LION ONE METALS

TUVATU GOLD PROJECT PRELIMINARY ECONOMIC ASSESSMENT

NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

17° 42' South
177° 35' East

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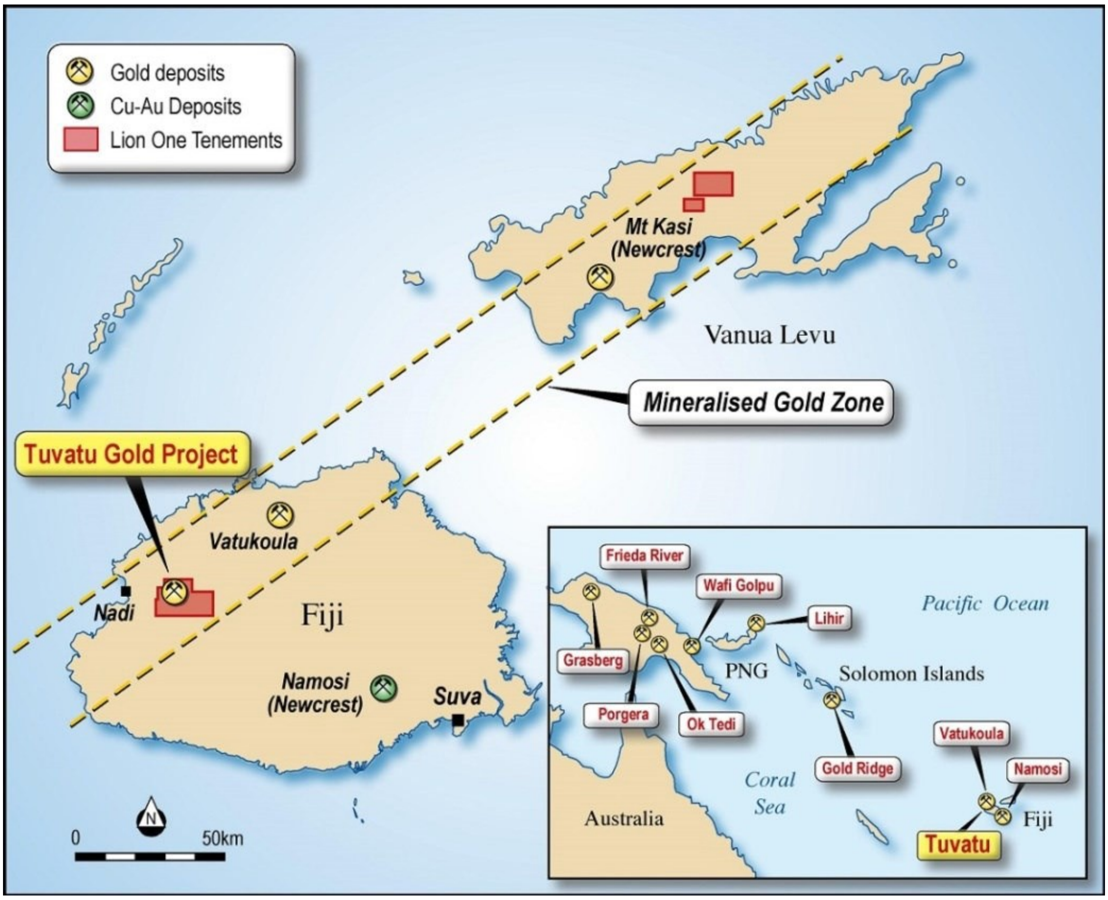
1.0 SUMMARY

This report presents a preliminary economic assessment of the Tuvatu Gold Project in Fiji that has been recently completed for Lion One Metals Limited (Lion One), which is a public company jointly listed in Canada and Australia (TSX-V: LIO, ASX: LLO).

The Tuvatu Gold Project is a high grade, narrow vein gold deposit located in the three tenements held by Lion One Limited, a subsidiary company of Lion One Metals Limited. The tenements are located in hilly topography in the upper reaches of the Sabeto River valley approximately 24 km north east of Nadi on the west coast of Viti Levu, and 15 km from the Nadi International Airport (Figure 1.1).

Figure 1.1 Project Location



Lion One engaged Mining Associates Pty Ltd (MA) to review the geology and exploration results to allow re-estimating of the mineral resource. Canenco Canada Inc. was engaged to review the metallurgy and produce a suitable process plant design for costing.

AMC Consultants Pty Ltd (AMC) was engaged to review the mining and geotechnical aspects of the project and to propose a preliminary mine design and indicative capital and operating costs for the project.

Knight Piésold Pty Ltd (KP) was engaged to propose a design for the tails storage facility (TSF) and to review the surface geotechnical conditions for the TSF and process plant sites.

This study is based on the January 2015 Mineral Resource report herein, which was in turn an update of the May 2014 mineral resource report.

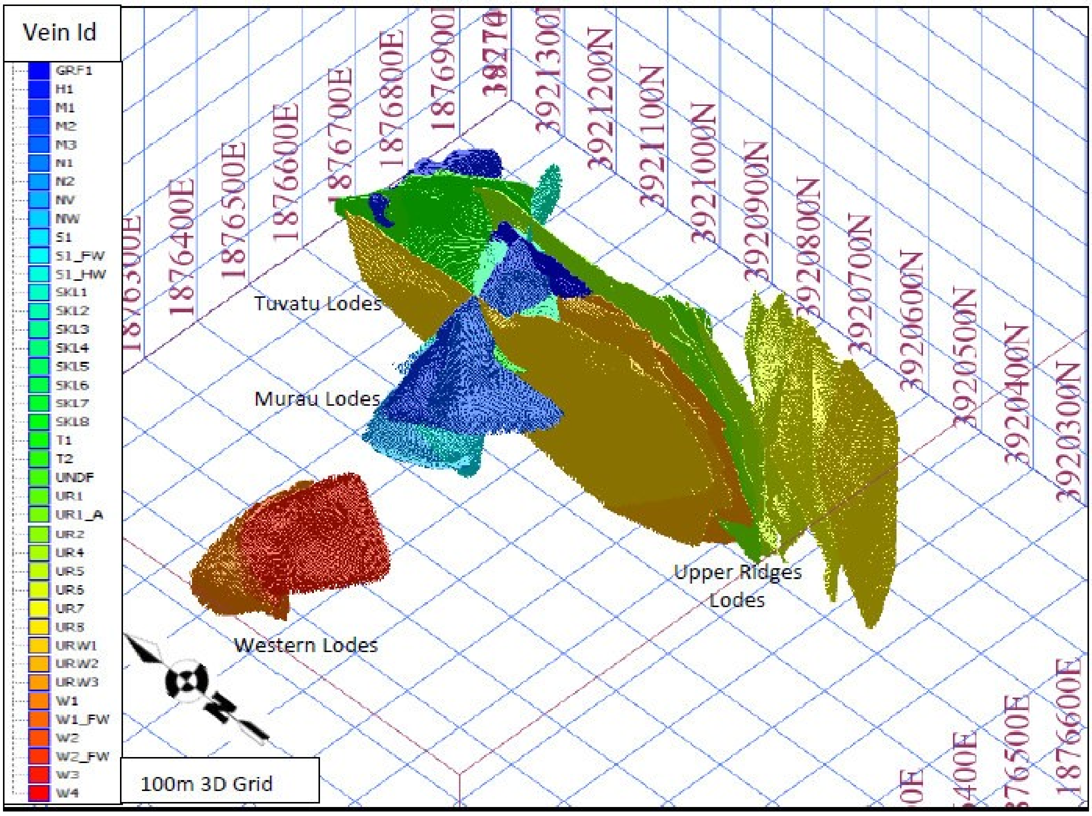
This study provides estimates of the Indicated and Inferred Resources. There are no Measured Resources. Inferred Resources comprise 55% of the mined tonnes and 63% of the contained ounces in the mine plan developed in this Study.

1.1 Geology

Local host lithologies in the Tuvatu area are a sequence of volcaniclastic units intruded by a monzonite intrusive complex. Gold mineralization is dominantly hosted in the monzonite units but also occurs in the adjacent volcanics. Mineralization is structurally controlled and is considered to have a close association with the emplacement of the monzonite intrusive body occurring as sets and networks of narrow veins and cracks, with individual veins as modelled in this study ranging from 0.04 to 9 m true width with a mean of 1.1 m. Lode mineralogy is varied, with most veins containing quartz, pyrite and base metal sulphides. A high proportion of the gold in the deposit occurs as either free gold or is contained in quartz or pyrite grains that can be extracted by simple floatation followed by cyanidation or direct leaching. Free gold present is both fine and coarse grained although sample assay repeatability is very good suggesting most is fine grained. Mineralization contains minor amounts of deleterious elements such as arsenic, selenium, and uranium.

The main mineralized zone (Upper Ridges) comprises eleven principal lodes with a strike length in excess of 500m and a vertical extent of more than 300 m (Figure 1.2). Another major zone of mineralization (Murau) strikes east-west and consists of two major lodes with a mapped strike length in excess of 400 m. A total of 39 different lode structures were identified in the resource area including 11 lodes in the Upper Ridges area, 3 lodes in the Murau area, 4 lodes in the West area, 2 lodes in the Tuvatu area and stockwork veins in the SKL area. A minimum of 5 intercepts are needed for a vein to be defined with a number of other lodes having been identified but remain to be further tested before inclusion in resource estimates.

Figure 1.2 Tuvatu Lodes (Oblique view (45°E -45°)



1.2 Mineral Resource

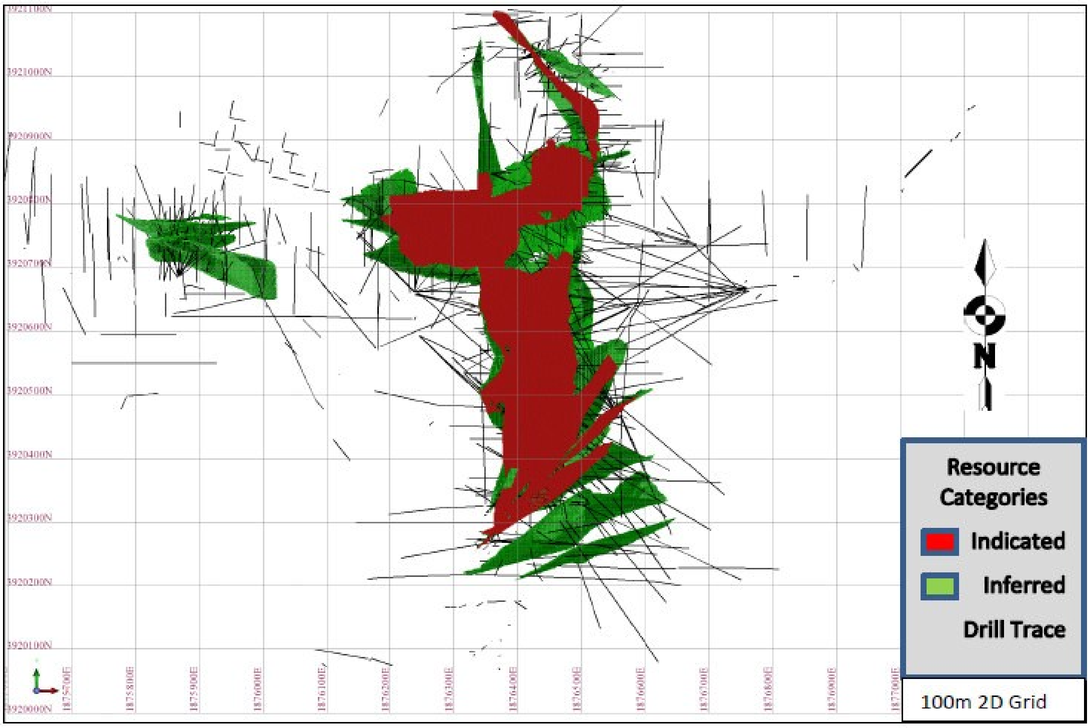
The resource has been estimated for each vein individually using Ordinary Kriging of width and grade, the latter using accumulations, into a 3D block model.

The Tuvatu resource is reported at one cut off (3g/t Au) representing a resource amenable to underground production. The total indicated resource is 1,120,000 tonnes at 8.17 g/t Au for 294,000 ounces of gold and an inferred resource of 1,300,000 tonnes at 10.6 g/t Au for 445,000 ounces of gold (Table 1.1).

Table 1.1 2014 and 2015 Tuvatu Resource Estimate

		Resource Category					
		Indicated			Inferred		
Cut off		Tonnes t	Grade Au g/t	Ounces Au oz	Tonnes t	Grade Au g/t	Ounces Au oz
3 g/t Au	May 2014	1,101,000	8.46	299,500	1,506,000	9.7	468,000
3 g/t Au	January 2015	1,120,000	8.17	294,000	1,300,000	10.6	445,000

Figure 1.3 Plan View of Lodes Colour Coded for Resource Category with Drilling



A total of 1,341 m of decline, strike and rise development have also been undertaken in the project area including a 600m exploration decline. The resource has been depleted by a total of 3,500 tonnes at 9.06 g/t Au (1,020 oz Au). No detailed historical production records are available.

The resource has been reported at various cut off grades to provide insight into the sensitivities of the resource (Table 1.2). Reporting all material above a cut off of 3 g/t Au provides an indicated resource of 1,120,000 tonnes at 8.17 g/t Au for 294,000 oz and an inferred resource of 1,300,000 tonnes at 10.6 g/t Au for 445,000 ounces of Au.

Table 1.2 Tuvatu Resource Reported Above Various Cut-Offs

Cut-Off g/t	Indicated			Inferred		
	material (t)	Au (g/t)	Au (oz)	material (t)	Au (g/t)	Au (oz)
1.0	1,933,000	5.52	342,900	2,569,000	6.4	524,500
2.0	1,442,000	6.90	319,700	1,903,000	8.1	492,900
3.0	**1,120,000**	**8.17**	**294,000**	**1,300,000**	**10.6**	**445,000**
5.0	689,000	10.80	239,300	793,000	15.0	382,100

The effective date for the resource estimate update is 8 January 2015, whilst the effective date of the original resource estimate is May 2014.

Qualified Person

The summary review of geology, resource models and estimates and the site visit were conducted by Mr Ian Taylor, BSc (Hons) MAusIMM (QP) who visited the site from 25 to 28 February 2014. Mr Taylor viewed the geological setting, located some drill collars, inspected drill core, and sample storage.

Mr Taylor has sufficient experience which is relevant to the Tuvatu style of mineralization and deposits under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (Australia) and is a Qualified Person as defined in NI43-101 (Canada). He is a Member and certified professional of The Australasian Institute of Mining and Metallurgy (Melbourne). Mr Taylor is employed by Mining Associates Pty Ltd of Brisbane, Australia.

1.3 MINING STUDY – AMC

1.3.1 Introduction

The Study is based on the January 2015 Mineral Resource estimate.

The Study considered Indicated and Inferred Mineral Resources. There are no Measured Mineral Resources. Inferred Mineral Resources comprise 55% of the projected mined tonnes and 63% of the projected contained ounces in the mine plan developed in the Study.

The Study is based on a metal price of US$1,200/oz gold.

Due to the high amount of Inferred Mineral Resources in the mine plan and the preliminary nature of the Study, no Mineral Reserves are reported.

There are four main zones contained in the Mineral Resource model. Snake/Marau zone is steep-dipping and strikes east-west. The Tuvatu zone is steep-dipping and strikes north-west by south-east, and is located north of the Coreshed Fault. The SKL zone is flat-dipping. The Upper Ridge zone is steep-dipping, striking north-south and contains over 80% of the conceptual mined gold.

1.3.2 Geotechnical

The geotechnical study involved data processing from 21 resource drill-holes followed by an empirical analysis for underground designs.

The underground geotechnical engineer undertook no site visit, and no dedicated mine geotechnical holes have been drilled.

Anticipated ground conditions can be described as "fair" to "very good". The majority of ground can be characterized as "good".

Very limited information is available on hydrogeology. The rock mass itself, particularly in the fresh rock exposed in the Upper Ridge zone, is impermeable and dry. Significant water inflows are reported from a major fault, the Coreshed Fault, located on the northern end of the mineralization.

The standard stoping panel size is based on a 60 m level interval and 60 m strike length. Sill pillars between the stope panels should be at least 6 m high.

The recommended standard ground support for development drives in good ground is friction bolts installed on a regular pattern. Areas where ground conditions are blocky will require surface support. Welded wire sheet mesh is recommended as the standard surface support in these areas. Larger spans are formed at the intersection of development drives. In these areas deeper anchorage for ground reinforcement using 6 m cable bolts is recommended.

Developing through and in the vicinity of the Coreshed Fault is expected to be challenging and require heavy ground support.

AMC has identified the presence of smectite (swelling clay) associated with mineralization. If present in sufficient quantities, this may adversely affect the mineralized material draw in shrinkage stoping panels. As the broken material forms the working platform for stoping activities, the presence of clay could lead to 'false floors' and pose a significant safety risk to personnel. AMC strongly advises that additional analysis be undertaken to understand the true extent, occurrences and the impact of the swelling clay.

1.3.3 Underground

Mine access will be via a main decline located in the footwall, with a gradient of 1:7 and dimensions of 4.5 mW x 4.5 mH. Access to the mineralization will be made from two declines from surface and internal declines. Two ventilation raises to surface are included.

Level access drives are designed at 4.0 mW x 4.0 mH and draw-points at 3.5 mW x 4.0 mH. These dimensions enable use of medium-sized loaders for improved productivity (width) and truck loading close to the stoping area.

The primary planned mining method is shrinkage stoping (air-leg method) with limited breast stoping (air-leg method) for flat dipping lodes.

Shrinkage stoping is a manual overhand method that relies on broken mined material being left in the stope to provide a 'working floor' and support the stope walls. Man and material access into the stope is from pre-developed raises. Production drilling is performed using air-legs (jack-legs). Mineral extraction is from crosscuts driven at closely spaced intervals into the bottom of the stope. During the mining cycle only 30% to 35% of the blasted material is extracted, equivalent to the broken swell factor. When mining to the top horizon is completed, the remaining material is extracted.

Breast mining is a method suitable for shallow-dipping orebodies of uniform narrow width and larger horizontal extent, usually represented in conglomerate gold reefs such as those mined in South Africa. This method is also used in flatter-lying lodes at the nearby Vatukoula mine. Material is extracted via a number of panels mined along strike from an initial raise line position, with drilling being accomplished using handheld rock drills. The blasted material is mucked using scrapers to slusher gullies and eventually to a muck-bay, from where it is loaded onto trucks and hauled out of the mine.

1.3.4 Schedules

The conceptual underground mineralization mining schedule is shown in Table 1.4. Indicated Mineral Resources comprise 45% of the material based on tonnes, and 37% of the material based on gold ounces.

Table 1.3 Conceptual mining schedule

Item	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Indicated + Inferred										
Tonnes	Kt	22.0	140.5	172.0	198.3	196.8	197.0	167.2	31.7	1,125.5
Grade	g/t	7.38	16.36	16.49	14.44	8.57	7.11	7.49	6.87	11.30
Contained Ounces	Koz	5.2	73.9	91.2	92.1	54.3	45.0	40.3	7.0	409.0
Indicated										
Tonnes	Kt	16.3	68.9	65.7	61.9	75.6	111.2	93.6	12.4	505.6
Grade	g/t	7.41	14.21	13.47	9.42	8.17	7.10	7.45	6.18	9.39
Contained Ounces	Koz	3.9	31.5	28.5	18.7	19.8	25.4	22.4	2.5	152.7
Inferred										
Tonnes	Kt	5.7	71.6	106.3	136.4	121.3	85.8	73.5	19.2	619.9
Grade	g/t	7.29	18.42	18.36	16.72	8.83	7.11	7.55	7.33	12.86
Contained Ounces	Koz	1.3	42.4	62.8	73.3	34.4	19.6	17.9	4.5	256.3
Inferred Mineral Resources of mined tonnes										55.1%
Inferred Mineral Resources of contained ounces										62.7%

The tonnage/grade estimates that comprise the conceptual production schedule do not represent estimates of Mineral Reserves. Values may not compute exactly due to rounding

1.3.5 Costs

All costs are expressed in US dollars (US$).
Mining costs are based on:

- Owner underground mining.
- Owner management and technical services.

Costs have been estimated using:

- Current supplier quotation.
- Lion One supplied inputs, including equipment costs, personnel costs and key consumables.
- Recent AMC information.
- Allowance for minor items on the basis of AMC database information or experience.

The estimated underground costs total US$123M, comprising of US$31M capital and US$92M operating, with an average estimated operating cost of US$82/t of mineralized material produced. The underground owner mining operating cost estimate was prepared from first principles.
Underground mining capital costs include:

- Purchase of the mining fleet.
- Rebuilds (major overhaul) of the mining fleet.
- Capital lateral development.
- Capital vertical development.
- Underground infrastructure.

The mining operations are envisaged as being supported by a technical services department. Costs for this department are allocated to underground capital and operating mining based on the cost ratio of these areas.

1.4 Mineral Processing

1.4.1 Metallurgy

A total of nine metallurgical testwork campaigns have been undertaken on variability samples, composites and bulk samples from the Tuvatu mineralized lodes. The mineralogy indicates a bimodal distribution of the gold particles and due to the variability in the test work results; gold recovery would potentially be more efficient by adopting a flowsheet with a combination of gravity, flotation and leaching. Comminution characteristics suggest that the mineralized material is relatively competent, requiring comparatively more crushing and ball milling energy to effect the size reduction required.

The gold recovery is based on variability results from samples with >3g/t Au head grade, in an effort to be representative of the resource and potential mill feed grades. This approach was found to be conservative; as to include the lower grade material increased the overall recovery result. Due to only one test being undertaken on leaching the flotation tails, returning a recovery of 75.3%, the leach stage recovery from the gravity/leach variability tests, from two campaigns was used to predict the variable leach response in the flotation tails CIL circuit. This brought the recovery for the flotation tails CIL circuit to 56.9%.

A gold recovery balance was then undertaken using the Metcon Laboratories Pty. Ltd. gravity/flotation/concentrate leach variability results, in order to calculate the gold that would be potentially recovered with the selected process flowsheet. The median gold recovery using this approach was 86.8%, which was decreased by 0.5% to account for solution losses that will typically occur in the commercial plant. This PEA recovery value of 86.3% is calculated only from consideration of the variability leach results and assumes that the samples tested are moderately representative of the material to be mined, but at this level of study, does not weight the results based on proposed mine plan.

1.4.2 Processing Facility

The processing facility flowsheet has been selected based on the criteria defined by the mineralogical characteristics of the mineralized material and metallurgical testwork undertaken to date. The comminution facility is a conventional two-stage crushing and screening circuit, followed by two-stage grinding to achieve a target grind P80 of 75µm. The grinding circuit, which includes gravity recovery, then feeds flotation where a sulphide concentrate is produced and reground to 80% passing 20µm prior to entering the Carbon-In-Leach (CIL) circuit. Both the flotation tails and concentrate are leached, with the concentrate CIL tails recirculating to the feed of the flotation tails CIL circuit, the combined discharge being pumped to detoxification and tails deposition.

The process facility is designed with a nominal capacity of 219,000tpa for a nominal design rate of 600tpd based on an overall availability of 91% with a life of mine average feed grade of

11.3g/t Au. The plant is designed to operate 365 days/year, 24 hours/day. The crushing circuit is designed with a mechanical equipment availability of 75%, however it has been sized to process 1000tpd to accommodate any future expansion.

The proposed process plant will include the following unit operations:

- Primary Crushing – A dump pocket, vibrating grizzly and jaw crusher in open circuit producing a final product of 80% passing 105 mm.
- Secondary Crushing - A vibrating double deck screen and cone crusher operating in closed circuit.
- Primary Grinding – A ball mill in open circuit producing a final product of 80% passing 1000µm.
- Secondary Grinding – A ball mill in closed circuit with hydrocyclones producing a final product of 80% passing 75µm.
- Gravity Concentration – Gravity concentration of cyclone underflow from the secondary milling circuit to produce a gold concentrate for tabling, followed by direct smelt.
- Flotation – Sulphide flotation of the hydrocyclone overflow to produce a gold concentrate for regrinding.
- Regrinding – A regrind mill fed by open circuit hydrocyclones producing a final product of 80% passing 20µm.
- Thickening - Both the flotation tails and concentrate are thickened to 50% solids prior to regrinding and leaching.
- Carbon-in-leach (CIL) – Gold leaching of the flotation tails and reground concentrate through the two CIL circuits, where absorption of solution gold onto carbon particles occurs. Leaching is facilitated by oxygen and air.
- Cyanide Detoxification – Detoxification of cyanide slurry via the SO_2/Air process with addition of copper sulphate, to produce tailings with a target of <1ppm CN_{WAD} (Weak Acid Dissociable) and disposal of detoxified tailings in the conventional tailings storage facility.
- Absorption, Desorption, and Refining (ADR) – The absorption occurs in the CIL circuit, acid wash of carbon to remove inorganic contaminants, elution of carbon to produce a gold rich solution for electrowinning (sludge production), filtration, drying, and smelting to produce gold doré, and thermal regeneration of stripped carbon to remove organic contaminants.

At the preliminary level of this study, test work results, industry data and assumptions have been used to make reasonable estimates for equipment sizing. These inputs will be reviewed in the next phase of engineering. A design criteria was developed based on the available testwork and a summary is Section 17. The design criteria include the calculations or information basis for each piece of major equipment.

Process plant tailings will be pumped to a tailings storage facility (TSF). Geochemical testing of the tailings has not yet been undertaken, but on review of the data available, it has been predicted by KP that the tailings will likely be potentially acid generating (PAG), however there is no information pertaining to the neutralization potential (NP) of the mineralized material, so for this study, storage of the tailings in a sub-aqueous facility is proposed to prevent oxidation of the solids.

1.4.3 Infrastructure

The Tuvatu project and mine lease area is 17km by road from Nadi international airport. The region is well serviced with port facilities at Ba and Lautoka. Lion One also maintains an operations office in Nadi that will continue to service the future site operations.

The Tuvatu project site comprises steep topography coupled with multiple creek lines, which flow into the Sabeto river which supports the community, agricultural and tourist activities downstream. The comparative reduced size of the proposed project will allow the surface infrastructure to be accommodated within the relatively flat areas available such that ground disturbance will be minimized as much as possible and site run-off will be managed readily.

There is an 11kV transmission line crossing the Tuvatu site from a nearby Fijian Electricity Authority (FEA) hydroelectric plant. Due to the national shortfall in power supply from the grid, despite supplementary thermal generating capacity, the project will generate its own power. A containerized diesel power station, including switchgear and transformers, with 1,500kVA generator units, is proposed to suit the load of approximately 4MW, in an N+1 or N+2 redundancy configuration to ensure reliability of supply and provide enough reserve to start the larger ball mill motors.

Power supply costs (including maintenance factors) is based on delivered diesel fuel cost of USD$0.90/L for a total cost of USD$0.24/kWh.

The mine dewatering from underground pumping, will discharge to an intermediate settling pond prior to pumping to the raw water storage tank adjacent to the process plant, and will be used to provide the raw water requirements to service the project needs. The Coreshed Fault, one of the major fault structures identified on site, is a significant water bearing fault and can provide a supply during the dry season. It has been determined that raw water can further be managed by controlling the flows from the tailings facility catchment to allow storage of raw water make-up to the process in the impoundment.

Proximity to Nadi and local villages provides sufficient accommodation for contractors. Local landowners will be contracted to provide transportation of workers to site. It is not envisaged construction accommodation will be required on site, as the workforce will be sourced from local communities, with only key components of the contractor’s workforce mobilizing from elsewhere.

1.4.4 Processing and G&A

The process operating cost or expenditure (OPEX) of the project has been estimated based on the scope defined in this report and is based on a variety of sources including cost service data, vendor quotes, first principle calculations, metallurgical testwork and reference projects.

Plant operating costs have been determined at $43.8/t milled, for a facility with a nominal throughput of 219,000tpa at a grind size of 80% passing 75 µm, based on a 24 hour per day operation, 365 days per year and a 91% overall plant availability.

The G&A costs include provisions for office administration, potable water supply and treatment, non-process related and off-site power costs, insurance, Health, Safety, Environment and Community (HSEC) equipment and related stakeholder engagement costs, contracts for IT, site security, transportation, off-site road maintenance, running costs associated with the site mobile equipment fleet, site fuel costs, PPE supplies and laboratory consumables. Costs associated with business travel and training have been allowed, as well as for ongoing use of consultants.

The G&A costs for the supporting facilities and administration are estimated to be $13.75/t milled. These costs are assumed to consist of both fixed and partially variable, changing to reflect the plant operations.

Operating and G&A costs are summarized in Table 1.4.1 and are considered to have an accuracy of +30 / -10%.

Table 1.4 Process and G&A OPEX Summary

Description	$/year	$/t (milled)
Labour	415,257	1.9
Power	2,666,880	12.18
Consumables - subtotal	5,001,263	22.84
Reagents	*4,561,989*	*20.83*
Liners	*323,867*	*1.48*
Media	*115,408*	*0.53*
Maintenance	1,021,576	4.66
Miscellaneous (e.g. Laboratory)	493,718	2.25
Total Process Operating Cost	**9,598,696**	**43.83**
G&A Labour	351,409	1.6
G&A Expenses	2,659,616	12.14
Total G&A Operating Cost	**3,011,025**	**13.75**
Total Operating Cost	**12,609,721**	**$57.58**

1.5 Economic Analysis

Financial Projections

A discounted cash flow model was prepared based on the mining schedule and estimated capital and operating costs. The early mine operating costs (pre-production) are capitalised. The current mining schedule results in a period of low production towards the end of the current mine life from the shrink stoping areas. Low grade stockpiles are not accounted for in this study and will be used to supplement throughput towards the end of the mine life (as currently defined).

At a gold price of US$1,200 per ounce (advised by Lion One), the project is estimated to have a pre-tax IRR of 67.1% and a pay-back period of 1.25 years after start of production. At a discount rate of 5%, the pre-tax NPV is estimated at C$116.99 million.

A sensitivity analysis on the financials indicates that the project is most sensitive towards gold price, followed by the project capital cost.

Table 1.5 Project Production Summary

	Basis of Estimate	
Total matieral mined and processed	1,125,548	t (dry)
Average head grade	11.30	g Au/t
Contained gold in mined	408,958	oz Au
Recovered gold	352,931	oz Au
Average gold recovery	86.3	%
Production mine life	6.16	years
Nominal production rate	219,000	t/y
Average annual production	182,802	t/y
	57,320	oz Au/y

Table 1.6 Project Cash Flow Summary

	Project US$ Million	US$/t *	US$/oz Au**
Mine operating cost	86.11	76.50	243.98
Processing cost	49.33	43.83	139.78
Exploration costs	1.73	1.53	4.89
General and Administration cost	21.94	19.49	62.16
Smelting and Refining cost	0.85	0.75	2.40
Subtotal Cash Operating Cost	**159.95**	**142.11**	**453.21**
Royalties and Export Taxes	40.23	35.75	114.00
Total Cash Operating Cost	**200.19**	**177.86**	**567.21**
Revenue	**423.52**	**376.28**	**1,200.00**
Total Cash Cost	200.19	177.86	567.21
Operating Cash Flow (EBITDA)	**223.33**	**198.42**	**632.79**

***** Basis is LOM tonnes

** Basis is recovered not contained ounces

Table 1.7 Project Financial Measures Summary

Basis of Estimate		
Revenue from gold (based on US$1,200/oz)	423.52	US$ M
Total cash cost excluding royalties	453.21	US$ / oz Au
Total cash cost (including royalties)	114.00	US$ / oz Au
All-in cost	778.60	US$ / oz Au
Capital expenditure (Life-of-Mine)	74.60	US$ M
Initial capital investment (excl working capital)	48.60	US$ M
Peak funding	55.83	US$ M
Deferred and sustaining capital	25.10	US$ M
Closure Cost	0.90	US$ M
Pre-Tax Economics		
Free cash flow after cost allocation (undiscounted)	148.73	US$ M
Internal rate of return (IRR)	67.1	%
Project NPV (discounted at 5.0%)	116.99	US$ M
Payback period	1.50	years
After-Tax Economics		
Free cash flow after cost allocation (undiscounted)	112.54	US$ M
Internal rate of return (IRR)	52.3	%
Project NPV (discounted at 5.0%)	86.54	US$ M
Payback period	1.50	years

2.0 INTRODUCTION

2.1 Issuer

This report is a preliminary economic assessment (PEA) of the mineral resource estimate, mining geotechnical, potential mine development and mineral processing for the Tuvatu Gold Project in Fiji.

At the request of Mr Stephen Mann, Managing Director of Lion One Metals Limited ('Lion One'), the original issuer, Lycopodium Minerals Pty Ltd ('Lycopodium') was commissioned on 11 August 2014 to prepare a PEA for the proposed mine and processing facility at the Tuvatu Gold Project. Following the completion of an initial draft, Canenco Canada Inc. (Canenco) was commissioned to finalize the final PEA report. As many aspects of the original draft have been reviewed and updated, this study is a revised update of the Lycopodium initial draft. Several sections or portions from that draft report have been used in this report with updates included as appropriate.

This assessment utilised the updated resource model and estimate by Mining Associates Pty Ltd ('MA'), May 2014 and a further update of that resource by Mining Associates Pty.Ltd dated January 2015 and includes contributions from AMC Consultants Pty Ltd ('AMC'), Knight Piésold Pty Ltd ('KP') and Canenco.

2.2 Terms of Reference and Purpose

Lion One Metals Limited intends that this report be used as an Independent Technical Report as required under Part 4 'Obligation to File a Technical Report', of Canada’s National Instrument 43-101 Standards of Disclosure for Mineral Projects ('NI43-101 Standards').

At Lion One Metals Limited’s request, this report includes:

- The MA May 2014 resource estimate encompassing all mineralized veins at the Tuvatu Gold deposit, and its subsequent update using smaller sub blocks dated January 2015.
- A preliminary new mine design and development schedule.
- A preliminary mineral processing plant and tailings storage facility design.
- A description of all related services and infrastructure to support the above.
- A report that satisfies Part 4 Section 4.2 of Canada’s National Instrument 43-101 Standards of Disclosure for Mineral Projects.

2.3 Information Used

This report is based on technical data provided by Lion One Metals Limited to the various subconsultants. Lion One Metals Limited provided open access to all the records necessary to enable a proper assessment of the project and resource estimates. Lion One Metals Limited has warranted in writing to MA that full disclosure has been made of all material information and that, to the best of the Lion One Metals Limited's knowledge and understanding, such information is complete, accurate and true. Readers of this report must appreciate that there is an inherent risk of error in the acquisition, processing and interpretation of geological and geophysical data, and MA takes no responsibility for such errors.

Additional relevant material was acquired independently by the subconsultants from a variety of sources. The list of references at the end of this report lists the sources consulted. This material was used to expand on the information provided by Lion One Metals Limited and, where appropriate, confirm or provide alternative assumptions to those made by Lion One Metals Limited.

Geological and metallurgical information at this level of study still contains a number of unknowns such that the true nature of any body of mineralization is never known until the last tonne of ore has been mined out, by which time exploration has long since ceased. Exploration information relies on interpretation of a relatively small statistical sample of the deposit being studied; thus a variety of interpretations may be possible from the fragmentary data available. Investors should note that the statements and diagrams in this report are based on the best information available at the time, but may not necessarily be absolutely correct. Such statements and diagrams are subject to change or refinement as new new data is made available, or new research alters prevailing geological or metallurgical concepts. Appraisal of all the information mentioned above forms the basis for this report. The views and conclusions expressed are solely those of the contributing subconsultants except where conclusions and interpretations credited specifically to other parties are discussed within the report, then these are not necessarily the views of MA, AMC, KP, or Canenco.

2.4 Site Visit by Qualified Persons

The summary review of geology and resource models and estimates and the site visit was conducted by Mr Ian Taylor, BSc (Hons) MAIG (QP) who visited the site from 25 to 28 February 2014. Mr Taylor viewed the geological setting, located some drill collars, inspected drill core, and sample storage.

Mr Taylor has sufficient experience which is relevant to the Tuvatu style of mineralization and deposits under consideration and to the activity which he is undertaking to qualify as a Qualified Person as defined in NI43-101 (Canada). He is a Member of The Australian Institute of Geoscientists. Mr Taylor is employed by Mining Associates Limited of Brisbane, Queensland.

The review of historical and current mine designs and the site visit was conducted by Mr David Lee, B.Eng Ming (Hons), FAusIMM (QP) who visited site from 8 to 11 September 2014. Mr Lee viewed the regional setting, proposed portal and open cut locations, the existing mine portal and the drill core in storage.

Mr Lee has sufficient experience which is relevant to the proposed mining operation at Tuvatu and the mine planning and economic evaluation that he is overseeing to qualify as a Qualified Person as defined in NI 43-101 (Canada). He is a Fellow of The Australasian Institute of Mining and Metallurgy. Mr Lee is employed by AMC Consultants Pty Ltd in Perth, Western Australia.

The site inspection of the proposed sites for development of the process plant, support infrastructure and tails storage and the geotechnical test pits and borehole cores was undertaken by Mr David Morgan, BSc Civil Engineering, MAusIMM (CP) who visited site from 8 to 10 September 2014.

Mr Morgan has sufficient experience which is relevant to the proposed process plant and surface infrastructure installation proposed for Tuvatu and the site development and environmental constraints to qualify as a Qualified Person as defined in NI43-101 (Canada). He is a Member of The Australasian Institute of Mining and Metallurgy. Mr Morgan is employed by Knight Piésold Pty Ltd in Perth, Western Australia.

The review of historical metallurgical testwork, and the design of the process plant for the proposed operation at Tuvatu was undertaken by Mr. Stacy Freudigmann P.Eng. who visited the site from the 8th to the 14th of April 2014.

Mr. Freudigmann has sufficient experience, which is relevant to the proposed processing route for Tuvatu that he qualifies as a Qualified Person as defined in NI 43-101 (Canada). He is a member of the Australian Institute of Mining and Metallurgy and the Canadian Institute of Mining and Metallurgy. Mr. Freudigmann is employed by Canenco Canada Inc. in North Vancouver, British Columbia.

3.0 RELIANCE ON OTHER EXPERTS

Canenco and the other sub-consultants involved in preparing this PEA report have assumed, and relied on the fact, that all the information and existing technical documents listed in the References section of this report are accurate and complete in all material aspects. While all the available information presented has been carefully reviewed, its accuracy and completeness cannot be guaranteed. We reserve the right, but will not be obligated to revise our report and conclusions if additional information becomes known to us subsequent to the date of this report.

In general, copies of the tenure documents, operating licenses, permits, and work contracts were not reviewed and an independent verification of land title and tenure was not performed. MA did not verify the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties.

Land tenure and other select technical data as noted in the report were provided by Lion One and the issuers have relied on the integrity of such data.

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

4.0 PROPERTY DESCRIPTION AND LOCATION

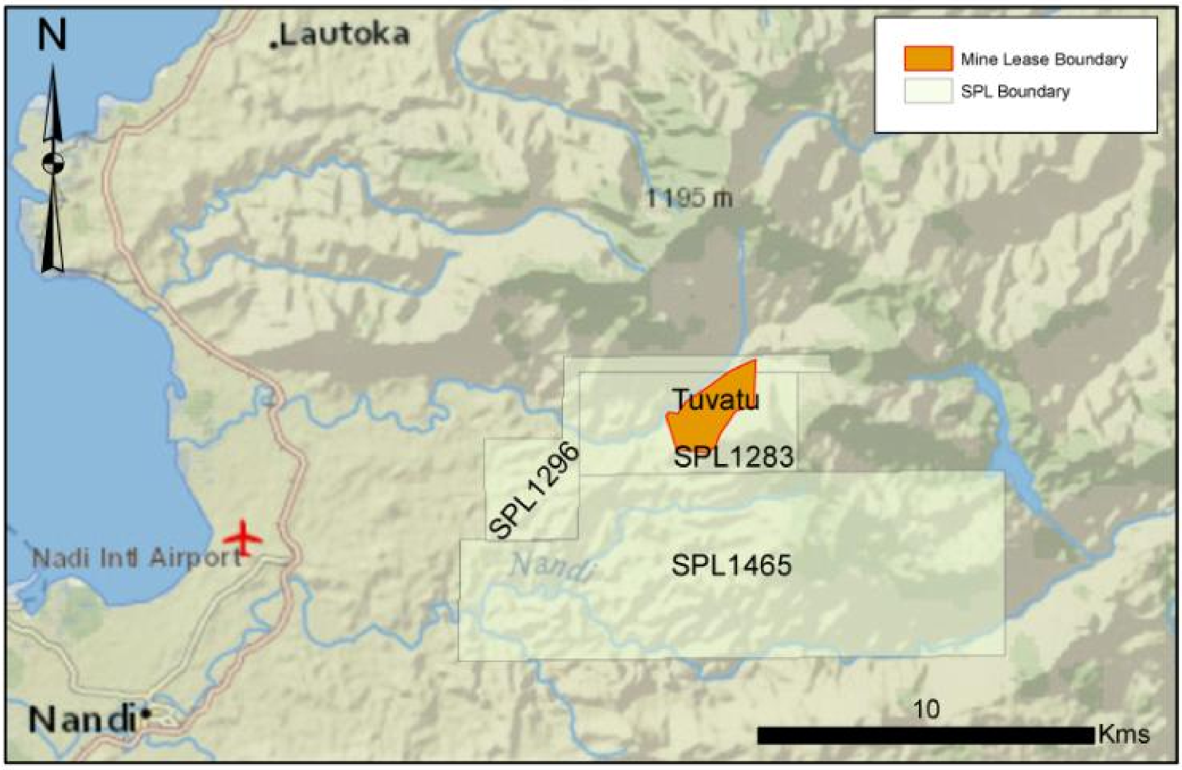
4.1 Property Area

Title to three Special Prospecting Licenses (SPL) (total area 11,794 ha) is held by Lion One Limited, a subsidiary company of Lion One Metals Limited (Table 4.1, Figure 4.1). A Special Mining Lease (371.68 ha) covering the current mineral resource at Tuvatu was granted to Lion One on January 22nd, 2015.The status of the tenements has not been independently verified by MA. Letters from the Mineral Resources Department, Fiji to Lion One Limited have been viewed.

Table 4.1 Tuvatu Tenement Details

Tenement	Area Hectares	Annual Expenditure Required	Date of Grant	Term	Interest
SPL1283	1,951	FJD$700,000	Sept 19, 2013	July 1, 2013 to June 30, 2016	100%
SPL1296	1,315	FJD$700,000	Sept 19, 2013	July 1, 2013 to June 30, 2016	100%
SPL1465	8,900	FJD$600,000	Dec 23, 2013	Dec 2, 2013 to Dec 1, 2016	100%
SML62	371.68		Jan 22, 2015	Jan 22, 2015 to Jan 21, 2020	100%

Figure 4.1 Project Location (Source: Lion One Metals, 2015)



4.2 Property Location

The tenements are located in the upper reaches of Sabeto Valley approximately 24 km northeast of Nadi on the west coast of Viti Levu, and 15 km from the Nadi International Airport. The Tuvatu gold deposit is located within SML62 which is surrounded by SPL1283 and SPL1296. SPL1465 is a contiguous lease with the existing Tuvatu leases extending to the south to cover additional prospective geology and to cover the area that was previously demarcated for a tailings dam by Tuvatu Gold Mines (TGM) in its 2000 mining study.

MA has not undertaken any title search (other than viewing on-line government tenement map databases) or due diligence on the tenement titles or tenement conditions and the tenement's status has not been independently verified by MA.

4.3 Property Ownership, Rights and Obligations

Title to the property is held by Lion One Limited, a subsidiary of Lion One Metals Limited.

There are three types of land in Fiji; native land, crown land and freehold land. The project area lies mostly, if not all, within native land, classified as native reserve land. This means that Lion One has to acquire consent through signatures of a minimum of 75% of adult members of the Land Owning Unit ("LOU") for the land to be de-reserved. Lion One must then negotiate for a land lease that will require the consent of 50% of adults in the LOU.

There are also native Fijian leaseholders in the project area with whom Lion One must consult in its acquisition plans. Compensation agreements must be finalized with these leaseholders to gain access to their lease areas.

All land covered by the SPLs is native land which comes under the control of the Native Land Trust Board (NLTB) on behalf of the native owners. About 5% of the SPLs are under cane lease, through the Agricultural Land and Tenants Act (ALTA).

Native land is vested in the NLTB under the Native Land Trust Act which means that only the NLTB may grant any legal interest in native land. Most, if not all, the land required by Lion One for its mining tenements and native leases are within native land reserve which cannot be leased out to any non-Fijian unless such land is de-reserved.

4.4 Royalties, Agreements and Encumbrances

To the extent known by MA, there are no option agreements or joint venture terms in place for the property.

To the extent known by MA, there are no known obligations on ground covered by claims comprising the property.

In the Republic of Fiji, a royalty is payable to the state government when a mineral is sold, disposed of or used. The Fiji Mineral Resources Act 1989 requires that the holder of a mining lease or mining claim lodge a royalty return and any royalty is payable at least annually for all leases and claims held, even if no production took place but saleable metal was won. The Minister allows samples with small quantities of gold to be sent for analysis, however, under the law in Fiji, trial mining and bulk sampling can be carried out and any significant gold won as determined by the Minister will be subject to royalties. Royalties for the Tuvatu Property will be 5% of the value of precious metal exported. This royalty is then split with parts compensating the community and other stakeholders.

Lion One has entered into a Surface Lease agreement with the iTaueki Land Trust Board (“TLTB”) which governs the native land ownership rights in Fiji. The TLTB manages the lease agreements between native land owners and tenants. The Surface Lease between the Company and the TLTB is required prior to obtaining a mining lease from the MRD. Under the terms of the Surface Lease, the Company must make a one-time payment of FJD$1,000,000 of which FJD$700,000 (CAD$427,348) has been paid upon acceptance of the Surface Lease agreement and the balance of FJD$300,000 (CAD$175,260) is due upon the first gold production from mining operations in Tuvatu. An additional lease payment of FJD$30,000 (CAD$17,526) is payable per annum to the local communities for education and community development over the 21-year term of the Surface Lease agreement.

Compensation agreements between Lion One and the native title landowners have been signed in respect of land disturbance within the SPLs in July 2013 and SML62 in June 2014 (Lion One, 2015).

4.5 Environmental Liabilities

To the extent known by MA, the Company has complied with the preparation and submission of all required environmental studies and documents, and there are no environmental liabilities on the property.

4.6 Required Permits for Exploration Work

In Fiji the investor's right to continue exploration/development programs is written in the Mining Act. The guiding principle of Fiji's mineral investment policy is that Government assumes that the grant of an exploration licence implies a right to proceed to eventual project development. This is subject to the licence holder maintaining a vigorous geological and/or feasibility study program approved by the Minister responsible for Mineral Resources.

4.7 Other Significant Factors and Risks

To the extent known by MA there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

Tuvatu lies on the west coast of Viti Levu, 24 km northeast of Nadi town and approximately 15 km by road from the Nadi International Airport. The area is steep and rugged, and access is via the Sabeto Road which is sealed for about half the distance.

The Sabeto road turnoff is located approximately 10 minutes north of the Nadi International Airport. The Sabeto road follows the Sabeto River on its western side. The electricity pylons of the Monasavu Hydro line can be seen. Further along the Sabeto road, the road forks, with the left fork going to Korobebe village, and on to Navilawa village. Tuvatu is located on the road to Navilawa.

SPL1283 and SPL1296 cover land areas in the upper catchment of the Sabeto River immediately south of Navilawa village. The tenements are bounded to the southeast by the Namotomoto ridge. Nagado village is located on this ridgeline. The Korobebe village is located on the banks of the Sabeto River about 4 km southwest of the Tuvatu prospect and further downstream are the villages of Naboutini, Koroyaca and Sabeto. On the opposite side of the river from Sabeto village is Natalau village. Indian cane farmers lease the land in between the Fijian villages.

Nadi is the closest city and is serviced by direct daily flights from Brisbane, Melbourne, Sydney and Auckland by several Australian airlines. Tuvatu is readily accessed from Nadi International Airport by the Sabeto Road. A network of local formed roads and pastoral tracks provides good access to most of the project area. During the wet season (November to March), major and minor creeks may be impassable for some days. In wet weather, four wheel drive vehicles are required to access the tenements. Creeks and adjacent areas are generally thickly vegetated while the spurs and ridges are dominated by open grasslands.

5.2 Climate

Fiji experiences a mild tropical South Sea maritime climate without great extremes of heat or cold. Winds are generally light to moderate and blow from E-SE during all seasons. Temperatures average 22°C for the cooler months (May to October) while November to April temperatures are higher with heavy downpours.

The islands lie in an area occasionally traversed by tropical cyclones. These are mostly confined to the period November to April, with greatest frequency around January and February. On average, some ten to twelve cyclones per decade affect some part of Fiji, with two or three causing severe damage. Specific locations may not be directly affected for several years but the dominant north-west tracks give some increased risk of damage in the outlying north-west island groups.

Viti Levu's climate is dominantly controlled by oceanic temperatures and winds, restricting the diurnal temperature range heavily and the average daily range is 8.5°C to 10.3°C. Average minimum temperatures for Nadi range from 18°C to 23°C while average maximums range from 28°C to 32°C; it can be expected that these are a good guideline for the Tuvatu area, given its close proximity to Nadi. Mean rainfall in the area varies from 50 mm in July to a high of 300-325 mm during the December to March wet season.

5.3 Local Resources

Tuvatu is located within the upper reaches of the Sabeto Valley. The area hosts a number of small villages that are dependent on the local waterways (e.g. Sabeto River) to supply water for local sustainable agricultural practices such as sugar cane, coconut oil and fruits and vegetables. English is the official language. However, Fijian and Hindi are also taught in schools as part of the school curriculum.

The major towns in close proximity to the Tuvatu area are Lautoka, Nadi and Ba (Table 5.1). Lautoka, Fiji's second-largest city, is located 30 km from Tuvatu. The local economy still relies heavily on the sugar industry and the Lautoka Sugar Mill has been operating since 1903. Nadi is Fiji's third-largest city and a tourist and business hub due to the presence of the Nadi International Airport.

The major land use in the Tuvatu region is pastoral, with most income generated from sugar cane, copra and rice production. Fishing, manufacturing and tourism industries are also employers in the region. Any skilled workforce for a mining development in the region would be expected to be drawn from coastal Nadi-Lautoka-Ba region. There are also experienced former mine workers from the Vatukoula Gold Mine.

Table 5.1 Population Centres (2007)

Town	Population	Principal Activity
Lautoka	52,220	Agriculture, tourism, fishing
Nadi	42,284	Tourism, manufacturing
Ba	18,526	Agriculture, fishing, mining

(Source: Fiji Bureau of Statistics)

5.4 Infrastructure

Fiji is one of the most developed of the Pacific island economies, although a large subsistence sector still exists. Sugar exports, remittances from Fijians working abroad, and a growing tourism industry (with 400,000 to 500,000 tourists annually) are the major sources of foreign exchange. Sugar processing makes up one-third of industrial activity.

Little infrastructure exists within the local area proximal to the Tuvatu project other than a small exploration facility. Local villages utilise a combination of traditional and modern practices but do not contain any significant infrastructure. The majority of regional infrastructure, such as transport, telecommunication and energy revolve around the nearby cities of Nadi and Lautoka.

Nadi is equipped with modern technology for both its internal and international telecommunications. All major towns have digital telephone exchanges and the islands are linked by cable and satellite to worldwide networks. The project area is covered by GSM mobile-phone reception.

The Fiji Electricity Authority ("FEA") holds the monopoly in all facets of the energy sector including generation, transmission and distribution. Hydroelectric and diesel are the two sources of power generation for the FEA. Its supply capacity currently stands at 180 megawatts, however rising use of electricity has prompted government to call for submissions from independent power producers. The FEA has an 11 kV line at Korobebe village, which could supply 2 MW of power. This line could be upgraded by the Fiji Electricity Authority to 33 kV from the Sabeto turn-off to the mine site. The villages around Tuvatu chiefly utilise fuel wood and small diesel generators.

5.5 Physiography

The upland areas of the Tuvatu project area are grassland. Stream valleys and their perimeters are heavily vegetated. Several intermittent and perennial streams are located within the prospecting licenses. Elevation of the Tuvatu property ranges from 50 m to a maximum of 700 m. The area is hilly with slopes of 15%-30 % being common.

6.0 HISTORY

6.1 Previous Ownership

Historical activities began during the early part of the 20th century with prospecting in the upper reaches of the Sabeto River with no evidence that the mineralized lodes at Tuvatu were discovered. Some pitting and limited underground work took place between 1945 and 1952 when Bayley and Bryant operated PL 689. Later work in the area was undertaken by the Nadele Syndicate.

In the period from 1977 to 1979 Aquitaine Fiji explored the Tuvatu area. In 1987, Geopacific Ltd pegged out SPLs 1283 and 1296. During the next ten years, Geopacific Ltd invested approximately $1.5M in exploration at Tuvatu. For three of these years, Geopacific Ltd was in association with Noranda Pty Ltd. In December 1995, Geopacific Ltd entered into an option agreement with Emperor Mines Ltd. and in June 1997, Emperor exercised its option to purchase 100% of the tenements. Emperor then incorporated the Tuvatu Gold Mining Company Limited (“TGM”), a subsidiary of Emperor Gold Mining Company, to manage the property.

In 2007 following the closure of the Vatukoula gold mine Emperor Gold Mining Company (at the time a subsidiary of DRD Ltd), sold its Fijian assets including the Tuvatu property to Westech Gold Pty Ltd and Red Lion Management Ltd. Licenses covering the Tuvatu property were re-issued in the name of Lion One by the Fijian Government. Subsequently American Eagle Resources gained control of Lion One Limited, the holder of the Tuvatu project. Lion One Metals is the product of the reverse takeover in January 2011 of X-Tal by American Eagle Resources.

6.2 Previous Exploration

All historical work described in this section was conducted within the tenements currently held by Lion One Limited.

6.2.1 Tuvatu Project Area

Some pitting and limited underground work was undertaken by Bayley and Bryant between 1945 and 1952 when they operated PL 689. Later geological work undertaken by the Nadele Syndicate included the pitting of two lodes, trenching and driving an adit but no records of the syndicate’s work have been located.

Aquitaine Fiji explored the area from 1977 to 1979 and located a soil anomaly of 1.4 g/t Au, which was not pursued. In 1987, Geopacific Ltd pegged out SPLs 1283 and 1296 in the area and investigated the soil anomaly previously identified by Aquitane Fiji. Geopacific discovered the outcrop of what is now called the Tuvatu lode in the vicinity of the soil anomaly.

