	1.8	Main
and	26	44	18	1.3	Main
NDD017	39	41	2	0.9	Main
and	53	65.8	12.8	1.2	Main
and	67.6	72	4.4	2.3	Main
and	76	83.8	7.8	0.6	Main
NDD11	17	20	3	4.2	Main
and	29	32	3	5.4	Main
and	40	50	10	2.8	Main
NDD024	9	14	5	0.6	Main
NDD02	14	30	16	2.4	Main

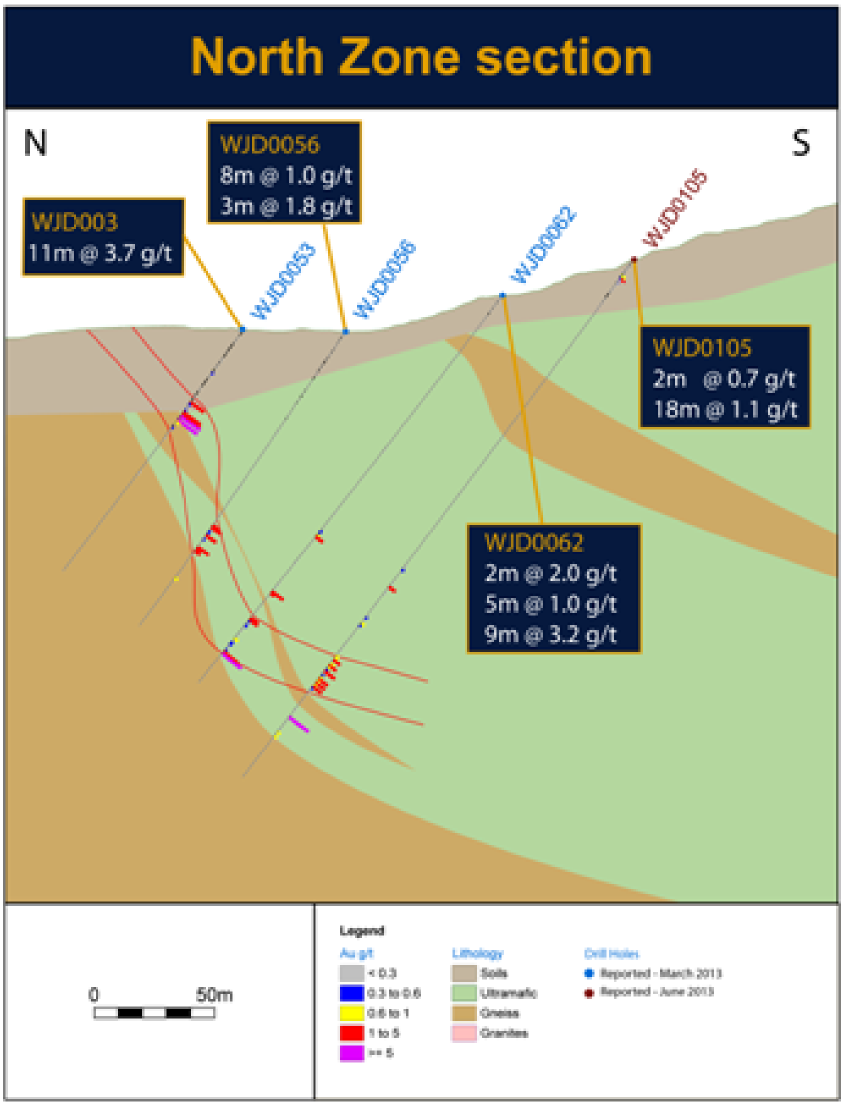
Borehole ID	From (m)	To (m)	Intersection Length (m)	Au g/t	Mineralized Zone
and	38	54	16	1.2	Main
and	64	67	3	8.7	Main
NDD027	17	25	8	0.9	Main
and	32	47	15	1.6	Main
and	50	59	9	0.9	Main
and	64	67	3	1.4	Main
NDD01	2	9	7	1.4	Main
and	49	53	4	1.4	Main
and	56	60	4	2.8	Main
NDD029	10	12	2	1.1	Main
and	23	28	5	1.2	Main
and	42	45	3	4.3	Main
and	47	51	4	1.9	Main
NDD032	12	17	5	0.8	Main
and	23	25	2	0.7	Main
NDD03	20	28	8	6.0	Main
and	45	49	4	1.9	Main
NDD030	23.7	27	3.3	4.8	Main
and	43.7	47	3.2	0.6	Main
and	58	61	3	0.5	Main
NDD031	12.6	14	1.4	4.7	Main
and	25.5	27	1.5	0.7	Main
and	31	37	6	0.6	Main
NDD04	26	28	2	1.7	Main
and	42	54	12	0.6	Main
NDD033	13.3	15	1.7	2.1	South Eastern
and	28	32	4	3.2	South Eastern
and	38	43	5	0.6	South Eastern
NDD035	42	47	5	2.0	South Eastern
NDD034	28.4	37	8.6	0.6	South Eastern
NDD036	30.8	33	2.2	7.6	South Eastern
and	41	43	2	1.5	South Eastern
NDD09	15	23	8	0.7	South Eastern
and	31	41	10	2.3	South Eastern

All drillhole values are reported from North to South within the Ndablama target
Assay grade data is un-cut
NSV - NDD005, NDD013, NDD015, NDD022

10.7.2 Weaju

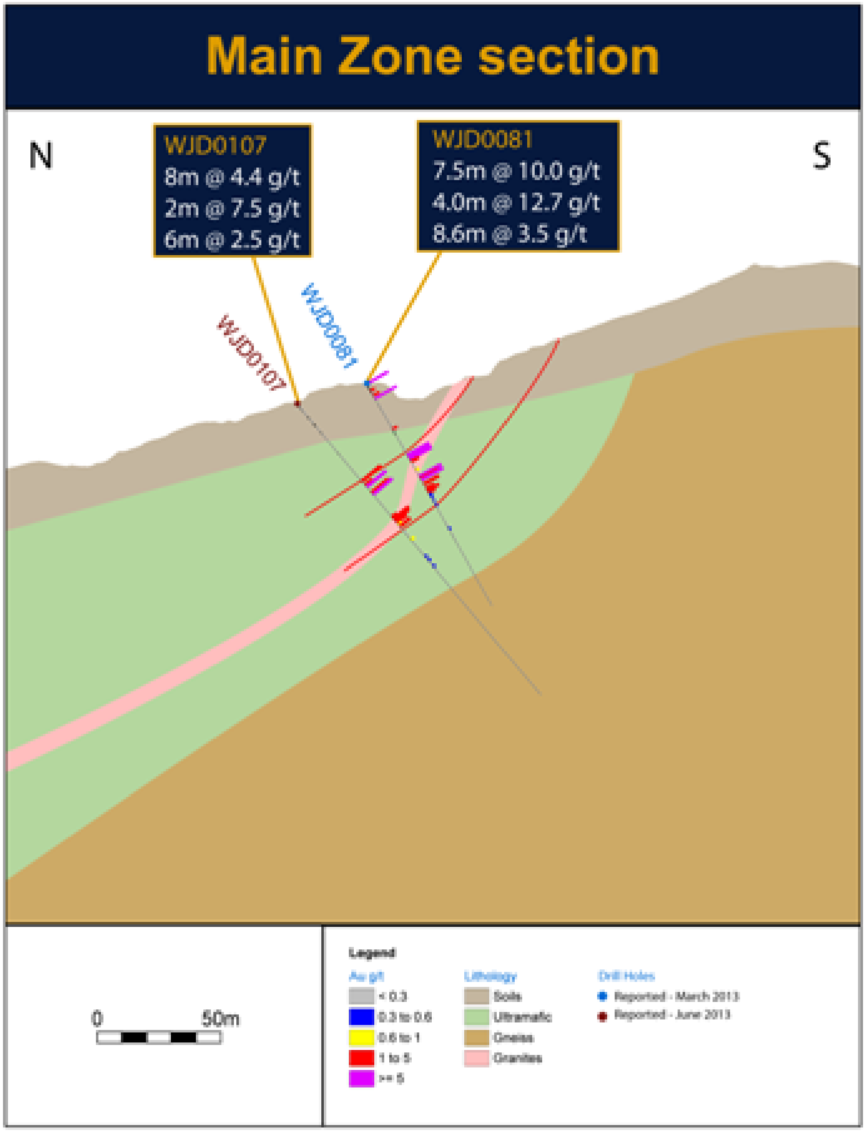
A total of 12,661 m of diamond core drilling has been completed in 110 drillholes, 3,935 m during two campaigns over the period 2000 to 2006, and a further 8,726 m during 2012/13. This drilling confirms the presence of four principal high-grade lenses of gold mineralization, designated as the North, Main , Ridge and Creek Zones, The North Zone dips at around 70° to the south (Figure 10.6) and the Main Zone zones dips north at around 60° (Figure 10.7). A strong plunge component to the SW is apparent in the mineralization, and this runs parallel to the fold axis in the area. The combined strike length of the zones is approximately 600 m, and the ridge and creek areas are where the two zones merge (Figure 10.8). Zones are typically around 8 m–12 m wide. A selection of intersections is presented in Table 10.6.

Figure 10.6 Weaju Section – North Zone



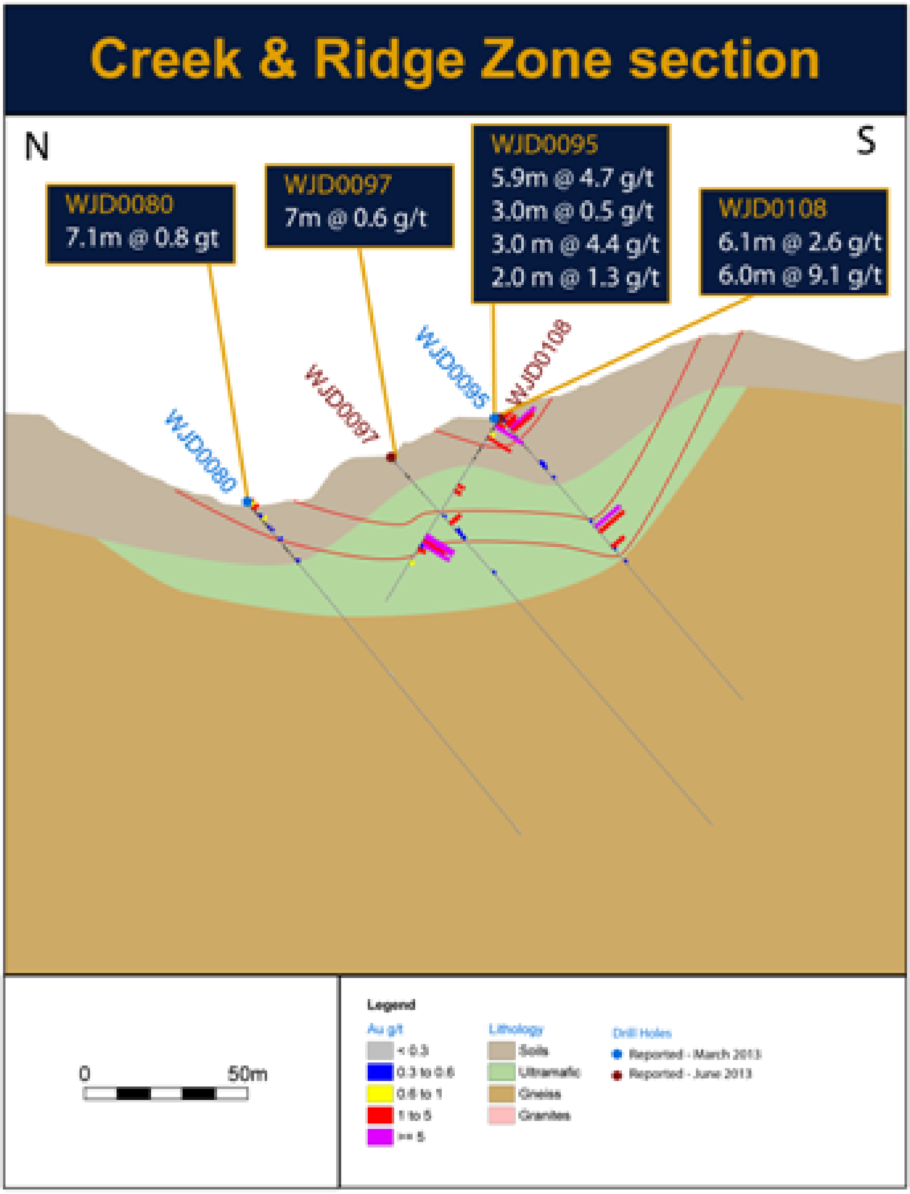
Source: *Aureus, 2013*

Figure 10.7 Weaju Section – Main Zone



Source: *Aureus, 2013*

Figure 10.8 Weaju Section – Creek and Ridge Zone



Source: *Aureus, 2013*

Table 10.7 Weaju Significant Intersections

Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Zone	Core Loss
2002-2006 Drilling						
WD1	0	24	24	33	Main	
WD1	48	50	2	3.5	Main	
WD2	8	30	22	4.5	Main	
WD5	14	48	34	19.9	Main	
WD7	19	23	4	15.1	Main	
WD9	26	44	18	4.5	Main	
WD38	80	82	2	3.8	Main	
WD41	86	87	1	9.2	Main	
WD13	16	28	12	10	North	
WD42	36	43	7	4.6	North	
WD45	17	27	10	6.1	North	
WD15	90	91	1	3.1	Ridge	
WD19	35	51	16	3.6	Ridge	
WD20	23	51	28	3.1	Ridge	
WD24	35	38	3	3.4	Ridge	
WD40	29	39	10	3.7	Ridge	
WD28	47	53	6	27.7	Creek	
WD47	19	21	2	3.2	Macenta	
2012/2013 campaign						
WJD0049	16	19	3	5.8	Main	
and	24	43	19	2.5	Main	
WJD0050	3	4	1	3.7	Main	
and	33	35	2	1	Main	
and	64	65	1	2	Main	
WJD0051	33	35	2	0.5	Main	
WJD0057	1.4	10	8.6	1.3	Main	48%
and	111.5	112.5	1	6.5	Main	
WJD0058	4.5	10.3	5.8	1.3	Main	45%
and	93	99	6	1.6	Main	
including	93	94	1	4.6	Main	
WJD0059	12.9	19.5	6.6	5.5	Main	28%
WJD0060	52	54	2	0.5	Main	
WJD0065	107	113	6	1.4	Main	
WJD0081	0.2	7.7	7.5	10	Main	49%
and	33.6	37.6	4	12.7	Main	
and	43.6	52.2	8.6	3.5	Main	
WJD0082	84.6	87.6	3	0.9	Main	

Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Zone	Core Loss
and	122.6	125.6	3	0.5	Main	
WJD0083	87.7	89.7	2	2.3	Main	
and	106.7	109.7	3	0.4	Main	
and	116.7	117.7	1	9.1	Main	
WJD0084	114.4	123.4	9	6.6	Main	
WJD0085	41.5	42.5	1	11	Main	
and	61.5	63.5	2	1.4	Main	
and	73.5	75.5	2	1.7	Main	
and	77.5	79.5	2	2.1	Main	
and	85.5	90.5	5	0.4	Main	
and	94.5	99.5	5	0.5	Main	
and	102.5	104.5	2	0.9	Main	
and	133.5	134.5	1	4.8	Main	
WJD0107	40.7	48.7	8	4.4	Main	
Including	46.7	48.7	2	7.5	Main	
and	59.7	65.7	6	2.5	Main	
WJD0109	3.8	5.8	2	1.3	Main	
WJD0053	36	47	11	3.7	North	9%
including	41	46	5	8	North	20%
WJD0054	23.2	27.8	4.6	2.5	North	33%
WJD0055	78	87	9	3.8	North	3%
WJD0056	97	105	8	1	North	
and	108	111	3	1.8	North	
WJD0068	63	68	5	5.7	North	
2012/2013 campaign						
WJD0069	6	10	4	5	North	40%
and	15	17.5	2.5	1.2	North	20%
WJD0070	78	82	4	1	North	
WJD0072	13.9	25.2	11.3	4.6	North	38%
WJD0073	0.9	8.3	7.4	1.9	North	78%
and	18.2	23	4.8	0.5	North	
WJD0074	4.6	12.4	7.8	0.9	North	50%
including	9.4	12.4	3	1.2	North	33%
and	42	48	6	1.1	North	
and	50	58	8	1.5	North	
WJD0075	44.2	47.2	3	2.3	North	
and	80.6	91.6	11	0.5	North	
WJD0078	28.9	33.1	4.2	1.6	North	23%
and	63.1	64.1	1	2.6	North	
WJD0079	15.3	32.8	17.5	1.3	North	30%

Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Zone	Core Loss
and	42.8	47.8	5	2.4	North	28%
WJD0086	28.9	32.8	3.9	0.9	North	23%
and	63.8	64.8	1	2.8	North	
and	79.8	84.8	5	4.4	North	
and	86.8	87.8	1	2.3	North	
and	90.8	94.8	4	1.3	North	
and	106.8	107.8	1	5.2	North	
WJD0087	86.9	87.9	1	5.9	North	
and	95.9	97.9	2	0.9	North	
and	103.9	105.9	2	1.2	North	
WJD0088	51.5	52.5	1	1.6	North	
and	64.7	66.7	2	1.1	North	
and	82.7	83.7	1	2.8	North	
WJD0089	126.2	127.2	1	3.2	North	
and	140.2	143.2	3	0.9	North	
and	152.2	157.2	5	2.3	North	4%
WJD0090	9.4	10.7	1.3	1.4	North	
and	19.9	21.1	1.2	4.5	North	
and	24.1	26.1	2	0.5	North	
WJD0092	15.9	19.6	3.7	0.6	North	25%
WJD0093	25.8	30.9	5.1	0.5	North	
and	51.6	52.6	1	1.2	North	
WJD0096	121.8	129.8	8	1	North	
WJD0098	0.3	3	2.7	1	North	
and	88.2	90.8	2	0.5	North	
WJD0099	96.4	98.2	2	7.8	North	
and	101.4	112.4	11	0.9	North	
Including	110.4	112.4	2	3.5	North	
WJD0105	9.3	12.4	3.1	0.6	North	29%
and	188.4	190.4	2	0.7	North	
and	205.4	223.4	18	1.1	North	
and	237.4	238.4	1	11.5	North	
WJD0110	162.7	164.7	2	0.4	North	
WJD0080	0	7.1	7.1	0.8	Creek	39%
WJD0094	0	8.7	8.7	6.9	Creek	55%
and	28.4	30.4	2	0.5	Creek	
and	52.4	57.4	5	0.5	Creek	
WJD0100	46.8	48.8	2	1.1	Creek	
and	66.8	75.8	9	1.8	Creek	
WJD0101	24.6	26.6	2	1.1	Creek	

Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Zone	Core Loss
and	49.6	52.6	3	2	Creek	
and	82.6	91.6	9	2.3	Creek	
Including	86.6	88.6	2	8.9	Creek	
and	102.6	104.6	2	0.7	Creek	
WJD0102	53.6	55.6	2	0.4	Creek	
and	123.6	125.6	2	1.8	Creek	
WJD0061	28	29	1	1.6	Ridge	
2012/2013 campaign						
and	40	41	1	4.5	Ridge	
WJD0062	155	157	2	2	Ridge	
and	169	174	5	1	Ridge	
and	180	189	9	3.2	Ridge	
WJD0063	0	3	3	1.2	Ridge	
and	6.5	9.5	3	1.3	Ridge	
WJD0064	7.2	11.6	4.4	5.1	Ridge	43%
and	41	56	15	1.3	Ridge	
including	41	46	5	3.2	Ridge	
WJD0066	9	16	7	3	Ridge	
including	12	16	4	4.8	Ridge	
WJD0067	95.4	96.4	1	1.6	Ridge	
WJD0077	6	8	2	0.4	Ridge	
WJD0095	0	5.9	5.9	4.7	Ridge	
and	17.9	20.9	3	0.5	Ridge	7%
and	43.7	46.7	3	4.4	Ridge	
and	51.7	53.7	2	1.3	Ridge	
WJD0097	27.3	34.3	7	0.6	Ridge	
WJD0103	39	53	14	5.6	Ridge	
Including	42	46	4	8.4	Ridge	
WJD0104	72	77	5	0.9	Ridge	
WJD0106	29.9	35.7	5.9	1.1	Ridge	12%
WJD0108	1.3	7.4	6.1	2.6	Ridge	31%
and	43.7	49.7	6	9.1	Ridge	
Including	43.7	47.7	4	13.3	Ridge	

Note: WD1 – WD47 were not NI 43-101 compliant and the results are shown only as information.

10.7.3 Gondoja

During 2012 a 13-hole, 2,699 m programme, was completed at Gondoja, and selected intersections are presented in Table 10.8. Core from the five drillholes previously drilled at Gondoja were lost during the period of civil unrest.

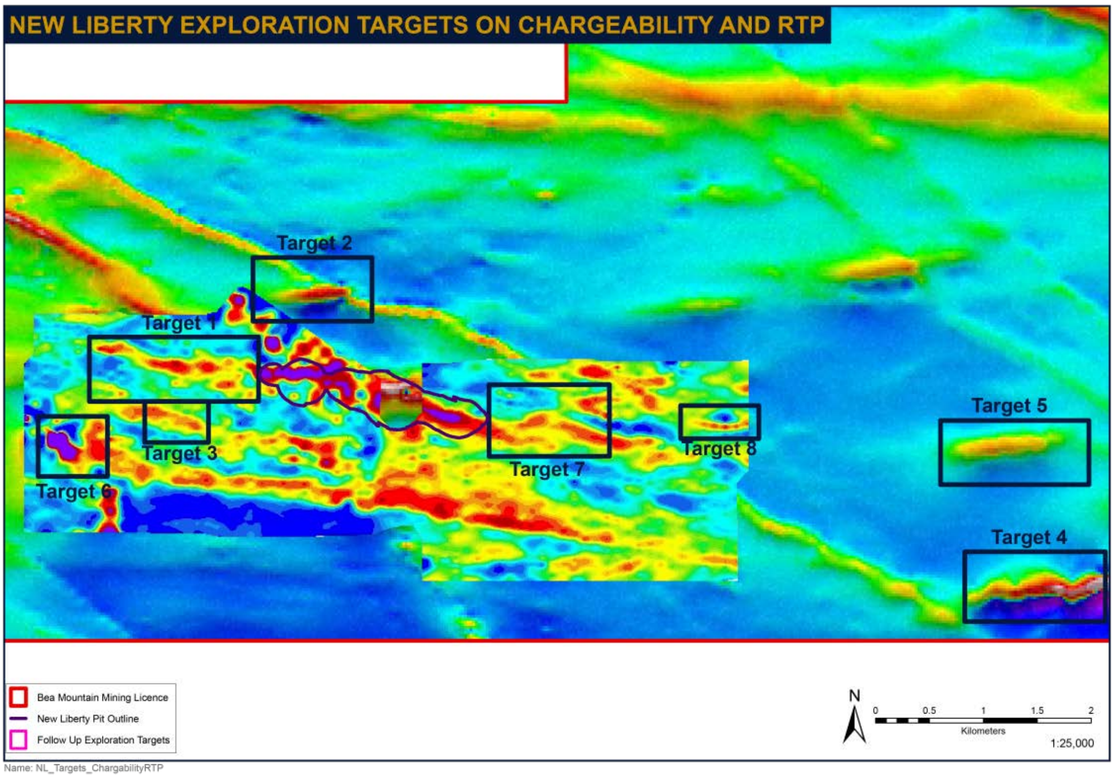
Table 10.8 Gondoja Significant Intersections

Borehole ID	From (m)	To (m)	Intersection Length (m)	Au Grade (g/t)
GD-4	74	110	30	3.9
GDD005	37.3	39	1.7	1.3
GDD010	72.8	75.8	3	14.8
GDD003	58.7	63	4.3	3.6
and	87	90	3	2.9
GDD004	36	37.85	1.9	0.9
and	40.9	48.8	7.9	0.6
GD-2	12	14	2	1.1
GDD001	62	65	3	2.2
and	73	76	3	0.8
and	90	96	6	11.7
and	104	106.6	2.6	0.7
and	114.8	116.3	1.5	1.6
GDD002	0	2.5	2.5	1.4
and	3.2	5.76	2.6	0.6
and	15.2	27	11.8	1.3
and	70.9	72.8	1.9	0.8
GD-1	38	42	4	3
GDD012B	25.8	27.8	2	1.2
and	30.8	33.5	2.7	0.9
and	73	78.8	5.8	0.7
and	79.1	81	1.9	0.7
GD-5	38	42	4	2
GD-5	58	64	6	1.2

Notes: Assay grade data is un-cut
NSV - GD-3, GDD006, GDD007, GDD008, GDD009, GDD011 and GDD013. GDD012 was abandoned.
Borehole values are reported from north-to-south.

10.7.4 Drilling Near the Project

Figure 10.9 Drill Targets Near to the Project



Near to the Project a further 12,153 m of drilling has been conducted on eight targets (Figure 10.9). This has identified continuations of the ultramafic host rock and parallel bands of the ultramafic units have been found to the north.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

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

11.1 Soils and Trenches

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

One metre-long samples were systematically collected in saprolite material from 10 cm square channels cut into cleaned trench walls near the floor of trenches and across the strike of mapped structures. For consistency the channels start at the southern end (collar) and numbered as such. Some trenches (and channels) were excavated in separate segments to traverse around large boulders, trees and unstable artisanal workings, to give continuity across the zone.

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

11.2 Diamond Drillhole Samples

Diamond-drilling activity at the Project is also supervised by Aureus geologists. Core and core blocks are placed in core boxes by the driller. Upon reception in the core shed on site, core was cleaned or washed (if required) and core blocks were checked by Aureus 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 is measured in situ and results recorded, in SI units (kappa), in the assay log sheet.

Geotechnical logging records casing size, bit size, depths, intervals, core loss/gain, core recovery with weathering index, RQD, fracture index, jointing and joint wall alteration and a simple geological description. Geotechnical logging covers holes up to K215 and K220, K226,K238,K239,K284,K304,K314,K320,K246,K248,K253,K256,K258,K263,K26 5K266,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 (Figure 11.1).

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

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

Figure 11.1 Structural Core Logging Using Jig

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

11.2.1 Bulk Density Measurements

Bulk density readings are taken at 2 m intervals within the same lithology and on every lithological break. This is carried out by weighing samples in air and water with a balance, where porous samples are wrapped in plastic. For drillholes K1-130, measurements were carried out on half core, i.e. post-sampling, but now whole core is used. Measurements are recorded using a balance with top and under-slung measuring capabilities with detection limit of +/-1 gm (Figure 11.2).

Figure 11.2 Measurement of Bulk Density

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

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

The range of bulk densities by geological unit is shown in Table 11.1.

Table 11.1 Dry Bulk Densities

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

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

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

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

11.2.2 Preparation and Analysis

A review of the historical and current procedures follows:

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 juncture, samples from the Project have been crushed, pulverized and split in Monrovia, before sample splits were shipped by DHL to OMAC.

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

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

2009/2010 and 2011/2012 Campaigns

Prior to shipment, final checking is carried out in the presence of a senior geologist and two field assistants to ensure sample identities are correct, samples are intact and there are no omissions. Quality control standards and blanks samples are inserted at pre-determined intervals at this point. Samples are sent from site, on a complete-hole basis, directly to the OMAC preparation facility in Monrovia, along with documentation, which acts as a receipt and sign back. Sample transfer and delivery to the OMAC laboratory in Ireland can be monitored and tracked via their website, until assay results are 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) were commissioned as a reference lab. OMAC, ALS Chemex and SGS, including the Monrovia sample preparation facility, are independent of Bea and Aureus.

The flow chart in Figure 11.3 summarizes sample collection, sample preparation, assaying and QA/QC procedures adopted during the 2009/2010 and 2011 drilling campaigns, including the following recommended modifications made after AMC's December 2009 site visit.

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

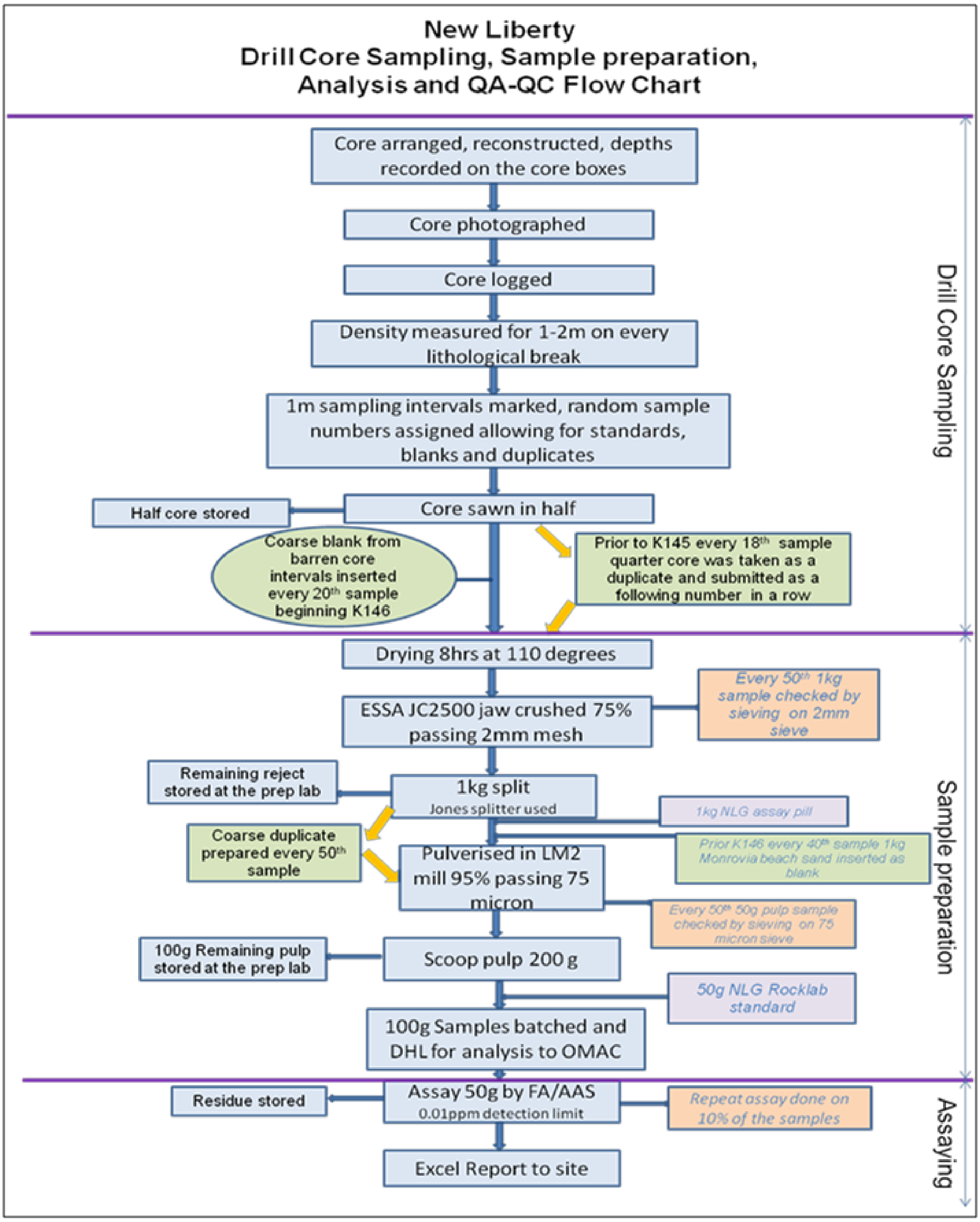
Samples are weighed, mixed with flux and fused in clay crucibles. Lead buttons produced after fusion are cupelled, forming dore prills that are digested in aqua regia, and digests are analysed for gold using a Varian AA Spectrometer.

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

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

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

Figure 11.3 Sample Preparation and QA/QC Flow Chart



11.3 Assay QA/QC

For the discussion below, the drilling campaigns have been combined into three periods, since little QA/QC work was carried during in the early campaigns. AMC has undertaken

QA/QC analyses for the periods 2005–2008, 2009–2010 and 2011; however only the 2009-2010 analyses are presented here in any detail, with the remaining periods covered in summary form only.

11.3.1 Period 1999–2000

11.3.1.1 Field Duplicates

Five quarter core samples from split core were collected by ACA Howe during their work in 2000 and sent for preparation and fire assay at OMAC laboratories. Table 11.2 compares the original samples and the Howe checks.

Table 11.2 1999–2000 Field Duplicate Comparison

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

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

11.3.2 Period 2005–2008

11.3.2.1 Blanks

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

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

11.3.2.2 Standards

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

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

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

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

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

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

11.3.2.3 Laboratory Repeats

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

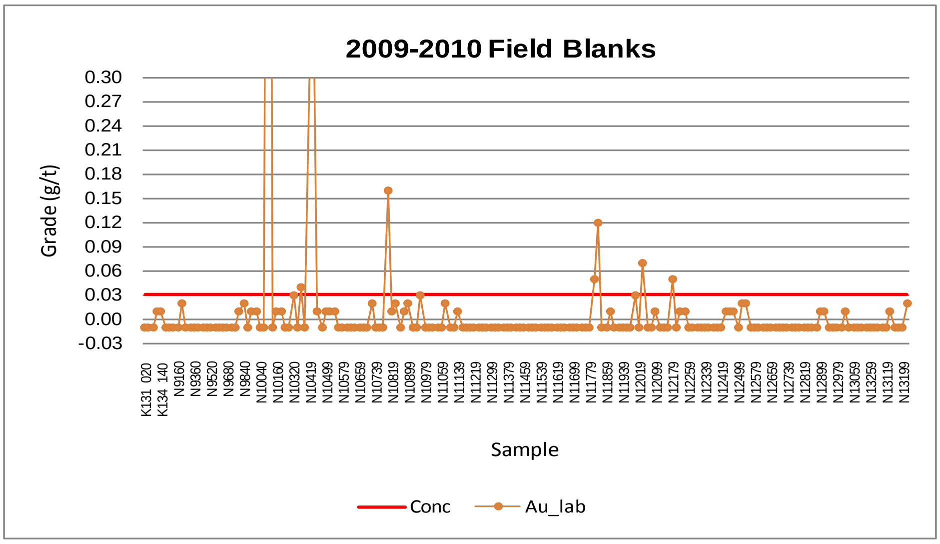
Statistical summaries and charts indicate that, while there is good linear correlation between sample pairs, the point cloud shows a relatively wide spread. A precision value of 18.5% was achieved, in the context of a recommended precision for pulp pairs of less than 10%. This suggests that a high nugget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

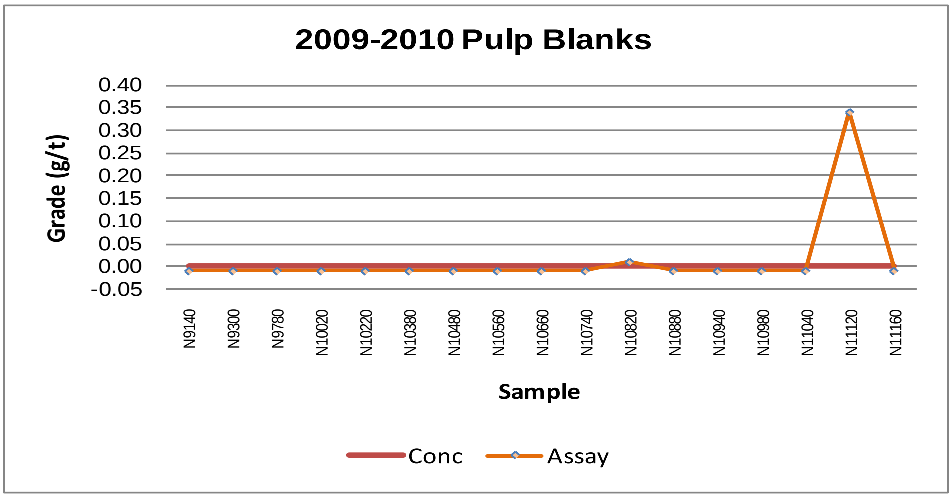
11.3.3 Period 2009–2010

11.3.3.1 Blanks

Initially in this period (from drillhole K131) Monrovian beach sand was used as blank samples, but from hole K146 onwards blanks were taken from barren hanging wall material, submitted as coarse samples which pass through all the preparation stages. The results included two outliers and five samples above three times the detection limit, while the remaining assays performed as expected (Figure 11.4). Pulp blanks recorded one outlier that most likely indicates a misclassification of a standard.

Figure 11.4 2009–2010 Blank Sample Analysis





11.3.3.2 Standards (CRM)

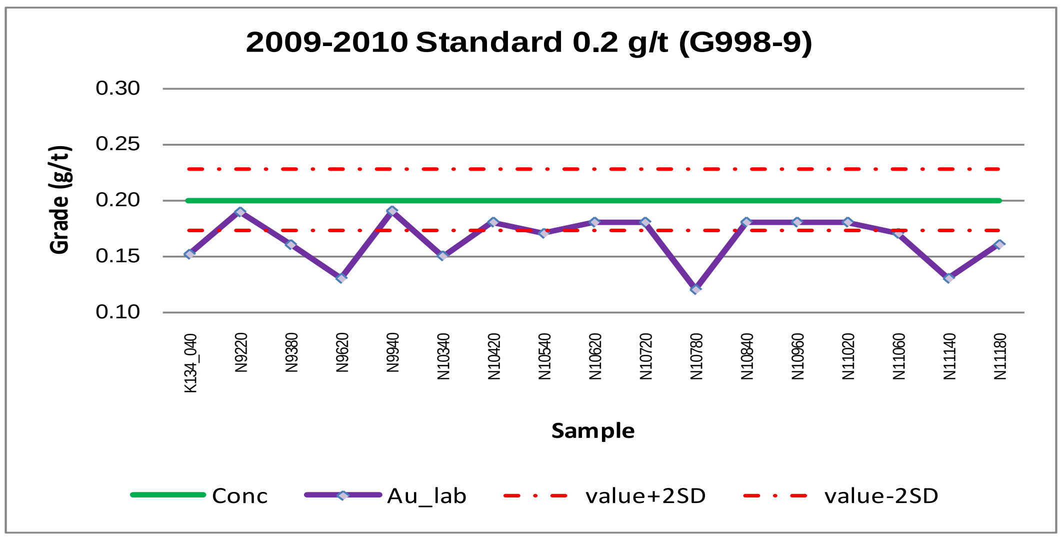
A total of eight standards were used during the 2009/2010 drilling campaign, which had the following suppliers and gold values:

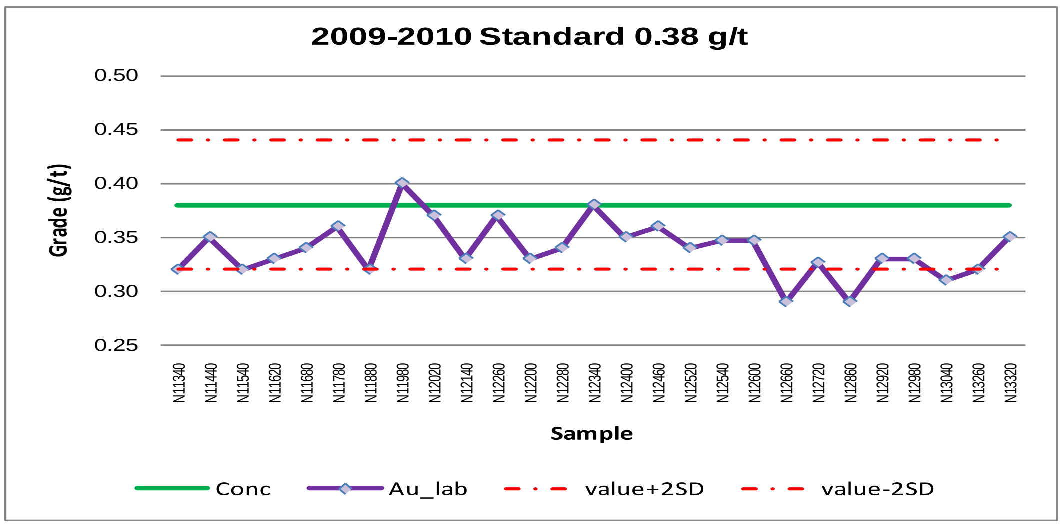
Rocklabs: 0.20 g/t, 0.99 g/t,1.031 g/t and 5.911 g/t.

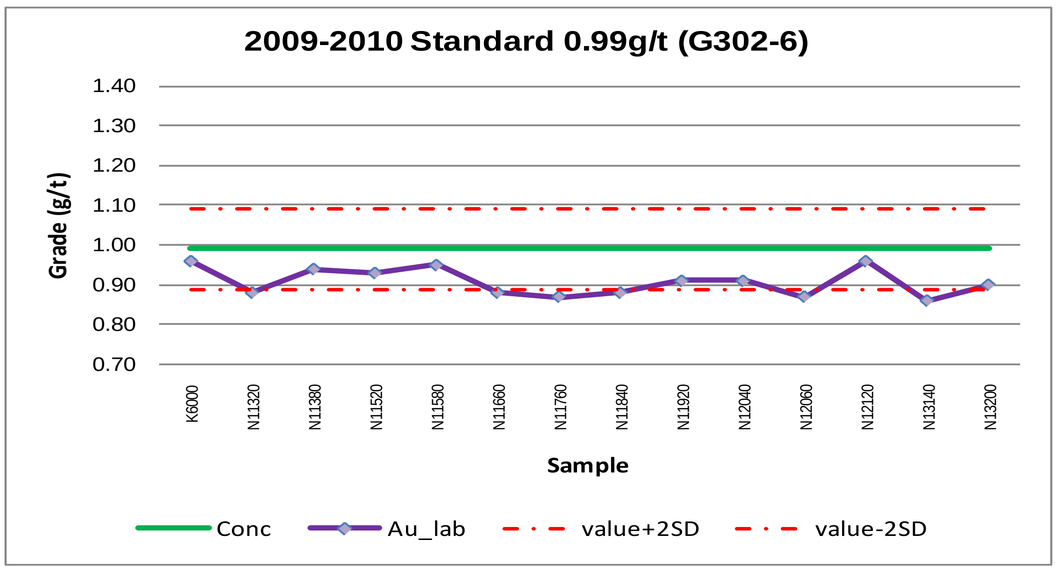
Geostats Pty Ltd: 0.38 g/t, 0.99 g/t,1.52 g/t and 1.96 g/t.

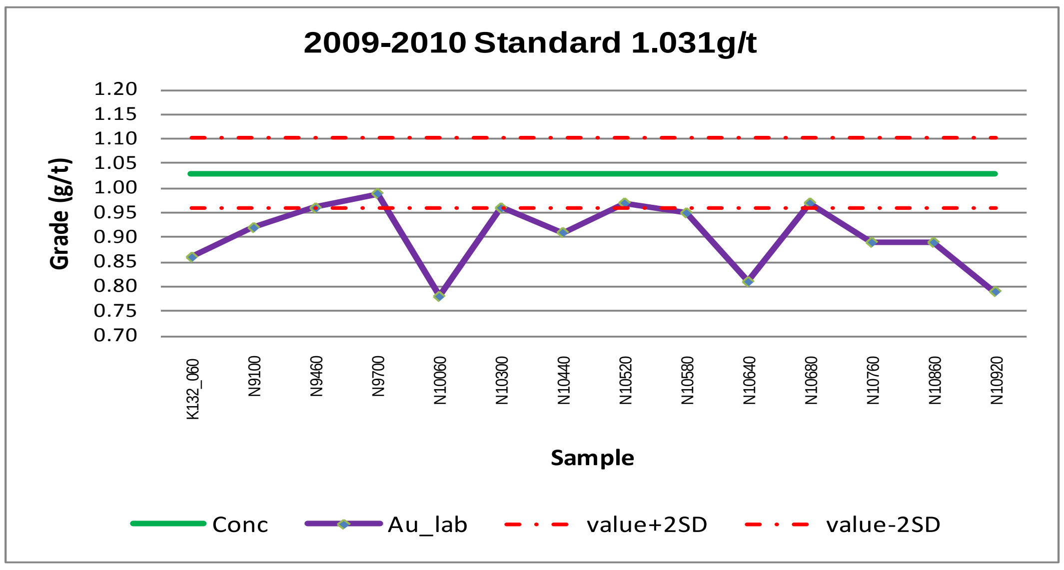
The performance of assays against all the standards, from both sources, was very poor, most clearly reflected in a strong negative bias (Figure 11.5). In addition, a small number of outliers were also recorded, suggesting mislabelling during sample submission or sample preparation.

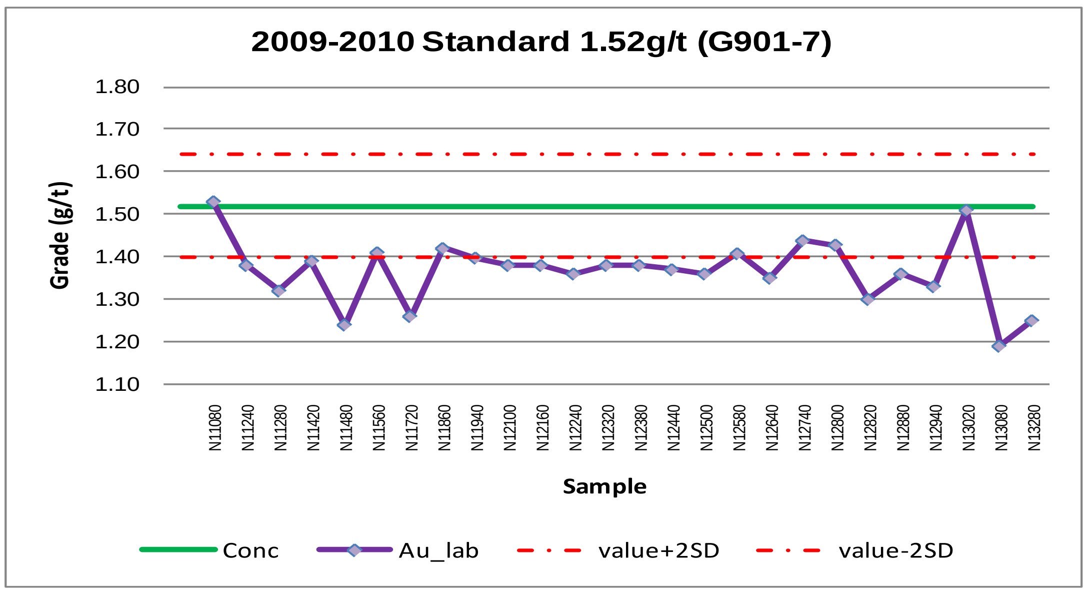
Figure 11.5 2009–2010 Standards Analysis

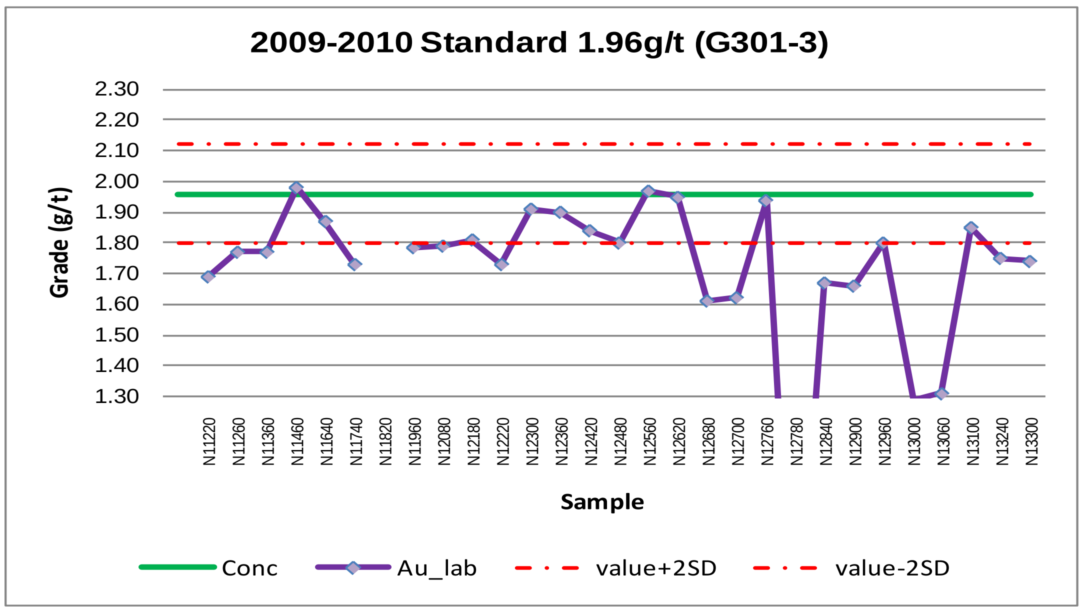


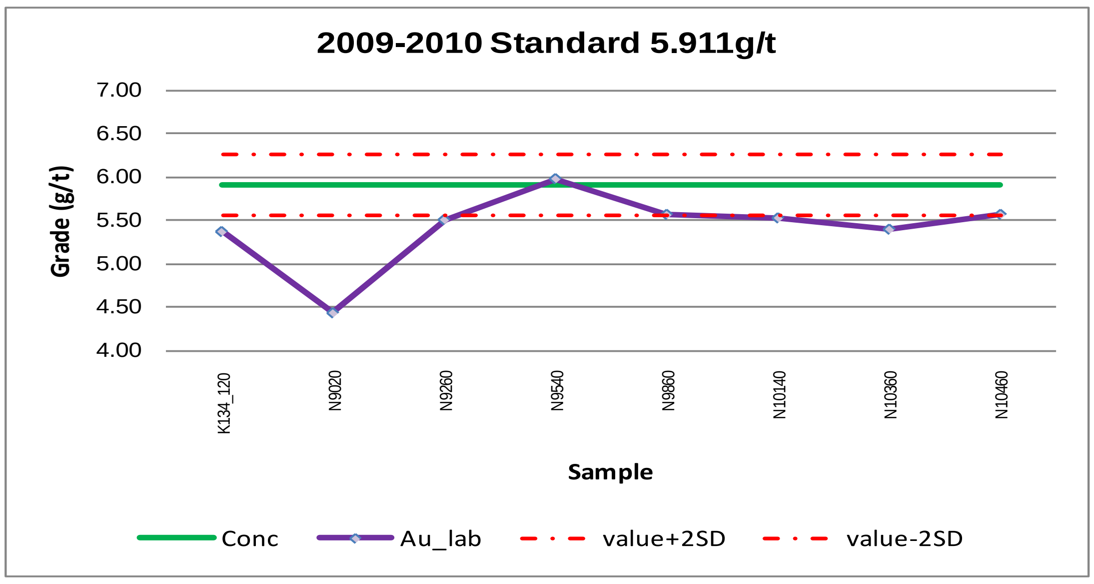







2009-2010 Standard 1.52g/t (G901-7)
Grade (g/t)
Sample
Conc
Au_lab
value+2SD
value-2SD


2009-2010 Standard 1.96g/t (G301-3)
Grade (g/t)
Sample
Conc
Au_lab
value+2SD
value-2SD


2009-2010 Standard 5.911g/t
Grade (g/t)
Sample
K134_120
N9020
N9260
N9540
N9860
N10140
N10360
N10460
Conc
Au_lab
value+2SD
value-2SD

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

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

11.3.4 Drilling Duplicates

11.3.5 Quarter Core Duplicates

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

11.3.6 Crushed Duplicates

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

A total of 49 samples were reported as crushed duplicates, only 10 of which locate within a mineralized interval (Table 11.3).

Table 11.3 2009–2010 Crushed Duplicate Pairs Value

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

11.3.7 Drilling Laboratory Repeats

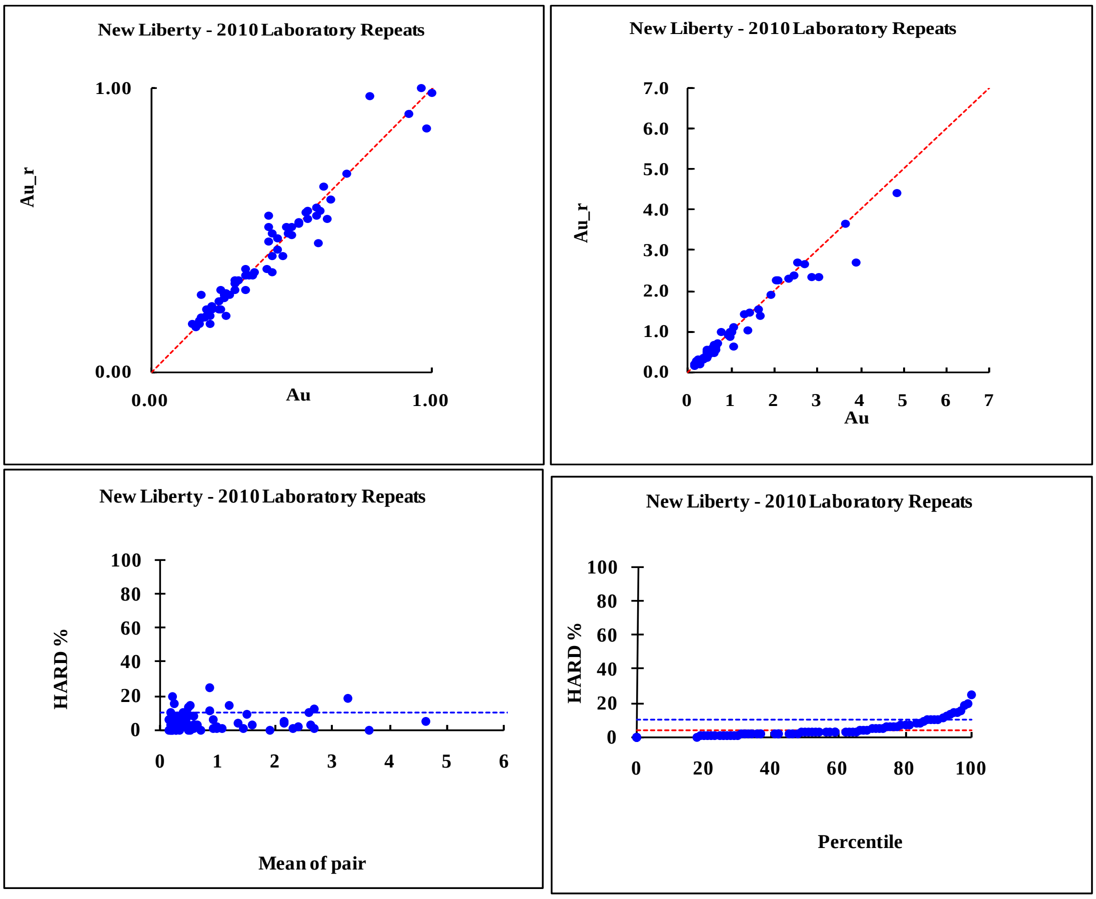
A total of 503 laboratory repeat assays were undertaken by the primary laboratory, OMAC, of which only 138 exceeded ten times the detection limit. For statistical analysis of laboratory repeats, all assays below fifteen times the detection limit and grades in excess of 10 g/t Au were excluded.

The laboratory repeat assay results are presented in Table 11.4 where the 12.5% precision statistic is considered high, since the precision for laboratory repeats is expected to be well below 10% (see also Figure 11.6). The poor precision could be attributed to inherent high nugget effect or poor preparation procedures.

Table 11.4 2009–2010 Laboratory Repeats Statistics

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

Figure 11.6 2009–2010 Laboratory Repeats Analysis



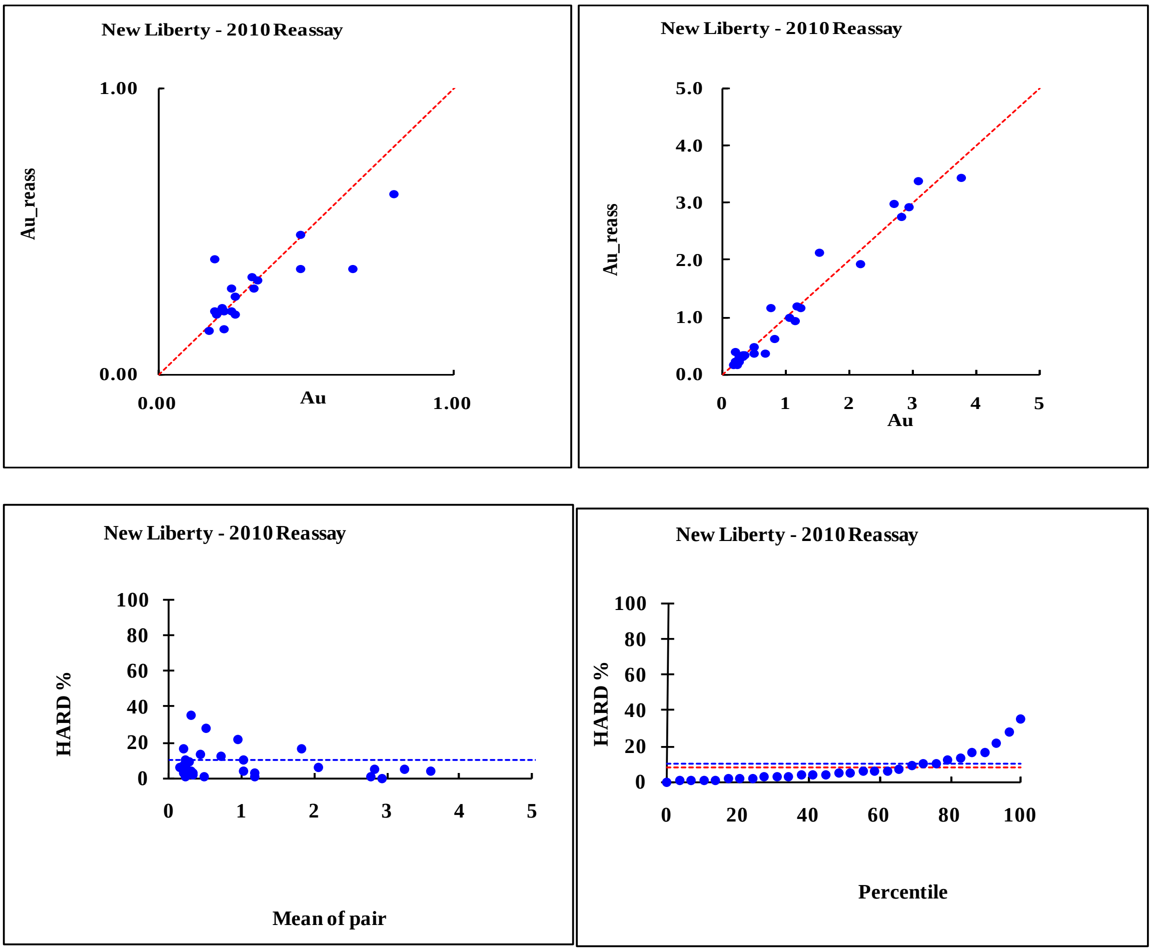
11.3.8 Drilling Re-assay Samples

Some sample batches were submitted for re-assay because of concerns arising from a QA/QC review. Of the 180 results generated, thirty pairs remained for statistical analysis after removing samples below 0.15 g/t Au and above 5.0 g/t Au. A poor precision of 22.9% was achieved (Table 11.5 and Figure 11.7).

Table 11.5 2009/2010 Re-assay Samples

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

Figure 11.7 2009–2010 Re-assay Sample Analysis



11.3.9 Umpire Laboratory Check Assays

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

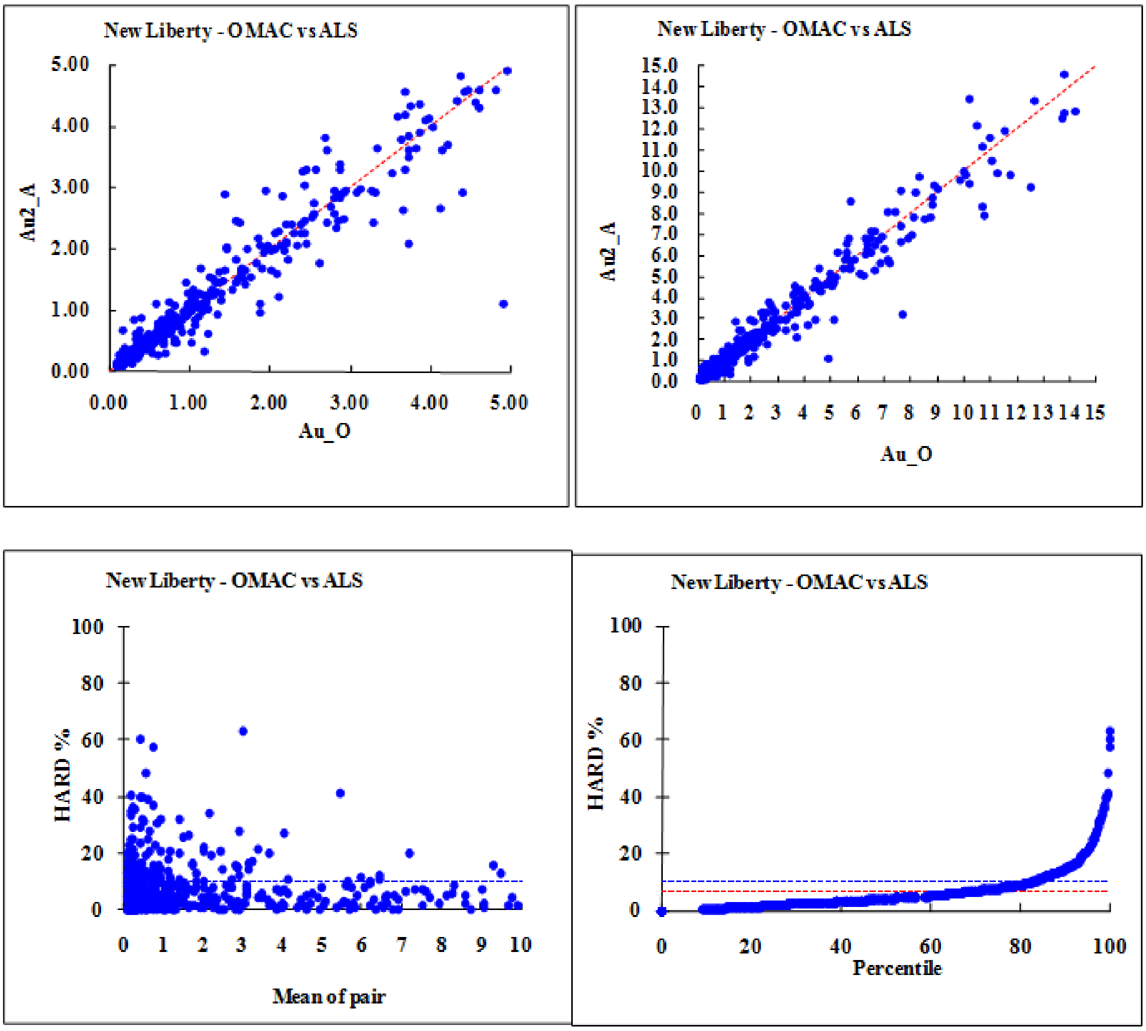
A total of 1,051 selected pulp samples were despatched to ALS Chemex for assaying, including 52 blanks and 50 CRMs. After removing outliers and values below ten times the detection limit 732 pairs were available for inter-laboratory comparison.

The results of the comparison are presented in Table 11.6 and charted in Figure 11.8. Considering that the pulps were re-prepared from coarse rejects a precision of 19.3% is within acceptable limits. The OMAC results show some negative bias (1.05%) relative to the ALS Chemex values.

Table 11.6 Inter-laboratory Comparison

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

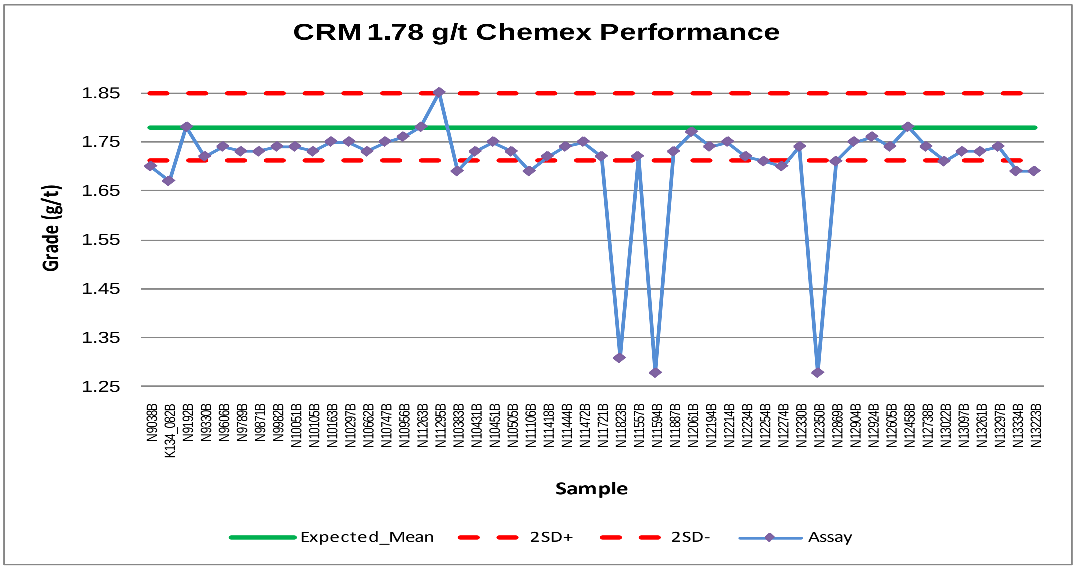
Figure 11.8 2009–2010 Inter-laboratory Comparison



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

The analysis of assay results from standard samples submitted during the ALS Chemex umpire laboratory programme show the presence of a 3 outliers, as well as a consistent low bias (Figure 11.8), albeit less of a bias than in the original OMAC results.

Figure 11.9 2009–2010 Umpire-laboratory Standards



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

11.3.10 Period 2011

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

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

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

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

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

11.3.11 Sample and Assay QA/QC Observations

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

Some evidence of sample mix-ups raises the concern that other less obvious cases may exist but go undetected. However, these appear to be relatively isolated cases and AMC has not observed instances of systemic sample misallocation.

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

There are remaining areas of concern relating to assay precision and apparent low bias, which have only been partially explained or resolved. Nonetheless AMC believes that the gold assay data is suitable for use for resource estimation at the confidence levels that have been assigned. Recommendations for further QA/QC improvements are included in Section 19.

In October 2011, AMC visited the OMAC sample preparation facility in Monrovia, used for the preparation of all Project samples. AMC noted that the facility is managed by an accredited laboratory which implements industry standard procedures. However, the state of the facility and the observed practices, whilst generally sound, suggest that there remains some risk for both sample mix-ups, due to the workflow layout, and contamination due to the quality of the ventilation.

AMC considers that these concerns are unlikely to be highly significant but, nonetheless recommends that Aureus commission an independent sampling and assaying specialist to conduct an audit of the sample preparation chain of activities.

During the course of 2012, Aureus changed laboratories to the recently opened SGS Monrovia facility. This decision does not affect the work covered in this report, but the recommended audit should include the SGS facility.

12 DATA VERIFICATION

12.1 Source Data Verification

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

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

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

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

12.2 Database Field Integrity

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

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

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

12.3 Data Verification Observations

As for the sample and assay QA/QC, procedures for data management and storage have improved over time. Remaining areas of uncertainty therefore relate mostly to the older data which cannot always be fully verified. AMC believes that the current data management is diligently undertaken and confidence in the data is enhanced by the close attention that field personnel apply to data management.