From 1995 to 2001 TGM conducted 3 phases of exploration at Tuvatu. The Phase 1 programmes carried out between April 1996 and February 1998, involved initial regional geological mapping and stream sediment sampling which located the Tuvatu gold deposit in the SKL-Nasivi area. A number of geophysical surveys were also completed including a dipole-dipole IP survey and airborne magnetics / radiometrics survey. Phase 2 followed in March 1999 with subsurface exploration and development, including limited trial mining and metallurgical testing.

Phase 3 commenced in 2000 with work on a feasibility study but the study was suspended in late 2000 as part of a general cost-cutting exercise by Emperor due to the low gold price at the time.

The Phase 3 evaluation of the Tuvatu resource area included surface diamond and percussion drilling to test some peripheral anomalies as well as down-dip extensions of the various Upper Ridges lodes. The program included mine and metallurgical design, environmental plans and social acceptance issues. In addition, re-mapping of the underground development took place in order to develop a robust structural model for the area. Further metallurgical test work was also completed.

Overall there have been three programs of drilling at Tuvatu from exploration through to resource delineation. Drilling has been completed both on the surface and from the underground exploration decline. Drilling methods included both diamond drill (DD) and reverse circulation (RC).

In total TGM completed 51,484 m of diamond core drilling and 9,265 m of RC surface drilling, as well as 13,407 m of underground drilling. A total of 1,341 m of decline, strike and rise development was also been undertaken in the project area including a 600 m long exploration decline developed to a depth of 240 m below surface in the region of the Upper Ridges lodes.

Further details of the TGM drilling are located in Section 10.1 of this report

6.2.2 Regional Exploration

Only limited regional exploration had been carried out in the area by explorers (primarily Aquitaine Fiji) before TGM's work. In the 2001-2003 period a regional exploration program was carried out by TGM that involved regional mapping, trenching, stream sediment and soil sampling. This work identified more than 10 new prospect areas outside the Tuvatu mine area.

Detailed exploration was carried out by TGM at Nubunidike, Ura Creek, Jomaki, Malawai, and Kubu prospects. The Nubunidike and Ura Creek prospects were the most advanced prospects. Exploration work commenced at Qualibua in June 2002. Subsequent ridge and spur soil geochemistry located high tenor gold-in-soil anomalies at the Korobebe prospect.

Upon gaining control of the Tuvatu property Lion One commenced detailed mapping and geochemical sampling. Work concentrated on the region south of the Tuvatu resource area and around Qalibua Creek to the north. Two surface diamond drillholes were completed in October 2008 at the Nubunidike prospect to test the Nubunidike / Hornet Creek / 290 Vein system.

6.3 Historic Resource and Reserve Estimates

A number of historical mineral resource estimates were carried out at Tuvatu by previous operators over the period from 1997 to 2000. A qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves. LionOne is not treating the historical estimates as current mineral resources or mineral reserves and the historical estimates are quoted for information and targeting purposes only.

The current resource statement presented in Section 14 in this document supersedes all previous resource figures

6.3.1 TGM 1997 Upper Ridges Resource

A resource figure was calculated internally by TGM in September 1997 for the Upper Ridges area as 904,000 tonnes at 5.1 g/t Au (149,272 ounces). This was a vein-style polygonal estimate with 25 m radius polygons being drawn on long sections in the plane of each hole. No lower cut-off was applied.

6.3.2 TGM 1998 Upper Ridges Resource

The resource was updated using similar methodology in February 1998. Using a lower cut-off for each intersection of 2 m-grams, a boundary was drawn around all intersections greater than 2 m-grams. Continuity of veining beyond 25 m was assumed where no conflicting evidence occurred. Equal weighting was given to each intersection within the model boundary of the lode when calculating average width and average grade of the lodes. A density of 2.7 g/cm^3 was used. An overall resource figure for the Upper Ridges lodes of 602,000 tonnes at 8.2 g/t Au was calculated for a total of 159,362 ounces (Table 6.1).

Table 6.1 Summary of Upper Ridges Resources, TGM Feb 1998

Lode	True Width	Tonnes	Grade (g/t)	Ounces
U Ridges West 1	1.00	16,462	24.74	13,093
U Ridges West 2	1.71	96,648	4.65	14,449
Standing Stone	1.57	147,979	7.00	33,303
Upper Ridges 1a	1.63	97,570	10.79	33,848
Upper Ridges 1b	1.81	17,201	14.41	7,947
Upper Ridges 2	2.39	101,441	11.45	37,343
Upper Ridges 3a	1.41	76,198	4.21	10,314
Upper Ridges 3b	1.32	18,194	8.45	4,943
Upper Ridges 4	1.00	30,672	4.18	4,122
Total	**1.69**	**602,365**	**8.23**	**159,362**

This estimation extrapolated the continuity of the veins over vertical and lateral distances of more than 25 m. If polygons around individual intersections were restricted to 25 m then the inferred tonnages decreased by the order of 50-60% reflecting the lower density of drilling in the southern part of the Tuvatu Mine. Additional drilling was considered to be required to infill this resource area in order to upgrade the area to Indicated Resource category.

6.3.3 Geoval 1998 Global Resource

Between September 1997 and February 1998, resource consultants Geoval completed a resource calculation for the Tuvatu Lode and Nasivi-SKL stockwork area. Estimated used a 3D “service variable” block modelling technique using 2.0 m composites and a 1.0 g/t cut-off. A revised resource figure for the Murau Flatmake was calculated using the September 1997 Geoval block model after the area included in the February 1998 Geoval re-calculation of the Nasivi-SKL stockwork was excluded. Global resource figures for the various lodes are summarised in Table 6.2.

Table 6.2 Summarised Tuvatu Resource (Feb 1998)

Lode	Lower Cut-Off	Tonnes	Grade (g/t)	Ounces
Upper Ridges	2.00 m-g	602,000	8.2	159,362
Nasivi / SKL	1.00 g/t	323,000	3.5	36,244
Murau	1.00 g/t	196,000	2.2	14,249
Tuvatu	1.00 g/t	103,000	2.2	7,186
Total	**1.00 g/t**	**1,225,000**	**5.5**	**217,041**

6.3.4 TGM 1999 Upper Ridges, Nasivi and SKL Resource

After the completion of the Phase 2 work programme by TGM in July 1999, an additional resource calculation was completed for the Upper Ridges area based upon data gained from underground development and surface and underground drilling (Table 6.3). A polygonal estimation was carried out internally by TGM with 25 m radius polygons being drawn on longitudinal sections in the plane of each interpreted lode. Using a lower cut-off for each intersection of 2 m-grams a boundary was drawn around all intersections greater than 2 m-grams.

In contrast to previous calculations an upper cut of 30 m-grams was applied where applicable. This figure was established by plotting a log normal cumulative frequency plot of all available Upper Ridges data and measuring the m-gram figure at the 95 percentile level. Continuity of veining beyond 25 m was assumed where no conflicting evidence occurred. Equal weighting was given to each intersection within the model boundary of the lode when calculating average width and average grade of the lodes. A density of 2.7 g/cm^3 was used.

Table 6.3 Upper Ridges Resource (July 1999)

Lode	Lower Cut-Off	Tonnes	Grade (g/t)	Ounces
Indicated	2.0 m-g	19,300	9.7	6,020
Inferred	2.0 m-g	964,000	7.8	241,775
Total	**2.0 m-g**	**983,300**	**7.8**	**247,795**

Two areas of indicated resource were calculated for the UR1 South strike drive area and the UR2 North strike drive area based on geological and channel sampling data on 2 m centres in the development underground. It was assumed that the structures could be extrapolated for a minimum of 25 m in the vertical orientation based upon results from the developed rise on the UR1 South and UR2 South lodes.

The resource figures calculated using development sampling data for the UR1 South and UR2 North development were found to be within 10% and 15% respectively of the figures calculated from drill holes. In addition it was found that the sampled grade of each of these areas was higher than indicated by drilling.

Upon completion of the Phase 3 drilling programme, the geological model was updated and a new manual resource calculation was completed by TGM using the same parameters as for the July

1999 estimate. Figures previously calculated by Geoval for the Nasivi-SKL area were superseded by resource figures for the GRF steep shear and the Murau lodes. Updated resource figures were not calculated for the SKL flatmakes. Preliminary figures were also calculated for the West lodes, located 500 m west of the Nasivi-SKL area.

6.3.5 Vigar 2000 Resource

In April 2000, Andrew Vigar of Vigar and Associates (“VA”) was commissioned to review the geology and resource estimates detailed by TGM. Further to this VA constructed a geological and resource model for the Upper Ridges lodes and estimated geological resources for each lode. Indicated resource estimates were subsequently converted to reserves using economic cut-offs, minimum mining widths and dilution.

In August 2000, following a verification of the Tuvatu database, it was found that a number of intercepts used in the April 2000 resource estimate had been excluded from the model and VA and Associates were commissioned to revise the resource calculation

Lodes, geological units, workings and resource zones in the Upper Ridges area (as well as the GRF lode) at Tuvatu were defined as a series of closed wire-frames. Each wire-frame is made up of a series of connected triangles which fully enclose a volume and is referred to as a “solid”.

Lodes were modelled as mutually exclusive wire-frames, one for each lode (Table 6.4). The lode widths were taken from the wire-frames. All drill-holes intersecting the structure were used, whether mineralized or not.

Table 6.4 Lode Wire Frame –Drillhole Details

Wireframe	Number of intercepts	WF Volume (m3)
UR1	69	117710.3
UR2	133	300234.5
UR4	45	79406.8
UR5	48	127694.4
UR6	36	102722.7
UR7	15	97685.6
UR8	11	104455.7
URW1	93	220146.2
URW2	72	151057.8
URW3	88	174019.1
GRF1	24	38271.1

A total of 41 lodes were identified of which 37 had sufficient intercepts to be modelled in the resource estimate. True widths were used and a mining width of 1.2 m allowed, fully diluted at zero grade.

One block model to accommodate the major lodes was created to contain the grade model and allow for tonnage, grade and reef width estimates. The blocks were set at 9.6 m x 9.6 m x 9.6 m, with sub-blocking down to 1.2 m. Ordinary Krige estimation was used for the grade of each block. Data used for the calculation were drill lode composites where an upper cut of 75.0 Au m-grams was applied on the raw drill data prior to lode compositing.

Lode blocks were filled with grades using the estimation of a width*grade accumulation using ordinary kriging and calculation of grade using the local block model width. This method also removed any bias with direct estimation of grades where wire-frame volumes were not adjusted. Each lode was filled separately only using drill intercepts from that lode. Estimates were made as width multiplied by grade and the grade back-calculated.

The extent of the search ellipse used in the Ordinary Krige modelling of the lodes was based on analysis of the level data, geological controls and test runs to create a grade distribution that, based on experience with narrow vein deposits, was likely to be realistic.

Resources were classified in regions as Indicated or Inferred based on drill spacing, kriging variance and number of holes used in estimation of each block. A density of 2.7 t/m^3, a cut-off grade of 3.0 g/t Au and a minimum width of 1 m were applied.

Table 6.5 summarises the total resources as reported by VA in August 2000 using a 3.0 g/t Au cut-off. The resource was stated as being JORC compliant at the time. The resource estimate for the Murau and West lodes was not recalculated by VA in 2000 and is an original estimate undertaken by TGM internally in February 2000 for which there is no documentation available. No further resource drilling was conducted after 2000 until 2012.

Table 6.5 Historical Resource Figures Vigar 2000

	Indicated Resource			Inferred Resource		
Lode	**Tonnes**	**Gold Grade g/t**	**Gold ounces**	**Tonnes**	**Gold Grade g/t**	**Gold ounces**
Upper Ridges	785,100	7.9	199,408	616,200	9.8	194,150
GRF	42,700	6.8	9,335	6,000	4.7	907
Murau				89,700	6.6	19,034
West Lodes				100,300	7.3	23,540
Total	**827,800**	**7.9**	**208,743**	**812,800**	**9.1**	**237,631**

Lion One is not treating the historical estimates as current mineral resources or mineral reserves and the historical estimates are quoted for information and targeting purposes only.

The current resource statement presented in Section **Error! Reference source not found.** in this document supersedes all previous resource figures

6.4 Historic Underground Development and Sampling

A total of 1,341 m of decline, strike and rise development has been undertaken in the project area including a 600 m exploration decline.

During TGM's Phase 1 an exploration decline was developed with minor crosscut and strike drive development to evaluate the continuity and grade of the gold mineralized structures. Underground development started in November 1997 and a total of 572.40 m of development was completed to a depth of 240 m below surface. Geological mapping of the underground development and

systematic channel sampling was carried out. A total of 588 samples were found to exceed 1.0 ppm and 214 samples were found to exceed 10.0 ppm Au. The maximum value was found to be 0.6 m at 840 g/t Au for a vertical sample taken from H-Lode. In total 32 samples were found to exceed 100 g/t Au.

A number of the lodes were intersected and sampled and an underground drilling programme was undertaken. In conjunction with the underground development, 17 underground diamond drill holes (TUG-01 to 17) were completed for a total of 1,108 m of HQ diameter core. Drilling was carried out using a Longyear LM-75 electric hydraulic drilling rig. The purpose of these holes was to infill surface drilling and to assist in planning future development.

Phase 2 of exploration work at Tuvatu started in March 1998 and involved deepening of the decline in order to access the Upper Ridges lodes in the southern part of the resource area. These lodes had previously been identified during Phase 1 by surface drilling at a broad spacing. In conjunction with the Phase 2 underground development, 26 more underground diamond drill holes (TUG-18 to 43) were completed for a total of 1,374 m of HQ diameter core. Drilling was carried out using a Longyear LM-75 electric hydraulic drilling rig. The purpose of these holes was to infill surface drilling and to assist in planning development. A bulk sample of Upper Ridges' vein material from the underground development was dispatched to Vatukoula for metallurgical test work. In addition a small trial mining exercise was carried out on veining associated with the Nasivi/SKL stockwork.

During Phase 3 a series of 69 underground diamond drill holes (TUG045–113) were completed for a total of 10,926 m. Drilling was carried out using a Longyear LM-75 electric hydraulic drilling rig and a Kempe rig. These holes were drilled to infill and expand the Upper Ridges resource and test peripheral mineralized zones in the Murau area. This programme successfully extended the Upper Ridges lodes (particularly UR2) and upgraded the Phase 2 resource.

Further details of the TGM drilling are located in Section 10.1 of this report.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

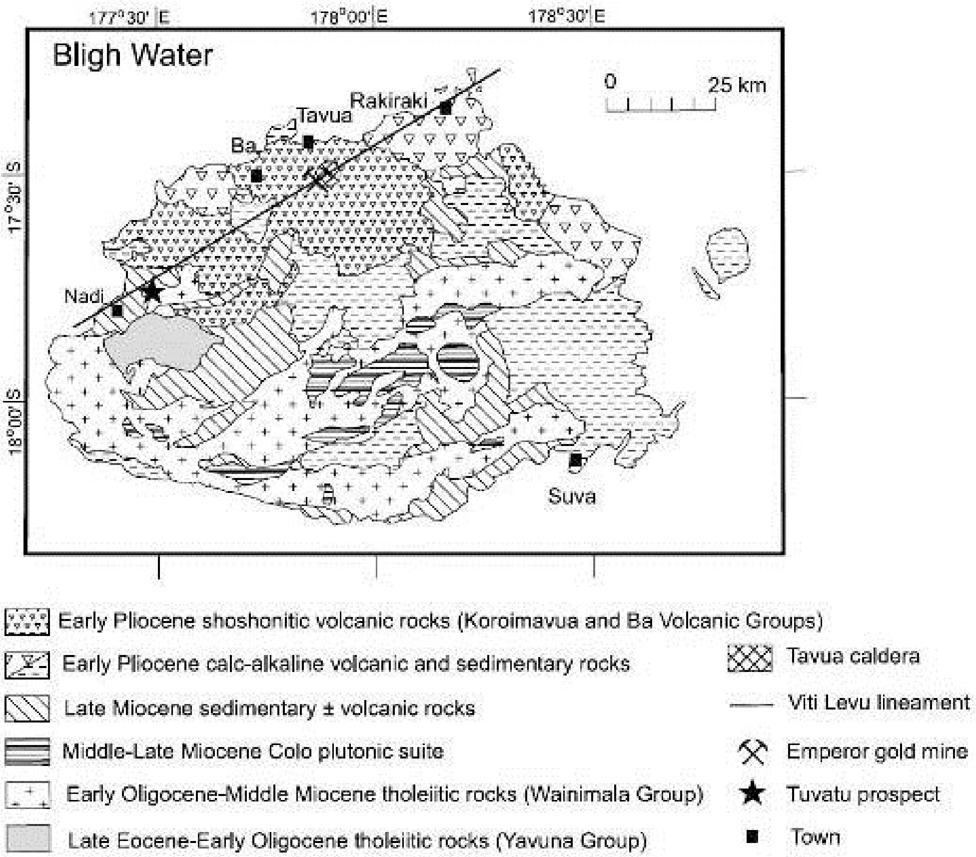
The information on regional geology is taken from Vigar, 2009.

Fiji lies on the boundary of the Indo-Australian and Pacific tectonic plates, a zone marked by seafloor spreading and transform faulting. The island is at the midpoint of the opposing Tonga Kermadec and New Hebrides convergence zones. It is separated from these actual convergence zones by two extensional back arc basins, the North Fiji Basin to the west and the Lau Basin to the east and a series of transform faults including the Fiji Fracture Zone and the Matthew Hunter Ridge. Approximately five million years ago (Miocene/Pliocene Period) the area was the site of a number of major shield volcanoes, formed along a northeast - southwest trend.

Tuvatu is one of several epithermal gold systems along the >250 km northeast trending Viti Levu lineament, which are genetically associated with alkalic magmatism (Figure 7.1). A number of gold deposits have been discovered along this trend including Tuvatu, Vatukoula and Raki Raki. The Vatukoula or Emperor Mine has produced some 7 million ounces since 1937.

Figure 7.1 Regional Geology. Location of Tuvatu Project with Respect to Viti Levu Lineament is Indicated

(Source: Spry and N. L. Scherbarth, 2005)



The oldest unit in the region is the Nadele Breccia (Late Oligocene -Middle Miocene, 29 to 23 Ma). Thin layers of sandstone and siltstone are interbedded with grits and often exhibit cross-bedding. Polymict breccias tend to be very coarse, compact and with generally angular clasts ranging from 5 mm to 200 mm (Niurou, 1997). Minor occurrences of limestone have also been noted. Pillow basalts occur as part of this sequence and can be seen in road cuttings on the project access road. The Nadele Breccia is part of the earliest volcanic activity in Fiji which took place during a period of island arc development. The volcanic units were deposited within an active fore-arc basin as proximal dispersal aprons of volcanic sediment derived from volcanic edifices (Hathway 1993).

Sabeto Volcanics (Late Miocene–Early Pliocene, 5.5 to 4.8 Ma) unconformably overlie the Nadele Breccia and represent the basal unit of the Korroimavua Volcanic Group, which is the oldest shoshonite volcanism in Fiji. The volcanics occur in a north east trending band across the north western side of Viti Levu and host a number of gold mines and prospects including Tuvatu, Vatukoula and Raki Raki. The unit consists of a series of interbedded andesitic volcaniclastics and flows. Hatcher (1997) subdivided this group into three units comprising a basal volcaniclastic breccia (30 m to 45 m), andesite porphyry flow (30 to 40 m) and volcaniclastic conglomerate (40 m). The contacts were observed dipping at 50° to 60° to the east-southeast.

A clear contact can be observed in the field at the position of the unconformity and is often accompanied by a distinct change in soil types with the red brown Nadele Breccia contrasting the grey sandy soils of the Sabeto Volcanics. High ridges and cliffs emphasise this gradation due to the resistance of the Sabeto Volcanics to weathering.

The Navilawa Monzonite (Late Miocene – Early Pliocene, 4.85 Ma) intrudes the Nadele Breccia in the northeast of the project area and hosts the majority of the mineralization. The intrusive has been divided into two phases, a central coarse to medium grain monzonite and peripheral micro monzonite. Abundant dykes cut the area ranging in composition from pegmatite to andesite, aplite and monzonite. The composition of the monzonite is equigranular with plagioclase (45%) and K-feldspar (45%) with lesser biotite and pyroxene. Considerable local variation in composition occurs with changes in grain size and inclusion of country rock. The overall intrusive complex is elongate in a north east orientation. Numerous small intrusive stocks, dominantly composed of micro monzonite also occur but tend to be elongated in a north northwest direction.

A-Izzeddin (1997) suggested that there is a spatial and temporal relationship between the emplacement of the intrusive complex and mineralization. The Tuvatu area appears to have had one to two kilometres of overburden removed since emplacement of the intrusive complex, which may represent the magma source for overlying volcanism. The gold mineralization therefore represents deep-seated hydrothermal fluids emplaced in the very upper portions of the magma complex during the waning phases of volcanism.

7.2 Local Geology

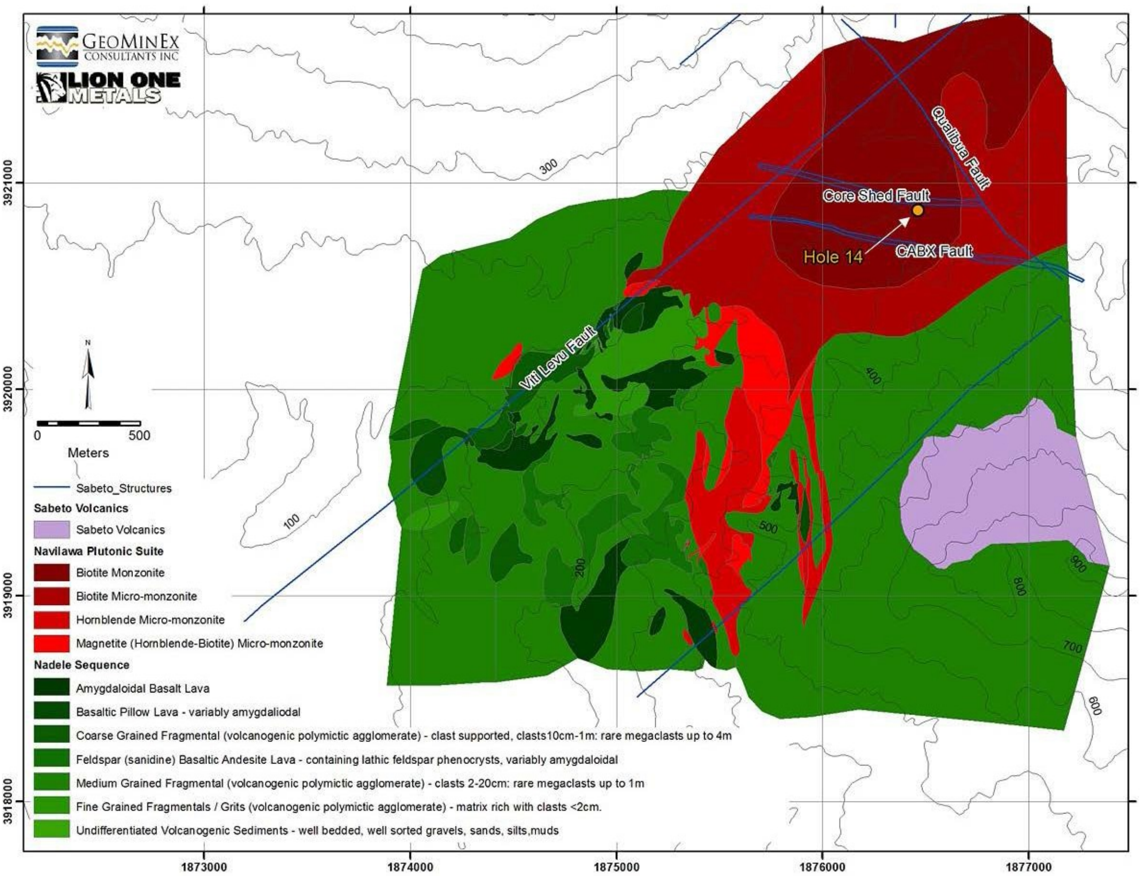
Tuvatu is one of several gold prospects known from the Sabeto area of north-western Viti Levu. Other gold and gold copper prospects in the local region are at Vuda, Navilawa (Kingston Mine and Banana Creek) and Nawainiu Creek, all associated with known or presumed centres of volcanic activity and/or volcanic core complexes within the shoshonitic Koroimavua Volcanic Group of late Miocene to early Pliocene age.

Basal units of the Sabeto Volcanics (part of the Late Miocene-Early Pliocene Koroimavua Volcanic Group) unconformably overlie Nadele Breccia in the Sabeto Valley. Members of the Sabeto Volcanics found outcropping in the area have shoshonitic affinities and include andesitic and biotite-bearing dacitic lithic and crystal tuffs, grits, agglomerates and minor flows. Shoshonites belonging to the Koroimavua Volcanic Group have been age dated at 5.88 Ma.

The volcaniclastic units were subsequently intruded by a monzonitic stock (Figure 7.2). Mapping by Emperor geologists indicated that it is a composite intrusive body with several different phases of intrusion associated with it. The monzonite within the Tuvatu prospect area is locally brecciated and varies in grain size. A series of pegmatite dykes, andesitic dykes and stocks have also intruded the area. The monzonite has been dated at 4.85 Ma, and is interpreted to be co-magmatic with the volcanic units of the Koroimavua Volcanic Group. It probably and represents the root of a caldera and is elongate in a northeast-southwest orientation.

Locally the geology is structurally complex with the area cut by a 60 m wide east-west striking fault zone referred to as the Core Shed Fault (CSF) which is exposed near the portal of the decline and can be traced for over 5 km along strike. Additional westerly striking structures locally offset veins.

Figure 7.2 Project Geology



7.3 Mineralization

Mineralization is structurally controlled and occurs as sets and networks of narrow veins and cracks, with individual veins generally ranging from 1 mm to 200 mm wide (Figure 7.3, Figure 7.4). Zones of veining which comprise the lodes may be up to 5 m wide.The main mineralized zone (Upper Ridges) comprises eleven principal lodes with a strike length in excess of 500m and a vertical extent of more than 300 m. Another major zone of mineralization (Murau) strikes east-west and consists of two major lodes with a mapped strike length in excess of 400 m.

Although gold mineralization is primarily hosted in monzonite it can also occur in the volcanic units. Veins are narrow, generally less than 1 m up to a maximum of 7 m, and gold grades are erratic. Lode mineralogy is varied, with most veins containing quartz, pyrite, and base metal sulphides.

Figure 7.3 Visible Gold in W3 Lode, TUDDH371 at 204.3 m (I.Taylor, 2013)

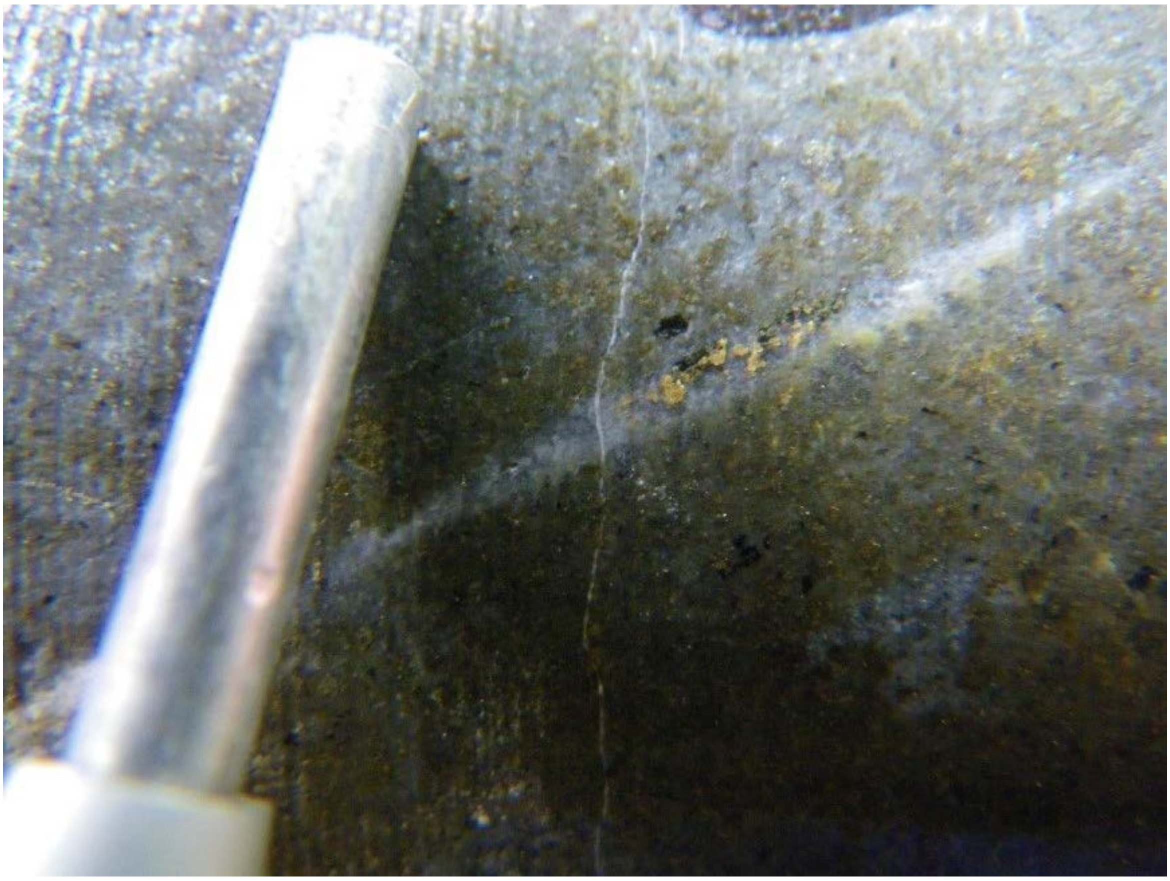


Figure 7.4 Murau Lode Outcrop (I.Taylor, 2013)

A very high proportion of the gold occurs as either free gold or is contained in quartz or pyrite composite particles that can be floated. Free gold present is both fine and coarse grained. Mineralization contains minor amounts of deleterious elements such as arsenic, selenium, and uranium.

A number of different lode structures were identified by TGM geologists in the Tuvatu resource area, and zones of veining which comprise the lodes may be up to 5 m wide. The main lode structures identified by TGM are shown in Figure 7.5 and comprise 10 lodes in the Upper Ridges area, 2 lodes in the Murau area, 3 lodes in the West (Plant Site) area, 2 lodes in the Tuvatu area and 3 lodes in the SKL area. Lodes were re-interpreted by Lion One geologists following infill and resource extension drilling (Figure 7.6).

Figure 7.5 Plan View of Tuvatu Lode Structures (historic interpretation)

(Source: P. G. Spry and N. L. Scherbarth, 2005).

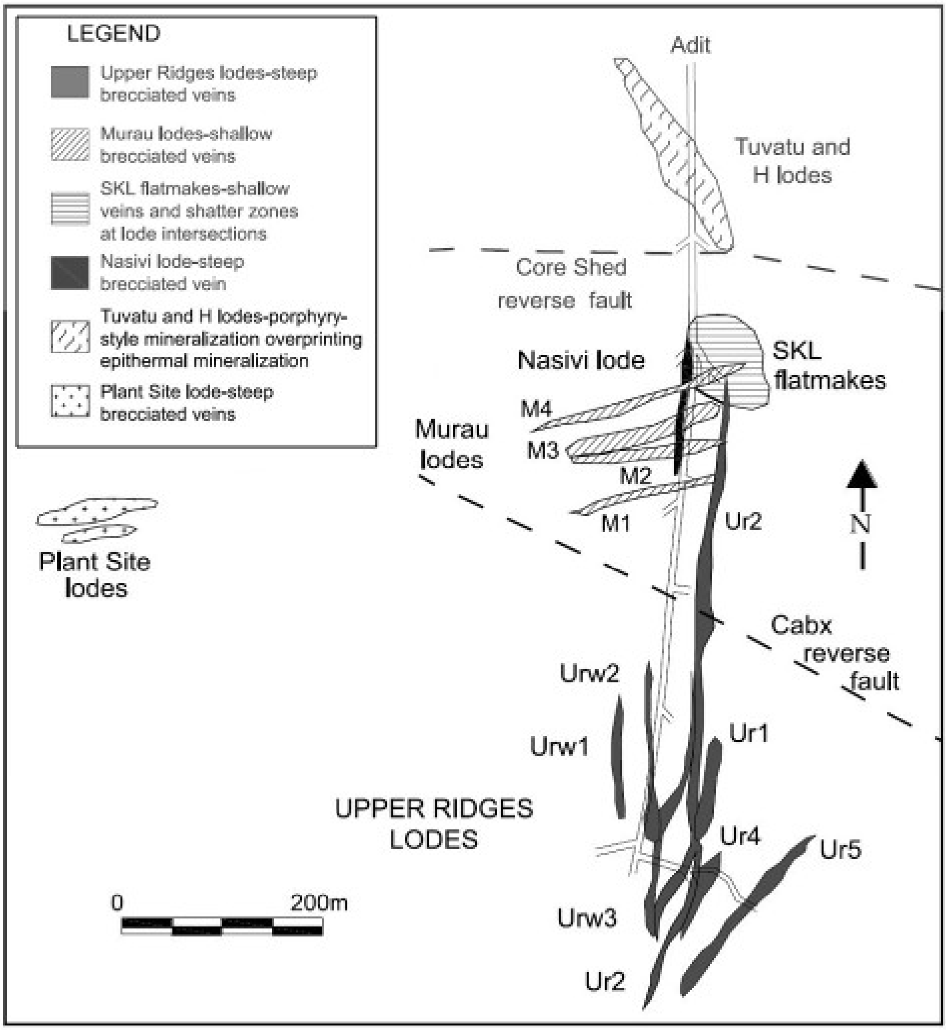
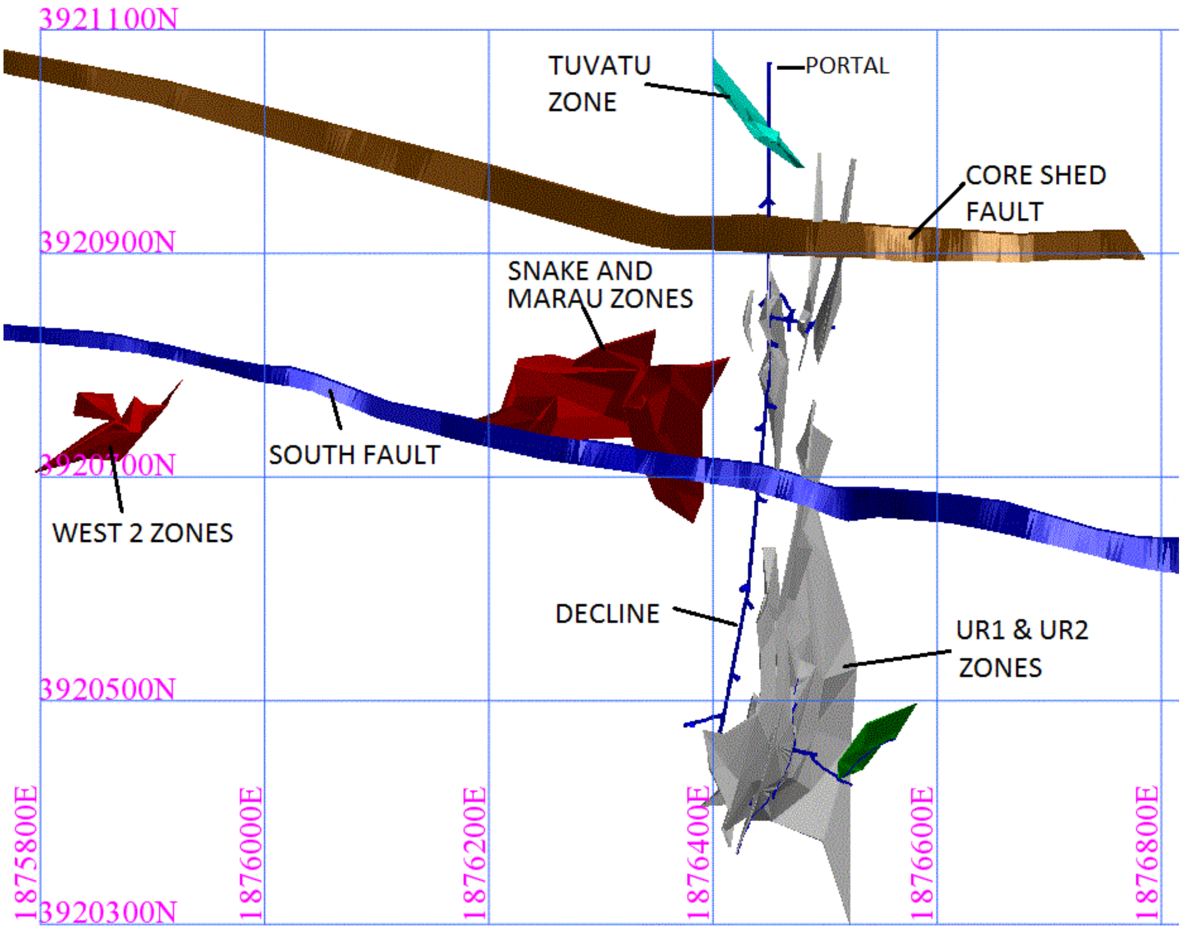


Figure 7.6 Current Lion One Interpretation of Mineralized Zones



In addition a number of other lodes have been identified in the local area but remain untested. The grades of individual lodes vary considerably due to the “spotty” nature of the gold and the variability in width of the host structures. Average grades for the lodes range from 2.0 g/t to 10.0 g/t. Gold mineralization tends to be quite coarse and visible gold is often observed in mineralized sections of core (Figure 7.3).

7.3.1 Structural Controls

Gold mineralization at Tuvatu is considered to have developed during an episode of northeast-southwest shearing and is intimately related to but postdates the emplacement of a high level monzonite intrusive.

7.3.2 Dimensions and Continuity

Mineralization is generally hosted in a series of sub-vertical, north and north-northeast striking trending veins as well as shallow, south dipping veins (locally referred to as “flatmakes”). In spite of the narrow widths of individual veins the gross lode structures appear to be continuous for over one hundred metres. The majority of lodes vary in width from 0.5 m to 5.0 m with an average width of 1.1 m. (individual vein intercepts have been recorded as low as 4 cm)

The Tuvatu and H Lodes are up to 5 m wide and are characterized by porphyry-style copper mineralization. The H Lode is crosscut locally by epithermal gold veins, and the Tuvatu Lode is characterized by potassic alteration and hosts chalcopyrite and biotite.

7.3.3 Paragenesis

Scherbarth and Spry (2006) suggest that the mineralized zone at Tuvatu may have originally developed as a porphyry copper system which was overprinted by epithermal gold mineralization. The style of mineralization is thought to have evolved as the local monzonite intrusives cooled and meteoric waters mixed with the magmatic fluids, resulting in the gradational changing of the mineralization and alteration styles.

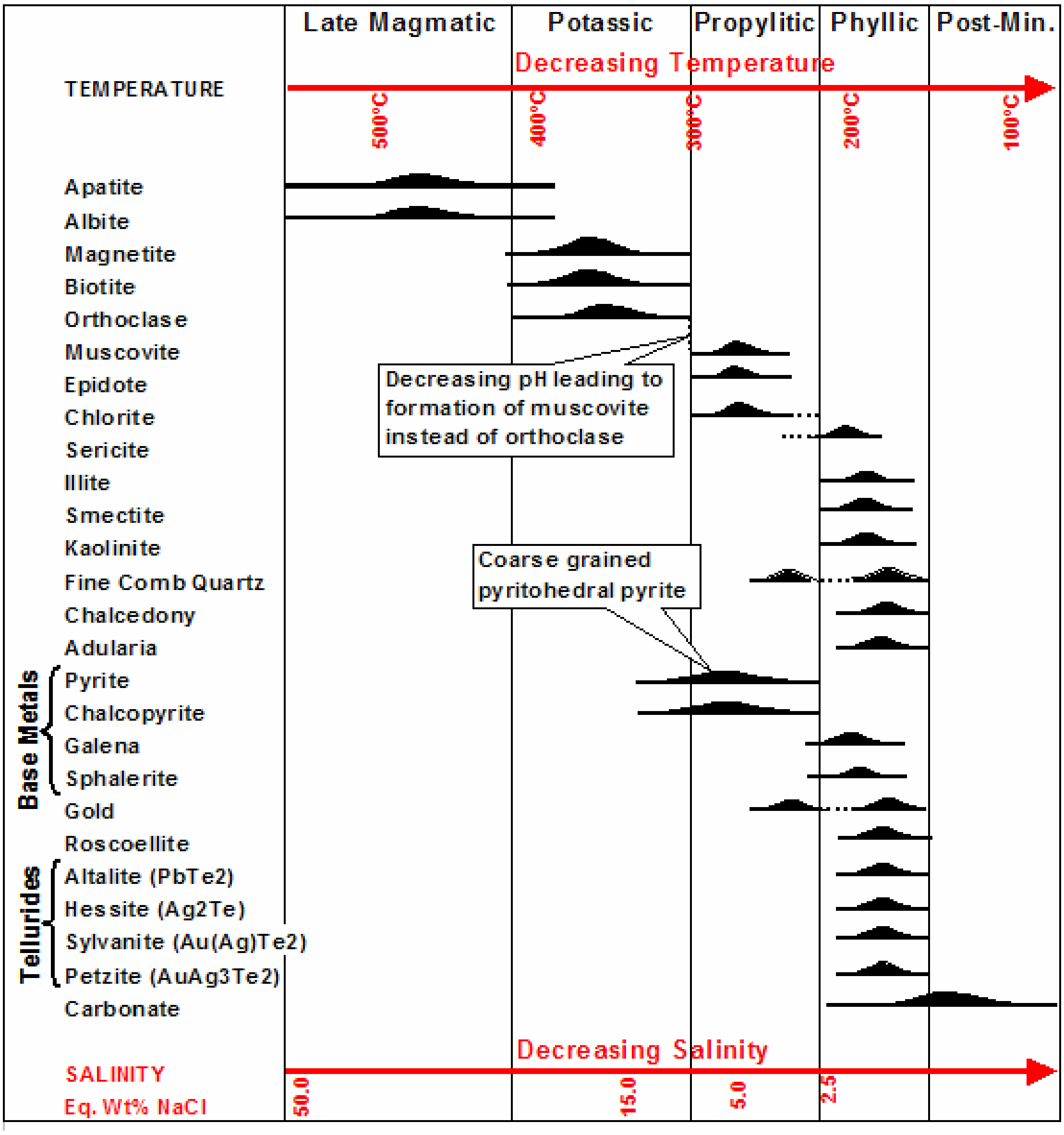
Mineralization associated with the porphyry copper system is characterised by apatite-k feldspar-magnetite-biotite veins with intense potassic alteration selvages. These veins are considered to have developed as the monzonite intrusive was in the final stages of crystallisation and early stages of cooling. As the system cooled it was overprinted by a phase of phyllic alteration which was characterised by a quartz-muscovite-pyrite assemblage. The system was then overprinted by a set of quartz-adularia veins accompanied by lesser amounts of calcite, chalcopyrite, pyrite, galena, tellurides and native gold. These veins generally have narrow chlorite-smectite selvages and commonly exhibit banded textures. A summary of the mineral paragenesis at Tuvatu is shown in Figure 7.7.

Minor roscoelite (vanadium K-mica) has also been observed in association with the quartz-adularia veins. Roscoelite is commonly observed at Vatukoula and many major deposits around the world (eg Porgera, Hishikari) and invariably has a close association with gold mineralization. The precipitation of roscoelite generally requires the reduction of a vanadium-bearing mineralising fluid. Reduction of the mineralising fluid may also lead to the precipitation of gold, tellurides and pyrite. Also rare occurrences of fluorite have been observed associated with the veins. The presence of fluorite further demonstrates the strong magmatic volatile content of the mineralising fluids.

The following is an overview of the mineralization, modified after A-Izzeddin (2000).

- Hosted in structurally controlled sets of narrow quartz veins (generally less than 0.5 m) which may form mineralized lodes up to 5 m wide.
- Early porphyry-related mineralization overprinted by late epithermal episode.
- Bleaching and alteration halo of sericite and clay minerals becomes more pronounced with weathering.
- Gold is free-milling and generally associated with silica/quartz, adularia and minor base metals (galena and sphalerite) and tellurides.
- High grades may be encountered in lodes, e.g. 0.5 m at 1,620 g/t and 0.3 m at 1,130 g/t Au.

Figure 7.7 Paragenesis of Mineralization, (A-Izzedin 2000)



A-Izzeddin (1997) suggested that there is a spatial and temporal relationship between the emplacement of the intrusive complex and the mineralization. The Tuvatu area appears to have had one to two km of overburden removed since emplacement of the intrusive complex, which may represent the magma source for overlying volcanism. The gold mineralization is interpreted to have been derived from deep-seated hydrothermal fluids emplaced in the very upper portions of the magma complex during the waning phases of volcanism.

7.4 Discussion

MA concludes the geological model is quite robust. Tuvatu is a low sulphidation epithermal deposit associated with the intrusion and subsequent cooling of a local monzonite. Stress regimes within epithermal/intrusive systems can be quite complex. The resulting veins and stockwork zones will pinch and swell along various strike orientations. This style of emplacement will always result in a risk to the tonnes and grade of any model developed. The mineralization is typical of epithermal deposits in being confined to narrow structures with little wall rock alteration which are hence “blind” outside of the mineralization. The grades decrease rapidly from very rich to barely detectable.

8.0 DEPOSIT TYPES

Scherbarth and Spry (2006) compare Tuvatu with Vatukoula. The Emperor deposit occurs along the margins of the Tavua volcano whereas the Tuvatu deposit may occur adjacent to an eroded shoshonite volcano. Both deposits are described as low-sulfidation, epithermal gold telluride mineralization occuring in flat-lying veins, steep faults, shatter zones, stockworks, and hydrothermal breccias. Mineralization formed in multiple stages and is characterized by the presence of quartz-roscoelite telluride veins in which gold-rich tellurides were deposited prior to silver-rich tellurides. Gold tellurides and vanadium minerals were deposited at approximately 250°C from moderately saline fluids.

The emplacement of epithermal deposits is characterised by late-stage, multiphase tectonic activity which creates a plumbing system and volcanic activity which provides a heat source. The general nature of these systems was first summarised by Buchanan (1981). The deposits were then divided on the basis of their alteration and mineralogy into two main types (Berger and Bethke 1985; Heald et al 1987) of acid-sulphate and adularia-sericite with a third minor but economically significant grouping of alkalic recently added (Bonham 1988; Richards and Kerrich, 1993). These types have now been included in a larger grouping low to high sulphidation systems and the links to gold and copper porphyries recognised.

9.0 EXPLORATION

Lion One has undertaken exploration activities in the Tuvatu project in two main phases: surface work and limited exploration drilling from 2008 to 2010 and more extensive drilling in 2011-2013.

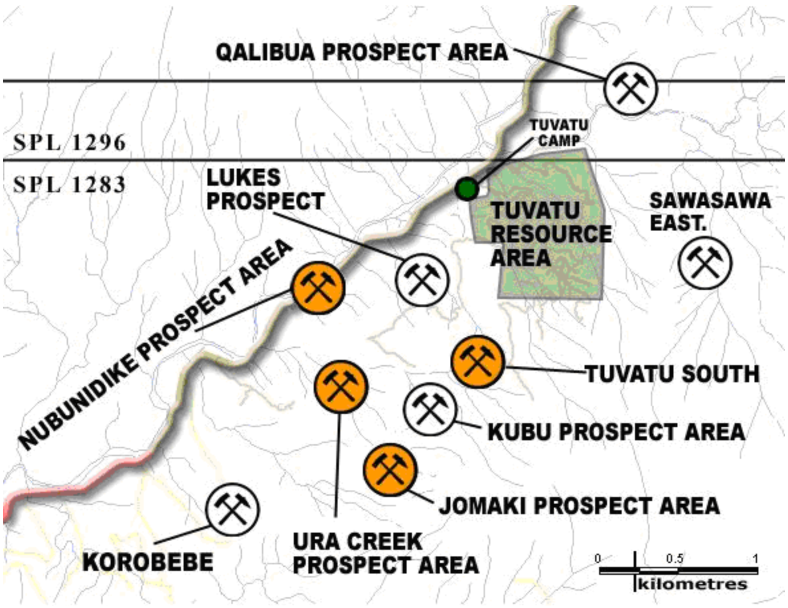
9.1 2008-2010 Lion One Exploration

During 2008, Lion One completed extensive mapping and geochemical sampling. Two surface drill holes were also completed. Field work was carried out by Lion One staff, W. Kuruisaravi, R. Sulua, and S. Bulu under the direction of various expatriate consulting geologists. The mapping, rock chip and channel sampling program involved the hiring of a trained team of permanent workers from Korobebe Village. Security staff at the Tuvatu Camp and core shed facility were hired from Korobebe, Nagado and Natawa Villages.

A number of highly prospective zones of mineralization that were identified in 2002-2003 by TGM were followed up (Figure 9.1).

- Nubunidike Prospect
- Ura Creek Prospect
- Jomaki Prospect
- Tuvatu South Prospect
- Qalibua Prospect

Figure 9.1 2008 Prospect Locations



Detailed geological mapping, rock chip and channel sampling in the region south of the Tuvatu Resource Area and Qalibua Creek was carried out with about 11.5 line-km of creek mapping completed. Detailed 1:1000 scale geological mapping and sampling covered the area from Veto Creek to the boundary of SPL 1296 just north of the Tuvatu Resource Area. Lion One submitted 1,309 rock chip and channel samples between November 2008 and May 2010 to ALS Chemex laboratories in Brisbane. MA has not seen any reports on this mapping or sampling detailing geology, vein widths and assay results.

Two surface diamond drill holes (TUDDH-338 & TUDDH-340) totalling 375.90 m were drilled during October 2008 at the Nubundike Prospect, 1.6 km southwest of the Tuvatu Resource Area. Drilling was planned to intersect the Nubundike/Hornet Creek/290 Vein system about 50 m below the surface over a strike length of 500 m and gain information on the dip and strike continuity of the vein system, as well as grade distribution within the structures.

9.2 2011 Lion One Exploration

Following a comprehensive review of historic data that began in August 2010, Cambria Geosciences ("Cambria") was contracted to assist in managing the exploration program at Tuvatu. In January 2011 Cambria mobilized a field team to the site to initiate a program of surface mapping, trenching and core re-logging and re-sampling of approximately 10,000 m of the total 60,000 m of core. In addition to the ongoing program of mapping, core re-logging and re-sampling, trenching and diamond drilling, this first phase exploration program was planned to include reconnaissance mapping, prospecting, stream sediment sampling, geophysical surveying, deposit modelling and dewatering of the decline.

Lion One reported that the review, along with ongoing mapping and prospecting conducted by Lion One geologists, resulted in the discovery of several near surface drill targets that became the focus of the trenching and surface mapping programs.

In excess of 1,200 m of trenching was completed to assess the near surface, open pit potential of the Tuvatu North area where drilling by previous operators had yielded several near surface high gold intervals in the northern portion of the Tuvatu Resource area. Principal objectives were to expose and confirm the presence of gold bearing veins and veinlets in the structures related to the Tuvatu Lode, H Lode and the Core Shed Fault (CSF).

Initial sampling was between the Core Shed Fault (CSF) and the Tuvatu and H Lodes from four benches and two trenches excavated adjacent to and directly south of the portal of the existing decline. Excavations were completed across the CSF, with subsequent trenching above the surface expression of the Tuvatu (1 and 2) and H Lodes. Trenches were up to 2 m deep with an average depth of 1.5 m. Several benches along road cuts were also sampled as a part of the program. Most samples were continuous or semi continuous chip samples with composite samples taken when necessary.

A core re-logging and re-sampling program was commenced with the objective of identifying mineralized intervals that were ignored by previous operators. As 3.0 g/t Au was the historical cut-off grade, Lion One geologists believed that the economic significance of many altered and mineralized zones within the hanging and foot walls were previously overlooked.

Lion One also completed 58 km of Induced Polarization ("IP") survey and prepared additional lines to obtain further readings over areas with prospective chargeability and resistivity anomalies, including five additional lines covering the First Porphyry Development Area. The survey was initially planned to cover known mineral occurrences before extension to outlying areas. Lion One also completed 36 line km of soil sampling across the IP survey grid area.

9.2.1 Results

Extensions of the Murau and Far West Lodes were mapped at surface over 500 m to the west displaying consistent lateral continuity typical of many epithermal lode systems. Multiple sub-parallel near-surface, high-grade veins were encountered.

The surface sampling program was reported by Lion One as confirming the presence of gold bearing veins and veinlets within the CSF and the Tuvatu and H Lodes. Five samples were reported to return grades over 100 g/t Au, including 210 g/t Au across 0.05 m, 188 g/t Au across 0.87 m and 188 g/t Au across 0.70 m. Significant intervals included 8.7 g/t Au over 4.8 m from the surface expression of the north-west striking Tuvatu Lode.

10.0 DRILLING

Drilling campaigns were completed in several phases by TGM from 1995-2001 and by Lion One between 2008 and 2013. Completed drilling is summarised in Table 10.1.

Table 10.1 Summary of Tuvatu Exploratory Drilling

Company	Surface RC Drilling		Surface Diamond Drilling		Underground Diamond Drilling	
TGM Phase 1	5,225 m (44 holes)	TURC101 to 171	42,783 m (193 holes)	TUDDH013 to 205	1,108 m (17 holes)	TUG01 to 17
TGM Phase 2					1,374 m (26 holes)	TUG18 to 43
TGM Phase 3	4,040 m (37 holes)	TURC172 to 208	8,702 m (24 holes)	TUDDH206 to 229	10,926 m (69 holes)	TUG45 to 113
TGM 1995-2000 Total	9,265 m (81 holes)		51,484 m (217 holes)		13,408 m (112 holes)	
Lion One 2008			376 m (2 holes)	TUDDH338 & 340		
Lion One 2012-2013			13,842m (65 holes)	TUDDH341 to 405		

10.1 TGM 1995-2001

TGM completed three phases of drilling at Tuvatu from exploration through to resource delineation. Drilling was carried out both on the surface and from the 600 m underground exploration decline which was developed to a depth of 240 m below surface. Drilling methods included both diamond drill core and reverse circulation (RC). Overall, TGM completed 51,484 m of diamond core and 9,265 m of RC surface drilling, as well as 13,407 m of underground drilling.

Up to six drilling rigs operated in the Tuvatu resource area during Phase 1. During this period 193 diamond holes (TUDDH-013 to 205) and 44 RC holes (TURC-101 to 171) were completed. A total of 42,783 m of diamond core (HQ and PQ diameter) and 5,225 m of 5¼” RC drilling were completed in the area. This programme delineated an area of mineralization that extends over a distance of 800 m. In conjunction with the underground development, 17 underground diamond drill holes (TUG01 to 17) were completed for a total of 1,108 m of HQ diameter core. The purpose of these holes was to infill surface drilling and to assist in planning future development.