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

Aureus has since employed a database manager and is in the process of organizing a new database system to cope with larger datasets, and to improve QA/QC procedures in general. In the interim, procedures have been established to review batch QA/QC assay results as soon as they are available.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Previous Metallurgical Test Work Phases

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

13.2 Optimization Test Work Scope

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

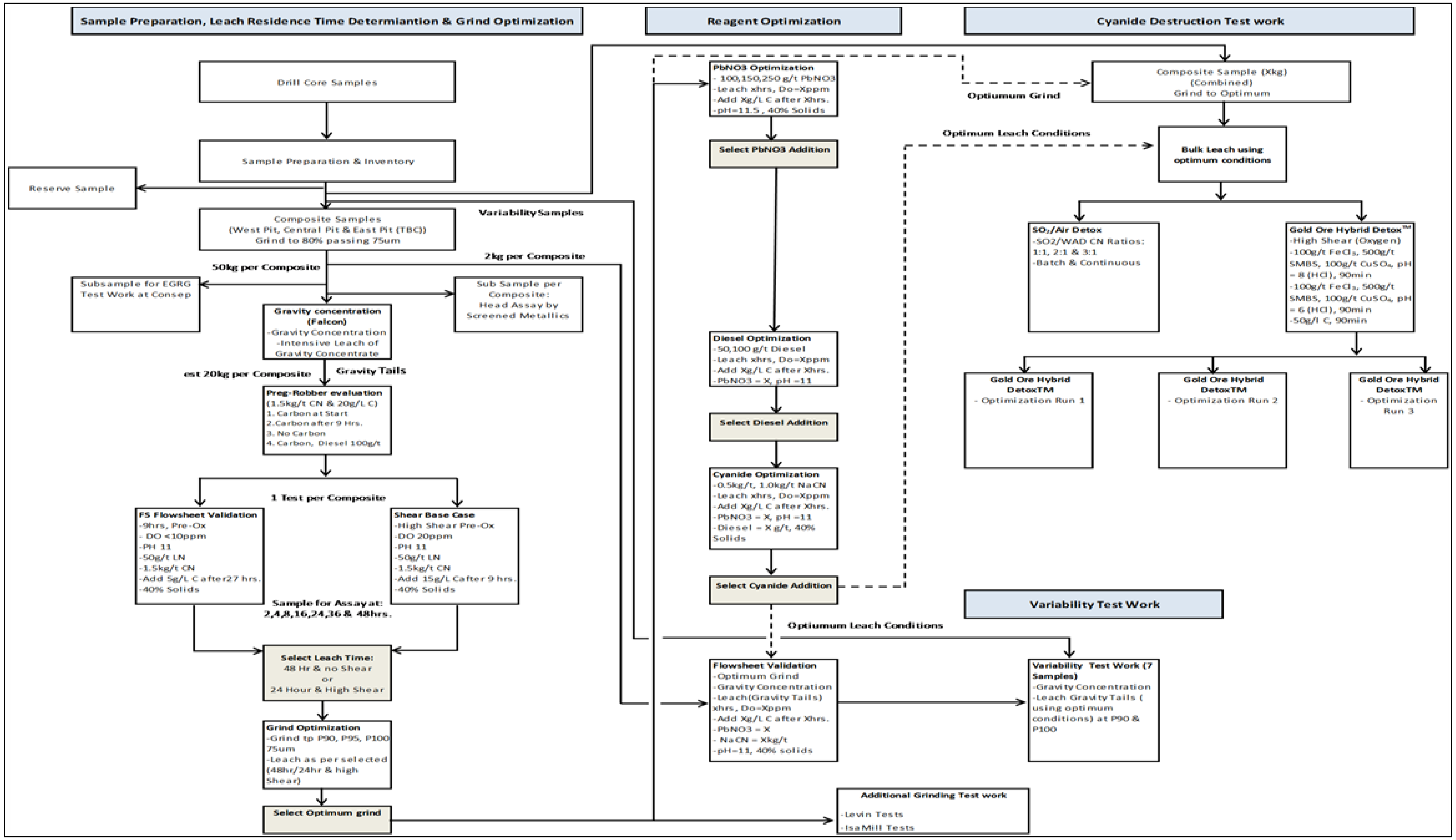
The metallurgical scope of the optimization phase was designed with the objective of completing all metallurgical test work required to finalize the process design criteria in order to finalize the process flowsheet, size mechanical equipment and determine plant capital and operating costs.

The test work programme included the following:

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

A flow chart for the optimization phase metallurgical test programme is detailed in Figure 13.1 below.

Figure 13.1 Optimization Phase Test Work Flow Chart



13.2.1 Test Work Samples

The metallurgical optimization test work programme was performed on both composite and variability samples. A summary of the sample inventory list is presented in Table 13.1 below:

Table 13.1 Optimization Phase Sample Inventory List Summary

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

13.2.2 Chemical Analysis

The screened fire assays for the teswork samples are presented in Table 13.2 below:

Table 13.2 Screened Fire Assay Results

Composite ID	+75 µm		-75 µm			Calc'd Head Au (g/t)
	Mass (g)	SFA Au (g/t)	Mass (g)	SFA Au$_1$ (g/t)	SFA Au$_2$ (g/t)	
Master	33.08	24.8	955.34	3.61	3.43	4.23
Comp #1	26.23	13.7	946.8	2.52	2.32	2.72
Comp #2	26.15	124	952.1	9.78	10.40	13.13
Comp #3	25.79	6.27	967.0	4.57	4.75	4.70
Comp #4	16.96	31.8	972.0	5.65	5.84	6.19
Comp #5	25.92	6.52	962.5	3.22	2.81	3.11
Comp #6	25.29	54.7	951.4	2.89	3.57	4.56
Comp #7	23.26	22.8	971.6	1.81	1.89	2.34
K485B	25.49	121.0	965.8	7.65	7.40	10.44
K490	18.75	32	972.7	2.13	2.05	2.66
K156	25.59	61.60	973.9	4.69	4.75	6.18
K233	20.51	11.4	964.8	4.42	4.89	4.80
K236	26.60	5.72	960.9	4.12	4.31	4.26
K287	22.70	41.2	964.2	3.75	3.99	4.73
K492	12.61	166.0	965.1	3.12	2.61	4.97

13.2.3 Composite Samples

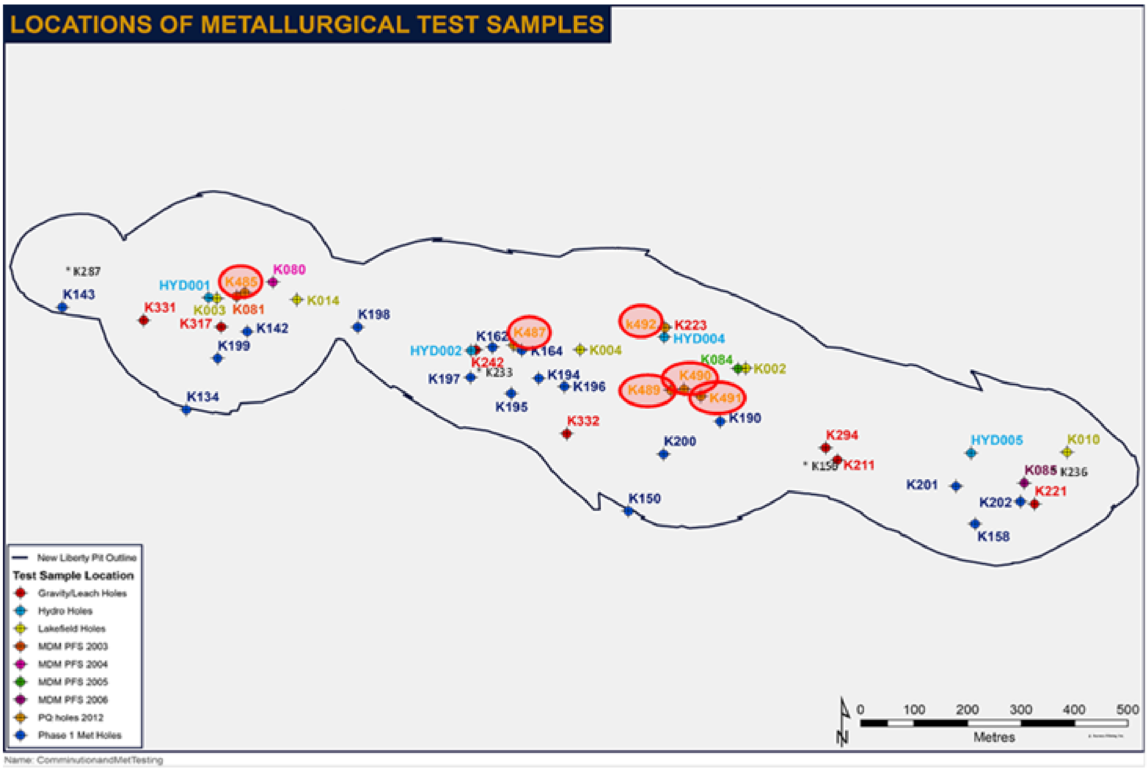
In October 2012, 800 kg of pre-crushed (12.7 mm) core sample material was delivered to ALS's mineral processing facility for metallurgical characterization test work. This material was the product material from comminution test work conducted on core samples at JKtech in September 2012. The sample material received from JKtech was composited to produce a 240 kg bulk master composite sample. This bulk sample was comprised of pre-crushed core samples from the western and central portions of the deposit at various depths. This composite represented the first six years of the mine schedule and did not contain material from the eastern portion of the deposit. The master composite had an assayed gold grade of 4.23 g/t as determined by screened fire assay (see Table 13.2 above).

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

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

The distribution of the optimization phase metallurgical composite test sample drill holes are presented in Figure 13.2 below (as circled in red).

Figure 13.2 Optimization Phase Distribution of Composite Test Sample Drillholes

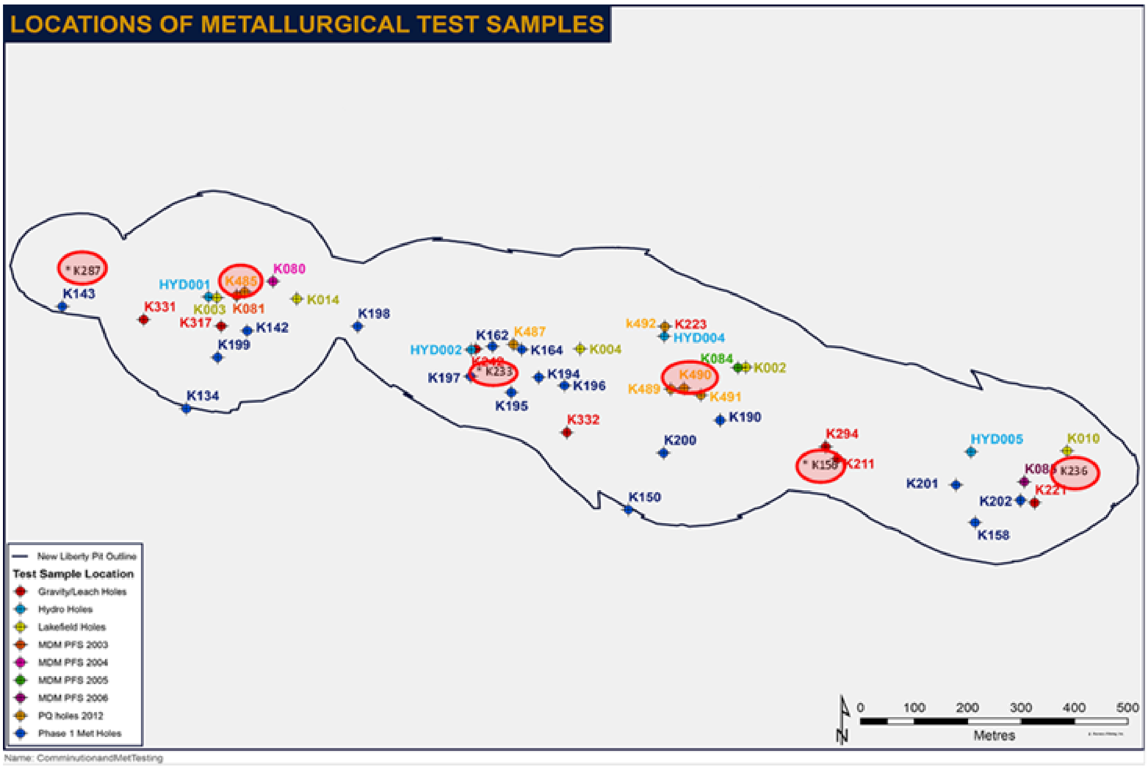


13.2.4 Variability Samples

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

The Distribution of the optimization phase metallurgical variability test sample drill holes are presented in Figure 13.3 below (as circled in red).

Figure 13.3 Opitmization Phase Distribution of Variability Test Sample Drillholes



13.3 Leach Optimization Test Work on the Master Composite Sample

Leach optimization test work on the master composite was performed on gravity tailings samples. The test work conducted on the master composite sample was aimed at determining the objectives as listed in Section 13.2.3 above. The detailed optimization phase test result can be viewed in the ALS leach optimization programme test report document titled: Metallurgical Test Work conducted for the New Liberty Gold Project (Report No: A14708)

A summary and interpretation of the results from this phase of test work are presented below.

13.3.1 Evaluation of Preg-Robbing

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

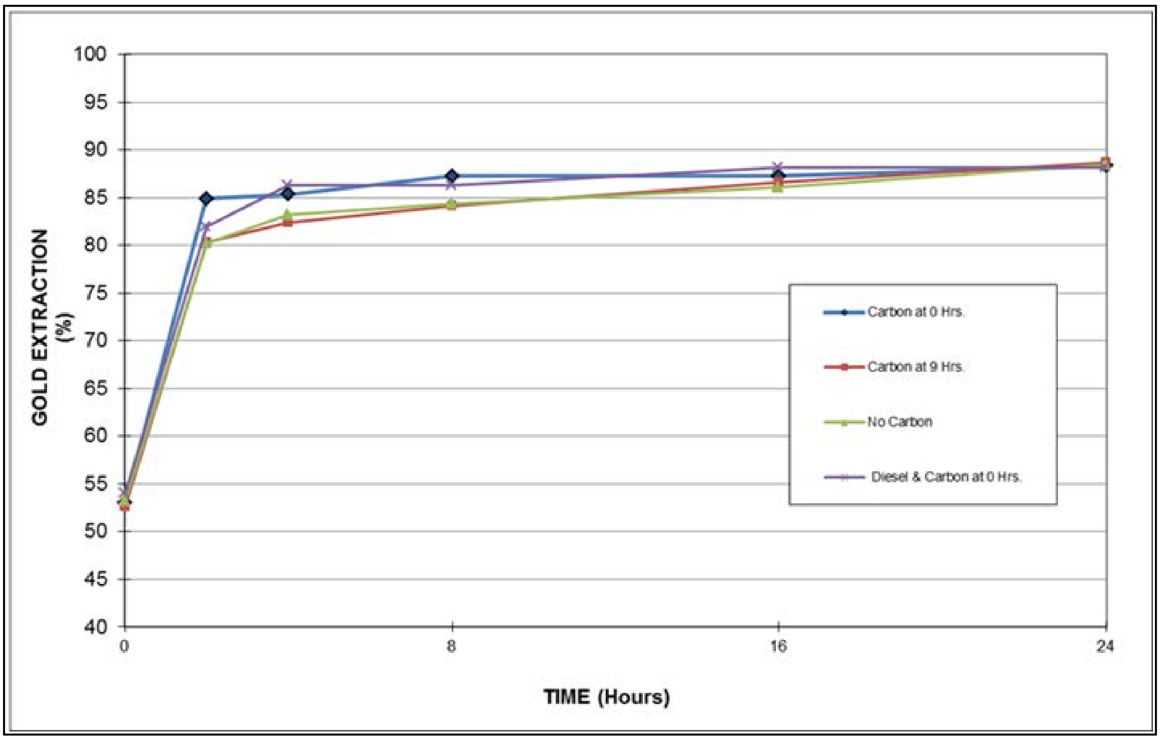
A Carbon in Leach (CIL) test was conducted on each composite at the target grind of 80% passing 75 µm. Comparison of the results of conventional cyanidation tests and carbon leach tests as presented in Table 13.3 and Figure 13.4 it can be seen that the gold assays in the final residues for both methods were very similar for each composite. The similarity of the results suggests that there is no preg-robbing in these composites.

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

Table 13.3 Metallurgical Results for Optimization Leach Tests to Evaluate Preg-Robbing

Test No	Feed	Test Conditions	Calculated Au Head Grade (g/t)	Residue Au Grade (g/t)	Au Extraction (%)
JR133	Master Composite	1.5kg/t CN, 24 hrs. Add Carbon at 0 Hrs.	4.20	0.49	88.34%
JR134	Master Composite	1.5kg/t CN, 24 hrs. Add Carbon at 9 Hrs.	4.23	0.48	88.67%
JR135	Master Composite	1.5kg/t CN, 24 hrs. No Carbon	4.19	0.49	88.42%
JR136	Master Composite	1.5kg/t CN, 24 hrs. Diesel and Carbon at 0 Hrs.	4.12	0.49	88.12%

Figure 13.4 Evaluation of preg-robbing for the New Liberty master composite sample at a grind of 80% passing 75 micron.



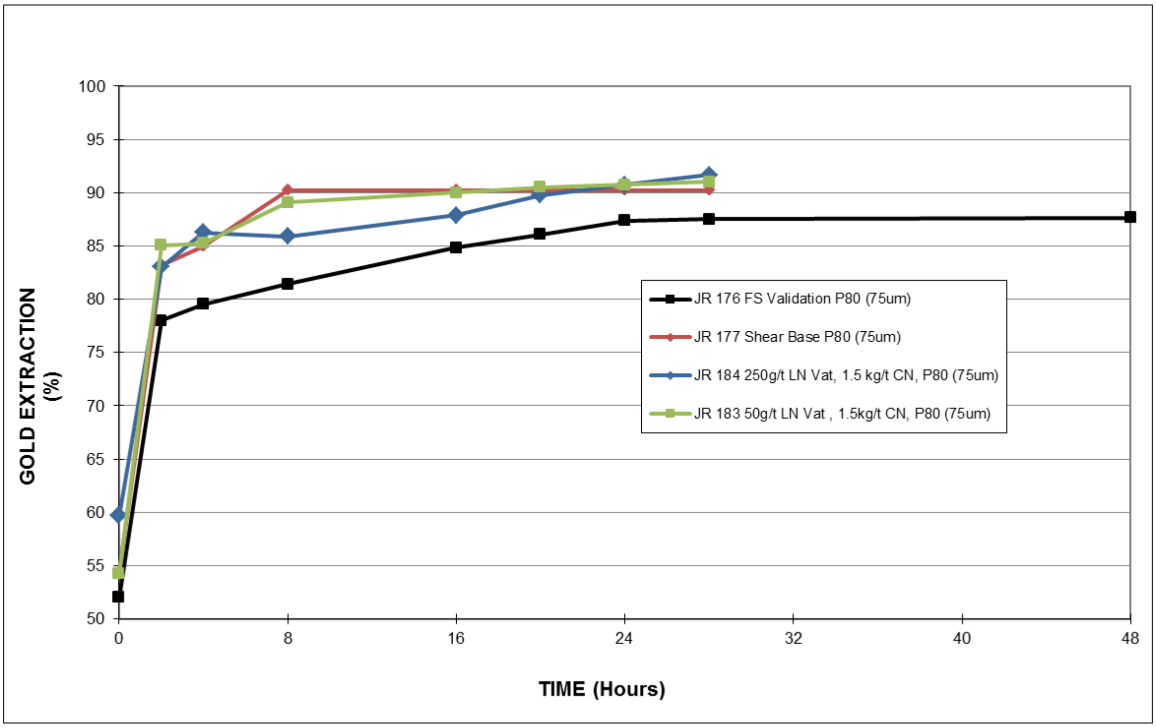
13.3.2 Effect of high-shear, pre-treatment with oxygen (JR 177/183/184) in comparison to the feasibility flowsheet performance (JR 176)

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

Table 13.4 Metallurgical Results for Optimization leach tests JR176, JR177, JR183 and JR184

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

Figure 13.5 Effect of high shear pre-treatment on gold recovery for the New Liberty master composite sample at a grind of 80% passing 75 micron.



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

13.3.3 Optmization of Cyanide Addtion

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

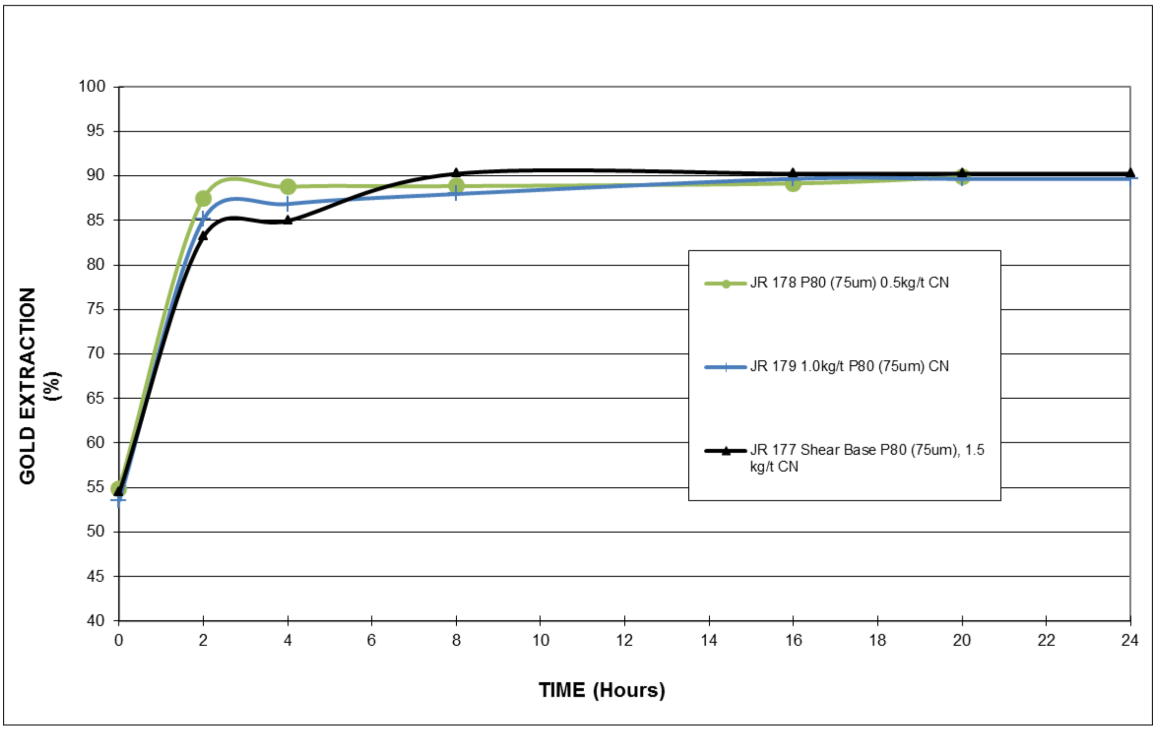
Comparison of the leach kinetics for tests as presented in Table 13.5 and Figure 13.6 indicated that at a cyanide addition of 0.5kg/t (JR 178), similar recoveries and kinetics could be achieved as those obtained for test conducted at a cyanide addition rate of 1.0 kg/t (JR 179) and 1.5 kg/t (JR177). The leach residue grades were in the range 0.40 g/t -0.43g/t.

Table 13.5 Metallurgical Results for Cyanide Optimization Leach Tests JR177-179

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

The results from the unmineralized zones were, unsurprisingly (since the average head grade was only 0.27gAu/t) poor and variable.

Figure 13.6 Effect of cyanide addition on gold recovery for the New Liberty master composite sample at a grind of 80%passing 75 micron.



13.3.4 Lead Nitrate Addition

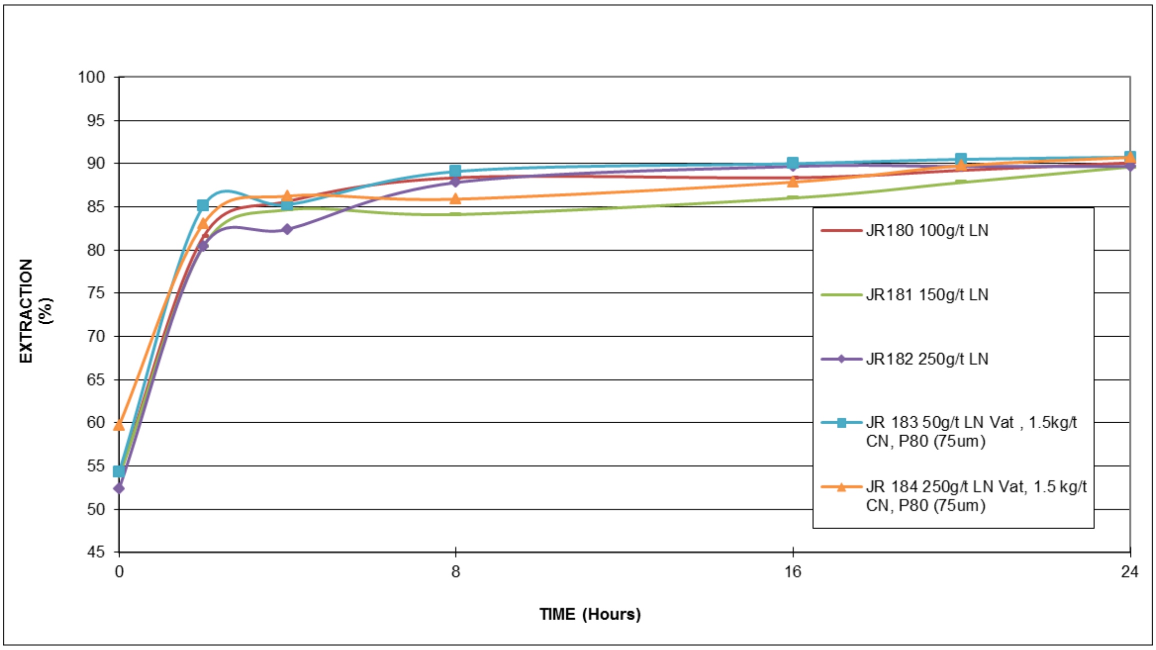
The effect of lead nitrate addition was evaluated at dosage rates of 250 g/t, 150 g/t, 100 g/t and 50 g/t.

The results of these tests are presented in Table 13.7 and Figure 13.7. Comparison of the leach kinetics for test JR180- JR184, did not show any significant difference in final gold extraction for the lead nitrate addition in the range 50 g/t – 250 g/t. The gold content of leach residues was found to be 0.35 – 0.44 g/t. Based on the kinetic curves as presented in Figure 13.7 below a minimum lead nitrate addition rate of 25 g/t was selected on the basis that higher addition rates of lead nitrate did not provide an improvement in recovery or leach kinetics.

Table 13.6 Metallurgical Results for Lead Nitrate Optimization Leach Tests JR180 - JR184

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

Figure 13.7 Effect of lead nitrate addition on gold recovery for the New Liberty master composite sample at a grind of 80% passing 75 micron.



13.3.5 Optimization of Lime Addition

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

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

A summary of the lime consumption for tests carried out in this manner is presented in Table 13.7 below. The lime consumption was found to be in the range 0.88 kg/t – 2.13 kg/t with an average consumption of 1.48 kg/t, which will be used as the optimized lime consumption.

Table 13.7 Leach Tests Conducted at an Initial pH of 11, with Lime Addition to Maintain pH 10

Test No	Feed	Test Conditions	Calculated Au Head Grade (g/t)	Residue Au Grade (g/t)	Au Extraction (%)	Lime Addition kg/t
JR 183	Master Composite	4 hr. Pre-OX, 24 hr. Bulk Vat Leach	4.38	0.395	90.99%	1.42
JR 184	Master Composite	1.5kg/t CN, 250g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear & Diesel, P80 75 um	4.20	0.35	91.67%	2.13
JR 298	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear, P95 75 um	4.34	0.31	92.86%	1.83
JR 344	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um	4.11	0.31	92.46%	0.88
JR 345	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL , Shear, P95 75 um	4.09	0.29	92.91%	1.14

13.3.6 Determination of Optimum Grind

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

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

Table 13.8 Metallurgical Results for leach tests on master composite samples to determine optimum target grind size.

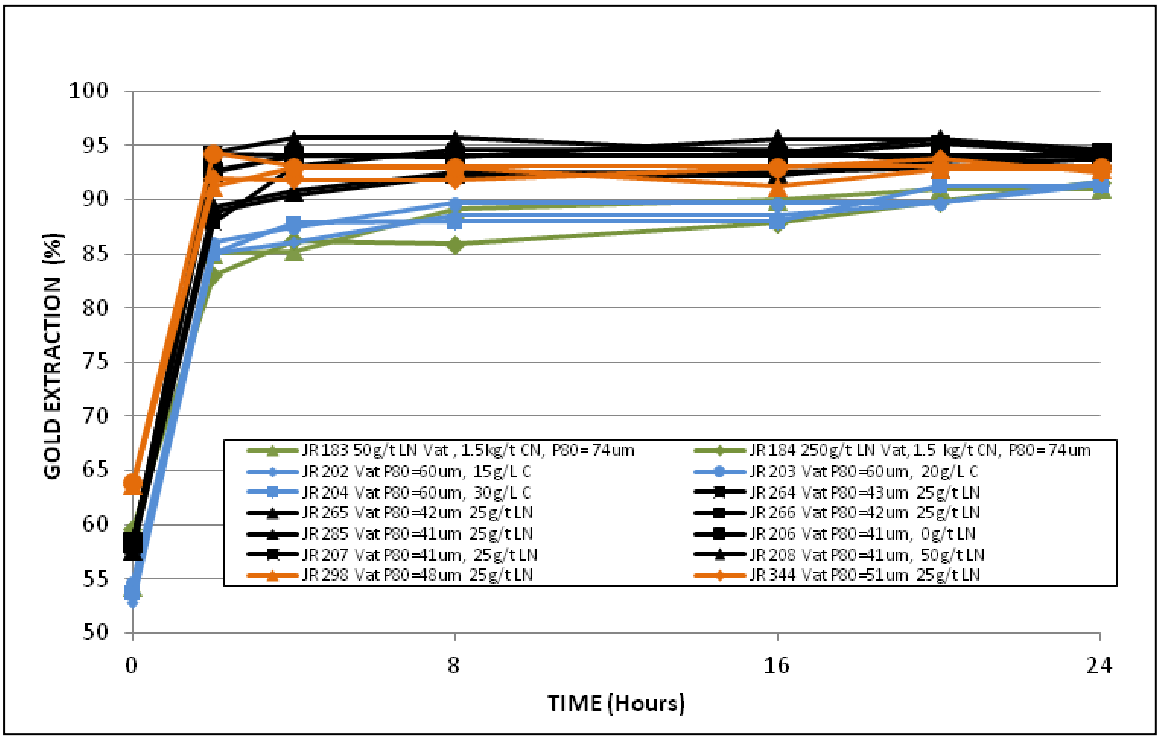
Test No	Feed	Test Conditions	Grind P80 (µm)	Calculated Head Grade (g/t)	Residue Au Grade (g/t)	Au Extraction (%)
JR 183	Master Composite	1.5kg/t CN, 50g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, P80 75um	74	4.38	0.40	90.99%
JR 184	Master Composite	1.5kg/t CN, 250g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear & Diesel, P80 75 um	74	4.2	0.35	91.67%
JR 202	Master Composite	1.5kg/t CN, 250g/t LN 4hr. Pre-Ox, 24hr. Bulk Vat Leach MidShear, P90 75um, 15g/L C	60	4.22	0.36	91.47%
JR 203	Master Composite	1.5kg/t CN, 250g/t LN 4hr. Pre-Ox, 24hr. Bulk Vat Leach MidShear, P90 75um, 20g/L C	60	4.07	0.34	91.65%
JR 204	Master Composite	1.5kg/t CN, 250g/t LN 4hr. Pre-Ox, 24hr. Bulk Vat Leach MidShear, P90 75um, 30g/L C	60	4.15	0.36	91.33%
JR 298	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 24 hr. Bulk Vat Leach for Detox, Shear, P95 75 um	48	4.34	0.31	92.86%
JR 344	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL , Shear, P95 75 um	51	4.11	0.31	92.46%
JR 345	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL , Shear, P95 75 um	51	4.09	0.29	92.91%
JR264	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C	43	4.30	0.24	94.42%
JR265	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C	42	4.27	0.23	94.61%
JR266	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C	42	4.27	0.24	94.38%
JR 206	Master Composite	0.5kg/t CN, 0g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um	41	4.25	0.27	93.65%
JR 207	Master Composite	0.5kg/t CN, 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um	41	4.28	0.26	94.04%
JR 208	Master Composite	0.5kg/t CN, 50g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um	41	4.33	0.27	93.76%
JR285	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um Scalped, 15g/l C	41	4.34	0.27	93.89%

The metallurgical results of all the vat leach tests conducted at grind sizes of approximately 80% passing 74 µm, 60µm, 42µm and 50µm are presented in Table 13.8 above. The results indicated the following:

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

As can be seen from Table 13.8, the average CIL recovery was found to increase at finer grind with an overall recovery increase of 2.8% when the fineness of grind increased from 80% passing 75 µm to 80% passing 42µm. Based on the results presented in Table 13.8 and the leach curves as presented in Figure 13.8 it was apparent that the recovery benefit was significant when the fineness of grind was increased to 80% passing 42 µm.

Figure 13.8 Effect of Target Grind Size on Gold Recovery for the New Liberty Master Composite Sample

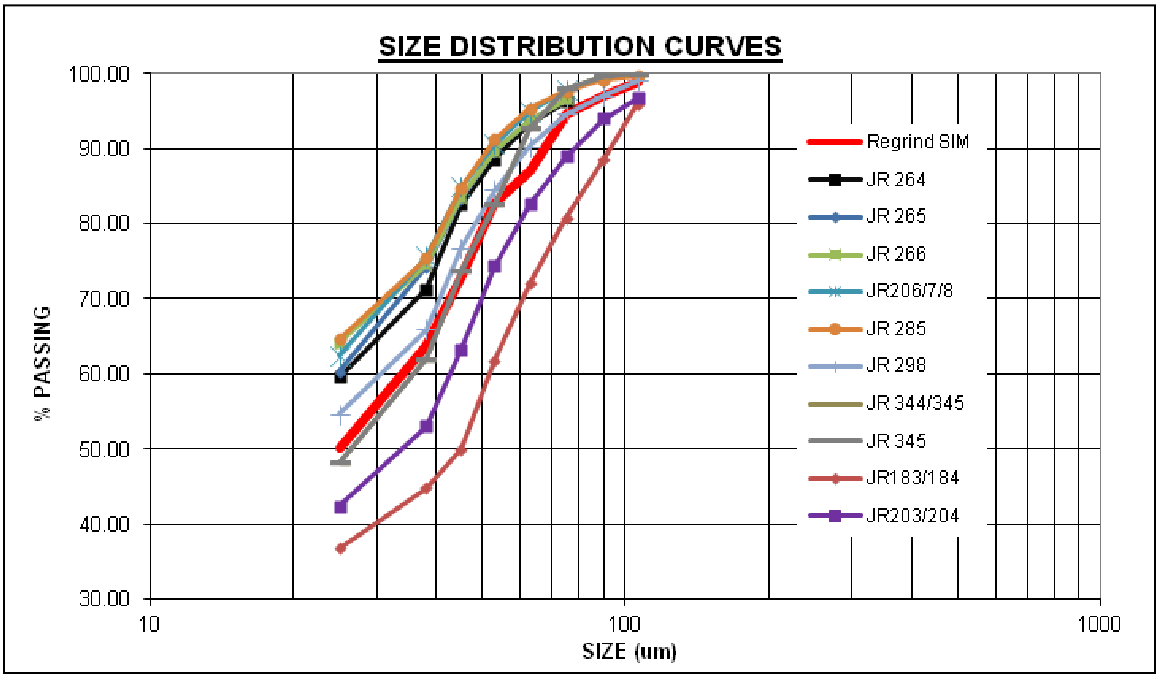


The full particle size distributions for each target grind size are presented in Table 13.9 below, with the corresponding size distribution curves as presented in Figure 13.9.

Table 13.9 Particle Size distributions for tests conducted on the master composite at various target grind sizes.

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

Figure 13.9 Size Distribution Curves for the Grind Optimization Tests



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

13.3.7 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.

13.3.7.1 Determination of Bond Work Index

The results of the Bond Work Index Test conducted on the master composite sample are presented in Table 13.10 below.

Table 13.10 Results of the Bond Work Index Test on the Master Composite

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

13.3.7.2 Levin and IsaMill Testing

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

- Levin tests
- IsaMill tests

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

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

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

13.3.7.3 Results of Levin Tests

Table 13.11 Levin Test Results

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

13.3.7.4 Results of IsaMill Tests

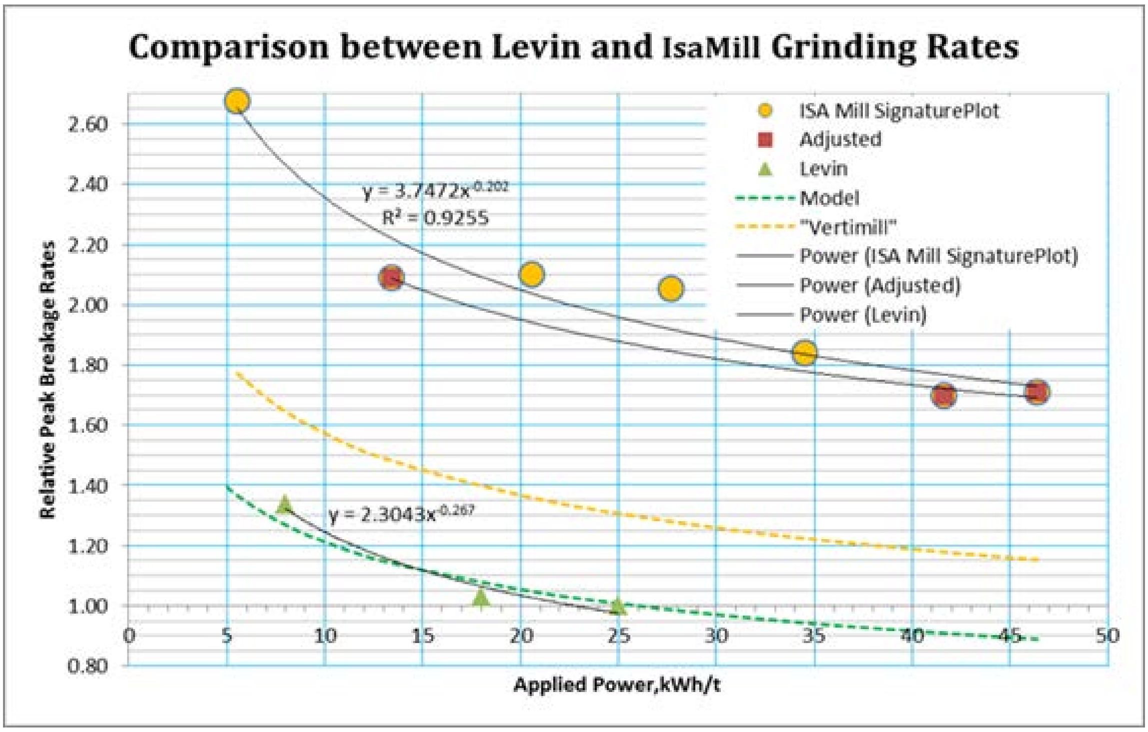
Table 13.12 Results of IsaMill Test Conducted 30 kWh/t

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

Table 13.13 Results of IsaMill Test Conducted 40 kWh/t

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

Figure 13.10 New Liberty Summary of Levin and IsaMill Test Work Result



Simulation of the Levin and IsaMill test work has shown significant improvement in milling efficiencies as compared to conventional mills. Figure 13.10 shows the relative difference in breakage rates (energy-based) on New Liberty regrind material. It can be seen that there are significant efficiency gains by 'matching media size' to feed size distribution. The Vertimill (using media 12.7 mm) estimated relative grinding rates have been shown in Figure 13.10 above. The relative grinding rates derived from test work were used to estimate the performance of a VertiMill unit, this has indicated that to achieve the target grind of 80% passing 50 µm the energy requirement is 6.7 kWh/t.

13.3.8 Evaluation of Leach Feed Density

At the end of the optimization phase two confirmatory flowsheet validation tests were performed to evaluate the effect of CIL feed density on overall gold extraction. The results of these tests are presented in Table 13.14 below.

Table 13.14 Evaluation of Overall Gold Extraction as a Function of Leach Feed Density

Test No	Feed	Test Conditions	Calculated Au Head Grade (g/t)	Residue Au Grade (g/t)	Au Extraction (%)	NaCN Addition kg/t	Lime Addition kg/t
JR 344	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 45% Solids, 24 hr CIL, Shear, P95 75 um	4.11	0.31	92.46%	0.50	0.88
JR 345	Master Composite	0.5kg/t CN, 25g/t LN, 4 hr. Pre-OX, 40% Solids, 20 hr CIL, Shear, P95 75 um	4.09	0.29	92.91%	0.50	1.14

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

13.3.9 Diagnostic Leach Tests

Multi-stage sequential diagnostic gold leach test work was conducted on two 1 kilogram subsamples of the mater composite sample. The results of these diagnostic leaches are presented in Table 13.15 below.

Table 13.15 Results of the Diagnostic Leach Tests Conducted on the Master Composite

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

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

13.3.10 Whole Ore CIL Test

In order to provide an indication of the expected plant recovery without the gravity concentration circuit in operation a whole ore CIL test was performed at a target grind of 80% passing 50 μm. The original master composite sample had been depleted so a new composite was generated using the same material that was used to generate the original master composite sample. The results of this whole ore CIL test are presented in Table 13.16 below.

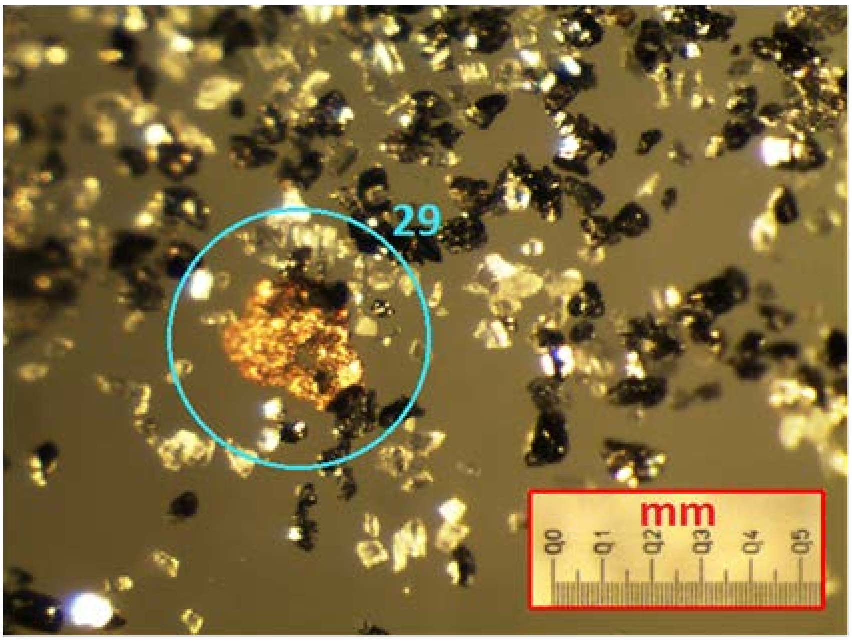
Table 13.16 Results of the Whole Ore CIL Test

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

The feed material was assayed four times to obtain an indicated head grade of 3.62 g/t – 6.51 g/t. The calculated head grade was determined to be 4.19 g/t. The large amount of variation in the assayed head grade and discrepancy between the assayed and calculated head grade is indicative of the presence of coarse gold. The leach residue grade for this test was 1.13 g/t with an overall extraction of 73.3%. The decrease in recovery was attributed to the presence of coarse gold particles that were not removed by gravity concentration and illustrates the importance of the gravity recovery circuit.

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

Figure 13.11 Coarse Gold Flake in Gravity Concentrate



13.4 Variability Test Work

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

The following optimized leach conditions were used:

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

During the variability test work, the presence of coarse nugget gold was evident from the comparison of the measured head assays (as presented in Table 13.2) and calculated head assays. For the purpose of determining the test recovery the calculated head grades were used, as this is the most accurate reflection of how plant feed grade and recovery will be determined for full scale plant operations. It is also a better calculation of recovery because it's based on more reliable product weights and assays.

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

The results of the variability tests conducted at a target grind of 80% passing 50µm (Refer P90 in ALS report) are presented in Figure 13.12 and Table 13.17 below.

Table 13.17 Metallurgical results for Variability Testing Conducted at Grind of 80% Passing 50 Microns

Test No	Sample ID	Test Conditions	Grind P80 (µm)	Calculated Au Head Grade (g/t)	Residue Au Grade (g/t)	Gravity Extraction (%)	CIL Extraction (%)	Overall Au Extraction (%)	NaCN Addition (g/t)	Lime Addition (g/t)
JR 279	K156	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	54	5.57	0.36	63.93%	82.33%	93.63%	0.61	4.87
JR 280	K233	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	47	5.44	0.36	72.48%	76.29%	93.47%	0.66	15.91
JR 281	K236	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	47	4.47	0.71	39.89%	73.58%	84.12%	0.61	5.89
JR 282	K287	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	71	4.24	0.34	72.84%	70.48%	91.98%	0.51	7.40
JR 283	K485B	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	52	7.66	0.22	78.98%	86.65%	97.19%	0.58	4.55
JR 284	K492	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	57	3.69	0.30	72.22%	71.22%	92.01%	0.54	13.43

The actual grind for the variability samples varied from 80% passing 47µm-71µm.The overall gold recovery for the six variability composite samples ranged from 84.1% - 97.2%. The overall recovery was comprised of the gravity circuit recovery and the cyanide carbon in leach (CIL) recovery. Gravity recovery ranged from 39.4% -79.0%, while the CIL recovery ranged from 70.5% -86.7%. The leach residue grades achieved ranged from 0.22 g/t to 0.71 g/t.

The residue grade of 0.71 g/t was achieved for test JR281 which was conducted on material from the eastern pit (K236). This residue grade was noted as being an outlier when compared to the variability testing conducted at P80 50µm. Further to this the result of this test was compared to previous Mintek phase I variability testing conducted at P80 75µm on material from the eastern pit which again confirmed this test as an outlier as can be seen in Table 13.18 below:

Table 13.18 Comparison of Variability Test Results for Material from the Eastern Pit

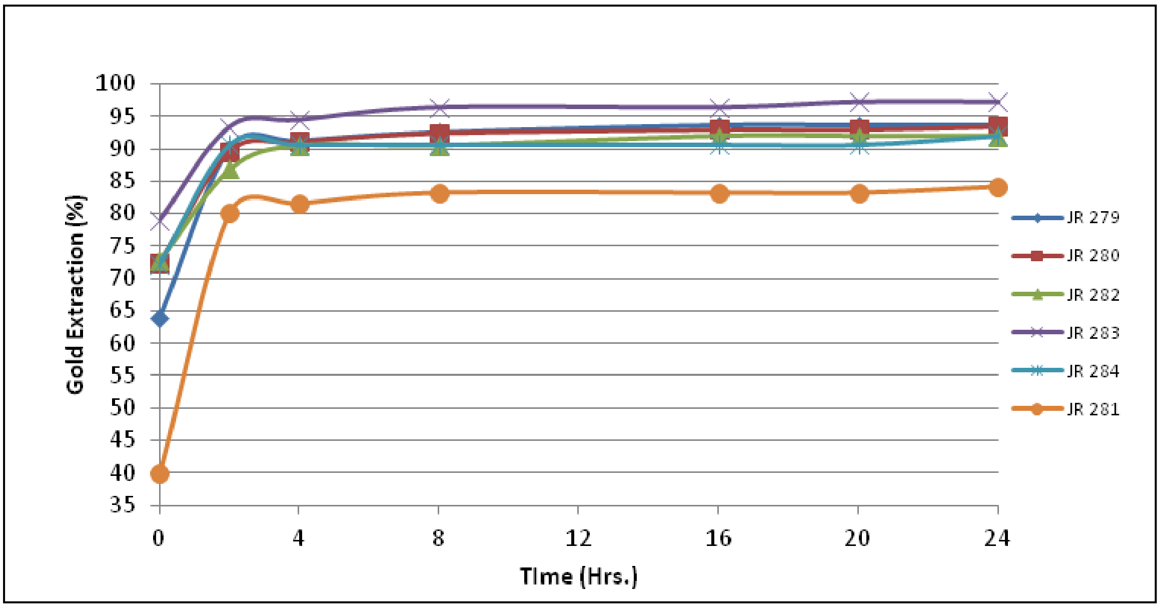
Test No	Feed	Test Conditions	Target Grind	P80	Calculated Head Grade (g/t)	Residue Au Grade (g/t)	Au Extraction (%)
Phase 1	K201	Whole Ore CIL	P80	75	4.33	0.19	95.61%
Phase 1	K202	Whole Ore CIL	P80	75	1.92	0.22	88.54%
JR281	K236	25g/t LN, 4hr. Pre-Ox, 24hr.	P90	47	4.47	0.71	84.12%

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

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

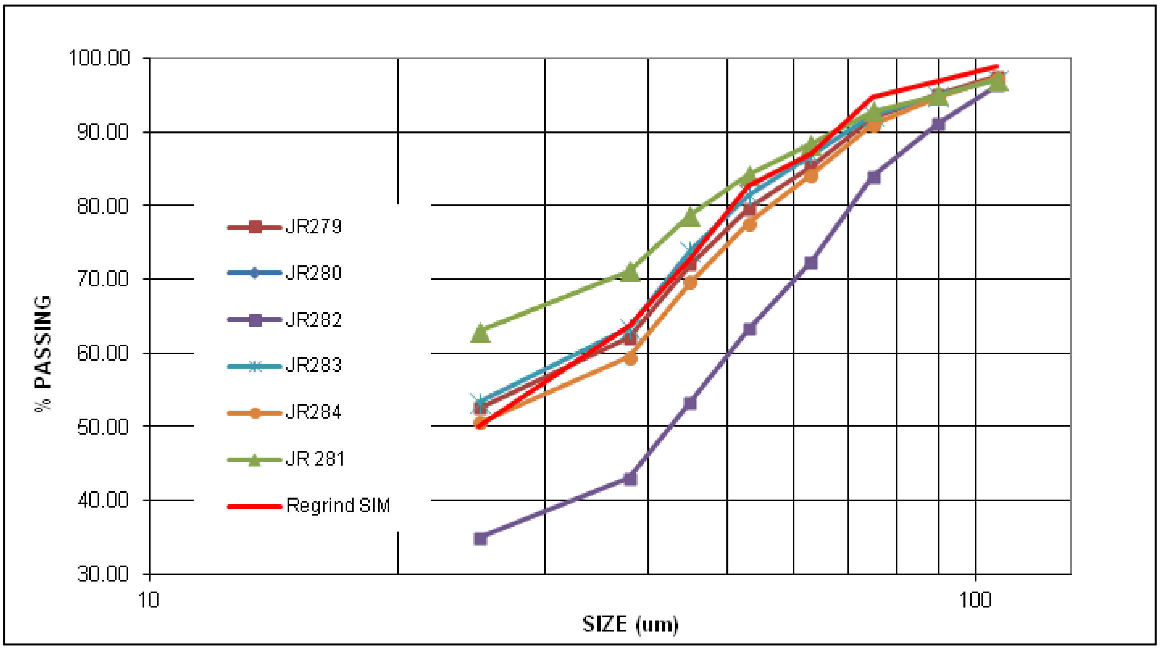
The results of the lime optimization testing are presented in Section 13.3.5.

Figure 13.12 Gold Recovery for New Liberty Variability Tests Conducted at a Target Grind Size of 80% Passing 50 µm



In order to establish the feasibility of achieving the recoveries as determined by the variability test work conducted at a target grind of 80% passing 50 µm the size distribution curves for the feed material to each test were plotted. These results where compared to the simulated 80% passing 50 µm PSD for a regrind application based on the results of Levin test work conducted at ALS. These curves are presented in Figure 13.13 below. It is apparent that the size distributions for these variability tests can be replicated in a full scale regrind application, as can be seen by comparing the simulated PSD to the actual PSD.

Figure 13.13 Size Distribution Curves for Variability Tests Conducted at a Target Grind of 80% Passing 50 µm



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

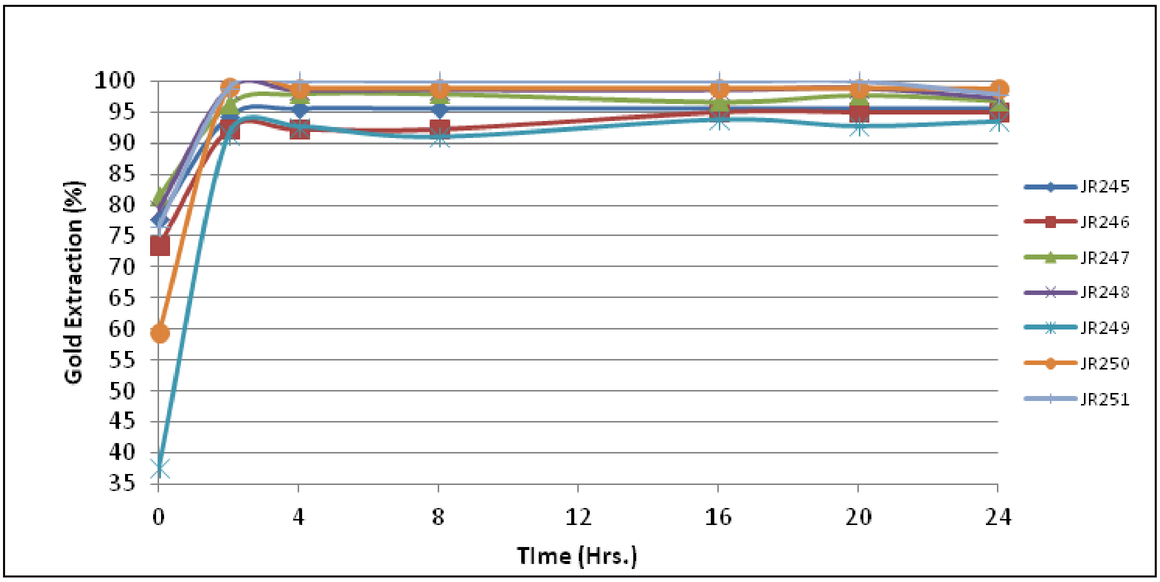
The results of the variability tests conducted at a target grind of 80% passing <20 µm (Refer P100 in ALS report) are presented in Figure 13.14 and Table 13.19 below.

Table 13.19 Metallurgical results for Variability Testing Conducted at a Target Grind of 80% Passing 20 Micron

Test No	Sample ID	Test Conditions	Target* Grind (µm)	Calculated Au Head Grade (g/t)	Residue Au Grade (g/t)	Gravity Extraction (%)	CIL Extraction (%)	NaCN Addition (g/t)	Lime Addition (g/t)
JR245	K287	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 11	21	4.72	0.21	77.77%	79.99%	0.50	7.12
JR246	K490	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 12	31	2.24	0.11	73.44%	81.51%	0.83	6.74
JR247	K492	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 13	17	4.59	0.14	81.42%	83.58%	0.92	15.92
JR248	K233	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 14	<15	7.07	0.20	79.31%	86.33%	0.63	5.00
JR249	K236	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 15	<15	4.15	0.27	37.66%	89.56%	0.75	7.89
JR250	K485B	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 16	<15	9.43	0.11	59.38%	97.13%	0.64	5.32
JR251	K156	25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, pH 17	<15	5.77	0.12	76.57%	91.12%	1.92	5.50

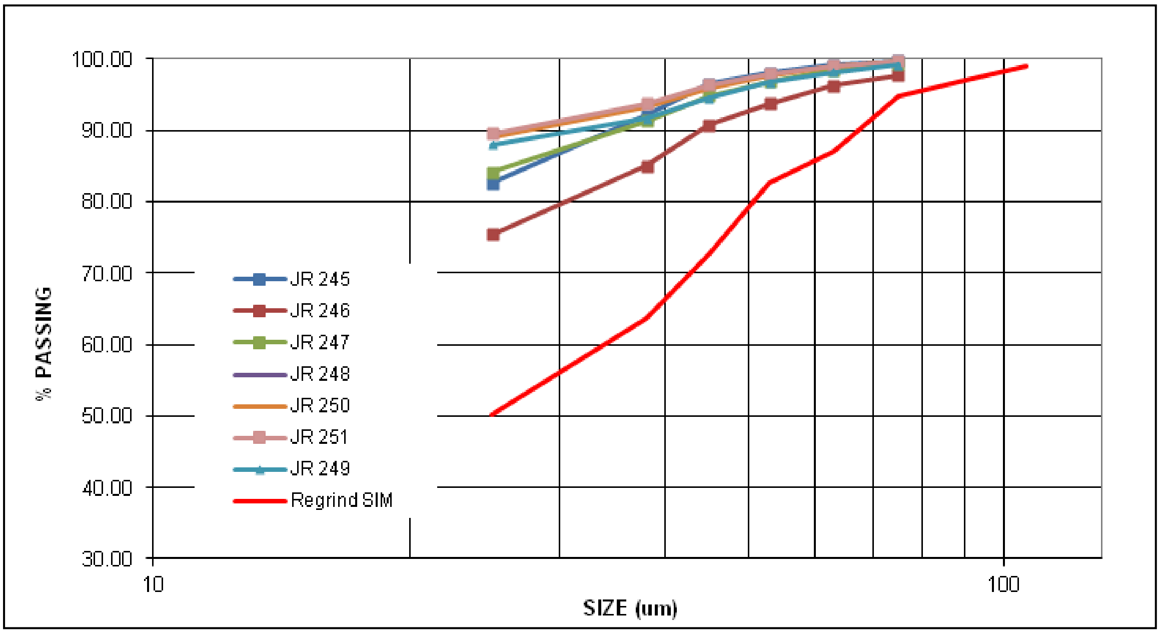
The actual grind for the variability samples varied from 80% passing 31µm to an estimated 15 µm for the remainder of the report this grind is referred to as 80% passing <20 µm. The overall gold recovery for the seven variability composite samples ranged from 93.5% - 98.8%. The overall recovery was comprised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

Figure 13.14 Gold Recovery for the New Liberty Variability Samples at a Target Grind Size of 80% Passing 25 Micron



In order to establish the feasibility of achieving the recoveries as determined by the variability test work conducted at a target grind of 80% passing <20 µm (P100) the size distribution curves for the feed material to each test were plotted. These results where compared to the simulated 80% passing 50 µm PSD for a regrind application based on the results of Levin test work conducted at ALS. These curves are presented in Figure 13.15 below. It was apparent that the size distributions for these variability tests would prove difficult to replicate in a full scale regrind application, due to the large proportion of fines present with more than 80% passing 20 µm as compared to the simulated 50% passing 25 µm. For this reason these recoveries will prove difficult to replicate as this grind almost represents pulverization.

Figure 13.15 Size Distribution Curves for the Variability Tests Conducted at 100% Passing 75 µm



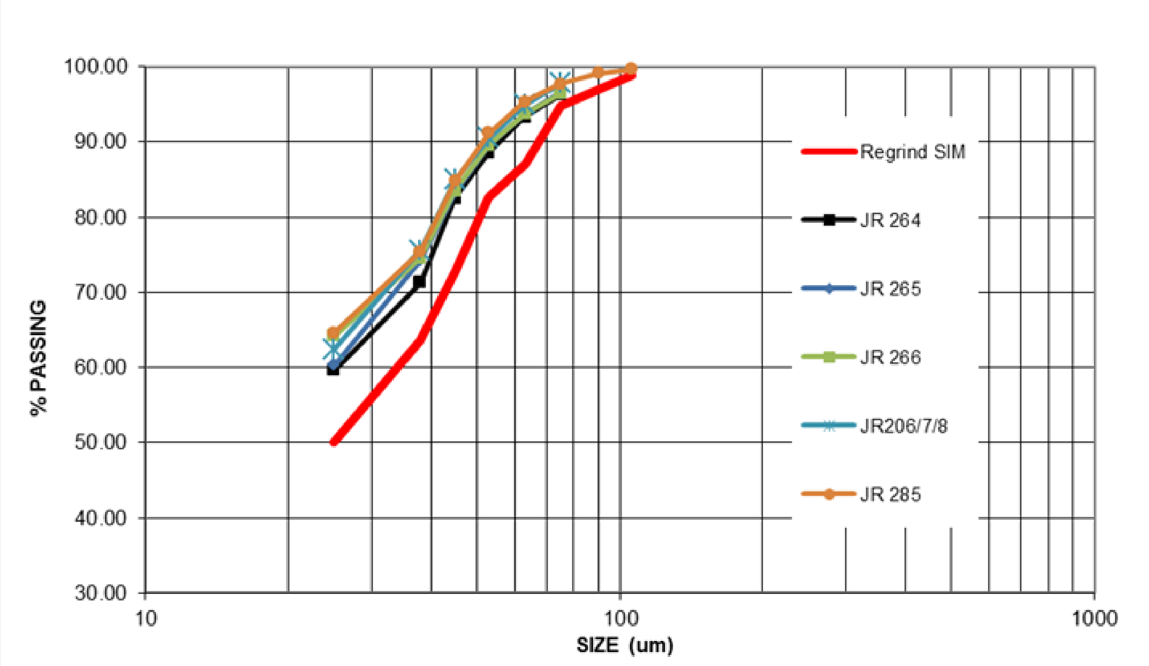
13.5 Selection of Mill Grind at 80% Passing 45 Microns

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

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

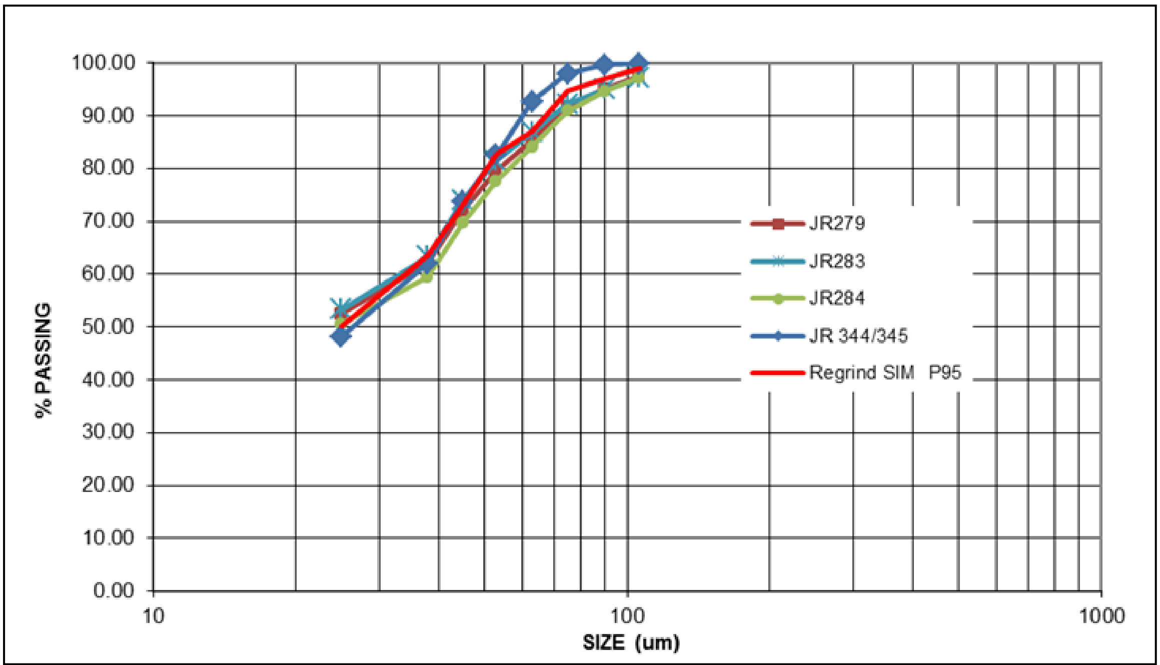
The simulated performance for the VertiMill with a power input of 6.7 kWh/t has been plotted relative to the PSD's achieved for all the tests conducted at an approximate grind of 80% passing 42 µm in Figure 13.16 below:

Figure 13.16 Size distribution Curves for the Composite Tests Conducted at 80% Passing 42µm



The simulated performance for the VertiMill with a power input of 6.7 kWh/t has been plotted relative to the PSD's achieved for all the tests conducted at an approximate grind of 80% passing 50 µm in Figure 13.17 below:

Figure 13.17 Size Distribution Curves for the Variability and Composite Testing at a Grind of 80% Passing 50 Micron



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

Testing conducted at 80% passing 42µm is presented in Table 13.20, while testing conducted at 80% passing 50µm is presented in Table 13.21. Comparison of Table 13.20 and Table 13.21 shows a recovery differential of 0.60% between the average recoveries achieved, which is attributed to the finer grind with a higher percentage of 25 micron material in tests conducted at 80% passing 50 micron.

The diagnostic leach tests as presented in Table 13.15 show that, there is 5.51% fine disseminated gold locked in the minus 20 µm size fraction, which supports liberation at finer grind sizes and this would have to be achieved with a Vertimill.

Table 13.20 Metallurgical Results for Composite Testing Conducted at a Grind of 80% Passing 42 Micron

Test No	Feed	Test Conditions	P80	Calculated Au Head Grade (g/t)	Residue Au Grade (g/t)	Au Extraction Test Work (%)
JR 206	Master Composite	0.5kg/t CN, 0g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um	41	4.25	0.27	93.65%
JR 207	Master Composite	0.5kg/t CN, 25g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um	41	4.28	0.26	94.04%
JR 208	Master Composite	0.5kg/t CN, 50g/t LN, 4hr. Pre-Ox, 24hr. Bulk Vat Leach Shear, P100 75um	41	4.33	0.27	93.76%
JR 264	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C	43	4.30	0.24	94.42%
JR 265	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C	42	4.27	0.23	94.61%
JR 266	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um, 15g/l C	42	4.27	0.24	94.38%
JR 285	Master Composite	0.5kg/t CN, 25g/t LN 4hr. Pre-Ox, 24hr Bulk Vat Leach Shear, P95 75um Scalped, 15g/l C	41	4.34	0.27	93.89%
Average						94.11%

Table 13.21 Metallurgical Results for Variability and Composite Testing at a Grind of 80% Passing 50 Micron

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

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

13.6 Cyanide Destruction Test Work

13.6.1 SO_2/Air Cyanide Destruction Test Work

An initial series of SO_2/Air cyanide destruction test work was conducted on the product of bulk leach tests conducted on the master composite at a cyanide addition of 1.5 kg/t. These initial tests are presented in Table 13.22 below and indicated that at a CNwad level in the leach effluent stream of 162.8 ppm an SO_2: CN ratio of 4:1 was not sufficient to reduce CNwad levels in the cyanide destruction product stream to below 50 ppm.

Table 13.22 Results of the Initial Scoping Tests Conducted for the SO_2/Air Cyanide Destruction Process

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

Once the optimum leach conditions had been established and a lower cyanide addition of 0.5 kg/t was determined to be optimum, a new SO_2/Air cyanide destruction test was conducted on the product of leach test JR298. The test as presented in Table 13.23 indicated that at a CNwad level in the leach effluent stream of 70.6 ppm an SO_2: CN ratio of 4:1 was sufficient to reduce CNwad levels in the cyanide destruction product stream to 10 ppm which is well below the 50 ppm target that is specified by the international cyanide code of practice.