During the 2nd phase of work by Emperor, 26 underground diamond drill holes (TUG-18 to 43) were completed for a total of 1,374 m of HQ diameter core. The purpose of these holes was to infill surface drilling and to assist in planning development.

During Phase 3 a reverse circulation drilling programme was initiated to test various anomalies in the local area as well as the near-surface potential of the Upper Ridges area. Thirty-seven holes (TURC172 – 208) were completed for a total of 4,040 m. Drill holes TURC174 and TURC179 encountered significant mineralization associated with a previously untested lode structure located approximately 500 m west of the resource area. Follow-up drilling and trenching demonstrated that mineralization was associated with two sets of veins trending E-W and NW-SE. The lodes may be up to 5 m wide. A series of 69 underground diamond drill holes (TUG045 – 113) were completed for a total of 10,926 m. These holes were drilled to infill and expand the Upper Ridges resource and test peripheral mineralized zones in the Murau area. This programme successfully extended the Upper Ridges lodes (particularly UR2) and upgraded the Phase 2 resource.

A series of surface diamond holes were also drilled to target various deeper drill intersections encountered in Phase 1 as well as the newly identified zone of mineralization located 500 m west of the current resource area. Twenty-four holes (TUDDH206–229) were completed for a total of 8,702 m.

10.2 Lion One 2008

Two surface diamond drill holes (TUDDH-338 and TUDDH-340) totalling 375.90 m were drilled during October 2008 to test the Nubunidike / Hornet Creek / 290 Vein system over a strike length of 500 m at the Nubunidike Prospect, 1.6 km southwest of the Tuvatu Resource Area. Drilling was planned to intersect the veins about 50 m below the surface and gain information on the dip and strike continuity of the vein system, as well as grade distribution within the structures.

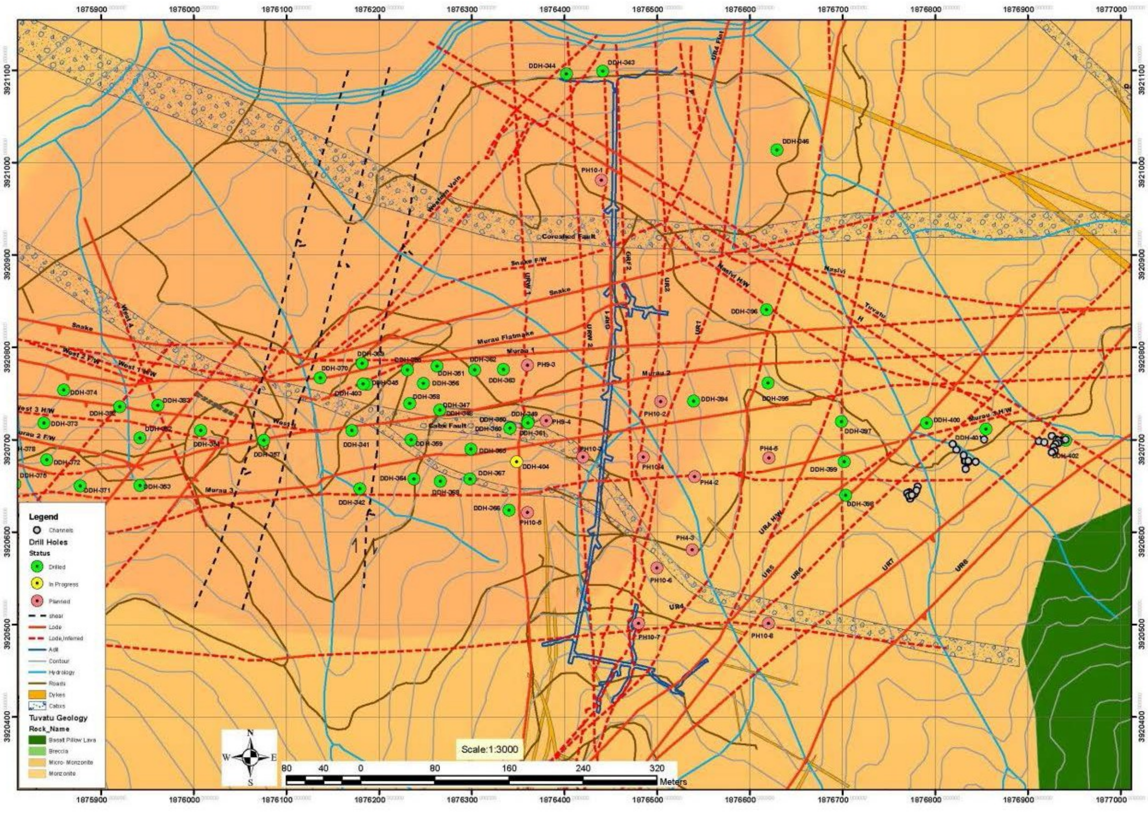
10.3 Lion One 2012-2013

In response to the results from the trenching program Lion One changed its focus to broad zones of low grade mineralization potentially exploitable by surface mining methods. Lion One commenced a systematic program to delineate the extent of near surface gold mineralization. The Lion One exploration team planned and executed the drilling program under the supervision of Lion One's Tuvatu Project Manager at the time, Mr. David Pals.

Drilling re-commenced in June 2012 with a combination of infill and step out holes. The program had three objectives: (i) infill drilling to increase the confidence level of the existing resource; (ii) step out drilling to expand the resource base; and (iii) exploratory drilling to test additional targets.

Infill drill holes were planned to test areas of the intersections of the east-west trending Murau-Far West Lodes with the N-S trending Upper Ridge Lodes west of the north-south trending UR structural corridor and current resource. Interpreted lode locations are shown in Figure 10.1.

Figure 10.1 Tuvatu 2012-2013 Drill Collars and Lode locations (Lion One,2013)



Step out holes tested for mineralized extensions of the Tuvatu and H Lodes in the northern portion of the Tuvatu resource area, where surface mapping has identified continuous mineralization along a strike length of 300 m.

10.4 Drilling Procedures

10.4.1 TGM 1995-2001

Up to six drilling rigs operated in the Tuvatu resource area during Phases 1, 2, and 3 but MA has no details on the type of surface rigs used.

During Phase 1 and Phase 2 the underground diamond drilling was carried out using a Longyear LM-75 electric hydraulic drilling rig from the Emperor Gold Mine in Vatukoula. Underground drilling in Phase 3 was carried out using a Longyear LM-75 electric hydraulic drilling rig and a Kempe rig.

Although Mr Vigar of MA originally worked on the deposit from 1999 to 2000, which included logging core, no detailed sampling or QAQC report is available on the sampling done by TGM.

However the following points are noted:

- Adjacent host rock material may be barren and forms internal waste within the lode structure. Where this internal waste was not assayed it was assumed to carry no grade.
- Individual veins within the lode structure were often sampled using half core samples.
- Selected drill core sections were halved with a core saw and samples were dispatched to the Emperor Gold Mining Company laboratory at Vatukoula.
- Waste intervals were not assayed.
- TGM used the assay laboratory at the Vatukoula mine operated by the Emperor Gold Mining Company. Monthly re-assays and checks on standards, mill products, mine and exploration samples are conducted with external commercial laboratories as part of the standard operating procedure at Vatukoula.
- The whole sample was pulverised in a 5 kilogram ring mill prior to splitting. A 50 g sub-sample was analysed for gold by fire assay with an AAS finish.
- All samples above 1 g/t Au were re-assayed.
- Samples within the interpreted lode structure were composited to obtain an overall grade for the lode.

All drill collars were picked up by TGM surveyors on a regular basis using a Leica TPS 300 theodolite. Data was downloaded in digital form and entered into the database. Where possible the collar azimuth and dip was also calculated by the surveyor to compare with the planned orientation and downhole survey data. The majority of diamond drill holes were also surveyed by downhole camera at 50 m intervals using an Eastman downhole survey camera. Percussion drill holes generally were not surveyed down hole due to the difficulties in surveying inside RC drill rods.

10.4.2 Lion One 2008 and 2012-2013

Drilling by Lion One was diamond core drilling from surface and the following procedures were used:

- Drill core was digitally photographed and placed onto the database.
- Core was logged manually onto log sheets and all data entered into the database.
- Information included hole number, date drilled, name of driller/company, location, coordinates, core recovery, lithology, structure, RQD values, alteration, gangue minerals, sulphide minerals, mineralization, sample numbers, intervals samples, analytical values, comments, date logged and by whom. Specific gravity of selected intervals and lithologies were measured.

- A summary log was prepared after the hole was logged.

Drill core was cut in half with a core saw for sampling and half-core samples were dispatched to the ALS sample preparation facility in Suva, Fiji. Samples were first crushed and pulverised at Suva, Fiji prior to analysis at ALS Minerals, an independent and qualified analytical laboratory in Brisbane, Australia. Gold is determined by fire assay and silver by Aqua regia digestion and AAS.

Consistent with industry standard practice, sample standards and blanks and other control methods are used to ensure quality control.

10.5 Results

10.5.1 TGM 1995-2001

A table of historical drill intervals listed by lode is contained in Appendix 1 of Vigar 2009.

10.5.2 Lion One 2008

Drill holes passed through bad ground with shear zones showing slickensided contacts. Faults are almost parallel to the core axis. The host rock is coarse grained to medium grained Nadele Breccia.

Only visually identifiable mineralized intervals were assayed for gold. A total of 59 samples ranging in length from 0.23 m to 1.0 m were collected from the 376 m of drilling. Best intersection was
0.25 m containing 1.06 g/t gold in hole TUDDH-338. Hole TUDDH-340 returned insignificant results.

Following a field inspection in 2009 Andrew Vigar of MA suggested that the two drill holes may have missed their intended target as the holes were not orientated properly relative to the vein being tested.

10.5.3 Lion One 2012-2013

Initial results confirmed high grades. Figure 10.2 shows the collar locations of drill holes with significant intersections at Tuvatu West. Figure 10.3 is a north-south representative section along 1876260E in the Tuvatu West locality.

Drill intercepts are listed in Appendix 1.

Figure 10.2 Tuvatu West – 2012-2013 Lion One Drill Significant Intersections

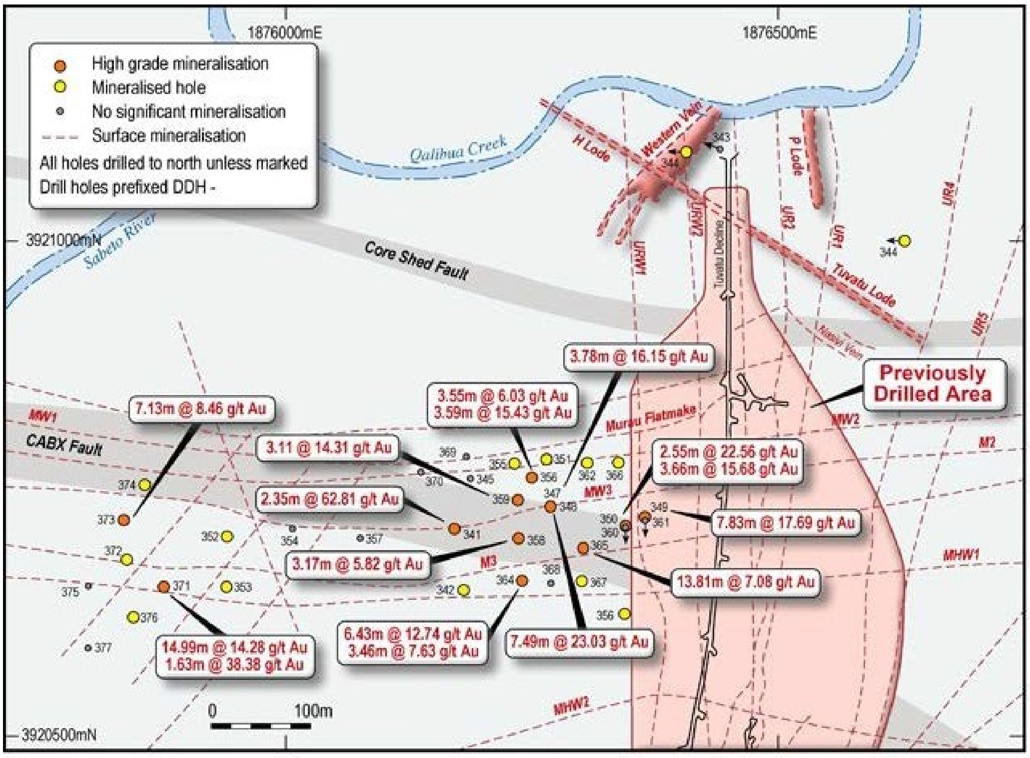
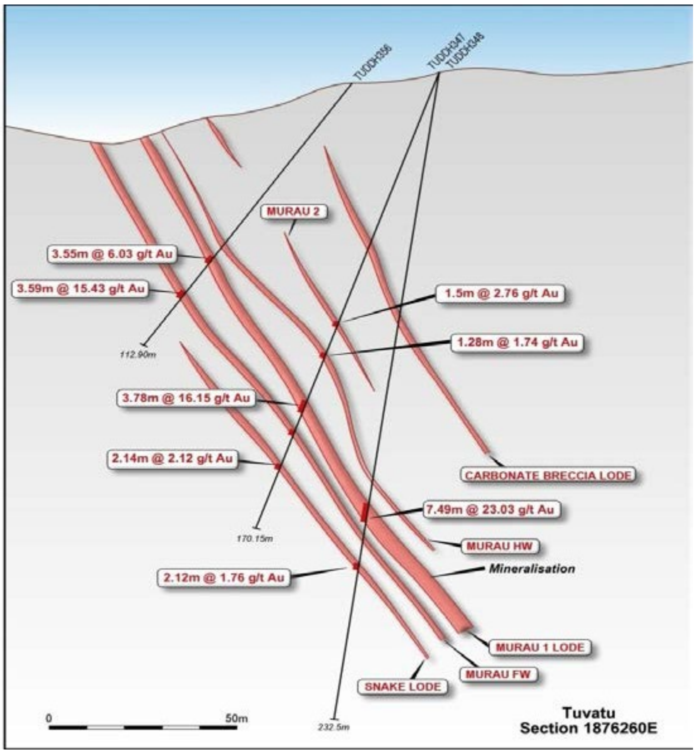


Figure 10.3 Murau Lodes Representative Cross Section (1876260E)

(Source: Lion One, 2013)



10.6 Discussion

There has been no independent review of the drill hole sampling, geological logging and geological interpretations done by TGM or Lion One. Although it is expected that this work was done to an industry acceptable standard there is always a risk involved with structural interpretations, grade and geological continuity. However, MA believes that the information revealed in the exploration decline mitigates the impact of this risk to a large extent.

10.6.1 Recovery and Quality

Core recovery is overall very high, although within sheared and broken intervals it is somewhat less so. Unfortunately these intervals may coincide with mineralized zones.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

During pre-2000 drilling by TGM all samples were dispatched to the Emperor Mine laboratory at Vatukoula for preparation and analysis. The whole sample was pulverised in a 5 kg ring mill prior to splitting.

In the Lion One 2008 and 2012-2013 programs diamond drill core was logged and sampled on site at Tuvatu by Lion One staff. Core samples were delivered by the Company to Suva, Fiji sample preparation facility of ALS Minerals (ALS), a division of Australian Laboratory Services Pty. Ltd., an independent accredited analytical laboratory.

The samples were finely crushed (>75% passing through -2 mm) and a 1 kg split then pulverized (>85% passing through -75 microns) prior to dispatch to ALS Minerals Brisbane, an independent accredited analytical laboratory in Brisbane, Australia, for analysis.

11.2 Sample Security

No particular security measures were used during the life of the Tuvatu project as visible free gold is rare and off-site laboratories have been used throughout.

Half-core splits of most drill core were retained on-site. This core is well catalogued and is available for inspection.

11.3 Sample Analyses

11.3.1 Laboratory Analysis Procedures

All pre-2000 assaying by Emperor for TGM used a 50 g sub-sample which was analysed via fire assay with an atomic absorption spectrometry (“AAS”) finish at the mine laboratory at Vatukoula. All samples above 1 g/t Au were re-assayed.

All analysis in the exploration programs by Lion One in 2008, and 2012-2013 was carried out by ALS Minerals at their laboratories, in Brisbane, Australia. Gold was analysed by fire assay with a 30 gram charge and AAS finish. Samples with higher grade gold (greater than 3 g/t Au) were re-assayed. Silver was analysed by Aqua regia digestion and AAS.

Exploration samples were analysed for 33 elements using a four acid digestion and Inductively Coupled Plasma Atomic Emission Spectrometry (ICP-AES).

11.3.2 Laboratory Independence and Certification

The laboratory at the Vatukoula gold mine used by TGM was a private laboratory operated by Emperor Gold Mining Company.

The ALS Minerals laboratories used by Lion One are part of the worldwide ALS Limited group of companies.

11.4 Quality Control

11.4.1 QC Program

There are no detailed sampling QA/QC reports available on the sampling carried out by TGM for the pre-2000 drilling. According to Vigar (2009), monthly re-assays and checks on standards, mill products, mine and exploration samples were conducted with external commercial laboratories as part of the Emperor standard operating procedure. Laboratory certificates for these assays and checks were not provided to MA. There was no evidence of the implementation of a QA/QC program utilizing field duplicates, blanks and standards.

The laboratory at Vatukoula is a private laboratory, and it is considered unlikely that they conducted an internal QC program for the samples submitted. However, the Vatukoula mine has relied on the results of its laboratory in order to run its operations since the 1930s and it can be reasonably assumed that the laboratory provides accurate assaying work.

No information was provided to MA regarding the QA/QC program for the 2008-drilling by Lion One.

The assay analyses performed during Lion One's 2012-2013 drilling programs was subject to a formal quality assurance and quality control (QA/QC) program that was under the supervision of Lion One's Tuvatu Project Manager at the time, Mr David Pals.

Certified reference materials (“CRM”), blanks, and field duplicates samples were inserted prior to shipment from site to monitor the quality control of the data. MA understands that 3 CRM samples were inserted every 100 samples and 2 field duplicates were inserted in every batch of 100 samples. MA received and reviewed QAQC summary reports (for CRM’s, field duplicates, and assay laboratory duplicates) from rOREdata Pty Ltd. database consultants for Lion One (Appendix 2).

11.4.2 QC Program Results

Standards Results - Accuracy

Accuracy is identifying the true grade of a sample, often achieved by submitting certified reference material (“CRM”) commonly referred to as standards (“STD”).

Ten different gold CRM standards supplied by Rocklabs Ltd. of New Zealand were used by Lion One for quality control in core sampling. Seven of the standards were submitted more than 5 times Table 11.1. A total of 216 CRM gold standards and 26 silver standards were submitted during the Lion One drilling program.

Table 11.1 Summary of CRM Used Lion One 2012-2013

CRM Code	Certified Value	Standard Deviation	Total Submitted	No Outside Limit
SL61	5.931	0.5418	44	9
SQ47	39.88	1.1774	26	6
OxJ80	2.331	0.5822	41	12
OxJ87	0.417	0.9296	31	5
B2	0.414	0.0646	27	3
B1	5.931	0.1189	15	1
A1	2.337	0.0590	25	1
SQ47 (Ag)	122.3	5.4751	26	2

Field Blanks - Contamination

Field Blanks are obtained from within the vicinity of the project area by selecting an un-mineralized outcrop of similar mineralogy and weathering as the samples being submitted. A representative number of blank material samples are submitted for analysis to provide reference concentrations of elements of interest. Ideally, Field Blank samples will look similar to the original material being submitted.

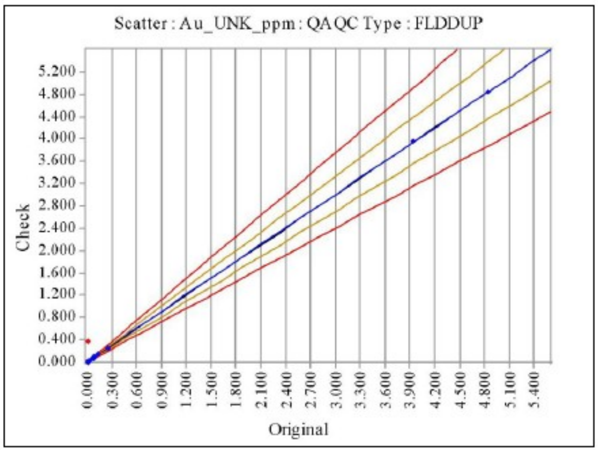
MA has not seen any results for field blank samples submitted to ALS Minerals with the Tuvatu drillcore samples.

Field Duplicates – Precision and Bias

Field duplicate procedures for diamond drill samples from Tuvatu have not been described in detail by Lion One.

Thirty-five (35) samples described as field duplicates were submitted by Lion One in the period 2012 to 2013. Field duplicate results are shown in Figure 11.1. Only one duplicate sample showed significant distance from the original value. One of the two check sample from TUDDH-384 assayed 0.64 g/t Au compared to the original sample which returned an assay of 0.16 g/t Au.

Figure 11.1 Field Duplicate Results for Au



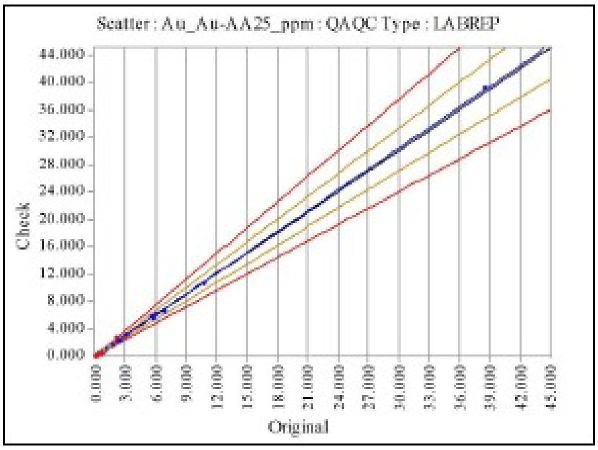
Laboratory QA/QC

Lion One instructed ALS Minerals to split secondary duplicates after the core had been crushed but prior to pulverizing. ALS assayed 173 of these secondary or laboratory duplicates (Figure 11.2). ALS Minerals conducts its own internal QA/QC consisting of CRM testing, duplicate assaying and repeats along with the primary sample analysis. MA has not seen these results.

Inter-laboratory Checks

No duplicate samples were sent by Lion One to a referee laboratory for analysis.

Figure 11.2 Laboratory Duplicate Results for Au



11.5 Adequacy Opinion

There has been no independent review of the drill hole sampling, geological logging and geological interpretations done by Lion One. Although it appears that this work was done to an industry acceptable standard, there is always a risk involved with geological interpretations and grade continuity.

Generally, the results of the QA/QC program implemented by Lion One are considered satisfactory for resource definition. It is MA's opinion that the sample preparation, security and analytical procedures were adequate and follow accepted industry standards for a mid-stage exploration property.

12.0 DATA VERIFICATION

12.1 Data Verification Procedures

The data verification involved database integrity checking, site visit, and independent sample collection.

12.1.1 Drill Hole Database

Lion One provided MA with a large amount of data relating to the Project. Lion One's current drill hole database, historic block models and geological wireframes were used, as were reports on resource estimation. MA also accessed archived data used for resource estimation in 2000 and 2009.

12.2 Drill Hole Database Review

MA was provided with an export of Lion One's current drill hole database in MS Access format, named Database ExportDrillHoles.mdb. The database contained tables for Collar details, Collar metadata, downhole surveys, assays, weathering, lithology, alteration, geotech, SG data and lode tags.

12.2.1 Database validation

MS Access queries were used to perform basic validation checks, and holes were then loaded into Surpac for a second round of validation. Table 12.1 summarises the basic validation checks performed and the results.

Table 12.1 Summary of Database Validation

Check	Results	Comment
Missing / out of range coordinate data	Six holes with missing z coordinates	All old DDH holes (1970s). Need to confirm locations and assign z coordinates from topography DTM.
Missing downhole surveys	Four holes with no downhole surveys	Lion One checking, one hole corrected. Holes without confirmed orientation must be excluded from resource estimates.
Sample overlaps / to < from depth / no depths	A few assay intervals with null from-to depths.	QAQC results mistakenly included in assay table.
Downhole survey validation check, Access query	71 drill holes with survey deviation >5° in azimuth or dip in adjacent surveys. Note that all RC holes have only collar surveys.	Data sent to Lion One for correction / checking.
Data not drill holes	Two trenches and one channel included in drill hole data	Removed from drill hole database.

12.2.2 Assessment of RC Drill Holes

Sample assay data from diamond (surface plus underground, all dates) and RC drilling were compared statistically by the following method:

1) Raw sample data composited downhole to 1 m intervals to create comparable samples of identical volume (reduce effects of sample volume variance)

2) 1 m composites were restricted to cover the same area (roughly corresponding to the 2000 resource model extents) as a crude method of compensating for possible spatial clustering of data. The following spatial filtering methods were used:

   a) Boundary drawn in plan view to cover extent of most RC drilling;

   b) DD data restricted in z extent to cover the same depth range as RC data;

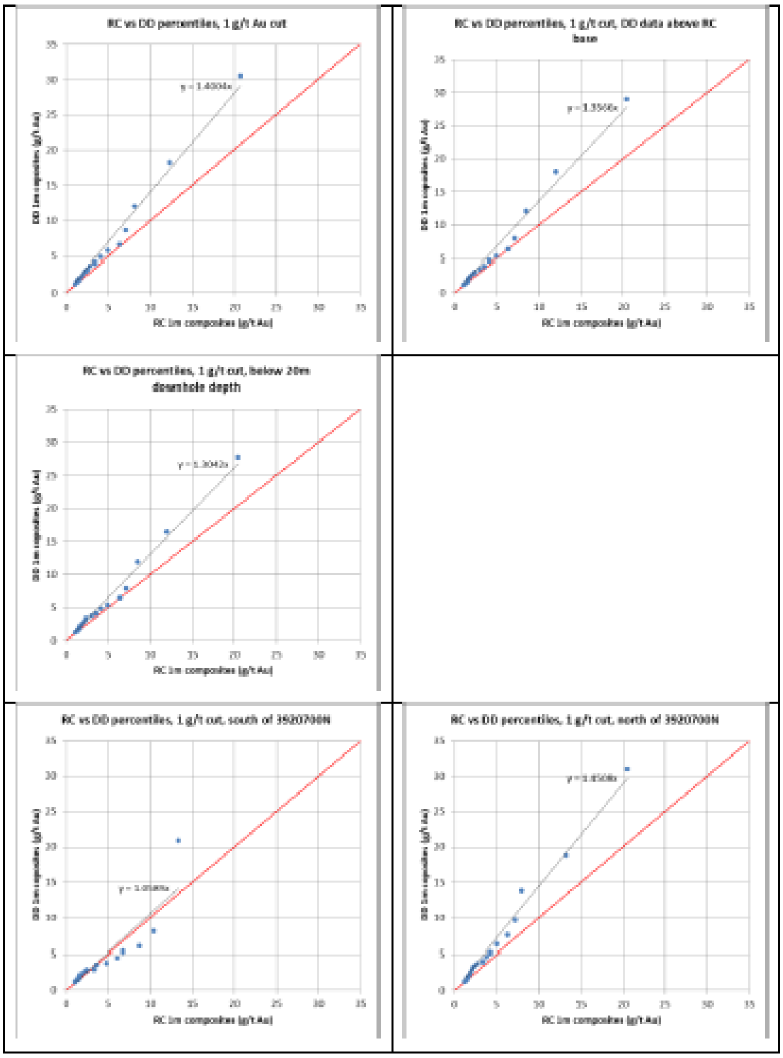
   c) DD and RC data restricted to depths > 20 m below surface (to reduce effects of shallow RC drilling near-surface);

   d) Data plotted north and south of 3920700N, which marks the approximate limit of clustered high grade DD intercepts.

3) Q-Q (percentile) plots generated for RC versus DD data above a cut-off of 1 g/t Au for each of the spatial filters described above. Cut-off was used to compensate for the effects of mostly selective sampling of DD holes.

The resulting charts are shown in Figure 12.1. In all cases the red line indicates a 1:1 correlation.

Figure 12.1 Q-Q Percentile Plots RC vs DD holes



In general, grade distributions match reasonably well up to the 50th percentile (about 2.5 g/t Au), with DD samples reporting slightly higher grades than RC. After the 50th percentile there is more positive bias towards DD samples, and after the 75th percentile the bias is more pronounced. There is a slight improvement in the correlation of percentiles across the first three graphs, corresponding to limiting the extent of DD data used. The data is difficult to assess in too much detail because the RC and DD samples are not exactly spatially equivalent. The last two graphs illustrate that spatial bias accounts for a significant part of the difference seem between RC and DD grade distributions, with the southern portion of the data better correlated (with a positive bias to RC data) and the northern part of the data showing positive bias towards DD data.

Spatially equivalent RC and DD samples were then selected via an approach that used a nearest neighbor method with a small search ellipse to assign grade values to a fine scale block model. Values were exported only for those blocks that contained values for RC and DD data and the results examined. This approach yielded a data set that was too small to draw any conclusions.

Limited conclusions can be drawn from the existing data. Spatial clustering appears to be a more important contributor to bias than drilling method. Other factors that may be important are:

- Sample recovery – it is not known how sample recovery compares between RC and DD samples. Some instances were identified where RC mineralized intersections coincided with not sampled DD intervals, presumably due to core loss.

- RC drilling sub-sampling – the method and possible introduction of bias during sub-sampling is not known.

- RC drilling QA/QC measures – in particular the efforts made to ensure that no sample contamination occurred during drilling and later sample processing.

- DD drilling sub-sampling – possible introduction of bias during core cutting, especially if core was not cut at a consistent orientation relative to veining.

- Directional bias – there are some examples of bias occurring where drill holes sample a vein at a low angle versus drill holes sampling veins in a perpendicular orientation.

From the available data, MA concludes that there appears to be no major problem with utilising RC samples as part of a resource estimate. However, the following should be taken into consideration:

- RC samples are inherently more likely to have lower grade variability and show less effects of high grade outliers due to the larger volume of sample taken compared with DD core.

- Due to the fixed 1 m sampling interval for RC, there will be a tendency for narrow, high-grade vein intersections to be over-estimated for thickness and under-estimated for grade (i.e. wall rock dilution will be included in the sample) compared with DD.

- For lower-grade veins the opposite problem will apply, with thickness under-estimated and grade possibly slightly over-estimated.

- Vein thickness in RC intersections can only be practically resolved to the nearest m using grades.

- The only way to compare DD and RC drill samples and assess potential risks to resource estimation is to undertake a small program of drill hole twinning.

12.3 Site Visit

The Tuvatu deposit was visited by Mr. Ian Taylor, BSc (Hons) MAusIMM (CP) during the period 25th to 28th February 2014.

In the course of the site visit, Mr Taylor viewed mineralized drill core and examined the drill core processing and storage facilities. He also viewed and sampled the mineralized vein systems and outcrops.

12.3.1 Independent Samples

For this report, Mr Taylor collected two independent samples, one from outcrop and one from core.

The selected samples were selected by Mr Taylor, at no time prior to the sampling were any employees or other associates of Lion One were advised of sample locations or identification of any of the samples to be collected. Sample remained in the custody of Mr Taylor, the samples were documented, bagged and sealed with packing tape and were shipped by DHL couriers to ALS Chemex Laboratories in Suva Fiji for sample preparation. The prepared samples were sent to ALS in Brisbane for analysis for gold by 30 g fire assay (ALS method Au-AA25). Table 12.2 lists the samples and description the gold assay results.

Table 12.2 MA Independent Sample Descriptions

Sample ID	Sample Description	g/t Au
MA_TV_01	Rock chip from Tuvatu Lode outcrop	13.65
MA_TV_02	Drillcore from TUDDH112; 282.4m to 282.8m	9.62

The assay results are consistent with the gold mineralization typical of the prospect. A previous sample (TU13421) assayed 12.5 g/t Au from the same section of drill core as MA_TV_02 although MA highlights that the sample lengths were different.

Figure 12.2 MA sampling Tuvatu Lode Outcrop

12.3.2 Limitations

Mr Taylor only visited selected surface drill collars and surface outcrops. Access to the underground workings was not possible.

12.4 Verification Opinion

Based on the data verification performed, it is MA's opinion that the data reviewed is adequate for the purposes used in this technical report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogy

The Tuvatu mineralization is structurally controlled, occurring in a number of discrete steeply dipping lodes within the monzonite host rock. Gold typically occurs throughout the variable lode mineralogy in veinlets containing quartz, pyrite and metal sulphides.

In October of 2013, APSAR (Applied Petrologic Services & Research) undertook a petrology study of samples distributed across the Tuvatu deposit. The report indicates that native gold, as well as gold-silver telluride comprise precious metal mineralogy within the banded quartz veins and stock-work zones. While some native gold, gold telluride and sulphosalt-bearing epithermal style veining locally crosscuts the porphyry copper-style mineralization observed to be present, most epithermal-style gold mineralization is spatially unrelated to porphyry-style mineralization.

Where structure may have allowed, the mixed magmatic and metamorphic fluids may have also interacted with meteoric fluids, with heating and circulation of the latter, resulting in the low-sulphidation epithermal systems with which precious metal mineralization is associated.

Central Mineralogical Services (CMG) in June 2000 also undertook a petrological study of samples from the Tuvatu gold project and observed that the gold deposited partly associated with enargite and argentite, partly as very small grains in gangue rimming and veining earlier formed sulphides. The gold associated with enargite, and with bismuth/galena and argentite, occurred as inclusions in these minerals, (1-200um in size) and as fine intergrowths with them. Other gold occurrences were observed to be variable, but with significant amounts of fine-grained (1-30um) gold occurring in silicate and carbonate gangue minerals. CMG also observed that the gold formed fine rims on and in veinlets, (grain sizes between 2-50um), and in earlier-formed sulphides such as pyrite, sphalerite, galena and chalcopyrite.

In 1997, AMMTEC (Report A5922) undertook a mineralogical examination, size-by-size analysis and diagnostic analysis of numerous gravity tails and leach residue samples of the composites from the Nasivi and Upper Ridges (UR) lodes.

Table 13.1: Size-By-Size Analysis

Master Composite Gravity Tails Identity	Calc. P80 (um)	Percent Gold Distribution at Size Range (%, um)					
		+90	-90 +75	-75 +63	-63 +45	-45 +38	-38
Nasivi Lode	90	22.6	3.1	3.5	4.6	2.2	64.0
Upper Ridges	96	16.3	4.6	4.8	6.9	3.4	64.1

Source: AMMTEC Report A5922, December 1997

The "size by size gold analysis data indicated a relatively uniform gold grade across the size ranges examined which may be due to the presence of fine gold". The data potentially indicates a bimodal gold distribution, with coarse gravity recoverable gold and a more finely disseminated fraction, being consistent with previous mineralogical and petrographic observations. AMMTEC reports that the optical mineralogical examination of the gravity tailings revealed that pyrite, magnetite and hematite were present as the major mineral phases in the Nasivi sample. The UR sample contained pyrite as the dominant mineral phase and magnetite as the major phase. The UR sample also contained free gold, gold as electrum and gold-silver telluride. Both samples contained minor traces of other sulphides.

AMMTEC also reports that the mineral phases of the leach residues were similar to those observed in the gravity tailings (leach feed), with no gold visible.

13.2 Testwork

The Tuvatu project has undergone numerous metallurgical studies assessing the conventional process methods, including gravity concentration, flotation followed by cyanidation, diagnostic leaching, and detoxification. Initial testwork has been undertaken in an attempt to define a number of process parameters and variability test work has been undertaken on a number of the different deposits (lodes).

The previous metallurgical testwork campaign reports are summarized as follows;

1. YANTAI JINPENG GROUP (YJG) - Fiji Gold Ore Mineral Processing Experiments - March 2015: Gravity, Flotation, Flotation Tails Cyanidation, Cyanidation, Process variable optimization (grind size, reagent selection and optimization).
2. GEKKO Testwork Report - July 2012: Gravity Recoverable Gold (GRG), Flotation, Vertical Shaft Impactor Amenability, Leach.
3. METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Gold Ore - July 2000: Grinding, Gravity, Cyanidation on 15 samples from UR1, UR2, UR West 1, UR West 2 and Murau.
4. METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Drill Core Samples – Variability Response Tests - May 2000: Variability testwork, Grinding, Gravity, Cyanidation, Flotation, Cyanidation of Flotation Concentrates.
5. AMDEL Ltd. - Comminution Tests - Tuvatu Ore - February 1998: AG media competency, Impact Crushing Work Index, SAG attrition, Abrasion, Bond Rod Mill, Bond Ball Mill, UCS.
6. EMPEROR MILL Batch Treatment Campaign - December 1997: Orway Mineral Consultants Pty. Ltd. (OMC) Grinding Survey, Cyanidation, Float/Cyanidation, Detox.
7. AMMTEC Ltd. - Metallurgical Testwork Conducted on Samples of Ore from the Tuvatu Gold Deposit - December 1997: UR and Nasivi Composites, Gravity, Mineralogy, Grinding, Flowsheet Determination, O2 Uptake, Rheology, Flotation, Variability, Diagnostic Work.
8. AMMTEC Ltd. - Comminution Testwork Conducted Upon Samples of Ore from the Tuvatu Gold Deposit - June 1997: UCS, Abrasion, Bond Rod Mill, Bond Ball Mill,

13.2.1 Recent Testwork

Recent test work, 2014-2015, has been performed by YJG in China and ALS Ltd. in Perth, Australia.

Yantai Jinpeng Group (YJG)

The YJG testing was undertaken in 2015 on a high-grade sample (29g/t Au, 14g/t Ag) composite of approximately 300kg. With little guidance, a non-representative sample, and no separate assays of the test charges, the test results are only indicative of the mineralized material response and were not included in any design parameters.

The gravity recovery was 9.8% at a 0.1% mass pull to a concentrate grade of approximately 3126g/t Au. YJG indicates that grinding finer, from 65% passing down to ~80% passing 74um, achieved a slight improvement in gold recovery to the flotation concentrate of around 2.5%, and that a natural pH achieved the best flotation recovery for the sample tested.

The addition of CuSO4, varying addition rates of blended collectors (Butyl-Xanthate/Ammonium-dibutyl-dithiophosphate), and comparing the frothers MIBC and "Oil", had no real effect on the flotation response.

YJG's assessment of cyanide leaching showed little difference in recoveries between the grind sizes of 85 - 95% passing 74um and only a marginal improvement in recovery by increasing the pH from 9.5 to 11.5. Increasing the cyanide dose from 1600ppm to 2000ppm NaCN indicated a slight improvement in recovery, however, with such a high-grade sample, most of the results are well within assay error.

ALS Ltd.

ALS Ltd. undertook 1hour LeachWELL™ tests on 29 trench samples from the UR2 Lode and an additional 98 samples from the Tuvatu Lode. The samples were grade controlled to >3g/t Au and ground to a P80 of 75um. The UR2 and Tuvatu lode average trench sample gold recovery was 91.8% and 91.5% respectively.

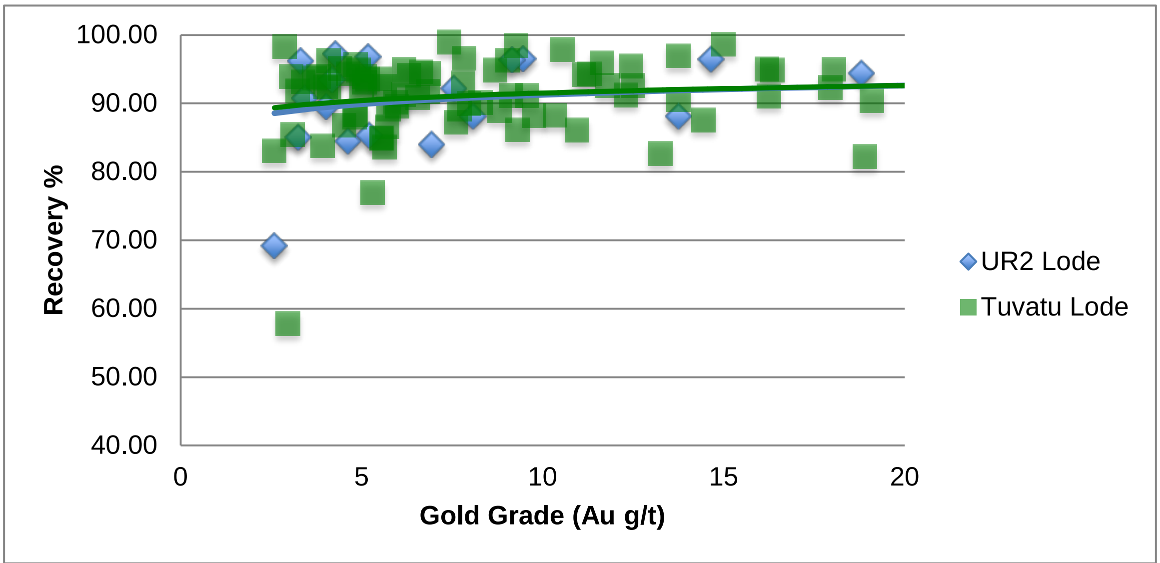


Figure 13.1 LeachWELL™ Tests on UR2 and Tuvatu Lode Trench Samples
Source: Canenco Canada Inc.

Although the LeachWELL tests are considered only indicative, the UR2 samples responded well, with only one sample returning a recovery less than 84% and only 9 of the 28 tests below 90% recovery. The Tuvatu lode trench samples also recovered well, with only seven of the 98 tests below 84% recovery. The variation between these recent results and historical testwork may be due to a level of oxidation of the trench samples.

13.2.2 Historical Testwork

Metcon Laboratories and AMMTEC Ltd. undertook the majority of the testwork aimed at determining the optimum processing route.

Metcon Laboratories Pty. Ltd. (Metcon)
Sample origins for this work and compositing procedures for both test campaigns (00840 & 00867) were documented well. Variability samples were sourced from the UR, UR West (UR 1, 2, 5, URW1, URW2 and URW3) and Murau lodes, representing a range of grades, location and depths within these areas.

The majority of the UR2 composite grades used for the variability test work were below the resource cut-off grade of 3g/t and consequently the UR1 composite was used for the bulk of the investigative processing testwork. Of the 39 variability samples tested, the head grades varied from 0.09g/t Au to 89.4g/t Au, with 23 having head grades above 3g/t Au.

The 00840 campaign variability testing ran parallel gravity, leach and gravity float concentrate leach process routes on each sample, excluding one from UR5.

Table 13.2: Summary of 00840 Variability Results

Summary of results - gravity/leach versus gravity/float/concentrate leach

Zone	URI	UR1	UR1	UR2	UR2	UR2	UR2	UR2	UR2	UR2	UR2	UR2	UR2	UR5	UR5	ave.
Hole	67	158	53 & 67	79	95	122 & 148	123	160	183	207	212	60 & 62	68 & 75	100	152	
depth	225.4-227.6	232.5-236.1	66.3-68.9 83.3-85.3	123.3-125.3	16.5-18.25	268.8-270.4 159.8-161.2	308-310.05	469.5-472.5	419.9-422.6	298-310.3	519.3-526.0	139.0-148.7 125.0-127.2	63.4-65.6 190.9-194	299.3-301.4	192.0-195.0	
head assays																
% S	6.55	0.85	2.44	2.09	3.76	4.5	1.17	0.25	2.22	2.51	3.5	2.71	3.55	3.54	0.22	2.66
ppm Cu	115	210	990	250	160	415	230	125	455	300	1180	430	285	440	350	396
g/t Au	7.30	2.32	10.9	5.45	26.4	4.70	6.95	0.09	2.60	3.00	17.8	4.00	2.65	3.75	0.36	6.55
Other gold head assays																
site g/t Au	3.97	3.87		3.30	18.08		13.76	2.86	2.65	12.68	7.77			8.73	4.51	7.47
leach g/t Au	7.17	3.14	12.21	5.00	4.50	4.26	7.23	0.57	1.73	4.23	6.11	6.50	2.87	-	0.79	4.74
float g/t Au	7.28	2.23	7.29	3.79	3.60	5.41	7.59	0.20	1.15	3.99	7.05	4.79	2.66	1.93	1.42	4.03
gravity/leach route results																
kg/t lime	1.10	0.72	0.94	1.16	2.73	1.41	0.53	0.66	0.66	0.58	0.4	0.39	0.50		0.40	0.87
kg/t NaCN	0.76	0.55	0.76	0.62	0.76	0.76	0.62	0.61	0.56	1.06	0.91	0.79	0.73	no	0.67	0.73
residue g/t Au	0.20	0.14	1.02	0.45	1.42	1.07	3.20	0.09	0.27	1.66	0.73	0.54	0.75	test	0.05	0.83
% gravity gold	51.9	11.3	18.1	34.1	53.3	57.9	34.4	68.0	22.7	29.1	67.1	62.7	22.1		70.2	43.1
% leach gold	45.3	84.4	73.5	56.9	15.2	17.1	21.4	16.2	62.0	31.6	21.0	29.1	51.9		23.5	39.2
% total gold	97.2	95.7	91.6	91.0	68.5	75.0	55.8	84.2	84.7	60.7	88.1	91.8	74.0		93.7	82.3
gravity/float/concentrate leach route results																
float tail g/t Au	0.27	0.09	0.23	1.05	0.33	0.20	0.28	0.01	0.11	1.32	0.30	0.06	0.91	0.15	0.04	0.36
leach tail g/t Au	0.99	4.78	8.9	1.82	13.75	6.8	42.6	1.87	3.39	16.5	3.85	3.3	3.25	1.64	1.72	7.68
kg/t lime	0.23	0.08	0.08	0.25	0.23	0.21	0.12	0.09	0.13	0.12	0.1	0.15	0.23	0.15	0.05	0.15
kg/t NaCN	0.23	0.1	0.18	0.23	0.23	0.28	0.17	0.08	0.21	0.2	0.29	0.19	0.29	0.18	0.08	0.20
% gravity gold	41.1	6.4	26.9	15.5	40.9	56.3	31.5	48.3	22.0	19.3	55.0	45.8	11.0	6.9	43.2	31.3
% gravity + float	96.9	96.1	97.1	76.0	91.6	96.8	96.6	95.2	91.0	68.9	96.2	98.8	71.0	93.3	97.2	90.8
% leach gold	53.4	80.8	62.0	47.1	6.7	11.5	13.9	24.2	51.5	24.9	36.1	47.6	43.1	77.9	51.3	42.1
% total gold	94.5	87.2	88.9	62.6	47.6	67.8	45.4	72.5	73.5	44.2	91.1	93.4	54.1	84.8	94.5	73.5

Source: METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Drill Core Samples – Variability Response Tests - May 2000

Gravity recoveries average approximately 40% across both campaigns, at a primary grind size of P80 75um. The flotation feed was ground to a P80 of 125um, which resulted in a lower gravity recoverable gold result. In the flotation tests, the concentrate was reground to a target P80 of 38um before leaching, however in reality, the grind sizes averaged 55-60um P80 which, based on the mineralogy, would have potentially decreased the concentrate leach stage recovery. The gravity/leach average recovery for 00840 and 00867 is 82.3% and 72% respectively.

The 00867 campaign only assessed gravity concentration followed by gravity tails leaching.

Table 13.3: Summary of 00867 Variability Results

Table 3 Summary of gravity/leach gold recovery tests

sample number (TU...)	146630	146631	146632	146633	146634	146635	146636	146637	146638	146639	146640	146641	148664	148665	148666	
test number	E15	E16	E24	E17	E25	E26	E18	E27	E28	E19	E29	E20	E21	E22	E23	
Ore body	UR West 1	UR West 2	UR West 1	UR West 2	UR West 3	UR West 3	Murau 1	UR West 2	UR West 1	UR West 3	UR West 3	Murau 1	UR 2	UR 2	UR 2	
drill hole	DDH207	DDH176	DDH160	DDH076	DDH144	DDH223	DDH225	DDH228	TUG057	TUG051	TUG040	TUG102	DDH176	DDH092	TUG73	AVE.
actual P80(µm)	73	82	79	76	75	79	75	64	65	65	66	67	75	75	73	72.6
final % NaCN	0.054	0.042	0.040	0.050	0.048	0.060	0.050	0.040	0.056	0.070	0.048	0.048	0.056	0.016	0.054	0.049
kg/t hydrated lime	0.34	0.38	0.32	0.91	1.65	1.16	0.32	0.57	0.41	1.06	0.69	0.47	0.36	1.40	0.44	0.70
kg/t NaCN consumption	0.71	0.89	1.66	0.76	1.17	0.99	0.77	0.91	0.68	1.21	0.94	0.79	0.67	2.01	0.70	0.99
head assays																
% S	1.87	1.73	2.7	1.86	3.24	0.62	0.97	0.98	0.40	5.16	2.25	0.81	1.04	2.87	1.84	1.89
ppm Cu	444	1320	4660	223	255	549	332	240	394	764	257	204	249	539	256	712
ppm Pb	332	132	3040	581	357	20	85	14	84	7540	944	30	70	1150	357	982
ppm Zn	2150	259	1870	910	2510	120	251	185	738	10300	835	360	1960	3040	513	1733
ppm As	231	583	1590	136	74	603	521	528	110	550	28	641	93	597	792	472
ppm Te	17.3	10.5	158	4.5	11.9	0.4	12.0	0.7	13.2	227	2.8	0.9	7.9	12.1	18.2	33
residue assay g/t Au	2.83	5.50	33.5	0.68	0.82	2.23	5.55	2.90	0.77	5.00	0.29	0.77	0.55	0.69	2.95	4.34
calc head g/t Au	12.61	14.39	89.4	13.31	3.47	5.55	12.17	6.29	7.26	42.93	8.16	1.84	1.47	4.61	7.04	15.37
% gravity gold	40.9	39.5	23.5	74.1	18.7	50.2	37.5	39.5	54.1	9.9	78.0	22.8	26.3	18.3	37.7	38.1
% leach gold	36.7	22.3	39.0	20.8	57.6	9.7	16.8	14.4	35.4	78.4	18.5	35.3	36.7	66.7	20.4	33.9
% gravity + leach gold	77.6	61.8	62.5	94.9	76.3	59.9	54.4	53.9	89.5	88.4	96.5	58.1	63.0	85.0	58.1	72.0

Source: METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Gold Ore - July 2000

Of the testwork conducted, the Metcon variability work is most representative of the range of mineralization to be treated and forms the basis of the recoveries in this report.

AMMTEC Ltd. (AMMTEC)

The samples for this work and compositing procedures are well documented. Samples were split, one half being set aside for variability testing and the remaining combined into two "Master Composites" (MC) representing the Nasivi and Upper Ridges (UR) Lodes. The Nasivi and UR MC head grades were 11g/t and 7 g/t Au respectively and the samples were received with a 93um and 95um P80 with the majority of the tests undertaken at this grind size.

The gravity recovery on the AMMTEC MC's is consistent with other testwork, achieving a 31-32% gold recovery. After processing the entire MC's through the laboratory concentrator, the gravity tails assayed 7.5g/t and 4.9g/t Au, and 2.1g/t & 9.0g/t Ag for the Nasivi and UR MC's respectively. The silver grades should be noted moving forward to the next phase of study as it may impact the Adsorption, Desorption, and Recovery (ADR) process selection.

The leach recoveries at different grind sizes is consistent with other work, showing a slight improvement with a finer grind, but like other test campaigns, only ground down to 45um P80, even though the mineralogy indicates a finer grind ~20um P80 would likely be required to achieve any significant recovery improvements.

A blend of the MC gravity tailings was made to test the flotation response. The blend was ground to 80% passing 75um and the float test achieved a 90.5% recovery to a 26.6g/t gold concentrate. The AMMTEC work is the only test work that undertook a fine regrinding of the flotation concentrate of 80% passing 20um and also the only test that leached the flotation tailings.

Table 13.4: Cyanidation Leach Testwork of Flotation Products

Leach Sample Description	Test No.	Grind Size P_{80} (μm)	Calc'd Head Au (g/t)	% Au Ext @ 72 Hrs
Concentrate	HS3107	As is	27.2	91.84
Concentrate	HS3108	20	26.3	93.20
Tailings	HS3109	As is	0.876	75.34

*It should be noted that LeachWELL™ was added at 2.0% and that all tests were oxygen sparged.
Source: AMMTEC Ltd. - Metallurgical Testwork Conducted on Samples of Ore from the Tuvatu Gold Deposit - December 1997

The recovery of the 80% passing 20um reground float concentrate sample was 93.2% while the flotation tails leaching recovery at 80% passing ~75um was 75.3%. No leach kinetic data was taken, so it is not possible to determine if a shorter leach time would have achieved similar recoveries with this sample, although when compared with the Metcon variability samples, some achieved comparable recoveries in 24hours. Due to this, the process design includes pumping the flotation concentrate tailings to the head of the flotation tailings leaching circuit, effectively achieving greater than 72 hours residence time for that stream.

13.2.3 Comminution

13.2.3.1 Crushing Work Indices

Gekko systems Inc. (Gekko) tested the Bond Crushing Work Index (CWi) on 20 specimens of Tuvatu core. The CWi describes the competency of the mineralized material at larger particle sizes and can be used to calculate crusher power requirements. The average CWi for Tuvatu samples of 12.5 kW/t is a good to average result, however a varying degree of competency was observed in the sample as shown by the minimum value of 2.8 kWh/t, the maximum value of 26.6 kWh/t and a standard deviation of 6.6.

13.2.3.2 Abrasion Indices

The Abrasion Index (Ai) provides an indication as to how abrasive a rock type might be and is used to calculate metal wear rates in process plant equipment such as crushers and ball mills.

Table 13.5: Abrasion Indices from Various Test Work Campaigns

Index	Gekko	OMC Nasivi Bulk Sample	OMC Nasivi C"	OMC UR Core	AMMTEC Tuvatu Lode Comp.	Amdel Tuvatu Lode Core
Abrasion	0.0585	0.145	0.169	0.184	0.0943	0.145

Due to recent industry discussions around the validity of the abrasion index test, the most conservative test result from the Upper Ridges at 0.184, was used for design in this study.

Even so, this is still a low abrasion index, indicating that the Tuvatu mined material are not expected to be abrasive.

13.2.3.3 Rod and Ball Mill Work Indices

The Bond Work index (BWi) used to size the ball mills was based on a review of the various work index determinations in the different test programs.

The work indices are quite variable, with BWi ranging from 16.3kWh/t in the Nasivi lode up to 20kWh/t in the UR lode. The results over the various test work programs show that there is a high degree of variability in the indices even within the same lode, such that UR ranges from 17.4kWh/t to 20kWh/t, the Tuvatu lode ranges from 16.3-18.1kWh/t. This is still a competent rock, with work index results suggesting a higher specific energy requirement for most of the mineralized material throughout the Tuvatu project.

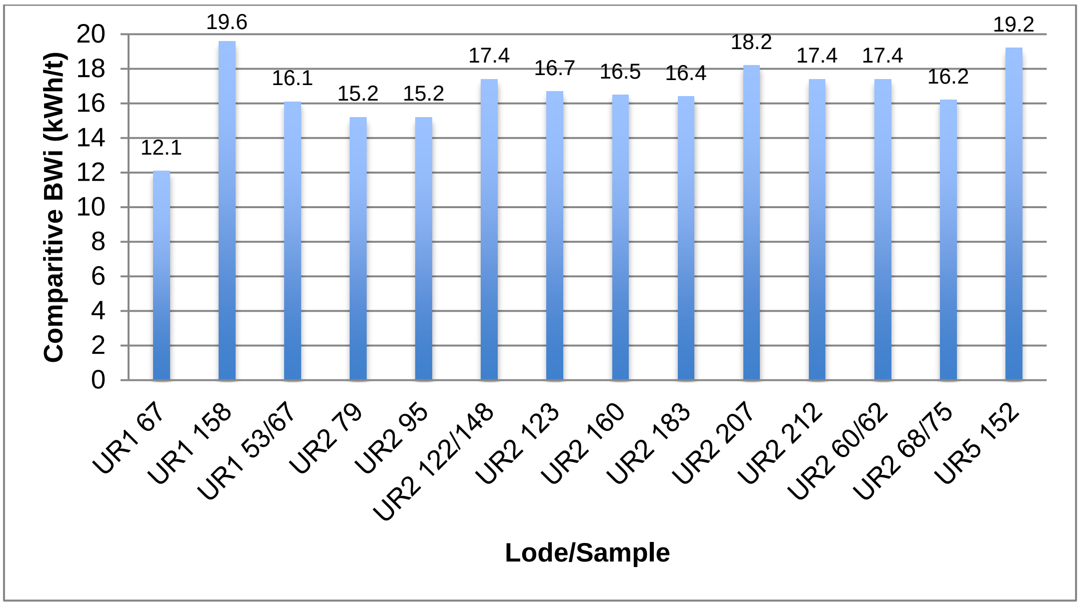
Table 13.6: Bond Rod Mill Work Indices (RWi) and Bond Ball Mill Work Indices (BWi) from Various Test Work Campaigns

Index	Metcon UR2 Core	OMC Nasivi Bulk Sample	OMC Nasivi Core	OMC UR Core	AMMTEC Tuvatu Lode Comp.	Amdel Tuvatu Lode Core	75th Percentile
RWi (kwh/t)	18.6	17.1	17.7	20.2	20.2	17.1	19.8
BWi (kWh/t)	17.4	16.3	17.6*	19.6*	18.1	16.3	18.6

* Average of two values

Metcon undertook relative work index testing on 14 of the 15 samples from the 00840 test work campaign. A range of indices from 12.1 kWh/t to 19.2 kWh/t was determined, suggesting there will be a degree of variability in the hardness between lodes. It is interesting to note that the there is only minor variability within the UR2 lode indicating that it may be a more consistently competent lode. These values can only be considered as indicative as they were not based on the formal work index determination approach and were only design to illustrate the work index variability. These values have not been utilized for design purposes in this study.

Figure 13.2: Comparative Wi Values (Metcon 00840)



Source: METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Drill Core Samples – Variability Response Tests - May 2000

The 75th percentile of the reported testwork data sets was utilized as the design value for sizing the ball mills.

13.2.4 Cyanide Detoxification

In December 1997, 968t of mineralized material from the Tuvatu adit was processed through the then Emperor Gold Mining Company Ltd. owned Vatukoula Mill as a batch treatment campaign. This is the only indication of any cyanide detoxification testwork being undertaken on Leach/CIP (carbon-in-pulp) tailings using the INCO, SO_2/Air detoxification method. The full report was not available, however, it was noted in the campaign report that "the detoxification of the Tuvatu sample is relatively simple with a reduction in CN_{Total} from 150ppm down to 2ppm achieved" and that "the detoxification using the INCO SO_2/Air method is effective."

The SO2/Air method of cyanide detoxification has been selected for the process in this study.

13.3 Process Flowsheet

The development of the process flowsheet for the Tuvatu project began with the Feasibility Study produced by Bateman Engineering Pty Ltd in 2000 for Emperor Gold Mining Company Ltd. where a gravity / flotation / CIL processing was chosen over a gravity / CIL route. In that study, only the flotation concentrate was to be leached with cyanide while the flotation tailings were to be cycloned for use as mine backfill. This process flowsheet selection was essentially driven on the basis of a reduced environmental and mine footprint, however recoveries via the flotation route have generally been lower than cyanidation in almost all-comparative testing.

With review of the mineralogy, it is reported that the more finely disseminated gold is associated with the sulphides, and consequently a much finer regrind size (<20um) is required for a more complete sulphide liberation in the flotation concentrate, combined with a longer leach time, in order to achieve higher and more consistent recoveries.

The processing facility flowsheet has been selected based on the criteria defined by the mineralogical characteristics of the mineralized material and metallurgical testwork undertaken to date. The comminution facility is a conventional two-stage crushing and screening circuit, followed by two-stage grinding. The grinding circuit, which includes gravity recovery, then feeds flotation where a sulphide concentrate is produced and reground to 20um prior to entering the Carbon-In-Leach (CIL) circuit. Both the flotation tails and concentrate are leached, with the concentrate CIL tails recirculating to the feed of the flotation tails CIL circuit, the combined discharge being pumped to detoxification and tails deposition.

13.4 Metallurgical Recovery

The three studies that form the basis of the metallurgical recovery prediction are;

- METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Gold Ore - July 2000.
- METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Drill Core Samples – Variability Response Tests - May 2000.
- AMMTEC Ltd. - Metallurgical Testwork Conducted on Samples of Ore from the Tuvatu Gold Deposit - December 1997

The gold recovery is based on Metcon variability results from samples with >3g/t Au head grade, in an effort to be representative of the resource and potential mill feed grades. This approach was found to be conservative; as to include the lower grade material increased the overall recovery result.

Due to only one test being undertaken by AMMTEC on leaching the flotation tails, returning a recovery of 75.3%, the leach stage recovery from the Metcon gravity/leach variability tests, from both campaigns was used to predict the variable leach response in the flotation tails CIL circuit. This brought the recovery for the flotation tails to 56.9%.

A gold recovery balance was then undertaken using the Metcon gravity/flotation/concentrate leach results, in order to calculate the gold that would be potentially recovered with the selected process flowsheet. The median gold recovery using this approach was 86.8%, which was decreased by 0.5% to account for solution losses that will typically occur in the commercial plant.

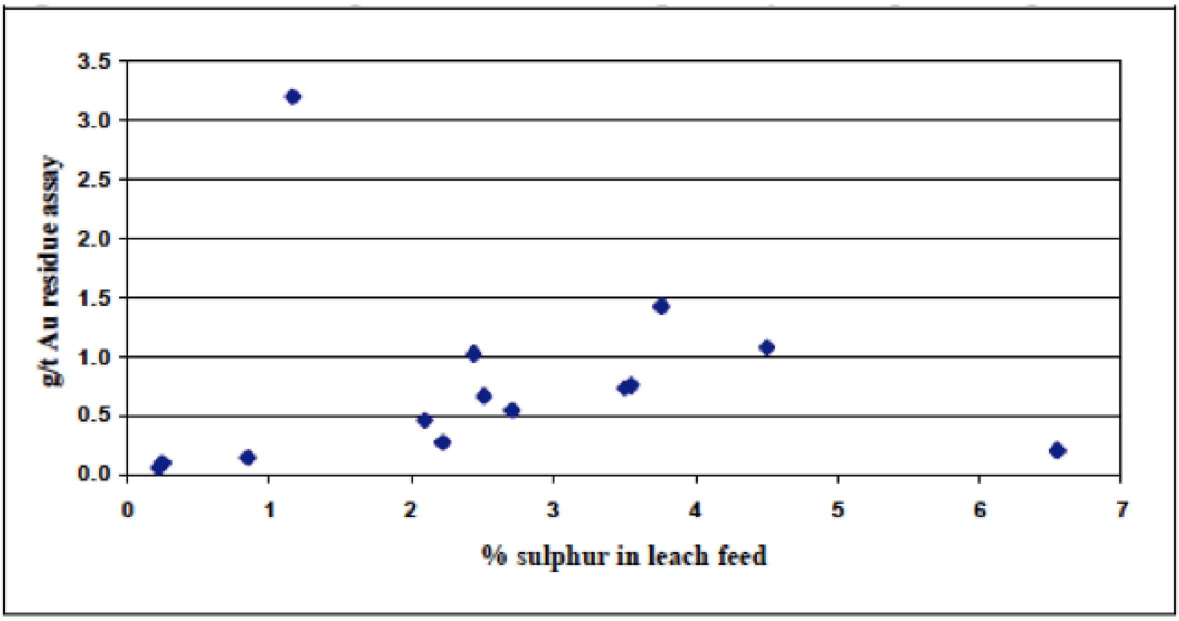
PEA Study Gold Recovery = 86.3%.

This recovery value is calculated only from consideration of the variability leach results and assumes that the samples tested are moderately representative of the mineralized material to be mined, but at this level of study, does not weight the results based on proposed mine plan.

13.5 Recommended Testwork

Through each test work campaign, a number of recovery improvement programs were conducted in an effort to explain the variable recoveries, however no conclusive reason was evident.

Figure 13.3: Leach Residue Grade vs. Sulphur Head Grades



Source: METCON Laboratories Pty. Ltd. - Metallurgical Development Testwork on Tuvatu Drill Core Samples – Variability Response Tests - May 2000

The samples with lower gold recoveries (higher leach residue grades), generally have relatively higher sulphur grades, but a relatively higher sulphur grade does not always result in lower recoveries.

With this in mind, the following metallurgical testwork programs are recommended moving forward:

- A more comprehensive mineralogical study, including gold deportment, in an attempt to characterize the gold association variability.
- Additional crushing and grinding work indices through the different lodes,
- BWi, CWi, abrasion testing.
- More detailed GRG work with vendors.
- Flotation testwork, (reagent selection, variable optimization, lock-cycle).
- Flotation Tailings and Concentrate regrind / leach characterizations.
- Aeration and leaching optimization.
- Thickening testwork.
- Cyanide detoxification testwork based on optimized process.

The estimated cost for this testwork is expected to be approximately $420,000.

14.0 MINERAL RESOURCE ESTIMATES

A number of historical mineral resource studies have been carried out at Tuvatu by previous operators over the period from 1997 to 2010 (see Section 6.3).

Previous resources were developed with classic techniques suited to broad zones of mineralization of relatively homogenous mineralization. Specifically the compositing of individual samples to one m down hole and utilising inverse distance cubed (ID^3) linear weighting techniques of the capped data.

MA considers that a two dimensional estimate using grade and thickness across the narrow vein is a better method. The model has to incorporate a level of conceptual interpretation (implicit modelling) as the veins are very narrow. Traditional cross section interpretation (explicit modelling) is near impossible.

The methodology used in this style of resource estimate is chosen as it facilitates better models of vein thicknesses and does not have the problems introduced by attempting to construct very narrow wireframes: vein walls crossing and too many small blocks. The 2D re-folded model provides a more realistic vein model ideal for underground design or open pit design where veins come to surface.

In May 2014, MA was commissioned by Lion One to review the geology and create a resource estimate for the Tuvatu Gold Project. The resource was estimated for each vein individually using Ordinary Kriging of width and grade, the latter using accumulations, into a 3D block model.

MA updated the May 2014 resource in January 2015. Changes to the block model are not considered material changes to the May 2014 resource estimate. The input data (drill hole data base) and estimation parameters for the block model have not changed. Minor adjustments to the vein tags occurred and several veins were split to remove below cut off material. The extrapolation of the vein extents was reduced to generally less than 30m. The parent block size of the model increased from $9.6m^3$ to $10m^3$ with sub-blocks reduced from $1.2m^3$ to $0.625m^3$. The reduced sub-block size allows more accurate design of stopes and dilution during the mine design phase.

14.1 Approach

The main consideration as to whether a 2D or 3D estimate should be performed is:

1) Are there differences in mineral domains horizontally within the vein and can separating the vein improve grade estimation and control of feed grades.

2) Does the mining equipment have the ability to mine smaller or multiple cuts across the vein? If not, but the estimates are better, then grades can be combined in a manner that makes the overall estimate more predictive. If there are great horizontal differences an engineering study could be considered to determine if there is an economic benefit to changing the mining method.

MA considers that there is no appreciable difference in mineralisation across the vein, which are very narrow (less than 1 m in places) and no mining selectivity across the vein is possible. Thus a two dimensional estimate of grade and thickness across the narrow vein is a better method to apply at the Tuvatu Gold Project. In essence the true thickness and grade (and geostatistics) of a vein domain are estimated in unfolded space, i.e. on a 2D grid. This vertical plane is sub-parallel to the vein direction, thus grades and thicknesses are absolutely tied to the informing samples/composites. The process of "unfolding" and "refolding" results in some smoothing of vein contacts, which may result in minor apparent spatial departures of the vein wireframes from some composite centroids.

14.2 Supplied Data

MA was supplied with Lion One's drill database, with the database name: Database ExportDrillHoles.mdb with the following structure shown in Table 14.1.

Table 14.1 Master Database Structure

Table Name	Description	Record Count
Assay	Assay intervals and associated gold and silver results	71,633
Collar	Collar information associated with drill type and location	1,131
Lithology	Logged lithological units	9,152
SG Data	Bulk Density data from drill core samples	1,955
Survey	Down hole survey data	4,054
Weathering	Logged oxidation codes	48,871
Site_Tags	Interpreted veins identification tags from site	1,833

MA created a new table to store the vein intercepts in, veins were check in three dimensions, cross section and long section. MS Access queries were run to ensure mineralisation was not excluded adjacent to composites and un-necessary waste samples were not included. There are examples of material below 0.3 g/t being included however these tags are required to constrain and ensure vein continuity.

There were mineralised samples outside vein tags (Table 14.2). The one exception to the high grade being excluded is in SKL8 where a 6.51 m mineralised sample is very improbable. Two additional "long" samples are recorded in the data base and are included in the estimate as the results are in-line with expected grades possible from a "long" sample. MA recommends confirming the sample lengths in log books or physically in the core trays.

Table 14.2 Mineralised Samples Outside Vein Tags

Hole_id	depth_from	depth_to	Au_ppm_BEST	Zonecode	Position
TUDDH-078	82.3	86	6.51	SKL8	Not included
TUDDH-42	32.5	37.5	1.58	M1	Included
TURC-21	7	10	0.63	M1	Included

14.3 Dimensions

Database extents (Table 4.3) are more extensive than the mineralised resource described in this report.

Table 14.3 Database Extents

Database	Min (m)	Max (m)	Extents (m)
Northing	3917051	3923892.0	6841.0
Easting	1874687.4	1878637.9	3950.5
RL	99.75	547	447.3
Hole Depth	0.88	600.6	NA

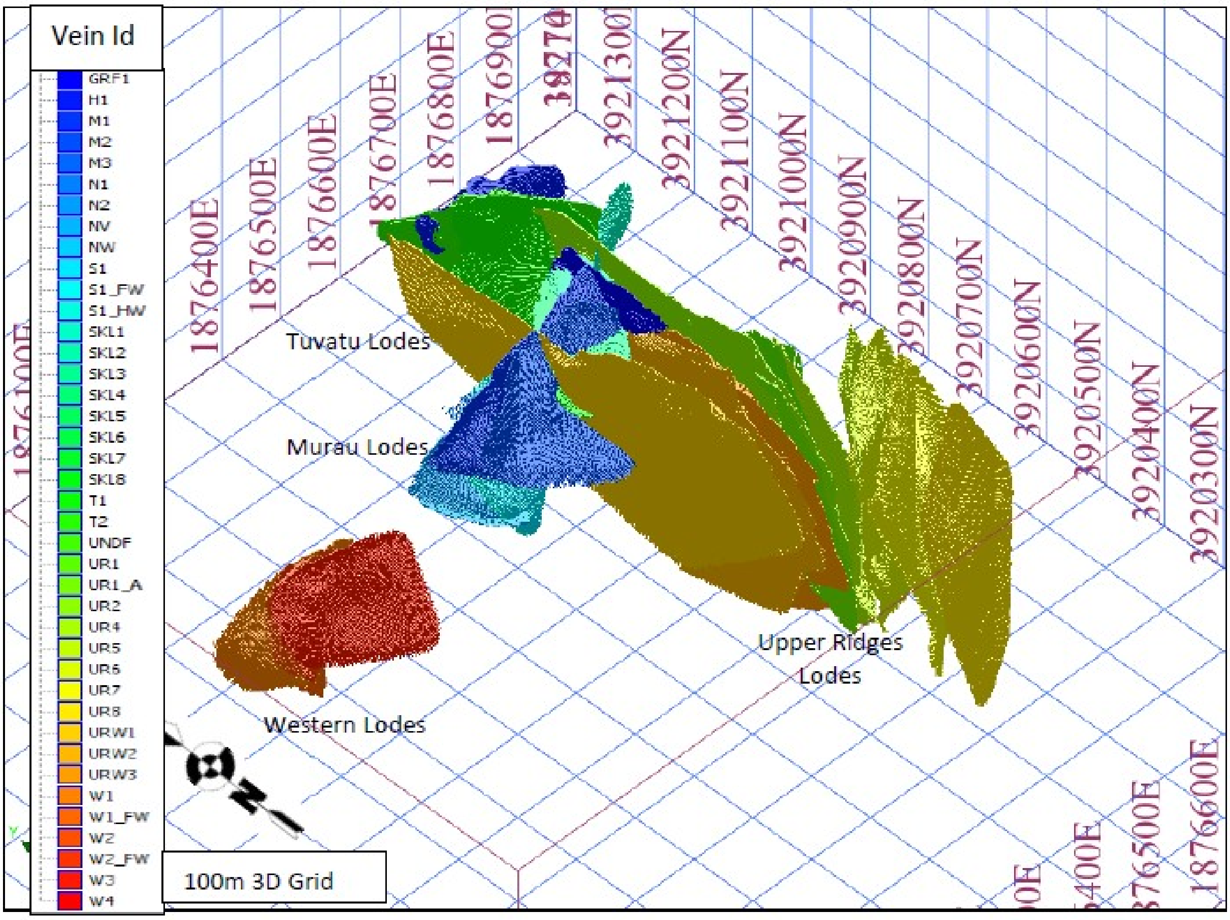
Although the Upper Ridges and associated north south veins cover a strike extent of 900 m (3920200 m N to 3921100 m N), individual veins have shorter strike lengths. The Western veins and Murau veins strike approximately 800m (1875750 m E to 1876650 m E) with a break between the Western veins and Murau of approximately 150 m.

14.4 Geologic Interpretation

Tuvatu is one of several gold prospects known from the Sabeto area of north-western Viti Levu. Mineralization is structurally controlled and is hosted by a series of sub-vertical veins, shallow dipping veins and stockworks. The main mineralized zone (Upper Ridges) comprises eleven principal lodes with a strike length in excess of 500 m and a vertical depth of more than 300 m (Figure 14.1). Another major zone of mineralization (Murau) strikes east-west and consists of two major lodes with a mapped strike length in excess of 400 m. Although gold mineralization is primarily hosted in monzonite it can also occur in the volcanic units. Lodes are narrow, generally less than 1 m up to a maximum of 7 m, and ore grades are erratic. Lode mineralogy is varied, with most veins containing quartz, pyrite, and base metal sulphides. A total of 39 different lode structures were identified in the resource area including 11 lodes in the Upper Ridges area, 3 lodes in the Murau area, 4 lodes in the West area, 2 lodes in the Tuvatu area and the stockwork veins in the SKL area.

Veins were identified as intercepts greater than 0.5 g/t Au, however due to the tight nature of the veins relatively few assays less than 1.0 g/t are incorporated. The low grade boundary allowed networks of narrow veins (1 mm to 200 mm wide) to be “bulked” into substantial vein intersections. In areas where the vein has propagated as a single thin veinlet assays as low as 0.3 g/t were incorporated as edge dilution, notably where veins/assay composites were less than 1m thick. Portions of the vein were selected based on lithology logs or interpreted strike extensions despite supporting assay data in these situations consisting of values below 0.3 g/t Au.

Figure 14.1 Tuvatu Lodes (Oblique View (45°E -45°)



14.5 Data Preparation and Statistical Analysis

Prior to a statistical analysis, grade domaining is normally required to delineate homogeneous areas of grade data, at Tuvatu the individual veins are assumed to represent sufficiently homogenous mineralisation. Statistical analysis does not take into account the spatial relationships of the data.

The purpose of statistical analysis is to define the main characteristics of the underlying grade distribution to assist with the geological and grade modelling work. This process is important as the statistics of the individual sample populations can influence how the grade data is treated and the application of the grade estimation techniques. For example highly skewed data may require special grade capping and indicator semivariogram analysis.

The drill hole database is stored in an MS Access relational database. The Tuvatu database is connected directly to Surpac for data display, vein compositing, wire-framing, unfolding, estimation refolding storing in a 3D block model.

Statistical analysis of the grade data was principally carried out using the SurpacTM Software package. SurpacTM was used to export composite drill hole data as a comma separated file (CSV) for importation into SupervisorTM. More detailed spatial analysis (variograms) was conducted within Supervisor. The Supervisor package is an internationally recognised geological and mining software toolbox which incorporates geostatistical tools that can be used at all stages of the mining process from initial feasibility studies though to production control.

14.5.1 Drill Hole Spacing

Drill hole data spacing is variable within each domain. Above 50 m RL the drill spacing in Upper Ridges (UR) is reasonably tight on a 25 m grid and below 50 m RL the drill spacing increases to approximately 50 m grid. UR western lodes are less well drilled. Development exists on UR2, UR5 and GRF veins. Murau veins are shallower and are generally drilled at 25 m spacing.

14.5.2 Domains and Stationarity

A domain is a three-dimensional volume that delineates the spatial limits of a single grade population, has a single orientation of grade continuity, and is geological homogeneous and has statistical and geostatistical parameters that are applicable throughout the volume (i.e. the principles of stationarity apply). Typical controls that can be used as the boundaries to the domains include structural features, weathering, mineralization halos and lithology. Within the Tuvatu deposit individual veins were used to define the domain. It is understood that the average grade of veins vary along strike and down dip as a result of high grade shoots, which are controlled by search ellipses, variography, and the number of informing intercepts selected.

14.5.3 Compositing

The two-dimensional technique used by MA to estimate resources at Tuvatu uses a single down-hole (or along channel) composite sample extracted from the drill hole database for each intercept within the vein. True thickness was calculated using the overall dip and dip direction of the vein. It is assumed that the grade of the vein at each location is the grade of the intercept thus reducing concerns of volume variance and negating the need for constant length samples. Scatter plots showed no correlation between grade and thickness, thus grade and thickness are treated as independent samples.

14.5.3.1 Channel Samples

Channel samples were used to guide the location, grade and thickness of veins at surface. In areas of intense channel sampling or where channels were sampled twice only, one channel was selected to inform the estimate. The following are examples where one channel is selected:

- Channel 17 and 18 are parallel only channel 18 is used.
- Channel 31 and 33 are parallel only Channel 33 is used.

14.5.4 Summary Statistics

Summary statistics for vein gold and thickness by area are presented in Table 14.4, Table 14.5 and Table 14.6. Informing sample grades (uncapped) for the upper ridges veins range from 2.16 g/t Au and 0.52 m for UR7 and 13.53 g/t and 4.65 m URW1 (Table 14.4), in the Murau Area veins range from 1.89 g/t Au and 1.95 m thick for Nasivi West to 3.74 g/t Au and 2.01 m thick for Snake vein

(Table 14.5) and SKL veins have a very high vein at 23.36 g/t (Table 14.6).

Table 14.4 Summary Statistics for Upper Ridges Veins

	Vein	GRF1	H1	T1	T2	UR1	UR1_A	UR2	UR4	UR5	UR6	UR7	UR8	URW1	URW2	URW3
Gold	Count	71	22	29	45	43	64	241	78.00	79.00	35.00	20.00	9.00	65.00	61.00	93.00
	Minimum	0.02	0.01	0.13	0.01	0.01	0.01	0.01	0.01	0.01	0.02	0.04	0.03	0.01	0.01	0.01
	Maximum	47.5	16.9	11.7	14.8	121.0	107.5	198.6	57.4	174.0	24.0	8.0	15.8	285.5	26.2	75.4
	Mean	8.17	3.49	3.15	3.64	7.38	7.10	7.41	5.47	7.67	3.32	2.16	5.28	13.53	2.73	8.02
	Median	3.12	2.02	2.55	2.48	2.28	3.35	2.51	1.84	2.12	1.80	0.83	2.70	1.85	0.68	2.61
	Std Dev	9.73	4.60	2.78	3.46	18.9	14.2	19.7	10.09	22.29	4.94	2.61	5.73	41.25	4.89	14.53
	CV	1.19	1.32	0.88	0.95	2.56	1.99	2.66	1.84	2.91	1.49	1.21	1.08	3.05	1.79	1.81
True Thickness	Count	68	22	29	45	44	62	237	78	80	35	22	9	67	57	89
	Minimum	0.04	0.09	0.09	0.08	0.09	0.00	0.11	0.01	0.07	0.01	0.13	0.02	0.01	0.15	0.07
	Maximum	3.45	4.59	6.24	6.18	5.58	3.93	6.24	4.52	2.66	2.07	1.35	0.57	4.65	4.50	5.81
	Mean	1.47	1.25	1.61	1.83	1.04	0.50	1.34	0.74	0.75	0.51	0.52	0.29	0.82	0.80	1.34
	Median	1.22	1.09	0.82	1.01	0.70	0.25	1.10	0.53	0.50	0.45	0.44	0.34	0.51	0.57	0.93
	Std Dev	1.09	1.03	1.75	1.76	0.99	0.70	1.04	0.74	0.62	0.37	0.34	0.17	0.95	0.74	1.26
	CV	0.74	0.83	1.08	0.96	0.95	1.40	0.78	1.00	0.82	0.73	0.66	0.59	1.15	0.93	0.94

Table 14.5 Summary Statistics for Murau Veins

	Vein	M1	M2	M3	N1	N2	NV	NW	S1	S1_FW	S1_HW	SKL1
Gold	Number of samples	52	37	13	7	14	5	6	19	18	13	12
	Minimum	0.01	0.02	0.12	0.14	0.01	0.50	0.71	0.10	0.03	0.10	0.40
	Maximum	12.52	30.08	11.73	7.15	15.34	7.20	3.20	51.04	8.84	22.80	36.75
	Mean	2.61	2.71	3.74	3.51	3.01	2.48	1.89	7.57	2.52	6.19	8.47
	Median	1.66	1.39	2.31	3.36	1.22	1.68	1.89	3.74	1.28	3.71	5.37
	Standard Deviation	2.86	4.87	3.60	2.62	4.21	2.45	0.92	11.80	2.54	6.84	9.86
	Coefficient of variation	1.09	1.79	0.96	0.75	1.40	0.99	0.49	1.56	1.01	1.11	1.16
True Thickness	Number of samples	46	35	11	7	14	5	6	19	18	13	12
	Minimum	0.09	0.02	0.00	0.16	0.08	0.47	0.57	0.24	0.19	0.21	0.17
	Maximum	5.51	9.86	0.93	1.53	5.43	2.21	4.44	4.93	3.73	4.53	1.55
	Mean	1.35	1.37	0.37	0.86	1.28	1.33	1.95	2.01	1.33	1.66	0.64
	Median	0.77	1.33	0.32	0.93	0.86	1.27	1.70	1.56	0.97	1.02	0.58
	Standard Deviation	1.37	1.60	0.29	0.47	1.39	0.68	1.27	1.31	1.13	1.30	0.45
	Coefficient of variation	1.01	1.17	0.78	0.54	1.08	0.51	0.65	0.65	0.85	0.78	0.70

Table 14.6 Summary Statistics for SKL veins

	Vein	W1	W1_F W	W2	W2_F W	W3	W4	SKL2	SKL3	SKL4	SKL5	SKL6	SKL7	SKL8
Gold	Count	22	9	25	12	25	5	8	16	38	24	24	11	6
	Minimum	0.07	1.22	0.02	0.31	0.09	0.21	0.83	0.31	0.01	0.12	0.01	0.54	0.55
	Maximum	15.85	10.74	39.1	15.9	44.8	7.04	7.68	11.3	32.0	32.3	79.4	172.9	10.5
	Mean	3.33	3.71	7.66	3.34	5.51	3.25	3.61	3.78	4.29	8.85	6.94	23.36	4.39
	Median	2.05	2.31	3.02	1.34	1.69	3.03	3.45	3.07	2.17	4.98	2.30	4.32	3.34
	Std. Dev.	4.34	3.25	10.4	4.53	9.98	2.31	2.07	3.34	6.40	9.65	16.1	48.35	3.49
	CV	1.30	0.88	1.36	1.36	1.81	0.71	0.57	0.88	1.49	1.09	2.32	2.07	0.80
True Thickness	Count	22	9	25	10	25	5	8	16	34	24	24	11	6
	Minimum	0.15	0.19	0.24	0.30	0.04	0.21	0.34	0.19	0.10	0.16	0.12	0.09	0.48
	Maximum	5.25	2.58	5.78	3.81	4.85	3.94	2.18	4.01	4.56	4.07	1.69	2.46	3.67
	Mean	1.78	1.23	2.51	1.73	1.24	1.66	1.03	0.99	0.98	0.98	0.62	0.72	1.98
	Median	1.28	0.52	2.02	1.62	0.74	0.86	0.98	0.77	0.70	0.71	0.45	0.49	1.68
	Std. Dev.	1.61	0.98	1.82	1.16	1.13	1.42	0.64	0.89	0.93	0.87	0.43	0.64	1.14
	CV	0.90	0.80	0.72	0.67	0.91	0.86	0.62	0.90	0.95	0.90	0.69	0.89	0.58

14.5.5 Grade Capping

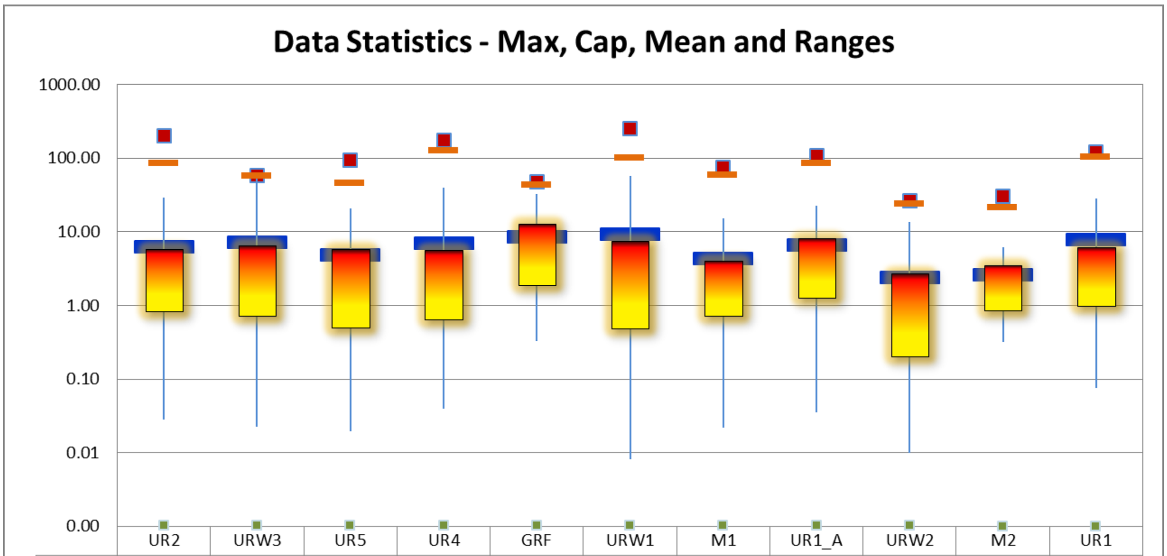
Capping is the process of reducing the grade of the outlier sample to a value that is representative of the surrounding grade distribution. Reducing the value of an outlier sample grade minimises the overestimation of adjacent blocks in the vicinity of an outlier grade value. At no stage are sample grades removed from the database if grade capping is applied.

Veins that contain more than 50 intercepts were assessed for outliers, via histograms log probability plots and metal loss. Uncapped and capped summary statistics are presented in Table 14.7 and graphically represented in Figure 14.2. Veins with less than 50 intercepts were considered unreliable representations of the distribution, and the grade cap was selected at the 97.5th percentile which often resulted in only one value being capped. Grade caps applied are presented in Table 14.8.

Table 14.7 Un-capped and Capped Summary Statistics Per Vein with More Than 100 Intercepts

	Uncapped Composite Data				Capped Composite Data					Grade	
Domain	Count	Mean	Maximum	CV	Count	# Cap.	Mean	Cap	CV	% Cap	% ∆
UR2	234	7.10	198.60	2.708	234	3	6.23	86.8	1.97	1.28%	-12%
URW3	91	7.30	57.50	1.771	91	1	7.30	57.4	1.77	1.10%	0%
UR5	80	5.50	94.29	2.171	80	1	4.90	46.3	1.63	1.25%	-11%
UR4	78	7.62	174.00	2.838	78	1	7.04	129.1	2.46	1.28%	-8%
GRF	64	8.62	47.46	1.174	64	1	8.57	44.0	1.16	1.56%	-1%
URW1	67	11.60	252.64	2.944	67	1	9.34	101.7	2.15	1.49%	-19%
M1	57	4.65	75.40	2.225	57	1	4.37	59.4	1.93	1.75%	-6%
UR1_A	62	7.05	107.48	2.049	62	1	6.69	85.1	1.79	1.61%	-5%
URW2	71	2.46	26.24	1.866	71	1	2.43	23.9	1.82	1.41%	-1%
M2	38	2.81	30.08	1.724	38	1	2.59	21	1.38	2.63%	-8%
UR1	39	8.28	121.00	2.413	39	1	7.88	105	2.24	2.56%	-5%

Figure 14.2 Box and Whisker Plot for Vein with More Than 50 Intercepts



	UR2	URW3	UR5	UR4	GRF	URW1	M1	UR1_A	URW2	M2	UR1
Series7	6.23	7.30	4.90	7.04	8.57	9.34	4.37	6.69	2.43	2.59	7.88
Max	198.60	57.50	94.29	174.00	47.46	252.64	75.40	107.48	26.24	30.08	121.0
Cap	86.84	57.41	46.26	129.10	43.98	101.73	59.37	85.06	23.92	21.43	105.23
25%	0.83	0.73	0.51	0.65	1.89	0.49	0.72	1.26	0.20	0.84	0.97
97.5%	28.96	52.83	20.52	39.59	32.16	57.12	15.28	22.88	13.41	6.24	28.56
2.5%	0.03	0.02	0.02	0.04	0.33	0.01	0.02	0.04	0.01	0.32	0.08
75%	5.88	6.50	5.81	5.64	12.66	7.56	4.00	8.22	2.71	3.55	6.13
Series8	234	91	80	78	64	67	57	62	71	38	39

Table 14.8 Grade Caps applied

Vein	Gold Cap	Vein	Gold Cap
GRF1	44	NV	uncapped
H1	16.93	NW	uncapped
T1	11.68	S1	51.04
T2	14.83	S1_FW	8.84
URW1	150	S1_HW	22.8
URW2	23.9	SKL1	36.75
URW3	57.4	SKL2	999
UR1	105	SKL3	11.3
UR1_A	85.1	SKL4	32
UR2	86.8	SKL5	32
UR4	129.1	SKL6	79.44
UR5	46.3	SKL7	114
UR6	24	SKL8	114
UR7	8.05	W1	15.85
UR8	uncapped	W2	10.74
M1	59.4	W2_FW	35.25
M2	21	W1_FW	15.91
M3	11.73	W3	44.8
N1	uncapped	W4	uncapped
N2	15.34		

14.6 Variography

The most important bivariate statistic used in geostatistics is the semivariogram. The experimental semivariogram is estimated as half the average of squared differences between data separated exactly by a distance vector ‘h’. Semivariograms models used in grade estimation should incorporate the main spatial characteristics of the underlying grade distribution at the scale at which mining is likely to occur.

The semivariogram analysis was undertaken in Surpac within each major vein; both gold and thickness were considered as separate variables. Variograms analysis was undertaken in unfolded space, thus only the plunge was defined through variography analysis. 2D experimental variograms are modelled using a nugget (C0) and two spherical models (C1, C2), occasionally one spherical model was sufficient, particularly for modelling the thickness variograms. The modelled variogram geometry is consistent with the interpreted mineralisation wireframes due to the unfolding process, a plunge component is incorporated where identified and modelled accordingly. The overall ranges modelled for the major axis are in excess of the drill spacing, and appropriate anisotropy fitted to the semi-major axis. The minor axis is not considered as all veins have been composited across the veins and it is not required.

Gold and thickness variogram sills were standardized to 1. Nugget effects for gold were generally low to moderate, ranging from 0.1 to 0.51, and the range (A2) of the variograms varied from 20 m to 85 m for gold variogram models. Thickness variogram nuggets ranged from 0.1 to 0.65 and more commonly only required one structure to model, with variogram ranges from 20 m to 90 m. UR and URW veins displayed far greater continuity for thickness with ranges of 200 m to 500 m for their second structure.

The major axis of the ellipse is orientated in the unfolded plane. North striking veins are unfolded to the X axis, east striking veins unfolded to the Y axis and flat veins unfolded onto the Z axis. The plunge is the angle above (<90°) or below (>90°) the horizontal.

Variogram parameters are summarised in Table 14.9 (gold) and Table 14.10 (thickness).

Table 14.9 Semivariogram Parameters for Gold Vein Domains

Vein Set	Variogram Model	Plunge	Max Range	C0	C1	A1	C2	A2	ratio1
GRF1	SPHERICAL	90	20	0.51	0.45	6.4	0.16	20	1
H1, T1 T2	SPHERICAL	100	50	0.25	0.75	50	-	-	1
URW1, URW2	SPHERICAL	20	85	0.14	0.86	85	-	-	1.75
URW3	GAUSSIAN	140	85	0.14	0.86	85	-	-	1.75
UR1	GAUSSIAN	160	65	0.13	0.87	67.7	-	-	2.2
UR1_A	SPHERICAL	90	50	0.51	0.45	6.4	0.16	20	1
UR2	GAUSSIAN	100	65	0.13	0.87	67.7	-	-	2.2
UR4	SPHERICAL	120	90	0.1	0.9	90	-	-	1.95
UR5	SPHERICAL	110	60	0.087	0.91	60	-	-	2
UR6, UR7 UR8	SPHERICAL	120	90	0.01	0.99	90	-	-	1.95
M1, M2, M3	SPHERICAL	100	100	0.12	0.88	100	-	-	1.5
N1, N2, NV, NW	SPHERICAL	100	50	0.12	0.88	100	-	-	1.5
S1, S1_FW, S1_HW	SPHERICAL	100	50	0.12	0.88	100	-	-	1.5
SKL1	SPHERICAL	45	20	0.51	0.45	6.4	0.16	20	1
SKL2, 3, 4, 5, 6, 7, & 8	SPHERICAL	100	25	0.51	0.45	6.4	0.16	20	1
W1, W1_FW, W2, W2_FW, W3, W4	SPHERICAL	100	50	0.25	0.75	50	-	-	1

Table 14.10 Semivariogram Parameters for Thickness Vein Domains

Vein Set	Variogram Model	Plunge	Max Range	C0	C1	A1	C2	A2	ratio1
GRF1	SPHERICAL	100	25	0.38	0.86	20	-	-	1
H1, T1 T2	SPHERICAL	100	50	0.25	0.75	50	-	-	1
URW1, URW2	SPHERICAL	20	100	0.2	0.58	26	0.22	200	2
URW3	SPHERICAL	100	100	0.2	0.58	26	0.22	200	2
UR1	SPHERICAL	160	100	0.39	0.15	31	0.36	500	1
UR1_A	SPHERICAL	100	50	0.38	0.86	20	-	-	1
UR2	SPHERICAL	100	100	0.39	0.15	31	0.36	500	1
UR4	SPHERICAL	120	90	0.1	0.9	90	-	-	1.95
UR5	SPHERICAL	100	50	0.65	0.35	45	-	-	1
UR6, UR7 UR8	SPHERICAL	120	90	0.1	0.9	90	-	-	1.95
M1, M2, M3	SPHERICAL	100	60	0.1	0.9	60	-	-	2.1
N1, N2, NV, NW	SPHERICAL	100	60	0.1	0.9	60	-	-	2.1
S1, S1_FW, S1_HW	SPHERICAL	100	60	0.1	0.9	60	-	-	2.1
SKL1	SPHERICAL	45	25	0.38	0.86	20	-	-	1
SKL2, 3, 4, 5, 6, 7, 8	SPHERICAL	100	25	0.38	0.86	20	-	-	1
W1, W1_FW, W2, W2_FW, W3, W4	SPHERICAL	100	50	0.25	0.75	50	-	-	1

14.7 Grade Estimation

The drilling and channel data were examined using Surpac™ software package, using the MA proprietary Narrow Vein Modelling system.

The MA Proprietary System for Narrow Vein Modelling estimates the grades and true widths of veins. This is done in unfolded space using 5 m x and y grid spacing. The estimation area is extended beyond the outer data points by expansion of a fixed distance to create a boundary perimeter; the boundary is then smoothed with the result that the expansion is reduced to less than the target thickness at the extremities. The expansion distance is therefore a maximum, rather than a fixed value. The expansion for Tuvatu veins is detailed in Table 14.10. Thickness at the extension boundary is set to 0.2 m.

Grade estimations are made using 5 different methods so that the results can be compared; these are Nearest Neighbour (capped), Inverse Distance Squared (capped), Ordinary Krige (uncapped) and Ordinary Krige (capped) and metal content (gram-metres). True widths are estimated directly using Ordinary Kriging (no capping).
One block model was created, covering the entire project. The final 3D block model utilised 10 m cubic blocks sub-blocked down to 0.625 m cubes.

Table 14.10 Vein Expansion Distances

Vein name	Projection Plane	Extrapolation (m)	Vein name	Projection Plane	Extrapolation (m)
GRF1	NS	25	T2	NS	20
H1	EW	25	T2A	NS	20
M1	EW	30	UR1	NS	30
M2	EW	30	UR1_A	NS	30
M2A	EW	30	UR2	NS	30
M3	EW	30	UR4	NS	30
N1	EW	15	UR5	NS	30
N2	EW	15	UR6	EW	30
NV	Z	15	UR7	EW	30
NW	Z	15	UR8	EW	30
S1	EW	15	URW1	NS	25
S1_FW	EW	15	URW1A	NS	25
S1_HW	EW	15	URW2	NS	25
SKL1	EW	15	URW2A	NS	25
SKL2	Z	15	URW3	NS	25
SKL3	Z	15	URW3A	NS	25
SKL4	Z	15	W1	EW	20
SKL5	Z	15	W1_FW	EW	20
SKL6	Z	15	W2	EW	20
SKL7	Z	15	W2_FW	EW	20
SKL8	Z	15	W3	EW	20
T1	NS	25	W4	EW	20

14.7.1 Methodology

MA Proprietary System for Narrow Vein Modelling was used which consists of the following steps.

1) Database – validation of the drill-hole database. Selection of down-hole composites lengths, for each vein.

2) Intercept Selection. The drill hole data is displayed in section and elevation slices showing assays. Intercepts are selected and coded for each vein based on the following selection criteria, in priority order:

    a) Grade – select intervals with a value above cut-off, in this case 0.5 g/t Au. Also, internal waste of < 0.3 g/t Au intervals and/or geologically continuous intervals just below cut-off may be included.

    b) Continuity – waste (<0.5 g/t Au) values in the projected plane of continuity of a particular vein being modelled will be coded as that vein.

    c) No assays but a "vein" lithology code in the expected location

3) Basic Statistics and Upper Cuts. The basic statistics of the vein composites for each vein are then examined using basic statistics for grades, true width and gram.metres (metal). The mean, median, standard deviation and variance are calculated for both normal and log-transformed data. A cumulative probability plot is prepared for each data set in both normal and log-transformed formats. Breaks in the plot indicating more than one population are highlighted and their spatial position relative to the total data set examined in 3D space. If more than one population is considered possible, the total population is decomposed into its component populations and these are highlighted again in 3D space. If a small high-grade population is indicated, and this cannot be physically domained from the remainder, then an estimate with an upper cut will be included in the resource estimates.

4) Unfolding and Variography. The vein composites are unfolded into a single plane, such that NS striking veins are projected to the X axis, EW veins projected to the Y axis and flat veins projected to the Z axis. The original coordinates are stored in the model so the model may be refolded post estimation. Variography is then undertaken in this 2D space. Values for anisotropy and a variogram models are recorded for gold and thickness. Where no directional variograms are clearly determined (as commonly happens with less than 50 data points, or where the data is unevenly distributed) isotropic variograms were used or variograms from similar veins sets where utilised.

5) Unfolded Grid Model and Extension – Generates a model of the vein centre using the coded intercepts, and estimates grades, vein true widths and gram.metres. This is done in unfolded space using selectable x and y grid spacings. The estimation area is extended beyond the outer data points by expansion of a fixed distance (in this case 20 m) to create a boundary perimeter, the boundary is then smoothed with the result that the expansion is reduced to less than the target expansion at the extremities. The expansion distance is therefore a maximum, rather than a fixed value. In extreme cases, say where the extension is based on a single drill hole, no extension will occur at all. Expanded wireframes are checked in 3D space to ensure the expansion does not intersect waste drill holes. The thickness of this boundary is set to 0.2 m. This prevents an overflow of grade contours past the limits of estimation. The grade estimates are made using 5 different methods so that the results can be compared. These are Nearest Neighbour Capped, Inverse Distance Squared Capped, Ordinary Krige Uncapped and Ordinary Krige Upper Capped and gram.metre estimates. The true widths are estimated directly using Ordinary Kriging.

6) Minimum Width application and consequent Grade Dilution – Every 10 x 10 m block in unfolded space with a vein width (in the perpendicular direction to strike) less than 1.2 m is set to a width of 1.2 m. The grades for each block are then diluted according to the original width and waste grade (0.0 g/t), using the following formula:

$$Diluted\ Grade = \left[grade \times \left(\frac{true\ thickness}{minimum\ thickness}\right)\right] + \left[0\ g/t\ \times \left(\frac{dilution\ thickness}{minimum\ thickness}\right)\right]$$

Blocks with a width greater than 1.2 m have no change. This dilution will raise the tonnes and reduce the grade of the model; however, the total ounces of gold will remain about the same. The process of applying a minimum width is to reflect the minimum mining width and apply an appropriate dilution where veins are thinner than the mining width.

7) Refolding and True Width Correction – The grid is re-folded to its original 3D position. This is done by replacing the unfolded coordinates with the stored real coordinates. Some smoothing of the surface using surface modelling algorithms (not geostatistics) is undertaken; this removes local spikes and steps due to clustering of data. Changes are small, generally less than half the grid spacing. The "slope" of the surface in 3D space relative to the 2D surface is then measured as a percentage gradient; this value is recorded as it is similar to that used in "Connolly Diagrams" (Schwartz 1986). The True Width value is then corrected using this factor. Note that "slope" value is measured at each node of the grid and is a function of the surface geometry; the more the surface moves from the projection plane the greater the correction – in effect an "auto-correction". This is much better than using an average strike and dip for the surface (too general), a drill core measurement (too local) or geostatistics (too smoothed).

8) Solid Creation – The 3D centre plane of the vein is then converted to a closed 3D solid. Footwall and hanging wall surfaces are created by translating the 3D centre plane half the width of the vein to create footwall and hanging wall surface. These are then joined at the edge, which is a common boundary, to create a vein solid. If more than one vein is being estimated, then the interaction between the resultant solids is examined and portions of the minor veins removed via "clipping".

9) Block Model – The volumes from the final closed 3D solids are used to flag blocks in the final 3D block model for each vein. The variables from the solids, including grades, widths, slope, kriging variance, number of informing samples, nearest drill hole name and distances, etc., are all stored in the block model. Each vein block is given a vein name and number.

10) Bulk Density – The bulk densities for each block below the topographical surface are set to a constant value.

11) Missing Blocks – Blocks that are not present are flagged as air (above the original topography), pit (mined out in an open pit), stoped (removed by underground mining).

12) Mineral Resource categories – The resource categories are defined in long-section view for each vein, based on a combination of the number of informing samples, sample distances and kriging variance. The mineral resource categories are stored in the block model field.

13) Validation – The values within the block model are compared to the informing drill composites. Basic statistics for block model and drill composites are compared. Distributions of grades in space (by elevation and northing) are compared. Blocks nearest to drill holes are compared with the informing drill holes. The estimates using the different estimation methods are compared in total and above cut-off.

14) Reporting – The resource can be reported by resource category, by vein, by cut-off grades, by different methods (sensitivity to method and upper cuts), by elevation (tonnes per vertical m), by x and y dimensions.

14.7.2 Block Model

The Tuvatu block model (tuvatu_20141220_veins.mdl) uses regular shaped blocks measuring 10 m x 10 m x 10 m (Table 14.11). The choice of the block size was patterned with the trend and continuity of the mineralisation, taking into account the dominant drill pattern and size and orientation of the veins. The orientation of the block model is normal to the direction of the local grid. To accurately measure the volume of the mineralized wireframe inside each block, volume sub-blocking to 0.625 m x 0.625 m x 0.625 m was used. Blocks above the topography were tagged and excluded from the model estimation.

Table 14.11 Block Model Extents

Type	Y	X	Z
Minimum Coordinates	3920000	1875500	-250
Maximum Coordinates	3921305.6	1876901.6	508.4
User Block Size	10	10	10
Min. Block Size	0.625	0.625	0.625

14.7.3 Informing Samples and Search Parameters

Informing samples are composited across the vein, providing a local average across the vein width before estimation. Using average grades across a vein requires careful consideration of the number of informing samples used to prevent over smoothing of the estimate. A minimum of one vein composite and a maximum of eight vein composites were permitted to inform a block. The number of samples per vein composites depends on the thickness of the vein and the orientation of the drill hole to the vein. Search radii were found to be optimal at or near the distance that the variogram reached the sill. Thus the variogram ranges will be utilised in the maximum search distances (Table 14.12). The isotropy apparent in the variogram analysis is reflected in the search ellipse. Only one pass was used to inform the blocks.

Table 14.12 Search Parameters

Veins	Search Distance (Au)	2D Anisotropic ratio (Au)	Search Distance (m)	2D Anisotropic ratio (m)
GRF1	20	1	25	1
H1, T1 T2	50	1	50	1
URW1, URW2	85	1.75	100	2
URW3	85	1.75	100	2
UR1	65	2.2	100	1
UR1_A	50	1	50	1
UR2	65	2.2	100	1
UR4	90	1.95	90	1.95
UR5	60	2	50	1
UR6, UR7 UR8	90	1.95	90	1.95
M1, M2, M3	100	1.5	60	2.1
N1, N2, NV, NW	50	1.5	60	2.1
S1, S1_FW, S1_HW	50	1.5	60	2.1
SKL1	20	1	25	1
SKL2, 3,4,5,6,7.8	25	1	25	1
W1, W2, W2_FW, W1_FW, W3,W4	50	1	50	1

14.7.4 Discretisation

The Krige estimate used a 4 x 4 x 4 discretisation (XYZ), giving discretisation nodes spaced evenly within the block, the projection plane direction has no thickness (2D unfolded space) thus one discretisation point is applied.

14.7.5 Block Model Attributes

Interpreted mineralized veins were coded to the block model. Sufficient variables were added to allow grade estimation, resource classification and reporting (Table 14.13). Blocks above the original topography were coded as air and not estimated. Blocks that have been mined were flagged in the final block model; these blocks were estimated for reconciliation purposes. To simplify and reduce the size of the block model several attributes were removed from the final model. Final block model (tuvatu_20141220_veins.mdl) attributes are defined in Table 14.14.

Table 14.13 Block Model Attributes

Attribute Name	Type	Background	Description
dist_near	Real	0	distance to nearest sample
gold_f_gm	Real	0	derived from g.m
gold_ids	Real	0	IDS value for gold uncut, diluted for min thickness
gold_krig_capped	Real	0	krig value for gold top c according to vein, diluted for min thickness
gold_krig_uncapped	Real	0	krig value for gold uncapped, diluted for min thickness
gold_nn	Real	0	gold value of nearest sample, diluted for min thickness
gold_undiluted_uncapped	Real	0	original gold krig uncapped value undiluted by minimum thickness
hole_id	Character	UNDF	name of nearest drill hole
hole_length	Real	0	length of nearest sample downhole
krig_var	Real	0	kriging variance
minimum_thickness	Real	0	horizontal width across whole vein set at a minimum
num_samp	Integer	0	number informing samples
silver_krig_capped	Real	0	krig value for silver top c according to vein, diluted for min thickness
silver_nn	Real	0	gold value of nearest sample, nearest neighbour
silver_undiluted_uncapped	Real	0	original silver krig uncapped value undiluted by minimum thickness
slope	Real	0	percentage gradient of the vein in 3D, capped at 300%
true_width	Real	0	true width across whole vein
vein_name	Character	UNDF	name of vein
x_width	Real	0	horizontal width across whole vein in x direction

Table 14.14 Final Block Model Attributes (exported)

Attribute Name	Type	Background	Description
density	Real	2.57	bulk density
dist_near	Real	0	distance to nearest sample
gold_f_gm	Real	0	derived from g.m
gold_krig_capped	Real	0	krig value for gold top c according to vein, diluted for min thickness
gold_undiluted_uncapped	Real	0	original gold krig uncapped value undiluted by minimum thickness
hole_id	Character	UNDF	name of nearest drill hole
krig_var	Real	0	kriging variance
mined	Integer	0	0 insitu, 1 mined
minimum_thickness	Real	0	horizontal width across whole vein set at a minimum
num_samp	Integer	0	number informing samples
res_cat	Integer	3	1 measured, 2 indicated and 3 inferred
silver_undiluted_uncapped	Real	0	original silver krig uncapped value undiluted by minimum thickness
true_width	Real	0	true width across whole vein
vein_name	Character	UNDF	name of vein
x_width	Real	0	horizontal width across whole vein in x direction

14.7.6 Validation

Block models were validated by visual and statistical comparison of drill hole and block grades and through grade-tonnage analysis. Initial comparisons occurred visually on screen, using extracted composite samples and block models. Two of the major veins are presented in Figure 14.5 and Figure 14.6, and additional vein long section views are presented in Appendix 3.