Table 13.23 Results of the SO_2/Air Cyanide destruction Tests Conducted for the on Leach Effluent Generated Using the Optimized Leach Conditions

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

13.6.2 Hybrid SO_2/Air Cyanide Destruction Test Work

An initial series of hybrid SO_2/Air (Includes a carbon contact stage) cyanide destruction test work was conducted on the product of bulk leach tests conducted on the master composite at a cyanide addition of 0.5 kg/t. These initial tests as presented in Table 13.24 indicated that for a 3 stage continuous test at a CNwad level in the leach effluent stream of 88.1ppm a reagent suite as indicated in the table below, was sufficient to reduce CNwad levels in the cyanide destruction product stream to below 2.5 ppm.

Table 13.24 Results of the initial scoping tests conducted for the hybrid SO_2/air cyanide destruction process (conducted on leach effluent generated using the optimized leach conditions).

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

Further to this a series of 4 stage batch hybrid SO_2/Air cyanide destruction tests were conducted on the leach effluent stream produced in each of the variability tests. The results of these tests are presented in Table 13.25 below. As can be seen from the results, low CNWad levels were achieved in all tests. It is worth noting that the feed material for these tests was produced from leaches conducted at a cyanide addition of 0.5 kg/t making these results incomparable to the results obtained for the initial series of SO_2/Air tests as presented in Table 13.23. These results are comparable to the results of the SO_2/Air test presented in Table 13.24.

Table 13.25 Results of the Hybrid SO_2/Air Cyanide Destruction Tests Conducted on the Effluent Stream from the Leach Variability Tests

GOLD ORE HYBRID CYANIDE DETOXIFICATION VARIABILITY TEST RESULTS								
Test No-Stage No.	Leach Residue Slurry	Sample ID	Test Conditions			Solution Assays		
			Slurry pH	Reagents Used: H_2SO_4 (as 100%) (kg/t)	Reagents Used: Lime (60% CaO) (kg/t)	Feed Effluent CN_P (mg/l)	Treated Effluent CN_P (mg/l)	Treated Effluent CN_{TOT} (mg/l)
Variability tests with nominally 550 kg/t SMBS, 300 kg/t copper sulphate pentahydrate, 300 kg/t ferric chloride and 500 kg/t HCl (as 100% HCl equivalent)								
H5 - S3	JR245	K287	6.63	7.41	0.0	126.2	1.01	2.41
H5 - S4			9.00	0.0	0.51	1.01	0.99	2.39
H6 - S3	JR246	K490	6.95	8.90	0.0	204.2	0.99	2.39
H6 - S4			9.00	0.0	0.53	0.99	0.89	2.29
H7 - S3	JR247	K492	6.71	17.99	0.0	139.4	0.82	2.22
H7 - S4			9.00	0.0	0.76	0.82	0.67	2.07
H8 - S3	JR248	K233	6.28	6.31	0.0	72.2	0.47	1.87
H8 - S4			9.00	0.0	0.66	0.47	0.94	2.34
H9 - S3	JR249	K236	6.68	9.55	0.0	77.0	0.39	1.79
H9 - S4			9.00	0.0	1.35	0.39	1.61	3.01
H10 - S3	JR250	K485B	6.95	8.97	0.0	104.4	0.62	2.02
H10 - S4			9.00	0.0	0.91	0.62	2.48	3.88
H11 - S3	JR251	K156	7.05	10.60	0.0	221.0	1.49	2.89
H11 - S4			9.00	0.0	0.81	1.49	4.44	7.24

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

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

13.6.3 Scoping Tests for Arsenic Preparation

Test work on a decant solution from the hybrid SO_2/Air cyanide destruction effluent stream was tested to determine if arsenic could be precipitated from solution using ferric chloride. The results of these scoping tests are presented in Table 13.26 below.

Table 13.26 Results of the Arsenic Precipitation Scoping Tests

FERRIC CHLORIDE ARSENIC PRECIPITATION TESTS						
Test No.	Fe^{3+} Addition (mol/mol As)	Reagent Dosage (g/l effluent)	pH	Retention Time (min)	Feed Effluent As (mg/l)	Treated Effluent As (mg/l)
Gold Ore Hybrid Cyanide Detoxified Decant Solution						
HY1758.1	2.52	0.014	7.10	30	2.59	<0.1
HY1758.2	5.04	0.028	6.69	60	<0.1	0.30
HY1758.3	10.07	0.056	4.88	90	0.30	0.20

The New Liberty process plant design is to include one additional tank, post cyanide destruction to allow for the precipitation of Arsenic from solution using ferric chloride.

13.7 e-GRG Test Work Performed by Consep

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

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

The results of the GRG test work are presented below.

Table 13.27 e-GRG Test Work Head Assays

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

As can be seen in Table 13.28, there was a large amount of variation in the assayed head grade with an average reported value of 7.5 g/t. The average assayed head grade is higher than the calculated head assay of 3.80 g/t as presented in Table 13.28 and 4.54 g/t as presented in Table 13.29. This variation is indicative of the presence of coarse gold.

Table 13.28 GRG Test Results

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

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

Table 13.29 GRG with Concentrate Leaching Test Results

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

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

* concentrate grades calculated from leach tests

** recovery = gravity distribution to concentrates x leach dissolution

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

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

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

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

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

13.8 New Liberty Resource Estimate

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

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

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

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

13.8.1 Derivation of a Correlation Between Grade, Recovery and Mill Grind

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

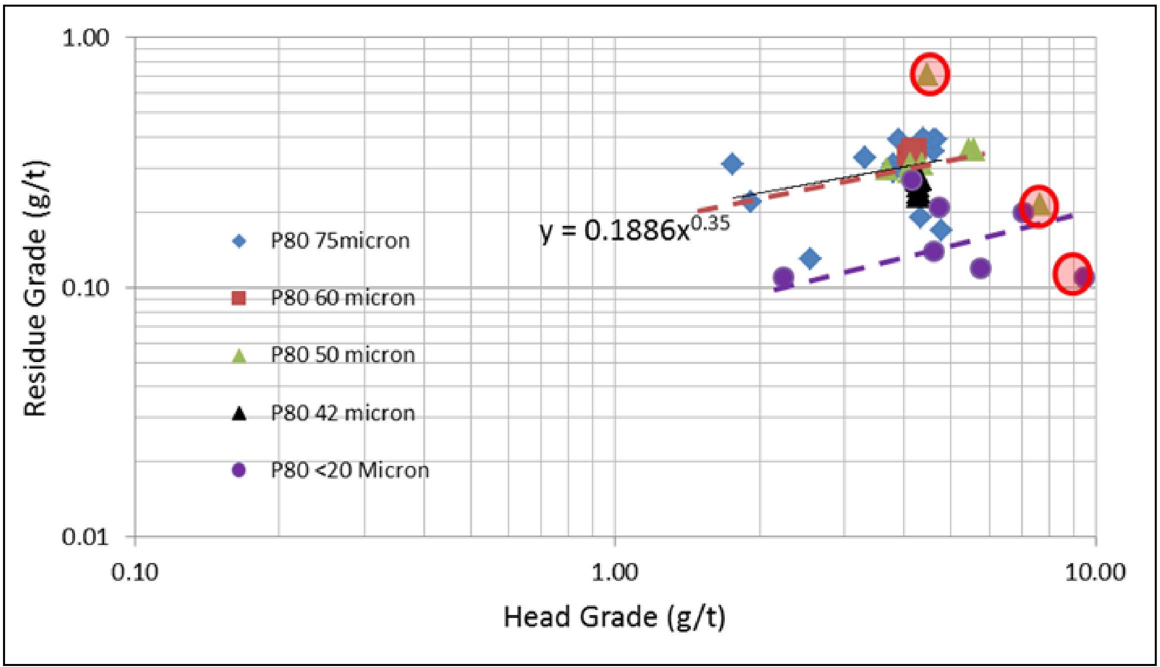
- Mintek CIL test work results for Phase 1, 2 and 7 from the previous feasibility study.
- ALS test work results on composite and variability samples

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

The relationship between final CIL residue grade and head grade for a target grind size of 80% passing 75 µm, 60 µm, 50 µm, 42 µm and <20 µm is presented in Figure 13.18. With the exception of the three data points as highlighted it would appear that there was an increase in final residue grade at higher head grades. Upon evaluation of the data it was decided that the correlation between final residue grade and head grade for each grind could be adequately described by the following equation:

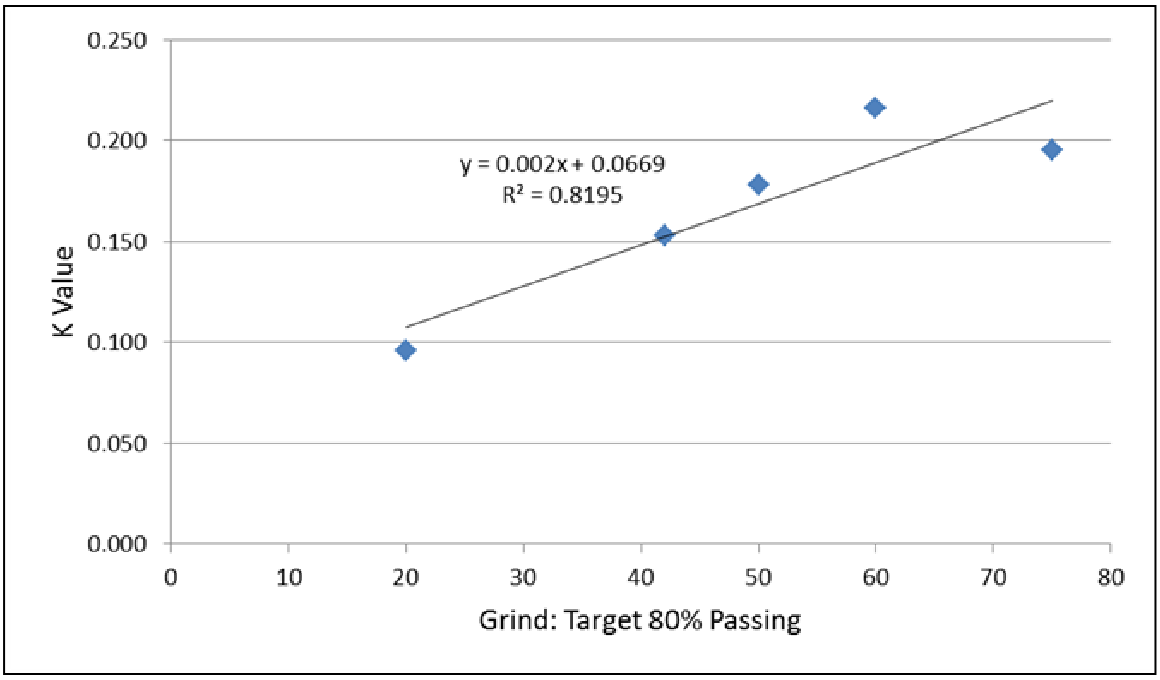
$$Final\ Residue\ \propto K(Head\ Grade)^{0.35}$$

Figure 13.18 Test Work Recovery as a Function of Grind



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

Figure 13.19 Derivation of Correlation Constants for each Target Grind



The correlation constants as determined from Figure 13.19 are presented in Table 13.30 below.

Table 13.30 Correlation Constants for Each Target Grind Size

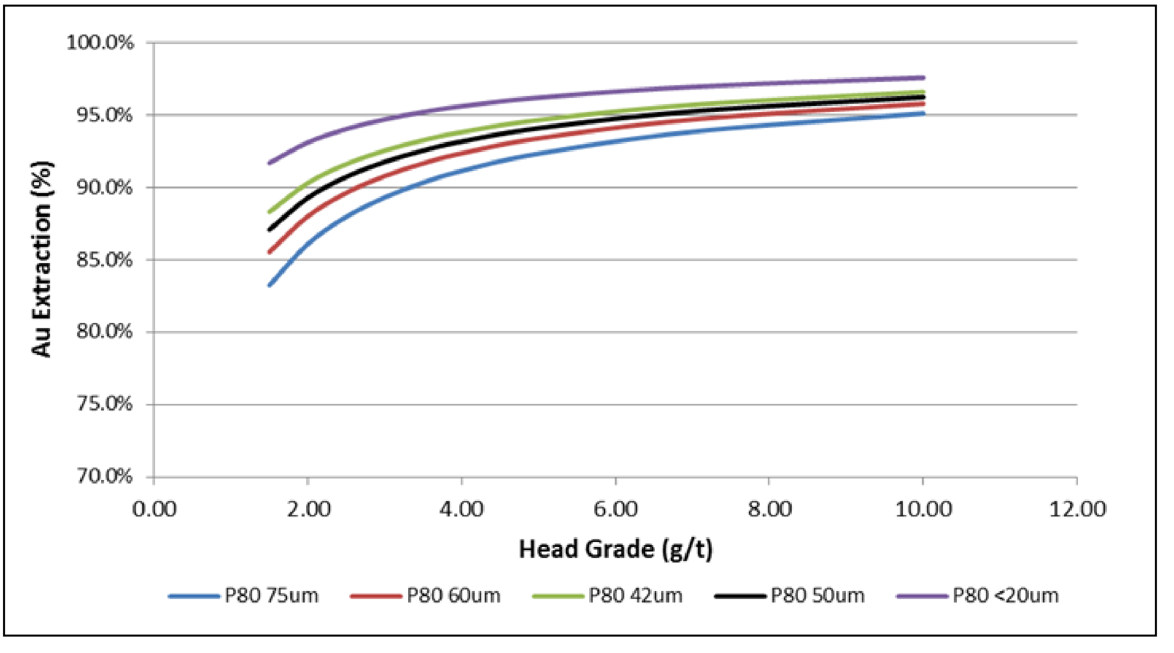
Grind	Constant (K)	
P80	Test Work	Model
75	0.195	0.217
60	0.216	0.187
50	0.178	0.167
42	0.153	0.151
20	0.096	0.107

Based on the constants as presented in Table 13.31 above the grade recovery correlation at each target grind size was found to be as follows.

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

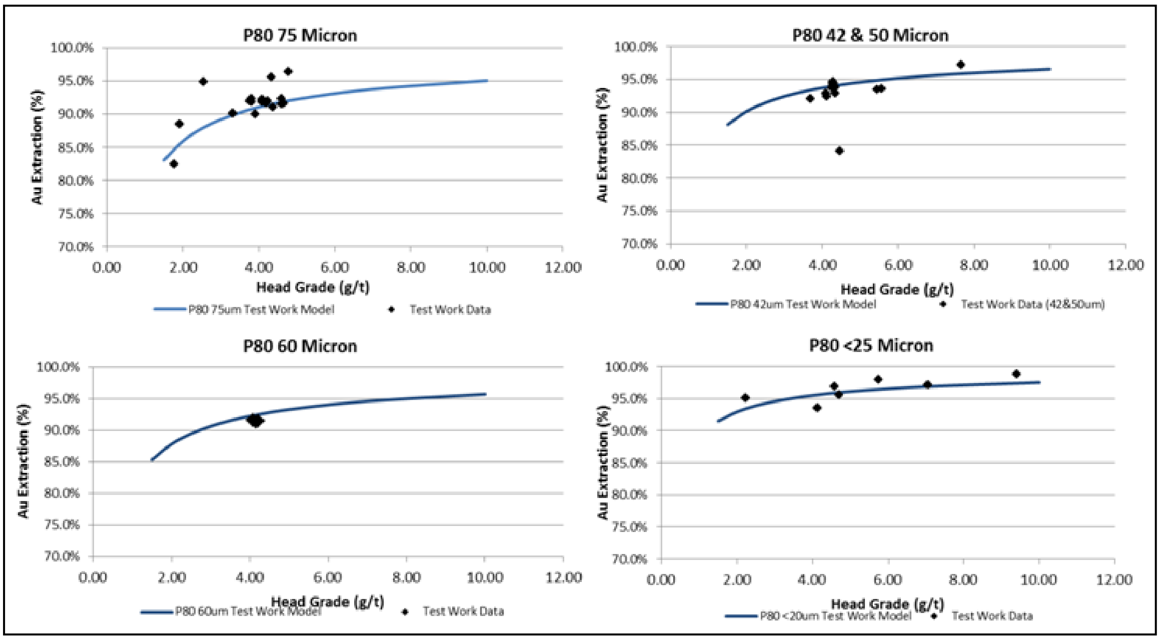
This modelled grade –recovery relationship is graphically presented in Figure 13.20 below.

Figure 13.20 Model Predicted Grade Recovery Curve at Each Target Grind Size



The grade recovery relationship as predicted by the model as derived above was compared to the Mintek Phase 1, 2 and 7 and ALS test work results and found to be in good agreement as presented in Figure 13.21 below.

Figure 13.21 Test Work Au Extraction Relative to Model Prediction



13.8.2 Monte Carlo Analysis

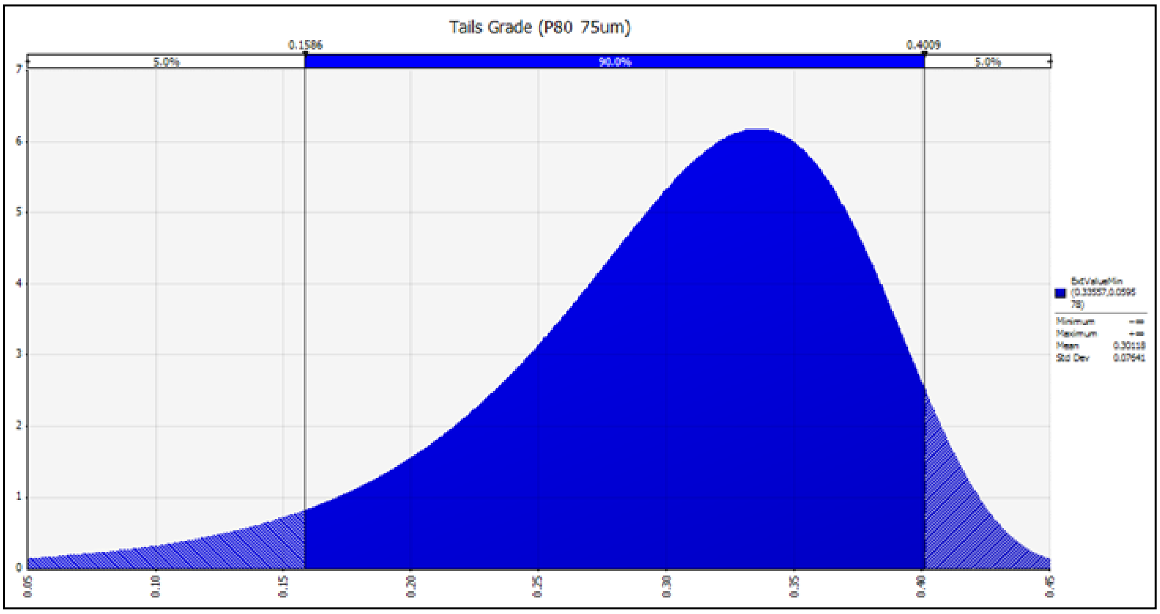
A Monte Carlo analysis was performed to confirm the recovery as determined from the grade recovery models derived from test work as presented in Section 13.8.1. The following data was used for the Monte Carlo analysis:

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

13.8.2.1 80% Passing 75µm Monte Carlo Distribution

The probability distributions for 80% passing 75 µm target grind are presented in Figure 13.22 below. The results show a mean residue grade of 0.30 g/t. The 90% confidence recovery range for all samples is shown on the graphs, and can be seen to provide a residue range of between 0.16 g/t – 0.40 g/t over this confidence interval.

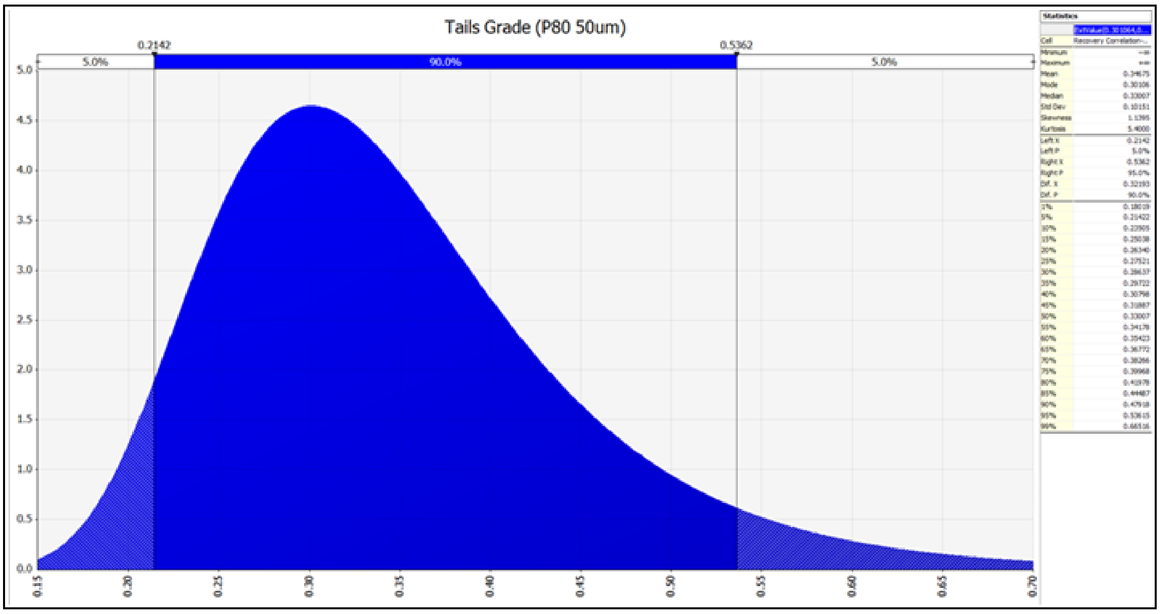
Figure 13.22 Monte Carlo Analysis of Test Work Residue Grades at 80% Passing 75 Micron



13.8.2.2 80% passing 50µm Monte Carlo Distribution

The probability distributions for 80% passing 50 µm target grind are presented in Figure 13.23 below. The results show a median residue grade of 0.33 g/t. The 90% confidence recovery range for all samples is shown on the graphs, and can be seen to provide a residue range of between 0.21 g/t – 0.54 g/t over this confidence interval.

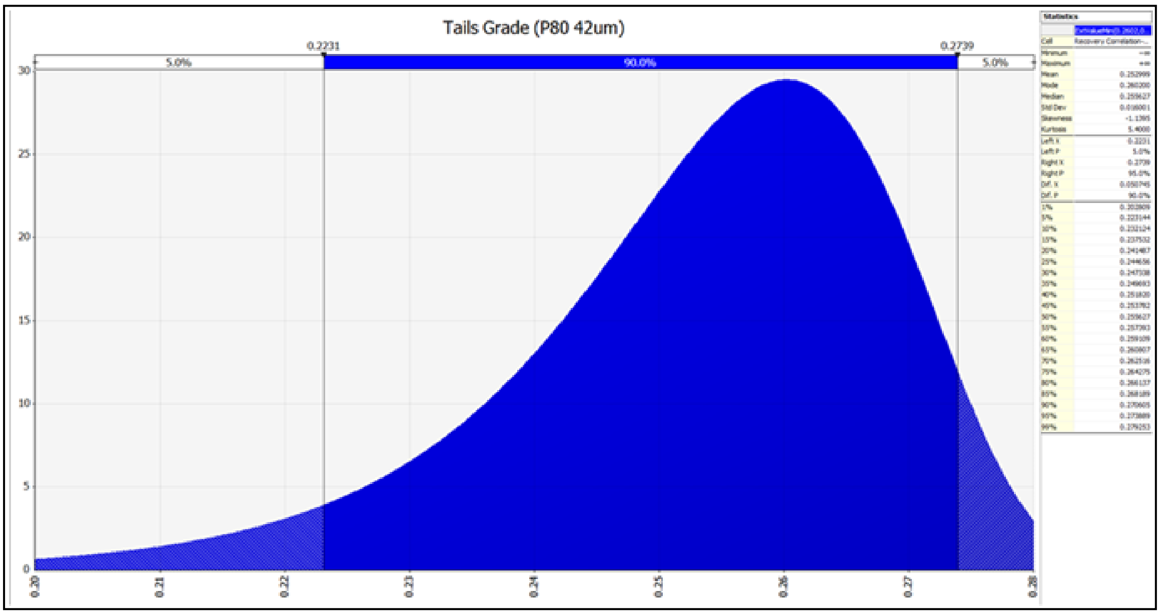
Figure 13.23 Monte Carlo Analysis of Test Work Residue Grades at 80% Passing 50 Micron



13.8.2.3 80% Passing 42 µm Monte Carlo Distribution

The probability distributions for 80% passing 42 µm target grind are presented in Figure 13.24 below. The results show a median residue grade of 0.25 g/t. The 90% confidence residue range for all samples is shown on the graphs, and can be seen to provide a recovery range of between 0.22 g/t – 0.27 g/t over this confidence interval. It should be noted that the recovery range provided here is based on all the composite tests conducted at a grind of 80% passing 41 µm-43 µm. Because these tests were conducted on the composite samples this analysis is an indication of repeatability.

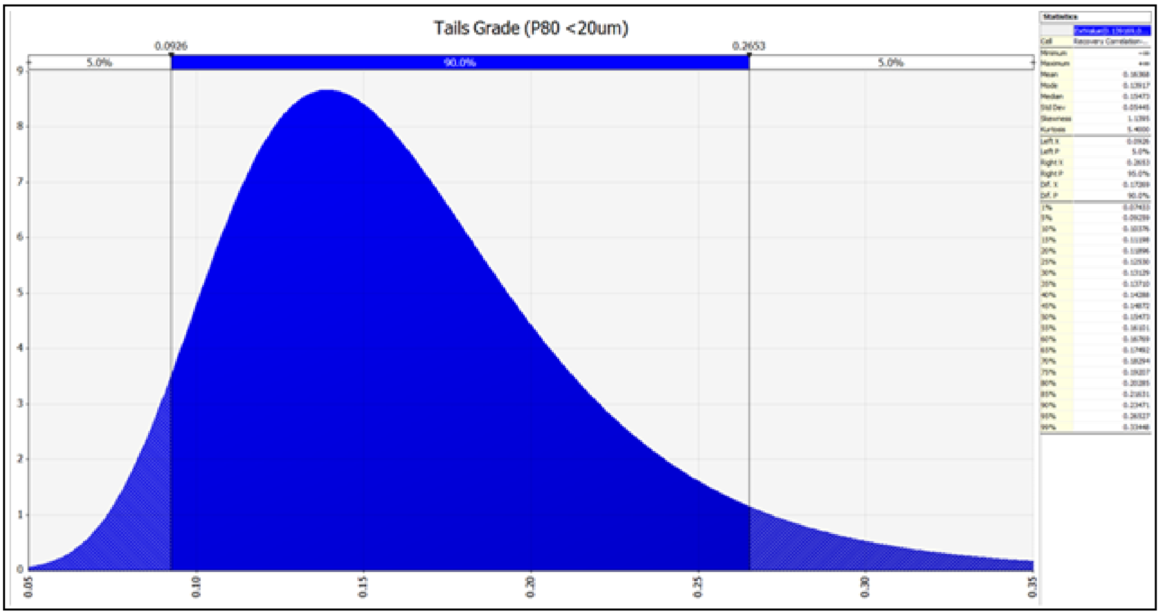
Figure 13.24 Monte Carlo Analysis of Test Work Residue Grades at 80% Passing 42 Micron



13.8.2.4 80% passing <20 µm Monte Carlo Distribution

The probability distributions for an 80% passing <20 µm target grind are presented in Figure 13.25 below. The results show a median residue grade of 0.15 g/t. The 90% confidence recovery range for all samples is shown on the graphs, and can be seen to provide a residue range of between 0.09 g/t – 0.27 g/t over this confidence interval.

Figure 13.25 Monte Carlo Analysis of Test Work Residue Grades at 80% Passing <20 Micron



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

The residue grade for each target grind size as determined by the Monte Carlo analysis was found to be in good agreement with the residue grade as predicted by the model presented in Section 13.8.1, for the range of head grades tested. This comparison is presented in Table 13.31 below.

Table 13.31 Comparison of Modelled CIL Reside Grades and Residue Grades as Determined by Monte Carlo Analysis

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

13.8.4 New Liberty Plant Recovery Estimate

Based on the current proposed mine production schedule, the estimated plant recovery for New Liberty at a target grind of 80% passing 45 µm is presented in Table 13.32. The recovery estimate is based on the expected range for the residue Au grade as determined by the correlation between head grade, grind and recovery as presented in Section 13.8.1. The recovery estimate presented in Table 13.33 includes a recovery discount figure which was calculated based on the following:

- Fixed carbon losses of 25 g carbon loss per ton milled at an estimated gold grade of 50 g/t.
- Fixed solution losses based on a CIL feed density of 45% solids (w/w) and a solution gold content of 0.05 g/L in the plant tailings stream.
- An assumed recovery loss of 0.5% for inefficiency of high shear oxygen addition in the pre-oxidation phase as compared to laboratory testing to account for Scale-up to plant conditions.

The three recovery discount factors above were combined to formulate a combined recovery discount for full scale plant operations as presented in Table 13.32 below.

Table 13.32 New Liberty Plant Recovery Estimate for a Target Grind of 80% Passing 45 Micron

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

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

14 MINERAL RESOURCE ESTIMATES

Aureus cannot give any assurance that estimated mineral resources will be recovered if a decision is made to proceed to production or that they will be recovered at the volume, grade and rates estimated. The failure of Aureus to achieve production estimates could have a material and adverse effect on its future cash flows, profitability, results of operations and/or financial condition. Estimated mineral resources could be materially affected by environmental, permitting, legal, title, socio-economic, political or other factors.

14.1 Overview and Approach

The primary estimation of mineral resources for the New Liberty deposits was undertaken in January 2012. Subsequently, following additional drilling mainly targeted between the Kinjor and Marvoe zones, an update of the mineral resource estimate was completed in April 2012 and reported in October 2012. Not all analytical tasks were repeated by AMC during the update (e.g. QA/QC and most statistical and variographic analyses), as the changes associated with the update drilling were not considered material to the manner in which the estimates were run.

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

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

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

14.2 Data Storage and Preparation

The sample database was provided to AMC by Aureus as a set of Microsoft Access database tables, as listed in Table 14.1. The final data available for use in the primary evaluation was received on 9 December 2011 and represents drilling up-to and including drillhole K375, whereas the updated database, received on 4 April 2012, represents drilling up to and including drillhole K438, as summarized in Table 14.2. Not all of the additional update holes were targeted on the resource mineralization.

Table 14.1 Sample Database Data Tables

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

The database tables were subjected to standard validation procedures. Geochemical fields provided in the database are summarized in Table 14.2. In the case of gold grades, the primary assay field was extracted for use in the resource estimate. The trench data was considered adequate for spatial viewing purposes but not of sufficient quality for use in the resource estimation.

Table 14.2 Geochemical Fields

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

14.3 Interpretations

14.3.1 Geology

Interpretations of geology have been restricted to the upper and lower boundaries which separate the ore-hosting ‘silicified metamorphosed ultrabasic suite’ (SMUS) from the enclosing migmatitic basement. The positions of the boundaries were determined as downhole drill pierce points based on logged lithological and stratigraphic unit codes.

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

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

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

The SMUS was modelled as an enclosed solid, with the hanging wall and footwall surfaces defined using downhole pierce point positions.

14.3.2 Mineralization

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

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

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

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

- Recognition that the eastern extent of the Latiff zone can be considered as a continuation of the Kinjor main hanging wall subzone.
- Justification for merging of Larjor with the western end of Latiff, although across the merged interval the mineralization is thin and poorly developed.
- Subdivision of the western (hanging wall) Marvoe zone into two subzones based on markedly different geometric and grade distribution characteristics.
- Treatment of the previously defined dual and parallel Marvoe zones as a single entity.

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

To facilitate both interpretation of the mineralization and subsequent resource modelling, a coding system, using the MINZONE field name, had been developed during previous modelling to distinguish the various zones and sub-zones. The assigned MINZONE codes have been modified to account for the changes listed above and are summarized in Table 14.3, along with the corresponding zone dimensions.

Table 14.3 Mineralized Zone Codes (MINZONE Field)

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

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

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

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

14.3.3 Oxidation

Interpretation of the weathered horizon has been based on geological logging from near-surface intersections in diamond core drilling, drawn on sections at 25 m intervals. The logged weathering codes record a progressive four-step diminution of weathering, as shown in Table 14.5, from completely weathered nearer to surface, down to fresh rock at depth.

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

Table 14.4 Weathering Zone Codes (WEAZONE Field)

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

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

14.3.4 Topography

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

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

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

Cell Model Construction and Coding

A cell model of the project deposits was constructed from a suite of sub-models using a base configuration of 10 m (Easting) × 5 m (Northing) × 10 m (RL) parent cells, as shown in Table 14.5. The cell geometry was selected on the basis of the overall drill spacing in the relatively near-surface portions of the deposit (to 200 m depth).

Table 14.5 Model Cell Parameters

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

For the mineralized zone sub-model, each wireframe was filled with cells such that parent cells were permitted to split along bounding surfaces down to the minimum dimensions shown in Table 14.6. Cells were coded with the relevant MINZONE code.

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

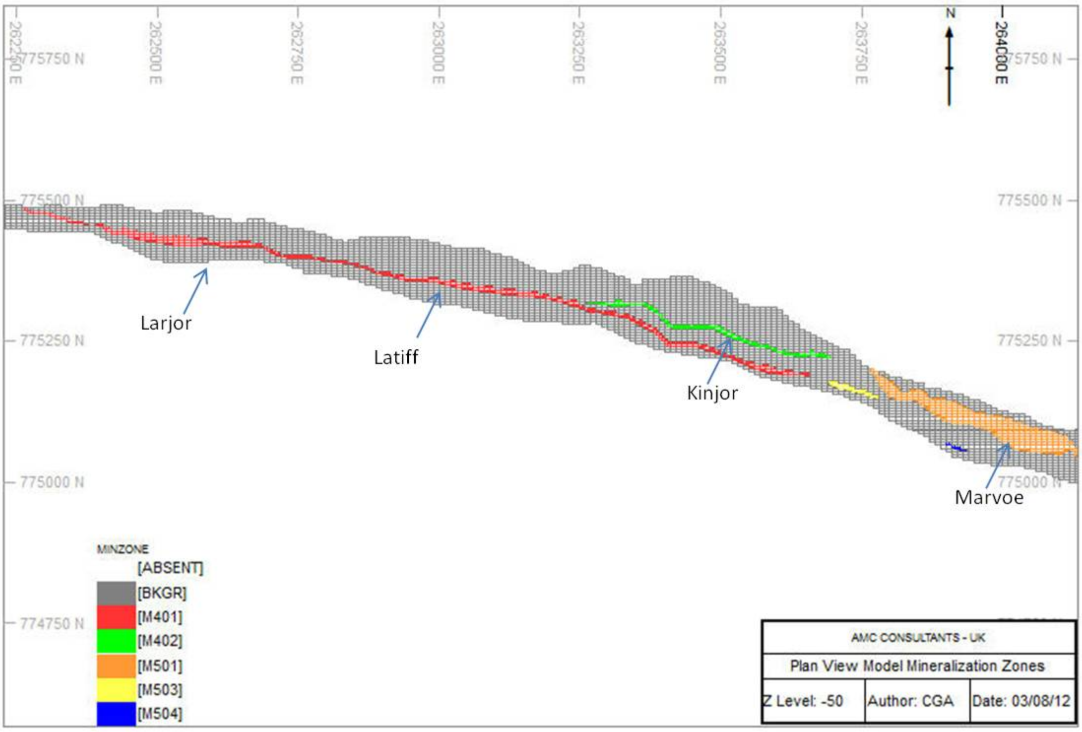
Sub-models representing both topography and weathered material were created by building cells above the respective triangulated surfaces, with horizontal and vertical subcell minima of 2 m. The weathering model was coded with the WEAZONE field set to ‘WEAT’.

The mineralization, stratigraphic, weathering and topographic (air) sub-models were combined to produce a unified and coded model consisting of mineralization zones set within the SMUS unit, flagged by weathering code, and trimmed along the topographic surface. The various model code and attribute fields are listed in Table 14.6 and Figure 14.1 shows a schematic relationship of the mineralized zones within the SMUS unit.

Table 14.6 Coded Model Field Descriptions

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

Figure 14.1 Schematic Plan View of Model Mineralized Zones



14.4 Sample Coding

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

Prior to coding, AMC excluded a number of holes, listed in Table 14.7, including some older ones which were rejected because of inconsistencies and which, based on more recent information, were indicative of data errors.

Table 14.7 Rejected Drillholes

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

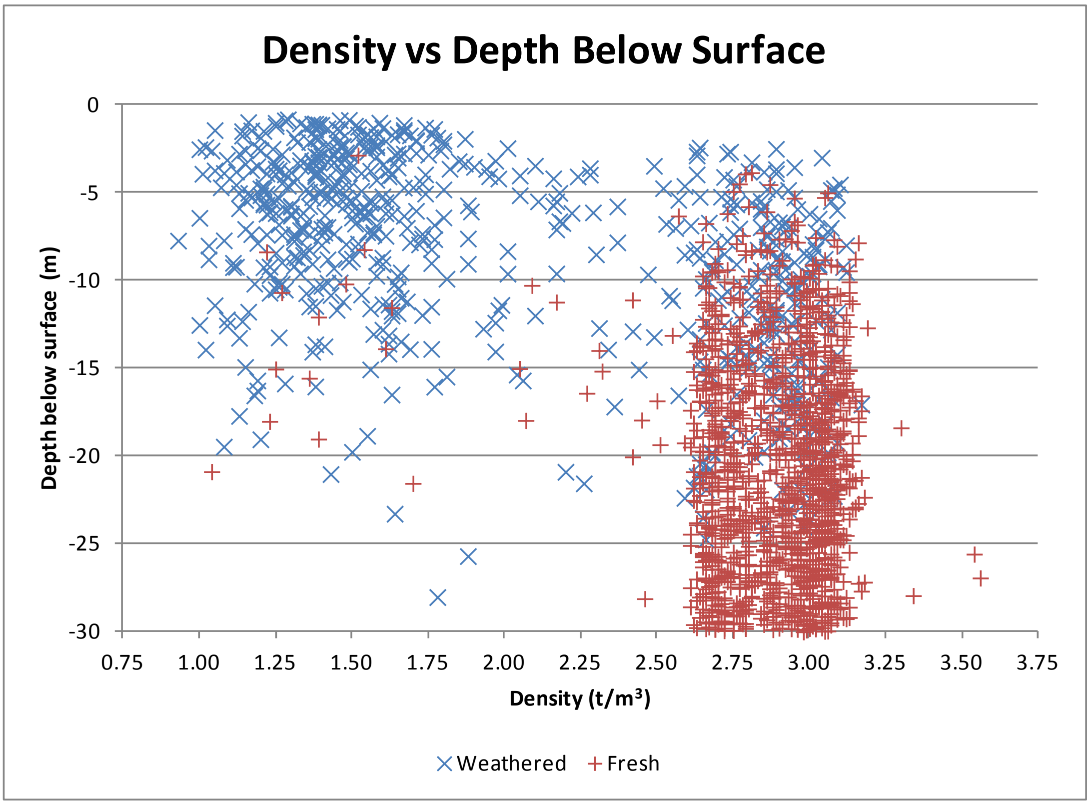
The remaining drillhole samples were coded according to the relevant MINZONE codes using a tagging process which incorporated the same table of downhole intercepts used to define the mineralized zone wireframes. To achieve stratigraphic coding of the field STRZONE, samples were ‘captured’ within the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

The resulting sample coding is consistent with the cell model codes shown in Table 14.7.

14.5 Bulk Density Evaluation

Of the 13,547 bulk density measurements used in the evaluation, 1,361 fall within fresh mineralized zones. A review of the spatial distribution of values showed a clear distinction between weathered zone and fresh sample densities, but exhibited no particular trends or associations internally within these horizons. Figure 14.2 shows the variation with depth of densities, for both weathered and fresh rock (in the first 30 m below surface). Overlaps between the two clusters of points reflect difficulties at times in partitioning drill intervals according to weathering, as well as the natural variability in weathering intensity.

Figure 14.2 Density Variation with Depth



Mean density values by mineralization zone and weathering horizon are summarized in Table 14.8.

Table 14.8 Mean Bulk Density Values

WEAZONE	Zone	Bulk Density	
		Number of samples	Mean (t/m^3)
WEAT	ALL	641	2.01
FRSH	M401	617	2.99
	M402	130	2.94
	M501	564	2.97
	M503	28	2.96
	M504	22	2.97
	BKGR	11545	2.90

14.6 Sample Compositing and Statistics

14.6.1 Composite Selection

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

14.6.2 Statistical Procedures and Characteristics

Table 14.9 shows the gold grade univariate statistics for 1 m composites and population characteristics for each of the mineralized zones. The corresponding sample distributions were plotted graphically as histograms, log histograms, and log probability charts. Figure 14.3 presents example charts for the dominant zones of Larjor, Latiff, Kinjor and Marvoe.

Table 14.9 Summary Statistics within Mineralized Zones

Description	Field	Number	Min.	Max.	Mean	Variance	CoV
Weathered Minzones	Au	359	0.01	23.04	2.22	12.6	1.3
Fresh Minzones	Au	4540	0.01	86.40	2.84	31.1	2.0
Minzone M201	Au	956	0.01	80.80	3.44	30.1	1.6
Minzone M301	Au	486	0.01	74.24	4.22	37.1	1.4
Minzone M401	Au	579	0.01	81.31	3.86	46.5	1.8
Minzone M402	Au	422	0.01	34.40	2.05	15.7	1.9
Minzone M501	Au	1609	0.01	48.00	1.54	11.9	2.2
Minzone M503	Au	297	0.01	53.76	5.15	64.1	1.6
Minzone M504	Au	63	0.01	86.40	5.84	233.7	2.6

The grade distributions for all the data subsets shown in Table 14.9 display some degree of multimodality, mostly bimodal; however the inflections between subpopulations are transitional and the clarity of the bimodality has generally reduced from 2010 with the addition of the 2011 drilling. Individual subpopulations generally tend towards log normality. The high-grade zone M503 displays the most marked bimodality, with a transitional inflection between 2.0 and 4.0 gt/ Au.

In all cases the spatial distributions of the subpopulations have been reviewed and the general conclusion is that the lower grade portions represent narrow low grade or anomalous intervals within the overall defined zones, and are not of a scale that would validly allow either partitioning from the interpreted mineralized zone or separate extraction during mining. The small-scale inter-fingering of high and low grade intervals probably reflects variations in both fluid pathways and suitable conditions for gold precipitation, although it is also possible that local thrusting could cause repetitions of individual intersections.

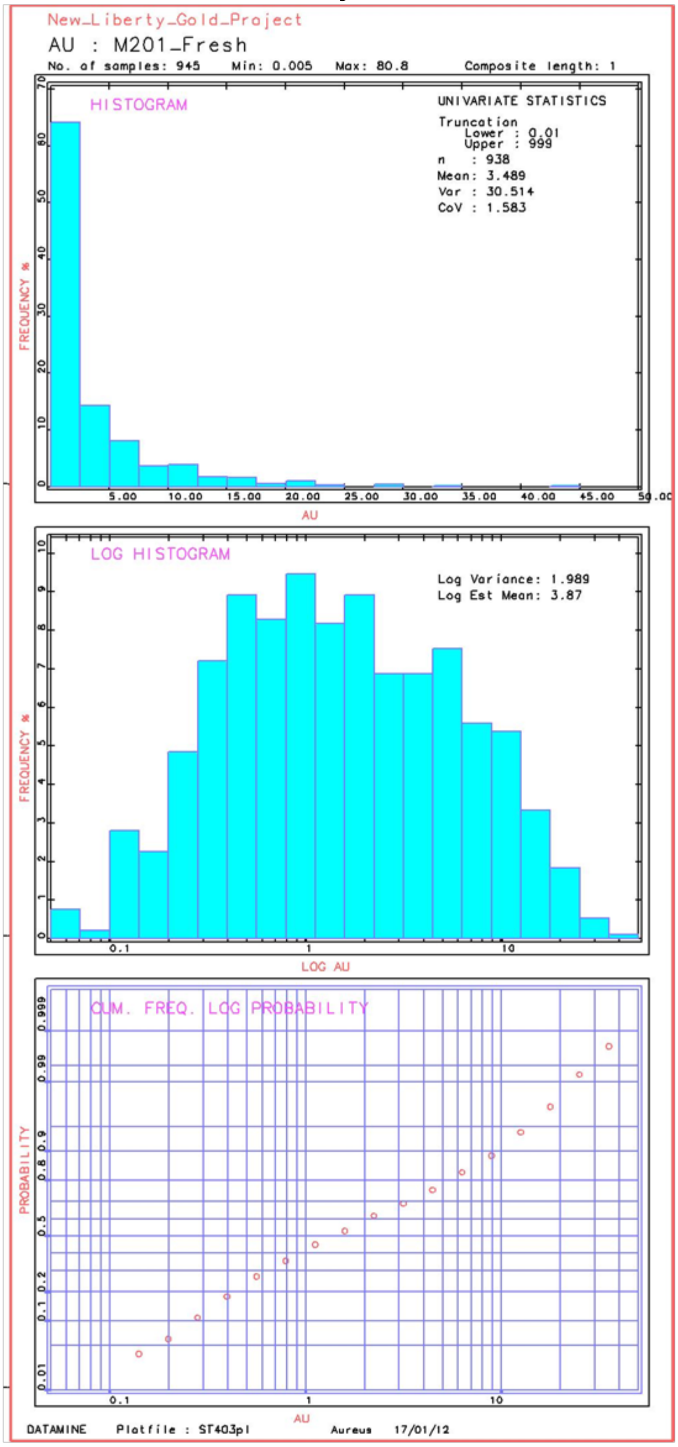
The mean gold grades for individual zones (excluding M504) are varied (1.54 g/t Au – 5.04 g/t Au). Nonetheless, the corresponding coefficients of variation (CoV) fall within a relatively narrow window (1.3 – 2.2), which may reflect a commonality of genesis for the different zones. The higher CoV of zone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

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

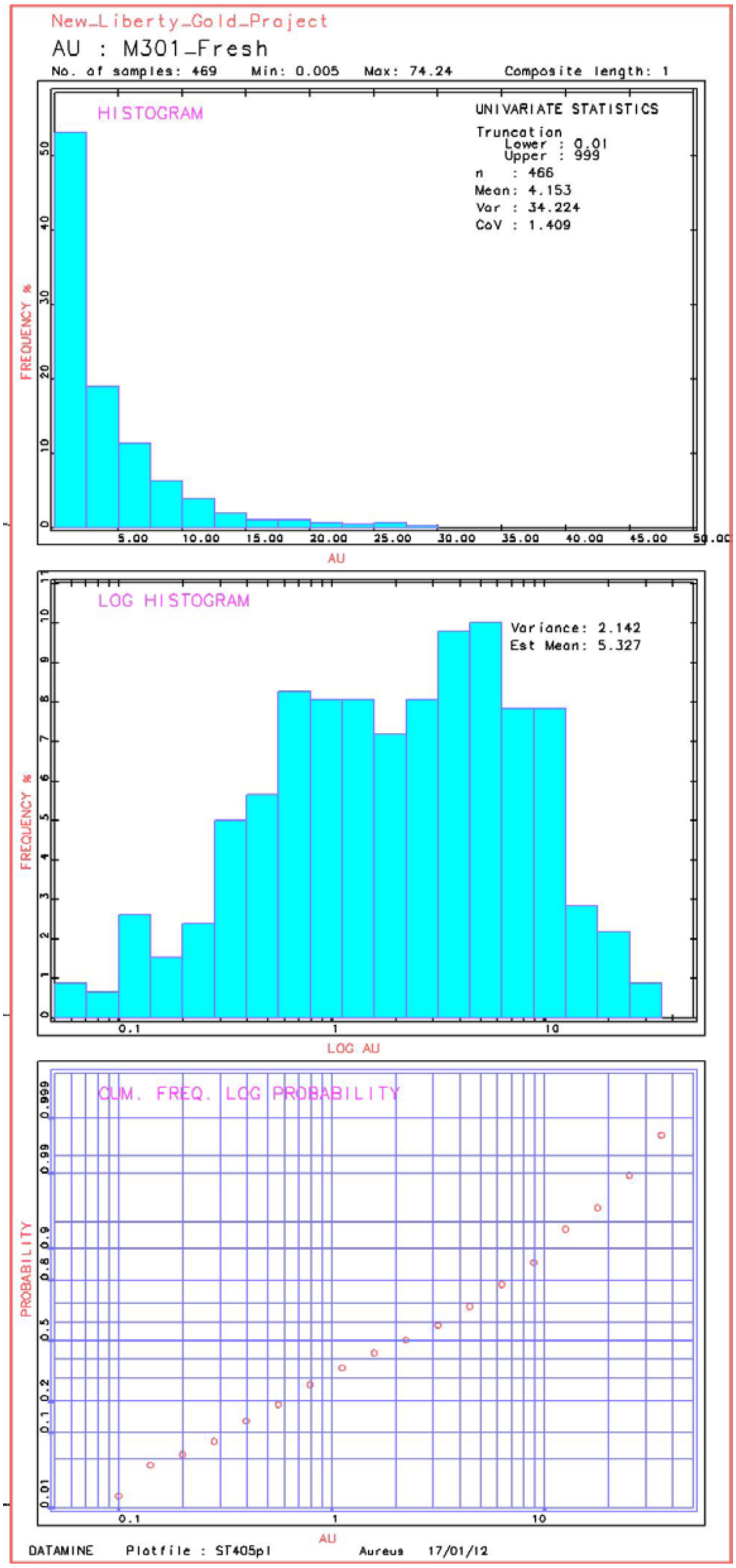
Overall, the grade distributions for the two (weathered and fresh) subsets exhibit similar characteristics; however the mean grade of the weathered population (2.22 g/t Au) is markedly lower than that of the near surface unweathered (2.57 g/t Au). Furthermore visualization of gold grades across the boundary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

Figure 14.3 Selected Statistical Charts

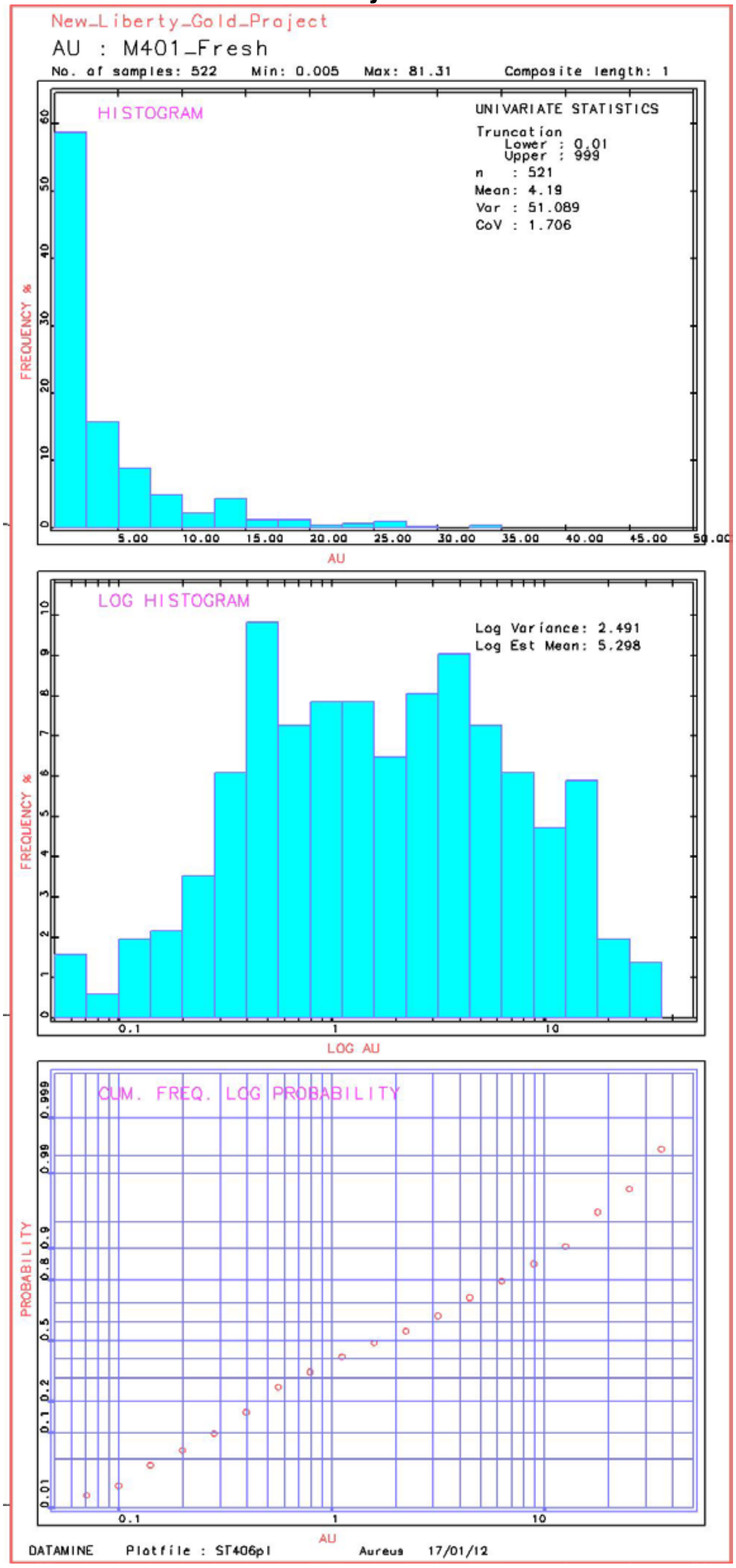
Larjor



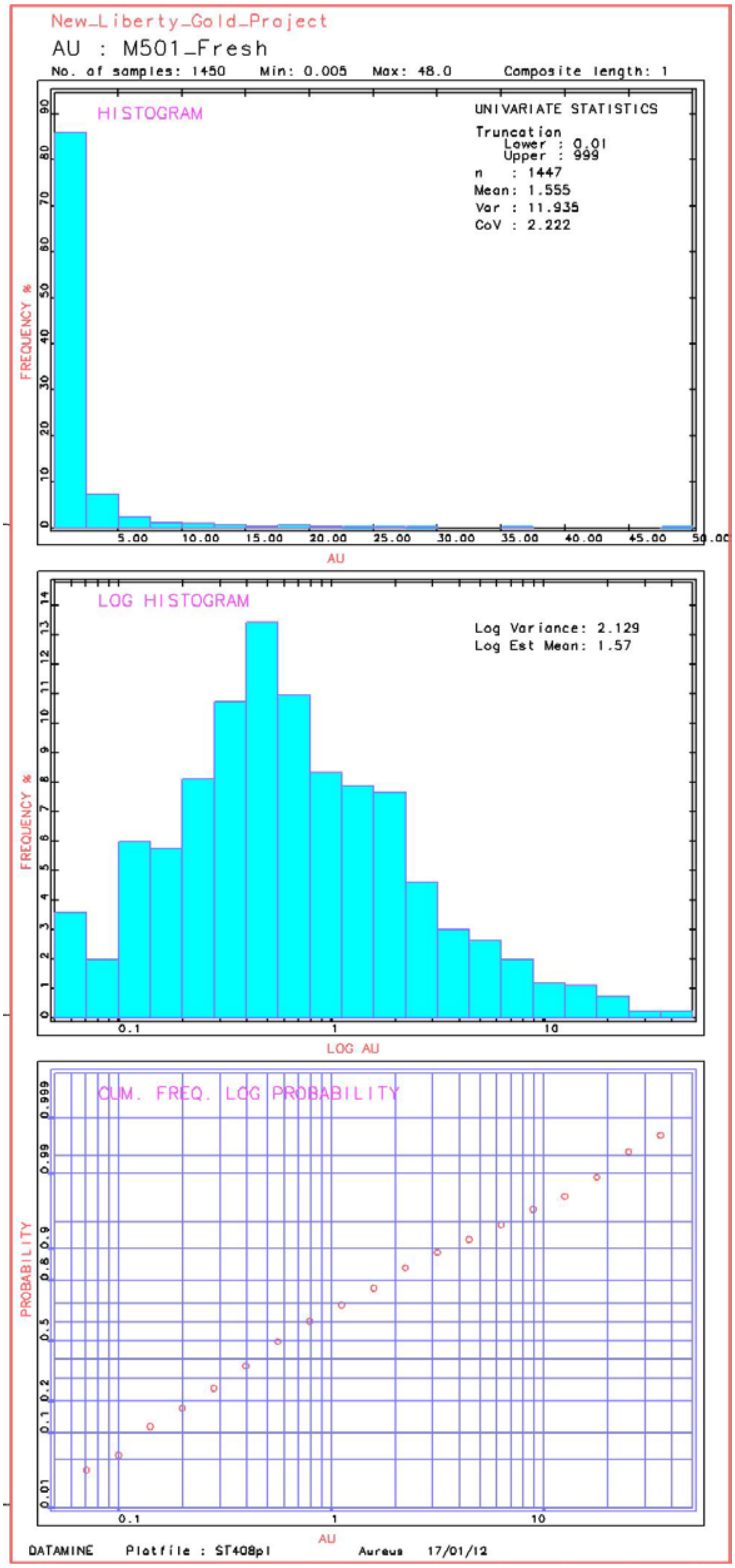
Latiff


New_Liberty_Gold_Project
AU : M301_Fresh
No. of samples: 469
Min: 0.005
Max: 74.24
Composite length: 1
HISTOGRAM
UNIVARIATE STATISTICS
Truncation
Lower : 0.01
Upper : 999
n : 466
Mean: 4.153
Var : 34.224
CoV : 1.409
FREQUENCY %
AU
LOG HISTOGRAM
Variance: 2.142
Est Mean: 5.327
LOG AU
CUM. FREQ. LOG PROBABILITY
PROBABILITY
AU
DATAMINE
Plotfile : ST405pl
Aureus
17/01/12

Kinjor


New_Liberty_Gold_Project
AU : M401_Fresh
No. of samples: 522 Min: 0.005 Max: 81.31 Composite length: 1
HISTOGRAM
UNIVARIATE STATISTICS
Truncation
Lower : 0.01
Upper : 999
n : 521
Mean: 4.19
Var : 51.089
CoV : 1.706
FREQUENCY %
AU
LOG HISTOGRAM
Log Variance: 2.491
Log Est Mean: 5.298
LOG AU
CUM. FREQ. LOG PROBABILITY
PROBABILITY
AU
DATAMINE Plotfile : ST406pl Aureus 17/01/12

Marvoe


New_Liberty_Gold_Project
AU : M501_Fresh
No. of samples: 1450 Min: 0.005 Max: 48.0 Composite length: 1
HISTOGRAM
UNIVARIATE STATISTICS
Truncation
Lower : 0.01
Upper : 999
n : 1447
Mean: 1.555
Var : 11.935
CoV : 2.222
FREQUENCY %
AU
LOG HISTOGRAM
Log Variance: 2.129
Log Est Mean: 1.57
LOG AU
CUM. FREQ. LOG PROBABILITY
PROBABILITY
DATAMINE Plotfile : ST408pl Aureus 17/01/12

14.7 Grade Capping Strategy

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

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

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

The list of selected high grade caps is shown in Table 14.10. The grades in zone M503 were not capped.

Table 14.10 Grade Capping

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

14.8 Variography

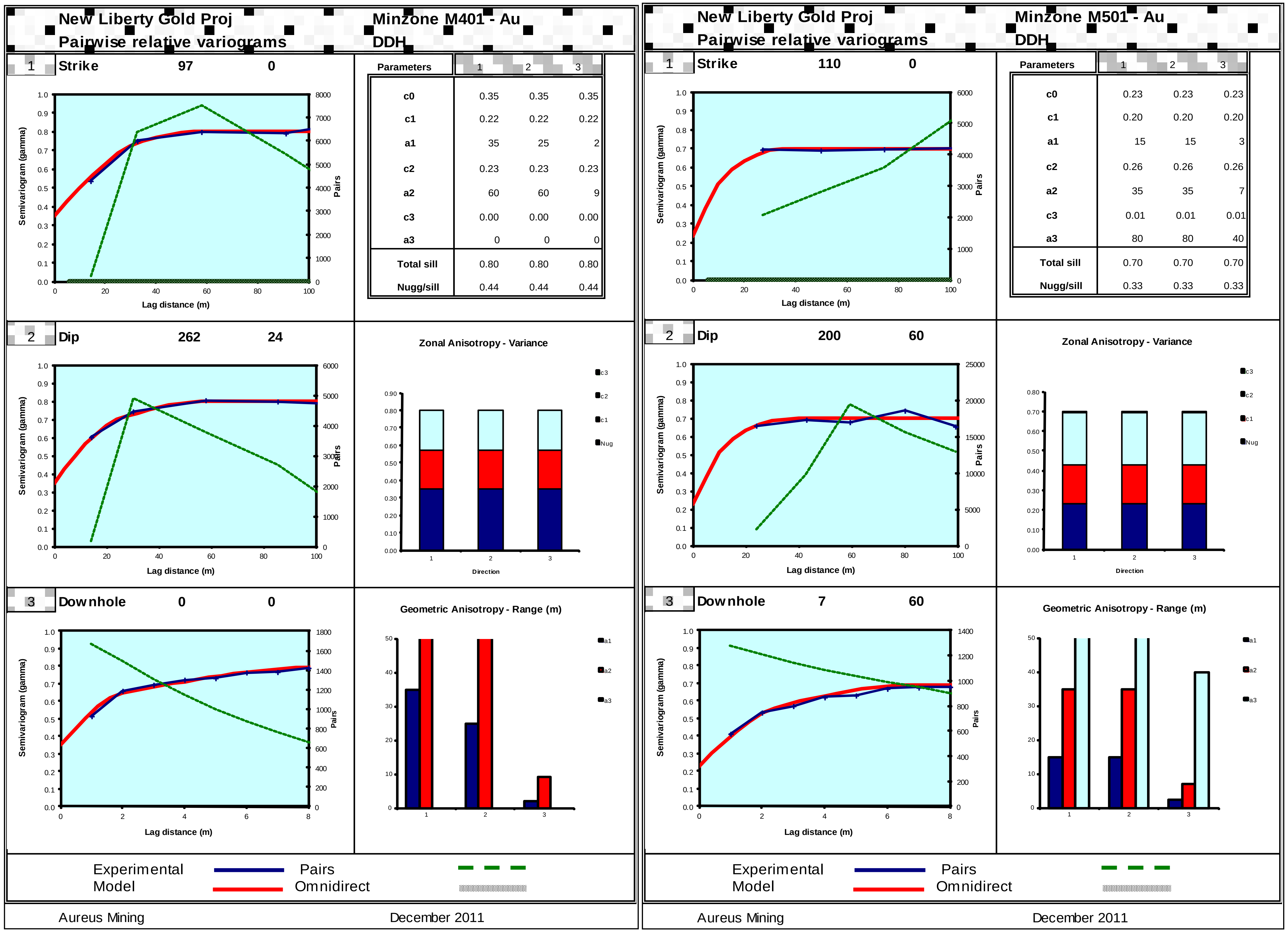
Variographic analysis was conducted for those mineralized zones with sufficient data points (i.e. excluding M504).

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

During variogram modelling (see example charts in Figure 14.4), the position of the nugget variance was fixed using the downhole variogram, and anisotropic variogram parameters were derived using two- or three-structure spherical models (Table 14.11).

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

Figure 14.4 Selected Variogram Charts



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

Table 14.11 Variogram Parameters

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

14.9 Estimation

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

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

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

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

Figure 14.5 shows a long. section view of the resource model, coloured on gold grades, while the four parts of Figure 14.6 present selected cross-sections through the resource model, one each for the main mineralized zones.