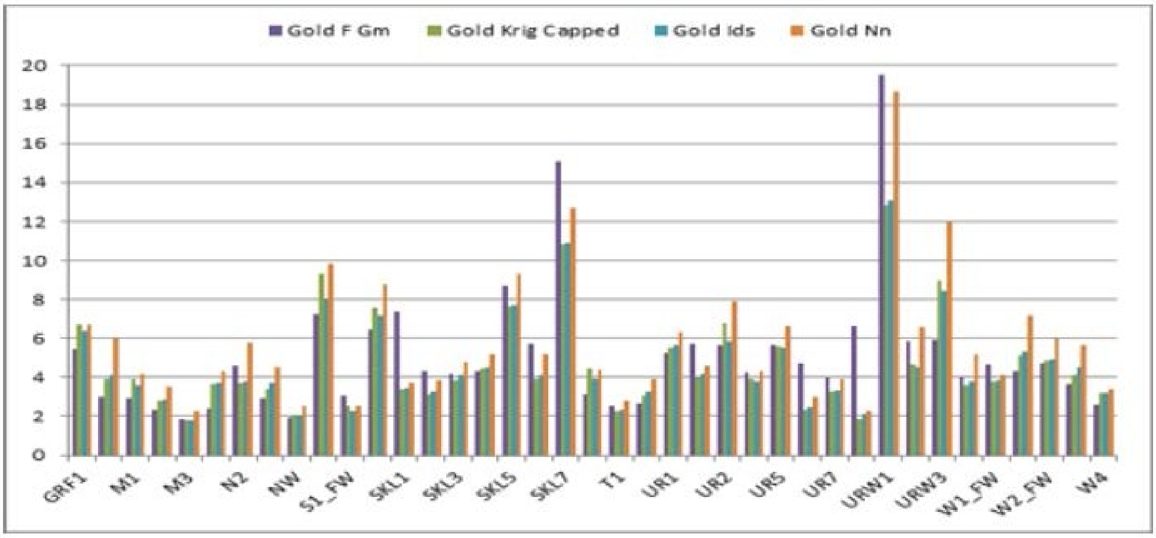
Alternative estimation methods (Table 14.15) were utilised to ensure the krige estimate was reporting a global bias, such as nearest neighbour and ID2 and back calculated grades from grams x m (g.m) estimates. The alternate estimates provided expected correlations. Nearest neighbour shows less tonnes and higher grade as it does not employ averaging techniques to assign the block grade. The ID2 estimate is closer to kriging as it does use averaging weighted by distance, but cannot assign anisotropy nor have the ability to decluster the input data. The gold grades back-calculated from g.m appeared over smoothed, a likely consequence of using the thickness variogram for both g.m and thickness. The ordinary krige estimate is the most reliable due to the ability of kriging to decluster data and weight the samples based on a variogram (which incorporates anisotropy).

Table 14.15 Alternate estimation results at nominated cut-offs

Cut-off grade g/t Au	Ordinary Kriging		Using g.m to back calculate grade		Inverse distance squared		Nearest Neighbour	
	Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Mt	Au g/t
3.0	2.43	9.5	3.06	9.4	2.48	9.2	1.97	12.1

The alternate estimation methods are presented by vein Figure 14.3. Note SKL7, URW1 with extreme outliers significantly affects the ID2 and NN estimates quite dramatically. SKL7 (11 comps, 172.9 g/t max) and URW1 (65 comps, 285.5 g/t max) have extreme outliers relative to the general grade of the veins.

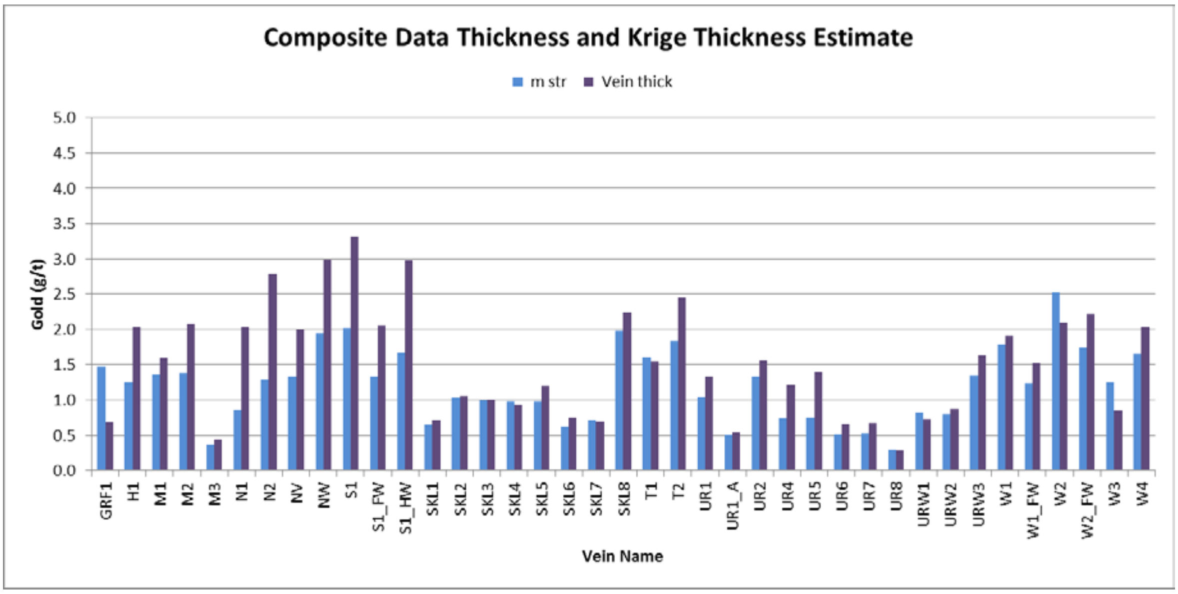
Figure 14.3 Veins Grades by gram.metre (purple), Krig Estimated Grade (green), Inverse Distance Squared (blue) and Nearest Neighbour (orange)



Vein thickness estimates are particularly troublesome for veins sets not striking north-south, east-west or flat, such as the Nasivi vein and Snake vein sets (“flatmakes”). These vein sets are oblique and shallow to moderately dipping, and informed by very few intercepts, one of which is usually thicker and high grade. These veins are over estimated in thickness and represent a small fraction of the overall tonnes. Thickness capping could have been applied to these vein sets to limit overestimation, but the low number of samples made assessment of caps difficult, the estimated grades within these shallower dipping veins appears reasonable.

Reflecting the uncertainty in estimation, Nasivi veins are classified as inferred. T2 and H1 are oblique veins but are steeply dipping thus the vein thickness is better estimated, but do show some overestimation. Using rotated block models would be a solution to the problem of calculating true thickness, however a compromise of block model orientation had to be reached as one un-rotated model is required for mining optimisation.

Figure 14.4 Vein thickness, drill data (blue) estimated (purple)



MA recommends further work (drilling) targeting oblique veins, such as Nasivi, Snake and Tuvatu lodes. At the completion of drilling the significance of the vein size should be assessed. If these veins become sufficiently large, a rotated block model should be considered to better enable the estimation of thickness, or a cap should be applied to the thickness estimate.

MA highlights a high grade shoot defined in URW1 (Figure 14.6). There is evidence for shoots within this lode but not as high grade as the two intercepts 100 g/t and 252 g/t Au might suggest, the supporting intercepts, (19.78 and 81.90 g/t Au) provide guidance on the size and potential grade. This vein is only 0.73 m wide on average, thus dilution will be a major factor in the final grade. Currently a minimum mining width of 1.2 m with 0 g/t Au dilution has been applied to the model. Only the top of this high grade shoot is classified as Indicated.

Figure 14.5 UR2 Estimated Gold (long section view)

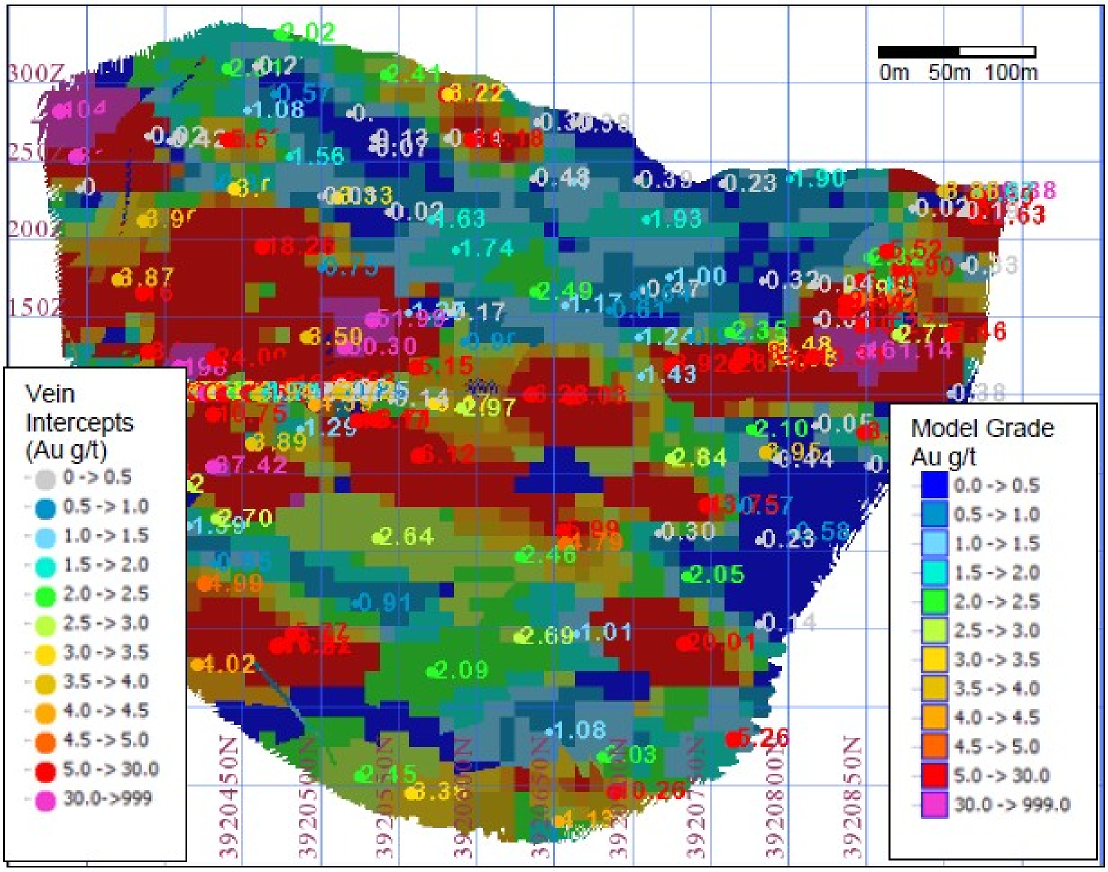
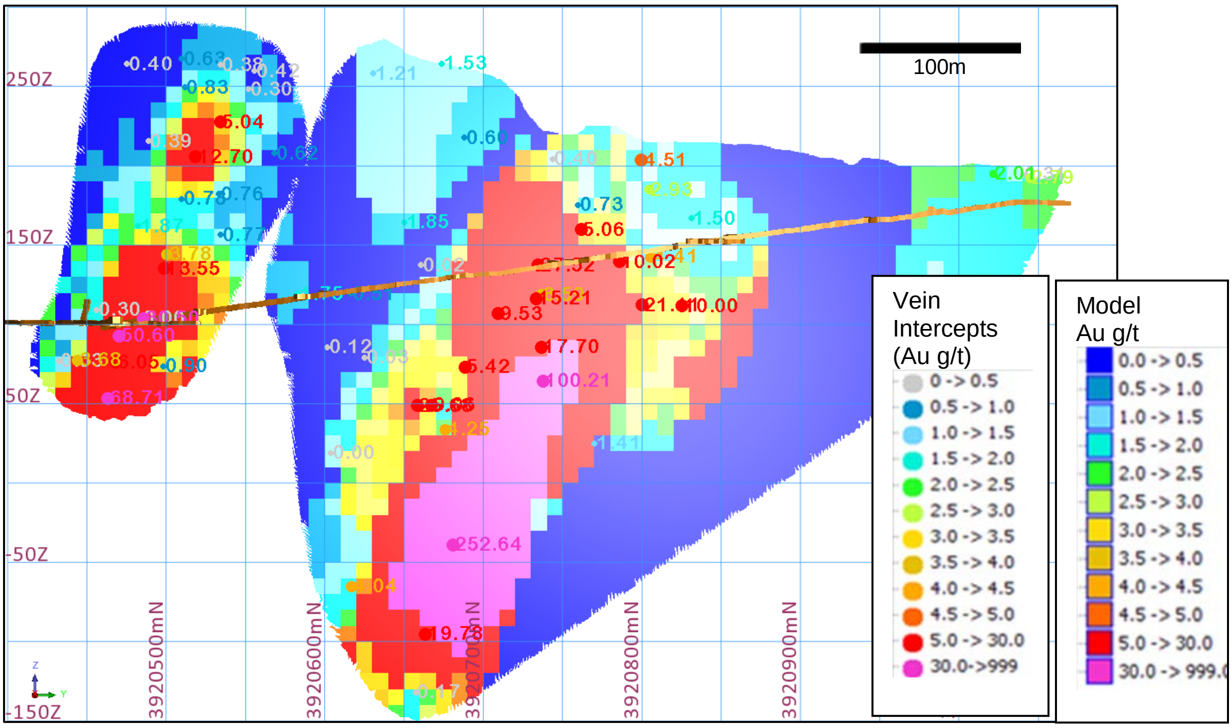


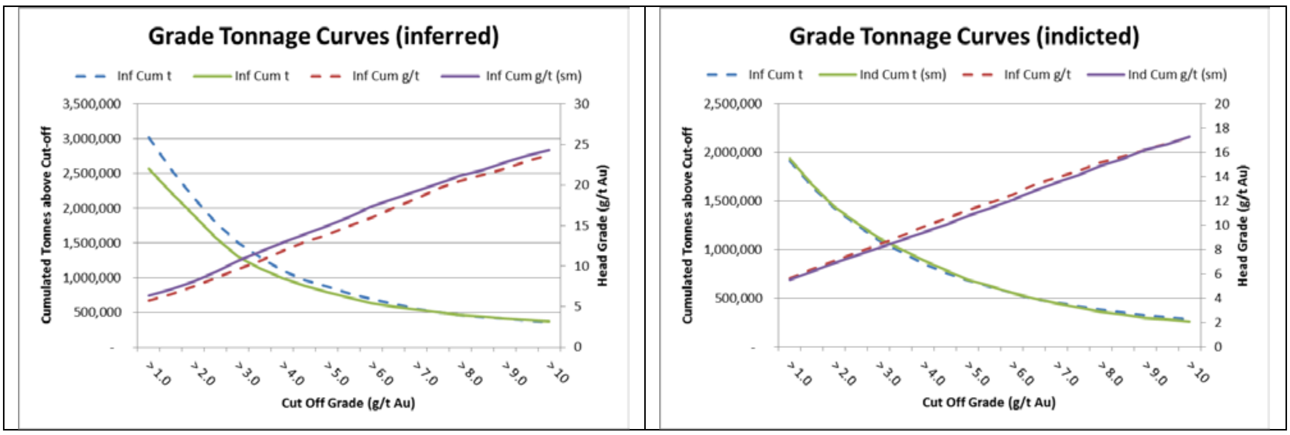
Figure 14.6 URW1 and URW1a Estimated Gold (long section view)



14.8 Economic Cut-Off Parameters

Resources have been reported above a 3 g/t Au cut-off assuming potential underground mining of veins with narrow widths from 1 m to 8 m. The assumed mining method is Shrinkage stoping or hand held miners: both methods are selective mining methods ideal for high grade, steeply dipping narrow deposits. Selective mining will maximize recovery and minimize dilution. An advantage of a shrinkage operation is that no back fill is required. The average stope parameters are 60 m long x 60 m tall x vein thickness. Assumed required head grade is 5.0g/t Au and as is shown in Figure 14.7 the average head grade above a 3g/t cut-off is 8.17 g/t in the indicated resource. It is assumed low grade ore (<5g/t) mined during development will be stock piled.

Figure 14.7 Tuvatu Grade Tonnage Charts compared to May 2014 resource (dotted lines)



The grade tonnage chart Figure 14.7 indicates the global indicated and inferred tonnes and grade above a cut off will provide a head grade of approximately 5 g/t Au.

The global resource reported above various cut off is presented in Table 14.17, at the higher cut off for all material of 3 g/t Au there is an indicated resource of 1,120,000 tonnes at 8.17 g/t Au for 294,000oz and an inferred resource of 1,300,000 tonnes at 10.6 g/t Au for 445,000 ounces of Au.

Table 14.17 Tuvatu Resource Reported at various cut-offs

Cut-Off	Indicated			Inferred		
g/t	material (t)	Au (g/t)	Au (oz)	material (t)	Au (g/t)	Au (oz)
1.0	1,933,000	5.52	342,900	2,569,000	6.4	524,500
2.0	1,442,000	6.90	319,700	1,903,000	8.1	492,900
3.0	**1,120,000**	**8.17**	**294,000**	**1,300,000**	**10.6**	**445,000**
5.0	689,000	10.80	239,300	793,000	15.0	382,100

The reporting of tonnages and grade figures reflects the relative uncertainty of the estimate, and due to rounding to appropriate significant figures , some discrepancy in the addition of rounded figures may occur.

14.9 Assumptions For 'Reasonable Prospects for Eventual Economic Extraction'

Assumptions for reasonable prospects for eventual economic extraction applied to this deposit include but may not be limited to the following:

- Gold pricing at US$1324.75, 12mth average to April 2014 (Kitco.com)
- Assumed open pit mining costs of $1.60 per tonne;
- Assumed underground mining costs of $40.00 per tonne;
- Assumed processing costs of $20.00 per tonne;

Mineralised veins come to surface providing potential for open pit extraction. This style of mineralisation typically only lends itself to small open pits, thus potential open pit material is limited to within 75 m of the surface and a 1 g/t Au cut off is applied.

Material below 75 m of the surface is considered amenable for underground mining, either hand held or shrinkage mining. Underground mining costs are higher and require a higher cut-off grade. A cut-off of 3 g/t Au is considered reasonable based on similar small scale underground operations.

14.10 Bulk Density

A total of 1955 bulk density measurements were reported from drill hole core at Tuvatu, with an average reported bulk density of 2.61 t/m^3 (Table 14.18). The statistical average of the bulk density measurements is assigned to all lithologies for this mineral resource estimate. This is lighter than the bulk density used in the 1998 NI43-101 (A-Izzeddin 1998) of 2.83 t/m^3 based on 171 samples.

Table 14.18 Bulk Density Statistics

Basic Statistics	All Density Readings	Vein Density (subset)
	t/m3	t/m3
Count	1,955	181
Minimum	1.86	2.07
Maximum	3.22	2.96
Average	2.61	2.57
Median	2.63	2.60
Mode	2.67	2.64
Standard Deviation	0.18	0.18

Bulk density data is stored in the drill hole database with a rock type code associated with each reading (Table 14.17), the majority of material is logged as either monzonite (MZ) or medium grained monzonite (MMZ), each reporting average densities of 2.61 t/m^3 and 2.62 t/m^{3}respectively. Mineralised samples are likely to be from vein breccia (VBX) or unaltered veins (UV) with a density of 2.58 t/m^3 and 2.50 t/m^3 respectively.

Table 14.19 Bulk Density by Rock Type

Rock Code	Count	Average t/m3
AN	224	2.63
AP	1	2.28
BL	32	2.58
CV	1	2.19
MMZ	906	2.62
MZ	687	2.61
PG	1	2.76
TF	1	2.49
UV	54	2.50
VBX	46	2.58
"Null"	2	2.72
Total	**1,955**	**2.61**

Density values for mineralization were extracted from the drill hole database and provided 181 samples within defined mineralization. The average of mineralised samples used in definition of wireframe interpretations is 2.57 t/m^3 (Table 14.16). This is a similar result to the average of 46 VBX and 54 UV samples. The bulk density assigned to mineralisation is 2.57 t/m^3.

Waste material is assigned the average bulk density of monzonite (2.61 t/m^3). Note: the average density of all measurements at Tuvatu is 2.61 t/m^3.

14.11 Moisture

No measurements were recorded; all reported tonnes are dry metric tonnes.

14.12 Mining and Metallurgical Factors

No mining factors have been applied to the in situ grade estimates for mining dilution or loss as a result of grade control or mining process. No metallurgical factors have been applied to the in situ grade estimates.

14.13 Resource Estimate and Classification

Based on the study herein reported, delineated mineralization of the Tuvatu Resource is classified as a resource according to the definitions from CIM Definition Standards (2010):

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

A breakdown of the Tuvatu Project resource estimate by resource category is provided in Table 14.20.

Table 14.20 2014 Tuvatu Resource Estimates

Resource Category	Indicated			Inferred		
Metrics	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
	t	Au g/t	Au oz	t	Au g/t	Au oz
3 g/t Au	1,120,000	8.17	294,000	1,300,000	10.6	445,000

For the classification of Mineral Resources for the Project, the following definitions were adopted and applied to each domain separately.

14.13.1 Measured Mineral Resource

No measured mineral resources are defined at Tuvatu. The underground development sampling shows highly variable grades over short (3 m) distances, indicating that local estimation of grades will be difficult.

14.13.2 Indicated Mineral Resource

Defined as those portions of the deposit for which grade, quantity and densities can be estimated with confidence sufficient to allow the appropriate applicator of technical and economic parameters to support mine planning and evaluation of economic viability. The indicated portions of the resource are based on detailed geological information gathered from surface and underground drilling, underground workings and mapping. The prescribed drill spacing required was 25 m x 25 m and can demonstrate a high level of confidence in the geological continuity of the mineralization. Estimation statistics were used to guide the decision, Krige variances of block within the indicated category fall within the range of 0.15 to 0.4 and must not exceed 0.5. A few higher variance blocks may be included if a structural trend is present. The majority of blocks must have a sample location within 40 m, the average distance to nearest samples for all indicated blocks is 20 m. Blocks are in informed by minimum of 6 vein composites.

14.13.3 Inferred Mineral Resource

Defined as those portions of the deposit which quantity and grade can be estimated on the basis of geological evidence and limited sampling, providing reasonably assumed continuity of quantity and grade. The estimates are based on geological evidence gathered from drill holes. The estimate inferred resource is defined with a drill spacing of greater than 25 m x 25 m. The inferred portions of the deposit are sampled with a fewer number of intersections but demonstrating a reasonable level of geological confidence. Inferred resources have an average distance to the nearest sample of 30 m and are informed by an average of 4 vein intercepts.

14.14 Comparison with Previous Estimates

In May 2014, MA was commissioned to review the geology and create a resource estimate for Lion One. The resource was estimated for each vein individually using Ordinary Kriging of width and grade, the latter using accumulations, into a 3D block model.

A total of 39 different lode structures were identified in the resource area including 11 lodes in the Upper Ridges area, 3 lodes in the Murau area, 4 lodes in the West area, 2 lodes in the Tuvatu area and stockwork veins in the SKL area. A minimum of 5 intercepts were used for a vein to be defined with a number of other lodes identified but not included in the resource estimate.

The Tuvatu May 2014 resource was reported at two cut offs representing resources amenable to open pit and underground production. The total indicated resource was 1,448,000 tonnes at 6.9 g/t Au for 320,000 ounces of gold and an inferred resource of 1,791,000 tonnes at 8.4 g/t Au for 485,000 ounces of gold.

Table 14.21 2014 Tuvatu Resource Estimate

		Resource Category					
		Indicated			Inferred		
		Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
Cut off	Selection Criteria	t	Au g/t	Au oz	t	Au g/t	Au oz
1 g/t Au	Surface Mining Potential - Within 75m of surface	609,000	3.5	69,000	513,000	4.2	69,000
3 g/t Au	Underground Mining Potential - Below 75m of surface	839,000	9.3	251,000	1,278,000	10.1	416,000
Total		**1,448,000**	**6.9**	**320,000**	**1,791,000**	**8.4**	**485,000**

Significant differences between the May 2014 and January 2015 resources include reducing the extrapolation of veins in densely drilled areas, excluding the western veins and splitting several veins where vein width and grade well below cut off. As shown in the comparative grade-tonnage charts in figure 14.7, the main impact at the 3 g/t Au cut-off is a decrease in tonnes and increase in grade of inferred resources, with very little change in tonnes or grade of inferred resources.

14.15 Discussion on Factors Potentially Affecting Materiality of Resources and Reserves

The following factors could potentially impact on the materiality of the mineral resource estimate:

- Infill drilling will tighten control on interpolated thicknesses of the veins;
- Distal proportions of the resource are extrapolated using parameters defined in

MA's Propriety Narrow Vein Modelling software. These projections were inspected by Lion One and MA and while there is limited geological evidence for the extrapolation, it is reasonable to expect that the veins continue. Further extension drilling is required to support the extension of the inferred continuity of the veins.

- Local faulting is identified at the Tuvatu project and these structures are considered to have minor off-sets;

- An assumed mining width of 1.2 m has been applied to the resource; mining dilution has been incorporated at a 0 g/t grade. The final mining width achieved with have an impact on the narrow veined mineralisation tonnes and grade.

- Underground development will provide close spaced channel samples and it is likely these samples will show highly variable grades within the veins as seen in the historical underground channel samples

MA notes Lion One has effectively complied with the requirements of the Department of Environment, (DE) Native Land Trust Board (NLTB) and the Mines and Mineral Resources Department (MRD)

- Lion One has completed an Environmental Impact Assessment (EIA) and has obtained approval by the Department of Environment in Fiji. The DE has recommended mining to the MRD.

- Lion One has finalised negotiations with the NLTB, who represent the landowners regarding the surface licence and they have also notified MRD.

- The proposed grant of the mining licence has been advertised in the papers for the statutory period of time and there were no objections.

- All community and stakeholder negotiations have reportedly gone well with community and stakeholder's supporting mining at Tuvatu.

- The EIA and the surface license were pre requisites to the grant of the mining license.
- The social implications were addressed in the EIA and were satisfactory to all stakeholders. Lion One is currently negotiating with the government regarding royalties etc. as some companies have received reductions in various payments and taxes in the past to start up operations.

14.16 Mineral Resource Estimate Statement

The categorised Mineral Resources for the Tuvatu Project have been classified as indicated and inferred confidence categories on a spatial, areal and zone basis and are listed in Table 14.22.

Table 14.22 Mineral Resources of the Tuvatu Gold Project (Jan 2015)

Resource Category	Indicated			Inferred		
Metrics	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
	t	Au g/t	Au oz	t	Au g/t	Au oz
3 g/t Au	1,120,000	8.17	294,000	1,300,000	10.6	445,000

14.16.1 Notes to Accompany Resource Statement

- The Tuvatu project comprises three ‘Special Prospecting Licenses (SPL1283, 1296, 1465) which have total area of 10,565 ha, and for which Lion One has a 100% interest. The Tuvatu deposit itself is situated on SPL1283.

- MA was provided with an export of Lion One’s current drill hole database in MS Access format.

- Ian Taylor (AusIMM(CP)) of Mining Associates visited the property in February of 2014.
- Field exposures and numerous drill holes were examined during this visit, and an assessment was made of the procedures for logging, sample preparation, quality control and SG measurement.

- Two independent samples were collected, (drill core and out crop) both returned expected gold values.

- The Tuvatu deposit consists of a number of zones of low sulphidation epithermal quartz veins and spatially associated stockworks. Most of the main veins are exposed and therefore have a well-understood geometry. The veins show minor variability in orientation both along strike and down dip.

- The majority of the mineralisation lies within two zones of veining; The Upper Ridges zone striking NS, which has extends of 700 m, and the Murau corridor striking EW, has extents of 400 m. Additional to these areas is the western lodes, striking EW for 200m. At the intersection of the Upper Ridges and Murau lodes occur the SKL veins which are smaller shallow southerly dipping veins.

- Estimation undertaken in Surpac, using ordinary kriging. (inverse distance squared and Nearest neighbour and gram m estimates were used as validation techniques).

- Kriging of 10 m x 10 m x 1 m blocks (2D grid). The 3D Block Model uses a 10 m x 10 m x 10 m block which considers vein orientations and drill pattern. (Approximately 1/3 in the drill spacing, in well drilled areas). Sub-blocking of 0.625 m x 0.625 m x 0.625 m approximating the selective mining unit. Cubic blocks were required to accommodate all vein orientations sufficiently. Ore loss and dilution have been applied to the vein; a minimum mining width of 1.2 m is applied (@ 0 g/t Au).

- Experimental Variograms were generated in Surpac. Nuggets were generally moderate to low, ranging from 0.1 to 0.51, and the range of the variogram ranged from 20 m to 100 m.

Geometric Anisotropy was adopted in the plane of the vein and ellipsoid ratios applied to reflect directional variograms.

- Estimation parameters: Veins used a max of 12 samples; minimum sample number was set to 1. Search distances reflect variogram ranges - 25 to 100 m.

- Average dry bulk density is 2.61 t/m^3 for all rocks and within interpreted veins the average dry bulk density is 2.57 t/m^3. Tonnages are based on dry tonnes. No moisture readings have been recorded.

- No other variables were considered in this resource estimate.

- Vein wireframes were constructed empirically from drill hole intercepts greater than at
- 0.5 g/t Au. Wireframes were generated from the mid-point of the drill hole intercepts, smoothed and gridded. Gold, vein thickness and gram m were estimated in 2D space, resultant grids were re-folded and expanded to the thickness of the vein. The resulting wireframe solids were used to constrain the individual veins in 3D space.

- High grade outliers within the vein composite data were capped. Veins with greater than 34 intercepts were individually asses for outliers; grade caps were applied as appropriate and ranged from the 97.5 percentile to the 99 percentile. Veins with less than 34 intercepts were capped at the 97.5 percentile. m (thickness) were not capped. (gram m were capped).

- Global mean grades for estimated blocks and drill hole samples compared well.

- Ordinary krige estimates were compared to nearest neighbour and inverse distance and gram m estimates, to assess the impact of data clustering and semivariograms.

- No reconciliation data is available for the Tuvatu project as no production records are preserved.

15.0 MINERALS RESERVES ESTIMATE

In accordance with NI 43-101, no Mineral Reserves were reported for this Preliminary Economic Assessment (PEA).

16.0 MINING

16.1 Introduction

The reader is cautioned that the mining study is part of a PEA that is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

The Tuvatu gold mineralization is dominantly hosted in monzonite units, but also in adjacent volcanics. The deposit comprises a series of narrow, epithermal veins over a strike length of about 600 m north-south and about 500 m east-west. The deposit is open at depth but extends at least 540 m vertically. The veins are predominantly sub-vertical north/south trending, although north-east/south-west and east/west trending sub-vertical and north/south and east/west trending sub-horizontal veins also occur. Veins range from 0.04 m to 8.0 m wide, with a mean width of 2.2 m.

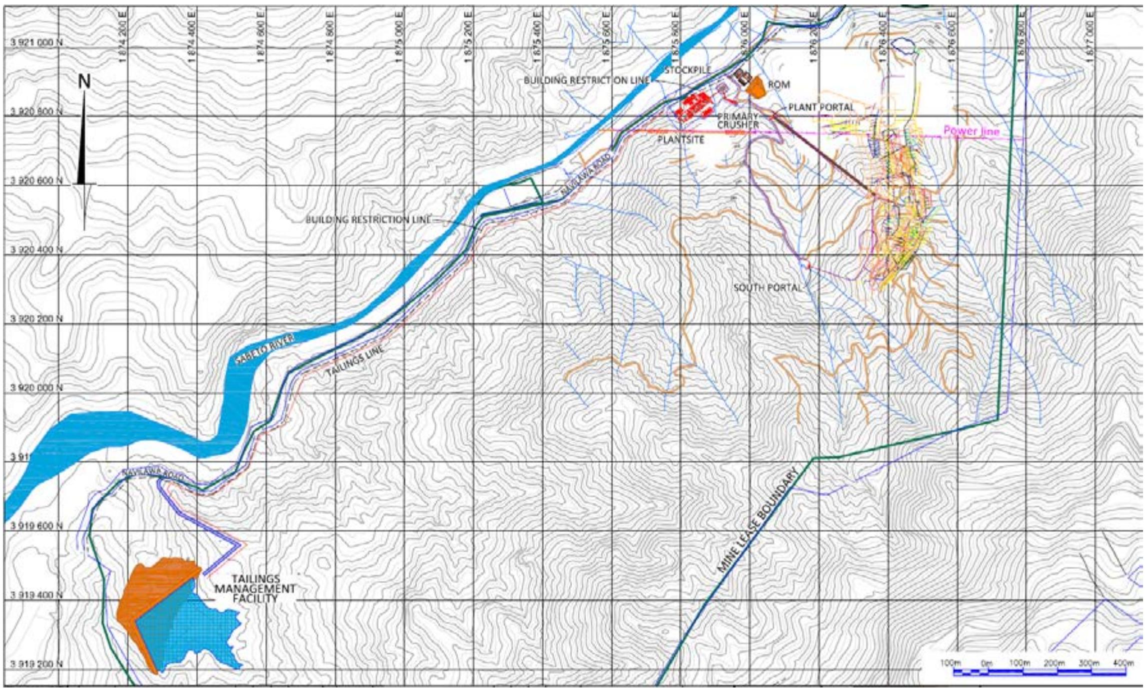
A total of 39 veins are defined in the resource model. The AMC conceptual mine design uses both Indicated and Inferred Resources. There are no Measured Resources. Inferred Resources comprise 55% of the projected mined tonnes and 63% of the projected contained ounces in the mine plan developed in this Study.

Mine planning indicates that optimal mining will be by narrow vein underground mining. This option is considered the most efficient for the narrow, steeply dipping nature of the mineralization and high overall average grade of the deposit, promising the optimum cost per ounce of gold mined and maximum extraction of the contained Mineral Resource ounces. Proposed mining methods will be exclusively conventional (air-leg) methods.

Two main declines from surface connected to two internal declines have been envisaged, with dimensions of 4.5 mW x 4.5 mH at a 15% (approximately 1:7) gradient, to a maximum depth of approximately 500 m below the surface (mbs).

All currencies are expressed in US dollars (US$) unless otherwise indicated.
A general layout of the project is shown on Figure 16.1.

Figure 16.1 General mine layout



16.2 Geotechnical

The geotechnical study involved data processing from 21 resource drill-holes followed by an empirical analysis for underground designs

.

No site visit was undertaken by the mine geotechnical engineer, and no dedicated mine geotechnical holes were drilled.

Anticipated ground conditions can be described as "fair" to "very good". The majority of ground can be characterized as "good".

Very limited information is available on hydrogeology. The rock mass itself, particularly in the fresh rock exposed in the Upper Ridges zone, is impermeable and dry. Significant water inflows are reported from a major fault, the Coreshed Fault, located on the northern end of the mineralization.

The standard stoping panel size is based on a 60 m level interval and 60 m strike length. Sill pillars between the stope panels should be at least 6 m high.

The recommended standard ground support for development drives in good ground is friction bolts installed on a regular pattern. Areas where ground conditions are blocky will require surface support. Welded wire sheet mesh is recommended as the standard surface support in these areas. Larger spans are formed at the intersection of development drives. In these areas deeper anchorage for ground reinforcement using 6 m cable bolts is recommended.

Developing through and in the vicinity of the Coreshed Fault is expected to be challenging and require heavy ground support.

AMC has identified the presence of smectite (swelling clay) associated with some mineralization. If present in sufficient quantities this may adversely affect the broken material draw in shrinkage stoping panels. As the broken material forms the working platform for stoping activities, the presence of clay could lead to 'false floors' and pose a significant safety risk to personnel. AMC strongly advises that additional analysis be undertaken to understand the true extent, occurrences and impact of the swelling clay.

16.3 Underground Mining

16.3.1 Mining method

A number of mining methods were considered for the potential underground operations at Tuvatu, including mechanized open stoping. AMC notes that mechanized open stoping can facilitate a high production rate at relatively low operating cost, and is a non-entry method where personnel do not enter the stope. However, due to the availability of a low cost and experienced local air-leg mining workforce, air-leg mining methods were ultimately decided upon for the Study. Unlike the nearby Vatukoula deposit, most of the veins at Tuvatu are steep-dipping in nature. Some flat-dipping veins are present, but account for less than 5% of the total potential mining areas. A method similar to that employed at Vatukoula is proposed for these areas, making use of air-leg mining and recovering mined material via winches.

Two methods are envisaged to be employed:

- Shrinkage stoping (air-leg method for steep dipping lodes).
- Breast stoping (air-leg method for flat dipping lodes).

The factors influencing the choice of mining method included:

- Narrow mineralization width, ranging from 1.2 m to 8 m, with an average of approximately 2.2 m.
- Mostly steep dipping, but with some flat dipping lodes.
- Moderate to high grades of mineralization.
- Lodes are visually difficult to identify.
- Veins undulate on dip and strike.
- Good geotechnical conditions.
- Interaction of mining methods within underground mining areas.
- Local experience in Fiji.

16.3.2 Methods description

16.3.2.1 Shrinkage stoping

Shrinkage stoping is used in narrow and steeply dipping lodes located in competent ground conditions that do not require backfill.

The material to be mined is taken in successive horizontal slices along the entire stope length (similar to cut-and-fill). A portion of the blasted material (swell) is removed from the stope through a series of draw-points located at the bottom of the stope. Enough broken material is left in the stope to provide a working floor from which the next cut is taken. The broken material also provides support for the stope walls. In order to avoid overdraw of the swell, adequate planning and coordination is required. This is a bottom-up mining method wherein the stope is mined to the level above. Once the level is reached, the remainder of the broken material is then extracted from the stope leaving behind an empty void.

Man and material access into the stope is from pre-developed raises. Production drilling is performed using air-legs (jacklegs). A typical layout for this method is shown in
Figure 16.2. During the mining cycle only 30% to 35% of the blasted material is extracted, equivalent to the broken swell factor. When mining to the top horizon is completed, the

remaining broken material is extracted. The mined-out void may or may not be filled with waste rock.

For Tuvatu, an in-situ sill pillar of 6 m wide is envisaged as being left at the top of each 60 m lift to protect the drive above the shrinkage stope, as well as provide regional stability to the stoping area along dip.

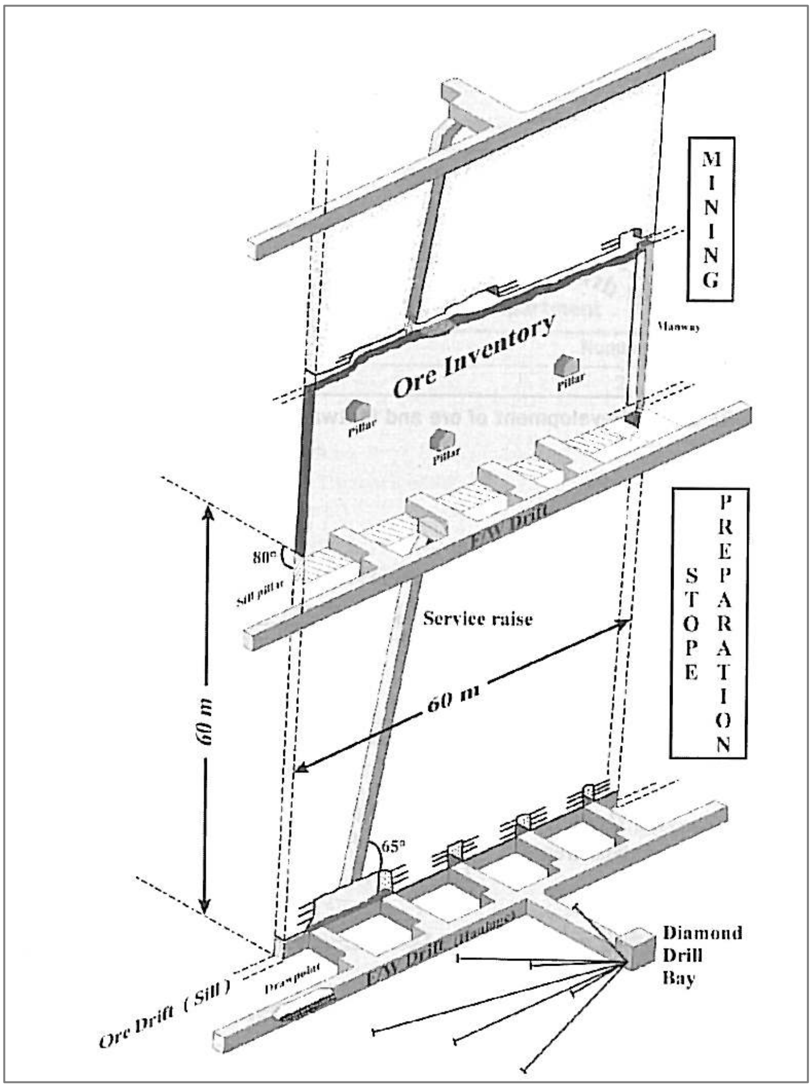
The advantages of shrinkage stoping include:

- Reduced sublevel development.
- Narrow mining widths and low levels of dilution are possible.
- Broken material reports to extraction draw-points.
- Hangingwall supported by broken material and can be bolted if required.

The disadvantages of shrinkage stoping include:

- Significant waste draw-point development.
- Potential operator safety concern due to exposure in the stope.
- Potential operator safety concern due to working off broken material.
- Potentially poor roof stability in blocky ground.
- Potential hang-ups during draw-down if mineralized material is reactive or contains fines.
- Potential hangingwall failures during draw-down.
- Low production rates during stoping.
- Possible material loss (reef-in-hangingwall or reef-in-footwall) if the mineralization is not easily identifiable.

Figure 16.2 Generic shrinkage stope layout



16.3.2.2 Breast stoping

This method is suitable for shallow dipping lodes of uniform narrow width and large horizontal extent, usually represented in coal seams, potash layers or conglomerate gold reefs mined in South Africa. The method has been in use at Vatukoula since 1991 (Gordon and Harmse, 1993). Mineralized material is extracted via a number of panels mined along strike from an initial raise line position. Drilling is done using handheld rock drills. The blasted material is mucked using scrapers to slusher gullies and eventually to ore-passes or a muck-bay. From there the mineralized material is hauled out of the mine or to shaft stations and hoisted out

.

A typical layout for this method is indicated in Figure 16.3. The extraction ratio for a typical shallow-dipping stoping block is estimated as 80%, but lower at Tuvatu due to the interaction of flat-dipping lodes with steeply-dipping ones. Where steeply-dipping lodes intersect flat-dipping ones, the stoping sequence allows the flat-dipping air-leg stoping to complete before the vertical stopes are extracted. The extraction ratio of flat-dipping stopes was reduced in order to simulate the leaving of in situ pillars between these stopes and the long-hole stopes. The breast stoping cleaning layout is indicated in Figure 16.4.

The mining sequence for this method is:

- Mine the underground accesses (declines and inclines) at 4.5 m high (H) x 4.5 m wide (W) to access a stoping area.
- Develop access drives at 4.0 mH x 3.5 mW.
- Develop muck-bays 12 m long at 4.0 mH x 4.0 mW.
- Develop raises at 2.4 mH x 1.5 mW up-dip to exploit the mineralization extents.
- Mine gullies along strike 20 m apart on dip, at 2.1 mH x 1.5 mW to form stoping panels.
- The stoping panels between adjacent gullies are mined along strike, with the face of the lowermost panel generally leading those located above it.
- The breast stopes in the Tuvatu footprint will mine in one direction only to a maximum distance of 100 m from the rise position due to the strike winch capability.

Figure 16.3 Generic breast stoping layout

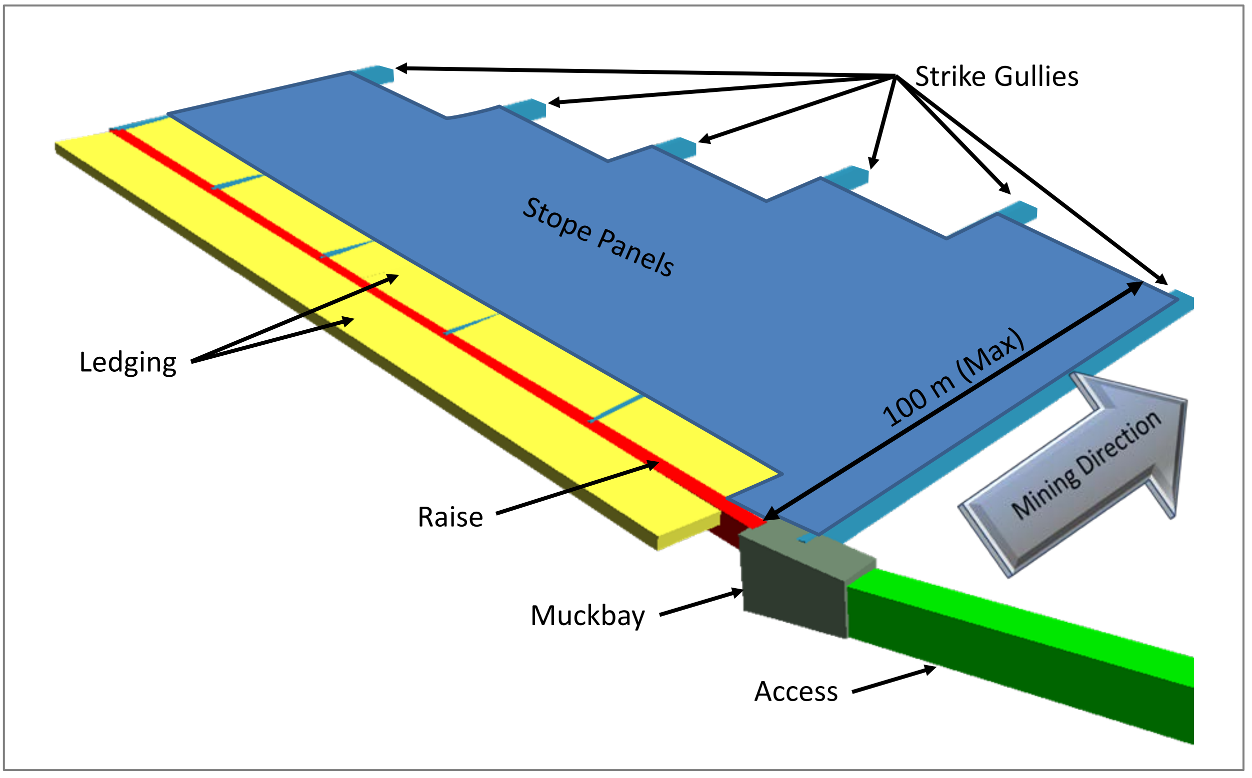
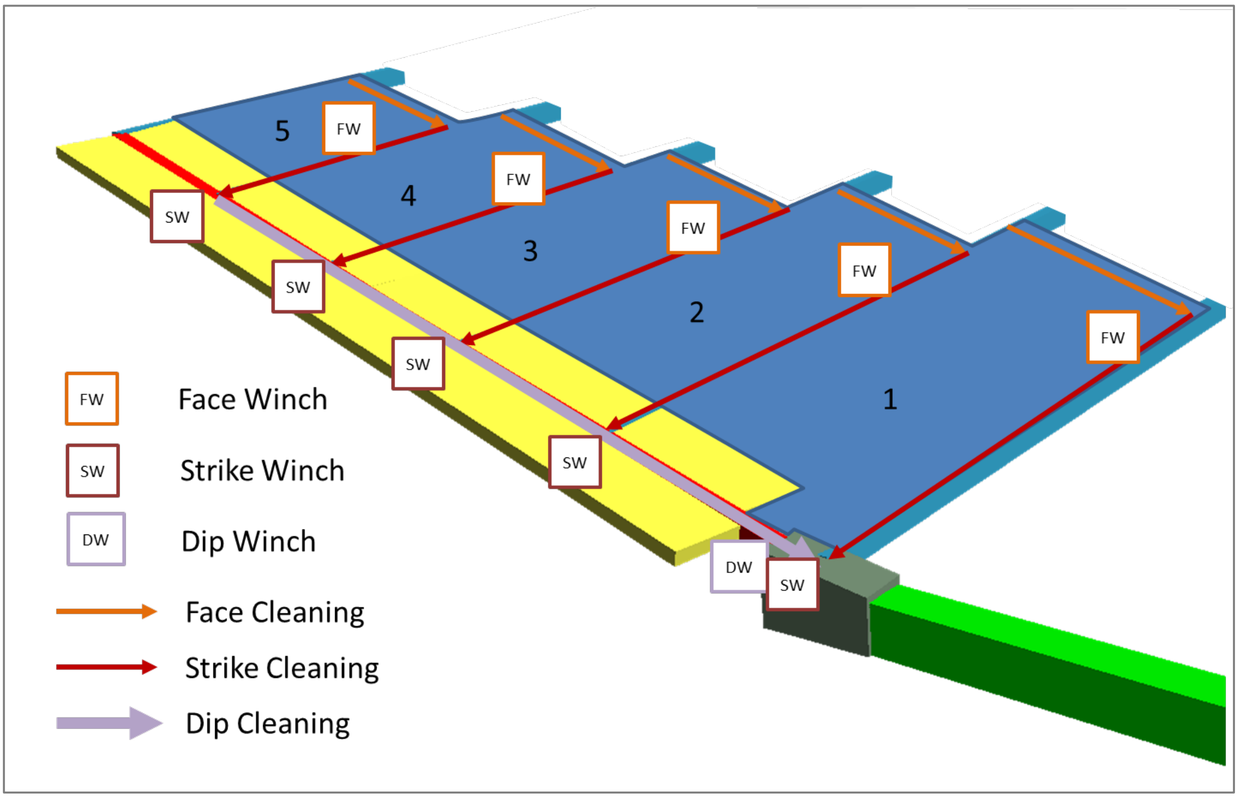


Figure 16.4 Breast stoping cleaning layout

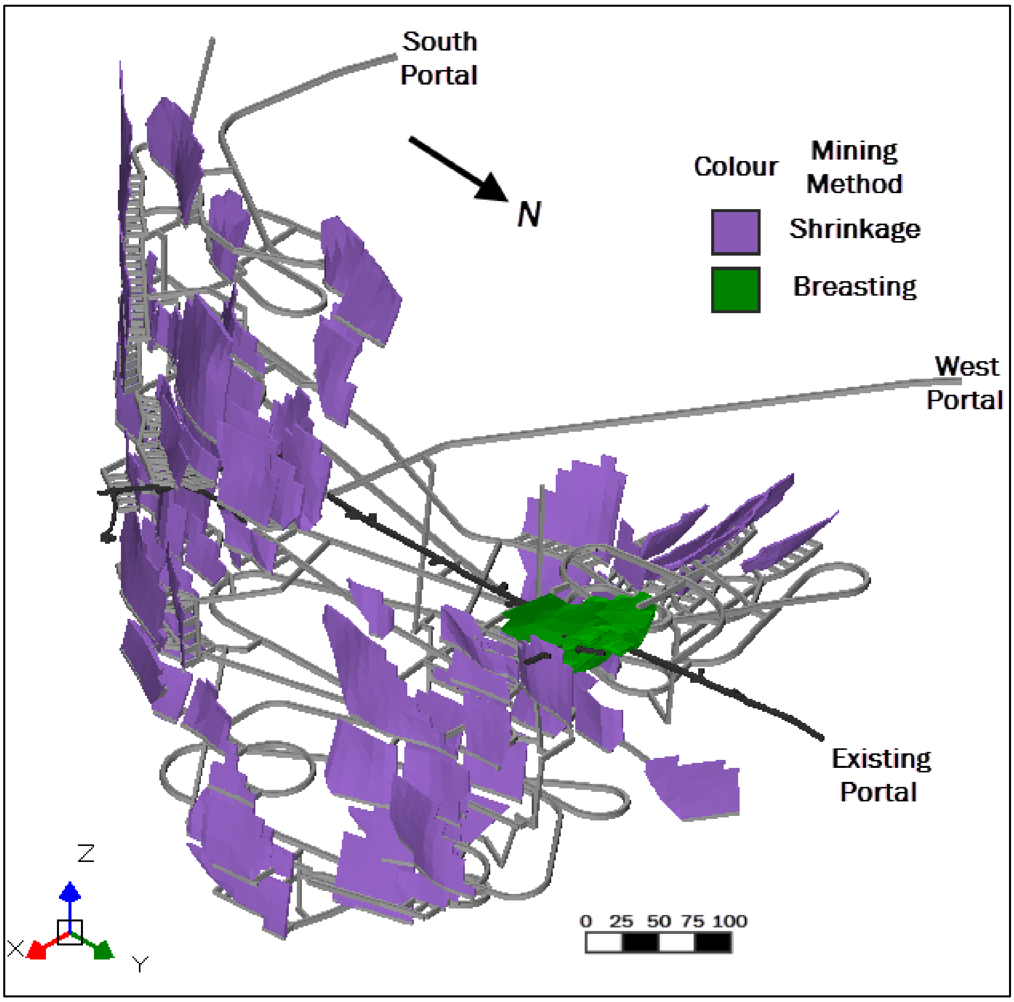


16.3.3 Methods conclusions

AMC considers both these mining methods are appropriate for the deposit given its geometry and the local labour skills available; the methods have historical and current application in the mining industry.

Due to the majority of the zones and lodes having steeply dipping geometry, AMC has assumed predominately shrinkage methods. Figure 16.5 indicates an isometric view of the envisaged mining by method. The Snake/Murau zone contains a small area of breast mining.

Figure 16.5 Isometric of envisaged mining methods



16.3.4 Other local mines

Vatukoula uses labour-intensive handheld methods to mine the generally narrow and flat-dipping lodes. The current mining plan is based on a conventional long-wall breast-stoping mining method.
Other mining methods used over the years include sublevel stoping and caving, cut-and-fill, shrinkage stoping and up-dip mining. The previously predominant room and pillar stoping has given way to conventional long-wall stoping. Attempts have been made to mechanize stoping but were met with limited success due to increased dilution incurring during mining.

At Vatukoula, broken material is scraped to muck bays and then removed from the stopes by diesel loaders and trucks. It is then hauled to the surface using trucks in the decline or hoisted via two haulage shafts.

16.3.5 Materials handling

Due to the projected low annual production rate AMC considers truck haulage the most suitable materials handling method.

16.3.6 Modifying factors

16.3.6.1 Cut-off grade

The diluted underground breakeven cut-off grade was estimated as 4.8 g/t Au and is detailed in Table 16.1.

Table 16.1 Estimated diluted underground breakeven cut-off grade

Input	Unit	Value
Mining cost	US$/t	80
Processing cost	US$/t	62
General and administration (G&A)	US$/t	19
Total costs	**$/t**	**161**
Gold price	US$/oz	1,300
Process recovery – gold	%	86.2
Received gold price (after royalties and other deductions)	US$/oz	1,209
Received gold price	US$/g	39.9
Breakeven cut-off grade	g/t Au	4.8

16.3.6.2 Recovery factor

A stope recovery factor of 95% was applied to allow for typical losses associated with stoping, such as broken material left in the stope and rib pillars where required.

A recovery of 95% was applied to in-lode drives to account for some potential loss in the hanging- and footwall due to difficulty in visually identifying mineralized material from waste.

16.3.6.3 Dilution

To account for unplanned dilution, such as blasting over-break, the projected tonnes from each shrinkage and breasting stope were diluted with 20% of zero grade material. This is in addition to any planned dilution within the stope shape. The in-lode development tonnes were diluted with 8% of zero grade material.

16.3.6.4 Stoping minimum mining width

The total minimum mining width includes allowances for planned and unplanned dilution. Planned, or internal, dilution is material below cut-off grade included in the stope shape to achieve a practical mining shape (stope minimum mining width). Unplanned dilution is additional material outside of the planned stope shape that results from the mining process.

AMC set the minimum mining width to 1.2 m for both the shrinkage and breasting stopes. This width is required for safe and practical operation for personnel to work in the stope. Unplanned dilution is included on all stope shapes regardless of width.

16.3.7 Mine design

16.3.7.1 Existing workings

The proposed mine plan accesses the underground working areas via the existing decline. This decline provides early access to the working areas to expedite the development and production schedule. AMC understands there is a fall of ground in the vicinity of the Coreshed

Fault that will require rehabilitation or a bypass drift. An allowance for this work and general rehabilitation of the decline is included in the schedule.

Before an additional breakthrough is made to surface, mining will be completed with force ventilation. After the initial use of this decline is completed, it is proposed to seal off the decline to reduce the risk of ingress of water to the main underground workings from the Coreshed Fault.

16.3.7.2 Main and South declines

Permanent access will be via two declines located on surface: the Main and South declines, which are additional to the existing decline. The Main and South declines will both be developed from a surface portal box-cut. The box-cut design is proposed to be similar to the existing 1997 Tuvatu adit box-cut. This box-cut is still in good condition 17 years after its construction.

The Main decline will be driven both from surface and from underground. The breakthrough of the Main decline to the existing decline creates an important ventilation and secondary egress connection. The South decline will be driven from surface to ultimately make a connection to the Main decline.

16.3.7.3 Development design

The mine development was designed to access stoping areas, provide ventilation, provide secondary means of egress from the mine, and to allow for haulage of broken rock to surface. The development design includes lateral and vertical excavations.

A ‘race-track’ decline layout was selected. This is a common layout that provides convenient access to multiple mining areas from a single ramp. Internal declines connected to the Main and South declines, access outlying stopes. The Main decline is the primary means of access and haulage over the life of the mine (LOM) and provides the shortest tramming route to the process plant.

The Main and South declines deliver fresh air into the mine. Internal fresh air raises, fitted with ladder-ways, are developed in parallel to the declines to distribute fresh air and provide secondary egress between levels. Two return airway (RAW) raises to surface are provided. Secondary egress to surface is facilitated by either the Main or South declines. The existing decline will be sealed off after its initial use and will not be used for ventilation purposes over the LOM.

The mine design and development schedule is prioritized to provide access to high grade stopes as quickly and practicably as possible. A parallel priority, in recognition of project risk, is early access to stoping areas that contain Indicated Mineral Resource material.
Figure 16.6 and Figure 16.7 show north-south and east-west sections, respectively, of the underground conceptual mining blocks within the Indicated and Inferred Mineral Resource categories.

Figure 16.6 North-south section – underground conceptual mining blocks

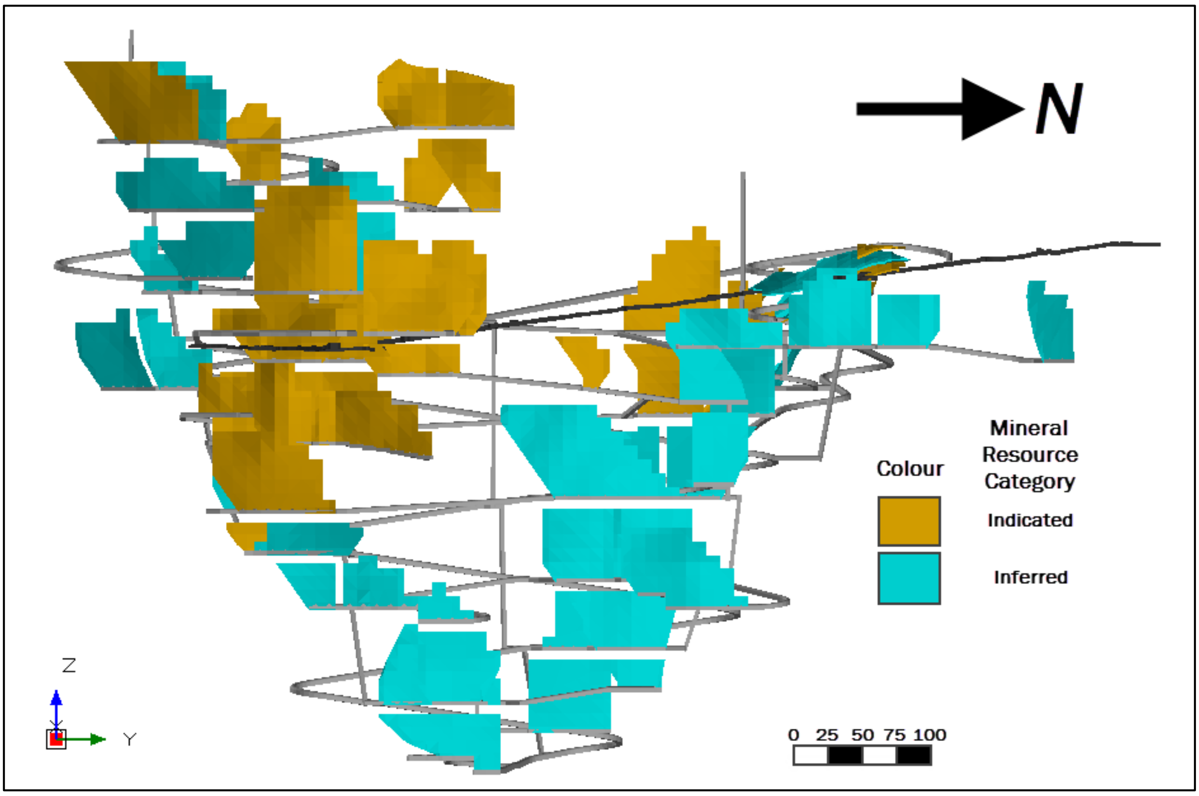
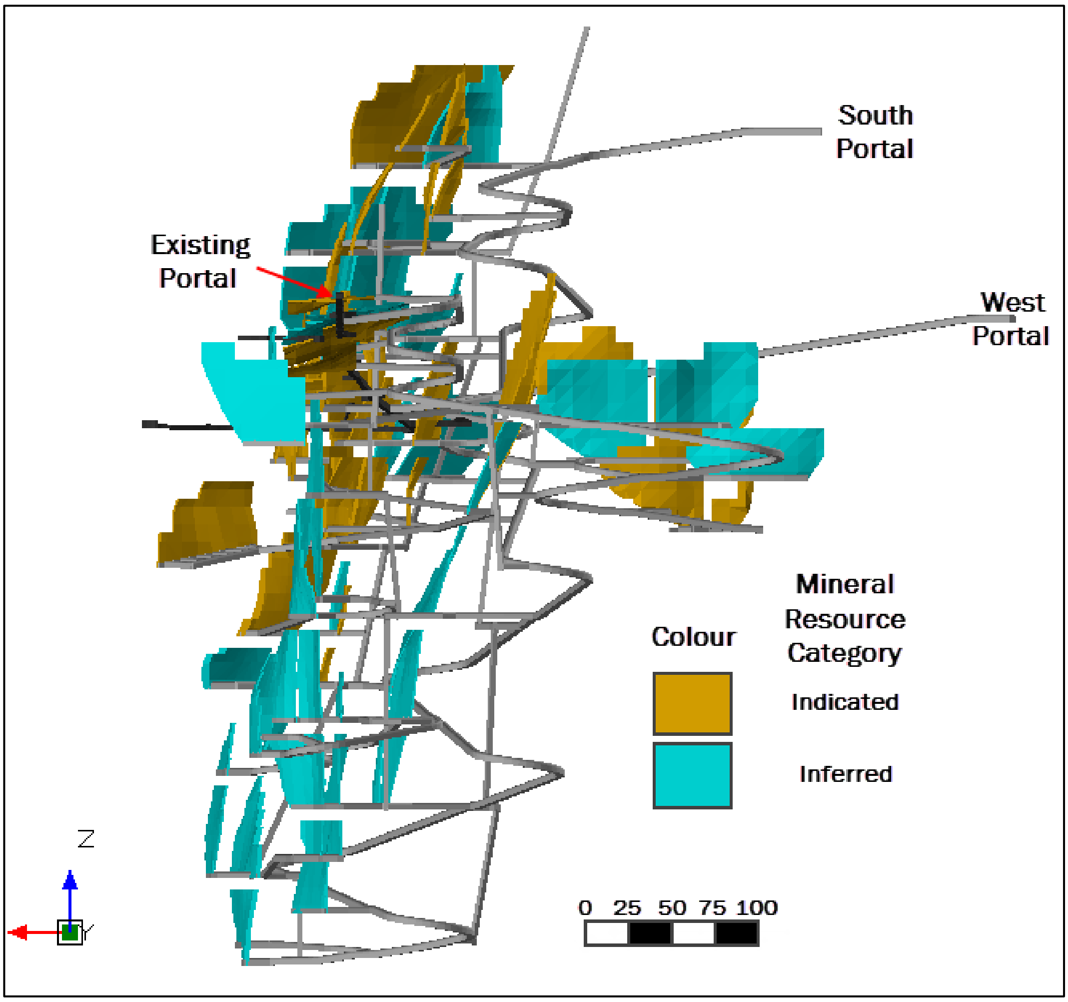


Figure 16.7 East-west section – Underground conceptual mining blocks



Internal ramps connected to the Main decline provide access to the flat-dipping SKL area, a section of the Tuvatu zone and the Murau lodes. The Main and South declines provide access to the majority of the remaining lodes.

The lateral development design includes:

- Decline development. The declines provide access to production levels and serve as the main haulage ways of the mine.
- Ventilation development. Developed parallel to the decline, and connecting ventilation shafts with ventilation rises in the vicinity of the tight spiral section of the declines.
- Level access development. Level access development connects the decline to in-lode drives on the production levels. The level access typically includes a stockpile and sump. Level access development also includes footwall drive development.
- Draw-points. Draw-point development is driven perpendicular to the conceptual stope strike.

Key parameters of the design include:

- A decline gradient of 1:7.
- A minimum decline radius of 25 m.
- A target minimum decline stand-off distance of 40 m from the stoping area.

- A typical decline stockpile spacing of 120 m.
- 8% unplanned over-break allowed for in all waste development drives.

16.3.7.4 Development dimensions

Development will be carried out using conventional drill and blast techniques. Drive dimensions are listed in Table 16.2. Dimensions provide minimum safe working clearance and are sized to fit all development and production activities.

Table 16.2 Lateral development dimensions

Development	Width (m)	Height (m)	Gradient	Profile	Capital/ Operating
Decline	4.5	4.5	-1:7	Arched	Capital
Return airway	4.5	4.5	1:50	Arched	Capital
Level access	4.0	4.0	1:50	Arched	Operating
Footwall drive	3.5	4.0	1:50	Arched	Operating
Draw-points	3.5	4.0	1:50	Arched	Operating

Vertical development will be carried out by Alimak raising. Dimensions are shown in Table 16.3.

Table 16.3 Vertical development dimensions

Development	Width (m)	Height (m)	Gradient	Profile
Fresh airway	3.0	3.0	Vertical	Square
Return airway	3.0	3.0	Vertical	Square

16.3.7.5 Development totals

A summary of the development metres is provided in Table 16.4 for lateral development and Table 16.5 for vertical development.

Table 16.4 Lateral development metres

Lateral Development	Total (m)
Decline	5,580
Return airway access	920
Fresh airway access	208
Total Capital Development Metres	**6,708**
Level Access	3,531
Footwall drives	4,655
In-lode Cross Cuts	4,900
Draw-points	7,661
Total Operating Development Metres	**20,747**
Total Lateral Development	**27,455**

Table 16.5 Vertical development metres

Vertical Development	Total (m)
Return airway raise	962
Fresh airway raise	411
Total Vertical Development Metres	**1,373**

16.3.8 Stope design

16.3.8.1 Methodology

The following sequence of activities was used to develop the conceptual stopes for the underground lodes:

Grade-shells were generated based on the selected cut-off grade (COG) and matched to the geological vein wireframe. The COG used to generate the grade-shells included an appropriate factor to account for unplanned dilution.

In long section, relative to the strike of the vein, an outline of the grade-shell was created. The outline was modified to meet stope design criteria to create a practicable mining shape. Stope design criteria include geotechnical constraints and the dip required to allow free running of muck.

The vein wireframe was then clipped to the modified grade-shell outline to create a final stope shape.

The quantities in the stope design shapes were exported to Microsoft Excel and conceptual economic evaluation completed through addition of access development costs. Areas perceived as being uneconomic were removed from the design.

16.3.8.2 Grade distribution

Long sections of the mine design showing estimated grade distributions are provided in the following figures. These distributions are based on the estimated Mineral Resource in situ gold content contained within the stoping shapes.

Figure 16.8 and Figure 16.9 shows the spatial gold content distribution for the north-south and east-west long sections of the mine. As described above, a scheduling priority was to access higher grade stopes as early as practicable. The South decline was included in the conceptual mine plan to expose higher grade lodes early in mine life, adding significant value to the project. Higher grade material adjacent to the existing decline is also targeted early in the schedule as it can be accessed readily after the rehabilitation is completed.

Figure 16.8 North-south long section - stoping block estimated gold content

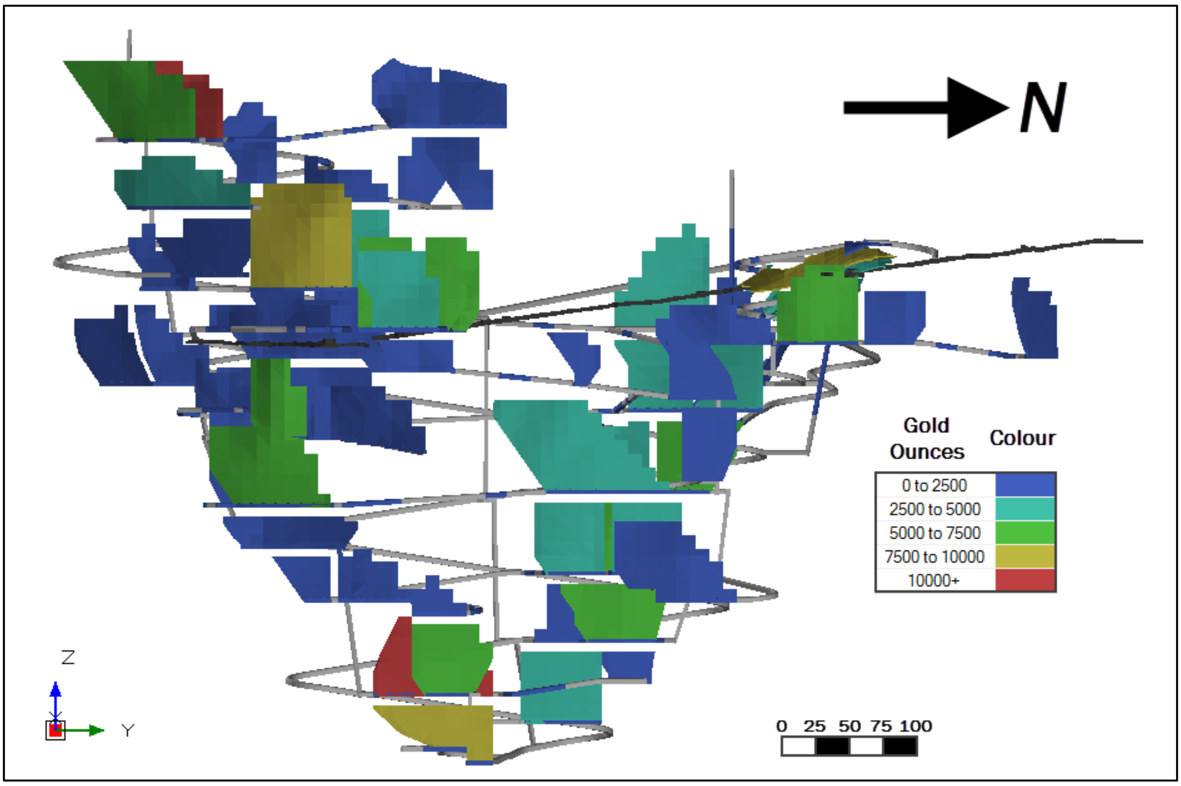
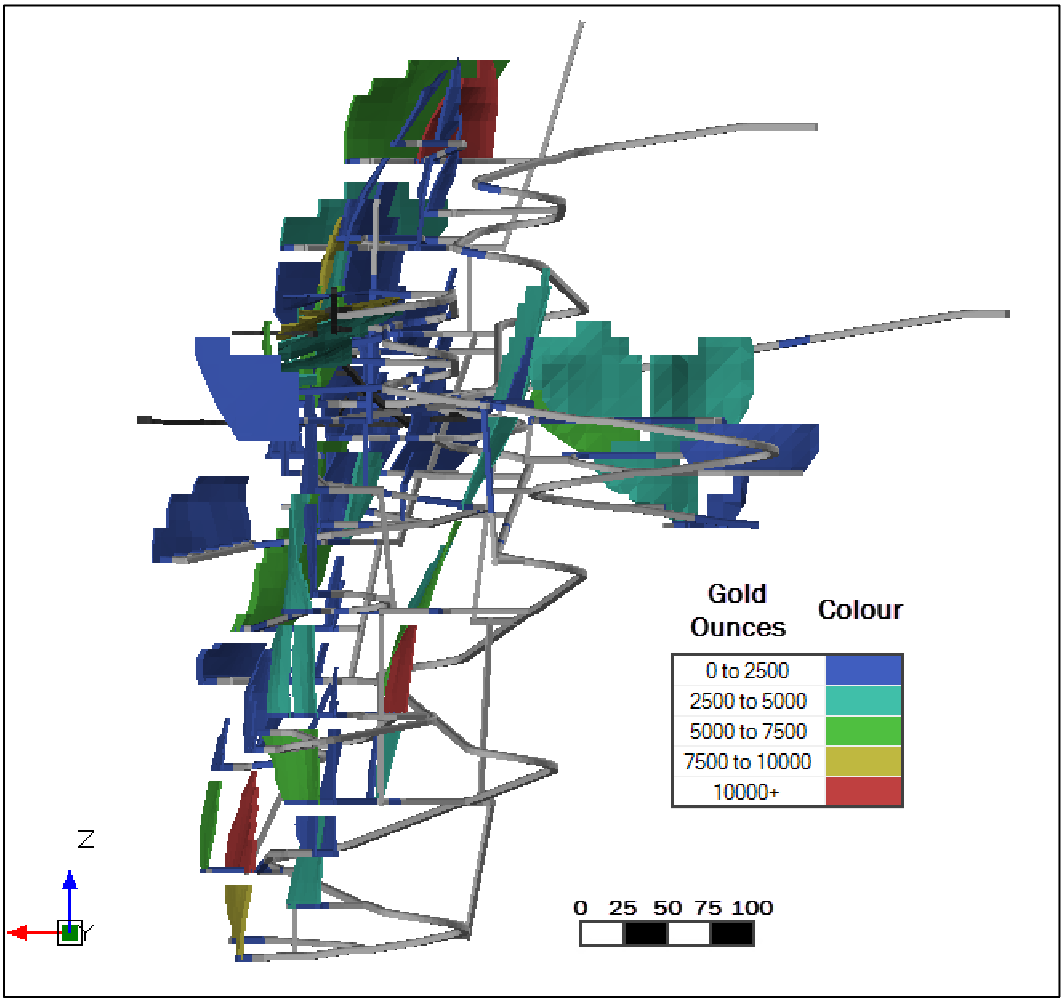


Figure 16.9 East-west long section – stoping block estimated gold content



16.3.9 Underground mobile equipment

Underground mining is based on using mechanized mobile equipment, and air-leg drills for stope production.

The projected equipment is considered appropriately sized for the degree of selectivity required and the equipment fleet size is considered appropriate for the quantities of material to be moved. The main fleet will be supported by an appropriate ancillary fleet. To reduce capital costs and take advantage of the local skills base with air-leg drills, Long-tom drills are proposed for the level development, along with air-leg drills for stoping. Jumbo drills will be used for the mining of the ramps and level accesses.

Equipment replacement requirements were assessed by modelling the equipment operating hours for each piece of equipment over the expected equipment life. No equipment requires replacing during the mine life. Equipment rebuilds are incurred at 50% of the equipment life.

The projected equipment numbers required are shown by type and average number in Table 16.6.

Table 16.6 Mobile equipment numbers required

Mobile Equipment Type	Capacity	Number
Jumbo	2 boom	2
Bolter	-	1
Long-tom	2 air drills	4
Charge-up unit	-	2
Loader 1	3 m^3	2
Loader 2	1.5 m^3	3
Loader 3	1 m^3	1
Haul truck	20 tonne	3
Light vehicle	-	12
Service vehicle	-	2
Grader	-	1
Water cart	-	1
Service truck		1

16.3.10 Mine services

Major underground infrastructure and facilities consist of the following:

- Ventilation system
- Electrical distribution
- Dewatering system
- Communications system
- Explosives magazine
- Maintenance service bay
- Lunchroom

16.3.10.1 Ventilation

The primary ventilation circuit will be a "pull" or exhausting type ventilation system. That is, the primary mine ventilation fans will be located at the primary exhaust airways of the mine. Fresh air will enter the mine via the Main and South decline portals and exhaust to the surface via two dedicated return airway raises.

A series of return air raises (RARs) and fresh air raises will be developed as the mine deepens, connecting at each level. Contaminated air from each active level will enter the RAR system via a drop board regulator installed in the access to the RAR on each level. The RAR system will connect to the surface through two vertical raises.

Single and dual stage axial ventilation fans will deliver fresh air to the working headings through ventilation ducting of various diameters.

The primary airflow quantities were developed based on consideration of the diesel fleet. The total airflow is planned to be 110 m^3/s. The total airflow is based on the following criteria.

- 0.06 m^3/sec airflow will be supplied per diesel kW. The size and number of diesel equipment units have been estimated as part of the Study.

- Utilization factors have been estimated by AMC for each piece of equipment dependent upon the type and its expected diesel operating time per shift. Additionally, for those pieces of equipment that are primarily electric/hydraulic operated units with a diesel engine used only for tramming from work site to work site, a percentage of time the equipment is expected to tram per shift has been allocated.
- An additional minimum amount of air required per person in the workplaces will be three cubic metres/minute. An assumption of a maximum of 40 persons underground during a shift has been applied.
- A contingency factor of 20% has been applied to account for leakage and other system inefficiencies.

16.3.10.2 Power

High voltage power will be supplied to the underground mine via a 3.3 kV overhead power-line from the (to be developed) site power station.

The high voltage power will be supplied to mine substations located at approximately 60 m vertical intervals underground. The substations will reduce the incoming high voltage to low voltage of 415 V. During the life of the mine, the substations will be moved as required to prevent efficiency losses. The outgoing circuits from the low voltage side of the substations feed a combination of individual 4-way and/or 3-way distribution boards. The distribution boards typically supply power to:

- Drill rig starters
- Pump starters
- Single and dual fan starters

The maximum connected load for the mine is expected to be approximately 1.5 MW.

16.3.10.3 Dewatering

Based on the limited hydrology data available, the estimated dewatering requirement from the underground workings due to ground water inflow is approximately 20 l/s. Additional water of approximately 6 l/s will be generated by the mining activities. A settling pond will be established on surface.

Considerable more water inflow could be expected in the wet season, therefore a primary pumping network capable of up to 40 l/s will be required to pump all water from the underground mine to the surface.

To facilitate early mining activities the existing exploration decline will require dewatering. A 37 kW submersible pump located on a raft will meet this requirement.

A main sump located at the base of the existing decline is envisaged to capture water from the Coreshed Fault and the upper levels of the mine. An additional main sump will be located approximately 150 m below the upper main sump. Main sumps will generally consist of two appropriately-sized pumps arranged in series. The sump excavation will be constructed to have the capability to settle fines from the water prior to pumping.

Small submersible secondary pumps and HDPE piping will deliver the water from sumps located on each level to the main sumps.

16.3.10.4 Compressed Air

Compressed air is required to operate various mining equipment such as long-tom drills, charge-up and small dewatering pumps. However, the primary users of compressed air will be the jacklegs operating in the active stopes. To support both development and production mining activities, compressed air capability will be established at the decline portals, from where air will be delivered via steel pipe to the working areas.

To meet the steady state conceptual production schedule, approximately 18 jacklegs will be required to run simultaneously. This equates to a compressed air demand of approximately of 2,500 cfm. To meet this demand as well as that from other users of compressed air, a 350 kW compressor or equivalent is specified.

16.3.10.5 Communications

A Wi-Fi communications system will be used as the primary means of communication at the mine. The system will allow for communications within the mine and with surface operations.

A tracking system will provide real time data on equipment and personnel locations throughout the mine. The system will also monitor the main ventilation fans.

16.3.10.6 Other underground infrastructure

Other underground infrastructure includes:

- A second egress from the mine consisting of an escape-way from the lower levels of the mine to the declines which provide access to surface. The escape-way will generally consist of a ladder-way in the lower mine fresh air raises.
- Refuge chambers will be placed at strategic locations.
- Explosives magazines located in the return airway.
- Maintenance service bay
- Lunchroom

16.3.11 Underground mine schedule

The underground life is projected to be approximately 7.4 years. The schedule is based on continuous operation, being 24 hours per day, 320 days per year.

The key schedule drivers are rates for development advance and stope production. These rates were established by first principles estimates calculated from activity cycle times, available work hours and equipment availability.

As the local workforce is jack-leg proficient but with very limited experience in trackless and shrinkage mining methods, the scheduling rates were adjusted for the first year to reflect a training period.

The key scheduling rates used in the schedule, after the initial training period, are listed in Table 16.7.

Table 16.7 Underground scheduling rates

Item	Unit	Maximum
Jumbo development – single heading	m/month	100
Jumbo development – multi heading	m/month	140
Longtom Development – multi heading	m/month	100
Vertical development (Alimak)	m/month	50
Stope preparation (raises)	months	2
Shrinkage stoping – initial stoping (draw-down)	t/day	45
Shrinkage stoping	t/day	150

Due to the deposit's overall narrow vein characteristic, as well as the dispersed nature of accessing multiple lodes on the same level, the project has a high lateral development requirement per stope production tonne. Just over 6.7 km of capital lateral development is estimated to be required, as well as just over 20.7 km of operating development, as identified in Table 16.2. Depleted stopes need to have development leading the mining front significantly at all times in order to replace depleted stoping areas. The capital vertical development is made up of fresh air and return airway raises.

The conceptual mining schedule is summarized in Table 16.8, Figure 16.10 and Figure 16.11
.

Table 16.8 Conceptual annual mining schedule

Item	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Physicals Development										
Capital	m	3,749	3,437	1,582	316	236				**9,319**
Operating	m	1,941	3,572	3,828	4,302	4,191	117	187		**18,137**
Total	**m**	**5,690**	**7,009**	**5410**	**4,618**	**4,427**	**117**	**187**		**27,456**
Vertical Development										
Capital	m	270	535	559	8					**1,373**
Mill-feed Mineralized tonnes	t	21,996	140,502	172,036	198,263	196,846	197,024	167,190	31,692	**1,125,548**
Waste tonnes	t	235,413	270,636	180,346	119,270	104,236	-	611	-	**910,512**

Figure 16.10 Conceptual annual underground development metres

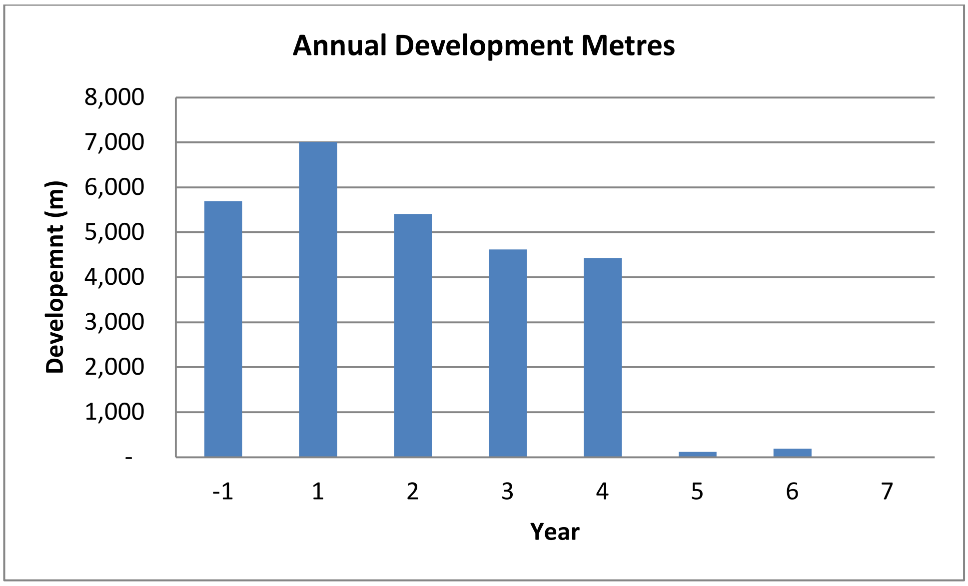
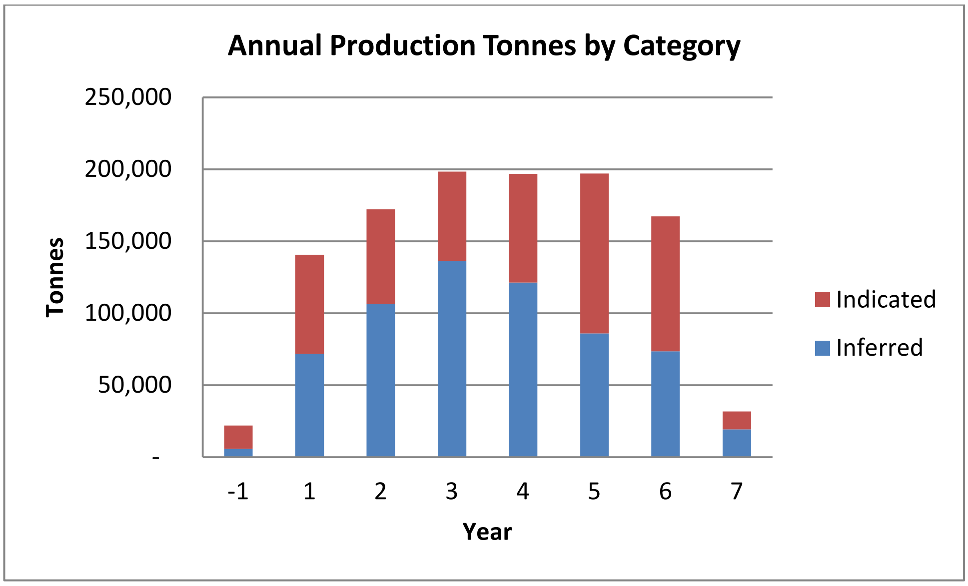


Figure 16.11 Conceptual annual mill-feed by resource category



16.4 Personnel

Personnel numbers have been estimated for all mining department personnel, including management and supervision, technical staff, operators and maintenance.

The underground operations personnel are to work a four panel, three shift roster. Features of this roster are:

- Continuous operation, being 24 hours per day, 320 days per year.
- Three 8 hour shifts per day.
- The schedule requires 4 shifts of operators.

Operator personnel numbers are driven by the equipment requirements and are variable over time.

Projected annual underground personnel numbers are provided in Table 16.9.

Table 16.9 Projected annual underground personnel numbers

Personnel Numbers	Year - 1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Management and Technical Staff								
Mining Manager	1	1	1	1	1	1	1	1
Technical Services Manager	0	1	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1	1	1
Senior Surveyor	1	1	1	1	1	1	1	1
Surveyor	1	2	2	2	2	2	2	2
Surveyor assistant	1	2	2	2	2	2	2	2
Mining engineer	2	3	3	3	3	3	3	3
Geotechnical engineer	1	1	1	1	1	1	1	1
Geologist	2	4	4	4	4	4	4	4
Sampler	4	6	6	6	6	6	6	6
Clerk	2	2	2	2	2	2	2	2
Subtotal	**16**	**24**	**24**	**24**	**24**	**24**	**24**	**24**
Underground Staff								
Underground Manager	0	0	0	0	0	0	0	0
Production Engineer	0	0	0	0	0	0	0	0
Ventilation	1	1	1	1	1	1	1	1
Mine Superintendent	1	1	1	1	1	1	1	1
Shift Supervisors	4	4	4	4	4	4	4	4
Training Officer	6	4	4	4	4	4	4	4
Storeman	2	2	2	2	2	2	2	2
Subtotal	**14**	**12**	**12**	**12**	**12**	**12**	**12**	**12**
Operators								
Jumbo and ground support	24	20	16	12	12	4	4	0
Production drill	0	0	0	0	0	0	0	0
Charge-up	8	8	4	4	4	4	4	0
Loader	24	24	24	20	20	12	12	8
Truck	12	12	12	12	12	8	8	4
Grader/WC	2	2	2	2	2	2	2	2
Air-leg	10	22	22	28	28	28	28	12
Services	28	28	28	28	28	16	16	16
Subtotal	**108**	**116**	**108**	**106**	**106**	**74**	**74**	**42**
Maintenance								
Maintenance Superintendent	1	1	1	1	1	1	1	1
Electrical Superintendent	1	1	1	1	1	1	1	1
Maintenance Planner	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1
Leading Hand Mechanic	4	4	4	4	4	4	4	4
Mechanic	22	22	22	22	22	22	22	22
Electrician	8	8	8	8	8	8	8	8
Subtotal	**38**	**38**	**38**	**38**	**38**	**38**	**38**	**38**
Total	**176**	**190**	**182**	**180**	**180**	**148**	**148**	**116**

Expatriates are proposed for the following positions:

- Mining manager
- Technical services superintendent
- Underground mine foreman
- Underground maintenance superintendent

- Underground electrical superintendent
- Trainers

The expatriates will provide management, technical direction and supervision of the mining and production and maintenance areas.

16.5 Production schedule

In accordance with Item 22 (b) of Form 43-101F1, this section discusses the conceptual production schedule that underpins the economic analysis. Readers are cautioned that the tonnes and grade estimates that comprise the production schedule include Inferred Resources.

The existing exploration decline will be used to provide early access to the underground workings. Rehabilitation of the exploration decline will be required. Early mining activities include establishing the first ventilation raise to surface and developing the Main decline towards surface. The Main and South declines will also be mined from surface.

Development is projected to commence approximately 15 months prior to process start-up.
An approximately 8-year mining operation is projected, as detailed in Table 16.10.

Table 16.10 Conceptual production mill-feed schedule

Item	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Underground										
Indicated										
Tonnes	kt	16.3	68.9	65.7	61.9	75.6	111.2	93.6	12.4	505.6
Grade	g/t	7.31	13.60	12.99	9.15	8.01	7.04	7.31	6.18	9.15
Contained ounces	koz	3.9	31.5	28.5	18.7	19.8	25.4	22.4	2.5	152.7
Inferred										
Tonnes	kt	5.7	71.6	106.3	136.4	121.3	85.8	73.5	19.2	619.9
Grade	g/t	6.53	18.05	17.96	16.60	8.70	7.07	7.39	7.33	12.66
Contained ounces	koz	1.3	42.4	62.8	73.3	34.4	19.6	17.9	4.5	256.3
Indicated and Inferred										
Tonnes	kt	22.0	140.5	172.0	198.3	196.8	197.0	167.2	31.7	1,125.5
Grade	g/t	7.38	16.36	16.49	14.44	8.57	7.11	7.49	6.87	11.30
Contained ounces	koz	5.2	73.9	91.2	92.1	54.3	45.0	40.3	7.0	409.0

The tonnage/grade estimates that comprise the conceptual production schedule do not represent estimates of Mineral Reserves.
Values may not compute exactly due to rounding

17.0 Recovery Methods

17.1 Introduction

The processing facility for the Tuvatu project is based on testwork described in Section 13 and consists of a flotation and carbon-in-leach (CIL) flowsheet comprising a two-stage crushing and screening circuit, two-stage grinding, gravity concentration, rougher flotation, cyanide leaching and carbon adsorption of both the reground concentrate and flotation tailings, cyanide detoxification, carbon elution and regeneration, gold refining, and tailings disposal.

The process facility is designed with a nominal capacity of 219,000tpa for a design (nominal) rate of 600tpd based on an overall availability of 91% with a life of mine average feed grade of 11.3g/t Au. The plant is designed to operate 365 days/year, 24 hours/day. The crushing circuit is designed with a mechanical equipment availability of 75%, however it has been sized to process 1000tpd to accommodate any future expansion.

The proposed process plant will include the following unit operations:

- Primary Crushing – A dump pocket, vibrating grizzly and jaw crusher in open circuit producing a final product of 80% passing 105 mm.
- Secondary Crushing - A vibrating double deck screen and cone crusher operating in closed circuit.
- Primary Grinding – A ball mill in open circuit producing a final product of 80% passing 1000µm.
- Secondary Grinding – A ball mill in closed circuit with hydrocyclones producing a final product of 80% passing 75µm.
- Gravity Concentration – Gravity concentration of cyclone underflow from the secondary milling circuit to produce a gold concentrate for tabling, followed by direct smelt.
- Flotation – Sulphide flotation of the hydrocyclone overflow to produce a gold concentrate for regrinding.
- Regrinding – A regrind mill fed by open circuit hydrocyclones producing a final product of 80% passing 20µm.
- Thickening - Both the flotation tails and concentrate are thickened to 50% solids prior to regrinding and leaching.
- Carbon-in-leach (CIL) – Gold leaching of the flotation tails and reground concentrate through the two CIL circuits, where absorption of solution gold onto carbon particles occurs. Leaching is facilitated by oxygen and air.
- Cyanide Detoxification – Detoxification of cyanide slurry via the SO_2/Air process with addition of copper sulphate, to produce tailings with a target of <1ppm CN_{WAD} (Weak Acid Dissociable) and disposal of detoxified tailings in the conventional tailings storage facility.
- Absorption, Desorption, and Refining (ADR) – The absorption occurs in the CIL circuit, acid wash of carbon to remove inorganic contaminants, elution of carbon to produce a gold rich solution for electrowinning (sludge production), filtration, drying, and smelting to produce gold doré, and thermal regeneration of stripped carbon to remove organic contaminants.

At the preliminary level of this study, test work results, industry data and assumptions have been used to make reasonable estimates for equipment sizing. These inputs will be reviewed in the next phase of engineering. A design criteria was developed based on the available testwork and a summary is included below. The design criteria include the calculations or information basis for each piece of major equipment.

17.2 Process Design Criteria

The Process Design Criteria (PDC) and mass balance have been completed and detail the annual feed and product capabilities, major mass flows, equipment capacities and availabilities. Consumption rates for operating and maintenance consumables can be found in the operating cost estimate described in Section 21. The design life for the Tuvatu process plant will be 10 years. Key process design criteria are shown in Table 17-1.

Table 17-1: Major Process Design Criteria

	UNITS	DESIGN
1.0 PLANT FEED CHARACTERISTICS, PRODUCTION AND OPERATING SCHEDULES		
1.1 PLANT FEED CHARACTERISTICS		
Physical Characteristics		
Specific Gravity	g/cm^3	2.7
ROM Moisture	%	5.0
Bulk Density	t/m3	1.6
Drop Weight Index (DWi)	kWh/m3	4.5
Crushing Work Index (CWi)	kWh/t	12.5
Abraision Index (Ai)	kWh/t	0.184
Bond Rod Mill Work Index (RWi) (75th)	kWh/t	19.8
Bond Ball Mill Work Index (BWi) (75th)	kWh/t	18.6
1.2 PRODUCTION		
General		
Operating Days per Year	d	365
Total Hours	h	8760
Throughput Tonnage		
Annual Throughput	t	219,000
Gold Recovery - LOM		
Total Recovery to Bullion	%	86.3
1.3 OPERATING SCHEDULES		
Crushing Schedule		
Days per year	d	365
Availability, Overall System	%	75
Throughput	t/d	1000
Grinding Schedule		
Days per year	d	365
Availability, Overall System	%	91
Throughput	t/d	600
2.0 CRUSHING		

2.1 CRUSHING SYSTEM		
Primary Crusher		
Type		Jaw Crusher
Number	#	1
Installed Power	kW	90
Product Size (P80)	mm	105
Secondary Feed Screen		
Type		Dual Deck: Deck 1 50mm, Deck 2 15mm
Secondary Crusher:		
Type		Cone Crusher
Number	#	1
Installed Power	kW	250
Product Size (P80)	mm	30
2.2 STOCKPILE		
Crushed Product Handling		
Primary Storage		Stockpile
Live Capacity	t	600
Number of Draw Points	#	2
3.0 COMMINUTION		
3.1 GRINDING		
Primary Grinding Circuit		
Type		Ball Mill
Circuit Configuration		Open
Number of Mills	#	1
Availability	%	91
Installed Power	kW	522
Product Size (P80)	µm	1000
Secondary Grinding Circuit		
Type		Ball Mill
Design Factor	%	100
Circuit Configuration		Closed
Number of Mills	#	1
Availability	%	91
Power Required	kW	522
Circulating load	%	300
Product Size (P80)	µm	75
Grinding Discharge Classification		
Type		Cyclopac
Cyclone Size	mm	254
Number Operating	#	3
Standby	#	1
Gravity Concentration		
Type		Gravity Concentrator
Number	#	1

		% of Underflow to Concentrator	%	25
		Reporting to Concentrate	%	38
4.0	**FLOTATION**			
	4.1 Flotation			
		Rougher Conditioning Tank		
		Number of Tanks	#	1
		Retention time – Total	min	12
		Tank Volume – Total	m^3	17
		Rougher Cells		
		Flotation Retention time	min	20
		Slurry Condition (Natural pH)	pH	7.6
		Cell Size	m^3	10
		Number of Cells	#	6
		Concentrate		
		Mass Recovery, % of Feed	%	20.0
	4.2 Float Concentrate Thickening			
		Thickener Feed		
		Feed Density	%	35
		Underflow Density	%	50
		Regrind		
		Availability	%	91
		Number of Mills	#	1
		Estimated Installed Power	kW	186
	4.3 Float Tails Thickening			
		Thickener Feed		
		Feed Density	%	31
		Underflow Density	%	50
5.0	**CARBON-IN-LEACH**			
	4.4 Float Concentrate Leaching (CIL)			
		Retention time – Total	h	24
		Number of Tanks	#	6
		Carbon content	g/l	20.0
	4.5 Float Tails Leaching (CIL)			
		Retention time – Total	h	24
		Number of Tanks	#	6
		Carbon content	g/l	20.0
		Residual Cyanide	ppm	200
	4.6 Metal Recovery – ADR			
		Carbon Management		
		Total Tons in Leach Circuit	t	22
		Carbon Movement	t/day	1.0
		Carbon Plant Size	t	2
	5.6 Cyanide Detoxification			
		Retention time – Total	min	85
		Number of Tanks	#	2

Target Discharge Solution CN (WAD)	ppm	1

17.3. Process Flow Diagrams

Simplified process flow diagrams (PFD's) for each unit operation can be found in the following sections). A conceptual overall flowsheet and the process plant layout are illustrated in Figure 17-1 and Figure 17-2 below. The process mass and water balances include flow to and from the tailings impoundments only.

Figure 17-1: Plant Summary Flow Diagram

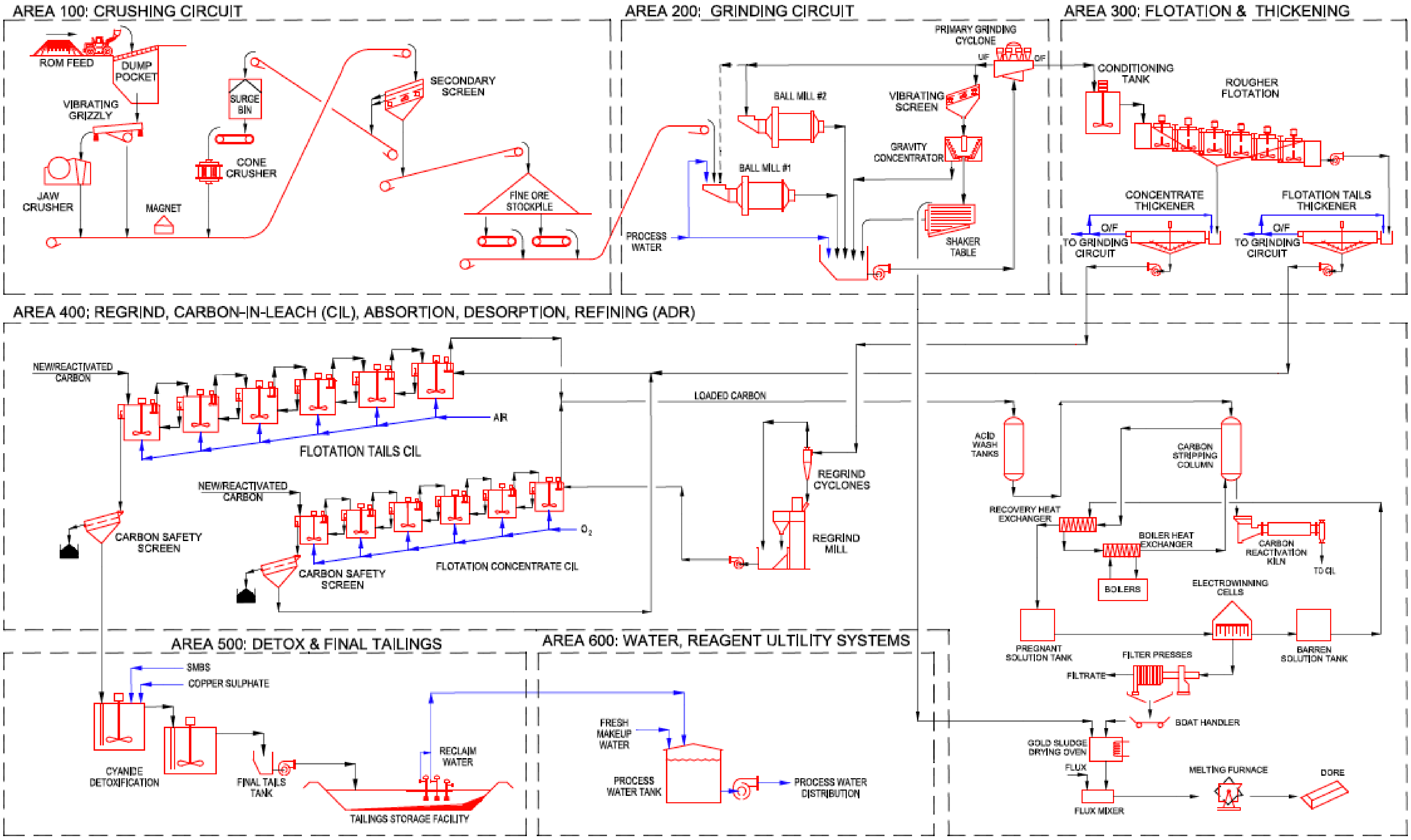
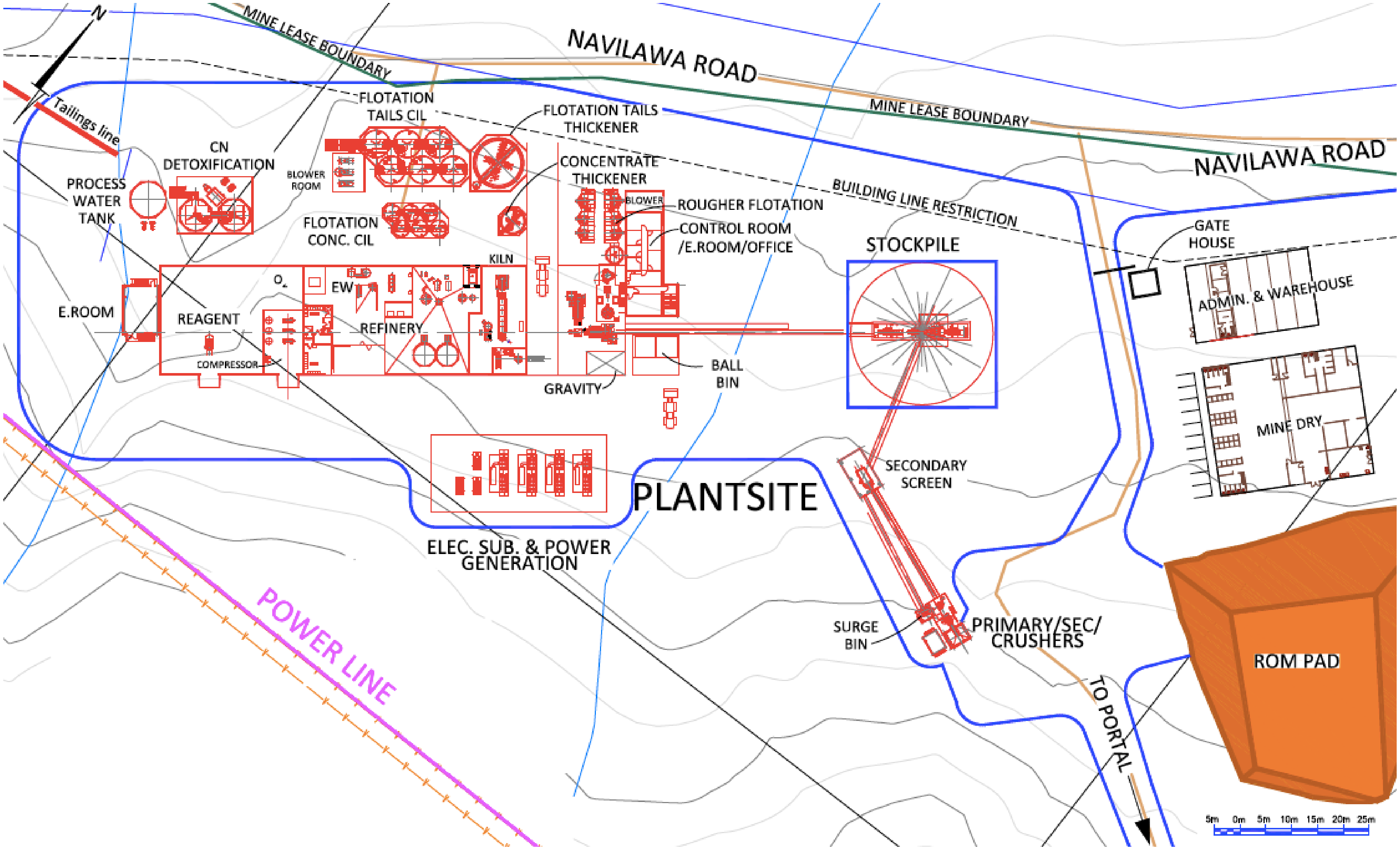


Figure 17-2: Process Plant Layout



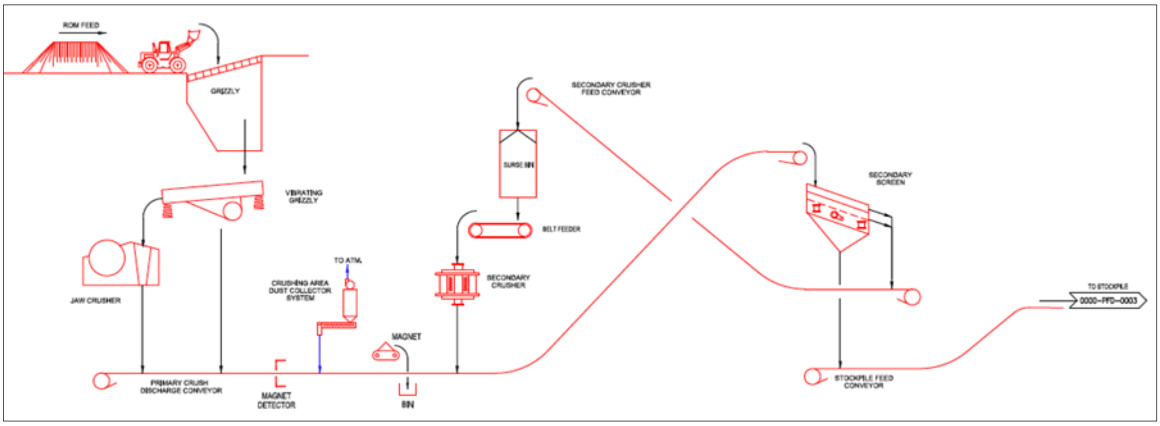
17.4 Crushing Circuit

Table 17-2: Crushing Circuit Equipment Summary

Description	Unit	Value
Static Grizzly	mm x mm	450 x 450
Vibrating Grizzly	mm x mm	1,020 x 3000
Jaw Crusher	mm x mm	1100 x 700
	kW	90
Screen	mm x mm	1500 x 5000
Cone Crusher	CSS (mm)	12 – 38*
	kW	250

*Depending on mantle/bowl profile selected.

Figure 17-3: Crushing Circuit



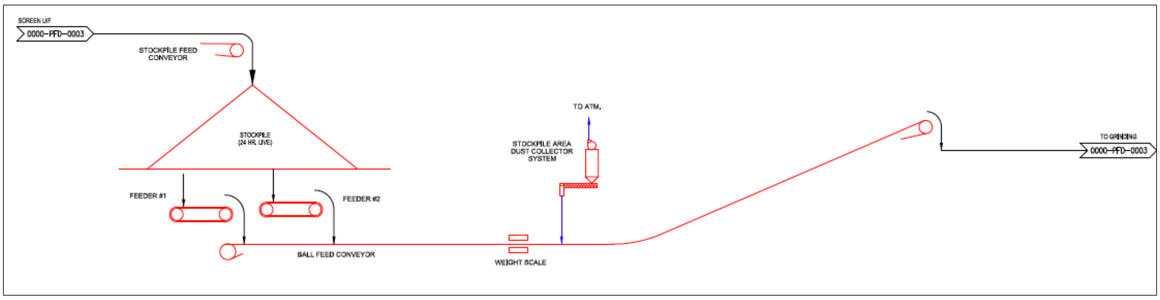
The proposed flowsheet for Tuvatu commences with a secondary crushing system designed for 1000tpd, and to operate 24 hours per day, at 75% overall system mechanical availability. The crushing system was designed at this capacity to allow for the potential to operate the facility in day shift only, for “over crushing” prior to relines or for future expansion.