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

Figure 14.5 Schematic Resource Model Long. Section Showing Gold Grades

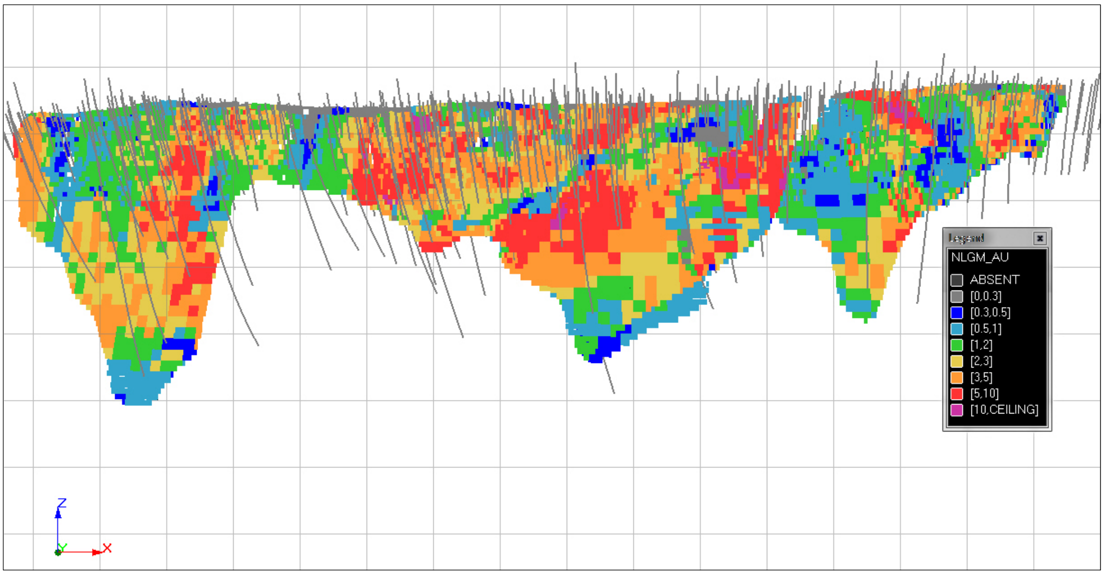
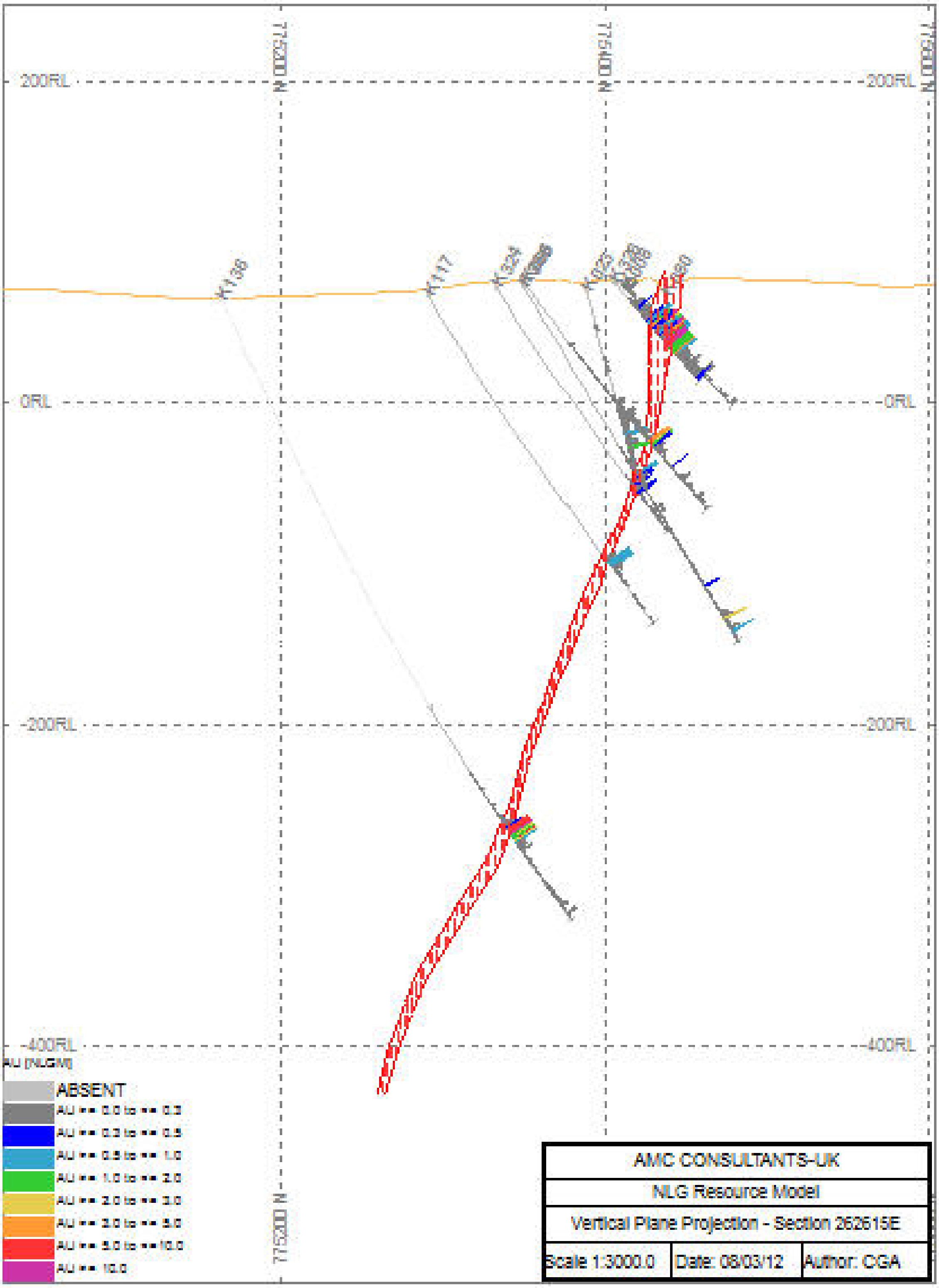
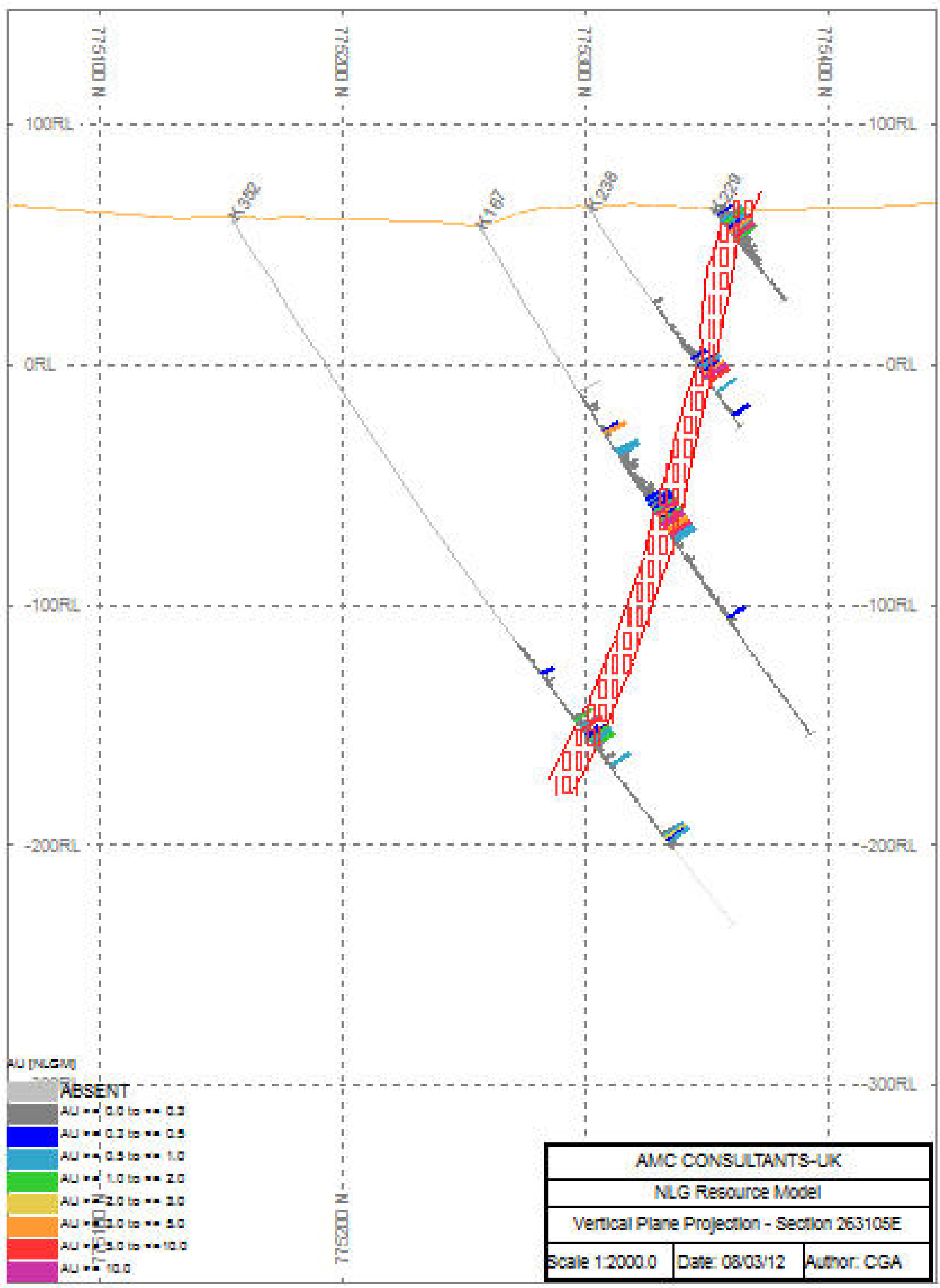


Figure 14.6 Model Cross-sections with Drillholes Overlay

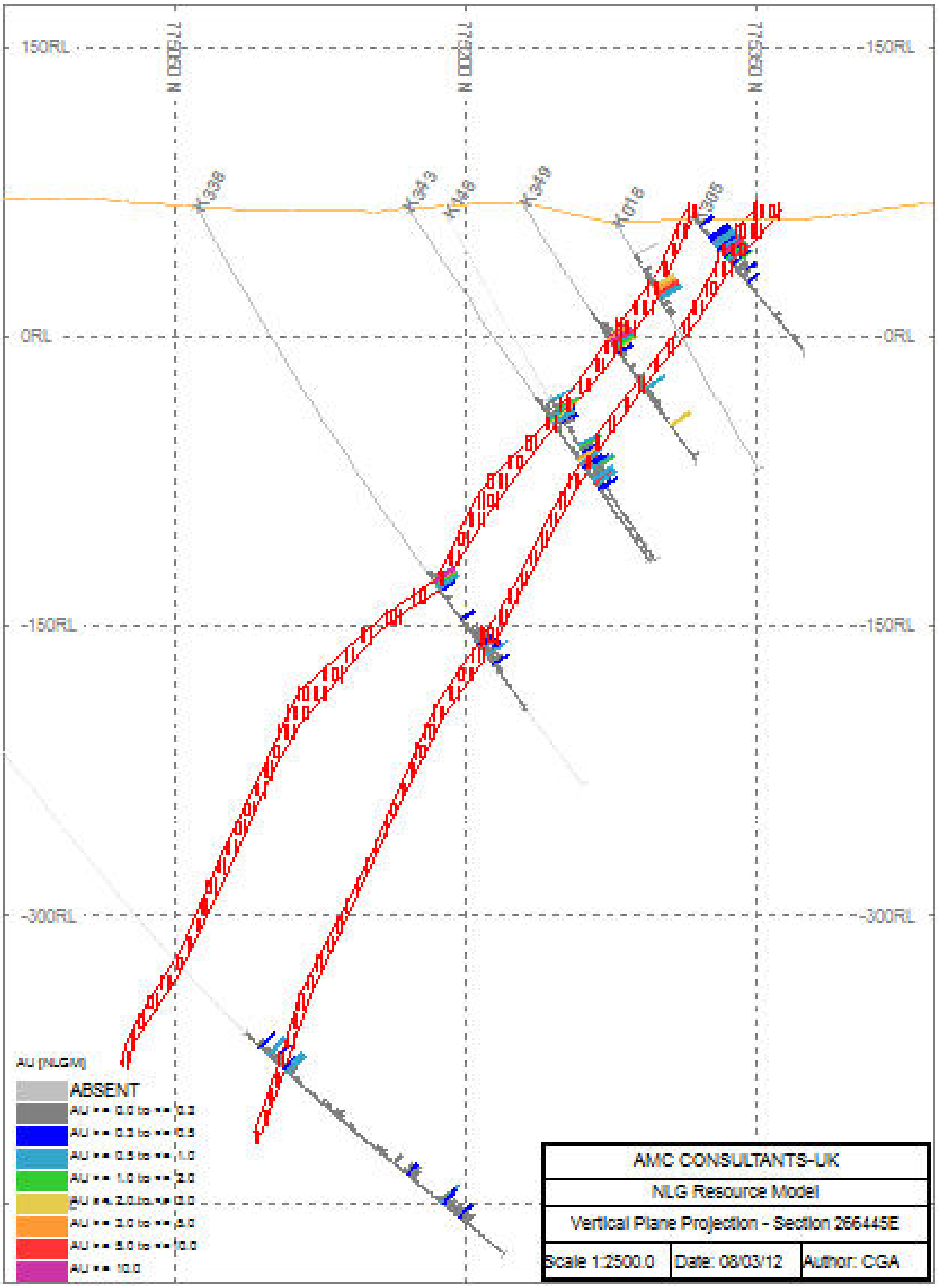
Larjor



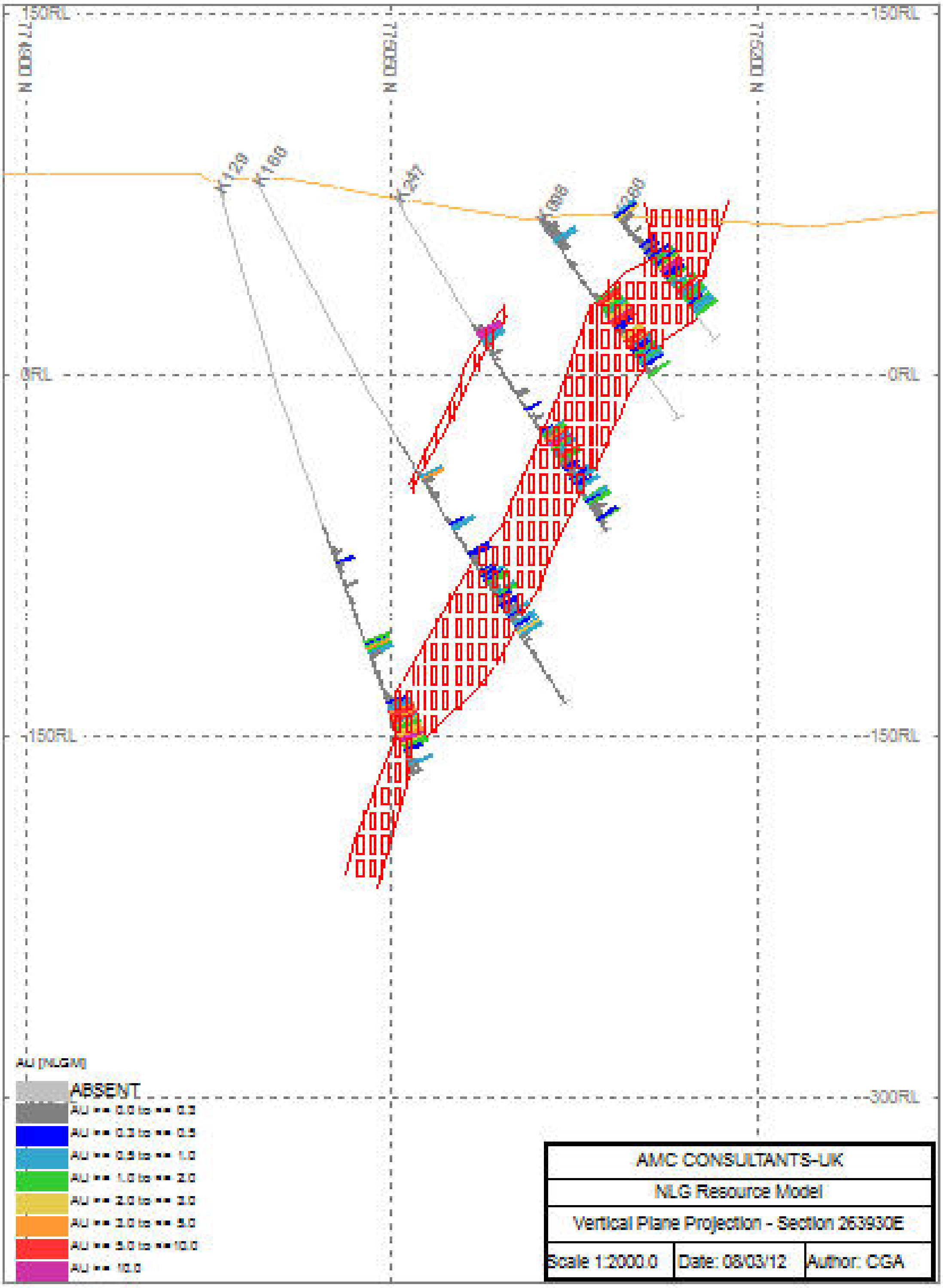
Latiff


775100 N
775200 N
775300 N
775400 N
100RL
0RL
-100RL
-200RL
-300RL
K392
K167
K238
AU [NLGM]
ABSENT
AU >= 0.0 to <= 0.3
AU >= 0.3 to <= 0.5
AU >= 0.5 to <= 1.0
AU >= 1.0 to <= 2.0
AU >= 2.0 to <= 3.0
AU >= 3.0 to <= 5.0
AU >= 5.0 to <= 10.0
AU >= 10.0
AMC CONSULTANTS-UK
NLG Resource Model
Vertical Plane Projection - Section 263105E
Scale 1:2000.0
Date: 08/03/12
Author: CGA

Kinjor


150RL
0RL
-150RL
-300RL
ABSENT
AMC CONSULTANTS-UK
NLG Resource Model
Vertical Plane Projection - Section 266445E
Scale 1:2500.0
Date: 08/03/12
Author: CGA

Marvoe


150RL
0RL
-150RL
-300RL
771400 N
771600 N
771800 N
K120
K160
K247
ABSENT
AMC CONSULTANTS-UK
NLG Resource Model
Vertical Plane Projection - Section 263930E
Scale 1:2000.0
Date: 08/03/12
Author: CGA

14.10 Resource Classification

Procedures for classifying the reported resources were undertaken within the context of NI 43-101.

- Estimated resources have been classified with consideration of the following criteria:
- Quality and reliability of raw data (sampling, assaying, surveying).
- Confidence in the geological interpretation.
- Number, spacing and orientation of intercepts through mineralized zones.
- Knowledge of grade continuities gained from observations and geostatistical analyses.
- The likelihood of material meeting economic mining constraints over a range of reasonable future scenarios, and expectations of relatively low selectivity of mining.

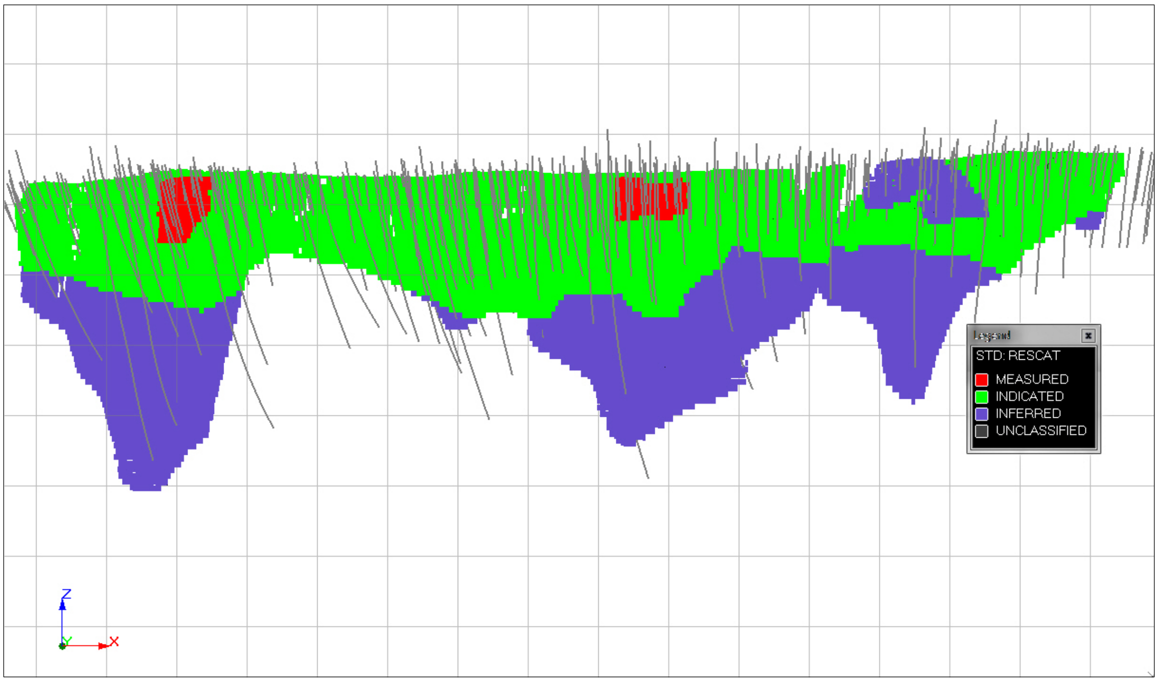
In general, the drill spacing is more closely spaced near surface, progressively reducing in density with depth as the drilling tracks each mineralized zone down dip. On drill spacing alone, therefore, the level of confidence in the resource declines with depth.

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

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

The process of coding the cell model by resource classification was facilitated by presenting the drill intersection data for individual mineralized zones in long. section and digitising outline strings to represent the limits of areas considered to constitute, as appropriate indicated or measured mineral resources. The model cells were by default assigned an inferred mineral resource classification (RESCAT field code 3), and those cells falling within the digitized strings were re-coded as indicated mineral resources (RESCAT field code 2) or measured mineral resources (RESCAT field code 1). The distributions of the different category materials are presented in Figure 14.7.

Figure 14.7 Resource Model 3D Schematic Showing Classification Areas



14.11 Model Validation

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

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

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

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

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

14.12 Tonnage-Grade Reporting

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

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

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

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

The measured, indicated and inferred mineral resources for the project at a 1.0 g/t Au cut-off are presented in Table 14.12. Figure 14.8 summarizes the grade and tonnage estimates at a series of cut-off grades. As higher cut-off grades are applied to the resource, the geometric and internal grade continuities of the mineralization will deteriorate, therefore the potential mineability of material as defined at higher cut-offs will be lower.

Table 14.12 Mineral Resources (as at 1 October 2012)

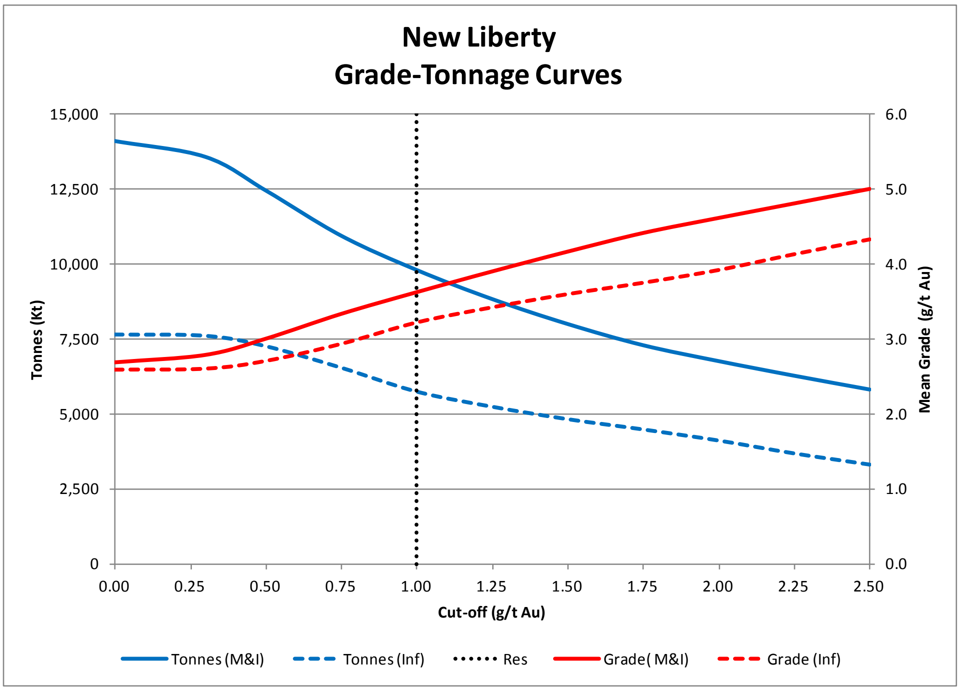
	Measured			Indicated			Measured and Indicated		
	Tonnes	Au		Tonnes	Au		Tonnes	Au	
Minzone	(Kt)	(g/t)	(Koz)	(Kt)	(g/t)	(Koz)	(Kt)	(g/t)	(Koz)
M401	651	4.77	100	5,468	3.88	683	6,118	3.98	783
M402				874	2.51	71	874	2.51	71
M501				2,317	2.43	181	2,317	2.43	181
M503				486	6.93	108	486	6.93	108
M504			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

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

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

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

Figure 14.8 Grade-Tonnage Profiles



15 MINERAL RESERVE ESTIMATES

15.1 Mining Approach

The following section summarizes the mining engineering design work carried out as part of the optimizations of the Project since the Feasibility Study reported in October 2012. This work is based on the same geological block model generated by AMC that formed the basis of the Feasibility Study, but includes additional geotechnical data and slope designs developed by AMC.

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

AMC prepared pit designs for the Project on the basis of pit optimizations carried out using the Mineral Resource estimate adjusted for dilution and ore loss, slope designs based upon the geotechnical assessments, and estimates of the contract mining costs, processing costs, site general and administration costs, metallurgical recovery.

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

15.1.1 Resource Block Model

The geological block model estimate presented in Section 14 of this report was used as the basic resource model for the pit optimization studies. The pit optimizations only considered the measured and indicated mineral resources. All inferred mineral resources were treated as waste for the purposes of mine planning. Table 15.1 details the mineral resource classifications that have been used in the optimization process.

Table 15.1 Summary of Mineral Resource (from Section 14)

RESOURCE CATEGORY	MINERALIZATION ZONE	Tonnes (kt)	Grade (g/t)	Au Ounces (kOz)
Measured	Larjor + Latiff + Kinjor main zone	651	4.8	100
	Kinjor footwall zone	-	-	-
	Marvoe main zone	-	-	-
	Marvoe western hanging wall zone	-	-	-
	Marvoe central hanging wall zone	-	-	-
Measured Total		651	4.8	100
Indicated	Larjor + Latiff + Kinjor main zone	5,468	3.9	683
	Kinjor footwall zone	874	2.5	71
	Marvoe main zone	2,317	2.4	181
	Marvoe western hanging wall zone	-	-	-
	Marvoe central hanging wall zone	486	7	108
Indicated Total		9,145	3.5	1,043
Total Measured + Indicated		9,796	3.6	1,143
Inferred	Larjor + Latiff + Kinjor main zone	3,059	3.2	313
	Kinjor footwall zone	127	3.6	15
	Marvoe main zone	1,122	2.6	93
	Marvoe western hanging wall zone	1,298	3.6	152
	Marvoe central hanging wall zone	122	5.1	20
Inferred Total		5,728	3.2	593

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 are inclusive of Mineral Reserves
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability

The mineral resource model was modified to reflect the dilution and mining loss expected for the mining method and the scale of mining machinery to be used.

15.1.2 Geotechnical and Hydrogeological Assessment

The geotechnical paramters from the Feasibility Study were refined to reflect:

- Drilling of 4 new targeted geotechnical drillholes.
- ATV logging of 19 drillholes, confirming the S2 foliation trend, and contributing to the improved definition of joint sets. Analysis of this data also allowed for the measurement of fault orientations.
- Photologging of core to identify areas of poor ground, and assess the spacing of breaks along the foliation. As a result of the investigation, Marvoe was identified as having the poorest ground conditions, a factor which was considered in the wall and slope designs.
- This new data was sufficient for AMC to develop a 3D structural model in collaboration with Aureus personnel. Geotechnical domains were redefined, based

on four lithological domains (weathered material, and three fresh rock domains), and four structurally distinctive areas. In addition, a further domain was developed in the ultramafic, based on the alteration (high magnetite ultramafic and low magnetite ultramafic).

- In order to integrate the new data into the the revised geotechnical model, a review of the Aureus logging data was completed:
- Following a report from Orefind (2013) that identified some cases where the orientation reference line had been poorly transferred onto the core, AMC filtered the Aureus database by removing all suspected poor quality data. The results of the data filtering process, confirmed the absence of S1 foliation fabric that was used during the FS study.
- Joint sets were assessed using the new database, indicating seven joint sets with a maximum of three joint sets in any particular domain.
- Following the identification of errors and inaccuracies in the database of RQD provided by Aureus to AMC, a new RQD database was developed. Comparisons were drawn between lithologies and pit areas, showing consistency across the pit areas, with a greater amount of poor ground in the low magnetite ultramafic, and also the granitioids.
- The variation in discontinuity condition ratings was analysed across the geotechnical domains, and a revised discontinuity strength was developed for each domain. Subequent numerical analysis considered the variation in strength across the geotechnical domains.
- Finally, rockmass rating parameters were analysed throughout the different pit areas, and the lithological domains.

Bench scale kinematic and deterministic analysis showed:

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

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

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

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

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

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

Water management at New Liberty is thought to be mainly driven by surface water components. The project is located within a wet climatic zone, with rainfall averaging approximately 3,300 mm/year. The high seasonal rainfall and intense storm events, which generally occur over the course of the rainy season (June – August), are key features for the design of an appropriate dewatering strategy.

Groundwater inflows to the pits are likely to be relatively minor when compared to the rainfall run-off generated inflows. However, the dewatering system will need to consider both elements as groundwater inflows will continue even during extended dry periods and may be locally significant in areas of enhanced permeability. The ongoing hydrogeological studies will attempt to quantify these inflows. The requirements for adequate drainage system for groundwater and rainfall have been incorporated into both the pit design and mining plan.

15.2 Open-Pit Optimizations

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

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

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

15.2.1 Dilution and Ore Loss

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

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

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

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

The effect of dilution and ore loss varies locally within the deposit as wider zones are less affected than narrow areas of mineralization. The effect on each mineralized zone is summarized in Table 15.2.

Table 15.2 Dilution and Ore Loss Effect

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

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

Overall Pit Slope Parameters

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

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

A summary of the mining bench parameters developed based on kinematic analysis is given in Table 15.3

Table 15.3 Pit Slope Design Domains

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

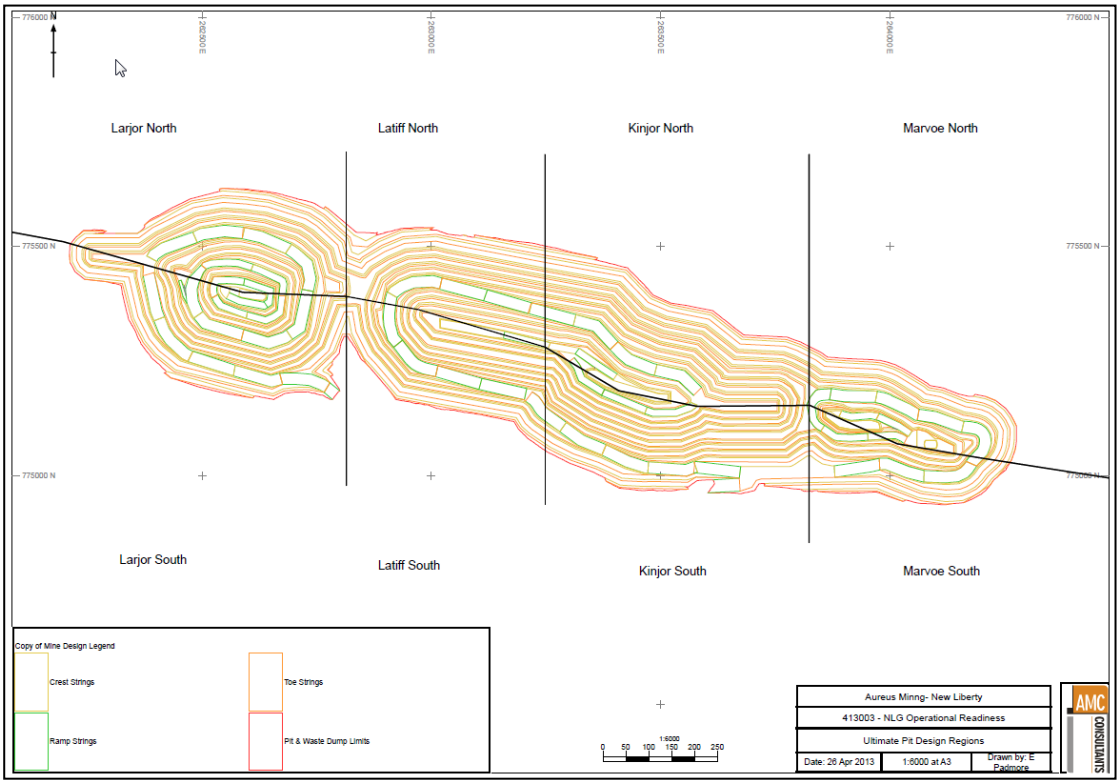
Numerical analysis of the overall slopes (Section 2.5.2) shows that 48° is an acceptable value for the upper limit of the overall slope angle.

The detailed design of the slopes were refined through the pit design process to reflect the slope design domains split to eight (8) different slope regions and to reflect the height of the slope and the integration of the access ramps required. The final design slope regions are summarized in Table 15.4 split according to the zones shown in Figure 15.1.

Table 15.4 Final Pit Design Slope Design Parameters

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

Figure 15.1 Design Sectors for the Pit Design



15.2.2 Mining Costs

The mine design, waste dump designs and mining schedule from the Definitive Feasibility Study and some preliminary haulage studies for a revised waste dump design were provided to Aureus' preferred mining contractor. Based upon this information, a revised set of mining rates was calculated by the contractor. Using these new mining contract rates the average cost of mining is $2.52/t mined.

15.2.3 Processing and General and Administration Costs

DRA Engineering provided the metallurgical processing costs and recovery factors per tonne of ore, and Aureus provided the overall general and administration (G&A) costs based on a nominal throughput of 1.1 Mtpa of fresh ore. The recovery and cost factors were also assumed to apply to the weathered near surface ore which contributes 2% of the Project ounces. A breakdown of the costs used in the optimization analysis is shown in Table 15.5.

Table 15.5 Processing, General and Administration Costs for Pit Optimization

Cost Centre	Cost/milled tonne US$
Processing	22.57
Administration	6.25
Total	28.82

15.2.4 Gold Price

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

15.3 Optimization Results

15.3.1 Cut-off Grade

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

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

- A gold price of US$ 1,300/ounce (US$41.80/gramme);
- A processing and administration cost of US$28.82 per tonne of ore treated; and
- A metallurgical recovery of 93%.

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

The calculation used is:

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

$$= \frac{222.57+6.25}{41.80*93\%} = 0.8 \text{ g/t Au}$$

15.3.2 Optimization Results

The key parameters applied in the pit optimizations were:

- A gold price of US$1,300/ounce
- A processing and administration cost of US$28.82 per tonne of ore treated
- An average mining cost of US$2.52/tonne
- A metallurgical recovery of 93%
- A minimum mining width for ore zones or waste partings of 2.5 m
- A 0.5 m dilution skin applied to the ore zones
- Overall pit slope angles of 35º in the weathered zones, and slope angles varying from 42º up to 51º in the hard rock zones of the footwall and hanging wall, respectively.

The results of the open pit optimization are shown in Figure 15.2 and Table 15.6.

Figure 15.2 Pit Optimization Results

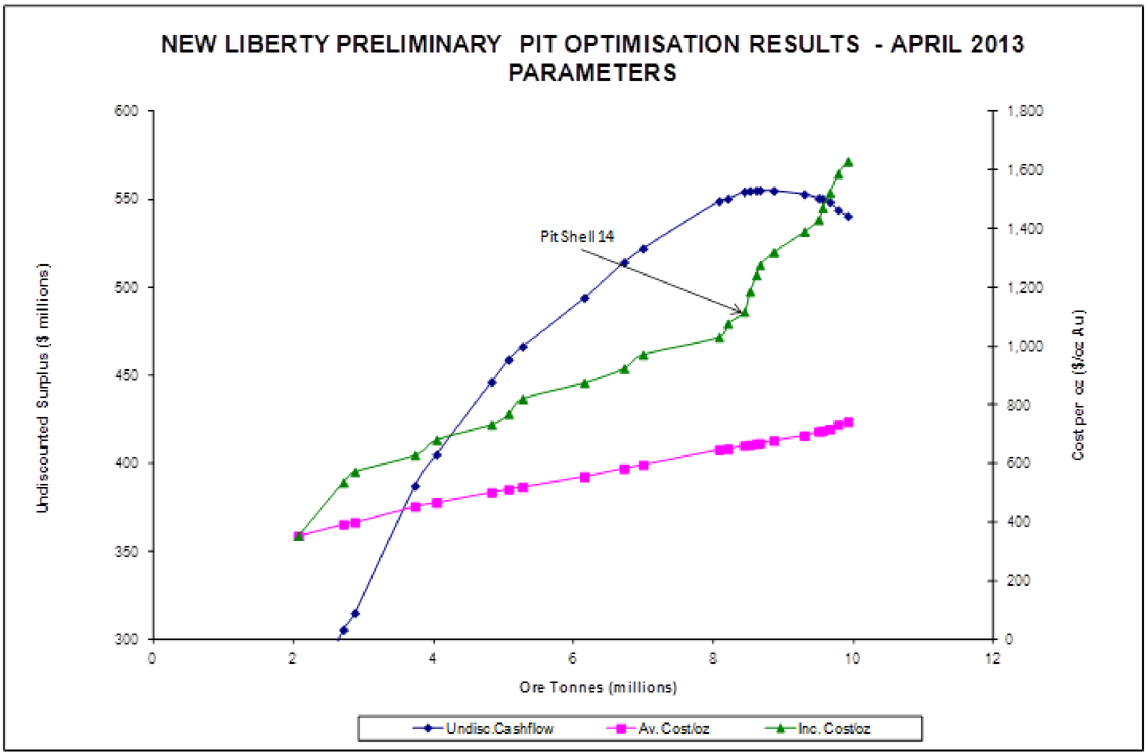


Table 15.6 Whittle Pit Optimization Results

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

15.3.3 Selection of Optimum Pit Shell

The graph in Figure 15.2 shows the 24 optimization shells that are generated using incrementally increasing revenue factors and hence the gold price used to generate the optimization shell. All optimization shells were then evaluated using the base costs and a gold price of US$1,300/oz. The optimization pit tonnage and revenue estimate details are shown in Table 15.6.

The results of undiscounted and discounted cash flow analysis are used to select the optimum pit for design purposes. After analysis and discussion with Aureus

Pit shell 14 (revenue factor = 0.9) was chosen as the optimal pit for design purposes as the incremental production cost rises steepely after this pit shell.

Pit shell 14 produces:

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

15.4 Pit Design

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

The final pit design is shown in Figure 15.3.

Figure 15.3 Final Pit Design

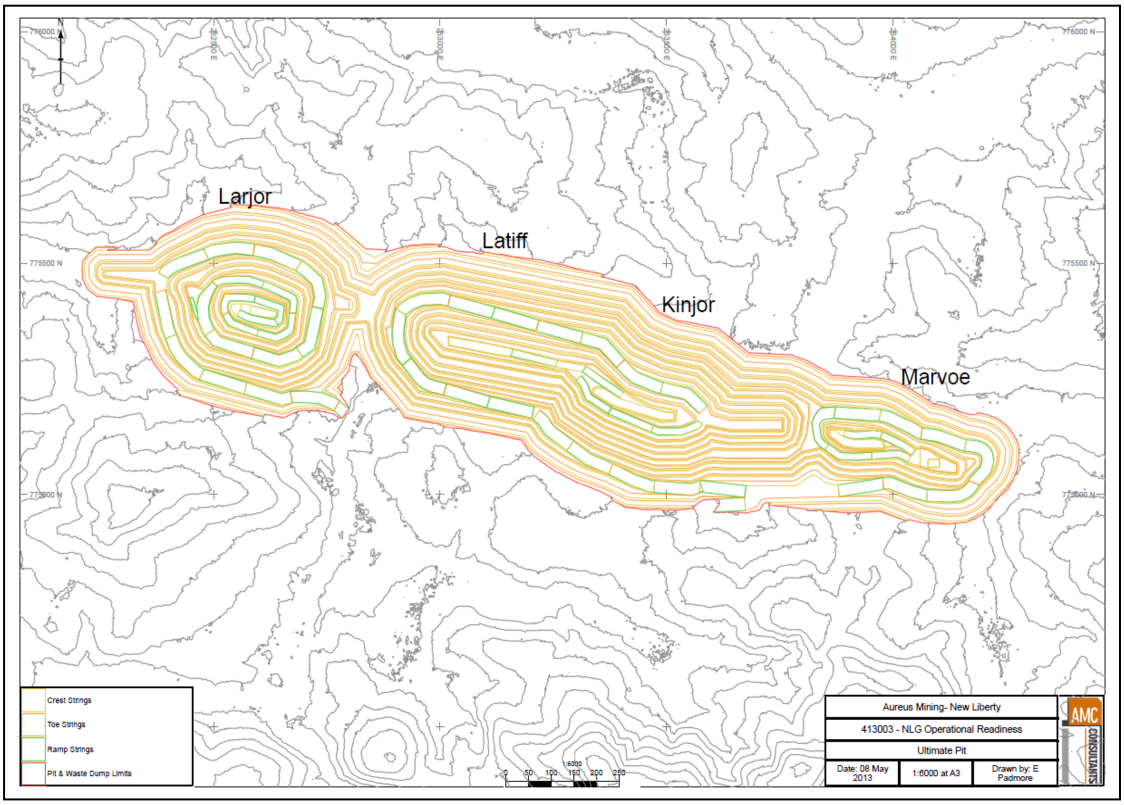


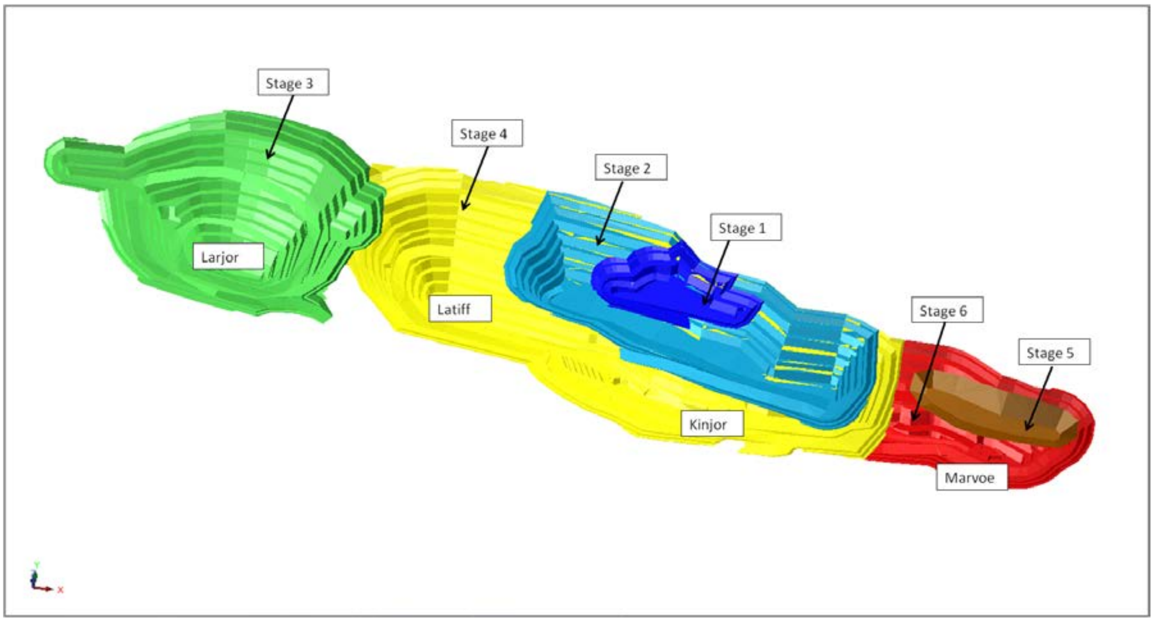
Table 15.7 shows the achieved design in comparison with the optimization pit shell 14.

Table 15.7 Optimized Pit and Designed Pit Comparison

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

In the initial stages of the pit operations it is planned to mine an interim starter pit in the Latiff-Kinjor area. This will be followed in the mining sequence by the Larjor pit and a cutback to the Latiff-Kinjor pit. There will be a second cutback at Latiff-Kinjor to take the pit to its final limits. Marvoe has a small starter pit to access some low stripping ratio ore earlier in Year 4, thereafter it deepens to its final limits. Figure 15.4 shows the main staging elements of the pit design.

Figure 15.4 Staging of Pit Design



15.5 Mineral Reserves

The pit design was evaluated using the diluted block model to estimate the mineral reserve of the Project. The mineral reserve estimate is summarized in Table 15.8.

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

Table 15.8 Mineral Reserve Estimate (as at 20 May 2013)

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

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

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

The project sensitivities suggest that this mineral reserve estimate is relatively robust.

16 MINING METHODS

16.1 Introduction

Four West-African experienced mining contractors were requested to submit mining proposals on preliminary designs of the open pit. Aureus have worked closely with their preferred contractor to develop the mining costs for the project. The mining contractor will be selected following negotiations and a contract adjudication/award process. The contractor will supply all of the capital mining fleet requirements, including pumps for pit dewatering, and be responsible for the site preparation, haul road construction, drill-and-blast, load and haulage of ore and waste material, oversize breakage, fleet equipment maintenance and pumping operations. Aureus will undertake and manage all mine planning technical aspects, including geology, grade control, mine planning, drill-and-blast planning, operational scheduling and contractor performance management.

The mining operations are based on conventional drill-and-blast, load and haul mining techniques. Loading of the ore and waste rock following drill-and-blast is envisaged to be by hydraulic excavators in backhoe configuration into nominal 100 tonne haul trucks with 50% of the ore reporting directly to the crusher tip and 50% to the ROM ore pad, and waste to the designated waste dump areas. It is has been assumed that the weathered portion of the ore and waste will be free dig or require light blasting as it transitions to fresh rock which will require blasting. Pit access roads will be designed and installed as required and will be constructed to a nominal width of 25 m inclusive of safety berms and water drainage controls. A ROM pad stockpile area will be constructed adjacent to the process plant crusher station/tipping point and all ore stockpiled will be sorted under a stockpile grade control management scheme. Stockpile reclaim will be undertaken by a front end loader and fed to the crusher as required.

All waste material will report either to the surface waste dumps which wraps around the open pit or be backfilled in the mined-out pit areas, depending on which facility is closest and available at the time of mining, and to suit the efficient scheduling of the mining fleet.

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

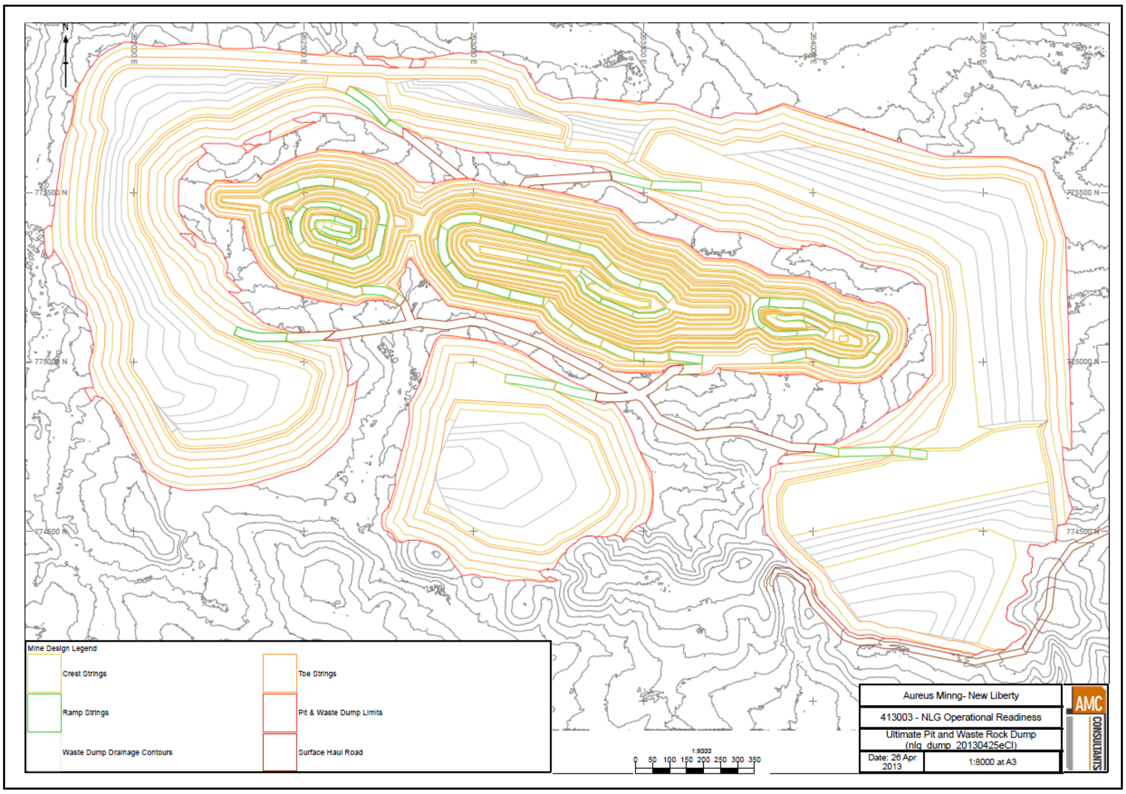
Aureus selected a preferred West-African experienced mining contractor and this contractor revised their budget estimate to reflect the feasibility study mine designs and schedules. This contractor has relevant experience and reflected detailed discussions of the project and preliminary designs and schedules in their estimates. The costs provided were based on a fleet of new mining equipment.

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

16.2 Waste Dump Design

The WRD was redesigned and the new designs are shown in Figure 16.1.

Figure 16.1 Pit and Waste Dump Design



The waste dumps were designed to have a capacity of 52 million cubic metres. The design now has two separate dumps. The gap between the dumps to the south of Larjor follows the existing course of the Marvoe Creek. This gap is for the accommodation of sedimentation ponds. The gap in the dump south of Marvoe is to provide haulroad access to the ROM.

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

The mining plan proposes backfilling the Larjor Pit with mined waste. It is estimated that approximately 10M BCM of broken waste material will be backfilled into the Larjor pit.

16.3 Mine Production Schedule

A pre-strip period of 6 months with relatively low material movement rates was scheduled to establish the pits and to build an ore stockpile ahead of plant commissioning. The schedule steps up an average material movement rate of 70,000 tpd from the end of the second quarter of Year 1 (Period 15).

In this schedule, processing commences at the start of Year 1, treating the small initial ore stockpile and pit ore production. Open-pit material movement achieves a steady state annual production of 1.1 Mt ore and 25.3 Mt total movement by the end of Year 1. This steady state production continues until halfway through Year 5. The total movement declines after Year 5 as the strip ratio declines from around 25:1 to around 10:1 towards the end of the mine-life.

The Larjor pit is completed at the end of Year 4, and waste is backfilled into the pit from this period until the end of the operation in Year 8.

Table 16.1 summarizes the annual production from the open pit and the treatment schedule.

The mining schedule reflects a review of a number of scheduling options with the focus on maximizing the grade mined to be treated in the early life of the mine whilst deferring mining costs. The phasing of mining the Larjor, Latiff-Kinjor and lastly the Marvoe areas achieves this aim.

Table 16.1 also details the mill treatment and gold production schedule which assumes a nominal treatment rate for ore of 1.1 million tonnes per annum. The mining and treatment schedules are also shown in Figure 16.2 to Figure 16.4.

Table 16.1 Mining and Treatment Schedule

		Total	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Ore	Mt	8.5	0.3	0.7	1.1	1.1	1.2	1.3	1.1	1.3	0.3
Grade	g/t	3.4	3.8	2.7	3.5	3.2	4.0	3.8	3.3	3.1	2.4
Contained Gold	000 oz	924	40.1	64.4	126.3	116.3	154.0	154.4	114.4	126.6	27.2
Waste	Mt	131	2.1	22.1	24.4	24.5	24.4	19.2	9.4	5.1	0.3
Total Material	Mt	140	2.4	22.9	25.6	25.6	25.6	20.5	10.5	6.3	0.6
Strip Ratio	t w:o	15.5	6.4	29.9	21.5	21.6	20.3	15.3	8.7	4.0	0.8
Ore Milled	Mt	8.5	-	1.0	1.1	1.1	1.1	1.1	1.1	1.1	0.8
Grade	g/t	3.4	-	3.1	3.6	3.2	4.0	4.0	3.5	3.3	2.0
Contained Gold	000 oz	924	-	101.1	127.0	115.2	142.1	142.2	126.3	118.2	51.6
Recovery	%	93%		93%	93%	93%	93%	93%	93%	93%	93%
Gold Produced	000 oz	859	-	94.0	118.2	107.1	132.1	132.2	117.5	110.0	48.0

The mining schedule has been generated on a monthly basis for all production periods. The schedule has been based on a practical sequence of mining of ore and waste blocks. All mineralized waste below the cut-off grade 0.8 g/t and above 0.65 g/t has been considered to be mineralized waste. The mineralized waste has been identified in the mining schedule as a separate stockpile to allow for possible future treatment of such material as financial conditions dictate.

Figure 16.2 Mined Tonnes and Grade

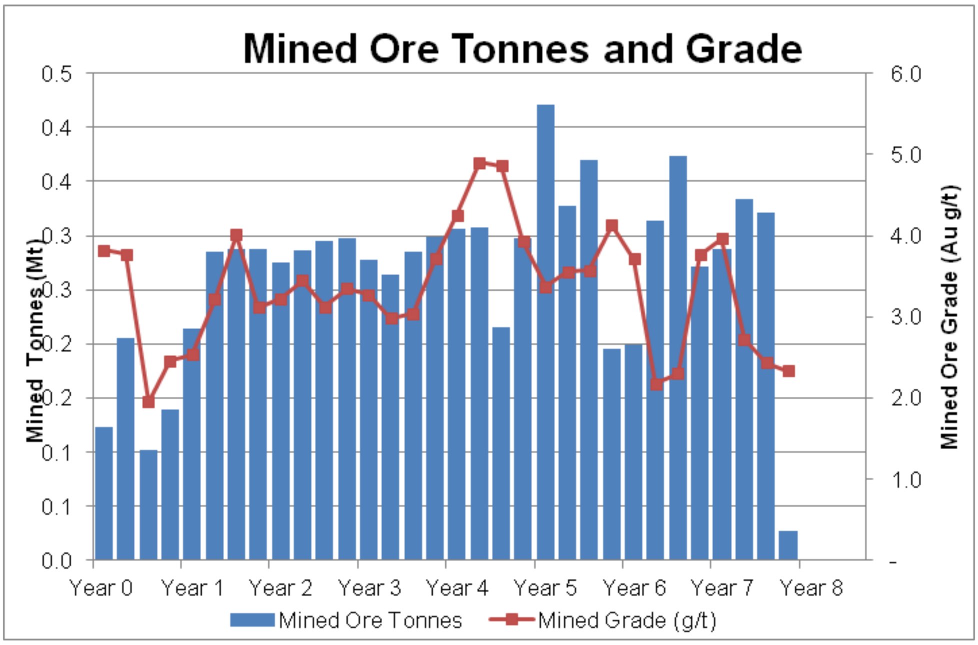


Figure 16.3 Process Plant Tonnes and Head Grade

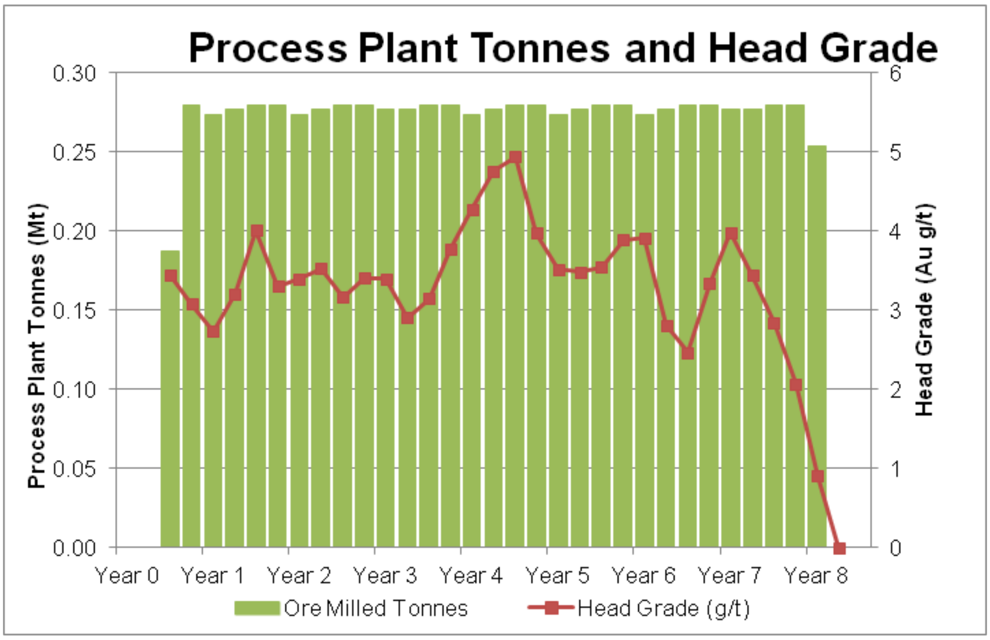
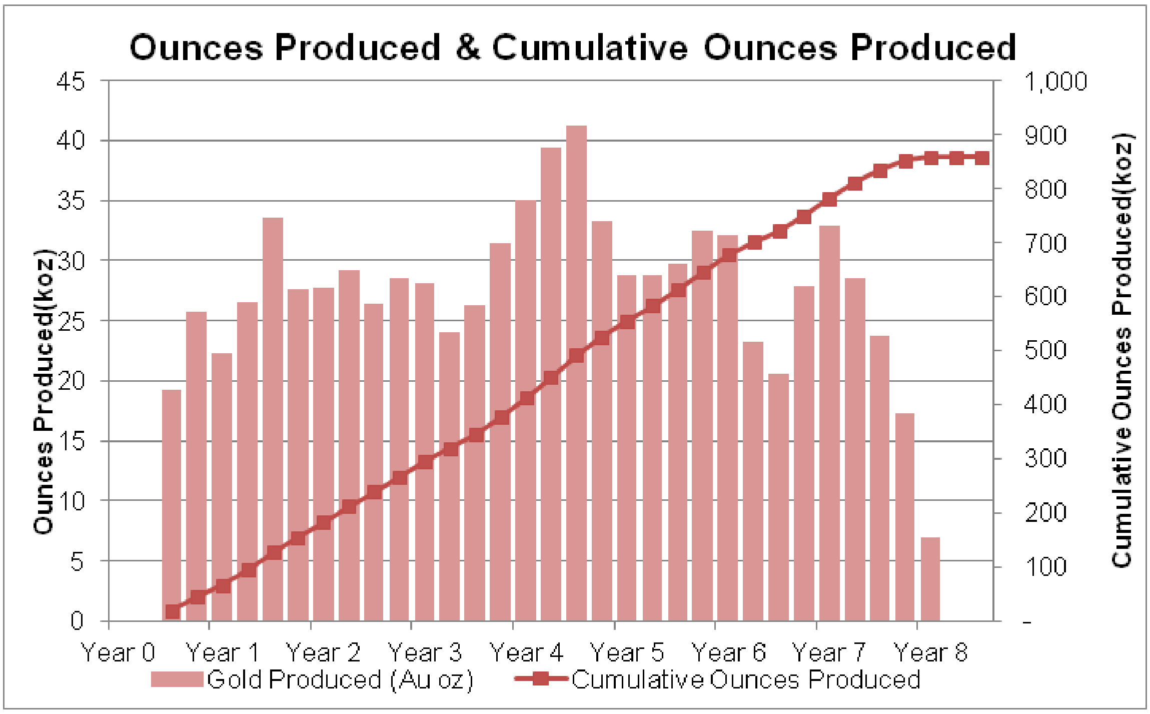


Figure 16.4 Ounces Produced and Cumulative Ounces Produced



16.4 Mining Equipment Requirements

The mining fleet equipment reflects the scale of the operation at peak mining rate from the production schedule – this is summarized in Table 16.2. There is a planned phased build up in mined tonnage to reflect both commissioning requirements of the process plant and the mobilization and training requirements for the mining contractor.

Table 16.2 Mining Fleet

Equipment	No of Pieces
Excavator 12 m^3 bucket	2
Excavator 6 m^3 bucket	1
Excavator 2 m^3 bucket	1
Haul trucks 100 t	18
Dozers	4
Wheel dozer	1
Graders 16 foot	2
ROM pad loader	2
Water carts	2
Fuel and lube trucks	4
Drills	7
Light vehicles	29
Buses	6
Pumps, lighting towers, tyre handler etc.	Various

It is anticipated that during the initial phase of the mining production the contractor will employ previously trained operatives as one-on-one operator/trainers from within the Economic Community of West African States (ECOWAS) region, who will provide training on a rotational basis, and then be replaced by local Liberian operatives.

16.5 Mine Work Schedule

Table 16.3 summarizes the mine work schedule planned for the Project. The mine is scheduled to operate 338 days per year which includes the time for inclement weather. Holidays will be worked on an overtime basis and form part of the two shift rotation system. Ore will be preferentially mined on day shift to allow greater supervisions on ore mining. The mine will operate two 12 hour shifts per day which allows 338 available shifts per year for ore and 676 shifts for waste. In order to achieve this, a three crew system will be adopted for all operating personnel, one shift day-shift, one shift night-shift and a roster off-shift. The operating time per shift will be the actual time during the shift that the equipment is ‘’productively’’ working at its rated capacity and is equal to the total scheduled time less all scheduled and unscheduled delays.

Table 16.3 Scheduled Working Periods

Parameter	Units	Value
Calendar Days	Days	365
Days per week	Days	7
Holidays	Days	0*
Weather	Days	27
Scheduled days	Days	338
Shifts/day ore	Shifts	1
Shifts/day waste	Shifts	2
Annual work shifts ore	Shifts	338
Annual work shifts waste	Shifts	676
Hours per shift - Ore	Hours	12
Hours per shift - Waste	Hours	12
Scheduled hours Ore	Hours	4056
Scheduled hours Waste	Hours	8112

*FULCO system i.e. pay overtime for holidays

16.6 Open-Pit Dewatering

The open pit dewatering strategy is based on a study by RPS Aquaterra in June 2013, and ongoing work. The total average monthly inflows have been estimated for the life of the mine and shows significant seasonal fluctuation, primarily due to fluctuations in rainfall runoff. The inflows for the entire pit typically range from approximately 25 l/s in the drier months, derived primarily from groundwater, to over 100 l/s in the wet months where the water is derived primarily from rainfall runoff. The pit pumps will be sized to handle double this amount of water.

The water management plan for the open pit will have two focus areas, firstly managing surface runoff water and secondly pumping groundwater from sumps inside the pit.

The WRD will be used to minimize surface water runoff to the pits. Drains and berms will be also constructed to surround the open pit, where possible, in order to divert drainage and minimize pumping requirements. The water from these will be collected and diverted to a Walled Sedimentation Basin (WSB) which will allow sediment to settle before the water is released via a wetlands area to the Marvoe Creek. A series of dewatering boreholes situated in the eastern and western limits of the pit may also be drilled where the main orebody shear zone can be intersected and dewatered. The water from these holes will be either pumped to the WSB or process plant, depending on water quality and water usage requirements. The details of this pit dewatering requirement will be determined through the ongoing hydrogeological investigations.

The second aspect of dewatering relates to rain/stormwater events which will be managed by dewatering pumps capable of handling the expected maximum ingress as described earlier. Diesel driven pumps will be used to pump to a sump inside each pit from where the water will be pumped out to the WSB by means of a pump station equipped with electric pumps. Successive pumping stations will be established as the pit extends deeper to ensure adequate pumping capacity at all times. The mining contractor will be responsible for the operation and maintenance of the entire pumping system.

16.7 Mining Manpower

The Aureus mining personnel will provide a managerial and technical services function to the operation. These include mine management, survey, geology, drill-and-blast planning and grade control as well as mine planning. The rest of the mining operational labour will be provided for by the mining contractor.

16.8 Mine Infrastructure

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

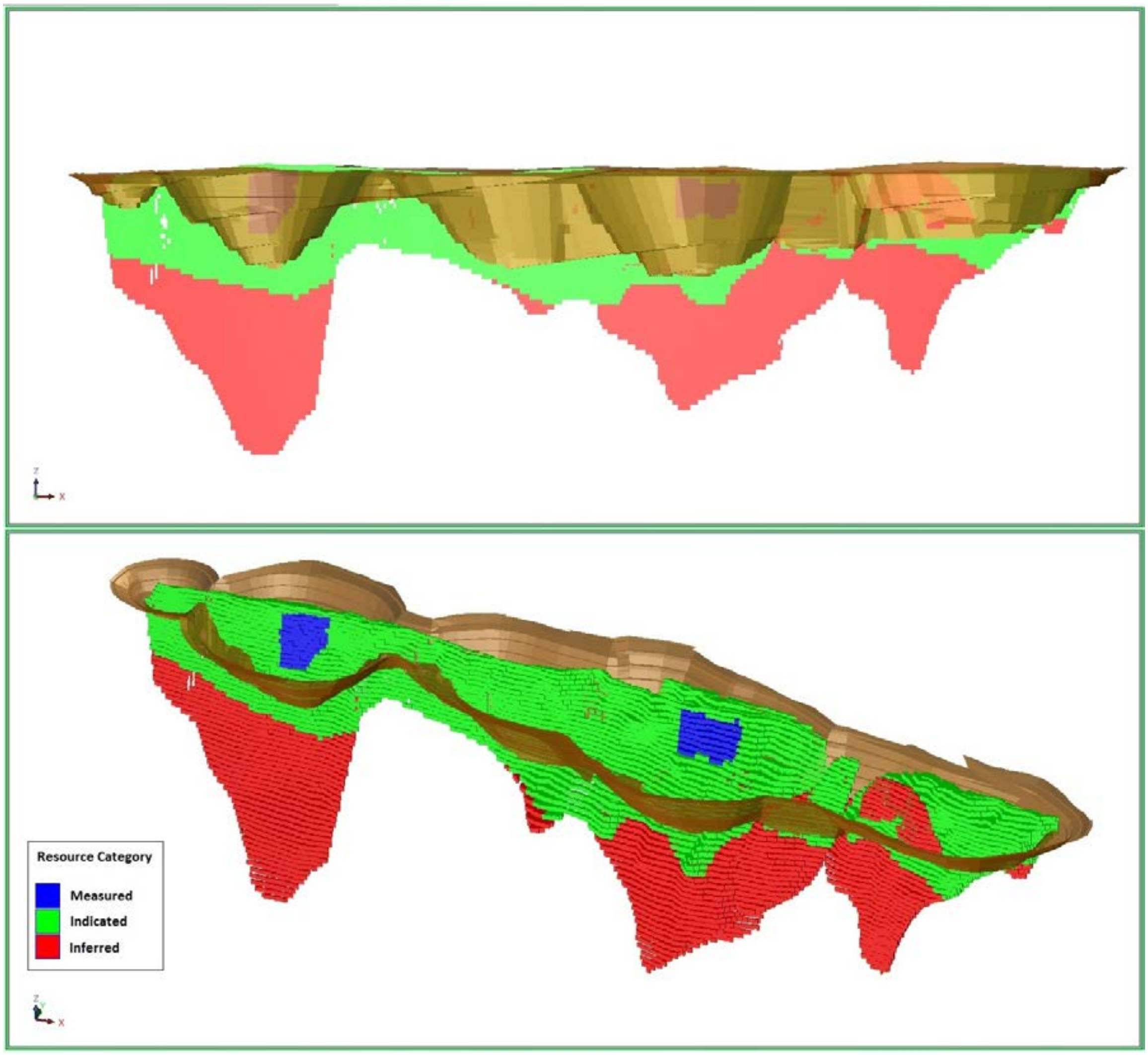
A layout of in-pit and surface access roads will be developed. These roads allow access between the pits, process plant, mine laydown area and workshops, explosives magazine, ROM stockpile and waste dumps area covering all the work activities associated with the mine operations. These roads will be constructed by the mining and civil works contractors.

16.9 Deeper Resources – Underground Potential

Substantial resources exist below the current designed pit bottom elevations used in the pit optimization models. Drilling has defined resources to a depth of 500 m below surface. Aureus is extending its exploration drilling programme to further explore the lateral and depth extents of the currently defined mineralized zones. Future studies will assess the viability of potential deepening of the open pits or potential underground mining operations during the operational phase of the mine.

The pit design is shown against the resource modelled in Figure 16.5.

Figure 16.5 Pits with Resource Model Showing Resource Classification



17 RECOVERY METHODS

17.1 Plant Design Criteria

17.1.1 Introduction

Metallurgical test work results and industry design principles, where necessary, were used to define the process design criteria for the New Liberty Gold Project. The process plant is designed to treat 1.1 million tonnes per annum.

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

Several references have been used to derive data used in the process design criteria which are shown in Table 17.1. These are the following:

- Aureus Mining;
- Metallurgical test work;
- Calculated data;
- Vendor data or recommendations;
- DRA standards or practices;
- Handbook (engineering handbook); and
- External consultants.

Table 17.1 Process Plant Design Criteria

Base Data	Units	Value
Plant feed rate	dmt/annum	1 100 000
Plant feed rate	dmt/month	91 667
Maximum lump size	mm	700
Moisture content	% w/w	10.0
Ore true density	tonne/m³	2.8
Ore bulk density	tonne/m³	1.88
ROM head grade	ppm Au	3.10 - 4.00
Overall Recovery Target	%	92.0% – 93.2%

Operating Times	Units	Value
ROM delivery	days/annum	350.0
ROM delivery (design)	hours/day	10.0
ROM delivery utilization	%	76.5
ROM delivery (design)	hours/annum	2,678.0
Crushing	days/annum	350.0
Crushing	hours/day	18.0
Crushing utilization	%	76.5
Crushing	hours/annum	4 820.0
Concentrator	days/annum	350.0
Concentrator	hours/day	24.0
Concentrator utilization	%	90.0
Concentrator	hours/annum	7,560.0

Feed Particle Size Differentiation		Value (mm)
Primary crusher feed PSD	P_{100}	700
(typical design values)	P_{80}	433
(typical design values)	P_{50}	209
(typical design values)	P_{25}	70

17.2 Ore Characteristics

Gold mineralization at New Liberty occurs in zones of variable thickness and is nearly continuous along 1.8 km of strike length. The Project deposits consist of high-grade gold mineralization. Ore characteristics are shown in Table 17.2.

Table 17.2 Ore Characteristics

Base Data	Units	Value
Ore Source	type	Open pit
Maximum lump size (F100)	mm	700
Maximum lump size (F80)	mm	433
Maximum lump size (F50)	mm	209
Maximum lump size (F20)	mm	70
ROM Characteristics	**Units**	**Value**
Moisture content	%	10.0
Ore SG	t/m³	2.80
Bulk density of crushed ore	t/m³	1.88
Raw Water Analysis	**Units**	**Value**
Source	-	Marvoe Creek
Magnesium	ppm	<0.5
Sulphate	ppm	<1
Chloride	ppm	0.900
Calcium	ppm	1.00
Total cyanide	ppm	<0.01
WAD cyanide	ppm	<0.01
Conductivity	mS/m	1.70
pH		5-5.2
TDS	mg/l	9-12

17.3 Operating Schedule

Table 17.3 summarizes the data used in compiling the operational schedule of the processing plant.