Caution should be exercised when reviewing the crushing plant sizing as the crushing index is based on a single test and the feed size is assumed. The circuit was simulated using PlantDesigner© and Bruno© simulation software, however, due to the hardness and equipment specifications (i.e., the closed side setting (CSS) may not be fine enough for a crusher that can handle the largest lump feed size), it may be that in practise a tertiary crushing circuit would be specified to achieve the necessary size reduction. The sizing of each crusher is an equivalent of typical equipment available from the respective vendors but does not necessarily need to be specified by those vendors.

Mineralized material from the underground mining operations will feed a vibrating grizzly - primary jaw crusher system in open circuit, which will produce a product size (P80) of 105 mm. A top size of 600mm has been assumed for the material from the underground mine. The method of feeding the crushing system can be by truck or front-end loader (FEL). The discharge from the vibrating grizzly underflow and primary crusher product are combined with the secondary crusher product and conveyed to the secondary screen.

The secondary screen is an inclined, dual-deck vibrating screen. The upper and lower panels have 50mm and 15mm openings respectively. The oversize from both screen decks reports to a secondary cone crusher feed bin, which in turn feeds the secondary cone crusher in closed circuit with the secondary screen. The feed bin is sized for approximately ten minutes of surge capacity at approximately 15t. The cone crusher has a maximum motor power of 250kW and a CSS range from 10-25mm in the throughput range required, contingent on the mantle profile selected. Crushed product from the secondary crusher joins the materials from the primary crusher and vibrating grizzly, and re-circulated to the secondary screen. The undersized materials from the screen are product size particles, which are conveyed to the stockpile.

A dedicated dust collector system will be installed in the crushing area to capture dust from the primary and secondary crushing system and screen. The system will include collection hoods, a bag-house complete with fan, a rotary valve and dust reclaim screw conveyor. The captured dust will be discharged onto the stockpile feed conveyor.



The crushed ore stockpile will be covered for protection from the elements due to the fine particle size and the increased precipitation during the wet season, and for dust control, with dozer or FEL access on one side. The stockpile will have a 600t live capacity that can support process plant operations for 24h when the crushing plant is not in operation. Two feeders will reclaim the crushed material from the stockpile, both sized to process the full mill feed should one require maintenance. The stockpile area will also have a dust collection system, which will discharge the dust onto the ball mill feed conveyor.

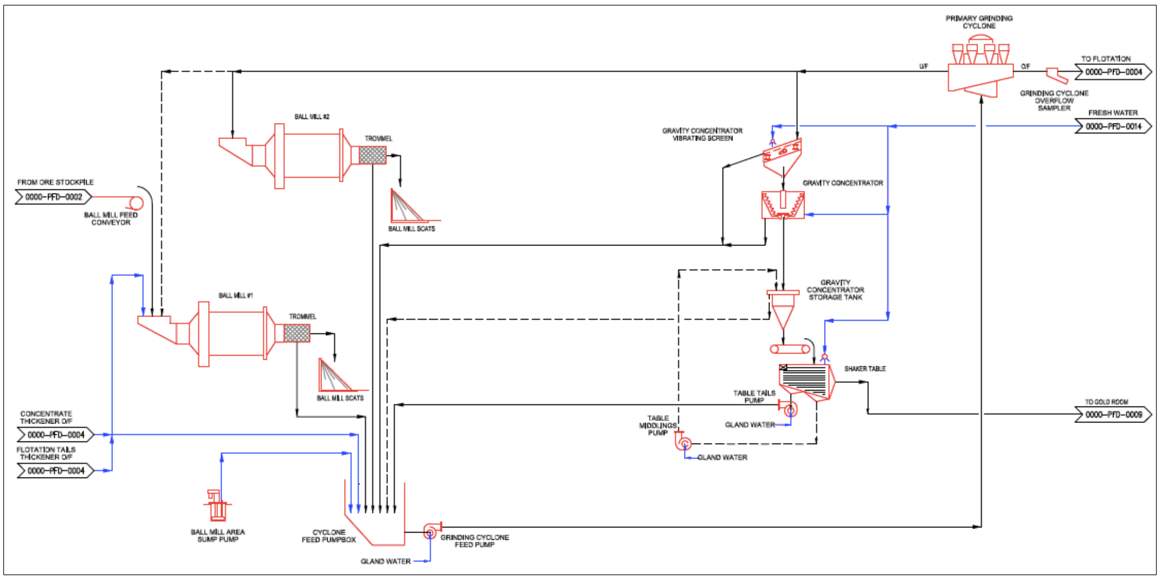
A self cleaning magnet will be situated on the vibrating secondary screen feed conveyor after the jaw crusher, which will remove any tramp metal from the belt, while a weigh scale on the ball mill feed conveyor will allow the operator to monitor conveyor loading and track material inventory.

17.5 Grinding Circuit

Table 17-3: Grinding Circuit Equipment Summary

Description	Unit	Value
Ball Mill Dimensions (1° & 2°)	m x m	3.0 (D) x 4.6 (EGL)
Ball Mill Power (1° & 2°)	kW / hp	522 / 700
Ball Mill Cyclone Diameter	mm	254
Ball Mill No. of Operating Cyclones	-	3
Ball Mill No. of Standby Cyclones	-	1
Gravity Concentrator Capacity	t/h	35

Figure 17-4: Grinding Circuit



The grinding circuit consists of an open circuit primary ball mill and a secondary ball mill and gravity circuit in closed circuit with the hydrocyclones. The circuit operates 24hr/day at an assumed mechanical availability of 91 percent with an annual throughput rate of 219,000tpa.

Mill sizing has been calculated based on the 75th percentile of the Bond Work test results, conducted through the various metallurgical campaigns. It is recommended that additional work should be undertaken in the next phase of engineering to confirm hardness indices and mill sizing parameters.

The primary mill is fed from the stockpile by dual feeders arranged in parallel at a new feed rate of 27-28t/h. The ball mill feed conveyor transports the feed to the primary mill feed chute where it joins with thickener overflow. Water will be added to both ball mills to maintain the charge in the mill at a constant slurry density. Slurry will overflow from the ball mills through a trommel attached to the ball mill discharge. The mills trommel screen oversize will overflow into a trash bin for removal from the system, while the underflow from both mills will discharge into a combined ball mill discharge box. The ball mills have been sized identically which will reduce spares inventory requirements. The mill size calculated has a diameter of 3.0mϕ and an effective grinding length (EGL) of 4.6m, and both mills are equipped with a 522kW (700hp) motor. The estimated power draw range for both mills is 425-450kW.

The ball mill cyclone feed pump will generally run at a fixed speed, being equipped with a VFD to select such to feed three operating cyclones. Operating the cyclone feed pumps at a fixed speed will regulate pressure within the cyclones and help optimise the cyclone efficiency. The ball mill hydrocyclone cluster will classify the feed slurry into coarse and fine fractions with the coarse underflow feeding the secondary ball mill for additional grinding or treatment in a gravity separation circuit for gold recovery. The ball mill hydrocyclones will be designed with a 300% recirculating load. The coarse cyclone underflow has been designed to allow recirculation to the primary mill should extra grinding be required. The cyclone overflow, of 80% passing 75µm, will flow by gravity to the flotation circuit.

The ball mill discharge box or cyclone feed pump box level control will be controlled by addition of dilution water. Any required dilution water will offset any water used in the secondary ball mill.

The single gravity concentrator is designed to remove free gold and silver particles from the slurry flow and will be located adjacent to the grinding area for efficient slurry transfer. Approximately 25 percent of the cyclone underflow is diverted to feed the gravity concentrator. Testwork indicates that approximately 38% of the gold will be recovered to a gravity concentrate. The concentrator is fed from the trash screen underflow, which removes oversized particles and dilutes the feed slurry to approximately 40 percent solids. The gravity concentrator tailings are discharged to the ball mill discharge box, which feeds the hydrocyclones. The concentrate is currently assumed to report the concentrating table, which will further reduce the mass while upgrading the concentrate grade for direct smelt to doré. Tailings from the secondary concentrate cleaning will be pumped back to the ball mill discharge box.

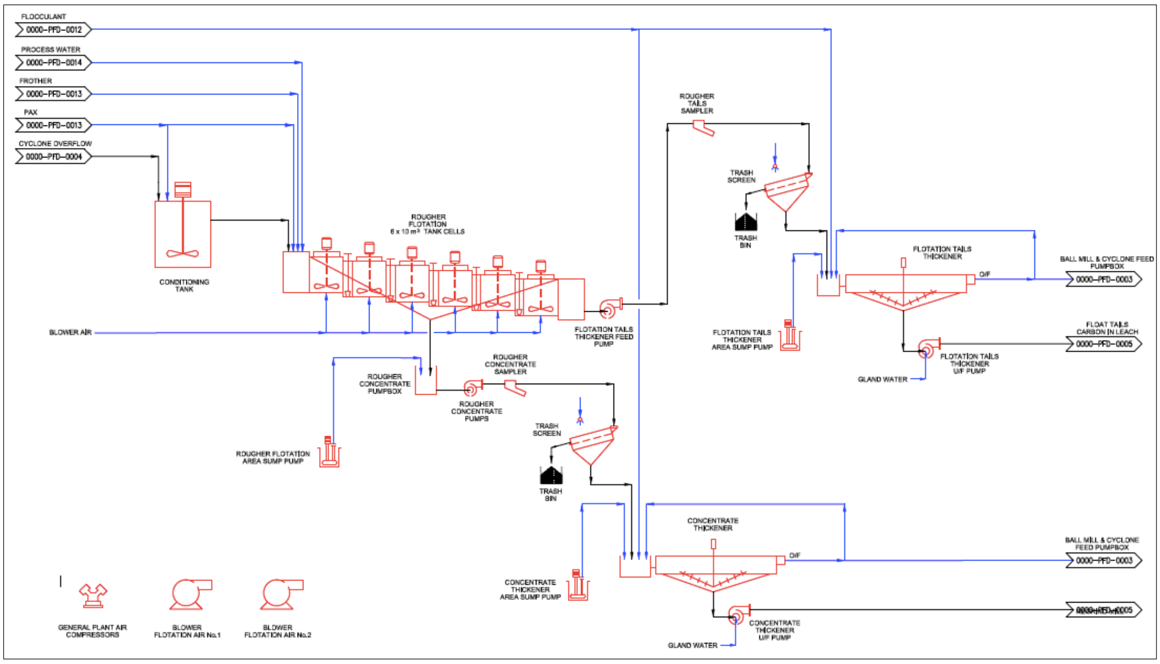
The grinding circuit will be contained within a sloped, concrete floor area equipped with sump pumps to transfer sump contents to the ball mill discharge box.

17.6 Flotation

Table 17-4: Flotation & Thickening Equipment Summary

Description	Unit	Value
Conditioning Tank Residence	min	12
No. of Rougher Flotation Cells	-	6
Flotation Cell Size	m^3	10
Flotation Concentrate Pre-leach Thickener Diameter	m	6
Flotation Tailings Pre-leach Thickener Diameter	m	7

Figure 17-5: Flotation and Thickening Circuit



The flotation circuit was sized based on testwork performed by METCON and AMMTEC, (refer Section 13). Because of the preliminary nature of the testwork, a scale-up factor of 2.5 was applied to the flotation time used to calculate the cell sizes. Common scale-up factors for sulphide rougher flotation range from 2 to 3, depending on amount of testwork undertaken and the level of confidence in those results.

The circuit consists of a conditioning tank with 12 minutes of conditioning time, and a single bank of six 10m^3 self-aspirating flotation cells arranged in series. This concept provides more than sufficient retention time as needed according to the limited testwork.

The material floats well at a natural pH of approximately 7.6. The collecting agent, potassium amyl-xanthate, or PAX, is added to the cyclone overflow slurry in an agitated conditioning tank prior to the flotation circuit. A frother, similar to Interfroth 50 or methyl-isobutyl carbinol (MIBC), is added as needed to each set of cells to enhance froth production. The overflowing concentrate from the flotation cells is collected in launders and reports through a trash screen to the flotation concentrate thickener.

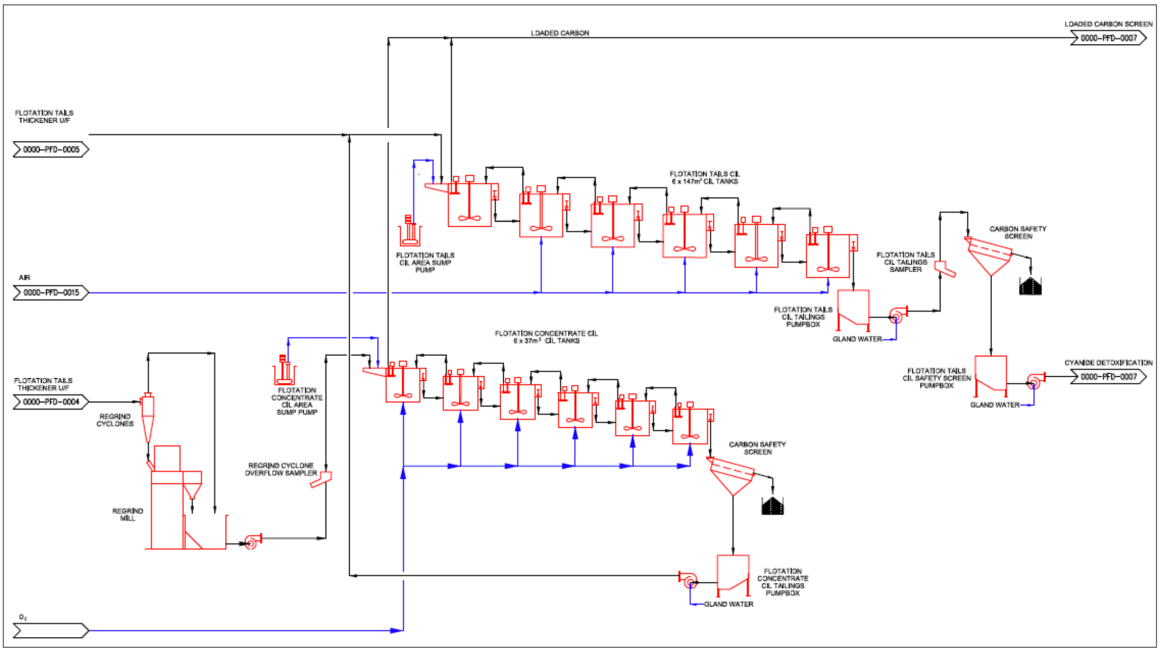
The tailings from the flotation circuit reports to the flotation tailings thickener. The thickener underflow is fed to the flotation tailings CIL circuit.

17.7 Carbon-In-Leach

Table 17-5: CIL Equipment Summary

Description	Unit	Value
Regrind Mill Power	kW	190
Regrind Cyclone Size	mm	152
No. of Float Concentrate CIL Tanks	-	6
Float Concentrate CIL Tank Volume	m^3	37
Float Concentrate CIL Residence	hr	24
No. of Float Tails CIL Tanks	-	6
Float Tails CIL Tank Volume	m^3	214
Float Tails CIL Residence	hr	24
No. of CN Detox Tanks	-	2
CN Detox Tank Size	m x m	4.6 x 5.1 (Dia. x Ht.)

Figure 17-6: Carbon-In-Leach (CIL) Circuit



The float concentrate thickened underflow proceeds to the regrind circuit while the overflow water is returned to the grinding circuit.

Testwork performed on concentrate indicates that regrinding to 20um P80 or less is necessary to extract the precious metals from the sulphide concentrate. The flowsheet proposes stirred mill circuit for regrinding the concentrate. The cyclones are fed from the concentrate thickener underflow. The underflow reports to the regrind mill while the overflow is combined with the regrind mill discharge and feeds the float concentrate CIL circuit. For this study, the target regrind size is 80 percent passing 20 microns, however, mineralogy indicates that the concentrate regrind size may need to be finer. Additional studies are required in the next phase of engineering to more clearly establish design parameters for the regrind circuit.

After regrinding, the slurry reports to CIL. CIL refers to a process where cyanide is added to slurries in the presence of carbon.

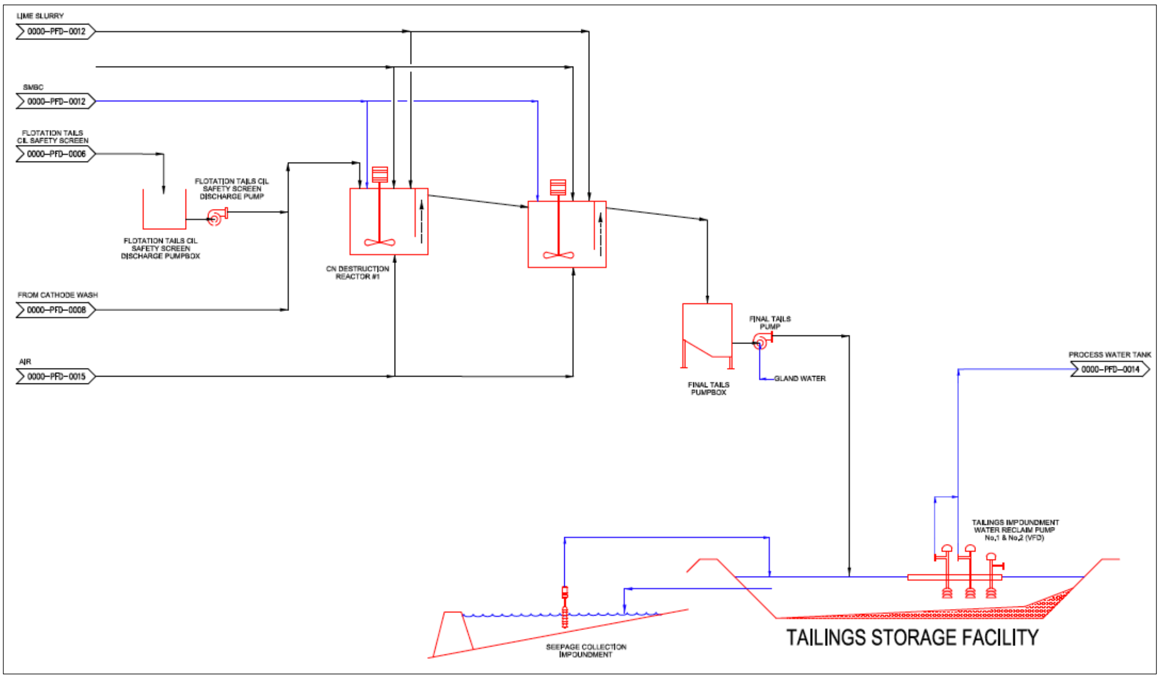
Slaked lime slurry will be added to the first and second leach tanks to maintain protective alkalinity at a design pH of 11.0 to prevent evolution of hydrogen cyanide gas (HCN).

In both the float concentrate CIL and float tailings CIL, a set of six tanks operating in a single train, provides 24 hours of leaching for the dissolution of precious metals. Both circuits will be arranged with overflow tanks complete with agitators and carbon screens. The slurry moves downstream by gravity and the carbon is pumped in a counter current arrangement. This type of carbon movement allows the most active carbon to be in contact with the lowest concentration solutions, improving the adsorption of metals to the carbon surfaces.

Oxygen will be sparged at the bottom of the float concentrate CIL tanks at an aeration rate of approximately 0.05 $Nm^3/h/m^3$. Process air will be added to the six float tailings CIL tanks. The sodium cyanide loop will be able to dose sodium cyanide solution to any of the CIL tanks to maintain cyanide concentrations through both trains. The float concentrate CIL tailings will report to the feed distribution box for the float tails CIL circuit, in order to achieve additional precious metal recovery from that stream through the float tails CIL circuit.

In both CIL circuits, the initial CIL tank carbon is pumped and screened from the slurry and will report to the absorption, desorption, and refining (ADR) plant.

Figure 17-7: Cyanide Detoxification Circuit



The combined leached flotation tailings will report to the cyanide destruction circuit before being pumped to the downstream tailings storage facility (TSF). A trade-off for a cyanide recovery thickener prior to detoxification is recommended for the next phase of study. Un-optimized detoxification testwork has been performed which indicates the SO_2/Air detoxification process can reduce the residual cyanide level to less than 2ppm CN_{WAD}. Optimization testwork will be performed in the next phase.

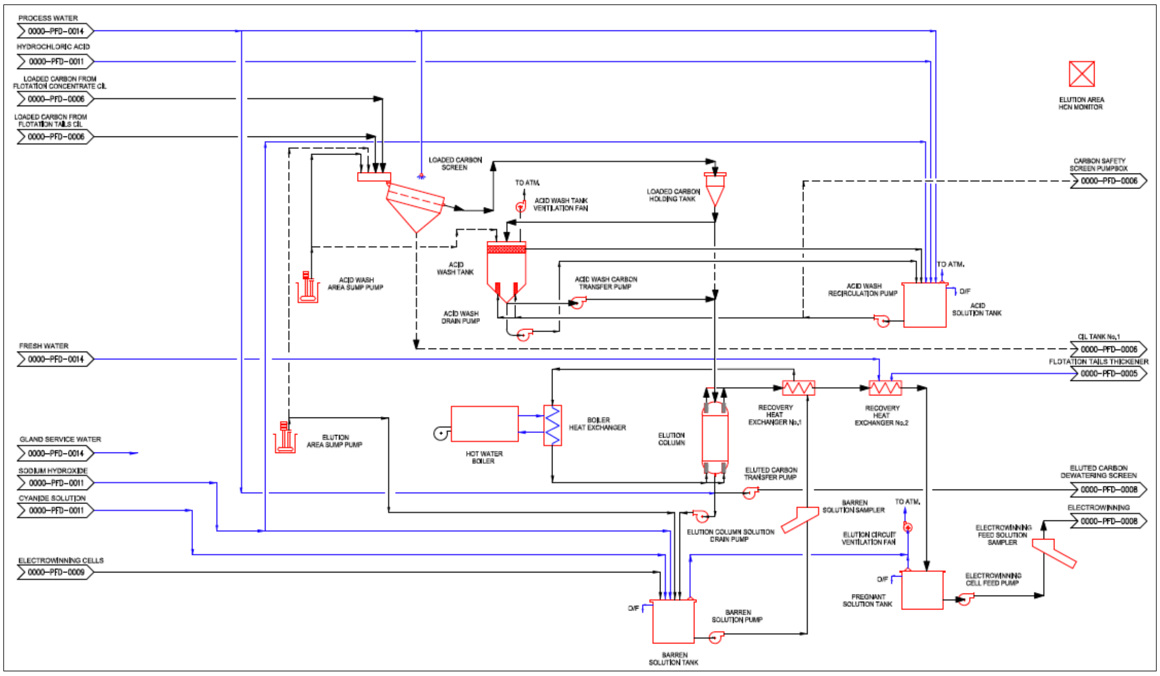
To capture any potential process spills, the CIL tanks will be contained by bund calculated at 1.1 times the tank volume. Platforms and walkways will be provided on top of the CIL tanks in both circuits to access equipment for maintenance and for routine operator rounds.

17.8 Adsorption, Desorption, Recovery (ADR)

Table 17-6: ADR Equipment Summary

Description	Unit	Value
Regeneration Kiln Sizing	t/d	1.0
No. Carbon Circuit EW Cells	-	2
Carbon Strip Cycles per Week	-	3

Figure 17-8: Acid Wash and Elution Circuit

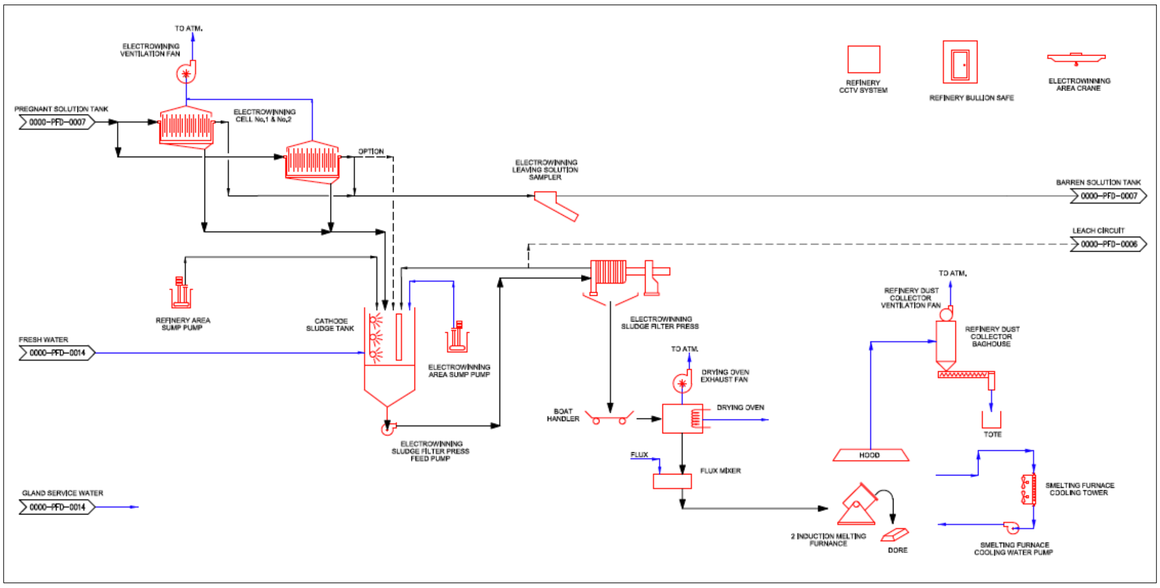


The screened carbon is pumped to the ADR plant. De-watered, loaded carbon will be washed with Hydrochloric (HCl) acid solution in the acid wash tank, and typically removes calcium deposits, some fine iron particles, silica, magnesium and sodium salts.

The carbon will initially be rinsed with fresh water. HCl acid will be pumped from the acid wash circulation tank, upward through the acid wash vessel and overflow back to the acid wash circulation tank. The carbon will then again be rinsed with fresh water. It is typical that the process steps where acid is used in the process facility be covered and vented to evacuate any hydrogen cyanide (HCN) gas that may potentially form where the pH could decrease below the target value.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel to the elution vessel. Under normal operation, only one elution will take place every second day. The strip vessel will be a carbon steel tank designed to contain approximately 2.0 t of carbon. During the strip cycle, solution containing approximately 1 % sodium hydroxide and 0.1-0.2 % sodium cyanide, at a temperature of 138-140 °C (280-284 °F) and 450 kPa (65psi) will be circulated through the strip vessel at approximately 2.0 bed volumes per hour. Solution exiting the top of the elution vessel will be cooled by the heat recovery heat exchanger.

Figure 17-9: Electrowinning and Refining Circuit

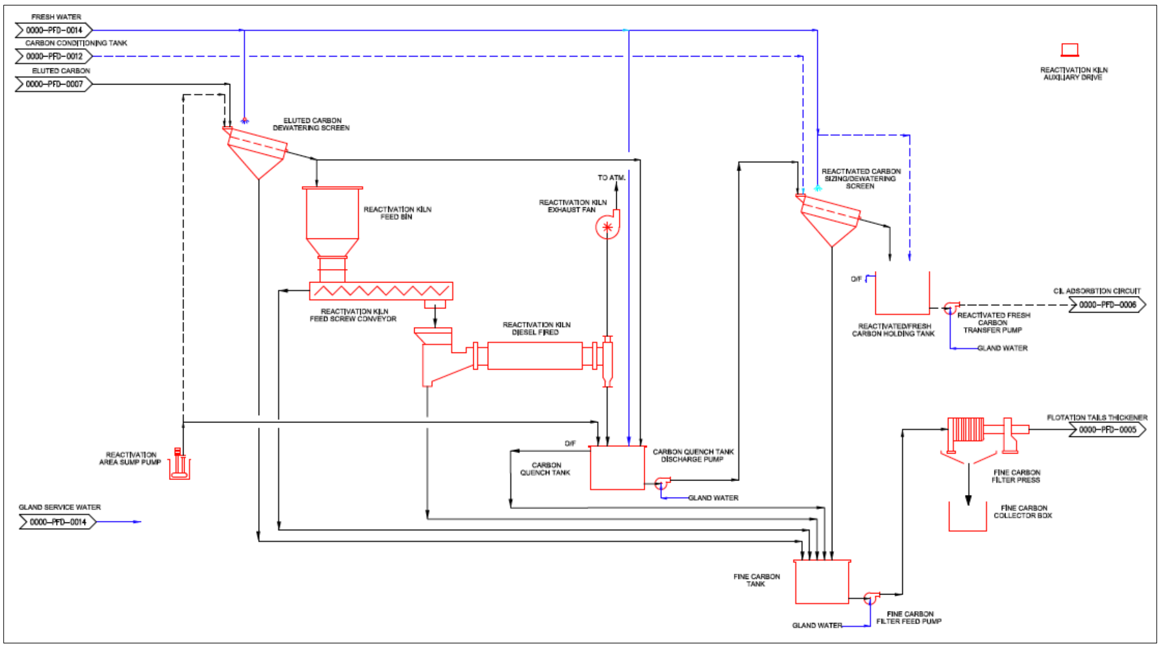


Barren solution flows via the heat exchanger and circulates through the strip column from the bottom of the vessel and the now loaded solution exits the top of the vessel and passes through the heat exchangers to the pregnant solution tank, prior to being sent to the gold room/refinery for electrowinning to produce a gold sludge. Resulting barren solution will be pumped back into the barren solution tank for reuse. Gold rich sludge will be washed from the steel cathodes into the sludge holding tank.

Periodically, the sludge will be drained, filtered, dried, mixed with fluxes and smelted in a diesel powered direct fire furnace to produce gold doré. The gold doré will be stored in a safety vault all within a secure and supervised area.

The reagent storage and sump pumps in the refinery will return any potential spills and clean-up solutions to the process.

Figure 17-10: Carbon Regeneration Circuit



The stripped carbon will then be transferred from the elution vessel to the kiln dewatering screen. The kiln dewatering screen also removes fine carbon particles. Oversize from the screen will discharge to the regeneration kiln feed hopper. A diesel fired horizontal kiln with residual heat dryer will be utilized to treat the carbon. The regeneration kiln discharge transfers to the carbon quench tank by gravity, and is cooled by fresh water prior to being pumped back into the processing circuit. New carbon, required due to carbon attrition losses, is added to the carbon attrition tank along with fresh water to mix and will be pumped to the processing circuit as required.

Other metals will also load on the carbon, most notably silver and copper.

17.9 Energy, Water, Reagents

Makeup water requirements for the plant are unknown at this time. In general, process facilities similar to the proposed Tuvatu plant require one tonne of makeup water per tonne of feed processed. This requirement depends largely on local climate and tailings disposal methods. An overall and process water balance should be performed in support of the next phase of engineering.

A number of reagents will be utilized in the processing facility. Reagent consumption has been based on the limited testwork performed in the METCON variability and AMMTEC developmental testwork. Although the dosages are considered accurate at this level of study, no relevant reagent optimization testwork has been undertaken.

Sodium Cyanide

Sodium cyanide will be used to dissolve the precious metals in the CIL circuits. Sodium cyanide is typically supplied to mine sites as solid briquettes or as flake in 1.1 tonne plastic bulk bags sealed within wooden boxes. A crane will elevate the bag from the box onto the top of the mixing tank where the plastic liner will be broken and the sodium cyanide dissolved with fresh water to make a solution (typically 20% w/w), always maintaining the solution pH greater than 10.0 to avoid generation of HCN gas. The mixed solution will be pumped to the supply tank, where it will be dosed to the CIL circuits. The sodium cyanide area will be designed for certification under the International Cyanide Management Code, where it will be located in a separate contained area isolated from the other reagents. A ventilation system will be required to exhaust any emissions from the tanks to atmosphere.

Lime

Lime supersacs will be broken on a bag breaker above the lime mixing tank, where it will be mixed to produce lime slurry, (typically 15% w/w). The lime slurry produced from quicklime ($CaOH_2$), will be pumped to the adjacent agitated lime slurry holding tank and will be used for pH control in the CIL and cyanide detoxification circuits distributed by pumping. The flotation circuit does not require pH adjustment.

The nominal lime consumption for the process plant CIL circuits will be 0.9kg/t of process plant feed, or approximately 0.54t a day.

PAX

For the proposed Tuvatu process, the collecting agent will be potassium amyl-xanthate (PAX), which will be mixed in an agitated tank and transferred to the storage tank before being pumped to the conditioning tanks and the flotation circuit as required.

PAX consumption based on testwork, is estimated to be 0.1 kg/t of process plant feed.

Frother

The frothing agent will be either MIBC or Interfroth®50 (IF50) which will be mixed and transferred to the storage tank before being pumped to the flotation circuit as required.

Consumption of IF50 based on testwork, is estimated to be 0.014 kg/t of process plant feed.

Flocculant

For Tuvatu, flocculant will be supplied in solid form in 25kg bags and mixed with fresh water. The solution (typically 0.5% w/w) will be aged in an agitated tank transferred to the storage tank before being pumped to the pre-leach thickeners. As is usual in most gold process plant flocculant systems, the flocculant solution concentration will be diluted to 0.05% w/w with process water through an inline mixer at each of the thickeners before dosing.

Flocculant consumption is estimated to be 0.05 kg/t of thickener feed.

Sodium Hydroxide (Caustic)

The carbon stripping process will utilize sodium hydroxide, which will be supplied to the mine site in solid form and typically in one tonne bags. The mixed solution at 20% w/w will be stored before being pumped to the carbon strip circuit.

Hydrochloric Acid

Hydrochloric acid will be supplied in 1000 L totes at 30% v/v strength and used in the ADR circuit. The hydrochloric acid will be pumped from the totes to the acid wash circulation tank where it will be diluted. Discharge from the acid wash tank will be pumped to the detoxification circuit where the acid will be neutralized.

Activated Carbon

New activated carbon will be required to maintain the carbon inventory in the CIL circuit. The activated carbon will be supplied in 0.5 tonne to the mix tank.

The estimated consumption is 0.2-0.3 kg/t of process plant feed.

Sodium Metabisulphite

Sodium Metabisulphite (SMBS) will be delivered in 2 tonne bags and used as the sulphur source for cyanide destruction. SMBS will be supplied in solid form and the dissolved/mixed solution (typically 20% w/w) will be stored before being pumped to the cyanide detoxification circuit.

The estimated consumption is 0.62 kg/t of process plant feed.

Copper Sulphate

Copper sulphate ($CuSO_4$) is used as a catalyst in the SO_2/Air cyanide detoxification process. Copper sulphate will be supplied in solid form and the mixed solution (typically 15% w/w) will be stored before being pumped to the cyanide detoxification circuit.

As there is no testwork to determine the $CuSO_4$ dosage, an industry normal consumption has been assumed at 0.02 kg/t of process plant feed.

Antiscalant

Antiscalant will typically be supplied to the plant in 250L isotainers in liquid form and be pumped directly to the carbon stripping circuit elution barren tank as required. The assumed consumption will be 0.004 kg/t of process plant feed.

LeachWELL

LeachWELL™60X is a reagent grade catalyst formulated to increase the cyanide leach dissolution rate of gold, copper and silver. Although not considered in the reagent suite for this study and often, dependant on mineralogy LeachWELL™ does not benefit, numerous tests indicate that the Tuvatu mineralized material responds well to the addition of LeachWELL™60X and it is recommended to be tested in future metallurgical work.

17.9.1 Water

The process plant will use the following different types of water:

Reclaim: Reclaim water is water that is "reclaimed" from the tailings storage facility will be used in the grinding circuit, flotation, and as dilution in the cyanide detoxification circuit.

Fresh: Fresh water for the process plant will be supplied from the core shed fault and will be used for reagent make-up, gland water, and for cooling water in the oxygen plant.

Process: Overflow from the pre-leach thickeners will be used as process water and supplemented by reclaim water as required. Process and reclaim water will be used in the grinding and flotation circuits, and as wash water on carbon and trash screens.

Fire Water: A fresh/fire water tank will be installed, in which a volume of fire water will always be maintained. The fire water will be supplied to all areas of the mine site including the administration offices, warehouse and mine maintenance facilities, and the process facility.

17.9.2 Air

Air will be supplied to the floatation tails CIL circuit, flotation circuit, plant instrument and process air. The crushing area will have a designated smaller compressor to provide air for that area. Two compressors will supply low-pressure process air to the floatation tails CIL tanks and cyanide detoxification tanks. Two blowers will supply air to the flotation circuit. An instrument and plant air system with compressors and receivers will be provided.

17.9.3 Oxygen Supply

The oxygen plant proposed will be a VSA system and have capacity to produce approximately 3-5tpd of oxygen at 90% purity and will be located next to the flotation concentrate CIL tanks.

17.9.4 Laboratories

The on-site laboratory will include assay and metallurgical laboratories. Environmental samples will be sent off-site for analysis.

The lab modules will include a sample preparation area, fire assay and associated equipment, wet lab, balance and equipment room and office. The assay lab will be on a separate HVAC system while water and electricity will be supplied from the main process generation.

17.10 Recommendations for Continued Work

In the next phase of study, metallurgical samples from the veins in the first 3 years of operation should be included in any testwork programs to better define the precious metal recoveries through this period. The flowsheet proposed in the current study will also need to be confirmed.

Engineering work should include:

- Updated design criteria
- Detailed mass and process water balance calculations
- Equipment sizing and specifications
- Detailed operational and capital cost estimates
- Updated flow sheets for each unit operation

Costs for engineering work can vary greatly, however, an estimated cost range should be between $750 and $850k for the process engineering.

18.0 SITE AND REGIONAL INFRASTRUCTURE

18.1 Tuvatu Site Description

The Tuvatu project and mine lease area is 17km by road from Nadi international airport. The region is well serviced with port facilities at Ba and Lautoka. Lion One also maintains an operations office in Nadi that will continue to service the future site operations.

The Tuvatu project site comprises steep topography coupled with multiple creek lines, which flow into the Sabeto River which supports the community, agricultural and tourist activities downstream. The comparative reduced size of the proposed project will allow the surface infrastructure to be accommodated within the relatively flat areas available such that ground disturbance will be minimized as much as possible and site run-off will be managed readily. Any discharges from the site will be controlled so the water quality in the river system meets proposed guidelines.

The core storage facility and associated infrastructure are maintained on site as is access to service the exploration activities. Emperor Gold Mining Co. Ltd. developed a decline in 1999, but the current mine development designs do not propose to use this as the primary access but rather a new access to the west.

A Fijian Electricity Authority (FEA) transmission line crosses the site, but no surplus power is available in the grid for use by the project.

18.2 Site Development

The approach taken in this study, with a nominal throughput of 600tpd, results in a process plant, tailings storage and mine surface footprint considerably more compact than the facilities proposed in previous studies. The overall site general arrangement is shown in Figure 18.1.

The plant site location will remain similar to previous studies, as no superior topographical location with similar proximity to the mine is available. It is proposed to construct the majority of the plant facilities on cut in an effort to minimize concrete foundation requirements. Small quarries may need to be developed to generate bulk rock for construction activities while a borrow pit will be developed within the final tails storage facility (TSF) impoundment footprint to provide materials for the embankment construction.

Pre-production mined material will be stockpiled on ROM pads and the relatively level area to the east of the plant site should it be required. The mine waste rock will be used in the TSF embankment or as mine fill.

Ground clearing, grubbing, and preparation works, pre-earthworks drainage and sediment control structures will be constructed prior to bulk earthworks activities initiating. Water courses impacting the construction of the facilities will be diverted. Slope stability and site water management will be key considerations in the project design.

18.3 Site Geotechnical Investigations

18.3.1 General

A geotechnical investigation of the TSF footprint and plant site was undertaken as part of this study. The purpose of the geotechnical investigation was to assess the sub-surface founding conditions within the TSF area and plant site. The investigations included the advancement of boreholes by rotary (diamond) drilling and excavation of test pits. The geotechnical investigations were conducted by Entec Limited with review by Knight Piésold Consulting (KP). Geotechnical samples were collected and submitted for laboratory testing. The test pit and borehole locations are shown in Figures 18.2 and 18.3.

The TSF will be founded on residual soils comprising moderately plastic silt up to 11.0 m deep, and underlain predominantly by sedimentary rock (breccia) interbedded with igneous intrusions. The igneous intrusions comprise mostly basalt, with beds of andesite. These materials provide a generally suitable foundation for the construction of the TSF as well as seepage control. Based on this preliminary assessment, the ground conditions at the proposed TSF site are considered to be suitable for the construction of the TSF.

The plant site is underlain by colluvium/residual soils and igneous intrusive monzonite rock. Due to the nature of rock formation, the weathering profile and rock head levels are very variable, undulating from location to location. Overlying the bedrock is primarily residual soils and/or extremely weathered rock with some colluvium in the lower part of the site.

18.3.2 Plant Site Foundation Assessment

A preliminary plant site geotechnical assessment was conducted in order to:

- evaluate founding conditions for the major process plant structures; and
- assess allowable bearing pressures for the major process plant structures.

Bearing capacities and settlements were estimated for major plant site structures using assumed foundation geometries, typical structure loadings, and design parameters. Safe bearing capacities were calculated to be above estimated loadings for all the key plant site structures.

Total settlements were estimated for selected foundations, summarised below:

Table 18.1 Plant Site Foundations

Structure	Foundation Bearing Pressure (kPa)	Estimated Settlements (mm)	Modulus of subgrade reaction (kPa/m)
Primary & Secondary Crushing	120	35 to 45	2,700 to 3,500
Fine Ore Bin	160	15 to 30	5,300 to 11,000
Ball Mill	150	15 to 30	5,000 to 1,000
Leach / CIP Tank Train	80	10 to 20	4,000 to 8,000

The following conclusions were drawn based on this preliminary assessment:

- The ground conditions are variable due to the nature of the geological formation in the area. The subsurface profile comprises a shallow depth of colluvium/residual soil (average thickness of 1.5 m) overlying extremely to highly weathered monzonite. The rock head level varies from 3 m to 12 m within the plant site.
- The proposed plant site is considered as suitable, provided that the top soft to firm clayey silt layer is removed and the formation level is compacted properly.
- A preliminary assessment of excavatability suggests that much of the near surface profile will be excavatable with conventional earthmoving equipment.
- The estimated foundation settlements are considered as not very large. Consolidation and differential settlements should be assessed in detail after confirmation of the plant site formation level.
- The top layer of soil is not suitable for structural fill. Subject to laboratory testing, structural fill may be sourced from the extremely to highly weathered rock layers.

18.4 Roads

18.4.1 Site Access Roads

The Tuvatu site is accessed via Sabeto Road, which follows the Sabeto River valley from its junction with the main road from Nadi international airport. The road is in good condition with the portion closer to the main road being sealed, however the last few kilometers closer to the plant site will potentially require upgrading and appropriate maintenance to accommodate larger loads and to ensure reliable access during the wet season.

Emperor Gold Mining Co. Ltd., commissioned Sinclair, Knight Mertz (SKM) to undertake a road upgrade study for the Feasibility Study completed in 2000. The proposed design remains valid with the route following the existing road alignment, while the upgrades would include widening to consider carriage width plus provision for the tails pipeline bench, elevating the roadway above Sabeto river flood levels while ensuring that stream crossings meet flood rating design standards. Provision for bridge and culvert replacement and upgrades in order to support 20t axle loads through the final 3km’s and general improvements to road design for operational safety is allowed in the project capital estimate.

18.4.2 Site Internal Roads

Haul roads will be constructed to service the plant portal and south portal sites. With the proximity of the portal and mine surface infrastructure to the plant site, traffic control guidelines will be in place given the relatively confined site in order to minimize interactions between haul fleet and light vehicles.

A new road will also be required to link the existing access roads to the tailings dam embankment (this includes improvements to off-site roads, including the run off diversions around the tailings dam as discussed in the relevant section).

18.5 Communications

The existing mobile tower and communications systems at the mine site will be upgraded during construction. An allowance has been made in the capital estimate.
The plant control room and sub-stations will be linked by a control network, with remote stop/starting of the reclaim pumps and seepage collection system via telemetry and back up by radio network for surface communications and emergency response.

18.6 Power Supply

18.6.1 Fiji Grid and Generating Capacity

There is an 11kV transmission line crossing the Tuvatu site from a nearby Fijian Electricity Authority (FEA) hydroelectric plant. Due to the national shortfall in power supply from the grid, despite supplementary thermal generating capacity, the project will generate its own power.

18.6.2 Project Power Requirements

Preliminary project power requirements are shown in Table 18.2 below.

Table 18.2 Predicted Project Power Requirements

Area	Total Connected Power (MW)	Average Power Consumption (MW)
Mine (surface, portal + u/ground)	1.21	0.97
Process plant	2.32	1.91
Infrastructure (allowance)	0.20	0.20
Total	**3.74**	**3.08**

AMC estimated mine power requirements. Canenco estimated process plant and infrastructure power requirements.

18.6.3 Power Generation

A containerized diesel power station, including switchgear and transformers, with 1,500kVA generator units, is proposed to suit the load of approximately 4MW, in an N+1 or N+2 redundancy configuration to ensure reliability of supply and provide enough reserve to start the larger ball mill motors. It is proposed to generate at 6.6kV for the main load centers such as the mill and for distribution to the mine motor control centers (MCC's) at the ventilation shaft and the process plant site portal, with step down to 415V for supply to the low voltage (LV) drives in the plant and as appropriate to supply the underground requirements. Remote facilities such as the TSF and low voltage distribution boards across the site will be equipped with standalone generators.

Power supply costs (including maintenance factors) is based on delivered diesel fuel cost of USD$0.90/L for a total cost of USD$0.24/kWh.

18.7 Water Supply

Reclaim, run-off and mine dewatering will supply water for the project site.

18.7.1 Raw Water

The mine dewatering from underground pumping, will discharge to an intermediate settling pond prior to pumping to the raw water storage tank adjacent to the process plant, and will be used to provide the raw water requirements to service the project needs. The Coreshed Fault, one of the major fault structures identified on site, is a significant water bearing fault and can provide a supply during the dry season. It has been determined that raw water can further be managed by controlling the flows from the tailings facility catchment to allow storage of raw water make-up to the process in the impoundment.

Historical studies were completed on abstraction of water from the Qalibua stream and Sabeto River, but with the reduced project scale, it is unlikely that implementation of these options will be required.

18.7.2 Potable Water

The existing potable water treatment plant (WTP) on site is limited in size and additional requirements will be required to support operations. With the ground water being of good quality and constant supply, treatment of that water for potable use will be restricted to filtration and chlorination. A potable water tank will be located adjacent to the potable WTP at the plant. Potable water will be distributed to points of use within the plant and the main buildings, ablutions and safety showers.

18.7.3 Sewage

Treated effluent, via a low maintenance biological contactor packaged waste-water treatment plant, will discharge to the plant tailings hopper.

18.8 Groundwater Assessment

Knight Piésold conducted a groundwater assessment based on a desktop evaluation of the available geological mining information and dewatering data. The main findings are summarised as follows:

- The existing adit passes through the Coreshed Fault and enters the proposed mine area.
  - As such the adit has the potential to transfer water from the Coreshed Fault into the mine area. The fault should therefore be sealed off within the adit.

- It is envisaged that initial dewatering of the upper part of the Coreshed Fault is best achieved by dewatering the existing adit and monitoring the response in exploration boreholes. Thereafter it is envisaged that the new underground workings should be dewatered using internal pumps within the workings as development occurs.

- Two areas of the underground workings pass through the Coreshed Fault in the north east section of the fault. Inflow from the Coreshed Fault to the tunnels should be pumped or sealed in a similar way to the exploration adit.

- The groundwater temperature will increase with depth; existing data indicates water temperatures up to 40°C.

- Mine discharge water from the fault zones may have a high suspended solids content which will need to be addressed by settlement prior to use as process make up water or discharge to the river.

- Previous pumping from the exploration adit at 4 L/s dewatered the adit to the Coreshed Fault in three months. Based on current information an allowance for pumping from underground at rates up to 20 L/s should be included in the operating costs.

18.9 Tails Storage Facility

18.9.1 Site Characteristics

The Tuvatu project site has a mild tropical maritime climate. The average annual rainfall is 2,202 mm and the average annual evaporation is 2,142 mm. The proposed Tailings Storage Facility (TSF) will occupy a valley with side slopes typically exceeding 25%. The stream at the base forms part of a small water course sloping approximately 1V:15H to the west.

A probabilistic seismic hazard analysis was carried out to determine appropriate seismic design parameters. The estimated peak ground acceleration for the 1 in 475 year earthquake is 0.19 g. The estimated peak ground acceleration for the Maximum Credible Earthquake (MCE) is 0.56 g.

18.9.2 Site Selection

Due to the mountainous terrain of the project area a number of cross-valley storage facilities were considered within a 2 km radius of the proposed plant site. These Tailings Storage Facility (TSF) site options were assessed and eliminated due to sitting within the potential pit extents or having significant catchment area upstream. Therefore the original TSF site was confirmed to be the most appropriate tailings storage option, due to its proximity to the plant site and open pits, ease of surface water diversion within a single water catchment and being slightly down gradient from the plant site.

18.9.3 Risk / Hazard Assessment

The proposed TSF is classified as a ‘HIGH B’ rating according to The Australian National Committee on Large Dams (ANCOLD) guideline on ‘Tailings dams planning, design, construction, operation and closure’ on the basis of a conceptual risk assessment as follows:

- A population at risk (PAR) of 10 – 100 due to the proximity of a local village across a ridge line downstream of the TSF.
- 'MAJOR' severity level of impact category from a large scale failure of the facility.

18.9.4 Tailings Characteristics

Tailings samples were not available for physical and geochemical testing at the time of this design. The following physical characteristics were assumed:

- Water release will be in the order of 50% of the water in slurry, not accounting for rainfall and evaporation.
- A typical achievable density range of between 1.00 and 1.20 t/m³ is expected over the life.

The likely tailings geochemistry was assessed based on a review of existing metallurgical test work and the following conclusion were drawn (subject to future testwork confirmation):

- The tailings were classified as potentially acid forming (High Capacity).
- No information pertaining to the carbonate content of the vein samples is available.
    - Therefore, the availability of neutralising capacity is not known.
- The multi-element analyses indicate high levels of enrichment within the vein samples, particularly gold, silver, arsenic, bismuth, cadmium, molybdenum and tellurium which were found to be significantly to highly enriched in all samples.
- Copper, lead, sulphur and zinc were also found to vary from not enriched to highly enriched, indicating a high degree of variability within the samples.
- The solution samples analysed would not be appropriate for release into surface waters due to elevated cadmium, copper, cyanide, molybdenum, nickel, silver and zinc.

Based on this preliminary evaluation of the tailings geochemistry it was decided that the tailings will be stored with a water cover to limit the tailings oxidation rate to mitigate the formation of acid.

18.9.5 Tailings Storage Facility Design

The TSF is designed as a cross valley impoundment. The facility will be located approximately 2.0 km southwest of the proposed plant site. The embankment will be constructed across a northwest southeast orientated valley with the design utilising ridges which form a natural constriction of the valley to reduce the volume of embankment construction materials required. The TSF general arrangement is shown in Figure 18.2.

The facility has been designed to store a total of 1,200,000 t of tailings at an average process plant discharge rate of 200,000 tpa (600 tpd), with capacity to contain all supernatant and runoff from wet years with an average recurrence interval (ARI) of 1 in 100 years.

The embankment will be constructed in two stages with the core and filter zones being constructed by a specialised earthworks contractor and the structural embankment being progressively constructed by the mining fleet as part of the mine waste operations from the open pits.

The tailings have been classified as potentially acid forming therefore the facility will be operated as a subaqueous facility with the embankment constructed as a water retaining structure. There will be no underdrainage located within the facility as this could allow the tailings to drain, become unsaturated and allow sulphide oxidation to occur. Water will be recycled from the TSF by a floating barge located in a valley to the southeast of the TSF embankment, from which process water will be pumped to the plant for reuse in the process cycle.

A surface water diversion will be constructed around the facility in Stage 1 to control the amount of surface runoff discharging into the TSF basin area. The surface water diversion will reduce the TSF upstream catchment area from 78 ha to approximately 12 ha. A water conditioning pond will be constructed to store water for monitoring and treatment prior to release into the surface water diversion system.

A spillway will be constructed to minimise the risk of overtopping of the embankment. It will be located southwest of the embankment and will be designed to discharge flows which result from storm events in excess of the design basis.

To reduce seepage losses from the facility a number of seepage control measures have been incorporated into the design. The site investigation results indicate vertical seepage losses out of the basin of the facility are expected to be relatively low. The seepage control measures include a cut-off trench and embankment drainage and a HDPE liner in the tailings basin area.

Tailings will be pumped from the processing plant to the TSF via the tailings delivery line. The tailings pipeline will be encapsulated within a separate pipeline in a pipeline to prevent any spillage. A series of dump pockets will be excavated at low points along the line to allow detection and containment of any tailings line failures within the containment pipe. Tailings will be deposited into the facility via a series of spigots located along the embankment crest throughout the life of mine. Two single point discharges will be brought online in Year 3/4 to facilitate the closure plan. The single point discharges will be located in the northern extent of the facility to slope the tailings beach towards the final spillway location.

18.9.6 Construction Materials

TSF embankments will consist of an upstream low permeability zone (Zone A), a filter media (Zone B), a downstream structural zone (Zone C) and a chimney drain. A typical embankment cross section is shown in Figure 18.4 and typical material specifications are summarised below:

- Zone A material will be selected mine waste or won from the basin area with a compacted hydraulic conductivity not greater than 1×10^{-8} m/s.
- Zone B material will be a transition / filter zone.

- Zone C material is expected to be any mine waste, placed as it becomes available on a continuous basis provided it is in advance of Zone A and the tailings levels. It is expected that sufficient mine waste will be available for the life of mine. The material will be hauled, spread and traffic compacted by the mining contractor.

- The Zone F material (central chimney drain): Drainage sand.

Construction of the TSF will commence in the dry season before the current stage is full, so that there is adequate storage volume available throughout the life of mine.

18.9.7 Stability Assessment

The stability of the proposed TSF embankment was assessed under both static and seismic loading conditions using 3D limit state equilibrium methods. Analysis was conducted for both upstream and downstream failures.

The stability of the embankment was analysed under a range of conditions representing various time periods during operation. The results of the analysis indicate that the embankment is stable under the range of conditions analysed.

18.9.8 Monitoring

A monitoring programme for the TSF will be developed to monitor for any potential problems which may arise during operations.

Three groundwater monitoring stations will be installed downstream of the TSF to facilitate early detection of changes in groundwater level and/or quality, both during operation and following decommissioning. Each monitoring bore station consists of one shallow bore, extending to a depth of approximately 5 to 10 m, and one deep bore extending to a depth of approximately 20 m or 5 m below the ground water table. It is recommended that the boreholes are constructed before commissioning the TSF to accumulate baseline data specific to the storage location.

Standpipe piezometers will be installed at five locations along the crest to monitor pore water pressures within the embankments to ensure that stability is not compromised. The piezometers will be monitored at regular intervals and any rises in water level noted.

Survey pins will be installed at regular intervals along the TSF crest and downstream face.