Table 17.3 Operating Schedule

Operating Schedule	Units	Value	
General			
Annual tonnage treated	Mtpa	1.10	1.10
Ore Processing tonnes per month	t/month	91 667	91 667
ROM Delivery			
Days per annum	days	350	
Hours per day	h	18.0	10.0
Overall utilization	%	76.5%	
Operating hours per annum	h/a	4 820	2 678
Crushing Circuit			
Days per annum	days	350	
Hours per day	h	18.0	
Overall utilization	%	76.5%	
Operating hours per annum	h/a	4 820	
Crushing circuit feedrate	t/h	228	228
Screening throughput	t/h	492	492
Screen O/Size % of new feed	%	115%	115%
Crushing throughput	t/h	263	263
Milling and Concentrating			
Days per annum	days	350	
Hours per day	h	24.0	
Overall utilization	%	90.0%	
Operating hours per annum	h/a	7 560	
Milling throughput	t/h	145.5	145.5

17.4 Plant Recovery

The overall plant recoveries are shown in Table 17.4.

Table 17.4 Plant Recovery

Recovery	Units	Value	
Head grade	g/t	3.10	4.00
Head gold content	g/h	451	582
Head gold content (maximum)	g/day	10 825	13 968
Gravity Concentration			
Estimated Gravity circuit feed grade	g/t	2.48	2.96
Gravity circuit feed gold	g/h	372	444
Gravity recovery (% Au of concentration unit feed)*	%	58.2%	62.9%
Gravity recovery from test work (% Au of head grade)*	%	48.0%	48.0%
Gravity concentrate gold	g/h	238	305
Dissolution reactor recovery (% Au of unit feed)	%	95.0%	95.0%
Dissolution reactor gold recovery	g/h	226	290
Overall recovery (% Au of head grade)*	%	45.6%	45.6%
CIL			
CIL feed grade	g/t	1.69	2.18
CIL feed gold	g/h	245	317
Solution tailings	ppm	0.005	0.005
Solid tailings	g/t	0.22	0.25
Estimate for Gold in CIL tailings (solids+solution+carbon fines)	g/h	33.9	37.0
High Shear Scale-up Recovery Discount	%	0.50%	
Leach rate constant for modelling	t/h.g	0.800	
Absorption rate constant for modelling	h-1	0.020	0.020
Equilibrium constant used for modelling	g/t	10 000	10 000
Freundlich exponent used for modelling	-	0.700	0.700
Total Overall Recovery (% Au of head grade)	**%**	**92.0% – 93.1%**	

*Denotes non- discounted recovery estimates

17.5 Process Plant Design

17.5.1 Ore Receipt and Crushing

ROM opencast ore received from trucks will be treated in a primary crushing circuit comprising of a ROM bin (100-BN-001) fitted with a static grizzly, apron feeder (100-FD-004) and primary jaw crusher (100-CR-010) operated in open circuit. A dust suppression system will be installed.

The primary crusher product and apron feeder dribblings will gravitate onto the jaw crusher conveyor (120-CV-014).

Secondary crusher product is combined with primary crusher product on the jaw crusher product conveyor (120-CV-014) which feeds the crushing circuit sizing screen (120-SR-016). The primary crusher product conveyor (120-CV-014) is fitted with a weightometer

(120-ZM-040), which will be positioned before the introduction of secondary crusher product.

Circuit screen oversize is weighed (120-ZM-603) and conveyed (120-CV-019) to the secondary crushing circuit comprising of a bin (120-BN-023), a vibrating feeder (120-FD-024) and a secondary cone crusher (120-CR-026), in closed circuit to produce a crushed product stream with a P100 of 22 mm. It should be noted that only one crusher will be installed but space provision allows for the installation of a second cone crusher. A dust suppression system will be installed. Circuit screen undersize is conveyed to a 3500 t mill feed stockpile (200-SP-050) via the stockpile feed conveyor (120-CV-034).

The key parameters of the crushing circuit are summarized in Table 17.5.

Table 17.5 Ore Receipt and Crushing

Primary Crushing	Units	Value	
General			
Crusher work index (CWi) used for crusher sizing	kWh/t	20.5	
Uniaxial compressive strength (UCS)	MPa	TBC	
Abrasion index (Ai)		TBC	
Primary Crushing			
Method of feeding ROM bin	type	Truck	
Truck type	type	TBC	
Truck load size	t	90t-100t	
Primary jaw crusher	type	C125	
Number of primary crushers	#	1.00	
Crusher CSS	mm	100	100
Primary crusher product size (P_{100})	mm	192	192
Primary crusher product size (P_{80})	mm	128	128
Primary crusher product size (P_{35})	mm	60.0	60.0
Primary Crusher Feeder			
Type	type	Apron feeder	
Drive type	type	VSD	
Capacity	t/h	228	
Primary Crusher Discharge Conveyor			
Dry capacity	t/h	228	
Maximum lump size (F100)	mm	192	
Moisture content	%	10.0%	
Nominal wet capacity	t/h	254	
Bulk density	t/m³	1.88	

Secondary Crushing	Units	Value	
Secondary crusher feed			
Circuit screen type	type	2.4 m wide x 6.1 m long, double deck, decline, dry	
Screen aperture (top)	mm	55x55 SQ	
Screen aperture (bottom)	mm	22x55 SWT	
Circuit screen federate	t/h	492	
Screening oversize % of new feed	%	115%	
Secondary Crusher Feed Bin			
Type	type	Bin	
Total bin volume	m³	25.0	
Selected bin capacity	min	5.00	
Emergency overflow	Yes/No	No	
Withdrawal method from bin	Type	Vibratory pan feeder	
Secondary Crusher Feeder			
Drive type	type	VSD	
Capacity	t/h	263	263
Secondary Crushing			
Secondary crusher type	type	HP500 Std, medium liners	
No. of Secondary crushers required	#	1.00	
Selected secondary crusher CSS	mm	24.0	
Crusher feed at crusher loading	t/h	263	
Final crushing product (P100)	mm	22.0	
Final crushing product (P80)	mm	18.0	
Secondary Crusher Discharge Conveyor			
Dry capacity	t/h	492	
Maximum lump size (F100)	mm	192	
Moisture content	%	10.0%	
Wet capacity	t/h	546	
Bulk density	t/m³	1.88	

17.5.2 Milling

A ball milling and regrind vertimill circuit will treat crushed ore at a design feedrate of 146 t/h dry solids producing a final product with a nominal P80of 45 micron and P60 of 25 micron.

The mill will be fed from the mill feed silo using belt feeders (200-FD-053/054) and into the milling circuit by a conveyor (200-CV-057) fitted with a weightometer (200-ZM-091). Milk of lime is added to the mill feed to control the CIL-circuit operating pH.

A 17.5ft x 22ft EGL ball mill (200-ML-059) will be operated in closed circuit to produce a target grind of 80% passing 75 µm. The mill feed will consist of fresh crushed ore, a portion of the classification cyclone underflow, gravity concentration classification screen oversize and a recycle stream from the gravity concentration circuit tailings. The mill density is controlled at ±72-75% solids by mass, by the addition of process water to the mill inlet. A trommel screen (200-SR-070) is installed on the mill discharge to remove scats from the mill discharge slurry. A scats bunker will collect the scats and a mechanical reclaim will dispose of scats.

Mill product will report to the mill discharge sump (200-TK-072) where it is diluted to 50%-55% solids (under normal conditions) before being pumped to the classification cyclone cluster (200-CL-075) operated at 60-100 kPa so as to achieve an overflow product of 80% passing 75 µm at an estimated 38-40% solids by mass.

Cyclone underflow will be routed to the gravity concentrator feed at a rate of 140 t/h dry solids, with any excess re-cycled directly to the mill feed noting that in the normal operation that this will be zero. .

The cyclone overflow is gravity fed to the secondary cyclone feed sump (200-TK-640) where it is joined by the product from the regrind mill and diluted to 50% solids before being pumped to the classification cyclone cluster (200-CL-076) operated at 110 kPa so as to achieve an overflow product of 80% passing 45 µm and 60% passing 25 micron at an estimated 29% solids by mass.

The cyclone underflow will be fed to the Vertimill. Provision has been made to allow the Vertimill to be bypassed. The vertimill product will join the primary classifier overflow for combined secondary classification.

Secondary classifier cyclone overflow is gravity fed to the pre-leach thickener (260-TH-120) via a linear trash screen (200-SR-078) and a primary cross cut and secondary vezin sampling system (200-SA-124/125). Trash screen oversize is dewatered in a basket (200-SR-081) situated in the mill sump and drains directly to the mill-area spillage pump.

Spillage in the milling circuit bund will be collected and pumped to the mill discharge sump.

The key parameters of the milling circuit are shown in Table 17.6.

Table 17.6 Milling

Milling and Classification	Units	Value	
Stockpile Feed Conveyor			
Dry Capacity	t/h	228	
Maximum Lump Size (F100)	mm	22.0	
Moisture Content	%	10.0%	
Wet Capacity	t/h	254	
Bulk Density	t/m³	1.88	
Mill Feed Stockpile			
Quantity	#	1.00	
Silo Live Capacity	hrs	24	
Silo Live Capacity	t	3 500	
Mill Feeders (stockpile Reclaim)			
Quantity of Feeders	#	2.00	
Drive Type	type	VSD	
Capacity	t/h	145.5	
Mill Feed Conveyor			
Dry Capacity	t/h	145.5	
Maximum Lump Size (F100)	mm	22.0	
Moisture Content	%	10.0%	
Wet Capacity	t/h	162	
Bulk Density	t/m³	1.88	
Milling Circuit Selection			
Circuit Type	type	Closed Circuit Ball Milling	
Primary Milling			
Ball Mill Work Index Range (BWi)	kWh/t	14.0-22.0	
Rod Mill Work Index Range (RWi)	kWh/t	14.7-18.3	
JK Tech Drop Weight Ranges	A	51.7 – 89.1	
	b	0.27-0.53	
	A x b	21.3-30.5	
	ta	0.21-0.59	
Quantity of Mills Installed	#	1.00	
Wet Or Dry Milling	type	Wet	
Open Or Closed Circuit	type	Closed	
Overflow Or Grate Discharge	type	Grate Discharge	
Feed Size (F_{80})	mm	18.0	18.0
Mill Discharge Product Size Prior to Classification (P_{80})	µm	212	212
Discharge Slurry Density	%Sw/w	70%-75%	
Scats Removal	type	Bunker	
Mill Discharge Screen	type	Trommel	
Aperture Size	mm	TBC	
Selected Mill Size			
Diameter (Inside Shell)	ft	17.5	
Effective Grinding Length (EGL)	ft	22.0	
Mill Speed as % of Critical Speed	%Cs	75.0%	
Drive Type	type	Single Pinion and Reversible Girth Gear	
Design Mill Ball Load (Process)	%	25.0%	
Ball Material (High Cr, Cast Or Forged)	type	Forged	
Mill Liner Material (Steel, Rubber, Polymet)	type	Polymet	
Mill Ball SG	t/m³	4.80	
Selected Ball Size	mm	90 mm	
Mill Power (absorbed)	kW	2 951	

Milling and Classification	**Units**	**Value**	
Motor Power (installed)	kW	3 500	
Mill Discharge Sump			
Cyclone Feed Density	%S w/w	50.0%	55.0%
Design Residence Time	min	2.00	
Cyclone Feed Grade	g/t	2.22	2.65
Classification			
Type of Classification	type	Hydrocyclone Cluster	
Quantity of Clusters	#	1.00	
Cyclone Operating Pressure	kPa	85	
Circulating Load (Vertimill Operating)	%	96.0%	
Circulating Load Vertimill (By-Passed)	%	200% - 250%	
Cyclone Overflow Density	%Sw/w	38-47%	
Overflow Product Size (P_{80})	µm	75.0	
Estimated Gold Upgrade Ratio in Underflow (Relative to cyclone feed)	-	1.12	
Regrind Mill			
Mill Type	type	VertiMill	
Quantity of Mills Installed	#	1.00	
Selected Mill Size	type	VM 1500	
Wet Or Dry Milling	type	Wet	
Mill Feed Slurry Density	%Sw/w	70%-75%	
Circuit Feed Rate	tph	146	
Mill Feed Size (F_{80})	µm	75	
Open Or Closed Circuit	type	Reverse Closed Circuit	
Ball Material (High Cr, Cast Or Forged)	type	Forged	
Mill Liner Material (Steel, Rubber, Polymet)	type	Steel	
Mill Ball SG	t/m³	4.80	
Selected Ball Size	mm	12 mm/19 mm	
Mill Power (absorbed)	kW	1000	
Motor Power (installed)	kW	1120	
Regrind Mill Classification			
Type of Classification	type	Hydrocyclone Cluster	
Quantity of Clusters	#	1.00	
Cyclone Operating Pressure	kPa	85	
Cyclone Tonnage U/F Relative to Mill Feed	%	70%	
Cyclone Overflow Density	%Sw/w	35.0%	
Overflow Product Size (P_{80})	µm	45	
Circulating Load		200%-250%	
Trash Screening			
Type of Screen	type	Linear	
Screen Feed Density	%Sw/w	35.0%	
Aperture Size	µm	600	
Underflow Slurry Transport Method	type	Gravity	

17.5.3 Gravity Concentration

The gravity concentrator feed is pre-screened on a vibrating screen (220-SR-100) to remove oversize material not suited for the concentrator. The screen sprays will be used to dilute the feed to the concentrator to 60%-65% solids by mass while also increasing screening efficiency. Screen oversize will return to the mill, with the underflow gravitating to the gravity concentrator (220-GS-104). Concentrator tailings will gravitate to the ball mill feed while the concentrate will report to a batch dissolution reactor. Fluidizing water will be supplied to the concentrator, which will be fenced for security reasons.

Concentrate from the gravity concentrator is collected in the Acacia concentrate de-watering cone (220-ZA-111). Dewatering cone underflow at ±60% solids by mass is fed to the intensive-disscolution reactor on a batch basis, once every 24 hours, while the overflow is returned to the ball mill discharge sump. A batch solution of cyanide and caustic is made up and this leach solution is used to achieve a target gold extraction in excess of 95%. The resultant pregnant leach solution is then pumped to the dissolution circuit, electrowinning cell, feed tank (520-TK-323). The solids in the reactor are rinsed with water after the batch leach. The rinse water with the solids is then pumped to the milling discharge sump.

Spillage in the dissolution area will be collected and pumped to either the mill discharge sump or the pre-oxidation tank feed box. The dissolution reactor will be enclosed with a fence for security reasons.

The parameters of the gravity concentration section are summarized in Table 17.7.

The layout has space has space provision for the installation of 2nd gravity concentrator in future.

Table 17.7 Gravity Concentration

Gravity Circuit	Units	Value	
Scalping Screen			
Type of Screen	Type	Vibrating	
Aperture Size Design	Mm	1.6 x 13 – Slotted	
Gravity Concentrators			
Gravity Feed Grade	g/t	2.48	2.96
Type of Gravity Unit	Type	Falcon	
No of Gravity Units	#	1	
Unit Feed	t/h/unit	140	
Unit Feed Density	%Sw/w	65-70%	
Unit Flushing Water Requirement	m^3/h	12-20	
Concentrate Mass Pull (% of Unit Feed) - Estimated	%	0.100%	0.150%
Concentrate SG	t/m^3	TBC	
Concentrate Density	%Sw/w	20.0%	
Gravity Concentrate Treatment			
Type of Concentrate Treatment	Type	Acacia	
Quantity of Units	#	1.00	
No of Leaches per Day	#	1.00	
Dissolution Reactor Volume Model		CS 2000	
Pregnant Solution Per Batch	m^3	5.00	

17.5.4 Thickening, Pre-Oxidation, Pre-Leach and CIL

Cyclone overflow gravitates to the pre-leach thickener (260-TH-120), where it is thickened to produce an underflow density of 45% solids by mass. The thickener underflow is pumped to the 1,000 m^3 pre-oxidation tank 300-TK-164 and spillage will be pumped (260-PP-123) back to the thickener feed box.

There is a step height of 600 mm between the pre-oxidation tank and the first CIL tank to ensure that slurry is transferred from one tank to the next by gravity. The pre-oxidation tank has a bypass facility to ensure continuity in production if the tank is taken offline for maintenance. Oxygen will be introduced to the bottom of the pre-oxidation tank by spargers and a pump with high shear reactor will increase the dissolved oxygen content. Lead nitrate will be added to aid the process and milk of lime for pH control.

The CIL circuit consists of 6 × 1000 m^3 (300-TK-127/128/129/130/131/132) tanks in series, with slurry transferred between tanks by Kemix MPS (P) inter-tank screens and launders. All tanks have a bypass facility to ensure continuity in production if a tank is taken offline for maintenance. Carbon concentrations of 12-15 g/L will be maintained in each tank, with counter-current carbon flow. In the CIL circuit, cyanide is added to the first and/or second CIL tank so as to effect the leaching of gold. Barren electrowinning solution and elution spillage is recycled to the first CIL tank to boost cyanide levels. The CIL circuit will be operated to achieve a carbon loading of 1,100-1,500 g/t, with daily inter tank carbon transfers to achieve constant carbon distribution. This translates to a loaded carbon batch size of ±5 tons per day. The CIL circuit will be operated to achieve a desired gold grade of less than 0.22 – 0.25 g/t in the tailings (dependant on mill feed grade) and a solution gold tenor of less than 0.005 ppm.

CIL tailings from the final interstage screen and launder gravitate's via the Carbon Safety Linear Screen (300-SR-156) to the cyanide detoxification circuit.

Loaded carbon from the first CIL tank will be pumped to a vibrating screen for washing. (300-SR-168). The screen oversize (washed loaded carbon) will gravitate to the elution circuit acid-wash tank (500-TK-255) and the undersize (slurry) will report back to the first CIL tank.

A spillage pump is installed in the CIL bund (300-PP-159). Intensive leach reactor spillage and CIL spillage are recycled to the pre-oxidation tank.

Hydrogen cyanide and ammonia gas detection will be installed on both the CIL and Detox circuits, together with cyanide and pH control equipment in required locations. A safety shower will be installed in the areaA maintenance bay with a screen frame and washing facilities will be supplied for the cleaning of the inter-stage screens as well as a tower crane (300-XL-161) for maintenance.

The key parameters relating to thickening and CIL are summarized in Table 17.8.

Table 17.8 Thickening, Pre-Oxidation, and CIL

Thickening, Pre-Oxidation and CIL	Units	Value	
Pre-Leach Thickening			
Type of Thickener	Type	High Rate	
Thickener Feed Density	%Sw/w	35%	
Thickener Feed Solid Flowrate Design	t/h	145.5	
Thickener Feed Slurry Flowrate Design	m³/h	322	
Specific Settling Area	t/m²/h	0.5	
Underflow Density	%Sw/w	40% - 45%	
Selected Thickener Diameter	m	21.0	
Flocculant Addition Rate	g/t	30.0	40.0
Pre-Oxidationand CIL			
Leach Feed Grade	g/t	1.69	2.18
Leach Feed Density	%Sw/w	40.0% - 45.0%	
Leach Feed Slurry Flowrate	m³/h	230 - 270	
No of Pre-Oxidation Tanks	#	1	
Selected Pre-Oxidation Tank Volume	m³	1000	
Calculated Pre-Oxidation Residence Time	hrs	3.59 – 4.22	
No of CIL Tanks	#	6	
Required Total CIL Residence Time	hrs	24	
Calculated Total CIL Residence Time	hrs	21.5 – 25.3	
Selected CIL Tank Volume	m³	1000	
Tank Volume Loss	%	3. 0%	
Carbon Concentration Design	g/L	12 - 15	
Gold Loaded on Carbon	g/day	5075 - 6710	
Carbon Movement Per Day	t	5.00	
Carbon Loading	g/t	1015 - 1342	
Final Carbon Loading	g/t	1065 - 1092	
Carbon Residence Time/Tank (Modelled)	hrs	72.5- 81.5	
Carbon Movement Rate	t/h	0.208	
Mass of Carbon per Stage	t	15.1 – 17.0	
Time Required to Move Loaded Carbon	hrs	6.7 - 8.3	
Mass of Carbon to Elution per Batch	t	5.00	
CIL Carbon Transfer Rate	t/h	0.60 - 0.75	
Slurry Flow During Loaded Carbon Transfer	m³/h	50	
Time Required to Move per Inter-tank Flow	hrs	6.7 - 8.3	
Carbon Inter-tank Transfer Rate	t/h	0.60 - 0.75	
Slurry Flow During Inter-tank Carbon Transfer	m³/h	50	
CIL INTERSTAGE SCREENS	**UNITS**	**VALUE**	
Interstage Screen Type	type	MPS(P)	
Interstage Screen Selected	-	MPS(P) 400	
Aperture Size	µm	630	
Leach Slurry Flow	m³/h	230-270	
Inter-tank Carbon Transfer	m³/h	50	
Eluted Carbon Transfer	m³/h	50	
Total Flowrate	m³/h	280 - 320	
Open Area	%	22.26	
Screen Volumetric Flowrate (Flux) Nominal	m³/m²/h	57.5 - 67.6	
Screen Volumetric Flowrate (Flux) Max	m³/m²/h	85.0	
Maximum Flowrate Available For Carbon Transfer	m³/h	70.0 – 110.0	
Selected Flowrate For Loaded Carbon Transfer	m³/h	50	
Selected Flowrate For Inter Tank Carbon Transfer	m³/h	50	
Quantity of Interstage Screens	#	7.00	
Dry Lifting Mass	t	2.13	
Typical Operating Mass	t	3.12	
Maximum Lifting Mass	t	4.03	

17.5.5 Acid Wash and Elution

The elution circuit processes one ±5 dry tonne batch per day of gold-laden carbon for subsequent gold recovery by electrowinning and smelting. Elution is based on the Anglo-American Research Laboratory (AARL), split-circuit process. The circuit essentially consists of loaded-carbon acid washing, elution and carbon regeneration.

Acid Wash

Hydrochloric acid (HCl) at a concentration of 33% w/w is transported in 1 m^3 IBCs (intermediate bulk containers) to the plant by road in bulk containers. Upon removal from the containers, the IBCs are stored in the chemical shed. An IBC container will be moved to the elution area as required. Hydrochloric acid at 33% w/w is pumped using a peristaltic pump (500-PP-462) as required to the dilute acid make-up tank (500-TK-286), where it is diluted to 3% w/w HCl for acid washing of the carbon.

Loaded carbon recovery screen oversize is delivered at a rate of ±5 t per daily cycle to the acid-wash hopper (500-TK-255) located directly above the elution column (500-ZM-256). Dilute hydrochloric acid is pumped (500-PP-287) into the acid-wash hopper. The loaded carbon is soaked for ±1 hour in the acid solution. The acid solution is then drained to the spent acid tank (500-TK-289). Wash water is then passed through the hopper and the carbon is washed until a neutral pH is achieved, with the rinse effluent also draining to the spent acid tank.

Pre-Treatment Cycle

After acid washing, the loaded carbon is discharged into the elution column (500-ZM-256). Sodium cyanide, sodium hydroxide and reagent water are mixed in the strip solution make-up tank (500-TK-257) to obtain a strip solution concentration of 3% w/w NaOH and 1% w/w NaCN. The strip solution, amounting to just short of one carbon bed-volume, is then transferred with a fixed speed helical screw strip solution pump (500-PP-260) to the elution column base. The strip solution travels via the column eluate recovery heat exchanger (500-XH-262) and the primary strip solution heat exchanger (500-XH-263/294) operating in conjunction with the strip solution heater (500-XH-264/295) in order to be preheated to a solution temperature of ±125°C. The heat exchangers and heater are provided as a vendor package.

Once the strip solution has been transferred into the elution column, a portion of the intermediate strip solution from the previous elution batch is pumped from the semi-pregnant solution make-up tank (500-TK-293) via the heat exchangers with the strip solution pump to fill the column and prime the associated pipe-lines.

Elution Cycle

When the “hot” column circuit is fully primed, the semi-pregnant solution make-up tank is by-passed and the column contents are re-circulated by the strip solution pump through the heat exchangers and the column until the column temperature is stable at ±125°C.

Once the column circuit temperature has stabilized at ±125°C, intermediate strip solution from the intermediate strip solution tank, which contains approximately four bed-volumes of intermediate strip solution from the previous elution batch, is pumped through the column via the strip solution heat exchangers with the strip solution pump. Pregnant solution escapes from the top of the column and passes via the column eluate recovery heat exchanger and the CIL pregnant strip solution in-line strainers to the CIL pregnant solution tank (520-TK-320) located in the electrowinning area.

Column Rinse Cycle

When the semi-pregnant solution make-up tank's contents have been emptied to the CIL pregnant solution tank (520-TK-320), wash water is pumped with the strip solution pump from the strip solution make-up tank to rinse and cool the stripped carbon in the column and fill the semi-pregnant solution make-up tank for the following batch elution process. During the cold rinse the strip solution heater is off. The total elution process takes approximately 6 hours.

Carbon Regeneration

On completion of the cold rinse cycle, water from the strip-solution make-up tank is pumped directly with the strip solution pump to the carbon regeneration kiln feed sieve-bend via a T-piece on the outlet at the bottom of the column. Pressure is then applied to the elution column in order to supply adequate energy to the column contents and once the pressure set-point is reached, the bottom outlet is opened and the eluted carbon is effectively pressure-educted from the column and transported to the carbon regeneration kiln feed sieve-bend. Sieve-bend oversize gravitates to the kiln feed hopper (500-TK-270) which discharges into the kiln feeder (500-FD-291). The water drained from the sieve bend and any additional drainage water from the kiln hopper or feeder reports to the carbon quench tank (500-TK-273). The carbon is regenerated in the rotary kiln (500-XF-272) and then discharged into the barren carbon quench tank, from where it is pumped to the CIL barren-carbon dewatering screen (500-SR-279) via the screen feed box.

Make-up carbon is delivered in bulk bags and is added to the carbon quench tank when required.

Spillage Handling and Services

Spillage in the discretely concrete-bunded elution area's spillage sump is pumped to the 1st CIL tank; spillage from the acid tank bund is pumped to the spent acid tank; whereas spillage accumulated in the discretely concrete-bunded carbon regeneration kiln area is pumped to the carbon quench tank. A safety shower supplied with potable water is located within the elution area.

The key parameters relating to acid wash and elution are summarized in Table 17.9.

Table 17.9 Acid Wash and Elution

Acid Wash and Elution	Unit	Value
Loaded Carbon Screen		
Type of Screen	type	Vibrating
Aperture Size	mm	0.630
Feed Slurry Flowrate	m^3/h	50
Feed Slurry Density	%Sw/w	40.0% -45.0%
Feed Slurry Density	%Sv/v	22.1%
Underflow Slurry Transport Method	type	Gravity
Acid Wash		
Type of Acid Wash Vessel	type	Conical Tank
Material of Construction	type	MSRL
Acid Wash Tank		
Minimum Acid Wash Volume	BV	2.00
Acid Wash Time / Recirculation	min	30.0
Acid Wash Solution Strength - HCl	%	3.00%
Rinsing		
Rinse Volume	BV	2.00
Rinse Time	min	30.0
Cone Emptying Method	type	Gravity
Elution		
Elution method	type	Pressure split AARL
Material of construction	type	Stainless steel
Operating temperature	ºC	125-140
Operating pressure	kPa	300
Carbon transfer method	type	Hydraulic
Required quantity of elution batches per day	#	1
Elution batch size	t	5
Elution column selected size	t	5.00
Design barren carbon loading	g/t	50.0
Bed volume	m^3	10
Elution pump type		Helical Rotor
Pre-soak flowrate	BV/h	1.00
Pre-soak volumes	BV	1
Cyanide strength in solution	%	1.00%
Caustic strength in solution	%	3.00%
Pre-soak solution volume	m^3	10
Elution flowrate	BV/h	2.00
Elution cycle time	min	150
Elution volume	m^3	50
Elution solution	type	Recirculation Tank solution
Rinse flowrate	BV/h	2.00
Rinse cycle time	min	150
Rinse volume	m^3	50
Rinse solution	type	Clean water
Cooling flowrate	BV/h	2.00
Cooling cycle time	min	30.0
Cooling volume	m^3	10
Total eluate volume	m^3	60
Minimum eluant tank volume	m^3	66.7
Elution Heating		
Elution heating medium	type	Thermic oil
Elution heaters type	type	Diesel
Elution heating required	kW	1 750

Acid Wash and Elution	**Unit**	**Value**
Regeneration		
Regen feed hopper		
Dewatering means	type	Sieve Bend
Feed hopper capacity	t	6.00
Feed hopper capacity	m^3	13.0
Regeneration Kiln		
Reactivation capacity	kg/h	300
Type of kiln	type	Horizontal
Type of kiln heating	type	Diesel fired
Temperature control	type	Automatic
Regen. temp	ºC	750
Filling ratio	%	10.0%
Kiln angle	°	0.700
Residence time at temperature	min	15.0
Residence time at total	min	25.0
Feed moisture	%w/w	50.0%
L:D ratio max		6.00
Regen. kiln running time per day	hrs	18.0
Type of regen. carbon storage tank	type	Open tank
Carbon transfer method	type	Hydraulic

17.5.6 Electrowinning and Gold Room

The electrowinning plant processes both intensive-leach reactor and CIL pregnant solutions to recover gold for downstream smelting.

The CIL pregnant solution tank (520-TK-320) and Intensive leach pregnant solution tank (520-TK-323) that supply pregnant feed liquor from the CIL elution ciruit and ACACIA reactor to the electrowinning cells, as well as barren solution tank (520-TK-331) that recycles barren effluent back to the plant, are situated in a discrete, concrete-bunded area immediately adjacent to the gold room.

High-security gold room processing comprises electrowinning of gold from the pregnant solutions, followed by drying and fluxed smelting of the resultant gold sludge to a final doré bar product to be transported to the refinery. The gold room layout design accommodates both full security guard surveillance and second-level surveillance by remote control CCTV cameras with viewing facilities in the process manager and security foreman offices.

Toilet and crib-room facilities are provided within the secure area to minimize the frequency of human-traffic passage. Gold room ingress and egress are controlled and monitored via a proximity card and turnstile system.

Of the four identical electrowinning cells, two are dedicated to the CIL circuit (520-EC-326/327), and one is dedicated to the gravity circuit (520-EC-329). The fourth cell is operated as a common standby unit (520-EC-328). The cells are equipped with stainless steel anodes and stainless steel wool cathodes. A direct current is passed through the cells between the electrodes, and the electrolytic action results in the gold in solution plating out onto the cathodes. The electrowinning cells are provided with a fume extraction fan and associated hoods and ducting which expel fumes generated during the process to atmosphere.

CIL pregnant solution is pumped from the CIL pregnant solution tank to the CIL electrowinning cells, and recirculated to the CIL pregnant solution tank for the duration of the 18 hour process. Similarly, ILR pregnant solution is recirculated from the ILR pregnant solution tank via its electrowinning cells to the ILR pregnant solution tank for the same duration. On completion of the electrowinning cycle, the barren solution from both circuits is directed, using the same recirculation pumps, to the barren solution tank (520-TK-331). Dissolution reactor flushing water is also received into the barren solution tank. Barren solution is pumped back to the CIL circuit tanks 1 and 2. Manual addition of sodium hydroxide to the two pregnant solution tanks from the sodium hydroxide manifold is provided for. Spillage accumulated in the electrowinning area bund is pumped into the Barren solution tank (520-PP-336).

The loaded cathodes are manually hoisted from the electrowinning cells and taken to the cathode wash table (540-ZM-351) where the gold sludge is removed from them by high pressure water blasting, with the sludge reporting to the cathode wash sludge settling tank (540-TK-352) which also receives loosened sludge from the EW cell drains. Excess water is decanted from the settled sludge which is then dried in a drying oven (540-XT-353), prior to direct-smelting with flux in a furnace (540-XF-356) to produce doré bars for further refining. The furnace is provided with a hood, and appropriate ducting to deliver furnace gases to atmosphere. The doré bars are stored in a safe while awaiting delivery to the refinery.

Gold room spillage accumulates in a dedicated sump within the area and is pumped (540-PP-365) via a gold trap to the cathode wash sludge collection tank.

A safety shower is located in the gold room area.

The key parameters relating to electrowinning and the gold room and summarized in Table 17.10.

Table 17.10 Electrowinning and Gold Room

Electrowinning	Units	Value	
Cell Type	type	Atm Sludging	
Mode Of Cell Operation/Feed	type	Parallel	
Electrolyte source		**Gravity**	**CIL**
Volume to Be Treated per Batch	m^3	5.00	60
Pregnant Liquor Au Concentration	mg/L	1266	96
Gold Extraction per Pass (Design)	%	15.0%	40.0%-50.0%
Total Design Gold Extraction per Electrowinning Batch Cycle	%	98.0%	98.0%
Barren Solution Grade (Au)	mg/L	25.3	1.93
Electrowinning Batch Cycle Time	Hrs.	18.0	18.0
Electrowinning Circulating Flowrate Required	m^3/hr	7.24	26.1 – 32.0
Recommended Specific Eluate Flow Rate	m^3/h/m^2	30.0	30.0
Rectifier Sizing	A	1 000	1 000
Selected Quantity of Cells	#	1.00	3.00
Cell Solution Temperature	ºC	40.0	60.0
Type of Cathode	type	S/Steel Mesh	
Type of Anode	type	S/Steel	
Quantity of Cathodes per Cell	#	6	
Cathode Sludge Removal	type	High Pressure Wash	
Total Barren Solution	m^3	65.0	
Barren Solution Grade (Au)	mg/L	2-4	
Barren Tank Capacity	m^3	82	
Sludge Handling			
Type	type	Sludge/Decant Tank	
Sludge Moisture	%	50.0%	
Drying and Smelting			
Oven Type	type	80L	
Quantity of Ovens	#	1.00	
Quantity of Trays	#	5.00	
Installed Power in Oven	kW	30.0	
Smelting Furnace Type	type	Diesel Fired	
Installed Power	kW	-	
Bullion Mould Capacity	-	800 Oz	
Type of Crucible	type	A200	
Cascade Trolley/ Cascade Mould Trays	#	4 Bullion and 2 Slag	

17.5.7 Cyanide Detoxification and Arsenic Precipitation

The cyanide detoxification and Arsenic precipitation circuit consists of 3 × 260 m^3 tanks in series (400-TK-178/187/185). Cyanide detoxification is performed in the first two tanks andis specifically achieved by an SO_2/air process, operated at pH 8.0 – 9.0. Each detox tank is fitted with a Chinese hat for the introduction of air, with an allowance for copper sulphate and SMBS addition to the first two tanks. pH control is achieved by the addition of Lime to each of the detox tanks as required.. The detox circuit will be operated to achieve a cyanide concentration in the final tailings stream of less than 50 ppm WAD.

In the third tank the addition of Ferric Chloride addition allows for the precipitation of arsenic from solution, the pH in this tank is controlled in the range 6-8 with the addition of sulphuric acid.

Detox circuit tailings gravitates to the final tailings disposal tank (400-TK-190) via a primary cross-cut/secondary Vezin sampler system. Plant tailings are comprised of the following streams; detoxification circuit tails, carbon fines, spent acid and acid spillage. Plant tailings are pumped to the tailings storage facility (400-DM-197) by a tailings pump train comprising of two pumps operating in series (400-PP-191/192). The design caters for the installation of a standby tailings pump train (400-PP-193/194).

The gland water requirements for the tailings pumps are serviced by high pressure GSW supply pumps. The tailings GSW supply pumps are designed to deliver 1 bar above the delivery pressure of the secondary pump.

Sulphuric Acid is supplied to the detox area in 1000 litre IBC's (intermediate bulk containers) which are transferred to two storage tanks within the detox bunded area by an IBC transfer pump 400-PP-727.

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

Return water is pumped (400-PP-199/200) backed to the process water pond via a silt trap. A spillage pump is installed at the tailings return water pumping station (400-PP-201).

The key parameters relating to the cyanide detoxificatnion and Arsenic Precipitaiton circuit are summarized in Table 17.11.

Table 17.11 Cyanide Detoxification and Arsenic Precipitation

Cyanide Detoxification	Units	Value
Cyanide Detoxification Method	type	SO_2/Air
Total Detox Air Addition Rate	Nm^3/hr/tank	500
Required residence time	min	90.0
Actual residence time	min	112-121
pH Set-Point	pH	8.0 -9.0
Quantity of tanks	#	2
Detox selected tank volume	m^3	260
$CuSO_4$ addition per tank	g/t	65
SMBS addition per tank	g/t	770-1100
Lime addition per tank	g/t	0.31-0.45
Cyanide in final detox tailings	ppm	<50ppm WAD
Arsenic Precipitation		
Qunaity of Tanks	#	1
Detox Selected tank volume	m^3	260
Ferric Chloride Addition	g/t	300
H_2SO_4 Addition	g/t	1000

17.5.8 Reagents

Caustic Soda Make-up and Storage

Caustic soda (NaOH) is delivered to the plant in 1,000 kg bulk bags in “pearl” form. The bags are lifted by the reagent area overhead crane and delivered as required to the caustic soda bag-splitter cabin (700-ZA-440) located above the mechanically agitated caustic soda mixing tank (700-TK-441). The hoist lowers the bag rapidly onto the caustic bag-splitter, and the contents discharge into the mixing tank, where they are diluted with reagent water to solution strength of 20% w/v solution. The caustic soda solution is pumped into the caustic soda storage tank. The solution is pumped to required points of use (cyanide make-up, intensive leach reactor, strip solution make-up tank and electrowinning) using fixed speed helical screw pumps (700-PP-445/446) (operating and standby) as required.

Sodium hydroxide and sodium cyanide make-up share a common, discrete, concrete bund. Spillage arisings gravitate to a dedicated sump, and are pumped to the detoxification circuit (700-PP-456).

Sodium Cyanide Make-up and Storage

Sodium cyanide (NaCN) is delivered to the plant in 1,000 kg bulk bags. The bags are lifted by the dedicated 2 t cyanide bag hoist and delivered as required to the cyanide bag-splitter cabin (700-ZA-447) located above the mechanically agitated sodium cyanide mixing tank (700-TK-448). The hoist lowers the bag rapidly onto the cyanide mixing tank feed hopper bag-splitter, and the contents discharge into the mixing tank, where it is diluted with reagent water to solution strength of 20% w/v solution. The cyanide solution is pumped into the cyanide storage tank, from where it is pumped to required points of use. Dosing pumps (700-PP-452/453) feed cyanide to the CIL circuit via a pressure relieved manifold, while the intensive leach reactor and the column elution strip solution make-up receives batched cyanide via a fixed-speed pump (700-PP-454) as required.

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

Copper Sulphate Make-up and Storage

Copper sulphate is delivered to the plant in 1,200 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the copper sulphate screen-hopper positioned above the agitated copper sulphate mixing tank. The reagent is washed through a screen into the mixing tank (720-TK-470) and mixed to a 20% w/v solution with reagent water prior to contents being dropped into the storage tank (720-TK-473). From here it is pumped to the cyanide detoxification circuit by a variable speed dosing pump (720-PP-474).

Copper Sulphate make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

SMBS Make-up and Storage

SMBS is delivered to the plant in 1,200 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the SMBS hopper positioned above the mechanically agitated SMBS mixing tank (720-TK-477). The reagent is washed into the mixing tank and mixed to a 20% w/v solution with water prior being pumped into the SMBS storage tank (720-TK-481). From here it is pumped to the detoxification circuit via a variable speed dosing pump (720-PP-482). A safety shower supplied with potable water is strategically located within the area.

SMBS make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

Lead Nitrate Make-up and Storage

Lead Nitrate is delivered to the plant in 1,000 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the Lead Nitrate hopper positioned above the mechanically agitated Lead Nitrate mixing tank (720-TK-671). The reagent is gravity fed into the mixing tank and mixed to a 20% w/v solution with water prior to being pumped into the Lead Nitrate storage tank (720-TK-676). From here it is pumped to the CIL circuit via a variable speed dosing pump (720-PP-677). A safety shower supplied with potable water is strategically located within the area.

Lead Nitrate make-up spillage is pumped by the common reagent spillage pump (740-PP-678) to the detoxification circuit feed tank.

Ferric Choride Make-up and Storage

Ferric Chloride is delivered to the plant in 1,000 kg bulk bags, which are lifted by the reagent area overhead crane as required and emptied into the Ferric Chloride hopper positioned above the mechanically agitated Ferric Chloride mixing tank (720-TK-661). The reagent is washed into the mixing tank and mixed to a 20% w/v solution with water prior being pumped into the Ferric Chloride storage tank (720-TK-664). From here it is pumped to the detoxification circuit via a variable speed dosing pump (720-PP-665). 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 (740-PP-678) to the detoxification circuit feed tank.

Flocculant Make-up and Dosing

Flocculant is delivered to the plant in 25 kg bulk bags and manually loaded into the flocculant powder feed hopper (730-ZA-497). From here it is fed by the flocculant screw feeder (730-FD-501) into a blower/venturi system for transfer to a wetting head (730-ZM-509/510) and mixed with water to achieve 0.25% solution strength and fed directly into the mechanically agitated flocculant transfer tanks (730-TK-503/507) From here it is pumped to the pre leach thickener using a variable speed pump (730-PP-506).

Flocculant area make-up area spillage is pumped by the spillage pump (740-PP-519) to the detoxification circuit feed tank.

Hydrated Lime Make-up and Distribution

Hydrated lime is delivered in 1,000 kg bulk bags which are transported to the plant by road in containers. The bags are lifted by the reagent area overhead crane and lowered rapidly onto the milk of lime addition bag cabin (740-ZM-511) as required, with the contents discharging into the feed hopper. The feed hopper is equipped with a vibrating system and variable speed rotary feeder (740-FD-514).

The rotary feeder meters the hydrated lime into the agitated milk of lime mixing tank (740-TK-516) where it is diluted to 20% w/v slurry. The lime slurry is pumped to the mill feed and the caynide detoxification circuit (740-PP-517/518) in a ring-main arrangement. A facility is installed to allow for the addition of lime to the pre leach thickener and CIL tank No. 1.

At the mill feed, the rate of primary dosage of lime into the CIL feed slurry is controlled by a /integrated-timer interface which operates a solenoid-controlled valve on the lime ring-main off-take.

Lime make-up area spillage is pumped by the common reagent spillage pump (740-PP-678)to the detoxification circuit feed tank. A safety shower provided with potable water is strategically located within the Lime make-up area.

The reagents to be used in the processing plant are listed in Table 17.12.

Table 17.12 Reagents

Raegents	Units	Value
Lime		
Delivery method	type	Bulk bags
Delivery size	kg/bag	1 000
Type of lime	type	Hydrated lime
Equivalent % $Ca(OH)_2$	% CaO	90.0%
CIL consumption as 100%	kg/t	1.8
Detox consumption as 100%	kg/t	0.31
Consumption as 100%	kg/h	310
Consumption at 90% activity	kg/h	340
Physical form	type	Powder
Lime solids SG	t/m³	2.25
Lime addition system	type	Milk of lime Ringmain
Lime make-up strength	%S w/v	20.0%
Number of make-up per day	#	1.00
Number of make-up/dosing tanks required	#	1.00
Total storage and dosing tank capacity	hrs	29.2
Selected storage tank capacity	m³	50.0
Lime Consumpiton	m^3/hr	1.71

Reagent	Unit	Value
Sodium Cyanide (NaCN)		
Delivery method	type	Bag in box
Delivery size	kg/box	1 000
Physical form	type	Briquettes
CIL consumption	kg/t	0.65
Elution consumption	kg/Batch	100
Dissolution	kg/Batch	50
Cyanide make up strength	%S w/v	20.0%
Number of make-up per day	#	1.00
Number of make-up tanks required	#	1.00
Number of dosing tanks required	#	1.00
Total storage and dosing tank capacity	days	2. 47
Selected storage tank capacity	m³	29.0
Selected dosing tank capacity	m³	30.0
Cyanide dosing method	type	Ring main
Velocity required in ring main (max)	m/s	1.50
Calculated consumption (nominal)	m³/h	0.51
Caustic Soda (NaOH)		
Delivery method	type	Bags
Delivery size	kg	1 000
Physical form	type	Pearls
Caustic consumption elution	kg/Batch	150
Caustic consumption dissolution	kg/Batch	12.0
Caustic consumption electrowinning	kg/Batch	intermittent
Caustic make up strength	%S w/v	20.0%
Number of make-up tanks required	#	1.00
Number of storage/dosing tanks required	#	1.00
Total mixing and dosing tank capacity	days	7.41
Selected mixing tank capacity	m³	5.0
Selected dosing tank capacity	m³	6.0
Caustic Soda dosing method	type	Direct dosing
Velocity required in pipeline	m/s	1.50
Calculated consumption (nominal)	m³/day	0.81
Hydrochloric Acid (HCl)		
Delivery method	type	IBC
Delivery size	kg/Iso -container	1 185
Physical form	type	Solution
HCl delivered strength	%	33.0%
33% HCl SG	t/m³	1.20
Acid wash consumption at 100% strength	kg/Batch	328
Cyanide dosing method	type	Direct from IBC
Calculated consumption at 33% strength (nominal)	m³/batch	0.829

Reagent	Unit	Value
SMBS		
Delivery method	type	Bags
Delivery size	kg/bag	1 200
Physical form	type	Granular
SMBS consumption (Nominal)	g/t	770
Number of make ups per day	#	1.00
Make up strength	%S w/v	20.0
Number of make-up tanks	#	1.00
Selected make-up tank volume	m³	8.0
Number of dosing tanks	#	1.00
Selected dosing tank volume	m³	15.0
Dosing method	type	Direct dosing
Velocity required in pipeline	m/s	1.50
SMBS solution consumption (Nominal)	m³/h	0.56
CuSO4		
Delivery method	type	Bags
Delivery size	kg/bag	1 200
Physical form	type	Granular
$CuSO_4$ consumption (Nominal)	g/t	65
Number of make ups per day	#	1.00
Make up strength	%S w/v	25.0%
Total storage and dosing capacity	days	-
Number of make-up tanks	#	1.00
Selected make-up tank volume	m³	5. 0
Dosing method	type	Direct dosing
Number of dosing tanks	#	1.00
Selected dosing tank volume	m³	6.00
Velocity required in pipeline	m/s	1.50
$CuSO_4$ solution consumption (Nominal)	m³/h	0.05
Reagent	**Unit**	**Value**
Lead Nitrate		
Delivery Method	type	Bags
Delivery Size	kg	1 000
Physical Form	type	Pearls
Leach Concumption	g/t	25
Lead Nitrate Make Up Strength	%Sw/v	20.0%
Number of Make-Up per Day	#	1.00
Number of Make-Up Tanks Required	#	1.00
Number of Storage/Dosing Tanks Required	#	1.00
Dosing Tank Capacity	days	13.70
Selected Mixing Tank Capacity	m³	5.50
Selected Dosing Tank Capacity	m³	6.00
Dosing Method	type	Direct Dosing
Calculated Consumption (Nominal)	m³/day	0.44
Ferric Chloride	**Unit**	**Value**
Delivery Method	type	Bags
Delivery Size	kg/bag	1 000
Physical Form	type	Granular
$FeCl_3$ Consumption	kg/t	0.30
Number of Make Ups per Day	#	1.00
Make Up Strength	%Sw/v	20.0%
Number of Make-Up Tanks	#	1.00
Selected Make-UpTank Volume	m³	5.00
Dosing Method	type	Direct Dosing
Number of Dosing Tanks	#	1.00
Selected Dosing Tank Volume	m³	6.00
Dosing Tank Capacity	days	1.15
Velocity Required In Pipeline	m/s	1.50
$FeCl_3$ Solution Consumption	m³/h	0.22
Flocculant		
Delivery method	type	Bags
Delivery size	kg/bag	25.0
Physical form	type	Granular
Flocculant consumption	g/t	30.0 50.0

Reagent	Unit	Value
Total flocculant consumption	kg/h	4.4 – 7.3
Number of make ups per day	#	1.00
Floc make up strength	%S w/v	0.250%
0.25% flocculant solution density	t/m³	1.00
Total storage and dosing capacity	days	2.0
Number of make-up/dosing tanks	#	2
Selected make-up/dosing tank volume	m³	42.0
Dilution factor	factor	1:10
Flocculant dosing method	type	Direct dosing
Velocity required in pipeline	m/s	1.50
0.25% flocculant solution consumption	m³/h	1.75 – 2.92
Activated Carbon	**Unit**	**Value**
Delivery method	type	Bags
Delivery size	kg/bag	600
Type of carbon in use	Mesh	8 × 16
Type of carbon in use	mm	1.68 × 2.38
Carbon bulk density	t/m³	0.480
Carbon dry SG	t/m³	0.800
Carbon wet SG	t/m³	1.37
Consumption rate	g/t	25.0

17.5.9 Water

Process Water

Return water from the tailings storage facility return water pond is pumped to the plant process water tank (600-TK-400) where it is joined by the overflow from the pre-leach thickener. Process water is supplied to the plant with two dedicated process water pumps (600-PP-402/403) (operational and standby).

Make-up water from the river system can be added to meet process water demand requirements.

Clean Water

Clean water is supplied to the plant from the clean water tank (600-TK-405), which receives make-up water from the river system via pumps 600-PP-420/422. The clean water tank provides for plant clean water and gland service water requirements with two dedicated clean water pumps (600-PP-406/407) (operational and standby) via a clean-water supply line.

Potable Water

Clean river water is supplied to the water treatment plant and potable water is supplied from the water treatment plant to the plant potable water tank (600-TK-412). Potable water is supplied to the plant via a dedicated potable water supply pump (600-PP-413).

Fire Water

Fire water is supplied from the bottom section of the clean water tanks (600-TK-405) to the plant via a dedicated vendor package fire water supply pump (600-PP-419), fire water jockey pump (600-PP-417) as well as fire water diesel pump (600-PP-416) in case of power failure.

The water usage is summarized in Table 17.13.

Table 17.13 Water

Water Services	Units	Value
Return Water		
Return water % of tailing disposal (wet season)	%	100%
Return water % of tailing disposal (dry season)	%	50.0%
Process Water Supply	type	Tank
Capacity	m^3	200
Process water storage	type	TSF pond
Capacity (nominal)	m^3	60 000
Clean Water Storage	type	Tank
Capacity (total)	m^3	500
Capacity (total) excl reserve	m^3	300
Capacity (total) excl reserve	hrs	4.3
Plant Potable Water Storage	type	Tank
Selected size	m^3	30
Fire Water Supply	type	Clean water tank reserve

17.5.10 Plant Services

High Pressure Air Services

The compressed air purification system consists of high efficiency filters (610-FL-419/435/421) and a dryer (610-DR-420) which is installed between the filters. The filters remove contaminants such as water, oil and solid particles from the compressed air stream. The dryer is responsible for the removal of moisture within the air delivering high-purity air to the receivers.

The instrument air receiver is designed to hold 3.85 m^3 of air at 7.5 bar.

Oxygen Plant

A Vendor-package, Pressure Swing Adsorption (PSA) oxygen plant, provides piped oxygen to the pre-oxidationand CIL tank spargers as well as the pre-oxidation high shear reactor.

Low Pressure Air Services

A low pressure air circuit is comprised on two low pressure air blowers (610-HA-647/648) which supply low pressure air to the cyanide detoxification circuit at a miniumum pressure of 171 kPa.

The plant services are summarized in Table 17.14.

Table 17.14 Plant Services

Plant Services	Units	Value
Compressed Air		
Plant general	Nm^3/h	2.5
Workshops	Nm^3/h	2.50
Instrument air	Nm^3/h	7.00
Instrument air receiver	m^3	3.85
Total compressed air	Nm^3/h	12.0
Air pressure (maximum)	kPa	750
Oxygen Requirement		
Usage	t/day	3.6 – 5.8
Oxygen Supply Pressure Required	kPa	650
PSA plant capacity	t/day	5.00 – 7.2
Low Pressure Air		
Air Supply Rate	**Nm^3/h**	**1500**
Air Supply Pressure Required (Minimum)	kPa	171.4

A process plant block diagram is shown in Figure 17.1 and a schematic of the process flow is shown in Figure 17.2.

Figure 17.1 Plant Layout

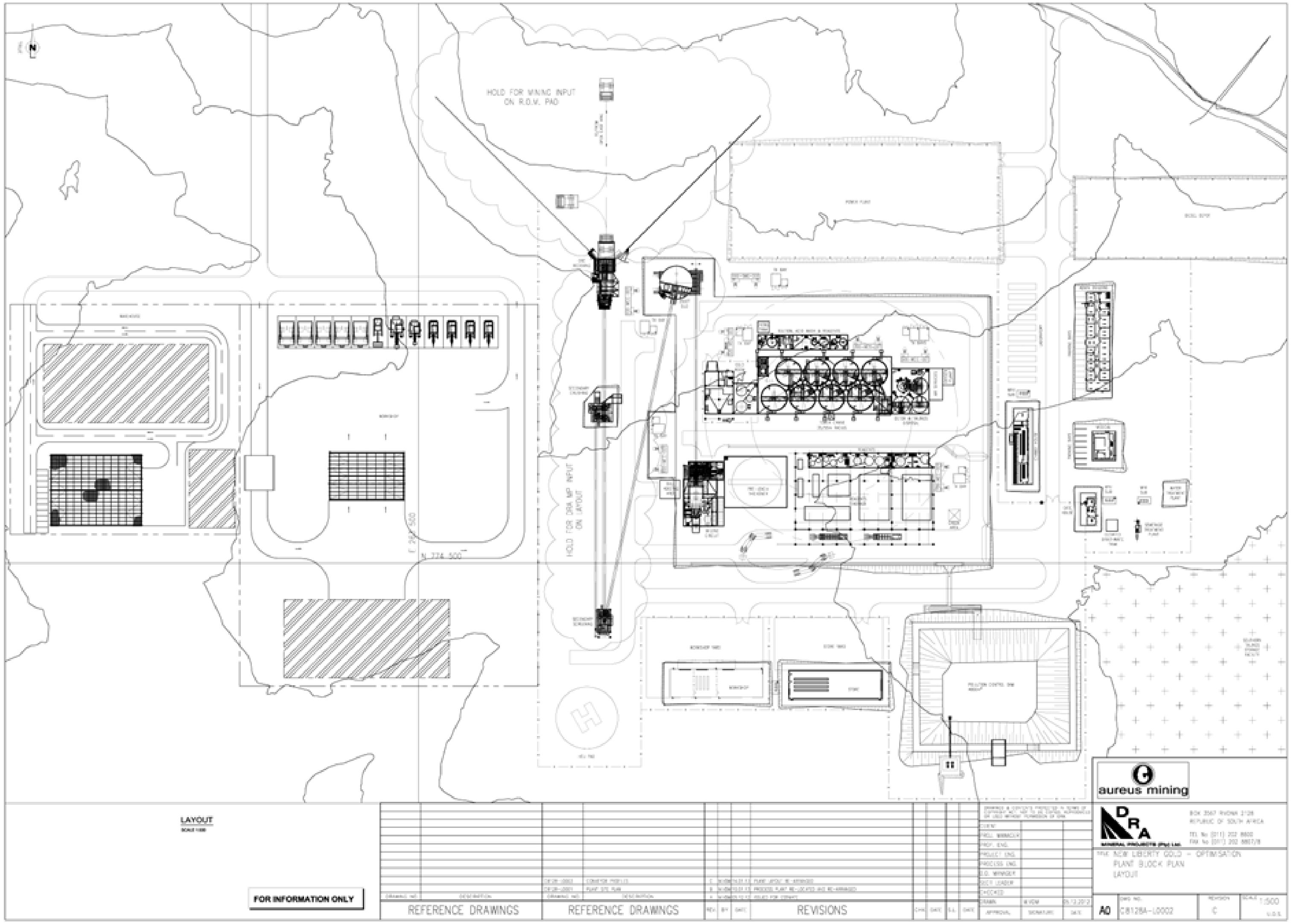
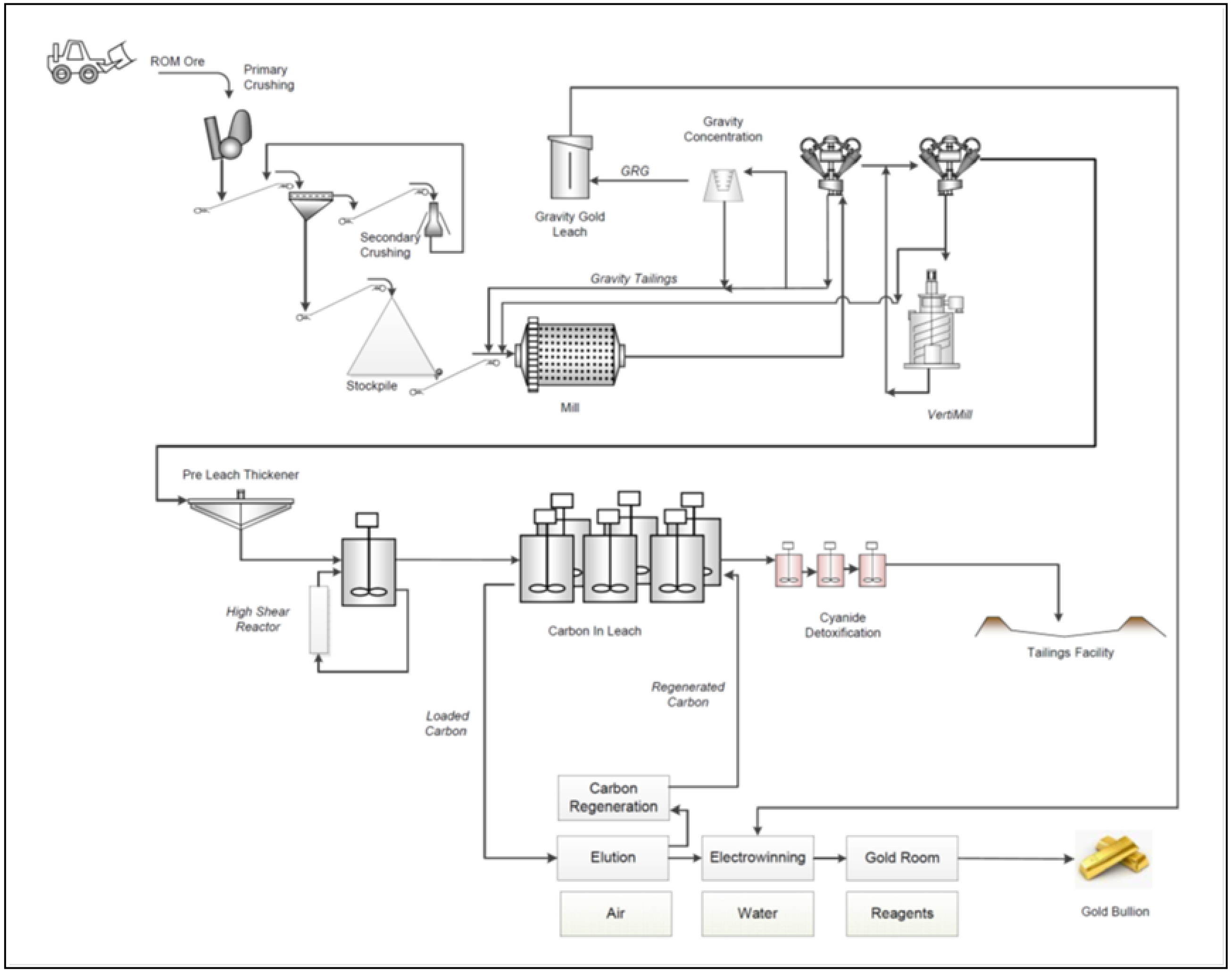


Figure 17.2 High Level Process Flow Diagram



18 PROJECT INFRASTRUCTURE

18.1 Introduction

The Project is located approximately 100 km north–north-west of the Liberian capital, Monrovia. The Freeport of Monrovia, a deep-water part, 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. It can accommodate third generation container ships.

Currently there is approximately 80 km of paved road to the town of Danieltown and then 20 km of laterite road to the Project site. The current road allows for easy access for larger cargo as the project infrastructure grows. Aureus has recently widened and re-graded this laterite road, made improvements to road drainage, and will install three concrete culvert type bridges. Secondary roads on the Bea-MDA licence area, built by Aureus, provide access across the property. Due to the sandy nature of the roads, access is all year round, including during the height of the rainy season.

The Project site is currently an established and existing exploration camp that does not have mining and processing infrastructure. The site has offices, staff accommodation, messing facilities, core storage facilities and is currently serviced with private company phone and internet services via satellite link.

The exploration camp is within the 500 m blast radius of the open pit operations. During the project build and mining pre-strip phase, the use of the camp will be maintained for storage and office facilities with it being subject to evacuation procedures for blasting. New office facilities are to be built as the Project moves from an exploration base to that of project implementation but it is planned for the core storage facilities to remain at the existing exploration camp.

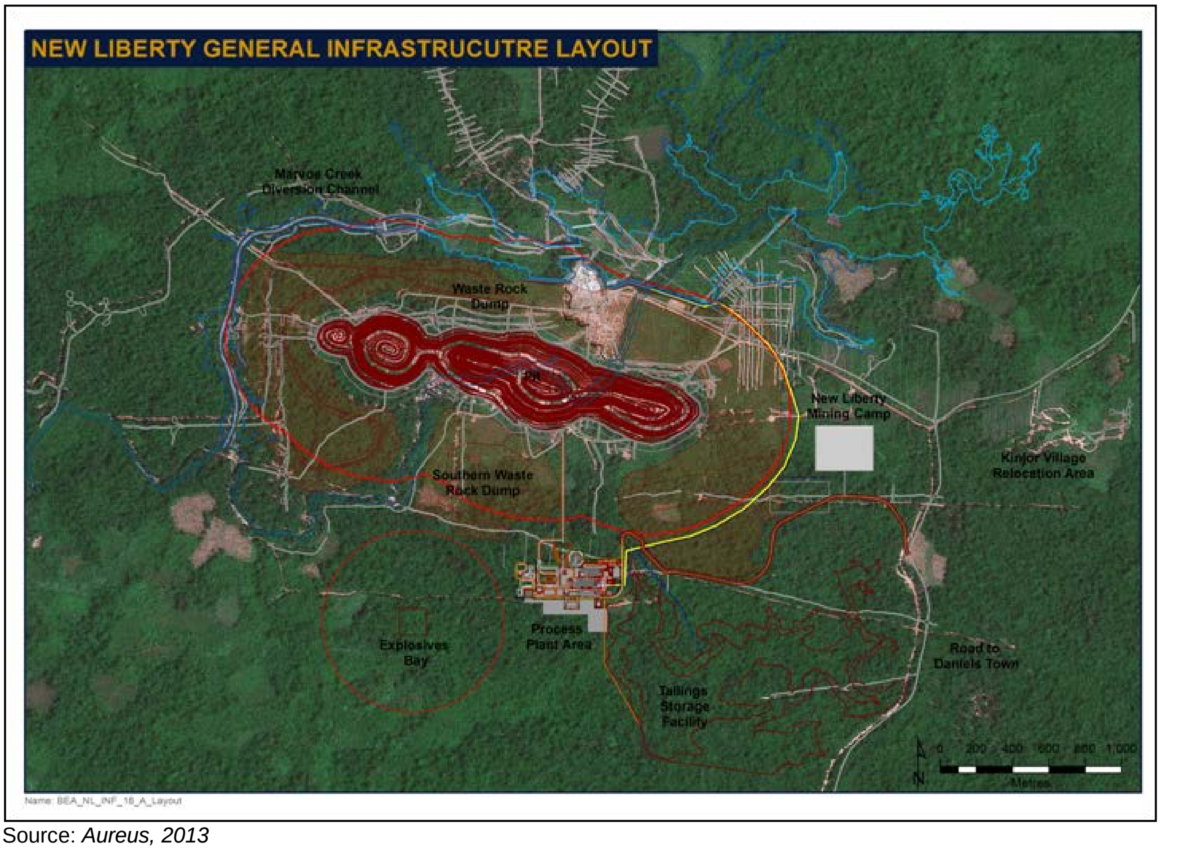
The proposed infrastructure will support the mining, plant and construction operations and can be summarized as follows:

- Mining infrastructure and general infrastructure.
  - Mining equipment workshops, fuel storage and explosives storage.
  - Processing plant, administrative facilities, security, assay laboratory and medical facilities.
  - Water services and waste control
  - Camp accommodation and facilities
  - Security
  - Communications
  - Access roads.
- Power supply and distribution
- Process tailings management - tailings storage facility (TSF)
- Marvoe Creek Diversion Channel (MCDC)

- Waste rock dump.

This section details the facilities that are envisaged for the Project. The site layout below shows the position of the process plant, tailing storage and water storage dams relative to each other and the surrounding topography.

Figure 18.1 General Infrastructure Layout



Source: *Aureus, 2013*

18.2 Mining Infrastructure and General Infrastructure

The mining infrastructure has been designed to provide adequate support for the duration of the life-of-mine and includes the following:

- Main workshop and repair facilities for the maintenance of the mining fleet and major mining equipment.
- Fuel storage for the mining fleet, equipment and the processing plant.
- Explosive storage, which will be located away from the main facilities as per relevant international codes of practice.
- Change house and security office.

Raw materials and other consumables, including, but not limited to, ammonium nitrate and explosive emulsions used in the explosive manufacturing process, will be brought to site by road by external contractors and stored until they are required.

18.2.1 Mining Equipment Workshop

This workshop is situated adjacent to the main warehousing facilities and will be fully equipped with all the facilities to maintain the mining fleet and ancillary equipment. Provision has been made for 6 service bays with enough head clearance to allow the servicing of 100 tonne dump trucks. These service bays will be serviced by a 20 tonne overhead crane and will have suitably engineered flooring (200 mm) to provide for heavy loads. One of the bays will be demarcated for use by tracked vehicles, and excavators and the floor will be suitably strengthened by means of rail sections. The drill rigs will have a dedicated area in the workshop. Other facilities include a hydraulic and hose repair room, tyre repair bay, washing bay, workshop office and ablutions. The fenced area outside the workshop will be suitably compacted and provision made for adequate drainage in order to ensure that heavy vehicles can negotiate the terrain in wet conditions. Parking areas for equipment awaiting service and collection is provided for inside the fenced area.

18.2.2 Fuel Storage Area

Diesel is required to operate the power generators which provide the power to the processing infrastructure and the camp as well as for the mining fleet. The forecast monthly consumption of diesel is approximately 2 million litres.

Fuel storage and the fuelling facility for both diesels and lubricants will be providing by an external contractor, which will include a fire prevention system. The fuel farm will be managed by the external contractor who will be responsible for the following:

- Storage facility
  - Design, construction and commissioning of the storage facility
  - Design, construction and commissioning of loading bay
  - Supply of products from Monrovia to the fuel storage facility at the Project
- Operations
  - Supply of diesel to the storage facility on the mine site
  - On-site offloading of fuels and lubricants
  - Transfer operations for the trucks to the main storage
  - Handling of used diesel and oil and general management of hydrocarbons
- Filtration
  - Filtration solution implementation
  - Filters and filtration equipment maintenance
- Technical support
- HSEQ – to specific guidelines.

The storage facilities will include the following:

- 8 mobile tanks with a capacity of 68 m^3 (544 m^3 storage capacity per depot facility, 1,088 m^3 across the project site)
- Bunded bulk lubes storage
- All civils, including a concrete bund
- Distribution piping and filtration equipment
- Bunded old fuel and oil storage area
- Fire fighting equipment.

Based on the diesel consumption rates, the fuel in the storage facility will be able to run the processing plant and camp for approximately 18 days before refilling (plus 9 days of safety stock). A fleet of trucks will be used by the external contractor to deliver product from the port at Monrovia to the Project site.

18.2.3 Explosives Storage

Bulk explosives will be supplied and stored by an external contractor. An area to the south-west of the pit, which is outside the pit blasting zone, has been demarcated for this purpose. Care has been taken to place all other infrastructure outside a 500 m radius of the explosives storage facility. This explosives storage facility will also be used by the explosives supply contractor to manufacture emulsion explosives. The area will be securely fenced and guarded and provision has been made for adequate lighting at night.

18.3 Processing Plant and Administration Facilities

The processing plant and administration facilities will include the following:

- Access roads within the plant site area
- Plant administration buildings, including, but not limited to, security office, change house, workshop, main administration offices, medical facility, assay laboratory and warehouses.
- Sewerage treatment and disposal
- Water services including of raw water abstraction, potable water and fire water.
- Accommodation
- Security
- Communications

18.3.1 Access Roads Within Processing Plant

The roads within the plant area will be stripped of organic material and compacted, which will facilitate access to the requisite areas within the plant. Drainage ditches and culverts will be created in accordance with the requirements for site drainage.

18.3.2 Plant Administration Buildings

Buildings located in the plant area will consist of security offices, change house, plant mess, process plant equipment workshop, control room, general administration offices, medical facility, assay laboratory, reagent warehouses and spares warehouse.

18.3.2.1 Security Office and Change House

The security office and gatehouse will be located at the main site entrance. The gatehouse consists of the following:

- Protection services office
- Protection services search area
- One Unisex Toilet
- One Turnstile

This gatehouse will control all vehicles in and out of the plant.

The plant ablutions will consist of the following:

- Ablutions for 50 males and 10 females. The plant will operate in shifts. There will not be more than 60 people per shift and the ablutions only need to accommodate one shift at a time.
- Cleaners store with shelving.
- Laundry area to wash overalls.
- Change area to accommodate 110 double lockers for 110 employees. The lockers will be 1,800 mm high × 300 mm wide × 450 mm deep.

There will be a security search zone located next to the change house. The security search zone will provide the security personnel with the means to conduct individual body searches, isolation rooms and general scanning. The main access gate to the plant will have a security office for the control of vehicle access to the plant.

18.3.2.2 Plant Control Room

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

18.3.2.3 Process Plant Equipment Workshop

A suitable workshop with an area of 480 m^2 will be established adjacent to the process plant to enable repair of plant equipment. The workshop will be a steel frame building equipped with a 3 tonne overhead crane and will have bays for servicing light vehicles. The workshop will have separate areas for mechanical and electrical repairs. Provision has been made for oil separation of any water leaving the facility. Offices for supervisory, workshop store, maintenance and planning personnel will be provided in the form of a modular building situated next to the workshop.

18.3.2.4 Administration Buildings

The administration building will be of a single-storey prefabricated panel construction. The building will include general areas for engineering, administration personnel and offices for the general manager, mining manager, plant superintendent, administration superintendent, chief geologist, plant maintenance superintendent and chief security officer.

18.3.2.5 Assay Laboratory

The assay laboratory will be in the form of a containerized lab supplied to the project and managed by an independent third-party laboratory service provider. This laboratory will conduct all of the onsite test work for the processing plant and the grade control.

18.3.2.6 Medical Facilities

Aureus will provide an equipped medical facility, which will allow for the treatment of any injuries during the construction and operational phases, as well as treatment of sick personnel.

18.3.3 Sewerage Treatment and Disposal

Sewerage from the plant and the Aureus camp will be treated separately with dedicated treatment plants. The sewerage plant will treat the water in accordance with South African DWEA General Limits for the release of treated water into the environment.

The sewerage plant will have covered sludge handling drying beds. The sewerage plant has been sized based on potable water requirements during the construction phase.

18.3.4 Water Services

The project is located in a net water surplus climate. To minimize the volume of non-contact surface rainfall run-off reporting to the TSF or the open pits, water diversion channels and ditches will be constructed. The preliminary water balance developed by DRA for the site indicates that there will be a surplus of contact water that will require discharge to the environment. This surplus contact water will be discharged to the Tailings Storage Facility.

18.3.4.1 Water Supply Dam

The design of this facility is based on meeting or exceeding agreed design criteria which comply with World Bank and other international standards. It has been incorporated in the design of the MCDC, under Epochs scope of work.

A water balance was developed and was used as the basis for sizing the water storage dam and the raw water requirements. Raw water stored in the water supply dam will be pumped to the process plant for make-up operations during the plant start-up and during periods where the return water from the tailings storage facility is insufficient to meet the requirements of the plant.

18.3.4.2 Potable Water Distribution

Potable water will be provided to the mine accommodation camp and plant area via dedicated potable water treatment plants. Raw water will be supplied to the potable water treatment plant from the plant raw water tank and in turn through a pipe line from the raw water dam. This plant is sized for a maximum of 100 persons per day

The plant for the mine accommodation camp is designed to supply sufficient water for up to 800 people per day and is fed from boreholes at the Village.

Potable water will be reticulated to all areas of the plant including a safety shower header tank.

In respect of potable water for human consumption, dedicated potable water supply boreholes will be drilled. This will form part of the open pit dewatering strategy. Current groundwater quality indicates that the resource is suitable for potable use according to the WHO Guidelines for Human Consumption.

18.3.4.3 Fire Water Distribution

The fire water system is a dual power system that can use electricity and diesel. The electric powered pump is used in the event of a fire and the diesel pump is used in the event of a fire where electrical supply is unavailable – the pump is also available as a standby unit. A jockey pump will be provided to maintain the pressure in the fire water header during normal plant operation.

The fire water system consists of a fire water distribution system with hydrants strategically positioned within or in close proximity to the plant site, ancillary buildings and the mine accommodation camp areas. In addition, portable fire extinguishers will be housed within the process plant facilities and the mine accommodation camp.

18.3.5 Accommodation

The contractor that will be performing the earthworks for the TSF, the MCDC, and the mine pre-stripping will provide a camp suitable for housing up to 800 persons during both the construction and the production phases.

The mine camp will be constructed to house up to approximately 750 individuals during the construction phase and will then be scaled back to be able to house approximately 75 individuals for the production phase. This camp includes the following infrastructure:

- Kitchen and camp dining room
- Entertainment area
- Laundry
- Potable water plant

- Sewerage disposal plant

The mine accommodation camp is designed as a combination of a modular system for the kitchens and sevices buildings, wooden cabins for senior accommodation from local timber, and tented accomodation for the temporary construction crews, which allows configuration in the most appropriate format for use. If required, the camp could provide additional office space.

18.3.6 Security

The plant site will be enclosed within a security fence. Access to the plant area will be via gates located on access roads to the site. Additional fencing will be provided for further safety and security within process plant areas, such as power plant, fuel storage, gold room area, transformers and substations, as required.

CCTV cameras will be installed at strategic locations in the plant for surveillance purposes. The cameras will be integrated with the plant's overall network, which will be the responsibility of the security manager.

18.3.7 Communications

The New Liberty exploration camp site is currently serviced with a private company phone and internet services via satellite link—this has already had an upgrade for additional bandwdth to service the construction activities.

A dedicated satellite system will be installed in the new plant area for operations.

18.3.8 Access Road to Site

The Project site is 100 km from Monrovia, 80 km of which is on a tarmac road with the remaining 20 km on a new laterite road from Danieltown. Aureus has widened and re-graded the laterite road in order that the transportation of the large volume of infrastructure required to be transported from the port at Monrovia to the Project site can be undertaken. Three concrete culverts and 22 cross drains will be constructed to manage rain water.

18.4 Power Supply and Distribution

18.4.1 Power Supply

An external contractor will provide the power generating capability at the Project site, which will be used for both the processing plant and the mine camp. Liberia has a limited power grid in Monrovia which does not serve the local district around Project, and as such, Aureus is responsible for generating its own power.

The external contractor will provide an 11 kV, 9 MW, diesel driven, build, own, operate and transfer (BOOT) power station at the Project. The generators will be housed in 12 m shipping containers.

The power plant has been designed to be self-sufficient and shall have its own fence line to allow for potential maintenance and servicing agreements to be executed with minimal disruption to the main processing facility. 525V motor control centres for auxiliary services shall also form part of the power plant infrastructure. The 11kV feeds from the generators will be run via cables to the plant main 11kV substation. Syncronization will be done on the main plant Substation, with control and protection of the supplies done by the power plant contractor Real estate has been allowed for in respect of the future inclusion of additional generator sets for power plant expansion should this be required.

The supply of diesel to the power plant shall be via the bulk diesel storage facility located adjacent to the power plant fence line. The bulk diesel storage facility shall be built and operated by the diesel supply contractor, with diesel fuel being free-issued to the power plant contractor. A diesel day tank has been allowed for within the power plant area.

Other power plant infrastructure includes: a local control room, a workshop, an oil change station and a transformer for supply of power to auxiliary loads. Power factor correction equipment is not required as no connection to a utility is being made and diesel consumption is not dependant on the power factor.

The workings for the 5-genset power station configuration can be summarized as follows:

Genset Units	
Prime power output	1,965 kW (m)
Prime power output	1,827 kW (e) alternator terminals
Unit prime power output	1,800 kW (e) alternator terminals
Output	
Average demand	7,505 kW (e)
Guaranteed power MD	7,200 kW (e) (99.5% availability)
Installed units	6
Installed capacity	10,800 kW (e) at 11kV busbar
1h overload capacity	11,880 kW (e) at 11kV busbar
Fuel Consumption	
g/kWh (m)	196 (mechanical at crankshaft)
Alternator efficiency	96.0%
g/kWh (e)	204.2 (electrical at alternator terminals)
Transformer losses	0% (alternators at 11kV)

Fuel consumption	204.2 g/kWh (at 11kV)
Fuel consumption	0.234 L/kWh

Power Levels with 6 Gensets Installed

11,880 kW	1 hour overload capability when all gensets available
10,880 kW	installed prime power capacity
7,200 kW	maximum continuous running capacity of 4 gensets (normal operation)
8,100 kW	continuous average load capability of 5 gensets (75% of 9,000 kW)
5,940 kW	capacity of 3 remaining gensets if 1 genset trips during normal genset operations (3 x 110%)
7,200 kW	guaranteed power output for 99.5% of the time
6,424 kW	annual average load (=4,021,826 kWh pm)

There may be further opportunities in the future to optimize the power generation capability.

18.4.2 Power Distribution

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

Medium voltage electrical power shall be distributed throughout the main processing facility via buried 11,000V XLPE cable. Low voltage electrical power distribution shall be distributed to loads (motors, distribution panels, light fittings etc.) via 525V PVC Cable, which shall generally be run above ground on cable ladder, or buried where the use of cable ladder is not appropriate.

The main electrical power consumer shall be the 11,000V 3.25MW ball mill and 950kW Vertimill motors, which shall be supplied from the plant 11,000V substation. In order to mitigate against the substantial increase in electrical current normally associated with motor starting events, the mill motor shall be of the wound rotor type and shall utilize a liquid resistance starter during the mill starting sequence.

Motor control centres shall nominally operate at 525V, 50Hz, for supply of electrical power to low voltage motors. Other lighting and small power loads shall be rated for 380V (3 phase) and 220V (single phase), 50Hz.

Power to the mining workshop infrastructure and the New Liberty accommodation camp shall be supplied from the plant 11,000V substation via approximately 2.5 km of overhead line.

Pit dewatering and raw water intale pumps will be diesel powered.

Emergency electrical power has not been provided for within the main plant area but has been allowed for in the accommodation camp where the diesel generator used during the construction phase shall be retained and act as the emergency backup generator.

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

18.5 Process Tailings Management – Tailings Storage Facility

18.5.1 Introduction and Design Criteria

As a part of the construction of the project, Epoch Resources (Pty) Ltd (Epoch) has been appointed to undertake an Optimization Study and Detailed Design of the Tailings Storage Facility (TSF) associated with the project.

Epoch's brief for the optimization study was to:

- Review the Golder's Feasibility StudyTSF option;
- Evaluate the terrain for alternative sites south of the open pit;
- Develop a proposed design philosophy for the selected TSF option; and
- Compile a BoQ and capital cost for the selected TSF option

There are no Liberian guidelines related to the design, operation and closure of tailings storage facilities and water diversion systems. In the absence local guidelines, the following internationally recognized publications and industry standards are used to develop site specific design criteria:

- International Financial Corporation guidelines (IFC, 2007);
- International Committee on Large Dams (ICOLD) - Various Manuals and Bulletins;
- Canadian Dam Association (CDA) - Dam Safety Guidelines (CDA, 2007);
- Mining Association of Canada (MAC) - A Guide to the Management of Tailings Facilities (MAC, 1998), and Developing an Operations, Maintenance and Surveillance Manual for Tailings and Water Management Facilities (MAC, 2003);
- Australian Committee on Large Dams (ANCOLD) (1999); "Guidelines on Tailings Dam Design, Construction and Operation";
- South African Committee on Large Dams;
- Department of Mines and Petroleum, Western Australia (1999); "Guidelines on Safe Design and Operating Standards for Tailings Storage";
- SANS Code of Practice for Mine Residue Deposits (SANS 10286); and
- The Cyanide Code Standard of Practice.

In addition to the above, cognisance of the following measures have been accounted for in the TSF design:

- DRA shall design the processing plant and detox circuit to conform to the Cyanide Code - Standard of Practice 4.4 by implementing measures to limit the concentration of WAD cyanide in the TSF to a maximum of 50 mg/l; and
- The TSF shall be designed to conform to the:
    - Cyanide Code Standard of Practice 4.3 and 4.5;
    - The IFC Emissions and Effluent Guidelines of the Environmental, Health, and Safety Guidelines – MINING. This will be achieved by implementing measures to limit discharges to surface waters not exceeding 0.5 mg/l WAD cyanide and 0.1 mg/l of Arsenic (95% of the operating time for IFC standard).

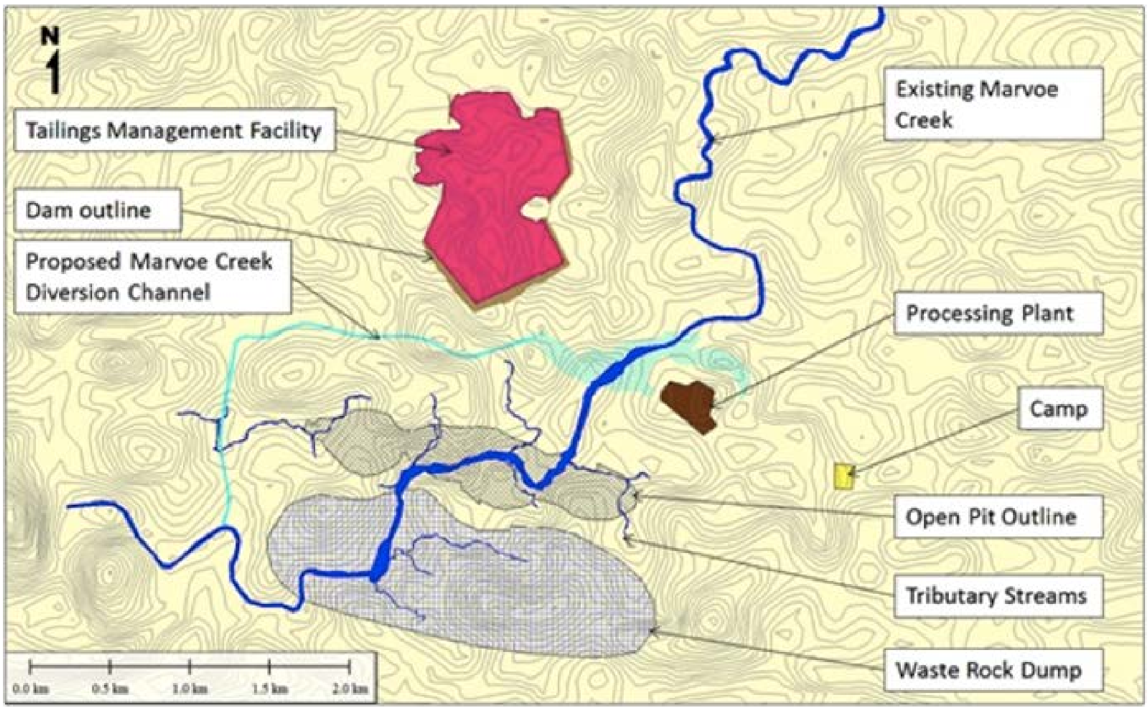
The mining process is expected generate a total of approximately 9.4 Mt of tailings over the life-of-mine. The dry density of the deposited tailings has been estimated at 1.36 t/m^3, requiring storage of a total of 6.9M-m^3 of tailings. The design criteria and assumptions developed for the TSF are as summarized below:

Parameter	**Design criteria**
Life-of-mine	8.5 years
Tailings production rate	1.1 Mt/annum
Specific gravity of tailings	2.8
Void ratio of tailings (assumed)	1.0-1.3
Dry density of deposited tailings	1.36 t/m^3
Slurry density	1.36 t/m^3
Tailings PSD	90% passing 75 micron
Percentage solids	41%
Basin lining	unlined
TSF dams slope stability	1.36 t/m^3
Percentage solids	41%

18.5.2 TSF Site Selection and Location

A Feasibility study was undertaken by Golders, which identified a preferred TSF site and associated design. The study identified a site north of the open pit as the preferred TSF site, refer to Figure 18.2.

Figure 18.2 TSF Layout as per Golder's Feasibility Study



Source: *Epoch, 2013*

During the optimization study various aspects of the mine layout changed in terms of location of infrastructure, including the process plant, waste rock dumps and TSF. TSF sites located in the south of the site and open pit had been considered during the Feasibility Study but were considered less favourable options, with some sites falling outside of the mining lease area. It should be noted that the survey information used during the Feasibility Study was not of a sufficient level of accuracy.

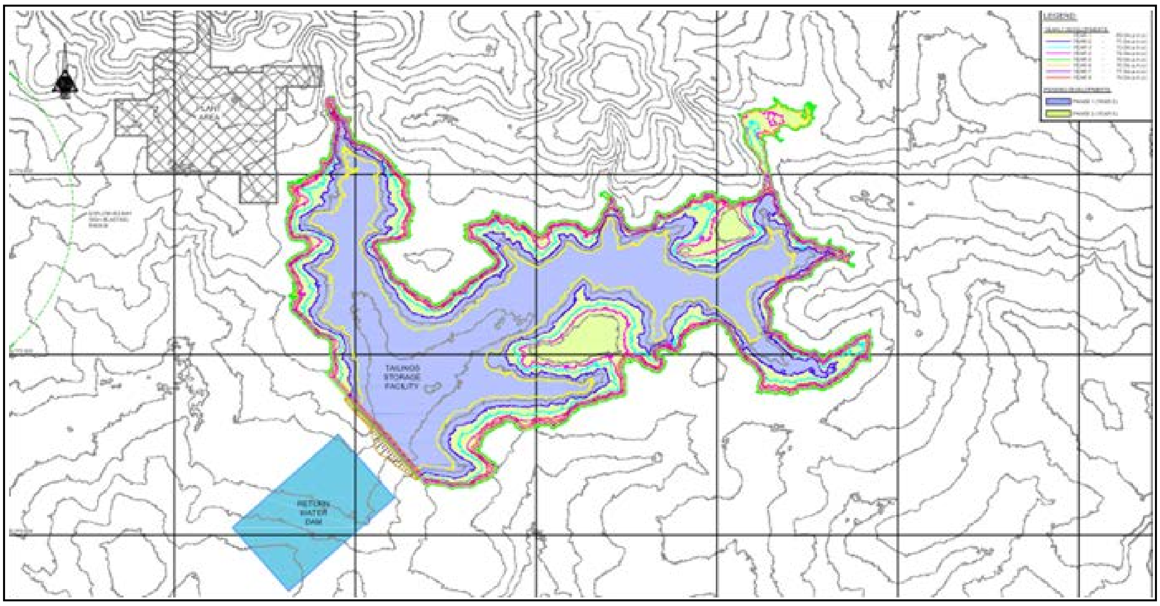
For the optimization study, an airbourne LiDAR survey dataset of the mine site was made available, and due to this increased level of accuracy and the changes in the mine infrastructure layout, a more favourable TSF site from a layout, proximity to plant, risk and capital cost perspective compared to the Feasibility TSF site was identified in the south of the mine lease area. Figure 18.3 shows the location and overall development of the southern TSF site over the life-of-mine (LOM).

The main advantages this site offered over the Feasibility Study site are:

- Valley dam TSF with a single embankment/starter wall of significant less earth fill volumes. The tailings dam is constructed as a self raising upstream dam in year 2 onwards as the rate of rise is below 2.5 m/yr and typically around 1.3 m/yr, refer to Figure18.4;
- Close proximity to the plant, thus less pumping distance and slurry piping;
- TSF is located downstream of the open pit and creek diversion, resulting in a reduced potential risk profile should a TSF failure occur; and

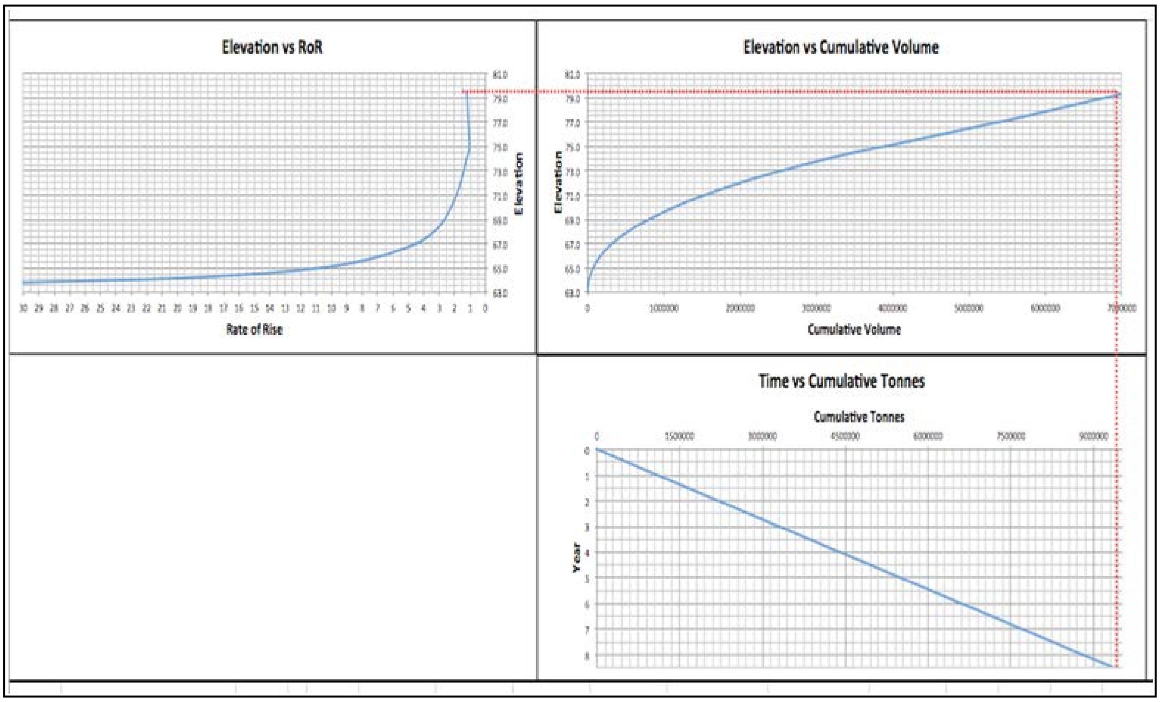
- The slurry pipeline does not cross the creek diversion, thus the environmental risk due to potential pipe bursts and leakages into the creek diversion are nullified.

Figure 18.3 Southern TSF Site Following the Optimization Study



Source: *Epoch, 2013*

Figure 18.4 Stage Capacity Curve for the Southern TSF Site Following the Optimization Study



18.5.3 TSF Design

The TSF has been designed to accommodate a volumetric storage capacity of 9.4 million dry tonnes over an 8.5 year LOM. The general arrangement of the TSF is shown in Figure 18.3. The key design features of the TSF is it is a valley single embankment dam with a compacted earth embankment wall of 40,000 m^3 of total fill catering for the first two years of operations when the rate of rise is greater than 2.5 m/year. Subsequent development of, and raising of the TSF embankment is by means of the upstream self-raising paddock system. The TSF has a total footprint area of 78 ha, a maximum elevation of 79.5 m.a.m.s.l (a maximum height of 16.5 m) and an average rate of rise of ±1.3 m/year above the elevation of the earth embankment.

18.5.4 Stage Capacity Curve

The stage capacity curve for the TSF, reflecting the relationship between tailings elevation, rate of rise, storage volume, footprint area, cumulative tonnage and time is shown in Figure 18.4. The initial 7.0 m high compacted earth starter embankment corresponds to a crest wall elevation of 70.0 m.a.m.s.l, at which point the average rate of rise of the TSF decreases to below 2.5 m/year, in year 2 of operation. The rate of rise continues to decrease with time and TSF height, until a rate of rise of 1.2 m/year is reached at closure. The indicated TD footprint has the potential to accept additional tailings beyond year 8.5 of operation. The staged development of the TSF is shown in Figure 18.4.

18.5.5 TSF Dam Preparatory Works

The preparatory works associated with the TSF comprise the following:

- Topsoil stripping to a depth of 300 mm over the entire TSF footprint;
- A box cut to a depth of 500 mm beneath the starter wall embankment;
- A vertical curtain drain located within the compacted earth embankment wall;
- A compacted clay key below the starter wall embankment with the following dimensions:
  - 5.0 m deep;
  - 4.0 m wide base; and
  - 1V:1H side slopes.
- A compacted earth starter wall embankment, constructed with suitably sourced material from within the immediate vicinity of the TSF, with the following dimensions:
  - 7.0 m high (i.e. crest elevation of 70.0 m.a.m.s.l);
  - 6.0 m wide crest;
  - 1V:2H internal side slope; and
  - 1V:3H external side slope.
- The self-raising portion of the TSF is battered at an overall side slope of 1V:3H;

- A 5.0 m wide elevated toe drain located on an elevated platform. The toe drain is positioned along the inside toe of the starter wall embankment and comprises the following:
  - 160 ND slotted HDPE drainex pipe;
  - Suitably graded filter sand;
  - 6.7 mm stone;
  - 19 mm stone; and
  - Non-woven geo-fabric.
- A 3.0 m wide NGL toe drain extending the length of the inside toe of the starter wall embankment above an elevation of 65.0 m.a.m.s.l. The toe drain is comprised of the following:
  - 160 ND slotted HDPE drainex pipe;
  - Suitably graded filter sand;
  - 6.7 mm stone;
  - 19 mm stone; and
  - Non-woven geo-fabric.
- 160 ND non-slotted HDPE drainex pipes, spaced at 50 m intervals and positioned along the perimeter of the elevated and NGL toe drains. Seepage water emanating from the drains is collected and channelled into the solution trench;
- A solution trench around the TSF from which water is directed towards collection sump. The trapezoidal solution trench has the following dimensions:
  - 1.0 m deep;
  - 1.0 m wide base; and
  - 1V:1.5H side slopes.
- An energy dissipater for the collection of supernatant water from the penstock outfall pipe as well as seepage water from the toe drains and curtain drain. The water is pumped back to the plant as make up water;
- A 1.0 m high catchment paddock wall extending the perimeter of the TD;
- A storm water diversion channel with its associated cut-to-fill berm wall with the following dimensions:
  - 1.0 m deep;
  - 4.0 m wide base; and
  - 1V:1.5H side slopes.
- A seepage interception drain downstream of the TD to an average depth of 5.0 m below NGL;
- A buried 900 ND Class 150D spigot-socket precast concrete penstock pipeline comprising single intermediate intakes and a double final vertical 750 ND precast concrete penstock ring inlets;

- A possible Return Water Dam/Decant sump from where water is pumped back to the process plant for re-use or discharged downstream under surplus water conditions; and
- A slurry deposition pipeline along the length of the TSF starter wall embankment.

The specified size of the penstock pipeline and the slurry delivery pipeline has been based on preliminary design calculations and should be re-evaluated during the next phase of the project.

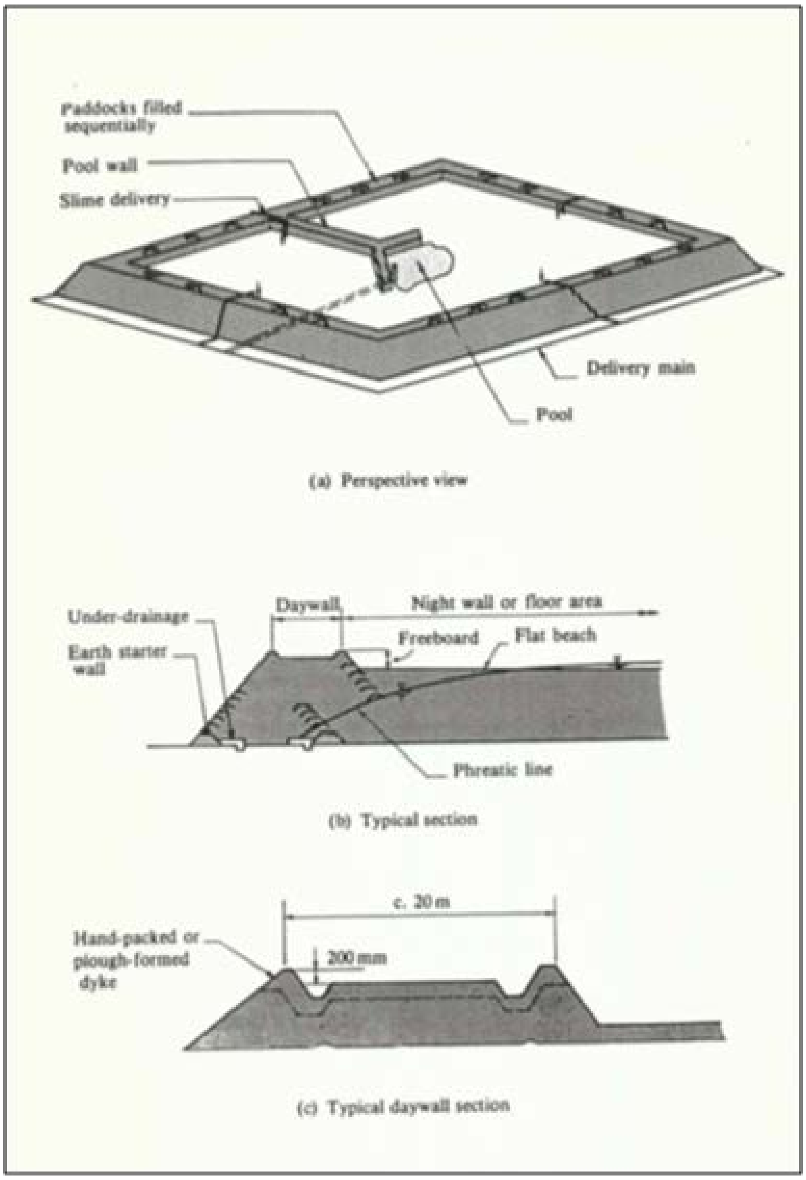
18.5.6 Tailings Dam Depositional and Operational Methodology

The proposed depositional methodology for the TSF is by means of the “self-raising upstream day-wall” system above the crest of the starter wall embankment. During the initial commissioning stage of the project, it remains crucial that the tailings not be deposited directly onto the elevated toe drain as this would lead to erosion and possible blinding of the toe drain system. Tailings shall be deposited into the basin of the TSF by means of an open-ended deposition technique whereby flexible hosing, positioned at approximately 30 m interval off-takes, is utilised. Prior to the tailings reaching the elevated toe drain, coarse tailings are to be used to cover the drains.

Open-ended deposition shall continue above the covered elevated toe drain, until the tailings elevation is within 1.0 m from the elevation of the starter wall embankment. Thereafter, the forming of 0.3 m high tailings paddocks, extending the length of the inner perimeter of the initial containment embankment, should commence so as to facilitate the shift in depositional methodology from an open-ended depositional strategy to a self-raising paddock system. Figure 18.5 illustrates the typical construction of a day-wall paddock system.

Supernatant and storm water collected on the TSF shall be decanted through vertical penstock inlets and a buried penstock pipeline to the enery dissipator/collection sump form where is shall be pumpsed back to the plant as make up water or discharged. During commissioning and initial development of the TSF, decanting occurs through single temporary/intermediate penstock inlets located along the migrating path of the pool from the starter wall embankment up towards the final location of the pool. It is at this final location where both the double penstock inlets and the pool wall/wing walls shall be situated. The intermediate inlets are progressively sealed as the pool is relocated to an adjacent upstream inlet.

Figure 18.5 Typical Construction of a Day-Wall Paddock System



18.5.7 Work in Progress or Planned

Geotechnical investigative work pertaining to the TSF and the tailings material itself has been initiated, but the outcome of this work is only expected at the end of June 2013. Currently based on the Feasibility Study, residual soil (laterite/saprolite) is the most common and typical soil encountered in the area and would thus be the material used in the construction of the of the TSF facilities, typically sourced from within the basin area and immediate surroundings.

Planned scope of work and activities going forward:

- On receipt of the geotechincal laboratory and site investigation reports, the TSF design shall be verified, finalized and all detail design drawings shall be produced;
- The design shall be verified against the Geochemical test results to confirm whether lining of the TSF is warranted and/or treatment of the surplus discharge water;
- Seepage assessment;
- Slope stability analysis;
- Design location and size of under drains;
- Slurry delivery and distribution piping design;
- Storm Water control trenches assessment around TSF;
- Update BoQ;
- Final design report;
- Operational manual; and
- Site visit(s) to confirm design intent has been met.

18.6 Marvoe Creek Diversion

18.6.1 Background

The Marvoe Creek is the dominant drainage feature in the Project area. It is fed by numerous small tributaries and is itself a tributary of the Mafa River, which lies 5 km south-west of the Project. The creek diagonally bisects the Project site and the alignment is such that it passes through the proposed open pit and waste dump sites. As a result, a permanent diversion channel is planned to route Marvoe Creek around the open pit and the waste dump. The drainage area of the creek upstream of the proposed diversion is approximately 109 km^2. Where it passes through the Project site, the Marvoe Creek is approximately 30 m wide with a mild slope (approximately 0.01-0.04%).

18.6.2 Guidelines on Safety in Relation to Flood for Dams

For this study we adopted the SANCOLD Safety Evaluations of Dams, Report No. 4, Guidelines on Safety in Relation to Flood, published by the South African National Committee on Large Dams, December 1991.

18.6.2.1 Classification and Categorization of the Proposed Dam

The proposed dams have a total storage capacity of approximately 11 million cubic metres, of which 7 million cubic metres is flood storage and a maximum vertical height of 11 metres; therefore both are classified as small size dams.

The following information is available to determine the hazard rating of the dam:

- The design life of the dam will exceed the design life of the mine, which is 8 years.

- The dam will be designed with a design life of 100 years.
- The mine pit is situated downstream of the dams, but will be protected by means of a waste-rock dump placed between the pit and the dams.
- A dam failure after mine closure will be into the pit, which will collect sediment and attenuate the flood.
- Both the spillway and the cutting between the two dams can be lowered to drop the full supply level in the dams with mine closure. This will reduce the long-term risks significantly.

The adopted hazard rating of the dam is low; this means that the potential loss of life is zero. Although there is a significant potential in economic loss to mining production should the dams fail when the waste-rock dump is still at a low level, this risk will reduces as the waste rock dumps progress. The dams are therefore category I dams of small size and have a low hazard.

18.6.2.2 Requirement in Respect of the Spillway Design

The requirements in respect of the spillway are dependent on the dam size and its hazard rating. In this instance the Recommended Design Flood (RDF) is the 100 year flood event and the Safety Evaluation Flood (SEF) is the Regional Maximum Flood (RMF). The Recommended Design Flood (RDF) must pass safely through the spillway with available freeboard and the Safety Evaluation Flood (SEF) must pass safely through the spillway with zero freeboard available.

18.6.3 Deterministic Flood Evaluation – (Rational Method)

The Rational Method was used to calculate the flood peak – frequency relationships for the Marvoe Creek. The Arial Reduction Factors (ARF) was used as recommended by Prof. D.C Midgley in the HRU 1/72 report, to accommodate the phenomena of convection storms which have high intensities but cover relative small areas.

The recommended flood peak – frequency relationships are listed in Table 18.1 below.

Table 18.1 Flood Peak and Volume Estimation Results for the Rational Method

Return Period (Years)	Marvoe Creek Flood Peak (m^3/s)	Marvoe Creek Flood Volume (Million m^3)
2	91	4.95
5	135	7.34
10	176	9.59
20	220	12.01
50	284	15.51
100	342	18.67
200	409	22.29

18.6.4 Regional Maximum Flood Evaluation

The maximum observed floods since 1850 in West Africa were plotted with their catchment areas on the “x”- axis and the flood peak on the “y” - axis. The following countries were included in the evaluation; Liberia, Ivory Coast, Mali, Benin, Ghana, Guinee, Sierra Leone, Togo and Burkina Faso.

Only two observations were done in Liberia, but seven were done in Sierra Leone. The largest flood recorded in Liberia was in the Mano River at Mano Mine. The catchment area of the river at the observation point is 5 540 km^2, the maximum flood peak recorded was 1 610 m^3/s, with a corresponding “K” value of 3.6.

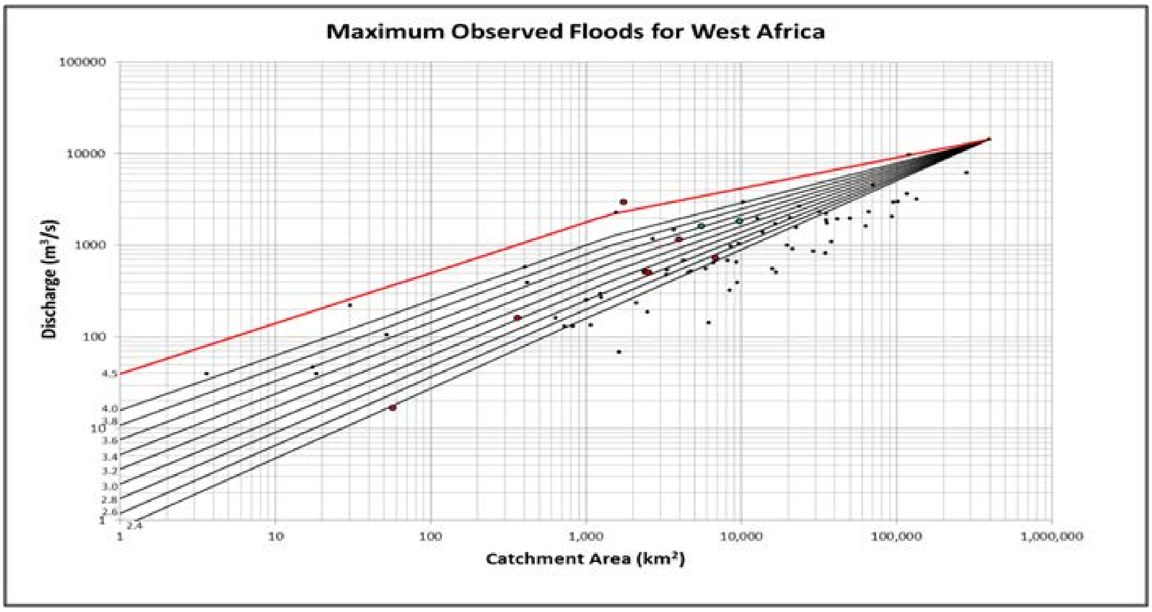
In neighbouring Sierra Leone the highest recorded flood had a “K” value of 4.68, which is also the highest in West Africa. Of the total of seventy-four floods evaluated, eight floods (11%) had a “K” value of equal or greater than 4.0, and two (3%) a “K” value greater than 4.5.

Since Liberia is one of the countries in West Africa to receive the highest annual rainfall, we adopted a “K” value of 4.5 for the estimation of the Regional Maximum Flood.

The calculated Regional Maximum Flood (RMF) for the Marvoe Creek is 525 m^3/s. If plotted on the same graph as the flood peak – frequency relationships from the Rational Method, the selected Regional Maximum Flood has an approximate return period of 650 years.

Figure 18.6 indicates the maximum observed floods in West Africa, the red line is the line associated with a “K” value of 4.5. The red dots represents floods observed in Sierra Leone and the green dots floods observed in Liberia.

Figure 18.6 Maximum Observed Flood in West Africa



18.6.5 Reservoir Routing

The Marvoe Creek diversion system will have two flood control dams / reservoirs, referred to as "Dam 1" and "Dam 2" on the drawings. "Dam 1", is the first dam that will be constructed in the Marvoe Creek and "Dam 2", will be constructed in a tributary of the Marvoe Creek. A trench will be cut between the two dams to ensure "Dam 1" overflows into "Dam 2".

For the purpose of routing floods through the dam, we assume that the dams will be at full supply level by the time the flood wave arrives. The full supply level (FSL) of the dam is 70.00 m.a.s.l. (based on the local benchmark on site). The invert level of the cutting between the two dams is 2 metres below FSL, at 68.00 m.a.s.l. The expected water level difference between the two dams during the Regional Maximum Flood event (RMF) is 200 mm based on the energy principal. For the purpose of flood routing we assume the water level in both dams to be the same during any flood.

The proposed dams have a total storage capacity of approximately 11 million cubic metres, of which 7 million cubic metres is flood storage and a maximum vertical height of 11 metres. The flood storage capacity relates to the temporal storage between full supply level (FSL) at 70.00 m.a.s.l. and non-overspill crest level (NOC) at 73.00 m.a.s.l.

Reservoir Routing was carried out using the level pool routing method. The attenuated flood peaks are tabulated below in Table 18.2.

Table 18.2 Expected Flood Peak Reductions Due to Flood Attenuation

Flood Event	Peak Inflow (m^3/s)	Peak Outflow (m^3/s)	Percentage Reduction (%)	Energy Head Required (m)
2 year flood	91	72	21	0.85
100 year flood	342	282	18	2.05
RMF	525	441	16	2.71

18.6.6 Hydraulics for Diversion Channel

18.6.6.1 Design Philosophy

The design philosophy of the diversion channel is simply to mimic the existing natural environment the Marvoe Creek functioned within before the existence of the mine. It is intended to create a system that will have an inner main channel with a floodplain wherein an ecosystem can develop over time.

The design has two main objectives namely; to safely and cost effectively divert the Marvoe Creek around the proposed pit and secondly to do it in such a manner that its impact on the local environment is limited.

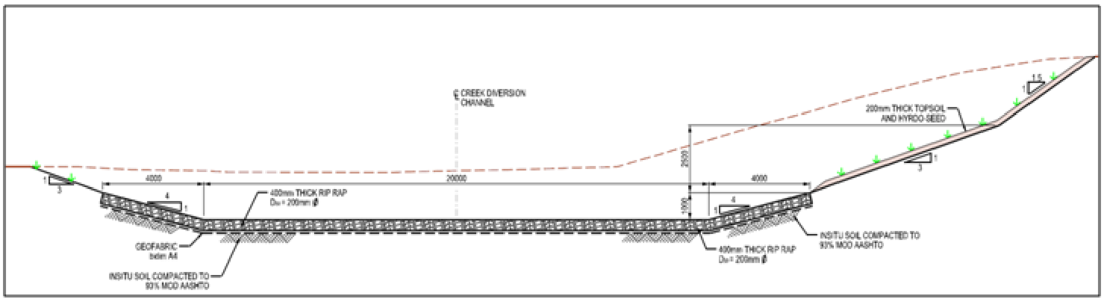
In terms of cost saving the approach was to design a system with the smallest possible footprint, least excavation requirements and require minimal scour protection. This was to be done within the confines of good hydraulic design principles, which includes large radius curves to avoid standing waves and maintaining sub-critical flow conditions with Froude numbers close to critical conditions. The reason for maintaining sub-critical flow conditions with Froude numbers close to critical conditions is to find a hydraulic efficient section that will produce a stable water surface profile and that can accommodate directional changes without the formation of standing waves or hydraulic jumps.

The bush clearing should also be limited into the flood plain, this will not only save costs but will be beneficial in creating a stable floodplain environment that can resist scour through providing flow resistance in the form of established vegetation.

18.6.6.2 Description of the Typical Cross-section

The typical cross-section has an inner main channel which has a 400 mm thick rip-rap lining underlay by a geotextile to act as a filter medium. The main channel has a bottom width of 20 metres, a depth of 1 metre and side slopes of 1:4 (V:H). The main channel was designed to be on average 1 metre below existing ground level and follow the average longitudinal ground slopes in general. Where the main channel passes though cut areas, excavations are extended at a slope of 1:3 to 3.5 metres above main channel invert level and thereafter a side slopes of 1:1,5. The reason for the 1:3 side slopes is that on the average flow depth of the 100 year flood event is 3.5 metres.

Figure 18.7 Typical Diversion Channel Cross-section

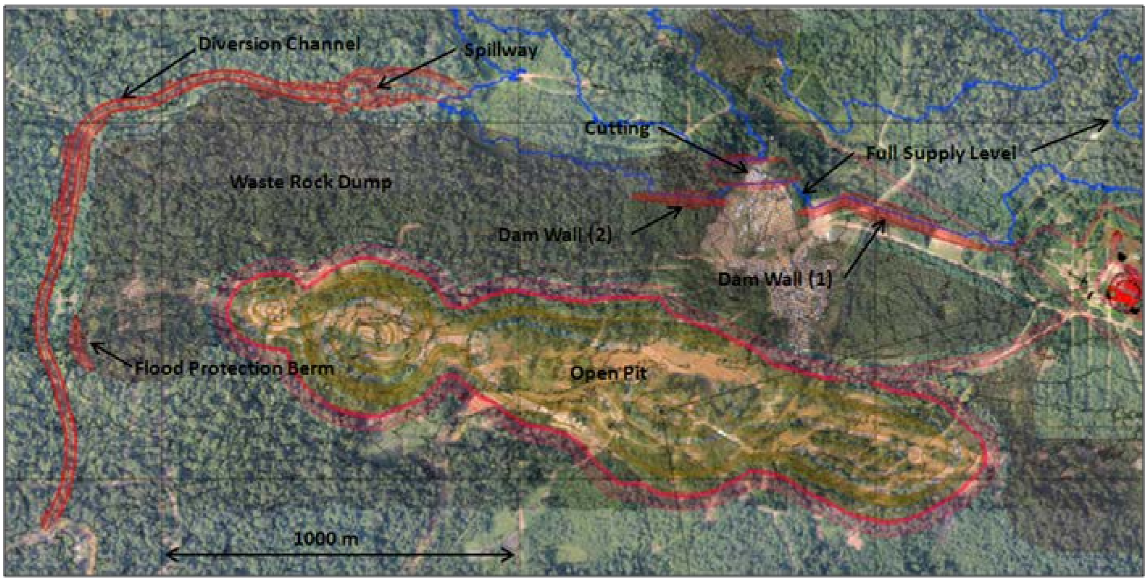


Source: *Epoch, 2013*

18.6.6.3 Typical Layout and Three-Dimensional Views of the Proposed System

The general arrangement of the Marvoe Creek diversion system is shown in the figure below, and it consists of two flood control dams with a cutting connecting the two dams, a By-wash spillway, diversion channel and flood control berms. The figure in this section is indicative with the purpose to provide the reader with an understanding of the system.

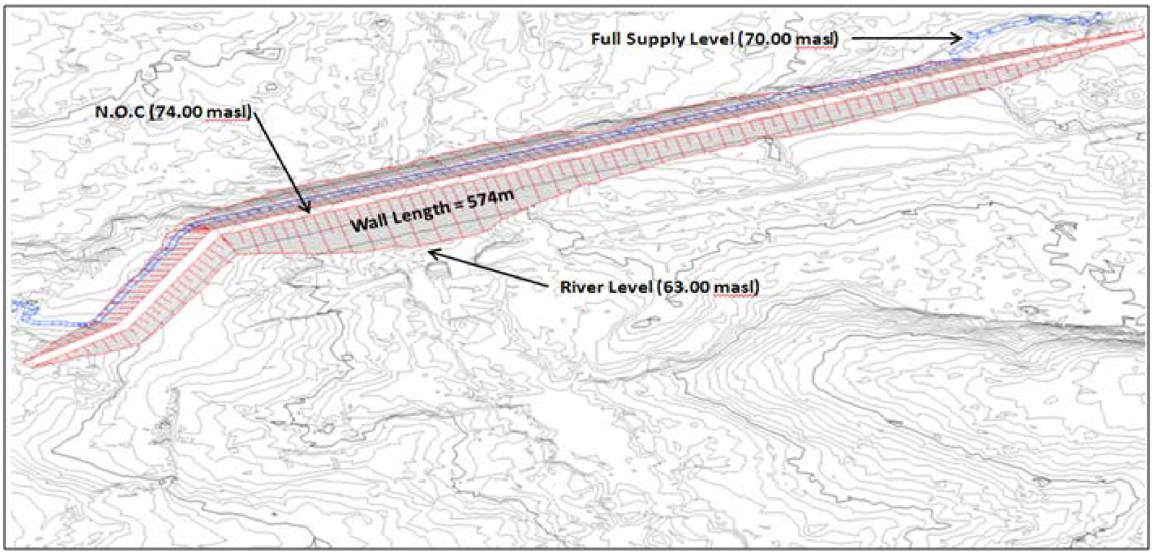
Figure 18.8 General Arrangement of Marvoe Creek Diversion System



Source: *Epoch, 2013*

Figure 18.9 below, shows the conceptual layout of Dam Wall “1”, this wall is currently in the design phase and both the alignment and non-overspill crest level (NOC) is subject to change as a result of geology and freeboard requirements with regards to wave actions. The design and construction drawings development for both dam walls fall under Epoch Recourses scope of works. The minimum allowable NOC level based on Safety Evaluation Discharge (SED) build up in the dam is 73.21 m.a.s.l.

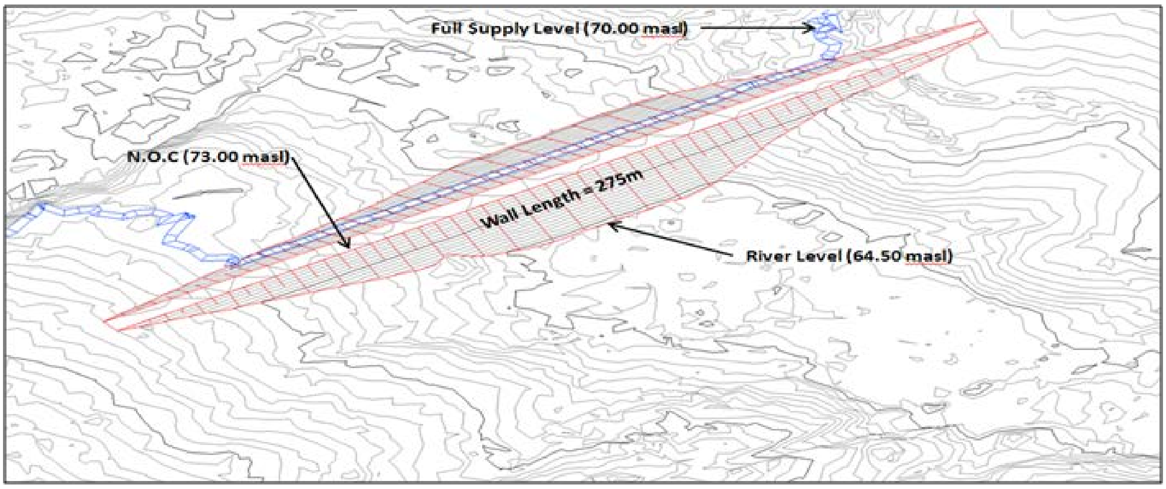
Figure 18.9 3D – View of Dam 1



Source: *Epoch, 2013*

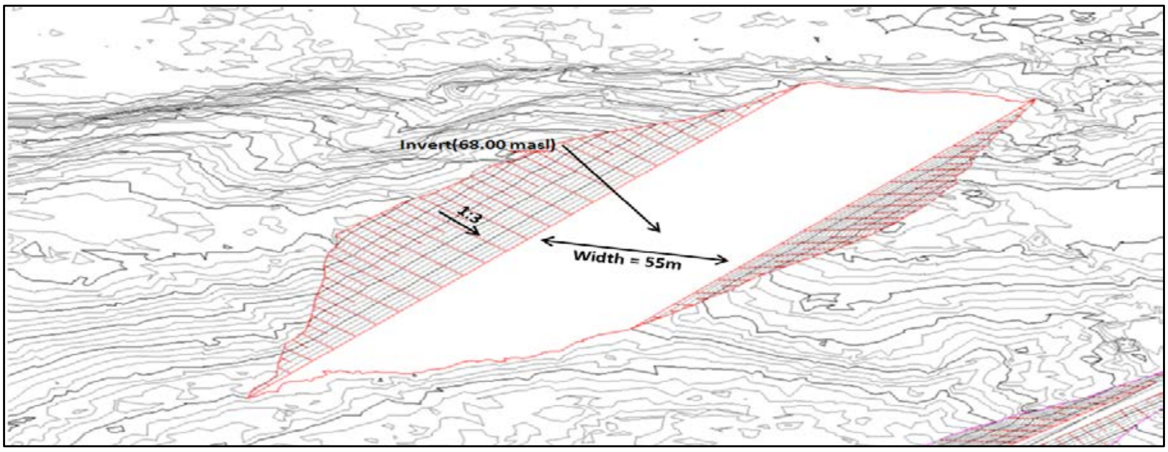
Figure 18.10 below, provides an overview of the intended configuration of the Dam Wall “2”, as for Dam Wall “1” the alignment may change during the final design development by Epoch Resources. The non-overspill crest level (NOC) is, however, the minimum allowed based on Safety Evaluation Discharge (SED) build up in the dam, which is 72.71 m.a.s.l.

Figure 18.10 3D – View of Dam 2



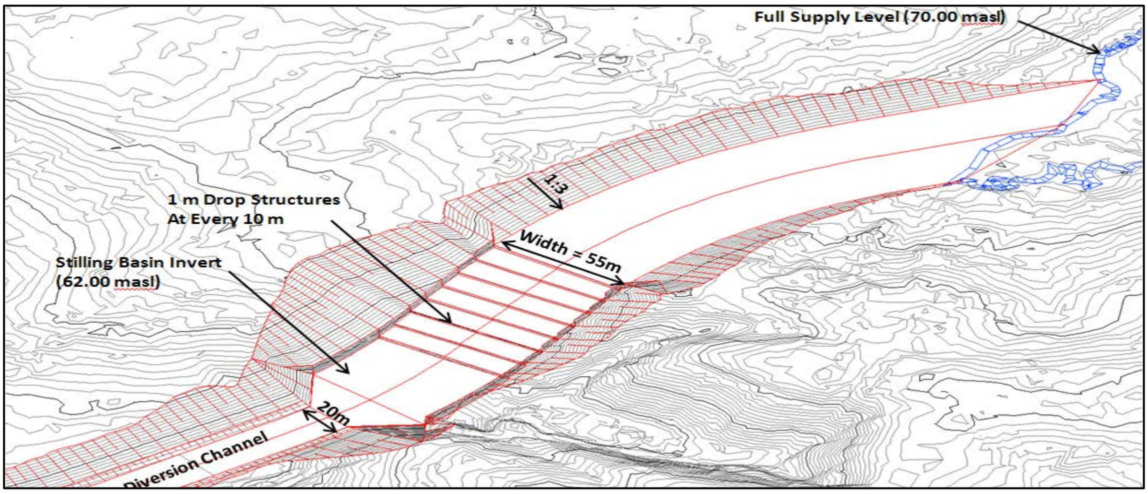
Source: *Epoch, 2013*

Figure 18.11 3D – View of Cutting Between Dam 1 And Dam 2



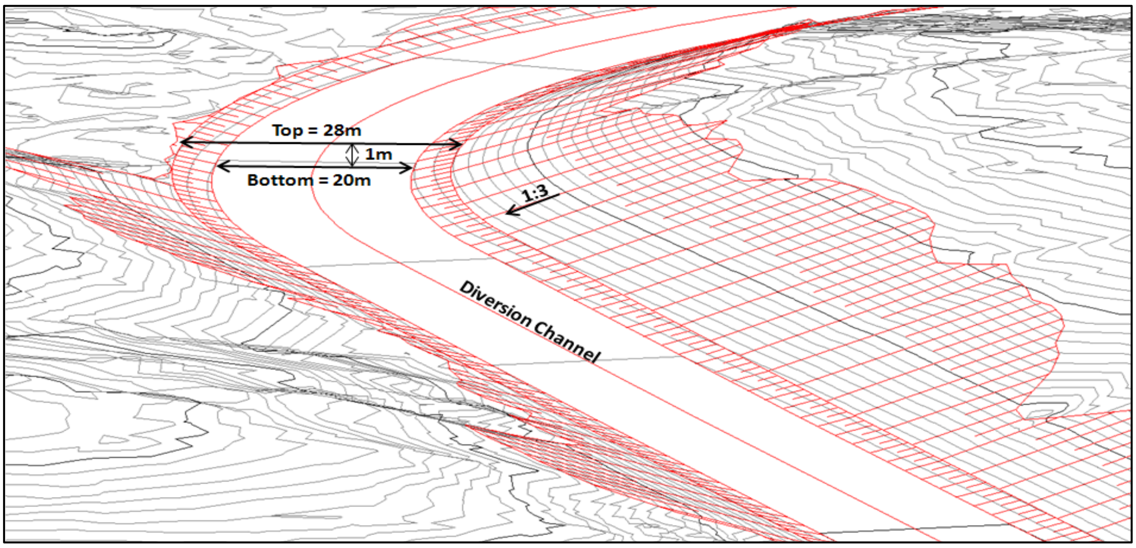
Source: *Epoch, 2013*

Figure 18.12 Spillway with Energy Dissipation System



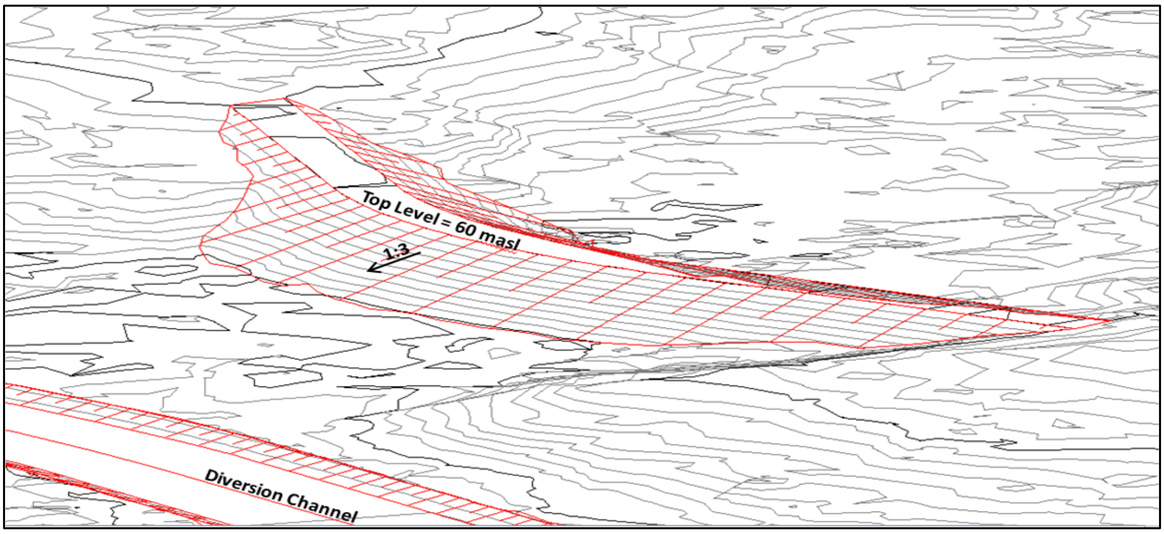
Source: *Epoch, 2013*

Figure 18.13 Typical 3D – View of Diversion Channel



Source: *Epoch, 2013*

Figure 18.14 Typical 3D – View of Flood Protection Berm



Source: *Epoch, 2013*

Except for Dam "1" and Dam "2", the rest of the 3D model as shown in the figures in this section was utilized for the final development of the construction drawings. On the deep cuttings for the spillway and diversion channel the sideslopes were made 1:1.5 where possible to save costs.

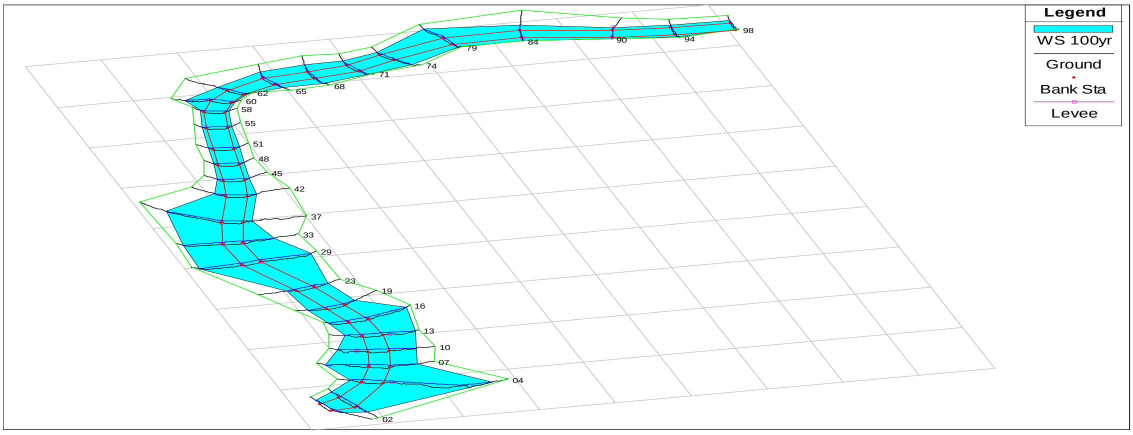
18.6.6.4 HEC-RAS Results

The results discussed in this section are only the most important outcomes that will confirm the specifications as shown on the construction drawings.

A three-dimensional CAD drawing was produced from which cross-section were extracted to build the hydraulic model in HEC-RAS. The results from HEC-RAS confirmed that flows less than the 2 year flood peak will remain within the main channel and larger floods will flow into the floodplains. The diversion channel's upstream section has a steeper longitudinal slope than the downstream section. The upstream section's slope is 1:130 and the downstream section's slope is 1:350.

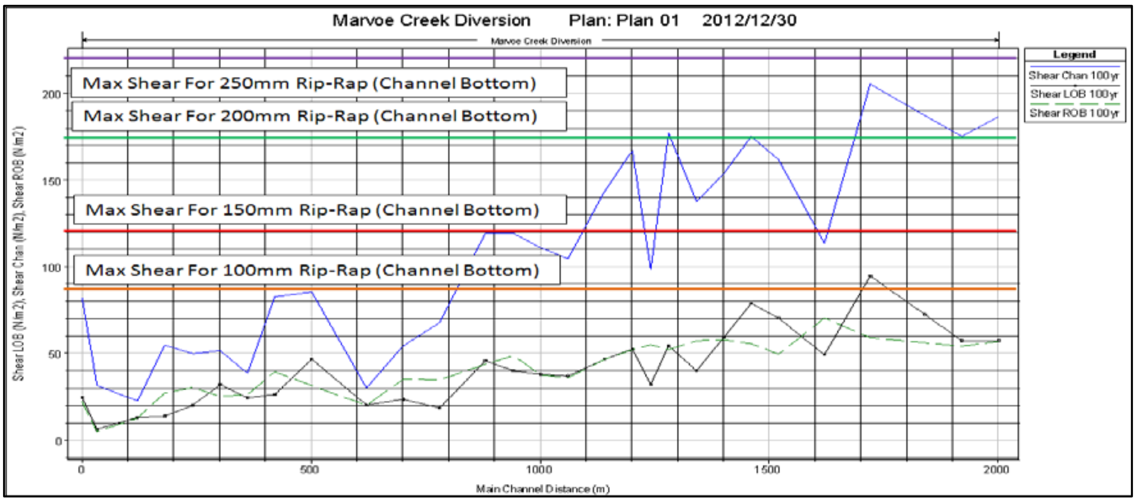
The average velocity in the upstream section of the main channel is between 3.0 m/s and 4.2 m/s during a 100 year flood event and in the downstream section between 2.0 m/s and 3.0 m/s. The shear forces in the upstream section of the main channel are between 120 N/m^2 and 200 N/m^2, for the downstream section between 30 N/m^2 and 120 N/m^2.

Figure 18.15 Three-Dimensional View of the Diversion Showing the 100 Years Flood Event



Source: *Epoch, 2013*

Figure 18.16 Minimum Scour Protection Stone Sizes Along the Diversion for the 100 Year Flood



18.6.7 Stability Analysis on the Gabion Walls

Stability analyses were carried out on the gabion walls utilised in the construction of the spillway dissipation system, i.e drop structures and wide walls. In this section only the input parameters and main results will be discussed. The input parameters with regards to the in situ soil conditions were obtained from a geotechnical investigations Golder and Associates undertook for the diversion during the pre-feasibility stage of the project.

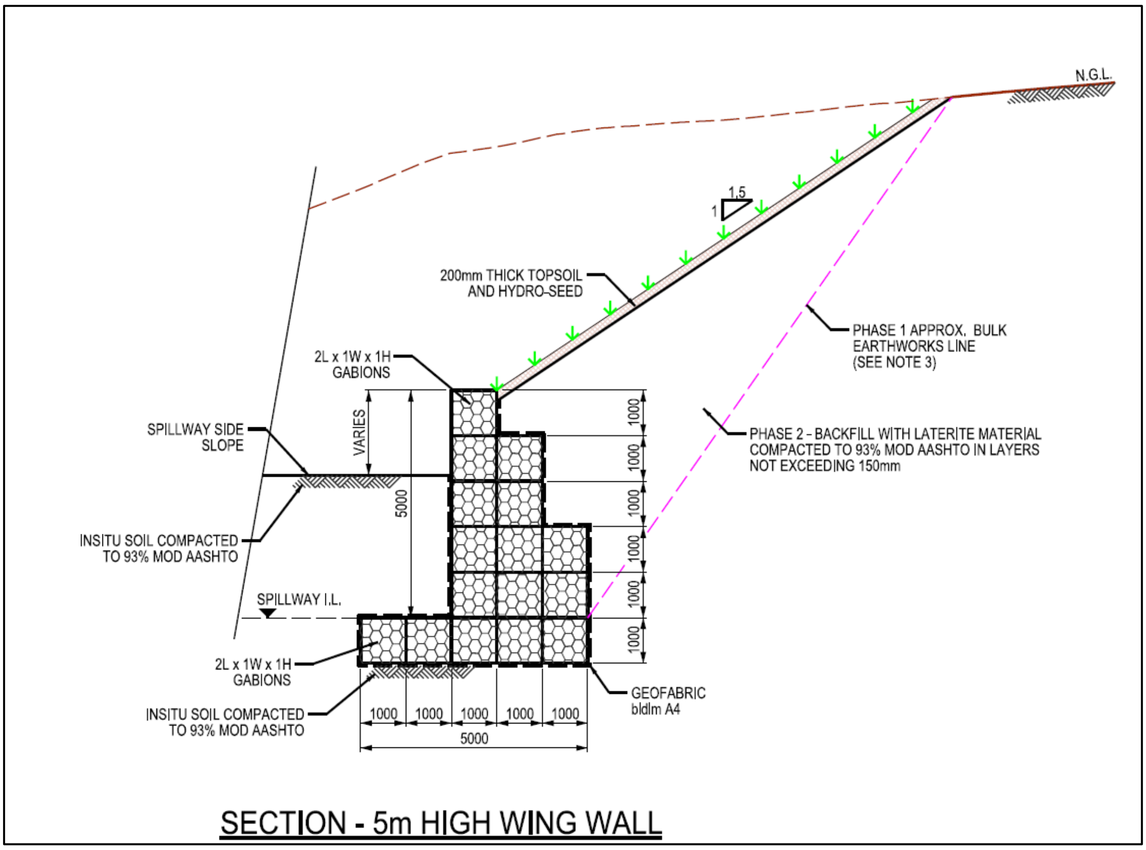
The basic input parameters are as follows:

- In situ soil unit weight is 18 kN/m^3
- In situ soil internal friction angle is 31°
- In situ soil Cohesion is 18 kN/m^2

- Rockfill unit weight is 23 kN/m^3
- Gabion porosity is 20%

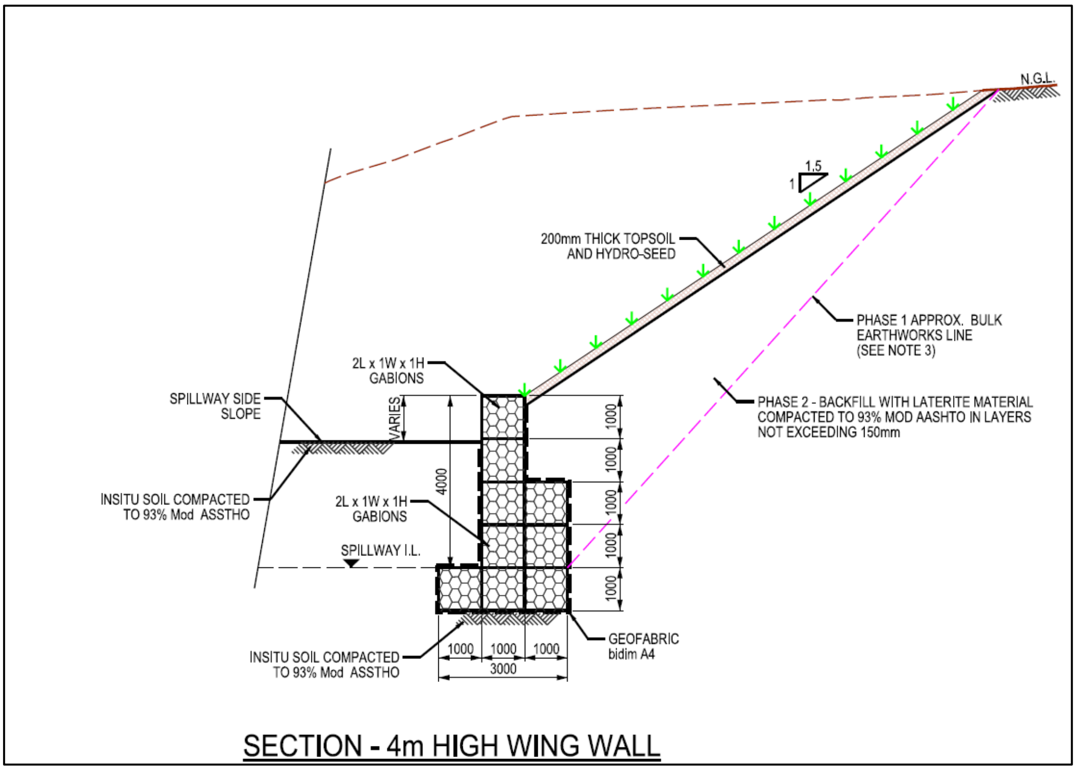
The highest retaining wall is 5 metres from the invert level of the stilling basin to the top of the wall. There are four configurations namely, 5 m, 4 m, 3 m, and 1 m (steps). The figure below will shows the typical dimensions of each configuration.

Figure 18.17 Typical Dimensions of the 5 Metre High Side Retaining Wall



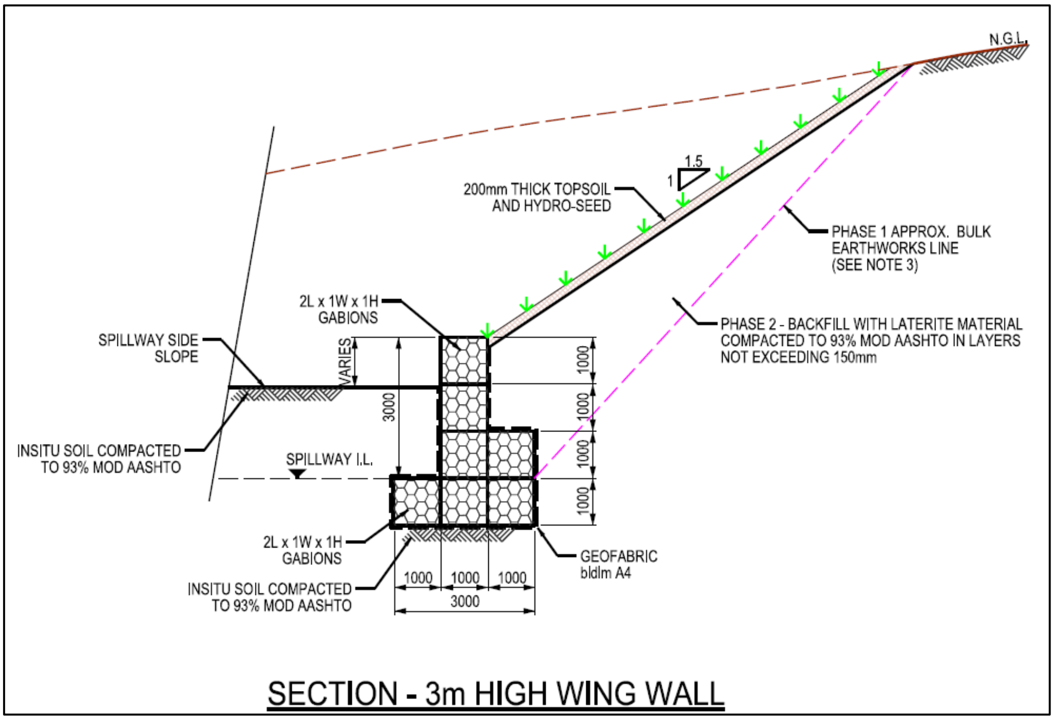
Source: *Epoch, 2013*

Figure 18.18 Typical Dimensions of the 4 Metre High Side Retaining Wall



Source: *Epoch, 2013*

Figure 18.19 Typical Dimensions of the 3 Metre High Side Retaining Wall



Source: *Epoch, 2013*

Table 18.3 Gabion Retaining Walls Stability Analyses Results

Retaining Wall Height (m)	Safety Factor (Sliding)	Safety Factor (Overturn)	Safety Factor (Overall Stability)	Base Normal Stress Left (kN/m^2)	Base Normal Stress Right (kN/m^2)	Max. Allowable Stress (kN/m^2)
5	2.62	5.21	1.81	49.93	102.07	493.83
4	3.04	3.66	2.05	64.63	75.56	427.42
3	3.52	4.91	2.30	34.18	82.08	417.84
1	Indefinite	Indefinite	8.23	N/A	N/A	439.77

18.6.8 Costing

Detailed quantities were taken from the three dimensional CAD model and construction drawings and forwarded to DRA for pricing. The design discussed in the report resulted in major savings from the budget allowed for by Golder during the pre-feasibility design.

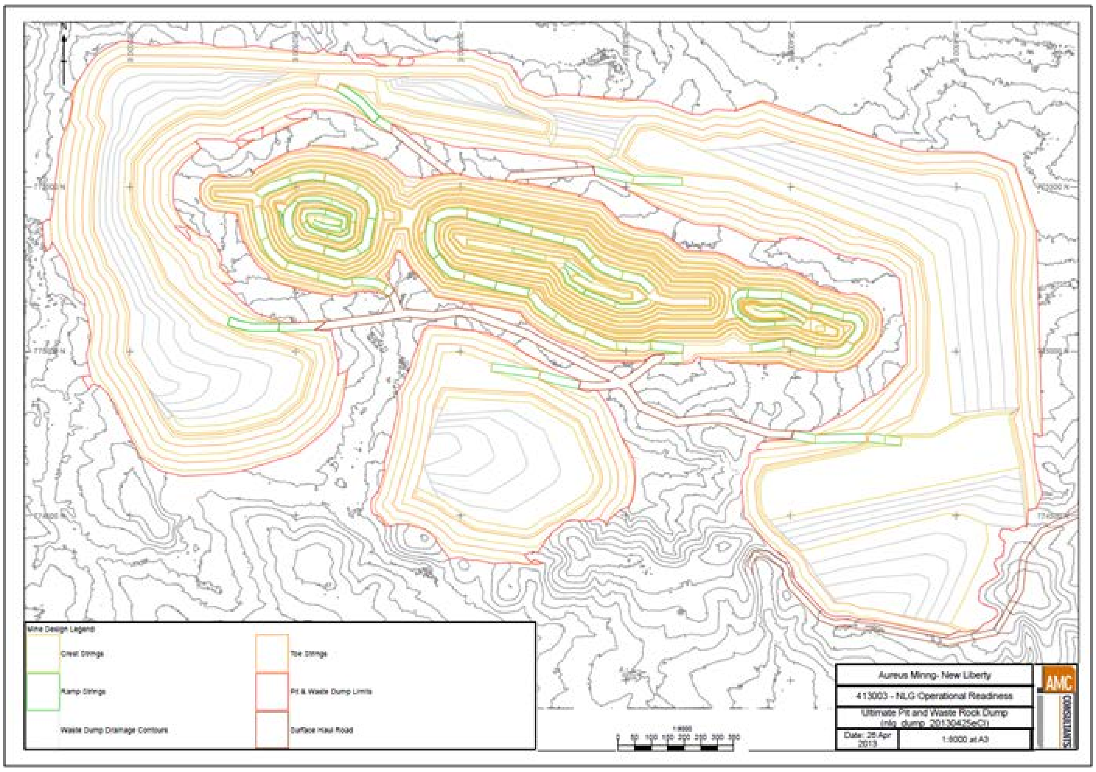
18.7 Waste Rock Dump

Figure 18.20 and Figure 18.21 show the final waste dump design for the project. The volume of material from the pit in bank cubic metres (BCM) is 48.3 million BCM. (mBCM). Considering an expansion factor of 35% the total waste dump volume required is 65.3 million m^3.

The following changes were incorporated into the design:

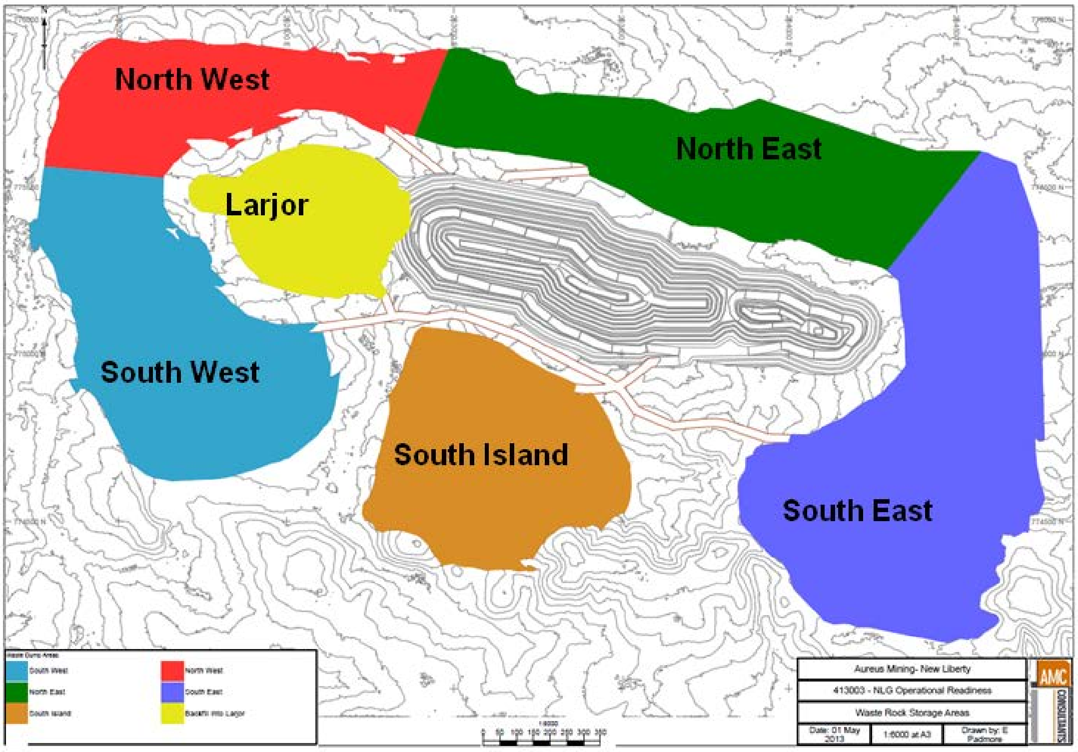
- The design now has two separate dumps. The gap between the dumps to the south of Larjor follows the existing course of the Marvoe Creek. This gap is for sedimentation ponds.
- The upper surface of the dump is profiled to shed water away from the pit catchment.
- The southern edge of the South East dump is positioned adjacent to the access road and at the drainage divide between the open-pit catchment to the north and the tailings facility catchment to the south.
- The northern toe of the dump has been positioned to abut the dams walls constructed to form the water storage dam and the Marvoe Creek Diversion Channel.
- An allowance has been made for a culvert under the western dump to facilitate drainage of the western open-pit catchment under the waste dump to the Marvoe Creek Diversion Channel.

Figure 18.20 Final Waste Dump Design



Source: *AMC, 2013*

Figure 18.21 Waste Dump Areas



Source: *AMC, 2013*

In addition to the construction of the waste dump, the project will use in-pit waste dumping. Once the Larjor pit is completed waste material will be placed in to it, rather than hauling to the waste dump south of the pit.

Table 18.4 shows the specifications of the waste dump.

Table 18.4 Waste Dump Design Specifications

Property	
Overall capacity (million m^3)	52.6
Overall Slope angles (degrees)	18
Lift height (m)	15
Berm width (m)	25
Batter angle(degrees)	33
Footprint area (Ha)	244
Elevation of top lift (m RL)	110
Maximum height (m)	45
Minimum height (m)	15

The waste dump surface has been designed with a gradient of approximately 1:40, draining away from the open pit catchment. The top surface of the dump is at 110 m Reduced Level (RL or elevation) grading to 100 m RL.

The dump has been designed to reduce infiltration and to minimize surface runoff towards the pit. Interim dump stages will be capped by saprolite to reduce infiltration and the dump will be graded to facilitate water flow away from the pit.

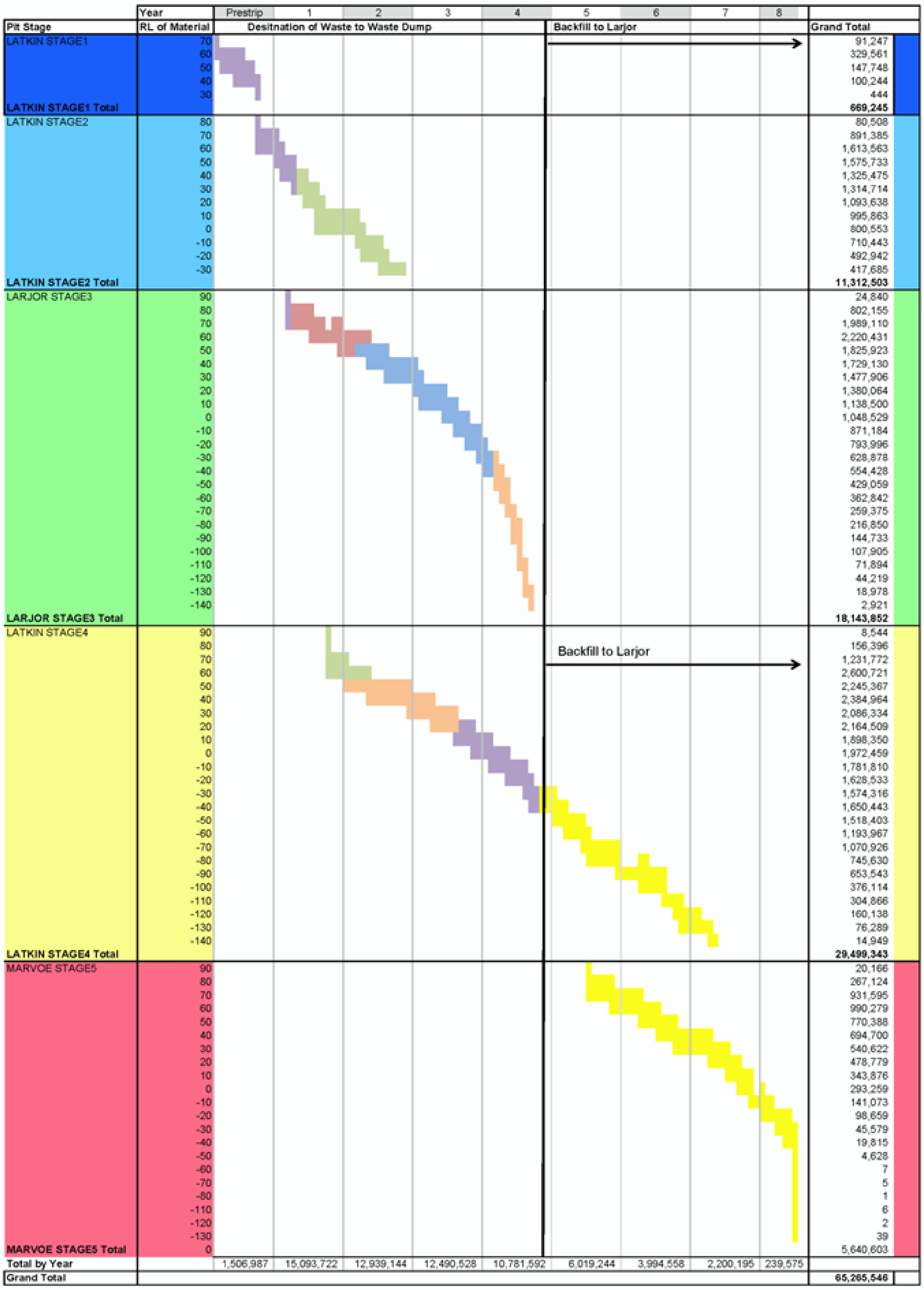
A waste schedule was constructed giving consideration to maintaining the shortest haul distances, where possible. The schedule takes into account the material required in Phase 1 of filling up the South East waste dump area. Table 18.5 shows the origin and destination of waste volume by year. The detailed schedule can be seen in Figure 18.22.

Table 18.5 Waste and Backfill Schedule

Source	Destination	Units		Totals	Year 0 Q3	Year 0 Q4	Year 1 Q1	Year 1 Q2	Year 1 Q3	Year 1 Q4	Year 2 H1	Year 2 H2	Year 3 Y	Year 4 Y	Year 5 Y	Year 6 Y	Year 7 Y	Year 8 Y
Stage 1	Waste Dump	Waste (Broken)	LCMs	0.7	0.5	0.1	-	-	-	-	-	-	-	-	-	-	-	-
Stage 2	Waste Dump	Waste (Broken)	LCMs	11.3	-	0.8	3.5	2.1	2.1	0.5	1.5	0.8	-	-	-	-	-	-
Stage 3	Waste Dump	Waste (Broken)	LCMs	18.1	-	-	0.3	1.7	1.2	1.0	2.6	2.7	5.7	2.9	-	-	-	-
Stage 4	Waste Dump	Waste (Broken)	LCMs	29.5	-	-	-	-	-	2.7	2.7	2.7	6.8	7.9	5.2	1.4	0.2	-
Stage 5	Waste Dump	Waste (Broken)	LCMs	5.6	-	-	-	-	-	-	-	-	-	-	0.8	2.6	2.0	0.2
Sub Total	Waste Dump	Waste (Broken)	LCMs	65.3	0.5	1.0	3.8	3.8	3.3	4.2	6.7	6.2	12.5	10.8	6.0	4.0	2.2	0.2
Stage 1-4	Waste Dump	Waste (Broken)	LCMs	50.8	0.5	1.0	3.8	3.8	3.3	4.2	6.7	6.2	12.5	8.8	-	-	-	-
Stage 4/5	Larjor	Waste (Broken)	LCMs	14.5	-	-	-	-	-	-	-	-	-	2.0	6.0	4.0	2.2	0.2

Volumes assume a 35% swell factor from in situ to broken volume.

Figure 18.22 Waste and Backfill Schedule

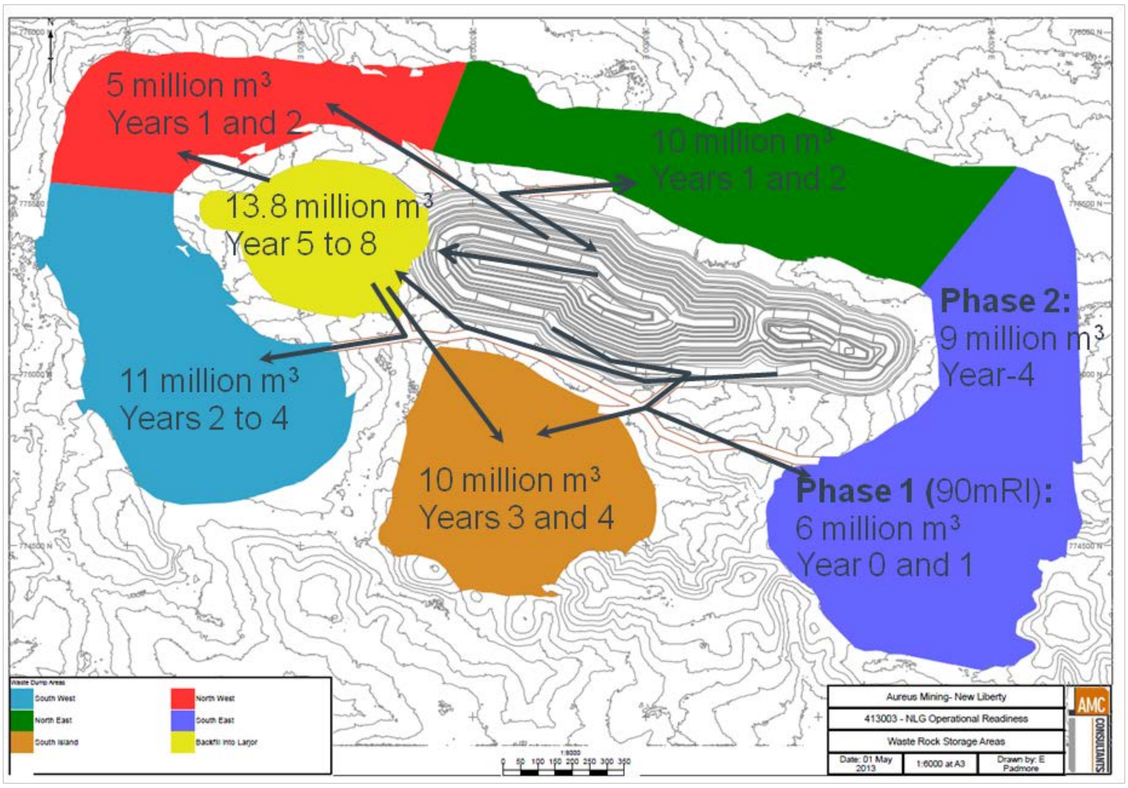


The capacity of different areas of the waste dump and the volume scheduled to each area is shown in Table 18.6, and in Figure 18.23.

Table 18.6 Capacity of Waste Dump Areas

Waste Dump Area Destination	Volume into Dump(million m^3)	Capacity of Dump Area (million m^3)
South East – Phase 1	6.0	6.0
South East – Phase 2	9.0	9.8
North West	**5.2**	**5.0**
North East	10.2	10.8
South West	**10.7**	**10.6**
South Island	10.3	10.5
Larjor	13.8	14.0
Total	**65.3**	**66.6**

Figure 18.23 Waste Dump Areas and Volumes



Source: *AMC, 2013*

The total area of top soil to be removed around the pit and waste dump area is 321 Ha. This is the area covered by the pit, dump and surface haul roads. Table 18.7 shows the respective surface areas. Assuming an average thickness of top soil of 0.3 m, a total of 1.22 million m^3 will require stockpiling as topsoil for later use in rehabilitation. This figure does not take into account topsoil from any other infrastructure such as the mine camp, process plant, MCDC or the tailings storage area.

This topsoil volume will form a significant dump if stored in one location. It may be more efficient to store topsoil in a number of smaller stockpiles locations around the site closer to from where it is stripped and where it will be used for rehabilitation.

Table 18.7 Areas of Topsoil Stripping

Topsoil	Surface Area (Ha)
Main dump	204.5
South Island Dump	39.7
Haul Road north of Pit	1.6
Haul Road south of Pit	4.6
Open Pit	70.8
Total	**321**

18.8 Waste Controls

Industrial and domestic waste will be recycled where practical. Any combustible waste will be burnt and the ashes buried with any non-combustible waste in the mine waste dump.

Treatment facilities for sewerage disposal will form part of the site accommodation requirements.

18.9 Closure Plan

18.9.1 Closure Objectives

The primary closure plan objectives are:

- To ensure long-term physical and chemical stability of the Project components (e.g. the residue management area and waste rock storage facilities) remaining on-site at closure;
- To minimize long-term care and maintenance requirements; and
- To minimize the health and safety hazards posed by the site with regard to local residents and their livestock.

18.9.2 Process Plant

The closure objectives for the Process Plant site are as follows:

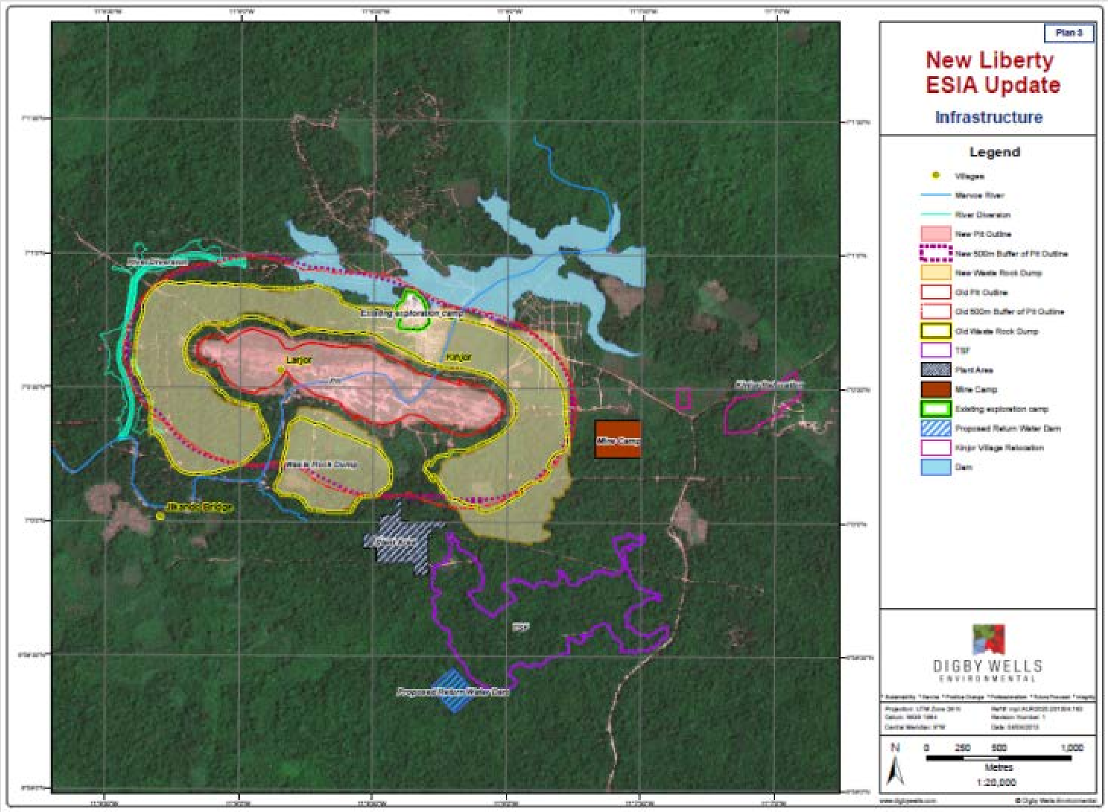
- Decommission unnecessary materials, equipment and infrastructures;
- Remove and remediate soil and restore original land use to the extent feasible;
- Handover some infrastructures for local use, if appropriate; and
- Encourage re-vegetation.

The closure measures, which will be undertaken in accordance with international accepted pratices and the Bea Mountain MDA, can be summarized as follows:

- Conduct environmental site assessment/risk analysis to determine clean-up requirements;
- Drain and store all valuable reagents for resale;
- Flush and clean all equipment to ensure that no residual toxic materials are left;
- Dismantle equipment and sell for re-use if possible or sell as scrap. Non-saleable material will be disposed at an appropriated disposal facility;
- Dismantle unnecessary buildings and resell. Non saleable material will be disposed of at an appropriated disposal facility;
- Remove impacted/contaminated soils and dispose at the TSF (beneath the saprolite/ cover. Treat or cover the impacted area to enhance satisfactory mitigation;
- Where feasible, promote re-vegetation of the footprint area of the demolished facilities;
- Scarify non-essential internal roads and other surface areas to encourage infiltration and natural re-vegetation;
- Culverts will be removed where necessary and the disturbed areas re-graded to allow for unobstructed drainage; and
- Where appropriate, construct diversion ditches/channels to route run-off to the open pit.

The project closure plan can be illustrated as follows in Figure 18.24.

Figure 18.24 Project Closure Plan



Source: *Digby Wells, 2013*

18.9.3 Open Pit

The closure objectives for the open pit are:

- Ensure physical stability;
- Ensure chemical stability; and
- Limit human and animal access.

The closure measures for the open pit are:

- Decommission pipelines, pumps, and electrical lines;
- Stop open pit dewatering;
- Block access ramps;
- Evaluate the stability of the pit slopes and where required, re-shape or stabilize pit slopes;
- Remove water diversion ditches to facilitate flooding of the open pit;
- Construct a 3 m high safety berm around open pit with warning signs;
- Construct spill structures and ditch for the floodwater to spill to the MCDC; and

- Continuously assess the water quality in the open pit and if required, provide water treatment system (e.g. pit lake treatment, constructed wetland, water treatment plant, etc.).

18.9.4 Tailings Storage Facility

The closure objectives for the TSF are:

- Creation of a stable landform;
- Ensure chemical stability;
- Minimize erosion and dust generation; and
- Minimize rainfall infiltration to extent practically possible.

The closure measures for the TSF are:

- Construct a permanent spillway to ensure physical stability of the facility during storm events;
- Fill the tailings pond area to eliminate water ponding (no water should pond on the surface of the TSF post closure);
- Place the tailings cover (which includes from top to bottom- vegetation, 0.5 m thick topsoil, and 1 m (minimum) thick saprolite layer);
- Reshape the downstream berms and encourage re-vegetation; and
- Maintain access road for continuous monitoring.

Note: Current designs for the tailings storage facility is a single cross-valley impoundment design.

18.9.5 Marvoe Creek Diversion Channel

The closure objectives for the MCDC are:

- The MCDC to remain at closure; and
- The MCDC to become stable and self-sustaining.

The closure objectives for the MCDC are:

- Inspect the MCDC and natural channel and make required repairs and upgrades to improve hydraulics and long-term morphologic stability;
- Post-closure, open pit overflow will report to the diversion; and
- Additional armouring may be required to handle the additional flows.

18.9.6 Waste Rock Dump

The closure objectives for the waste dump are:

- Creation of a stable landform
- Confirm chemical stability
- Minimize erosion and dust generation
- Minimize rainfall infiltration to the extent practically possible; and
- Apply stripped topsoil and vegetate or allow natural colonisation of vegetation. Sufficient topsoil will be stripped and stockpiled during mine construction for use in rehabilitation of the waste dump as detailed in Section 18.7.

The closure measures are:

- Reshape the crest (2%) of the waste dump and allow drainage channels on the side slope for the run-off to flow to the open pit;
- Where necessary, reshape the side slopes of the waste dump to 3H:1V for ease of cover placement and provide 10 m wide benches on the side slope for water management; and
- Place the waste rock dump cover (which includes vegetation and 0.5 m topsoil).

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

Liberia allows for the direct export of gold doré to refiners. The Government of Liberia has the right, but not the obligation, to purchase a portion or all of the production at fair market value. As such, it has been assumed that all gold shall be sold, after refining, on the open market. For the economic evaluation Aureus has used a flat gold price of US$1,400/oz. This price was selected by Aureus based on the spot gold price during compilation of the DFS and taking a view of industry gold price forecasts.

There is currently no gold refining capability in Liberia. As such, the New Liberty operation will produce gold doré which will be air freighted from site to refineries in either Europe or South Africa.

19.2 Contracts

No EPCM, mining, smelting, refining, transportation, handling, or sales contract, or arrangements, have been entered into.

Moving into the project implementation phase, Aureus intend to engage DRA as EPCM contractor later this year. In January 2013, Aureus appointed International Construction and Engineering (ICE), an international company with strong experience in Liberia and South Africa, to perform the civil and earthworks component of the development of New Liberty commencing with the MCDC.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The following section provides an overview of the legislative process surrounding the Project, the permitting requirements of the Project, and the environmental impacts and implications of the Project together with Aureus' commitments to the environment and local community.

An Environmental and Social Impact Assessment (ESIA) was undertaken from Q4 2010 to Q2 2012 to investigate the local environmental and social situation existing prior to the development of the Project and to determine the likely positive and negative impacts of the Project. The timing, extent, intensity and cumulative effects of these impacts were investigated. The ESIA also identified the necessary management measures required to mitigate the identified impacts. These form the basis of the Environmental Management Plan (EMP), Resettlement Action Plan (RAP) and accompanying Community Development Plan (CDP). The ESIA was approved by the Liberian Environmental Protection Agency in October 2012 and the RAP and CDP during April 2013.

Subsequent to the completion of the ESIA, a mine optimization study was conducted to identify better positions for the plant, tailings dam and mine village. The newly-identified positions are all within the area permitted for mining and are therefore covered by the ESIA. Detailed studies are ongoing in these areas of changed infrastructure positions to establish baseline conditions, and an updated ESIA report is being compiled. The updated ESIA will be submitted to the Liberian Environmental Protection Agency as per the MDA requirements.

20.2 Liberian Legislation and Guidelines

In November 2002, the Liberian Government adopted the National Environmental Policy, the Environment Protection Agency Act and the Environment Protection and Management Law (EPML). The three documents became law in April 2003.

Under Part III of the Act creating the EPML of the Republic of Liberia (2002), an ESIA Licence or Permit is required from the Environmental Protection Agency (EPA) prior to commencement of activities specified under Annex 1 of that Law. Consideration of the Project's Project Brief (submitted to the EPA in November 2010) by the EPA identified the need for a full ESIA.

Terms of the Mineral Development Agreement

The Mineral Development Agreement (MDA) between Aureus Mining and the Liberian Government is of relevance to the Scoping Report which reads "*16.1 Environmental Impact Statement [EIS]: The Parties recognise that Operations may result in some pollution, contamination or other environmental damage to land, water and the atmosphere within the Contract Area and elsewhere. The EIS illustrates the adverse effect operations will have on the environment and review plans to mitigate such effects.*"

The MDA and its commitments were taken into account during the development of the ESIA.

International Standards and Best Practices

The ESIA refers to relevant international standards, guidelines and best practice documents notably those of The World Bank Group (WBG), including the International Finance Corporation (IFC) Performance Standards, the Equator Principles (EP's) and the World Health Organisation (WHO) Guidelines.

International Finance Corporation Standards for the ESIA Process Followed

The IFC Performance Standards on Social and Environmental Sustainability address social and environmental issues and potential impacts associated with project development. The Performance Standards require that social and environmental impacts and risks of a project are identified and assessed in the early stages of project development and continue to be managed throughout the life of the project.

The Environmental, Health and Safety Guidelines provide general and industry-specific best practice guidance and numerical limits for occupational and community health and safety, noise, gaseous emissions, effluent discharges and other waste products.

The IFC Performance Standards consist of eight performance standards. Of particular importance is the introduction of Performance Standard 1 of the IFC Policy on Social and Environmental Sustainability. This standard requires the Project proponent to assess the social and environmental impacts of the proposed Project and to ensure the continued management of social and environmental performance throughout the lifecycle of the Project. The ESIA has now been completed and subsequently approved. Procedures for long-term monitoring and reporting on the effectiveness of the risk management measures are also required in terms of meeting Liberian and international standards. Performance Standard 5 on Land Acquisition and Involuntary Resettlement is also relevant in terms of the Project's Resettlement Action Plan (RAP) for the villages of Kinjor and Larjor.

Equator Principles

The 'Equator Principles' are a set of standards for determining, assessing and managing environmental and social risks in project finance transactions. It comprises a set of principles adopted by the Equator Principles Financial Institutions (EPFI's) to ensure that the projects they finance are developed and implemented in a manner that is socially responsible and environmentally sound. These principles apply to all project-financing by EPFI with total project capital costs of US$10 million or more, and across all industry sectors.

Minerals and Mining Act

The Minerals and Mining Act (2000) states that minerals on the surface of the ground or in the soil or subsoil, rivers, streams, watercourses, territorial waters and continental shelf are the property of the Republic of Liberia. This act allows for the establishment of a Minerals Technical Committee consisting of: Ministry of Lands, Mines and Energy, Ministry of Justice, Ministry of Finance, Ministry of Planning and Economic Affairs, National Investment Commission, Ministry of Labour, Council of Economic Advisors, and Central Bank of Liberia. This committee has the power to negotiate agreements for Class A Mining Licences.

Project Permitting Process

The primary permit/licence required for the development of the NLGM Project is an Environmental Permit issued by the Minister of Environment. This was granted for the Project in October 2012 and is valid for three years subject to an annual renewal by the Liberian EPA. Various other operational permits from other governmental departments will be required (e.g. construction of buildings etc.) but are relatively procedural in nature.

In order to gain the Environmental Permit, the following permitting process steps were undertaken:

- Application for an Environmental Impact Assessment Licence by BMMC;
- Notice of Intent completed;
- Development of a Project Brief submitted to the EPA (carried out in November 2010);
- Conducting the Scoping Phase of the ESIA and submitted a Scoping Report to the EPA (carried out in April 2012);
- Conducting the Impact Assessment Phase of the ESIA, notably the applicable specialist assessments and baseline studies;
- Based on the findings of the specialist assessments, the EIS was developed and was submitted to the EPA at the end of July 2012; and
- The EPA and other relevant Liberian governmental agencies have reviewed the EIS and the Environmental Permit was granted for the Project in October 2012.

20.3 ESIA Study Area

The New Liberty Project is located in the Grand Cape Mount County of Liberia, approximately 90 km north-west of the capital, Monrovia. Liberia lies almost entirely within the Upper Guinea forest block, which forms the western part of the West African Guinean Forest.

There are numerous hills, valleys and watercourses in this zone. Vegetation in Grand Cape Mount County and in the eastern part of the country consists mostly of forests. Most of the private agricultural concessions are located in this belt where both agriculture and forestry are favoured by the prevailing topographical and climatic conditions.

The forest ecosystems can be divided into closed forest and transition or secondary forest. The closed forest can further be sub-divided into evergreen and semi-deciduous forest.

The vegetation surrounding the New Liberty Project can be broadly classified into four vegetation communities, namely:

- Pristine primary forest;
- Impacted primary forest;
- Secondary forest; and
- Transformed landscapes.

From past baseline assessments that have been conducted, a total of 329 plant species have been identified to occur in the area. A large portion of these species are indigenous to the region. 216 insect species, 16 reptile species, 6 amphibian species, 56 bird species and 10 mammal species have also been recorded. In terms of re listed species, the African Grey Parrot (Pisttacus erithacus) and the three-cusped Pangolin (Phataginus tricuspis) have also been recorded as occurring within the region.

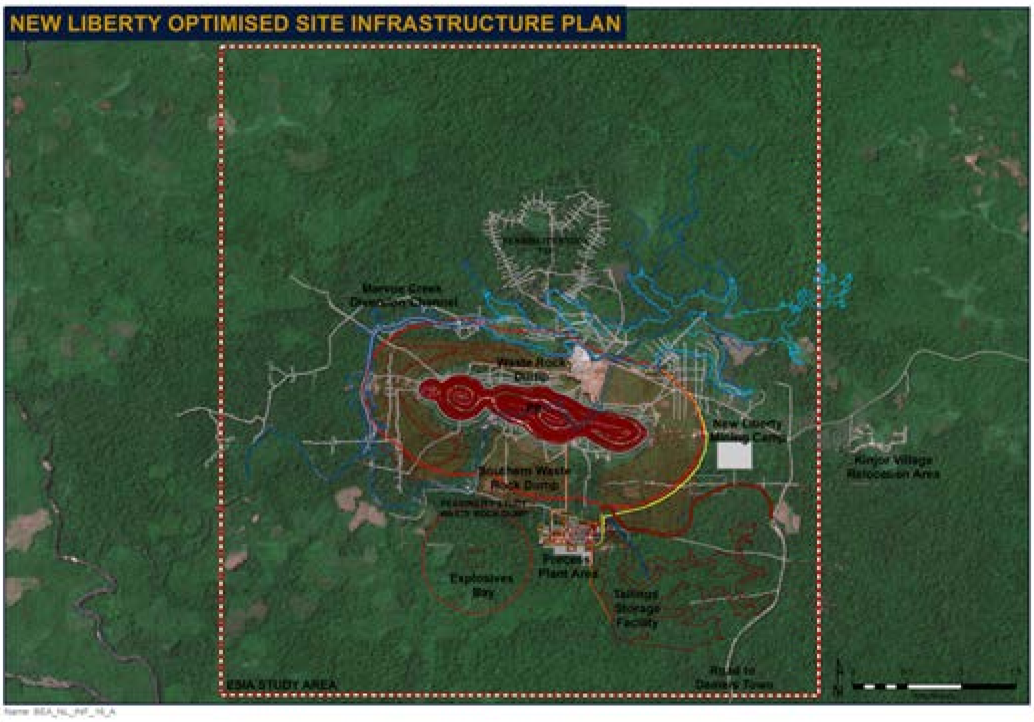
With respect to land use, the area is mainly characterized by tall tropical forests or disturbed forest area. A significant portion within the study area has experienced impacts from artisanal mining. About 30% of the area is arable, with the remaining proportion of the either marginal or unsuitable for agricultural purposes.

The demographic and socio-economic conditions of the project area, as well as its political and traditional structures, are extensively documented in the RAP. The project area includes two affected villages (Kinjor and Larjor); the main economic activities of local communities are subsistence agriculture and artisanal mining.

The study area for the ESIA consists of the footprint of the proposed Project (approximately 8 km^2); upstream and downstream areas; local topography (studied as part of the infrastructure design process and fed into the ESIA); the directly-affected villages of Kinjor and Larjor; neighbouring villages within a radius of approximately 5 km of the site; and the 20 km gravel road via Danieltown in the South that has been regraded and serves as the Project's main access road. In addition, consideration was given to the wider geographical context where applicable.

As mentioned above, a project optimization study was undertaken subsequent to completion of the ESIA, and new locations selected for the plant, tailings dam and mine village. These changes necessitated modification to the areas subject to detailed baseline studies, which are currently underway and will be finalized by the end of June 2013. The findings of these studies will be submitted as an addendum to the approved ESIA. Figure 20.1 shows the new areas of infrastructure and of detailed baseline investigations.

Figure 20.1 New Liberty ESIA Study Area



Source: *Aureus, 2013*

20.4 The Environmental and Social Impact Assessment

The EIS details the findings of the environmental and social baseline studies and specialist studies conducted during the ESIA; it also presents the Environmental Management Plan (EMP). The format for the EIS is taken from and aligned with the Liberian EPA "Environmental Impact Assessment Procedural Guidelines" (2006) and commitments are in line with international standards and practices.

The key factors considered in the ESIA were as follows:

- The existing/baseline environment, and obtaining relevant baseline data;
- The potential direct and indirect environmental and socio-cultural impacts of the proposed Project;
- Measures that are technically and economically feasible and that would mitigate any significant adverse environmental or social impacts related to the proposed project;
- Project alternatives that are technically and economically feasible; and
- Closure objectives and costing associated with the Project.

The following specialist studies were undertaken as part of the ESIA:

- Soils and land use;
- Surface water, also including surface water quality;

- Groundwater;
- Geochemistry;
- Air quality;
- Greenhouse gas emissions;
- Noise;
- Terrestrial ecology;
- Aquatic ecology (water quality);
- Socio-economic;
- Cultural heritage;
- Visual aesthetics; and
- Closure objectives and closure costing.

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

20.5 EMP Commitments

Aureus is committed to limiting the negative impacts of operations at the proposed Project site on the environment. The EMP was developed in line with Liberian legislation and international good practice standards and principles, in order to put this commitment into practice. The EMP translates the findings and recommendations of the ESIA into measures for management and monitoring of impacts of the proposed Project activities. The mitigation measures that have been identified in the EMP are for the exploration, construction, operational and decommissioning and closure phases of the Project.

Aureus will implement this EMP, and will update it every three years after its approval, as per Liberian legislative requirements. Environmental management of the Project will be an evolving process over the life of the mine. In particular, the environmental management and mitigation measures and the monitoring programme outlined in the EIS will be updated annually for continual improvement and for management practices to remain current and aligned with Liberian legislation and industry good practice. An annual report is to be submitted to the Liberian EPA and is a condition of the Environmental Permit.

The key recommendations highlighted in the ESIA include:

- A significant socio-economic impact associated with the proposed Project is the involuntary relocation of two communities, as well as associated economic displacement. The RAP must therefore be completed and implemented in line with Liberian legislative requirements and IFC Performance Standard 5. The development and implementation of the RAP should be a consultative process involving affected persons and other relevant stakeholders;
- Aureus needs to undertake quantitative groundwater modelling to ascertain the potential impact on groundwater levels, quality and contamination in the surrounding streams, rivers and boreholes, and committing to establish a comprehensive ground and surface water monitoring programme on site;
- In association with the quantitative groundwater study, Aureus needs to undertake a Phase 2 Geochemical Assessment, which includes kinetic testing;
- Aureus needs to undertake quantitative air quality modelling, aimed at further assessing and managing air quality and dust impacts associated with the Project;
- Aureus needs to implement the EMP mitigation measures and the associated environmental and social monitoring measures as outlined in this EIS; and
- Health and community programmes supported by Aureus should be culturally appropriate and sustainable beyond the life of the project.

20.6 Progress with the Actions Outlined by the ESIA

Since the approval of the ESIA, the following progress has been made in terms of implementing the recommendations listed above:

- The RAP and the CDP have been completed and approved by the Liberian Government in April 2013 (discussed in more detail below);
- A comprehensive Stakeholder Engagement Plan (SEP) for the project as a whole is currently being developed;
- Groundwater studies and modelling are far advanced and a comprehensive ground and surface water monitoring programme has been established;
- Geochemical Assessment studies have been conducted, which includes kinetic testing. Further kinetic testing is underway. Results to date indicate that there is a low potential for the waste rock or the tailings to become acidic; however, this is not uncommon, and suitable mitigation measures have already been included in the project design.
- Air quality modelling studies are underway; and
- Numerous management plans have been developed to manage the identified impacts.

20.7 Resettlement Action Plan

The Project will involve relocation and resettlement of two villages (Kinjor and Larjor) encompassing 325 property owners and their households, as well as the relocation of some households along the access road. Relocation of the latter is primarily motivated by potential safety impacts associated with increased traffic volumes caused by the Project. International best practice for resettlement related to private sector projects is guided by the IFC's Performance Standards on Social and Environmental Sustainability, and particularly defined by the IFC's Performance Standard 5: Land Acquisition and Involuntary Resettlement.

The RAP has been developed to specify how relocation and compensation for affected assets and livelihoods will be undertaken. In accordance with international best practice, the RAP identifies all laws of Liberia that are applicable to land acquisition and involuntary resettlement, including relevant local customs and traditions that govern affected communities.

The legal framework lays the foundation for the four key elements of the RAP:

- Approach to Land Access and Management
- Establishing rates of compensation
- Determining eligibility for compensation and resettlement assistance, including livelihood initiatives.; and
- Establishing mechanisms to resolve grievances among affected persons related to compensation and eligibility.

The purpose of developing a RAP at this stage of the Project cycle is to outline the framework for execution of the NLGM Resettlement Project as early as possible within the project development cycle. This will allow for effective disclosure to key stakeholders, and subsequent feedback and inputs, prior to completion of resettlement negotiations and implementation.

As indicated above, the RAP has been approved by the Liberian EPA and the relevant owners' compensation packages approved. The RAP stakeholder engagement process expands on the EIS process in that it includes the regulatory authorities that are expected to be involved in the development and implementation of the RAP. The primary focus of the RAP stakeholder engagement process is on the directly affected property owners in the Kinjor and Larjor communities and along the main access road, as well as their representative bodies (including the traditional authorities concerned).

In order to involve affected communities in the resettlement planning and implementation processes, a Resettlement Working Group (RWG) was established and comprises representative members of the resettlement-affected communities and households, as well as the relevant local government structures, town councils, traditional authorities, women's group, the youth, and non-governmental organizations. The RWG is chaired by Aureus and the chairperson nominated is available at all formal RC meetings.

Resettlement Site Selection

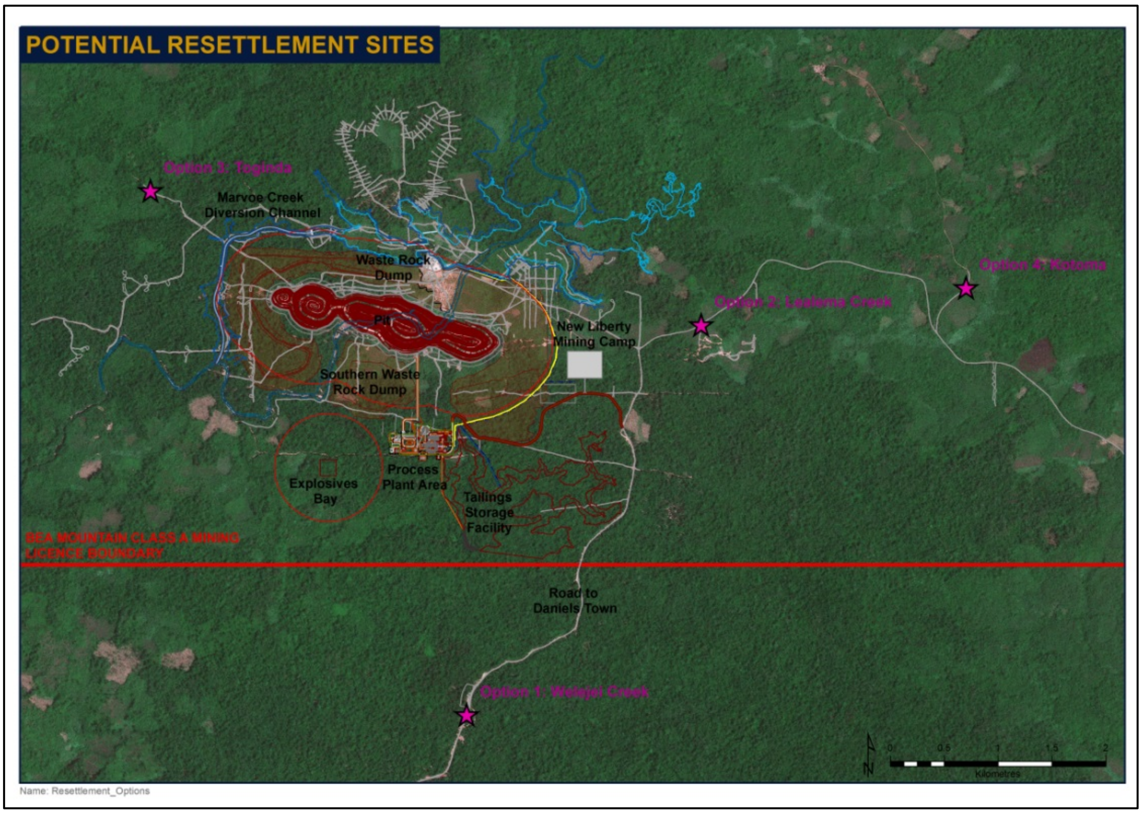
The following general criteria were applied in the selection of a preferred resettlement site:

- Availability of land and the quality/production potential of the land;
- Current land ownership and land acquisition conditions;
- Location of resettlements in relation to the licence area;
- Proximity to the NLGM in order to:
  - Accommodate local recruitment processes;
  - Minimize travel distances to the mine for the local workforce;
  - Prevent/control human settlement in too close proximity to the mine camp;
- Proximity/access to public infrastructure and community facilities;
- Availability of land to develop agriculture-based livelihood restoration projects;
- Settlement patterns and land use at the resettlement site (integration potential);
- Availability and quality of natural resources (particularly water); and
- Compatibility with the socio-cultural practices and economic activities of host community.

During an initial resettlement site selection workshop with the RWG, four site options were identified as shown in Figure 20.2. Site Option One was the preferred option of the RWG based on the outcome of a workshop on site options. However, this site is located on an adjacent concession licence area, which would have significantly complicated the land acquisition process. During discussions with the RWG, it was agreed that Site Option One should be discarded based on these limitations. Site Option 2 (Lealema), was subsequently selected by the RWG as the preferred site. Lealema is located approximately 4.5 km from Kinjor and 3 km from the proposed new Project mine camp.

The compensation required for the acquisition of the replacement land has been finalized. The area of the new resettlement site has been cleared and an area constructed where bricks are currently being manufactured to be used for the construction of new houses and other buildings.

Figure 20.2 Potential Resettlement Sites



Source: *Aureus, 2013*

Community Development Plan

Aureus is committed to developing measures and action plans in line with its policy to conduct its business activities in a manner that promotes sustainable development and social welfare in the areas in which it operates. Moreover, Aureus has committed to mitigate and manage the resettlement impacts associated with the development of the Project in accordance with the requirements of IFC PS with regard to the following:

- Improve, or restore, the livelihoods and standards of living of displaced persons;
- Ensure that affected vulnerable people receive additional assistance, if required; and
- Provide opportunities to displaced people to derive appropriate development benefits from the project.

Aureus commissioned independent consultants Digby Wells and Associates (Pty) to develop a Community Development Plan (CDP) for the resettlement of affected households and communities. This plan was completed in December 2012 and was subsequently approved by the Liberian EPA in January 2013.

Based on the development opportunities assessed, a list of potential CDP projects was identified, including:

- Vegetable production;
- Seedling and expanded plant nursery;
- Vegetable pack house;
- Lowland rice production, drying and milling;
- Cassava production and value-adding demo;
- Mobile cassava grater business;
- Corn production and milling demonstration unit;
- Chicken broiler unit;
- Plantain and banana production;
- Abaca and banana fibre production;
- Pineapple production and value-adding;
- Cashew nut production and value-adding; and
- Coconut production.

Where feasible, the final CDP will be expanded to incorporate the community development aspects of the BMMC MDA. The MDA requires that operations shall be carried out in a manner that is consistent with the continuing economic and social viability of centres of population that have formed, and/or may form as a result of the NLGM operations during the term of the MDA.

Socio-Economic Benefits of the Project

The development and operation of the Project will have both positive and negative impacts on the socio-economic structure of the Project area and its environments, as well as impacts at a District and National level.

The positive impacts will relate mainly to the economic advantages which will have immediate and long-term benefits for the socio-economic environment. This will be achieved in various ways at National, District and Local levels through the payment of taxes and royalties, increased employment opportunities, training, purchase of goods manufactured and supplied in Liberia, cash compensation for farms, commercial opportunities and an improvement in local infrastructure by the establishment of the Resettlement site/town. The development of the Project will bring much-needed investment and development opportunities with consequent positive impacts on employment and the affected communities.

Negative impacts relate to the disruption of the local social dynamics and increased pressure on local infrastructure and resources, mostly due to the influx of people to the area.

20.8 Rehabilitation and Closure

A closure study was undertaken for the Project as part of the ESIA. This study focused on developing closure objectives for the proposed mine site, as well as producing a high level closure costing table for the site.

The overall closure goal for NLGM is to progressively re-instate native forest areas that are safe, stable and non-polluting, mimicking the current land use, and taking into account the unavoidable remaining mining residue and/or disturbances towards leaving behind a positive post-mining legacy. The Closure costs also reflect the conditions spelled out in the MDA, whereby the Government of Liberia take ownership of all associated project infrastructure.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following section summarizes the capital and operating costs of the Project based upon a conventional open-pit mining operation, a two-stage crushing process, ball milling and a CIL circuit. The plant design had been based on the treatment of 1.1 million tonnes per annum of ore.

The capital and operating costs presented below form the basis of the economic analysis performed in Section 22.

21.2 Operating Cost Estimate

21.2.1 Accuracy and Basis of Estimate

The mine operating costs are based upon discussions held with Aureus' preferred mining contractor who are an experienced operator in West Africa.

The process operating cost estimate was completed from a zero base. All labour, materials and consumables have been included in this estimate. The bulk of the inputs were generated by DRA based on the work of ALS Metallurgy, using quotations from typical suppliers as well as current data base information. The exceptions were general and administration costs that were determined by the Owners Team.

This estimate excludes the cost of transporting product materials from site.

21.2.2 Base Date and Exchange Rates

The base date of this operating estimate is March 2013. The following exchange rate has been used for this estimate:

- ZAR/US$: 8.84

21.2.3 Definitions of Costs

The costs in this estimate can be defined as all costs that will be incurred in the life cycle of the operation.

Fixed Cost: This component can be defined as the costs that will be incurred irrespective of production rates. These costs would typically include the following:

- Labour
- Lease Costs
- Environmental and Social

Variable costs can be defined as costs that are only incurred during production. It can be said that these costs are based on rates per tonnage and the total costs are incrementally incurred as production rates increase.

The costs would typically include the following:

- Reagents
- Maintenance Spares
- Diesel
- Tyres
- Oil
- Consumables
- Power
- Water
- Explosives

21.2.4 Plant Costs

The following section provides a description of the operating costs for the New Liberty Gold metallurgical processing facility. The following are examples of those items excluded from this estimate:

- All VAT, import duties and/or any other statutory taxation, levies and/ or national and local institutions
- Contributions to social programmes
- All owner's budget costs, head office, administration charges, payroll etc.
- Contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements
- All costs associated with grade control, blending and stockpile management. These costs are excluded from the plant estimates and included in the mining costs
- Final product transport, marketing and sales agreement costs.
- Tailings Facility Operational Costs

The basis for the estimated operating cost is given in each of the sections to follow. All costs are provided in United States dollars.

21.2.4.1 Labour

The labour costs have been based on the organogram for the plant, as presented in Figure 21.1 below. The cost for labour has been determined based on a labour cost model derived for the project using information from the owner's team.

Table 21.1 New Liberty Process Plant Labour Requirements

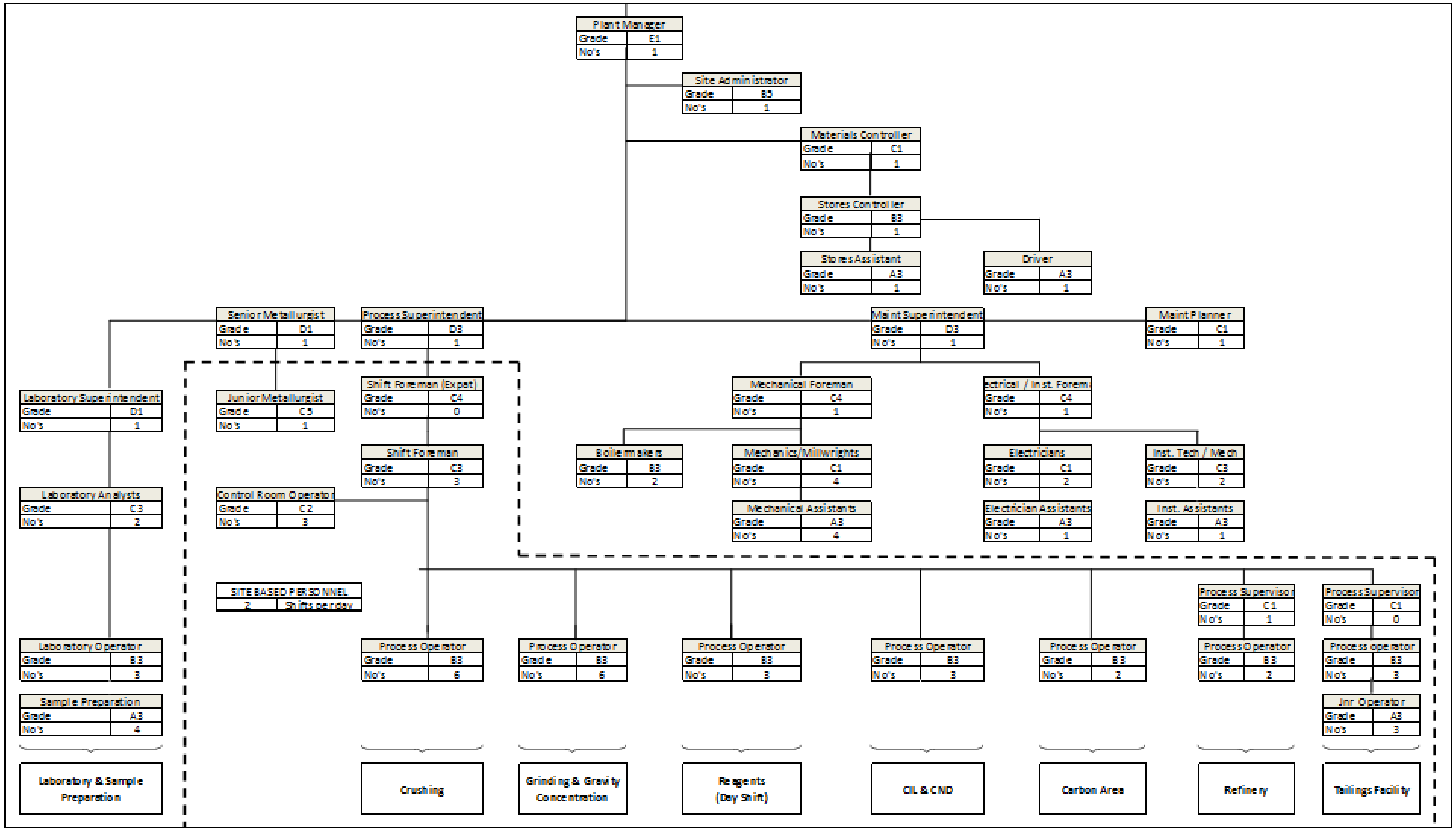
Personnel Complement Breakdown	Section	Responsibility	Grade	No.	Local/ Expat
Management					
Plant Manager	Management	Plant	E1	1	Expat
Maintenance Superintendent	Management	Engineering	D3	1	Expat
Laboratory Superintendent	Management	laboratory	D1	1	Expat
Process Superintendent	Management	Process	D3	1	Expat
Senior Metallurgist	Management	Process	D1	1	Expat
		Total		5	
Administration					
Site Administrator	Administration	Secretary	B5	1	Local
Materials Controller	Administration	Stores	C1	1	Local
Stores Controller	Administration	Stores	B3	1	Local
Stores Assistant	Administration	Stores	A3	1	Local
Driver	Administration	Driver	A3	1	Local
		Total		5	
Laboratory					
Laboratory Analysts	Services	Laboratory	C3	2	Expat
Laboratory Operator	Services	Laboratory	B3	3	Local
Sample Preparation Assistants	Services	Laboratory	A3	4	Local
		Total Laboratory		9	
Process Supervision					
Shift Foreman	Process	Plant	C3	3	Expat
		Total		3	
Process					
Control Room Operator	Process	Plant	C2	3	Local
Process Operator (2 shifts)	Process	Crushing	B3	6	Local
Process Operator	Process	Grinding & Gravity Concentration	B3	6	Local
Process Operator	Process	Reagents (Day Shift)	B3	3	Local
Process Operator	Process	CIL & CND	B3	3	Local
Process Attendant	Process	Carbon Area	A3	2	Local
Process Supervisors	Process	Refinery	C1	1	Local
Process Operators	Process	Refinery	B3	2	Local
Process operator	Process	Tailings Facility	B3	3	Local
		Total Process		29	
		Day Shift Process		8	
		Total per shift		11	
Maint Planner	Engineering	Plant	C1	1	Local
Mechanical Foreman	Engineering	Mechanical	C4	1	Expat
Mechanics/Millwrights	Engineering	Mechanical	C1	4	Local
Boiler makers	Engineering	Mechanical	B3	2	Local
Mechanical Assistants	Engineering	Mechanical	A3	4	Local
Electrical / Inst. Foreman	Engineering	Electrical	C4	1	Expat
Electricians	Engineering	Electrical	C1	2	Local
Electrician Assistants	Engineering	Electrical	A3	1	Local
Instrument Tech / Mech	Engineering	Instrumentation	C3	2	Expat
Instrument Assistants	Engineering	Instrumentation	A3	1	Local
		Total Maintenance		19	
		Total Staff Complement		70	
		Expat Complement		14	
		Local Complement		56	

The total cost for labour includes all production and engineering staff directly associated with the processing plant and laboratory, and covers a complement of 70 people. The total cost shown is a total cost to company inclusive of all allowances, medical contributions, life cover, training and expat contingent travel costs as provided by Aureus. The processing plant organogram is presented in Figure 21.1, where the number of employees and proposed grading band is shown.

Table 21.2 New Liberty Process Plant Labour Requirements

Employee Group	Complement	Annual Cost
Management	5	$806 961.50
Administration	5	$24 145.00
Laboratory	9	$260 722.69
Process Supervision	3	$59 265.00
Process	29	$256 815.00
Maintenance	19	$412 177.69

Figure 21.1 New Liberty Process Plant Organogram



21.2.4.2 Power

The total connected electrical load, inclusive of all standby units in the plant, is around 12.7MW by summation of all the equipment specified on the mechanical equipment list. The estimated average running load has been calculated using expected power draw from the equipment sizing calculations. The design operating power draw for the plant has been estimated at 8.1MW.

The plant power is to be generated by a diesel power generation plant. The power cost ($/kWh) for plant operations was calculated based on the quoted diesel consumption figures and pricing as presented in Table 21.3 below. A figure for maintenance costs for the operation of the power generation plant as determined by DRA was also included in the calculated power costs. The estimated calculated power costs are summarized in Table 21.4.

Table 21.3 Basis for Determination of Plant Power Costs

Basis for Determination of Power Costs		
Diesel Cost	$/L	$1.13
Diesel Consumption	g/kWh	204.2

Table 21.4 Power Cost Estimate

Power Requirements		**Optimized P95 75um**
Primary Crusher	kWh/t	0.369
Secondary Crusher (2 x HP500)	kWh/t	1.017
Ball Mill	kWh/t	19.62
Regrind Mill	kWh/t	6.69
Plant Other	kWh/t	16.78
Total Power	kWh/t	44.48
Total Power	**$/t**	**12.08**

21.2.4.3 Water Cost

Water consumption was calculated using the nominal mass balance as indicated on the Process Flow Diagrams (PFDs). Water can be sourced from a local river for the New Liberty project and there was no cost associated with this water supply. The water treatment costs (reagents and flocculant) for the treatment of potable water is insignificant and has been excluded from the analysis. The pumping costs for supplying water from the various storm water impoundments and the river supply system are included in the plant power costs.

21.2.4.4 Reagent Consumption and Costs

The following section provides a summary of the expected nominal reagent consumption rates, based on results obtained from the laboratory scale test work, vendor specifications and mass balancing. The prices include freight to port, and include all clearance charges and taxes that may be incurred. All reagent costs were quoted on the basis of deliveries to site being made ex Monrovia. A minimum of 60 days of stock will be held in a store on site. This will ensure that total stockholding on site will be equal to or exceed the typical sourcing lead times for relevant imported product. The unit costs are based on quoted supply costs as detailed in Table 21.5 below.

Table 21.5 Reagent Supply Pricing used for OpEx Determination

Chemical Reagent	Supplier	Operting Cost Basis
		$/t
Sodium Cyanide (NaCN Briquette 98%)	Nowata	3500
Sodium Hydroxide (NaOH Pearl 99%)	BK Afriquip	650
Sodium Metabisulphite (Na2S2O5 Solid)	Nowata	440
Copper Sulphate (CuSO4.5H2O 98%)	BK Afriquip	2200
Hydrated Lime (Ca(OH)2 85%-90%)	BK Afriquip	300
Lead Nitrate (PbnO3 Solid)	CN Mining Chemicals	2200
Flocculant (Magnafloc 10 /equivalent)	Protea Chemicals	3200
HCL (30%-33% solution in IBC containers)	Nowata	335
H2SO4 (98% solution in IBC containers	Nowata	445
Activated Carbon	Protea Chemicals	2397
Borax	Protea Chemicals	1748
Silica	Nowata	350
Sodium Nitrate	Protea Chemicals	1261

Table 21.6 Reagent Cost Estimate

Reagents		Optimized P95 75um
Sodium Cyanide	$/t	2.42
Caustic Soda	$/t	0.04
Lead Nitrate	$/t	0.06
Lime (Leach)	$/t	0.49
Lime (Detox)	$/t	0.10
HCL	$/t	0.10
H2SO4	$/t	0.45
SMBS	$/t	0.34
CuSO4	$/t	0.14
Flocculant	$/t	0.13
Carbon	$/t	0.06
Ferric Chloride	$/t	0.13
Smelthouse Consumables	$/t	0.04
Total Reagent	**$/t**	**4.50**

It is clear from the table the primary cost drivers for the plant, in respect of reagents, are Sodium Cyanide, Hydrated Lime, Acid and Sodium Metabisulphite (SMBS). Together these reagents contribute 82% of the total reagent costs.

21.2.4.5 Liners and Grinding Media

Crusher Liners

The maintenance cost for the jaw crusher has been based on replacing one liner set per year, while the cone crusher costs have been based on replacing two liner sets per year.

Mill Liners

The liner life for the ball mill was estimated using the abrasion index test results, mill dimensions and anticipated grinding media load. Unit costs (for full sets) were given by various mill suppliers in their tenders submitted for the project.

Table 21.7 Liner Cost Estimate

Liners		**Optimized P95 75um**
Jaw Crusher	$/t	0.03
Cone Crusher	$/t	0.45
Ball Mill	$/t	0.35
Regrind Mill (Estimated)	$/t	0.08
Total Liners	**$/t**	**0.91**

Grinding Media

The ball mill grinding media wear rates were determined by the DRA in-house milling consultant, while the vertimill media consumption figure has been estimated based on a consumption figure of 0.05kg/kWh as provided by Metso.

Table 21.8 Grinding Media Cost Estimate

Grinding Media		**Optimized P95 75um**
Ball Mill	g/t	510.0
VertiMill	g/t	334.7
Grinding Media	**$/t**	**0.95**

21.2.4.6 Diesel Consumption and Costs

The following section provides a summary of the expected nominal diesel consumption rates, based on diesel consumption figures from vendors for plant equipment that is diesel driven namely the elution heating system, carbon regeneration kiln and gold room furnace.

Table 21.9 Diesel Cost Estimate

Diesel		Optimized P95 75um
Diesel	$/t	0.37
Total	**$/t**	**0.37**

21.2.4.7 Mechanical Maintenance

An estimate for the plant maintenance costs was determined by summation of the vendor specified operational spares required for large critical equipment.

Table 21.10 Maintenance Cost Estimate

Equipment/Description	Optimized P95 Cost
Ball Mill Gear spray Lube System Spares	$2 880
Apron Feeder	$7 044
Interstage Screens	$99 870
Regen Kiln	$4 320
Electro Winning Cells	$14 464
Screens	$6 977
Linear Screens	
6 m^2	$20 218
9 m^2	$23 440
Gravity Concentration System	$14 296
Elution Heating	$6 651
Slurry, Spillage and Water Pumps	$318 583
Carbon Pumps	$28 511
Helical Pumps	$32 739
Double Diaphragm Pumps/Chemical Pumps	$13 882
Cyclone Cluster	$30 945
Magnet	$3 191
Ventilation	$2 145
Conveyors	$50 004
Annual Maintenance Costs	$680 162

21.2.4.8 Overall Plant Operating Cost

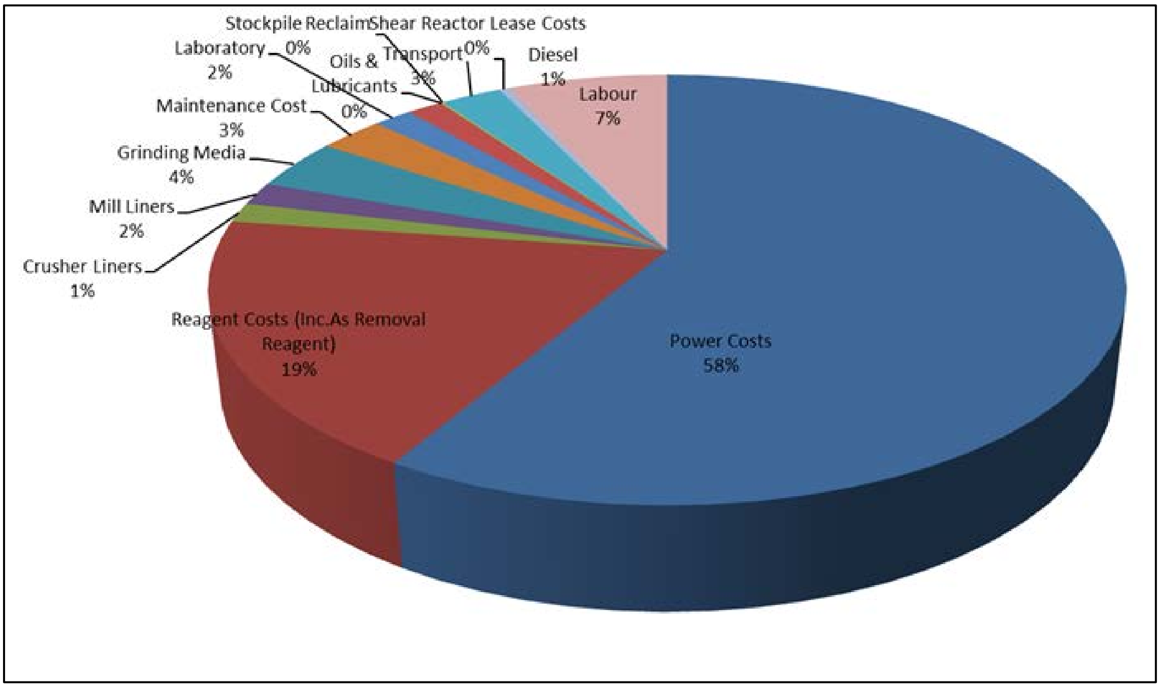
The overall plant operating cost estimate is shown in Table 21.11 below. The breakdown shows all the costs discussed above and includes costs for the laboratory and other miscellaneous items.

Table 21.11 Overall Operating Cost Estimate Summary

Variable Cost		**Optimized P95 75um**
Power Cost (Excell VertiMill)	$/t	10.26
Power Cost (VertiMill)	$/t	1.82
Reagent Cost	$/t	3.92
Reagent Cost (As Removal)	$/t	0.58
Crusher Liners	$/t	0.48
Ball Mill Liners	$/t	0.35
VertiMill Liners	$/t	0.08
Grinding Media (Ball Mill)	$/t	0.48
Grinding Media(VerteMill)	$/t	0.47
Maintenance Cost	$/t	0.62
Laboratory	$/t	0.40
Diesel	$/t	0.37
Transport	$/t	0.63
Shear Reactor Lease Cosrt	$/t	0.10
Labour	$/t	1.65
Total Estimated Plan OpEx (Excel G&A)	$/t	22.21

An allowance of US$0.36 / tonne ore feed has been included in the processing cost to cover ROM re-handle costs.

Figure 21.2 New Liberty Optimized Flowsheet Operating Cost Breakdown



21.3 Mining Operating Costs

Mining operating costs are discussed in Section 16. The mining contractor costs were based upon discussions held between Aureus and a number of mining contractors. The owner's costs for the mining operation were included in the Administration costs.

21.4 Capital Cost Estimate

The capital cost estimate has been developed based on the engineering work performed by DRA. Pricing was obtained for equipment via a formal enquiry process. Requests for quotations were generally sent to at least three suppliers, and their tenders formally adjudicated and signed off in the Definitive Feasibility Study. The selected vendors were contacted and their costs revalidated for this estimate base dated March 2013.

The general approach to estimating was to measure/quantify each cost element from the engineering drawings, process flow diagrams (PFDs), mechanical equipment list, motor lists, cable schedules, and instrument lists.

The estimate for the plant has been based on an assumption of a continuous engineering, procurement and construction effort with no interruption of the implementation programme after funding approval has been obtained.

Table 21.12 Capital Cost Estimates

	US$ million	%
Processing plant	56.0	41%
Infrastructure – earthworks and buildings	26.0	19%
Indirect costs – EPCM fee, pre-production costs, consumable and spares	27.2	20%
Initial mining pre-strip	6.3	5%
Tailings dam construction	7.2	5%
Marvoe Creek diversion channel	6.0	4%
Village relocation	3.5	3%
G&A and owner costs	3.8	3%
TOTAL	**136.0**	**100%**
Contingency	13.6	10.0%

Deferred CapEx will be incurred following the commencement of production. Sustaining capital includes the mine closure costs. The mine closure costs cover environmental aspects at the mine and process plant sites. Mining operations will be undertaken on a contract basis. The diesel generators, fuel farm and mining fleet equipment are covered by lease agreements over the LOM. The deferred capital costs are summarized in Table 21.13.

Table 21.13 Deferred Capital Cost Estimates

	US$ million
Sustaining capital and mine closure	5.7
Diesel Generators, fuel farm and mining fleet over LOM	77.5
Mine pre-strip	18.0

21.4.1 Earthworks

The earthworks estimate can be divided into five main sections:

- Marvoe Creek Diversion Channel (MCDC)
  - Epoch Resources was appointed to do a preliminary design and quantity take off for the diversion of the Marvoe Creek around the open cast mining area.
- Tailings Storage Facility (TSF)
  - Epoch Resources was appointed to design the tailings facility and produce a bill of quantities.
- Plant Terrace
  - DRA produced the bill of quantities for the plant terrace. The earthworks were quantified by DRA from the Block Plan and General Arrangement Drawings. The quantities shown on the drawings were based on computer modelling of the proposed terrace and surveyed ground conditions.
- Crushing Facility

The following will be procured from a local crushing facility and inported to site:

- Crusher Sand
- 6 mm stone
- 19 mm stone
- G5 Material
- Rip Rap

A combined Epoch and DRA bill of quantities was issued to major civil construction companies to obtain rates and overall costs.

21.4.2 Civil Works

The civil works bill can be divided into two sections:

- Infrastructure

Infrastructure quantities were quantified from the drawings and layouts by DRA.

- Processing Plant Civils

Civil quantities were estimated based upon the raft foundations and preliminary sizing from a Professional Engineer. This sizing was based on a preliminary assessment soil conditions for laterite soils based on site investigations by a Geotechnical Engineer. Subsequent laboratory testing and calculations have confirmed the estimating assumptions to be correct. A bill of quantities was issued to major civil construction companies to obtain rates and overall costs.

21.4.3 Building Works

A building contractor will supply modular buildings. The foundations for these buildings were included in the civil works bill of quantities. No laboratory buildings were included in the estimate as these will be containerised.

21.4.3.1 Structural Steelwork Supply and Erection

Steelwork quantities for major and minor structures were estimated by DRA from General Arrangement and layout drawings produced by DRA. A preliminary bill of quantities was produced by DRA to cover all steelwork and ancillary equipment (sheeting, grating, stair reads, hand railing, sheeting etc.) and this was issued on enquiry to prominent steelwork fabricators/erectors. A commercial adjudication was carried out and one of the quotations was selected, based on cost, but more importantly based on experience and capability of undertaking work in West Africa. The estimate was populated with the final steelwork quantities from DRA and the rates from the chosen contractor. Structural steel preliminary and generals (P&Gs) have been based on quoted P&G's.

21.4.3.2 Platework and Lining

Platework and lining items were quantified by DRA from the equipment list, PFDs, layouts and drawings. Rates for platework and liners were obtained from the fabrication/erection contractor used for the structural steelwork. The plate work P&Gs were based on quoted P&G's

In accordance with the Engineering Design Criteria, all steelwork and platework will be painted. The rates for painting were quoted by the fabrication contractor and these have been included in the estimate as part of the fabrication rate. The cost of galvanizing or painting for piping is included in the piping price. An allowance has been provided for the touch-up of steelwork and platework on site after installation. The Corrosion Protection P&Gs have been included in the fabrication P & G's.

21.4.3.3 Mechanical Equipment

From the Equipment List and PFD's, an enquiry register was compiled and enquiries were issued to multiple vendors for each piece of mechanical equipment. Commercial and Technical adjudications of the quotations were carried out and the pricing information for each item was incorporated into the estimate. The erection cost for the mechanical equipment was based on rates received from the steelwork and plate work fabrication/erection contractor. For minor and/or ancillary items the erection costs were taken from previous quotations and/or the DRA historical data base. The mechanical

P&Gs were taken at 61% of the erection component only, as quoted by the steelwork and plate work contractor.

The design of all belt conveyors was carried out by the DRA engineering department in accordance with the belt profiles as depicted on the general arrangement drawings, the duty to meet the process requirements and the design to meet the general engineering design criteria. Calculations for each conveyor were carried out and the mechanical equipment components and steelwork content were quantified. Vendor quotations for the mechanical equipment were used and the steelwork fabrication rates were utilized from the steelwork vendor. The erection cost for the mechanical equipment and steelwork was based on rates received from the steelwork and platework fabrication/erection contractor. For minor and/or ancillary items the erection costs were taken from previous quotations and/or the DRA historical data base.

21.4.3.4 Piping and Valves

The cost for the process plant piping was derived from a detailed Piping Material Take Off done by way of preliminary routings of all lines shown on Revision F of the P&ID's and using the Plant Area GA's to quantify lengths and quantity of fittings. All actuated valves were costed as part of the Control and Instrumentation Bill of Quantities. Overland and non-process piping was quantified and priced in the infrastructure section of the estimate. Piping P&G's were assigned at 61% of the total supply and installation cost.

21.4.3.5 Electrical and Instrumentation

It is anticipated that an 11kV, 9MW, diesel driven Build, Own, Operate and Transfer (BOOT) power supply will be operated on a deferred terms agreement.

A Bill of Quantities was compiled for the electrical and instrumentation scope of works. This was forwarded to five installation contractors to be priced. The tenders were compared and adjusted where necessary to account for excluded equipment. The total cost for the supply, installation and P&G's portion for this works was included in the estimate.

The medium voltage switchgear requirements were taken off of the main single line diagram, these take-offs were issued to vendors to price the works.

Take-offs were performed on site layout drawings for all medium voltage cables and overhead lines sized according to the load schedule. Rates for the supply and installation of these cables were included in the installation contractor's scope of works. The contractors' tenders were evaluated on price, technical adherence and experience in similar environments. The selected vendors' rates were included in the estimate.

Power transformers were sized based on the load schedule and the key single line diagram. Various vendors were issued with enquiries for the supply of the transformers.

Motor control centre sizes were based on the Motor List and available container space, three vendors tendered on the supply of the works, including the control room container.

Mechanical layout drawings of plant and infrastructure buildings were used to design a lighting scheme. This design was then priced based on supplier prices and the installation contractors' installation rates. Both of these rates were included in the estimate.

A medium voltage isolator was priced for the ball mill motor and included in the estimate.

Earthing and lightning protection specialists were approached to provide a price to conduct a soil resistivity survey, complete a design and to perform an audit on the completed system. The price obtained for this work was combined with the contractors' rates to install earthing materials as per the quantities defined by estimated requirements.

A high level design for the electrical and instrumentation racking requirements was completed and take-offs made from site layouts and structural drawings. These requirements were then priced based on the installation contractors' rates for supply and installation.

A low voltage cable schedule was compiled from the load schedule and MCC location drawings, take-offs were included for the lighting and small power cables required for the plant and infrastructure. The cable schedule was then priced using the installation contractor's rates.

A single 250kVA diesel generator has been allowed for the construction phase of the project. This will be distributed to four construction distribution boards for reticulation to the various areas of the plant. The contractors will provide their own "Ready Boards" for low voltage distribution. The electrical and instrumentation installation contractor provided rates and P&Gs for this work to be performed before the full electrical and instrumentation team mobilizes.

The plant control and instrument quantities were taken from the plant P&IDs. Instruments were grouped by relevance and forwarded to various suppliers to price. The tenders were adjudicated on price, technical adherence and past work.

General Arrangement drawings were compiled based on the IO List and site layout drawings. These were then sent to panel manufacturers for the pricing of junction boxes.

Multicore cable sizes and quantities were calculated based on the IO requirements for each junction box, lengths were measured from site layout drawings. Take-offs were done for each instrument based on the IO list.

The PLC and control hardware design was done according to the IO requirements and plant layout. These requirements were forwarded to vendors of cabling and terminators to be priced. The vendor was selected based on price, technical adherence, previous experience and support availability.

A single SCADA software vendor was preselected due to previous experience and cost saving opportunities. Two distributors were approached to offer this SCADA solution for the project.

A single supplier was chosen for all of the IT equipment on the plant as preferential pricing was available to DRA.

A containerized control room layout was completed based on the plant requirements.

An enquiry was compiled based on similar requirements from previous projects while taking into account the specific needs of this site. Included in the specification was the requirement for fire protection and time and attendance tracking. Vendors tendered complete offers to design, install and commission the whole of the security works.

The requirements for satellite communications were taken from previous projects with the site specific conditions taken into account.

A fibre optic cable schedule was compiled based on the location of the MCC's and control room. This schedule was then forwarded to two contractors to price the splicing and OTDR testing of the cables.

21.4.3.6 Transportation

Transport for mechanical equipment, steelwork and platework has been priced on a tonnage or volume basis using transport and shipping rates from the selected logistics service provider.

21.4.3.7 Project and External Services

The EPCM costs cover the project management, detailed engineering, procurement and construction management costs directly associated with the implementation of the project.

Project Management and EPCM costs relating to the Plant have been based on EPCM man hours calculated by DRA. The man hours were calculated using the implementation programme and project team organogram, which also includes reimbursable and sundry costs as well as a contractor's fee. Input was obtained from the engineers responsible for the disciplines and the drawing offices for the number of drawings and engineering hours required for the execution of the project. The external services costs allow for the utilization of external consultants in the fields of Tailings Storage Facility and Marvoe Creek diversion, mass flow and silo design, quantity surveying services, geotechnical investigation, miscellaneous consultants and Hazop consultants.

Allowance has been made under disbursements for travel and accommodation based on the manpower histogram for implementation, the requirement for travel for inspections, printing and general disbursements.

21.4.3.8 Pre-production

Pre-production costs include all G&A capital costs, contractor camp costs, owner's costs and pre-production stripping costs for approximately 2.4 million tonnes of material.

21.4.3.9 Consumables and Spares

The requirements for the first fill and for consumables have been calculated based on DRA's estimate of the requirements. The first fill and consumable allowances apply to lubricants and grinding media and these have been costed and listed in the estimate. Spare parts costs have been included in the estimate based on DRA's understanding of the requirements to cover commissioning, strategic and operational spares.

An opening stock of two months of process reagents has been allowed for.

The mechanical spares costs have been derived as a factor of the mechanical equipment supply cost where they were not directly quoted by the selected vendors. The electrical and instrumentation spares cost has been derived as percentage (7%) of the electrical and instrumentation supply cost and the piping and valves spares cost have also been derived as a percentage (9%) of the piping and valves supply cost.

21.4.3.10 Village Relocation

The village relocation costs have been based on the requirement to relocate approximately 325 households (approximately 1,800 individuals).

21.4.3.11 Preliminary and General Costs (P&Gs)

P&G costs include all contractors' overheads such as contractual requirements (safety, sureties, insurance, etc.), the site establishment and the removal thereof, and company and head office overheads. They also cover supervision, travel to and from the site, contractor supplied temporary facilities, offices and lay-down areas, tools and contractors' equipment. P&Gs costs have been allowed for based on quotes from selected contractors of the works to be executed.

21.4.4 Assumptions

The following assumptions have been made in the preparation of this estimate:

- All material and equipment purchases and installation sub-contracts have been competitively tendered on a lump sum basis.
- The Project would proceed on an EPCM basis as per the execution programme.

22 ECONOMIC ANALYSIS

22.1 Economic Model

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

22.1.1 General Assumptions

The financial model is pre-finance, allows for working capital and is based on a detailed analysis of gold processing throughput as detailed in Section 16. The financial model is based on the following assumptions:

- Currency base is the US$.
- The financial model is in real Q2 2013 terms.
- A discount rate of 5%.
- The financial model uses a flat gold price of US$ 1,400/oz across all periods.
- Royalty is 3% of net revenue, the financial model does not account for the Republic of Liberia's retention of a free of charge equity interest in Bea's operations equal to 10% of its authorized issued and outstanding share capital without dilution (i.e. a 10% "carried interest").
- The financial model includes an estimated US$60m of capital expenditure (sunk costs) prior to the start of project execution mine construction.
- No contingency costs have been allowed for in the capital cost estimates.
- Cashflow forecasts are calculated on a quarterly basis.

All mining and processing tonnages and grade are as shown in Table 16.1. Gold recovery has been assumed to be 93%.

22.1.2 Project Economics

A net present value has been calculated for the Project through the application of Discounted Cash Flow (DCF) techniques to pre-financing cash flows derived from the inputs and assumptions presented in this and previous sections of the report. All figures are presented in Q2 2013 real terms.

For the base case analysis a flat gold price of US$1,400 has been used.

A government royalty of 3% of net revenue has been assumed. The financial model is reported on the basis of 100% of the Project, with no consideration of the free carried interest. The model assumes a corporation tax rate of 30% which is taken from the recent amendments made to the Revenue Code of Liberia Act of 2000. There is a possibility that this corporation tax rate may be revised, which may result in a reduction of the corporation tax rate to 25%. This would provide additional upside on the post-tax economics presented in this report.

A summary of cash flow modelling is presented below in Table 22.1 with the annualized cash flow model shown in Table 22.3. The average life-of-mine cash cost per ounce of gold is estimated at US$668 with an expected pre-tax IRR of 29% and a pre-tax NPV of US$230 million using a 5% discount rate .The expected payback period for the Project is 3.4 years.

Table 22.1 Cash Flow Modelling Summary

Description	Units	Project Totals/Averages
Recovered gold	koz	859
Mill processing life	Years	8
Net smelter revenue	US$M	1,163
Operating costs	US$M	574
Net operating cash flow	US$M	589
Initial capital costs	US$M	136
Net pre-tax cash flow	US$M	353
Pre-tax NPV (5%)	US$M	230
Pre tax IRR	%	29
Post-tax NPV (5%)	US$M	165
Post tax IRR	%	24
Payback	years	3.4
Average cash cost per ounce	US $/oz	668

As a further sensitivity the economic assessment was repeated assuming a flat gold price at a value close to the price at the effective date and at US$1,500/oz. This is compared with the initial analysis in Table 22.2.

Table 22.2 Gold Price Sensitivity

Gold price	US $/oz	1,300 flat	1,400 flat	1,500 flat
Gross revenue	US$ M	1,116	1,202	1,288
Net smelter revenue	US$ M	1,080	1,163	1,247
Net operating cash flow	US$ M	505	589	673
Net pre-tax cash flow	US$ M	269	353	436
Pre-tax NPV (5%)	US$ M	166	230	293
Pre-tax IRR	%	23	29	34
Post-tax NPV (5%)	US$ M	119	165	210
Post-tax IRR	%	19	24	28
Payback	years	3.9	3.4	2.9

Table 22.3 Annualized Cash Flows

	Units	Totals	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Ore mined	kt	8,494	-	329	741	1,136	1,134	1,198	1,261	1,079	1,267	349	-
Waste mined	kt	131,486	-	2,095	22,136	24,414	24,486	24,353	19,250	9,396	5,071	285	-
Strip ratio	x	15.5	-	6.4	29.9	21.5	21.6	20.3	15.3	8.7	4	0.8	-
Ore processed	kt	8,494	-	-	1,017	1,109	1,112	1,109	1,109	1,109	1,112	815	-
Grade	g/t	3.4	-	-	3.1	3.6	3.2	4	4	3.5	3.3	2	-
Gold contained	koz	924	-	-	101	127	115	142	142	126	118	52	-
Gold recovered	koz	859	-	-	94	118	107	132	132	117	110	48	-
Revenue													
Gold price	US$ / oz	1,400	-	-	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	-
Gross revenue	US$ '000	1,202,460	-	-	131,573	165,369	149,917	184,909	185,054	164,442	153,918	67,279	-
Freight and refining	US$ '000	(-2,989)	-	-	(-325)	(-412)	(-372)	(-463)	(-464)	(-409)	(-382)	(-162)	-
Royalty	US$ '000	(-35,984)	-	-	(-3,937)	(-4,949)	(-4,486)	(-5,533)	(-5,538)	(-4,921)	(-4,606)	(-2,014)	-
Net revenue	US$ '000	1,163,487	-	-	127,310	160,008	145,059	178,913	179,053	159,112	148,930	65,103	-
Operating Costs													
Mining costs	US$ '000	(-328,833)	-	-	(-42,989)	(-57,609)	(-58,668)	(-61,461)	(-47,728)	(-27,588)	(-11,536)	(-21,254)	-
Processing costs	US$ '000	(-191,780)	-	-	(-22,960)	(-25,046)	(-25,116)	(-25,046)	(-25,046)	(-25,046)	(-25,116)	(-18,406)	-
General and Administrative expenses	US$ '000	(-53,108)	-	-	(-6,596)	(-6,896)	(-6,906)	(-6,896)	(-6,896)	(-6,896)	(-6,906)	(-5,117)	-
Total operating costs	US$ '000	(-573,722)	-	-	(-72,545)	(-89,550)	(-90,690)	(-93,402)	(-79,669)	(-59,530)	(-43,558)	(-44,777)	-
Capital costs													
Capital and deferred capital costs	US$ '000	(-237,465)	(-66,473)	(-69,603)	(-19,763)	(-14,060)	(-13,360)	(-10,360)	(-10,360)	(-13,360)	(-10,360)	(-7,770)	(-2,000)
Pre-tax cash flow		**352,300**	**-66,473**	**-69,603**	**35,002**	**56,399**	**41,010**	**75,151**	**89,024**	**86,222**	**95,013**	**12,556**	**(-2,000)**

22.1.3 Taxes and Royalties

Based on the base case of a gold price of US$1,400 per oz the government of Liberia will receive corporate tax revenues of US$87 million and gold royalties of US$36 million.

22.1.4 Project Sensitivities

Table 22.4 illustrates an analysis of the project NPV sensitivity to variations in gold price, operating cost and capital cost estimates used in the base case.

Table 22.4 Project Sensitivities

Sensitivity	NPV 5% Discount Rate	Variance to Base Case
Gold Price	US$m	%
+10%	319	+39
-10%	141	-39
Initial Capital Costs		
+10%	218	-5
Operating Costs		
+10%	184	-20
Grade		
-10%	141	-39

The financial model sensitivities indicate that the project is economically robust and when ranked shows that the Project is most sensitive to gold price or grade and least sensitive to initial capital cost variations.

AMC has verified that the financial model inputs reflect accurately the technical and financial costs reported in the study.

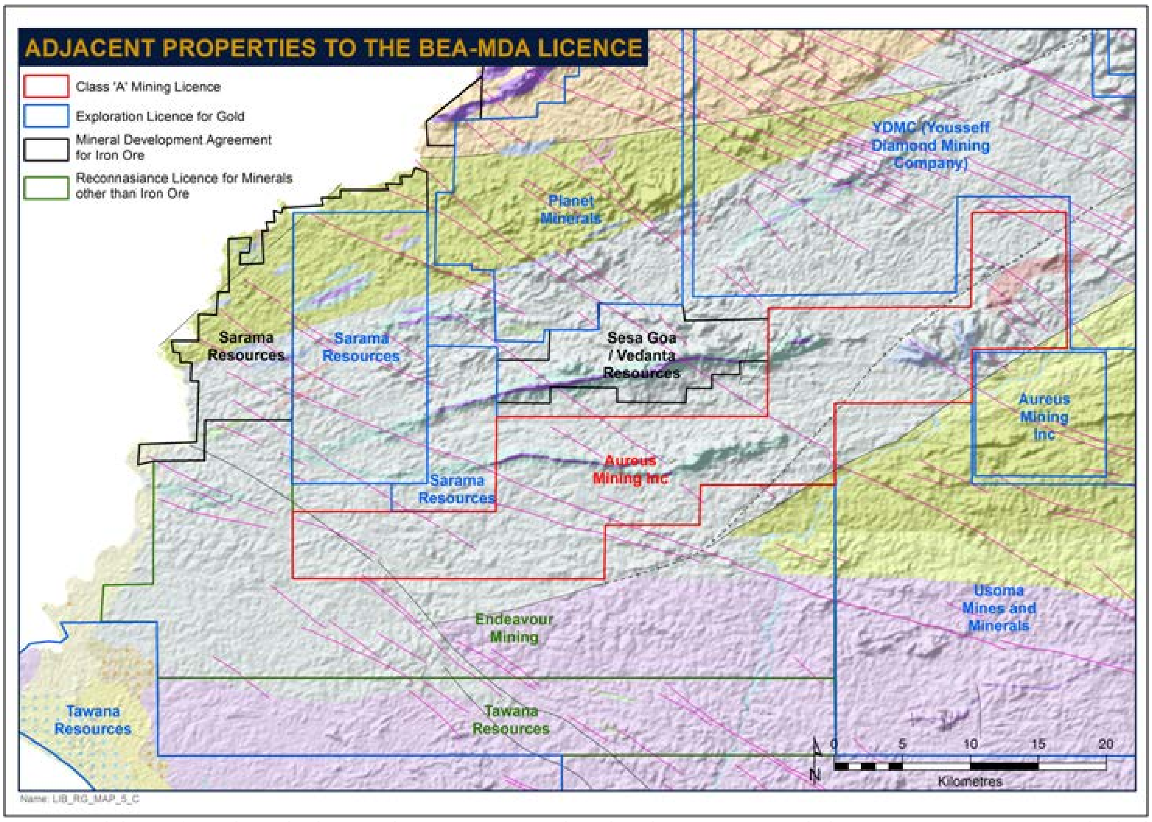
AMC has reviewed the basis of the technical assumptions applied to the economic assessment, together with the operating and capital cost estimates and they are considered appropriate support the definitive feasibility study.

23 ADJACENT PROPERTIES

23.1 Overview

The properties adjacent to the Bea-MDA licence area are illustrated in Figure 23.1.

Figure 23.1 Adjacent Properties to Bea-MDA Licence



Source: (*Ministry of Lands and Mines and Energy 2011 and exploration company websites*) Geology from *US Geological Survey, 2007* (Reproduced by: *Aureus, 2013*).

The immediate property neighbours from the most recent April 2011 update of the Mineral Land holding map of the Ministry of Lands Mines and Energy and various exploration/mining company websites are shown in Table 23.1.

Table 23.1 Table of Adjacent Licence Holders

Licence Holder	Licence type	Commodity
Youssef Diamond Mining Corp.	Mineral Exploration	Diamonds
Usoma Minerals and Mines	Mineral Exploration	Gold
Archean Gold Limited	Mineral Exploration	Gold and Base metals
Endeavour Mining	Mineral Exploration	Gold
Samara Resources	Mineral Exploration	Gold and Base metals
Vedanta Resources	Mineral Exploration	Iron Ore

Since the publication of this government data, Aureus Mining Inc. has acquired an exploration licence, covering area 72, from Archean Gold Limited, as announced on 21 September 2011.

There is no publicly available data from Usoma Minerals and Mines, or the Youssef Diamond projects.

23.2 Archean Licence

Aureus has a 100% interest in the Archean licence, within which Aureus has conducted geological mapping (Figure 23.2) and completed a 27 hole (4,293 m) diamond drilling programme on the Leopard rock deposit (Figure 23.1 and Table 23.2), and the results were announced on June 12, 2012. The diamond drilling cores demonstrate that the gold mineralization is associated with disseminated pyrite, pyrhotite and arsenopyrite, located within sheared and altered ultramafic rocks at the contact with metabasalt rocks. The ultramafic and metabasalt rocks have been intruded by granitic dykes. Geologically, Leopard Rock is thought to be the continuation of the same system seen at Ndablama and continuity between the two has been demonstrated by a ground IP survey (Figure 23.3).

Figure 23.2 Leopard Rock Geology and Drilling Location

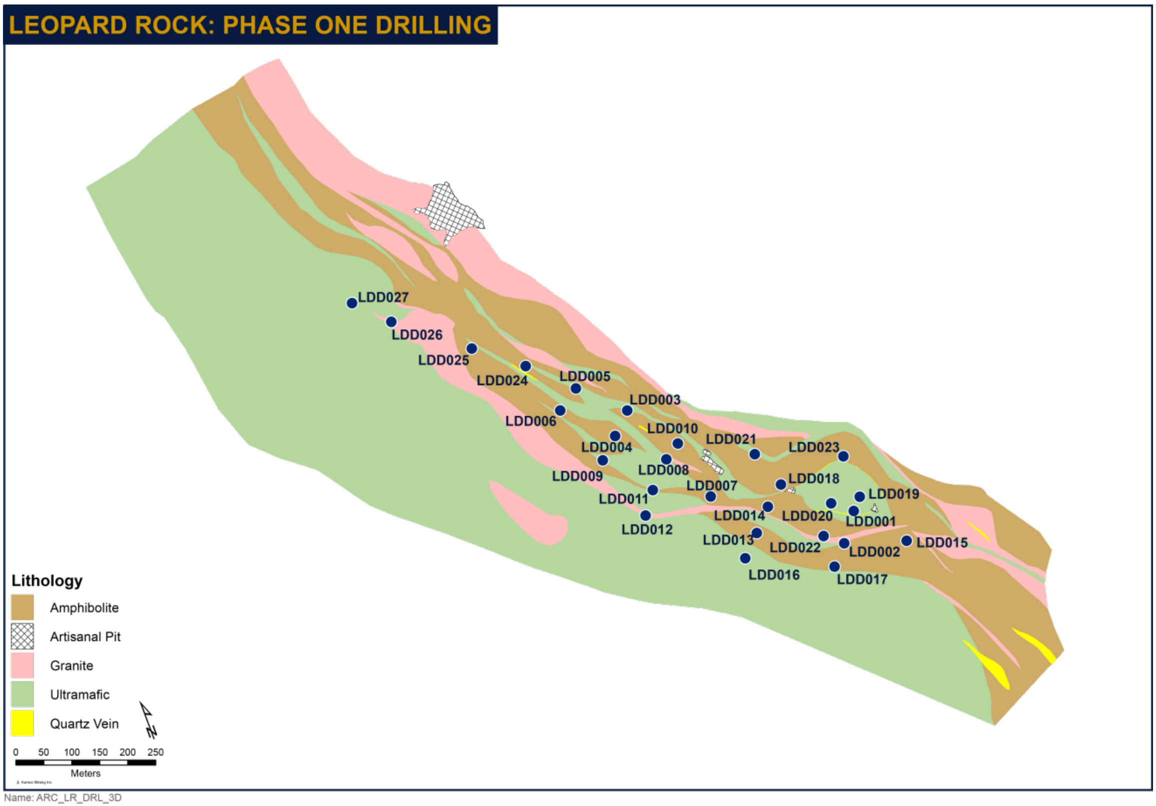
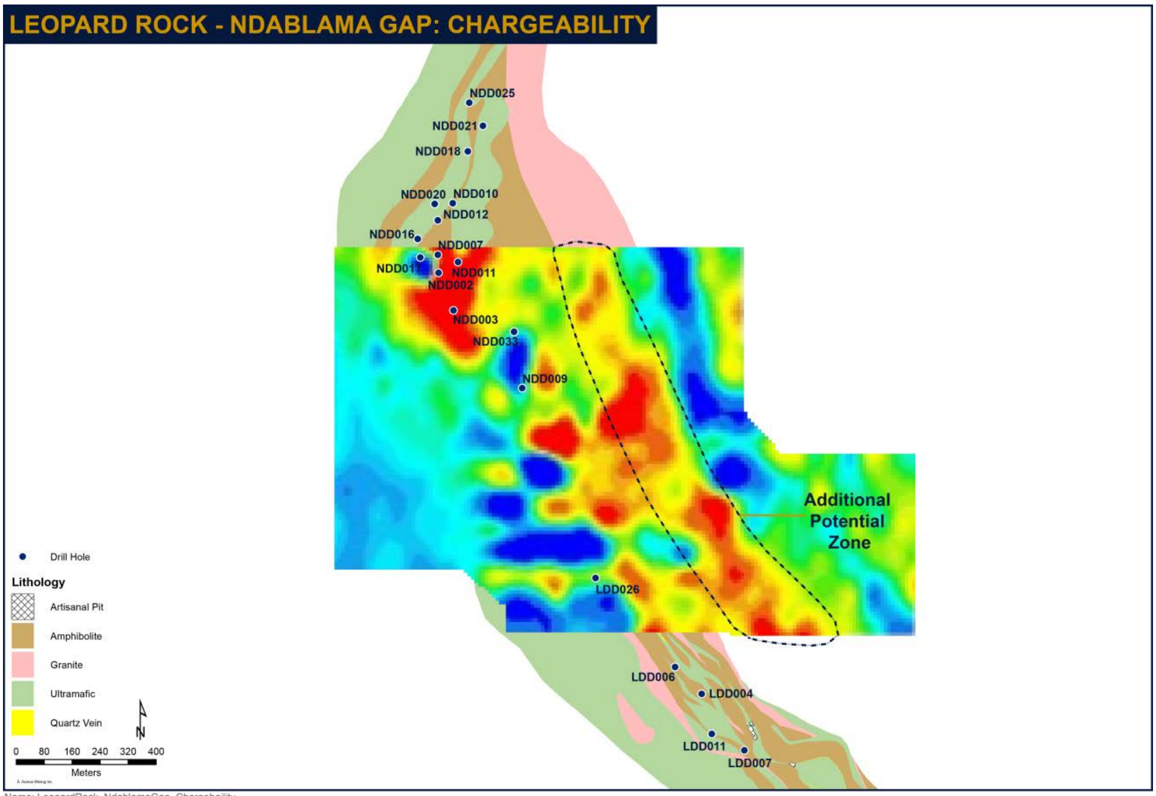


Figure 23.3 Leopard Rock - Ndablama Gap Ground IP Survey



Name: LeopardRock_NdablamaGap_Chargebaility

Table 23.2 Leopard Rock Diamond Drillhole Results

Hole ID	From (m)	To (m)	Intersection Length (m)	Au (g/t)
LDD001	2	22	20	1.9
and	52	61	9	1.9
including	56	59	3	5.0
LDD002	52	54	2	0.6
LDD004	79	83	4	5.5
LDD001	2	22	20	1.9
and	87	91	4	17.6
LDD005	2	10	8	1.0
and	42	44	2	10.4
LDD006	24	35	11	1.7
and	39	43	4	1.1
and	56	59	3	9.5
LDD007	26.6	33	6.4	1.0
and	45	46	1	1.4
LDD008	40.4	45	4.6	0.7
and	51	54	3	0.7
and	74	78	4	1.9
LDD009	77	84.2	7.2	0.5
and	107	113	6	9.4
including	107	111	4	13.9
LDD010	21.5	26.6	5.1	0.7
LDD011	67	80	13	0.5
and	116.5	126	9.5	1.3
LDD012	102	106	4	0.9
LDD013	48	74	26	0.6
including	48.6	52	3.4	1.5
including	55	63	8	0.8
LDD014	15	25.3	10.3	1.0
and	35	39	4	4.8
and	56	58	2	1.8
LDD016	83.7	98.2	14.5	0.8
including	93.2	95.2	2	4.4
and	115.2	117.2	2	1.2
LDD017	86.6	103	16.4	0.7
and	126	131	5	1.0
LDD019	40.8	50	9.2	1.4
LDD020	57	64	7	1.0
LDD022	71.2	78.6	7.4	0.6
and	81.6	85	3.4	0.5
and	92.5	108.7	16.2	0.9
LDD023	0	4	4	0.5
and	17.1	23.3	6.2	0.4
LDD024	0	11.5	11.5	1.0
LDD026	67	77	10	2.8

Assay grade data is un-cut.
NSV - LDD003, LDD015, LDD018, LDD021, LDD025 and LDD027Values shown in italics previously reported in February 2012

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

24.1.1 Introduction

The overall Project implementation schedule is based on the assumption that first gold will be produced in December 2014. The Project execution schedule reflects the work required, from detailed engineering at the planning stage, to the construction phase and then onto the commissioning and production phases.

The Project major milestones include the following:

- EPCM contract award
- Mining Contract award
- Long lead items ordered in H2 2013
- Commencement of construction, specifically earthworks and civils
- First gold pour in December 2014.

Aureus intends to appoint DRA Mineral Projects as an independent engineering company to execute the Project on an Engineering, Procurement and Construction Management (EPCM) basis.

24.2 Execution Strategy and Owner’s Team

24.2.1 Project Manager

The execution of the construction of the Project will be led by Thinus Strydom, Aureus GM Construction. The EPCM contractor and its project manager will report directly to the GM Construction.

24.2.2 Owner’s Team

The Owner’s Team has been created from a combination of Aureus’ existing employees and recent new appointees. The specialist technical, operational and project execution skill sets required for the Owner’s Team include:

- Project engineering and mining engineering – monitoring and approving EPCM contractor’s design and engineering work.
- Electrical, construction, earthworks and procurement – monitoring and approving the EPCM contractor’s work.
- Geology, Mining engineering and Survey
- Environmental and health and safety
- Community relations
- Government liaisons for permitting and legislative matters
- Back office support, including, but not limited to, accounting and legal matters.

24.3 EPCM

24.3.1 Engineering

The EPCM Contractor will be responsible for the following engineering aspects of the Project execution:

- Project specifications
- Design criteria
- Plant and infrastructure designs
- Data sheets
- Drawings
- Technical reports
- Engineering schedules.

24.4 Procurement

The EPCM Contractor is also responsible for the following procurement aspects of the Project execution:

- Generation of enquiry documents
- Obtaining and control of quotations
- Preparation of commercial adjudications
- Bid clarification and negotiations
- Preparation of orders and contract documentation and any modifications
- Expediting
- Delivery
- Quality assurance and inspections
- Freight forwarding control
- Preparation of control documents and schedules.

24.4.1 Construction Management

The EPCM Contractor is responsible for the following procurement aspects of the Project execution:

- Performing required scope of work in accordance with timetable and budget
- Adherence to requisite quality and workmanship standards, including occupational health, safety and environmental practices.

24.5 Commissioning

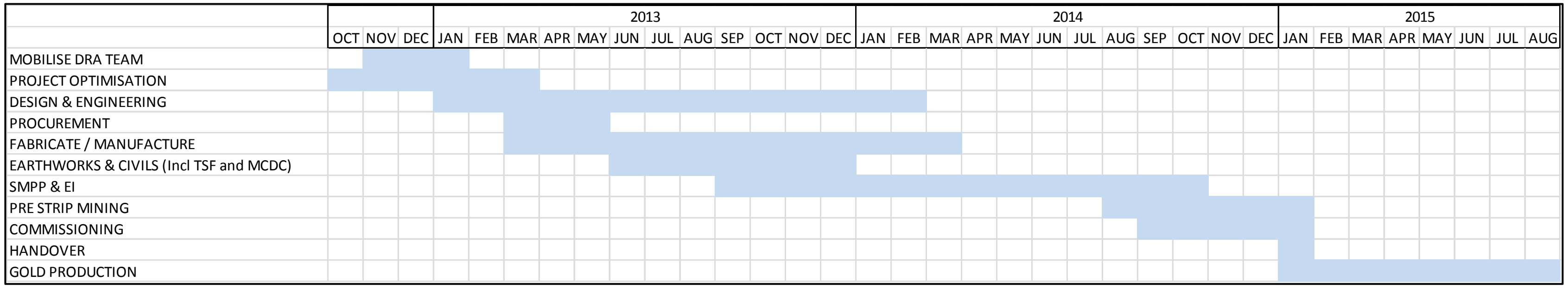
The commissioning approach will be based on four basic phases as described in a typical enquiry document:

- Pre-commissioning
- Cold commissioning
- Hot commissioning
- Training and operational assistance period.

24.6 Project Schedule

The Project schedule is summarized in Figure 24.1.

Figure 24.1 Project Schedule



25 INTERPRETATION AND CONCLUSIONS

25.1 Exploration

In spite of some adverse working conditions, the various campaigns of exploration conducted on the licence area have generally exhibited the characteristics of comprehensive and systematic evaluation programmes, with sound field professional and technical input. The quality of some work has varied over time and by technical area, and selected comments are made below concerning observed deficiencies, and opportunities for improvement are noted in the recommendations section.

There is evidence that the exploration work has been diligently undertaken and there has been a willingness to implement changes where potential improvements have been identified.

Exploration at the Project has confirmed the persistence of gold mineralization, including concentrations of potential economic value, within defined zones that extend from known surface occurrences in both artisanal workings and exploration trenches down to drill intersections more than 500 m below surface.

25.2 Drilling

Diamond core drilling over a series of campaigns has consistently intersected gold mineralization in a broad zone representing a predominantly southerly-dipping schist belt remnant. Within this stratigraphic interval, the defined concentrations of higher grade mineralization have been intersected in what has become an increasingly predictable distribution with each campaign. The 2011/2012 infill drilling campaign further verified the general character of the mineralization, and provided increased levels of confidence in local interpretations and grade estimates in the upper portions of the mineralized zones.

In 2010 it was recognised that the nearer-to-surface mineralization (weathered and upper fresh horizons) had been inadequately tested and the recommendations for further drilling of this interval were followed up by Aureus in 2011. Consequently the density of data in the upper levels has been markedly improved. This includes the long-recognised ‘gap’ between Kinjor and Marvoe which now appears to be straddled at depth by an extension of the previously identified Marvoe hanging wall mineralization.

Inherent continuities in both geometry and grade are variable across the deposit, with corresponding spatial variations in confidence levels. This has been exacerbated by problems with achieving uniform drill intersection spacing due to both drill site access difficulties and drillhole deviations. Nonetheless, the overall increase in confidence arising from the 2011/2012 drilling campaign is reflected in the expansion of indicated mineral resource and the inclusion of some areas of measured mineral resource.

The sequence of drill campaigns has also corresponded to a general and progressive improvement in QA/QC activities so that these are largely now embedded in standard procedures.

25.3 Sample Preparation

During the 2011/2012 campaign, Aureus addressed a number of the sample QA/QC recommendations made in 2010 (AMC Consultants, 2010). The time lags associated with the logistics of sample movement between the Project, the Monrovia sample preparation facility, and the Ireland-based laboratory has been resolved by Aureus' decision to move sample analysis to SGS laboratories, which undertake sample preparation and assay analysis in their Monrovia facility.

AMC's October 2011 visit to the OMAC sample preparation facility in Monrovia revealed generally sound practices; however it was concluded that the workflow layout and quality of the ventilation represent potential risks for sample mix-up and contamination respectively. Subsequently to the data used for this report, Aureus have moved the sample preparation to a facility operated in Monrovia by SGS. Nonetheless, AMC recommends that Aureus commission an independent sampling and assaying specialist to conduct an audit of the SGS sample preparation facility.

The 2011/2012 drilling focussed on infill objectives; however some extension drilling, along strike and below the Latiff Zone was only partially successful in extending the mineralization beyond the previously defined limits. With depth the Latiff Zone was shown to continue, but in the central and western extensions, identified zone thicknesses and grades are poor. The Latiff Zone has now been shown to correlate with the Larjor Zone in the west and the Kinjor Zone in the east, but it is only in the deeper portions of the Latiff-Kinjor interface that economically significant grade intersections have been found.

The additional drilling data used for the April updated mineral resource estimate includes an intersection in drillhole K427 which is of significantly greater width and grade relative than surrounding intersections. The location and general character of this intersection are consistent with the assigned interpreted mineralized zone; hence AMC has no basis for concluding that the intersection is other than a valid representation of the mineralization. AMC cautions, however that the combined effects of the marked grade and thickness have been shown to significantly impact on the local estimates of gold metal, and AMC recommends that further drill data be collected to clarify the economic impacts associated with this single intersection.

Aureus continues to explore the Project for the purposes of both resource definition and extension; however the substantial amount of drilling conducted during the 2011/2012 campaign has largely fulfilled the requirements of resource definition for the current level of evaluation.

This report describes recent exploration works carried out by Aureus on the Project and which formed the basis on a new mineral resource estimate and an initial mineral reserve estimate developed jointly by Aureus and AMC.

25.4 Mineral Processing and Metallurgical

Subsequent to the completion of the Feasibility Study metallurgical test work, a further metallurgical test programme was undertaken as part of the optimization phase of the project. The test work programme was undertaken by the Australian, Perth based, ALS Laboratories.

The test work schedule was designed with the objective of finalizing the plant process flowsheet and optimizing reagent consumption while achieving a test work recovery of 93%.

The CIL circuit residence time could be reduced to 24 hours without negatively impacting on gold recovery by including a high shear pre-oxidation stage prior to CIL.

- The overall gold recovery was found to be strongly dependant on mill grind, test work on the composite sample indicated that gold recovery could beimproved by 2.8% when the grind was increased from 80% passing 75µm to 80% passing 42 µm. 80% passing 45 µm was thus selected as the optimum target mill grind.
- The target mill grind of 80% passing 45 µm with the inclusion of a regrind milling stage, using a VertiMill with steel grinding media.
- Test work on the variability samples indicated cyanide addition requirements of 0.50 kg/t –1.92 kg/t with an average addition requirement of 0.65 kg/t. This was based on an initial cyanide addition of 0.5 kg/t and further addition to maintain a solution concentration of 100 ppm cyanide.
- Test work indicated that the optimal pH control regime was to target a pH of 11 prior to the pre-oxidation stage and control the pH at 10 in the CIL circuit. Based on the optimized pH control regime lime addition requirements were found to be in the range 0.88 kg/t–2.13 kg/t with an average requirement of 1.48 kg/t.
- The SO_2/Air process produced a tailings stream with less than 50 ppm CNwad at a CNwad:SO_2 ratio of 5:1. This translated to an SMBS consumption of 0.77 kg/t for a CNwad level of 80 ppm in the CIL effluent stream.

The results of the optimization phase test work were used in conjunction with the test work results from Mintek phase 1, 2 and 7 to generate a correlation between head grade, grind and overall recovery. The test results from the Mintek phases 3, 4 and 5 did not include CIL testing. The Mintek phase 6 testing was not authorized by MDS and these results were not representative. This resulted in MDS requiring the additional phase-7 test work on the composites from phase 6 to verify leach recoveries. The correlation for a target grind size of 80% passing 42µm was then used to determine the expected plant recovery for full scale plant operations based on the mine plan.

Based on the optimization phase of the metallurgical test work undertaken on the New Liberty depoisit an average of 93% gold recovery should be achievable for years 1–6 under steady state conditions, post-commissioning and optimization of recovery.

25.5 Mineral Reserve Estimates

AMC has produced a detailed pit optimization analysis from which a pit design and life-of-mine schedule was produced. Included are a waste dump design and waste dump and backfill schedule.

A break even cut-off grade of 0.8g/t was calculated base on processing and General and Administration costs, processing recovery and gold price.

Dilution was applied to the resource based on a minimum mining width of 2.5 m, and a dilution skin of 0.5 m, which is considered appropriate for the scale of mining equipment anticipated. Dilution and ore loss parameters increased the reserve tonnes by 8.0% and decreased the reserve grade by 7.9%.

Overall pit slopes in the design are between 41° and 50°. This is within the geotechnical recommendations for overall slope design. Additional ramps or wider berm were included in the design to reduce overall slopes to meet geotechnical requirements.

The pit was designed as three sub pits which coalesce into one pit on strike. The ramps were designed to meet at saddles between the pits, and to exit to the south at the waste dump location.

The waste dump is designed to wrap around the open pit with a capacity of 52 million cubic metres. The mining plan proposes backfilling the Larjor Pit with 10 million cubic metres of mined waste.

In the initial stages of the pit operations it is planned to mine a starter pit and with the remainder mine in a further five phases. A steady state mining rate is planned, after an initial period of waste pre-stripping; at an annualized plant feed mining rate of 1.1 Mt tonnes and a total mining rate peaking at 26 million tonnes per annum.

The total mining cost (including all capitalized mining costs) over the life-of-mine is US$353 million. This works out to an average mining cost of US$2.52/t mined.

Over the life-of-mine the total ore tonnes processed is 8.5 Mt with an average head grade to the plant of 3.4 g/t Au producing 859koz of gold metal recovered with an average processing recovery of 93%. The average strip ratio is 15:5.

Table 25.1 details the proven and probable mineral reserve estimate.

Table 25.1 Mineral Reserve Estimate (as at 20 May 2013)

Reserve Category	Tonnes (Mt)	Au Grade (g/t)	Au ounces (koz)
PROVEN	0.7	4.4	99
PROBABLE	7.8	3.3	825
TOTAL	8.5	3.4	924
Waste tonnes (Mt)		132	
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.

25.6 Project Infrastructure

Following on from the recommendations in the Feasibilty Study, Aureus commissioned a LiDAR survey to be flown at the project site in December 2012. Following the receipt of an accurate site wide LiDAR topography, Aureus embarked on a process of optimizing the site infrastrucre arrangements in order to increase operational efficiencies, decrease construction and operational costs, and to mitigate against any environmental risks.

As a part of this optimization process, the following improvments have been made:

- The process plant has been relocated to a sloped position more central to the orebody and south of the open pit;
- The TSF has been relocated to the south of the open pit, in a natural valley, downslope from the processing plant, both reducing the level of earthworks required, lowering pumping costs and also removing the environmental risks associated with the tailings line crossing the Marvoe Creek;
- The Marvoe Creek diversion has been optimized by utilising an existing natural channel, identified during the LiDAR survey, which reduces the level of earthworks required; and
- The Waste Rock Dump now encircles the open pit to provide additional flood protection from the Marvoe Creek and additionally has a lower height profile, reducing rehabilitation requirements.

25.7 Environmental

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

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

Subsequent to the completion of the ESIA and the granting of the projects Environmental Permit, Aureus embarked upon an optimization process which resulted in the relocation of various mining associated infrastructure. The revised infrastructure locations remain within the area permitted for mining and as a result of this, Digby Wells were engaged to revise and update the ESIA. Detailed studies are currently underway and the results of which will be submitted to the Liberian Environmental Protection Agency as per the Bea Mountain MDA requirements.

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

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

The primary permit/licence required for the development of the Project is an Environmental Permit issued by the Minister of Environment. This permit was granted for the Project in October 2012 and is valid for three years, subject to an annual renewal by the EPA. Various other permits from other governmental departments will be required (e.g. construction of buildings etc.) but are procedural in nature.

25.8 Resettlement Action Plan

Development of the Project will require the resettlement of two relatively small villages, Kinjor and Larjor, which are located within the proposed mine pit. The resettlement involves 325 property owners and their households, as well as some households along the main project access road.

The Project Resettlement Action Plan (RAP) to address the above resettlement impactswas granted by the Liberian Environmental Protection Agency in March 2013.

As indicated above, the RAP has been approved the Liberian EPA and the relevant owner’s compensation packages have been approved and finalized. The primary focus of the RAP stakeholder engagement process is on the directly affected property owners in the Kinjor and Larjor communities and along the main acess road, as well as their representative bodies (including the traditional authorities concerned).

In order to involve affected communities in the resettlement planning and implementation process, a restellement working group (RWG) was established and comprises representative members of the resettlement-affected communities and households, as well as the relevant local government structures, town councils, traditional authorites, womens groups, the youth and known governmental organisations, The RWG is chaired by Aureus and the chair person nominated is available at all format relocation committie meetings.

The construction of the relocation village has commenced and the local community is fully involved in the construction process.

25.9 Community Development Plan

The CDP was completed in December 2012 and was approved by the Liberian EPA in January 2013.

Based on the development oportunites assessed, a number of potential CDP projects were identified, which include vegetable production, cashew nut production and coconut production as well as other further agricultural activities. Where feasible the final CDP will be expanded to incorporate the community development aspects of the BMMC MDA.

The development and operation of the project will have both positive and negative impacts on the socio-economic structure of the project area and its environments.This will be achieved in various ways at national, district and local levels though the payment of taxes and royalties, increased employment activities, training, purchase of goods manufactured and supplied in Liberia, cash compensation for farms, commercial opportunities and an improvement in local infrastructure by the establishment of the resettlement site/village.

The development of the project will bring much needed investment and development opportunites with consequent impacts on the employment and the affected communites.

26 RECOMMENDATIONS

26.1 Mineral Resources

In view of the significantly greater width and grade of the main mineralized intersection in drillhole K427, relative to surrounding intersections, and hence the potential economic impacts of this single drillhole, AMC strongly recommends that the lateral extent of the interpreted mineralization be tested with at least three drillholes in the immediate vicinity.

AMC is aware that Aureus has implemented more rigorous sample QA/QC management and tracking procedures. AMC recommends that this process should form an ongoing priority for all exploration and resource definition programmes, and that periodic reviews by independent specialists should be conducted.

AMC notes that Aureus has changed the laboratory facility used for sample preparation from OMAC Monrovia to SGS Monrovia. This decision preceded implementation of a previous AMC recommendation that the OMAC facility be audited. Nonetheless, AMC recommends, as good practice, that an audit by an independent sampling and assaying specialist be conducted of the full chain of sample preparation activities, including sample preparation at SGS Monrovia.

Over the 2011/2012 drilling campaign, Aureus employed a dedicated database geologist and upgraded the drill data repository to a single coherent MS Access database. AMC considers this to be a significant step forward, but notes that the expanding quantity of information will require increasing rigor in the management of drilling and other information related to resource definition and evaluation. AMC therefore recommends that all aspects of resource data management be reviewed.

26.2 Mining

There is scope to increase current mineral reserves through the drilling of Inferred Resources on hanging wall lenses within the pit as well as drilling of inferred mineral resources just below the bottom of the current optimized pit.

Once the hydrogeological studies have been completed and a groundwater model is developed the pit slopes should be reviewed in the light of the hydrogeological modelling.

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AMC Consultants (UK) Limited
Registered in England and Wales
Company No 3688365

Level 7, Nicholsons House
Nicholsons Walk, Maidenhead
Berkshire SL6 1LD
UNITED KINGDOM

T +44 1628 778 256
F +44 1628 638 956
E amcmaidenhead@amcconsultants.com

CERTIFICATE OF QUALIFIED PERSON

Martin W Staples
AMC Consultants (UK) Limited
Level 7 Nicholsons House
Nicholsons Walk
Maidenhead
Berkshire
SL6 1LD

Telephone: +44 (0) 1628 778 256
Fax: +44 (0) 1628 638 956
Email: mstaples@amcconsultants.com

This certificate applies to the technical report titled "New Liberty Gold Project, Liberia, West Africa, Updated Technical Report" (the "Technical Report") for Aureus Mining Inc. with the effective date July 03, 2013.

I, Martin W Staples, do hereby certify that:

1. I am a Director for AMC Consultants (UK) Limited, Level 7 Nicholsons House, Nicholsons Walk, Maidenhead, Berkshire, SL6 1LD, United Kingdom.
2. I graduated with a BSc in Mining Engineering from the University of Newcastle upon Tyne, in 1980.
3. I am a Fellow of the Australasian Institute of Mining and Metallurgy – FAusIMM.
4. I have practiced my profession continuously since 1980, and have been involved in the mining industry for a total of 32 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 1, 2, 3, 15, 16, 19, 22, 24, 25, 26, 27, and parts of Section 18 and parts of Section 21 of the Technical Report.
7. I visited the property between 19–25 November 2012.
8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as a Director for the preparation of the work which forms part of the Technical Report.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and Sections 1, 2, 3, 15, 16, 19, 22, 24, 25, 26, 27, and parts of Section 18 and parts of Section 21 have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Sections 1, 2, 3, 15, 16, 19, 22, 24, 25, 26, 27, and parts of Section 18 and parts of Section 21, of the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 3rd July 2013

Martin W Staples BSc, FAusIMM
Director

AMC Consultants (UK) Limited
Registered in England and Wales
Company No 3688365

Level 7, Nicholsons House
Nicholsons Walk, Maidenhead
Berkshire SL6 1LD
UNITED KINGDOM

T +44 1628 778 256
F +44 1628 638 956
E amcmaidenhead@amcconsultants.com

CERTIFICATE OF QUALIFIED PERSON

Christopher G Arnold
AMC Consultants (UK) Limited
Level 7 Nicholsons House
Nicholsons Walk
Maidenhead
Berkshire
SL6 1LD

Telephone: +44 (0) 1628 778 256
Fax: +44 (0) 1628 638 956
Email: carnold@amcconsultants.com

This certificate applies to the technical report titled "New Liberty Gold Project, Liberia, West Africa, Updated Technical Report" (the "Technical Report") for Aureus Mining Inc. with the effective date July 03, 2013.

I, Christopher G Arnold, do hereby certify that:

1. I am a Principal Geologist for AMC Consultants (UK) Limited, Level 7 Nicholsons House, Nicholsons Walk, Maidenhead, Berkshire, SL6 1LD, United Kingdom.
2. I graduated with BSc (Hons) in Geology from Natal University, South Africa in 1979, and an MSc in Natural Resource Management from the University of Western Australia in 1986.
3. I am a Chartered Professional member of the Australasian Institute of Mining and Metallurgy.
4. I have practiced my profession continuously since 1980, save for a two year interval of postgraduate study, and have been involved in mineral exploration, mine geology and mineral resource consulting for a total of 31 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 4–12, Section 14, and Section 23 of the Technical Report.
7. I visited the property between 1–3 December 2009 and between 1– 5 October 2011.
8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as a Director for the preparation of the work which forms part of the Technical Report.
9. I am independent of the issuer as described in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and Sections 4–12, Section 14, and Section 23 have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Sections 4–12, Section 14, and Section 23 of the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 3rd July 2013

Christopher G Arnold MAusIMM CP(Geo)
Principal Geologist

CERTIFICATE OF QUALIFIED PERSON

R M Welsh
DRA Mineral Projects
Suite 502
80 Strand Street
Cape Town 8000
South Africa

Telephone: +27 21 285 0119
Email: robw@drasa.co.za

This certificate applies to the technical report titled "New Liberty Gold Project, Liberia, West Africa, Updated Technical Report" (the "Technical Report") for Aureus Mining Inc. with the effective date 3 July 2013.

I, Robin Mark Welsh, do hereby certify that:

1. I am a Senior Project Manager for DRA Mineral Projects, Suite 502, 80 Strand Street, Cape Town, 8000, South Africa.

2. I graduated with a BSc Electrical Engineering from the University of Natal, South Africa in 1990.

3. I am registered as a Professional Engineer with the Engineering Council of South Africa since 1999 (Registration number 990118) and I have been a member of the South African Institute of Electrical Engineers since 1993.

4. I have practiced continuously as an Electrical Engineer and Project Manager since 1990, and have been involved in industrial and mining projects for a period of 22 years.

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

6. I am responsible for the preparation of parts of Sections 17, 18 and 21 of the Technical Report.

7. I visited the property on 21-22 May 2012, 21-25 January 2013, 05-08 March 2013, 14-17 May 2013 and 11-14 June 2013.

8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as a Project Manager for the preparation of the work which forms part of the Technical Report.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and parts of Sections 17, 18 and 21 of the Technical Report have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, parts of Sections 17, 18 and 21 of the Technical Report and its supporting documentation contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 3 July 2013

Robin Mark Welsh
Senior Project Manager
DRA Mineral Projects

CERTIFICATE OF QUALIFIED PERSON

G Bezuidenhout
DRA Mineral Projects
3 Inyanga Close
Sunninghill 2157
Johannesburg
South Africa

Telephone: +27 11 202 8686
Email: glennb@drasa.co.za

This certificate applies to the technical report titled “New Liberty Gold Project, Liberia, West Africa, Updated Technical Report” (the “Technical Report”) for Aureus Mining Inc. with the effective date 3 July 2013.

I, Glenn Bezuidenhout, do hereby certify that:

1. I am a Process Director for DRA Mineral Projects, 3 Inynga Close, Sunninghill, Johannesburg, South Africa.

2. I graduated with a National Diploma in Extractive Metallurgy from the Witwatersrand Technicon South Africa in 1979.

3. I have been a Fellow of the South African Institute of Mining and Metallurgy since 2012 (Membership Number 705704).

4. I have practiced continuously as a Process Engineer since 1992, and have been involved in mineral processing and mining projects for a period of 21 years.

5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 13, as well as parts of Sections 17, 21 and 25 of the Technical Report.

7. I visited the property on 21-22 November 2012.

8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as a Process Director for the preparation of the work which forms part of the Technical Report.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and Section 13, as well as parts of Sections 17, 21 and 25 of the Technical Report have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Section 13, as well as parts of Sections 17, 21 and 25 of the Technical Report and its supporting documentation contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 3 July 2013

Glenn Bezuidenhout
NDT Ex. Met, FSAIMM
Process Director
DRA Mineral Projects

CERTIFICATE OF QUALIFIED PERSON

Graham Trusler
Digby Wells Environmental
Fern Isle, Section 10
359 Pretoria Avenue
Private Bag X10046
Randburg 2125
South Africa

Telephone: +2711 789 9495

Fax: +2711 789 9498

Email: graham.trusler@digbywells.com

This certificate applies to the technical report titled "New Liberty Gold Project, Liberia, West Africa, Updated Technical Report" (the "Technical Report") for Aureus Mining Inc. with the effective date 3 July 2013.

I, Graham Trusler, do hereby certify that:

1. I am the Chief Executive Officer of Digby Wells Environmental, Fern Isle, Section 10, 359 Pretoria Avenue, Randburg 2125, South Africa.

2. I graduated with a B.Sc Chemical Engineering (University of Natal, 1986), M.Sc Engineering (University of Natal, 1988) and B. Comm (University of South Africa, 1994). I completed the Integrated Environmental Management course (I.E.M.) (University of Cape Town, 1993).

3. I am a Professional Engineer registered with the Engineering Council of South Africa, (Pr.Eng, Reg. No. 920088).

4. I have practiced my profession continuously since 1990 and have been involved in Environmental and Social Impact Assessments for 20 years.

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

6. I am responsible for the preparation of Section 20 of the Technical Report.

7. I visited the property on 20-21 March 2013 and 13-17 May 2013.

8. I have not had any involvement with the property that is the subject of the Technical Report prior to my engagement as Chief Executive Officer of Digby Wells Environmental on technical matters, the results of which form part of the Technical Report.

9. I am independent of the issuer as described in Section 1.5 of NI 43-101.

10. I have read NI 43-101 and Form 43-101F1, and Section 20 of the Technical Report has been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my information, knowledge and belief, Section 20 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the use of my name and to the public filing of the Technical Report by Aureus Mining Inc.

Dated the 3 July 2012

Graham Trusler
M.Sc (Eng.), Pr.Eng.
Chief Executive Officer
Digby Wells Environmental