18.9.9 Rehabilitation

The TSF will be designed to be decommissioned as a wet facility. A pond will be maintained on the facility on decommissioning by increasing the size of the TSF catchment by removal of the surface water diversion channel located upstream of the TSF. The catchment of the TSF will increase from 12 ha during operation to 78 ha at closure. A closure spillway will be constructed to carry flows resulting from the probable maximum precipitation (PMP) event.

The TSF embankment will have an overall final slope of 1V:3.00H (18.5°) with inter bench slopes of 1V:2.5H (22°). Benches will be constructed every ten vertical metres and will slope back into the embankment at a minimum grade of 5%. It is recommended that revegetation trials are conducted during operation to establish suitable vegetation for rehabilitation. The steep nature of the natural terrain indicates that many local plant species should be well adapted to the steep topography.

18.10 Site Water Management

The tailings storage facility will have a surface water diversion system upstream of the basin area comprising a surface water diversion pond and diversion channels to reduce surface water runoff into the tailings storage basin area. The TSF will be designed as a ‘treatment and release’ facility. Over the life of the mine, supernatant water quality will be continuously monitored and discharged to a water conditioning pond where it will be treated and then discharged into the surface water diversion pond upon meeting the release criteria.

Based on the high rainfall expected at the site and the requirement for full pond coverage of the tailings to mitigate acid generation, a site water management model for the project was developed, incorporating the following aspects:

- Tailings facility rainfall runoff and evaporation.
- Predicted tailings supernatant release and achieved densities from subaqueous deposition.
- Plant site water demand.
- External stormwater runoff from the upstream catchment.

Under average climatic conditions with an upstream diversion system in place, the TSF will operate slightly water positive during the life of mine. The supernatant pond will generally increase over time and decant return will be sufficient to supply the water demand of the plant. Therefore, an external water supply is not required.

The supernatant pond will cover the deposited tailings over the life of the mine, therefore undrained layer density is expected and ponding against the embankment will occur.

Under extreme wet conditions, additional rainfall will control the required embankment height to prevent any spillway flow.

Construction of the surface water diversion system will be completed prior to constructing the TSF in order to protect the works. It will be beneficial to conduct preparatory works during the wet season to allow embankment construction to commence early in the dry season. To allow excavation of the cut-off trench across the base of the valley a coffer dam will be constructed which will divert surface water runoff into the diversion system which will discharge around the embankment.

A sediment control structure (SCS) will be constructed in the valley downstream of the TSF embankment to reduce the impact of the proposed construction activities on the surface water quality and sediment loadings, thereby limiting disruption of natural drainage (runoff) patterns to natural catchments. Discharge from the SCS will be to the environment downstream. Water collected in the SCS may be used for dust suppression (if suitable).

Surface water runoff from the plant site and ROM pad may also need an SCS. Runoff from the plant site will be collected in the SCS for the removal of sediment, following which it can be pumped back to the plant and used for process make-up water requirements and dust suppression (if suitable), or released upon meeting the release criteria.

The proposed pits will have a berm and / or diversion channel around the perimeter in order to collect any upstream runoff to limit entry of runoff into the pit. Runoff will be classified as undisturbed and will be diverted around the pits into existing watercourses. Surface water collected in the pit and seepage from the pit walls will be collected and be pumped to the plant.

A flow diagram block model has been developed and is shown in Figure 18.5.

18.11 Rehabilitation and Closure

18.11.1 General

The project will plan for closure commencing from the early stages of project development so that the project area is left in an environmentally acceptable condition and reconstructed landforms are safe and stable after completion of the mining operations. The plan will ensure that progressive and final rehabilitation activities achieve the long term goals of stable landforms, maintenance of public safety, provision of compatible land use and sustainable ecosystems.

18.11.2 Schedule

A detailed mine closure schedule will need to be established prior to operations ceasing. The schedule will include the major mine closure activities Including demolition of the process plant, rehabilitation of the TSF, pit closures and removal of contaminated material to be undertaken progressively both during and beyond the final year of operations.

The duration of monitoring works beyond this preliminary schedule would be assessed during operations and based on data obtained.

18.12 Logistics and Other Infrastructure

18.12.1 Administration, Security and Emergency Medical Facilities

The administration building will be located adjacent to the gate-house, providing ready access to the process plant, warehouse, maintenance facilities and the mine dry and services area.

The gate-house, with manned boom gate, will include a medical room which will manage First Aid. The site ambulance and fire response vehicles will be located adjacent in dedicated parking bays. Buildings are planned to have smoke, carbon monoxide and heat detectors, overhead sprinklers, hydrants / hoses and appropriate chemical fire extinguishers

18.12.2 Maintenance Shop and Warehouse

The maintenance shop and site warehouse access will be from within the secure plant compound. A warehouse facility, including secure yard storage for lubricants and oils, will be adjacent to the main workshop structure.

18.12.3 Mine Infrastructure

The mining operations will require surface facilities such as the mine dry (personnel facilities), the mine maintenance workshop, including a vehicle wash down pad and explosives magazine. Explosives would be stored at a secured and monitored site located away from the main plant and populated, high traffic areas. All infrastructure items required for storing, loading and unloading of explosive are assumed to be owned by the contractor providing this service.

18.12.4 Accommodation

Proximity to Nadi and local villages provides sufficient accommodation for contractors. Local landowners will be contracted to provide transportation of workers to site. It is not envisaged construction accommodation will be required on site, as the workforce will be sourced from local communities, with only key components of the contractor's workforce mobilizing from elsewhere.

18.12.5 Laydown

Land availability, suitable for equipment and materials laydown on site is limited. Scheduling of the timing of construction site deliveries will need to be well managed. It is recommended that the project establish additional receiving areas in suitable locations proximate to the site, where items can be issued from as required for the construction.

18.12.6 Fencing

The process plant area, mine portal and TSF will be fenced, with double security fencing around the warehouse yard and gold room with 1.8 m high chain link fences.

18.12.7 Sea Freight

All sea freight for the project will arrive in Fiji via the port of Lautoka, located approximately 40 km by road from the project site.

The port of Lautoka is one of two major ports on Viti Levu, Fiji's main island, the other being at Suva in the southeast of the island. The port of Lautoka is capable of handling bulk, containerized and break-bulk cargo, and has a 100 t shore crane to assist with vessel unloading. Regular freight services call at Lautoka, and limited issues are anticipated in landing any of the cargo required for the proposed project at Lautoka from any point of origin.

18.12.8 Ground Transport

Transport to site from the port of Lautoka is approximately 40 km by road. The last 12 km (approximately) to site is unsealed, but considered passable in all but extreme weather conditions. Up to 32 t loads may be transported on a normal 40' semi-trailer. Any loads above 32 t must be transported via low loader, of which there are several available in Fiji.

There are not anticipated to be any issues with weight restrictions on bridges on the route to site. There are, however some height restrictions due to low power lines and loads up to 4.2 m high may travel without restriction. Above this height, a permit from the FEA will be required.

18.12.9 Air Freight

Any items required to be air-freighted to the project for logistical or scheduling reasons may be routed via Nadi international airport, located approximately 17 km by road from the project site. Regular international passenger and freight services arrive at the airport.

18.12.10 Logistics Service Providers

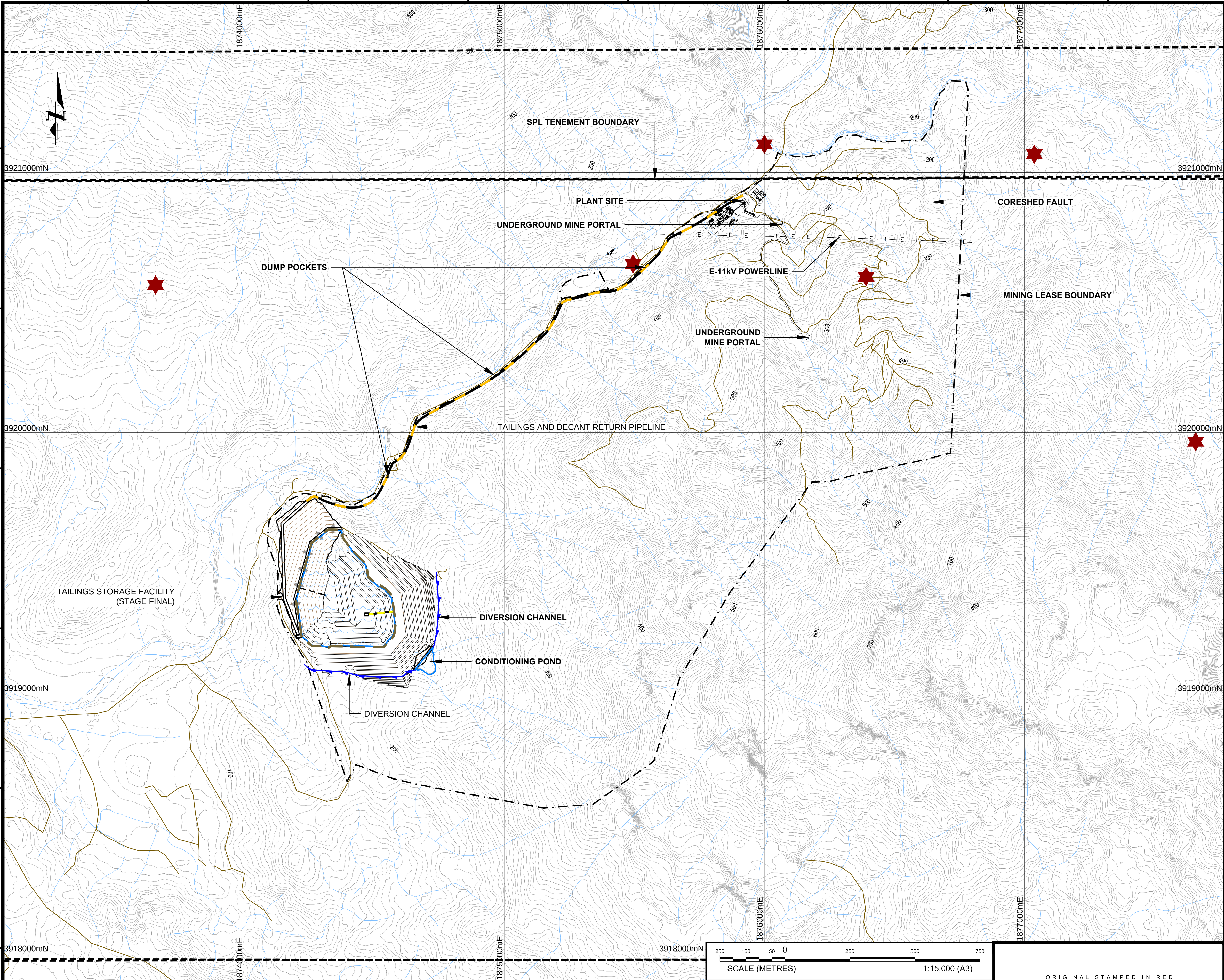
There are several established logistics / transport / customs clearance agents in Fiji that could manage all project freight from either port to delivery at the project site.

Carpenter Fiji Ltd. (Carpenters) provides such a service having operated in Fiji for years. Carpenters maintain freight storage at the Lautoko port and an office at Nadi. It is proposed that Carpenter's could look after all customs clearance and assist with permit applications for special cargo.

18.12.11 Import Duty

It is assumed that Lion One would apply for import duty exemption from the Fijian Customs authorities for all project related cargo.

Figure 18.1 Site General Arrangement



LEGEND:

- SITE OF CULTURAL SIGNIFICANCE
- WATER COURSE
- ROAD/TRACK
- MINING LEASE BOUNDARY
- SPL TENEMENT BOUNDARY
- E-11kv POWERLINE
- TAILINGS & DECANT RETURN PIPELINE CORRIDOR
- SUPERNATANT POND
- DIVERSION CHANNEL

NOTES:

1. ALL COORDINATES SHOWN IN FIJI MAP GRID (WGS72).
2. 5m CONTOUR INTERVALS SHOWN.
3. FINAL TAILINGS STORAGE FACILITY SHOWN. EMBANKMENT CREST AT 123.7 mR.L..

REV	DATE	DESCRIPTION	DRN	CKD	DESIGN	APP	CLIENT APP
A	09/12/2015	ISSUED FOR REVIEW	JJT				
B	22/05/2015	ISSUED FOR PRE-FEASIBILITY DESIGN UPDATE	PRK	MJS	MJS	BAS	
C	22/06/2015	REVISED PLANT SITE LAYOUT	PRK	BAS	BAS	BAS	

REVISIONS

DRAWING No.	DRAWING TITLE

REFERENCES

SHEET SIZE A3

ORIGINAL STAMPED IN RED

DATE	CHECKED	DESIGNED	APPROVED	CLIENT APP'D

Knight Piésold CONSULTING

CONSULT AUSTRALIA Member Firm

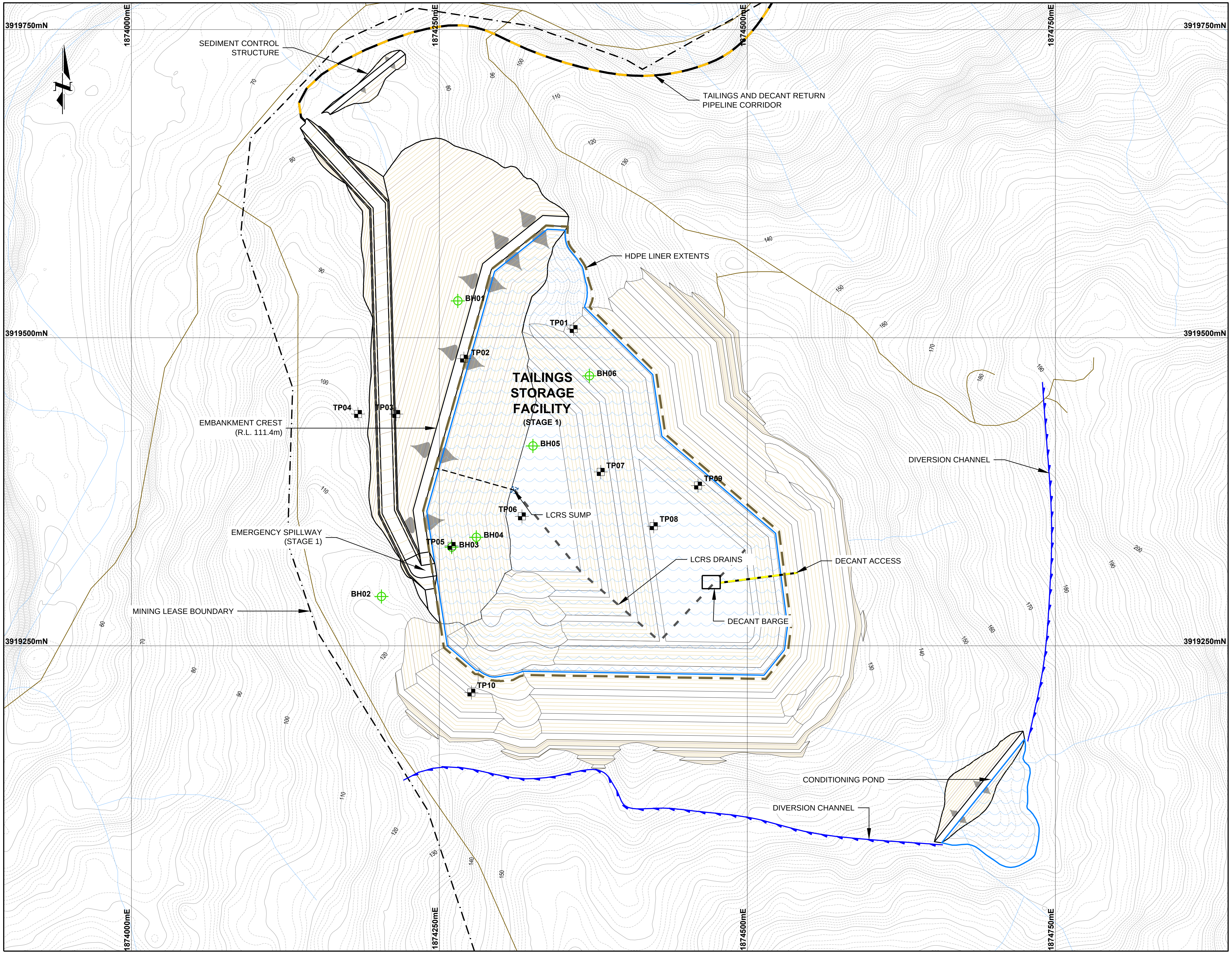
THIRD ANGLE PROJECTION

LION ONE METALS LIMITED
TUVATU GOLD PROJECT
TAILINGS STORAGE FACILITY
SITE GENERAL ARRANGEMENT

SCALE 1:15,000
STATUS IFR
DRAWING NUMBER **701-080-C200-004**
REV **C**

287 410

Figure 18.2 TSF General Arrangement



LEGEND:

- WATER COURSE
- ROAD/TRACK
- MINING LEASE BOUNDARY
- TAILINGS & DECANT RETURN PIPELINE CORRIDOR
- SUPERNATANT POND
- HDPE LINER EXTENT
- DIVERSION CHANNEL
- LCRS DRAIN
- RISER PIPE
- COMPLETED BOREHOLE
- COMPLETED TEST PIT

NOTES:

1. ALL COORDINATES SHOWN IN FIJI MAP GRID (WGS72).
2. 1m CONTOUR INTERVALS SHOWN.
3. STAGE 1 TAILINGS STORAGE FACILITY SHOWN. EMBANKMENT CREST AT 111.4mR.L..

COMPLETED BOREHOLE LOCATIONS

BOREHOLE I.D.	EASTING	NORTHING
BH01	1874265	3919530
BH02	1874203	3919290
BH03	1874260	3919330
BH04	1874280	3919338
BH05	1874326	3919412
BH06	1874372	3919469

COMPLETED TEST PIT LOCATIONS

TEST PIT I.D.	EASTING	NORTHING
TP01	1874359	3919507
TP02	1874270	3919483
TP03	1874215	3919438
TP04	1874184	3919438
TP05	1874260	3919331
TP06	1874317	3919355
TP07	1874381	3919391
TP08	1874424	3919347
TP09	1874460	3919380
TP10	1874276	3919212

160 45 30 15 0 30 60 90 120

SCALE (METRES) 1:3,000 (A3)

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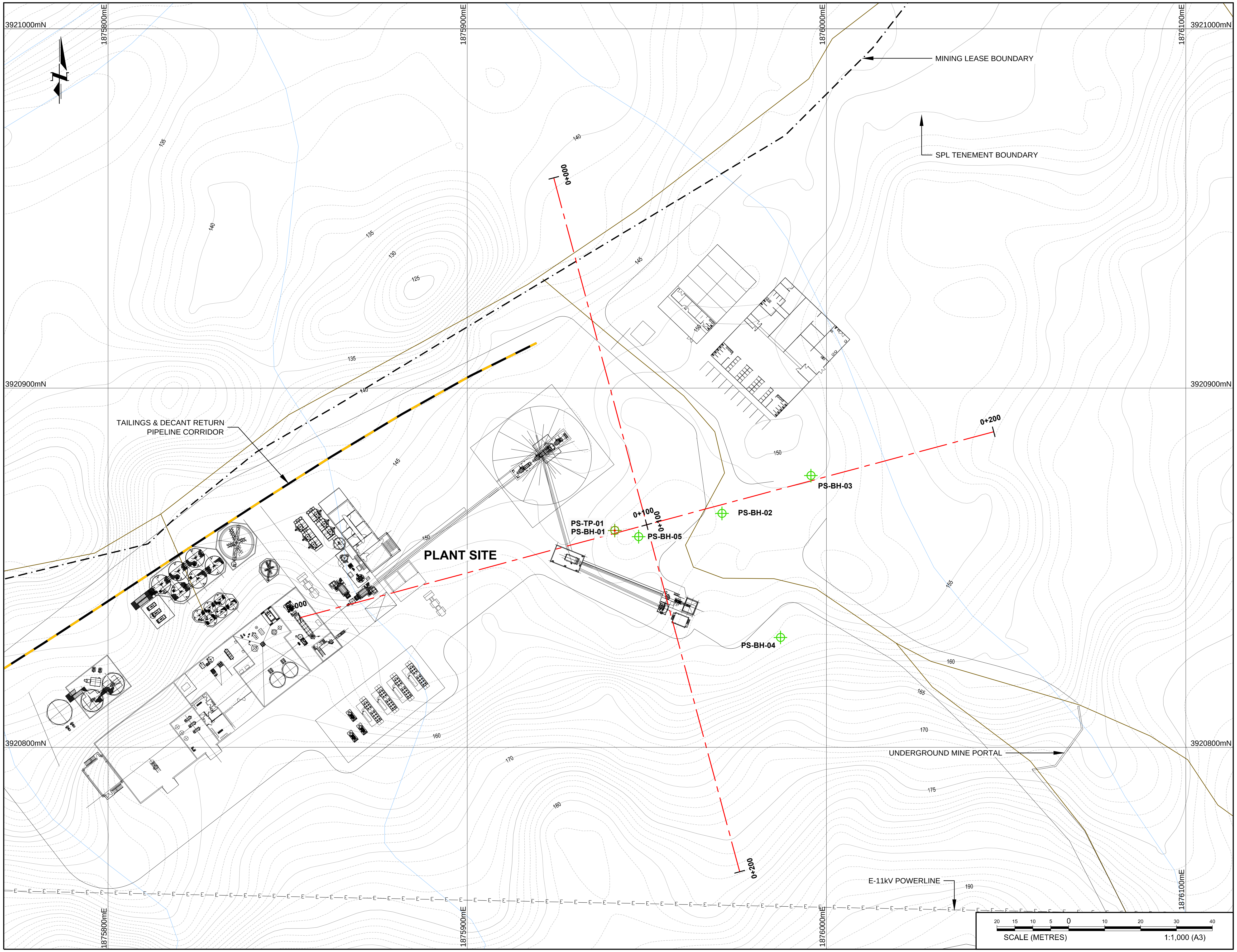
TUVATU GOLD PROJECT
TAILINGS STORAGE FACILITY PRELIMINARY FEASIBILITY STUDY
TSF GENERAL ARRANGEMENT

Figure 18.3 Plant Site General Arrangement



LEGEND:

WATER COURSE

ROAD/TRACK

MINING LEASE BOUNDARY

SPL TENEMENT BOUNDARY

TAILINGS & DECANT RETURN PIPELINE CORRIDOR

COMPLETED BOREHOLES

COMPLETED TEST PITS

NOTES:

1. ALL COORDINATES SHOWN IN FIJI MAP GRID (WGS72).
2. 1m CONTOUR INTERVALS SHOWN.

COMPLETED BORE HOLE LOCATIONS

BORE HOLE	EASTING	NORTHING
PS-BH-01	1875941	3920860
PS-BH-02	1875971	3920865
PS-BH-03	1875996	3920876
PS-BH-04	1875987	3920831
PS-BH-05	1875948	3920859

COMPLETED TEST PIT LOCATION

TEST PIT	EASTING	NORTHING
PS-TP-01	1875941	3920860

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TUVATU GOLD PROJECT
TAILINGS STORAGE FACILITY PRELIMINARY FEASIBILITY STUDY
PLANT SITE GENERAL ARRANGEMENT

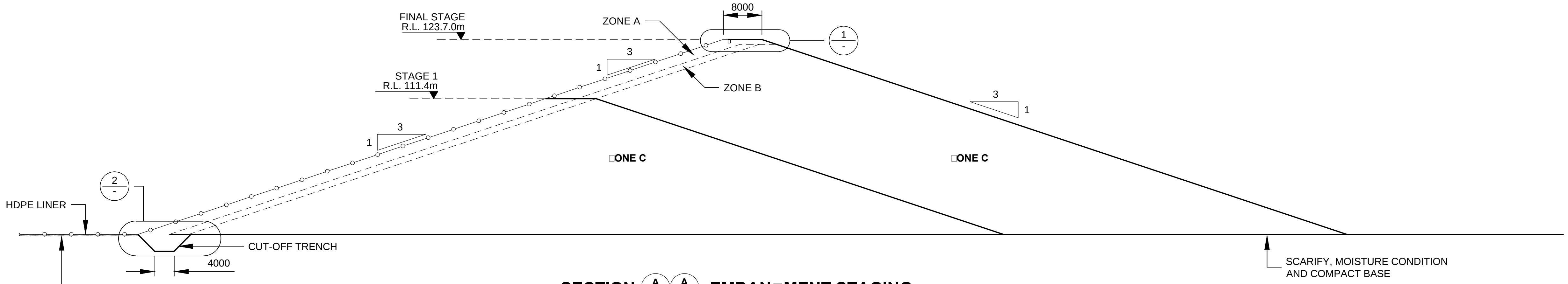
Figure 18.4 Typical Sections and Details

TABLE 1:
□ONE SPECIFICATIONS S□MMARY

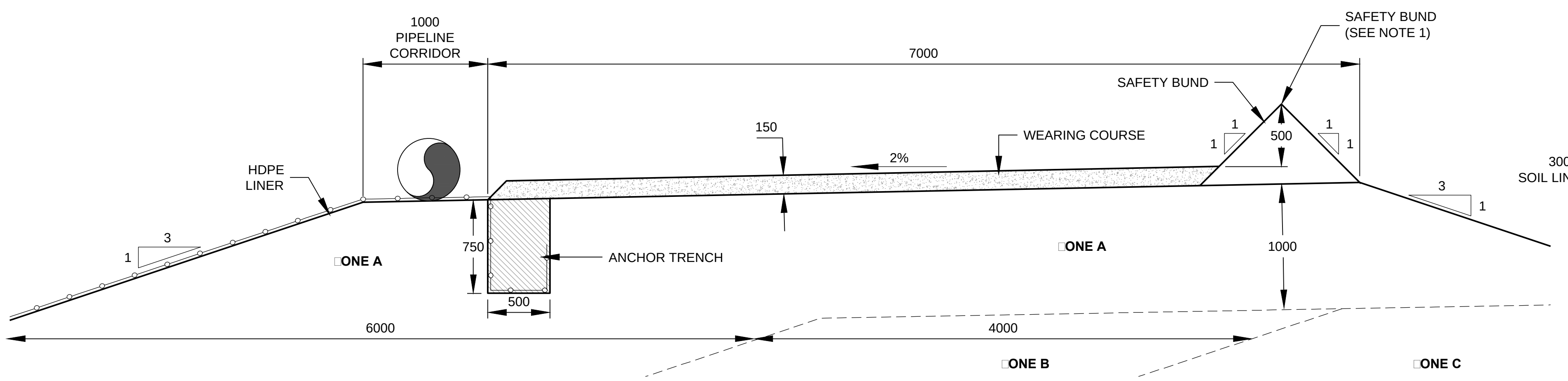
ZONE TYPE	DESCRIPTION	MAXIMUM LAYER THICKNESS	COMPACTION SPECIFICATION	GRADING
ZONE A	LOW PERMEABILITY FILL	300mm	98% OF STANDARD MAXIMUM DRY DENSITY -2%<OMC<+3%	D_{max}=150mm % FINES > 30 PI_{fines}>8
COMPACTED SOIL LINER	LOW PERMEABILITY MATERIAL	300mm	98% OF STANDARD MAXIMUM DRY DENSITY -3%<OMC<+3%	D_{max}=150mm % FINES > 20 PI_{fines}>8
ZONE B	TRANSITION FILTER	500mm	95% OF STANDARD MAXIMUM DRY DENSITY	TRANSITIONA L MATERIAL % FINES > 10
ZONE C	STRUCTURAL FILL	1000mm	TRAFFIC COMPACTED	-

NOTES:

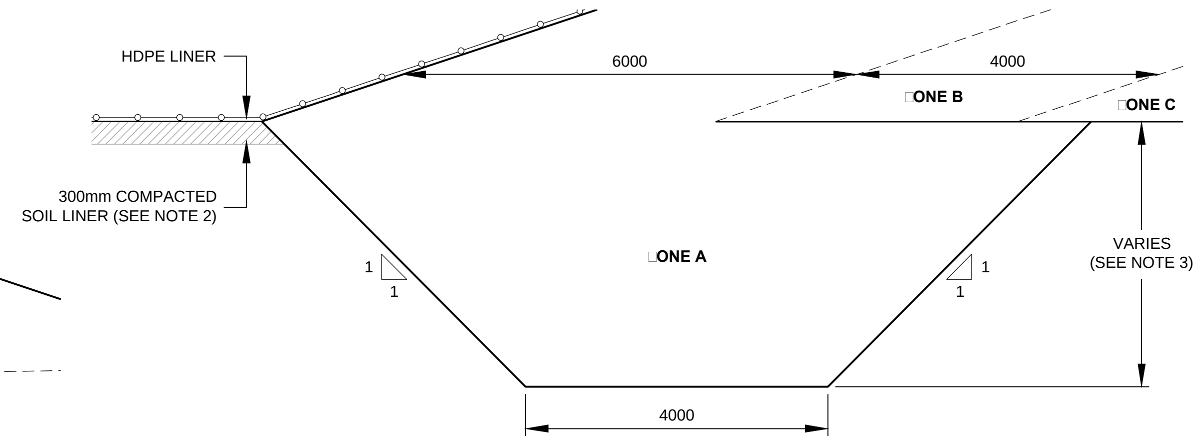
1. SAFETY BUND TO BE CONSTRUCTED OF GENERAL FILL MATERIAL. BREAKS IN UPSTREAM SAFETY BUND AT 50m CENTRES TO ALLOW DRAINAGE OF RAINFALL RUNOFF.
2. COMPACTED SOIL LINER THROUGHOUT TSF BASIN AREA.
3. CUT OFF TRENCH TO BE EXCAVATED 1m (MIN.) INTO COMPETENT MATERIALS AS DETERMINED BY ENGINEER ON SITE.
4. ZONE C PLACEMENT BY MINING FLEET TO SUIT MINE PLAN, PROVIDING IT ACHIEVES STAGE LEVEL PRIOR TO CONSTRUCTION RAISE REQUIREMENT



SECTION A/005 A/006 **EMBAN□MENT STAGING**
SCALE 1:1,000



DETAIL 1 **TYPICAL EMBAN□MENT CREST**
SCALE 1:50



DETAIL 2 **C□T□OFF TRENCH**
SCALE 1:100

REFERENCES DRAWING No.	DRAWING TITLE
701-080-C200-005	GENERAL ARRANGEMENT - STAGE 1
701-080-C200-006	GENERAL ARRANGEMENT - FINAL STAGE

REV	DATE	DESCRIPTION	DRN	CKD	DESIGN	APP	CLIENT APP
A	09/12/2015	ISSUED FOR REVIEW	JJT				
B	22/05/2015	ISSUED FOR PRE-FEASIBILITY DESIGN UPDATE	PRK	MJS	MJS	BAS	

SHEET SIZE A3

ORIGINAL STAMPED IN RED

DATE	CHECKED	DESIGNED	APPROVED	CLIENT APP'D

Knight Piésold CONSULTING

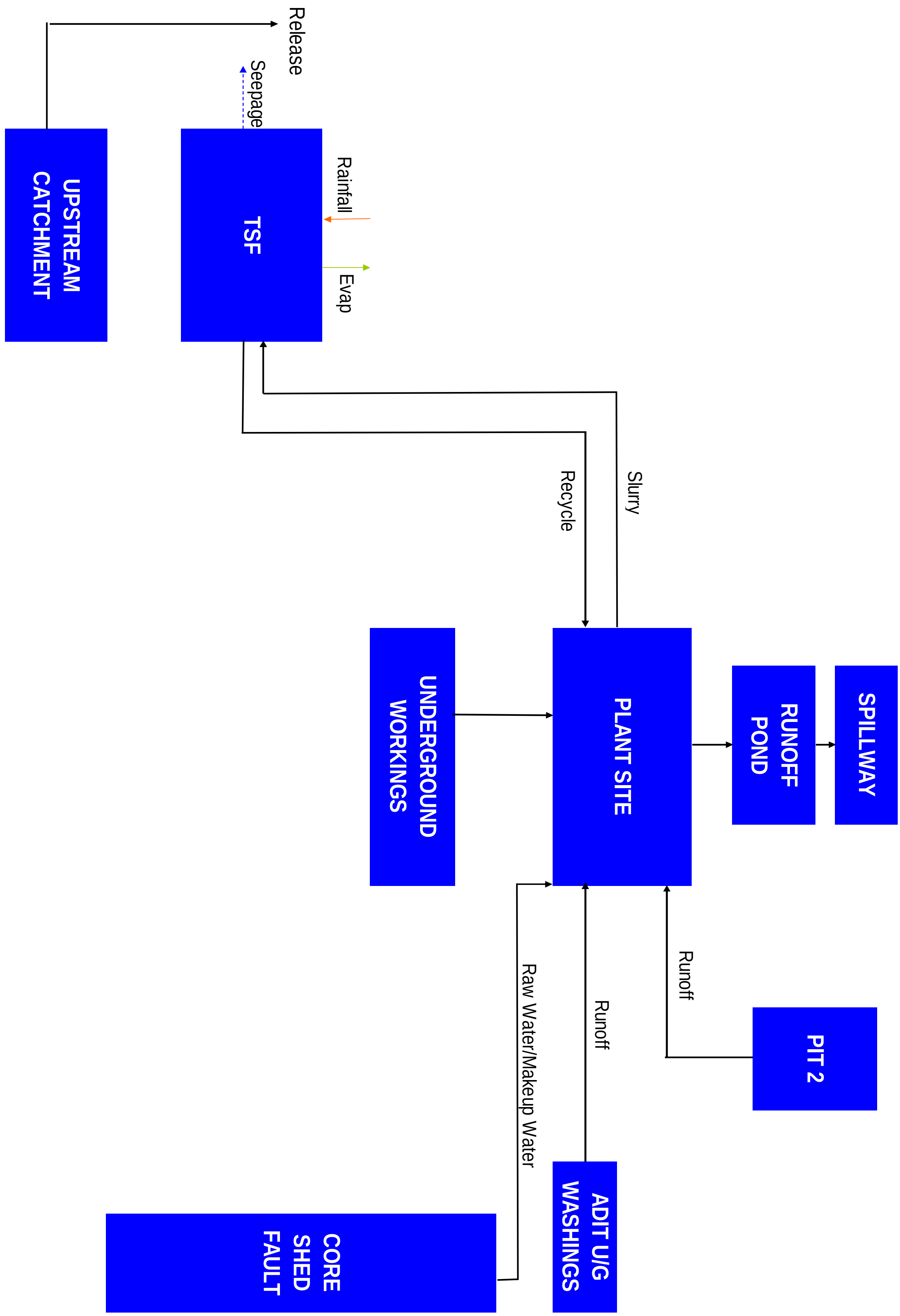
CONSULT AUSTRALIA Member Firm

THIRD ANGLE PROJECTION

LION ONE METALS LIMITED
T□□AT□ GOLD PRO□ECT
TAILINGS STORAGE FACILITY
EMBAN□MENT SECTIONS AND DETAILS

SCALE AS SHOWN
STATUS IFR
DRAWING NUMBER 701□080□C200□011
REV B

Figure 18.5 WB Block Model


UPSTREAM CATCHMENT
Release
Seepage
TSF
Rainfall
Evap
Recycle
Slurry
UNDERGROUND WORKINGS
PLANT SITE
RUNOFF POND
SPILLWAY
Runoff
PIT 2
Runoff
Raw Water/Makeup Water
ADIT U/G WASHINGS
CORE SHED FAULT

19.0 MARKET STUDIES AND CONTRACTS

This item is not applicable for this report.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Lion One has completed an Environmental Impact Assessment (EIA) for the project and this has been approved by the Department of the Environment (Fiji).

Lion One has further lodged the Construction Environment Management Plan, the Operations Environment Management Plan and the Rehabilitation and Closure Plan.

On 3rd March 2015, Lion One received notice from the Director of Mines of MRD that the Minister of Lands and Mineral Resources has approved the grant of a special Mining Lease Number 62 covering Tuvatu.

Lion One lodged an application for a surface lease covering the whole 371.68 hectare area of the mining lease. The surface lease was granted on 19th June 2014.

These documents can be made available for review on request.

21.0 CAPITAL AND OPERATING COST ESTIMATE

21.1 Mining Costs

All currencies are expressed in United States dollars (US$) unless otherwise indicated.

Mining costs are based on:

- Primarily owner underground mining.
- Owner management and technical services.

Costs have been estimated using:

- Current supplier quotation.
- Lion One supplied inputs, including equipment costs, personnel costs and key consumables. AMC considers these costs as appropriate.
- Recent AMC information.
- Allowance for minor items made on the basis of AMC database information or experience.

Key consumable costs, supplied by Lion One, and used in the cost estimates include:

- Diesel US$0.90/litre
- Power US$0.24/kWhr, based on on-site power generation using diesel generators

The estimated underground costs total US$123M, comprising of US$31M capital and US$92M operating, with an average estimated operating cost of US$82/t of mineralized material produced. The operations are envisaged as being supported by a technical services department. Costs for this department are allocated to underground capital and operating mining based on the cost ratio of these areas

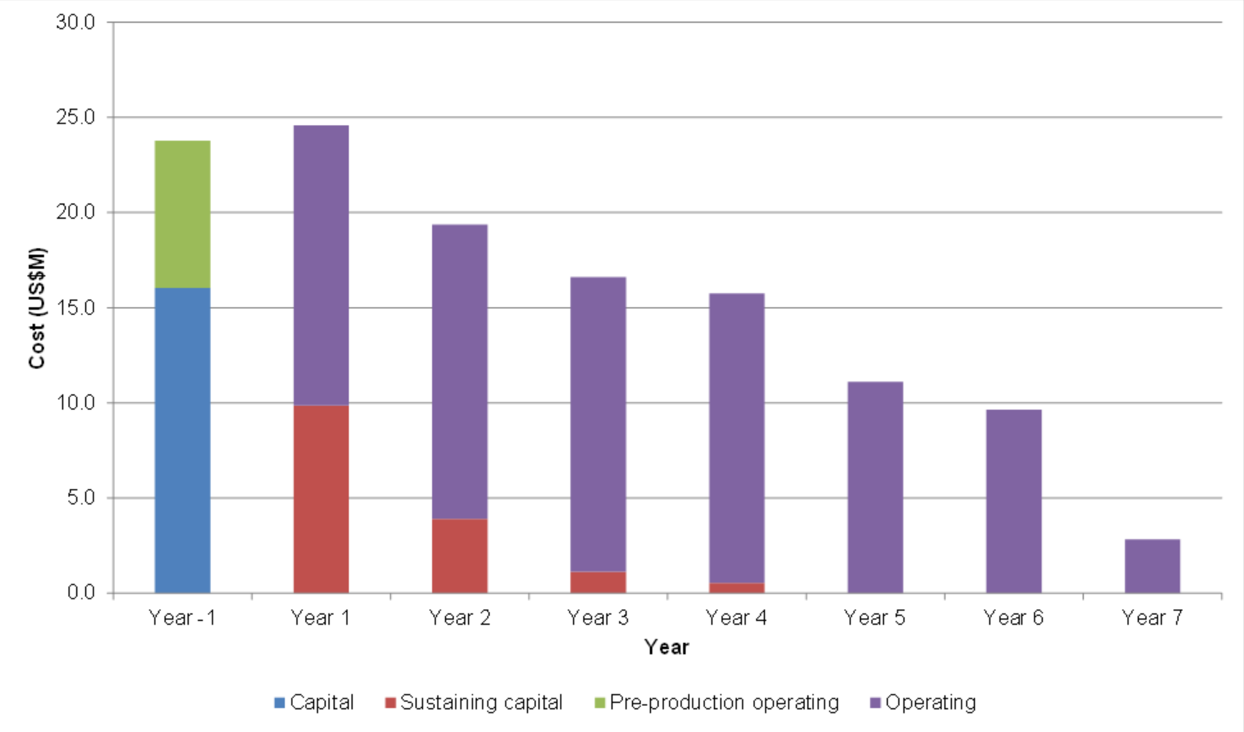
.

A summary of the estimated mining costs is provided in Table 21.1 and Figure 21.1. Capital and pre-production operating costs are incurred in Year -1, with sustaining capital and operating costs from Year 1 onwards.

Table 21.1 Total mining cost estimates

Area	Underground (US$M)
Capital	16.0
Sustaining capital	15.5
Pre-production	7.7
Operating	84.4
Total	**123.6**

Figure 21.1 Total mining cost estimates



Mining capital costs

Underground mining capital costs include:

- Purchase of the mining fleet.
- Rebuilds (major overhaul) of the mining fleet.
- Capital lateral development.
- Capital vertical development.
- Surface and underground infrastructure.

Rebuilds or major overhauls were allowed for at 50% of the equipment life, and costed at 50% of the purchase price.

Capital lateral development includes all development required to access the deposit, such as declines, primary airways, escape-ways, diamond drill platforms, stockpiles and sumps. Costs for the lateral development were built-up from first principles as part of the operating costs, and then allocated to capital based on the split of equipment engine hours. Costs allocated this way include plant (mining fleet) maintenance, personnel, diesel, electrical power, consumables, and general and administration mining costs.

Vertical capital development includes all primary airways and escape-ways.

Infrastructure covers:

- Portal establishment
- Primary high voltage supply and reticulation
- Primary and secondary ventilation system
- Primary and secondary dewatering facilities
- Raw water supply
- Underground lunchroom
- Underground safety equipment, including mine rescue equipment
- Mining staff computer software, survey equipment and other items
- Sustaining capital on the above at 2.5% per year

A contingency of 10% was applied to the underground capital costs.

A summary of the estimated underground capital cost is provided in Table 21.2.

Table 21.2 Underground capital cost summary

Capital Cost ('000 USD)	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Plant - purchase	4,768	993	0	300	250	0	0	0	**6,311**
Plant - rebuild	0	0	119	506	130	0	0	0	**755**
Capital lateral development	5,732	5,276	1,927	153	121	0	0	0	**13,210**
Vertical development	0	538	1,105	0	0	0	0	0	**1,642**
Infrastructure and minor plant	4,104	2,179	413	85	0	0	0	0	**6,780**
Contingency (10%)	1,429	876	345	101	48	0	0	0	**2,799**
Total	16,033	9,862	3,909	1,144	549	0	0	0	**31,498**

Mining operating costs

The underground owner mining operating cost estimate was prepared from first principles covering the following activities:

- Grade control
- Lateral development of in-lode drives
- Stope production covering preparation, rising, drilling, and blasting
- Load, haul and dump of production to the process ROM pad and waste to the waste dumps
- Mine services including dewatering, electrical power supply, raw water and ventilation
- Maintenance of the mine fleet
- Underground mining department management, supervision and technical services
- Operating and maintenance personnel
- Consumables including fuel, electrical power, parts, explosives, etc.

A summary of the underground mining operating cost is provided in Table 21.3.

Table 21.3 Underground mining operating cost estimate summary

Operating Costs ('000 USD)	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Plant - maintenance	992	1,746	1,805	1,886	1,786	909	797	182	**10,105**
Personnel	988	1,397	1,692	1,878	1,881	1,670	1,670	711	**11,888**
Technical Services	216	353	461	538	557	576	576	288	**3,564**
Diesel	341	657	696	736	703	464	413	113	**4,124**
Electrical power	569	1,045	978	837	835	794	689	234	**5,981**
Development consumables	2,548	3,429	3,331	3,414	3,197	69	110	0	**16,099**
Air-leg Stoping	344	2,831	3,402	3,988	4,079	4,526	3,477	664	**23,310**
Vertical development	1,082	1,604	1,132	31	0	0	0	0	**3,849**
Grade Control	149	893	1,091	1,263	1,265	1,216	1,036	197	**7,111**
General and admin	501	763	871	896	897	879	878	439	**6,124**
Total	**7,732**	**14,718**	**15,459**	**15,468**	**15,201**	**11,103**	**9,645**	**2,828**	**92,155**
Unit cost (USD/tonne)	**352**	**105**	**90**	**78**	**77**	**56**	**58**	**89**	**81.9**

Estimation accuracy

The estimated level of accuracy of these projected operating and capital costs is ±25%. The main areas of uncertainty are:

- Purchase terms and conditions for the mining equipment.
- Achieving the proposed personnel numbers.
- Skill level of the personnel.
- Achieving the estimated equipment availability and productivities.
- Achieving the design mining parameters.
- Achieving the designed mining layouts.
- Confirmation of vertical development costs.
- Completion of detailed engineering design and costing for infrastructure items.

21.2 Processing Capital Cost Estimate

The purpose of the capital cost estimate is to provide a substantiated cost estimate, which can be utilized to assess the economics of the Project.

The capital costs are presented in US dollars as at the second quarter 2015 (2Q15) to an accuracy of -20% +35%.

21.2.1 Processing Estimate Summary

Table 21.4 Process Plant Capital Cost Estimate Summary

Main Area	$, Million
Process Plant	12.28
Reagents and Plant Services	0.98
Infrastructure	7.58
Subtotal - Directs	**20.84**
Construction Indirects	2.56
Management (EPCM)	3.06
Owner's Costs	1.12
Subtotal – Indirects, EPCM, Owners	**6.74**
Contingency	4.68
Total	**32.26**

The capital cost for the process plant includes the earthworks, concrete, steel, piping, electrical and instrumentation, civils, fencing, a quote for the treatment plant which itself included the plant buildings, processing equipment, test equipment, laboratory, workshop, installation, and the plant control system.

21.2.2 Processing Estimate Basis

The capital cost estimates for each option are based on the following: Currency: USD

Period: 2Q15

Accuracy: +35% / -20%

Exchange Rates: US$1.00 = FJD$2.04

US$1.00 = RMB $6.20

The capital cost estimate has been prepared with the approach outlined in Table 21.5.

Table 21.5 Cost Estimate Basis

Description	Basis
Project Definition Information	
Maps and Surveys	Lion One provided/available/developed by consultants.
Geotechnical Investigation	Preliminary, by Knight Piesold
Process Selection	Metallurgical Testwork To Date
Design Criteria	Preliminary as per Section 17
Flow sheets / Plant Capacity	Fixed for study at a nominal 600tpd (219,000tpa)
P&IDs	Not developed for this level of study
Metallurgy	Eight Testwork Programs.
Mass Balances	Preliminary, developed by consultants.
Equipment List	Preliminary, developed by consultants.
General Arrangement/Drawings	Preliminary, developed by consultants.
Existing Infrastructure Services	Site Visit, Information provided by Lion One or otherwise known by consultants.
Capital Cost Estimate	
Earthworks – Plant Site	Budget quote based on quantity calculation made from plant site topographical data.

Earthworks – TSF	Quantities and costs estimated by geotechnical engineer, Knight Piésold.
Structural Steel, Plate-work, Piping, Concrete	Budget quote based on factored quantities from previous projects.
Mechanical Equipment	Preliminary quotations for major equipment and buildings. Remainder based on database, factors and allowances from other projects of similar scale.
Roads	SKM study, typical rates applied.
Mining	Advised by AMC
Borrow Sources	Assumed to be available from within the project limits.
Power Generation and Electrical (HV)	Sized from equipment lists, in an N+2 configuration, Estimate: Budget Quote
Raw Water Supply	Advised by AMC.
Overland Piping	Sketches developed. Estimate: Budget Quote
Electrical General	Factored off mechanical and plate work.
Installation Rates	Legislated/Lion One provided Fijian labour rates applied based on typical work scopes; including burden, contractor profit, training, supervision, tools, accommodation and commercial costs.
Cranage Supply / Hire	Crane requirements identified for heavy lifts over and above contractor supplied cranage. No allowance has been made for lifting stops due to strong winds.
Freight General	Factored estimate for sea freight and in country costs. Plant Equipment: Budget Quote.
Contractor Mobilization / Demobilization	Allowance based on similar projects at comparable scale. It has been assumed that no extremes in weather would be experienced during the construction phase and as such, no allowances are included for construction-labour stand-down costs.
Site Establishment and Construction Facilities.	Requirements estimated using base rates.
Fencing	Budget Quote.
EPCM	Budget estimate based from preliminary manhour estimates with current EPCM rates with expenses estimate based on historical data.
Consultants (mining, geotechnical)	Allowances provided by key consultants.
Site Survey / Soils Testing, Surveying QA, Vendor Reps, First Fills, Spares	Allowance based on similar projects at comparable scale.
Working capital	Based on 6 weeks' operating costs.
Labour	It is assumed that suitable construction labour would be available at the time of execution of the project.
Owner's Project Costs (including recruiting, relocation, training, etc.)	Based on labour rates, predicted staffing build-up and expenses or allowances based on comparable projects, advised by Owner.
Project Insurances and Permits	Quote provided by Owner.
Duties and Taxes	Allowance for import duties and withholding taxes based on advice by Lion One. VAT is assumed to be recoverable.
Public Road Maintenance	Excluded.
Sterilization Drilling	Excluded.
Escalation	Excluded.
Sunk Costs (including future engineering, any land acquisition or compensation costs, metallurgical testwork or any other engineering or permitting costs)	Excluded.
Exchange Rate Fluctuations	Excluded.
Scope Changes (including unidentified environmental requirements, relocation or preservation costs, delays and redesign associated with archaeological sites)	Excluded.
Force Majeure Issues	Excluded.

21.3 Tailings

The costs incurred to construct the initial starter TSF which were developed by Knight Piesold, are included in the pre-production capital at $5.1M. An additional $7.6M will be incurred over the

LOM to raise the facility embankment, which has been included in the sustaining capital estimate.

Table 21.6 TSF Cost Estimate Summary

Description	Pre-production ($M)	LOM ($M)
General (mob/demob)	1.08	2.66
TSF Site Preparation	0.14	0.26
TSF Embankment Construction	2.48	6.81
TSF Underdrainage System	1.21	2.64
TSF Spillway, Monitoring, Instrumentation	0.17	0.23
Conditioning Impoundment	0.13	0.13
Sediment Control	0.04	0.04
TSF Access	0.12	0.50
Contingency	0.85	2.12
Total	**5.12**	**12.71**

***** Totals may not sum due to rounding.

21.4 Indirects

21.4.1 First Fills

A cost allowance of $0.35M has been made in the estimate to cover the cost of the charge grinding media to charge the grinding mills, consumables and reagents required to fill the circuit and to provide the materials to sustain operations during start up.

21.4.2 Construction Equipment and Field Indirects

Heavy construction equipment costs have been estimated to be $0.14M. Costs are intended to cover any miscellaneous heavy equipment hires required for supporting the construction efforts. Due to the reduced size of the plant and the associated equipment, it is currently envisaged that large heavy equipment requirements would be minimal.

A nominal allowance has been made for field indirect costs to cover the temporary construction facilities operation and surveying, pre-operational testing and start-up. Costs generally associated with recruitment of key operating personnel, training and establishment of operating procedures and administration, on discussion with Lion One, have been carried in the owners' costs.

21.4.3 Vendor Representatives

Vendor Representative (Rep) costs typically allow for specialist vendor reps to oversee commissioning of their equipment and have been estimated at $0.17M. Vendor Rep costs are expected to be nominal as there are no specialist equipment in the process flowsheet. The allowance for training operations and maintenance staff has been carried in the owners' costs.

21.4.4 Capital Spares

An allowance of $0.24M has been made to cover the commissioning and capital spares required to support the commencement of operations.

21.4.5 Mobile Equipment

The mobile fleet required to support plant operations is based on used equipment. A 60% value of new equipment was assumed for locally sourced, used equipment in the cost estimate. This includes the light vehicles, cranes and forklifts required on site. An allowance has been made for miscellaneous mobile equipment requirements in addition to the mobile equipment estimate.

21.4.6 Freight and Logistics

Freight and logistics for the mining process equipment, buildings, reagents etc. have been quoted at $0.66M. Costs include ocean freight and inland freight with delivery to site or yard laydown.

21.5 EPCM

For the purpose of the PEA estimate, $2.1M or 10% of the process and infrastructure direct costs was estimated to cover the cost of the engineering, procurement, and construction management services required to develop the project.

21.6 Closure

At this level of study, an allowance of $0.95M has been made for the mine closure and rehabilitation cost. It is recommended that this cost be developed more fully in the next phase of engineering.

21.7 Owners Costs

These are costs incurred by the owner and are associated with the pre-production expenses from the commencement of site activities, through to the start of processing operations and gold production. These are included in the capital cost estimate, but have been developed from the operating costs. They include pre-production labour and training costs for both plant and administration, totaling $2.1M.

21.8 Sustaining Capital

A total of $10.9M was estimated as the process and infrastructure sustaining capital required and the costs are assumed to occur between Year 2 and Year 6 of the mine life inclusively. Process sustaining capital costs were estimated as 3% of the direct process plant capital costs which is in line with previous project experience.

Sustaining capital costs estimated during the mine life for the TSF amount to $7.6M.

Capitalized development for the underground mine has been estimated at $10.5M with an additional $3.3M for mining equipment sustaining capital.

An additional $1.3M contingency has been allowed for other sustaining capital items at this level of study.

21.9 Contingency

The purpose of contingency is to make an allowance or provision for uncertain elements that are understood, but not yet measured at the current level of study, which have been historically shown in the industry, to impact the engineering designs and the associated cost within the project scope; such as:

- Accuracy or variation in materials or equipment pricing and labour rates and productivities, (excluding extreme changes in these items).
- Accuracy of the estimate assembly, consolidation and inputs including the completeness of the quantity take-offs, design and level of engineering undertaken at the associated level of study.
- The information available at the associated level of study, such as variance in site conditions, sub-surface conditions and availability of suitable excavated materials and their engineering properties.

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

Due to the relatively higher number of budgetary quotes received than is typical for this level of study, a preliminary contingency analysis was undertaken which identified and analyzed estimate items such as the permanent equipment quotes, material purchase and installation, pipelines, subcontracts, level of confidence in the indirect costs and EPCM quotes, etc. The result of the analysis indicated a contingency of $6.1M was appropriate at this level of study.

21.10 Process Operating Cost

The process operating cost or expenditure (OPEX) of the project has been estimated based on the scope defined in previous sections of this report and is based on a variety of sources including cost service data, vendor quotes, first principle calculations, metallurgical testwork and reference projects. The operating cost estimate developed for this study is utilized in a preliminary assessment of the economic viability of the Tuvatu Project.

Plant operating costs have been determined for a facility with a nominal throughput of 219,000tpa at a grind size of 80% passing 75 µm, based on a 24 hour per day operation, 365 days per year and a 91% overall plant availability.

The operating costs have been developed from sources including: metallurgical testwork reagent consumptions and mineralized material hardness and abrasiveness, assumed and industry typical reagent consumption design criteria where testwork was not available, budget quotes for reagents, modeling of the comminution circuit, power developed form the preliminary equipment list, current site delivered diesel invoicing costs and Canenco's database of costs for similar sized projects.

Costs are presented in US dollars ($) and are based on prices for the second quarter of 2015 (2Q15) and the following exchange rate has been used for the preparation of the operating cost estimate: US$1.00 = FJD$2.04

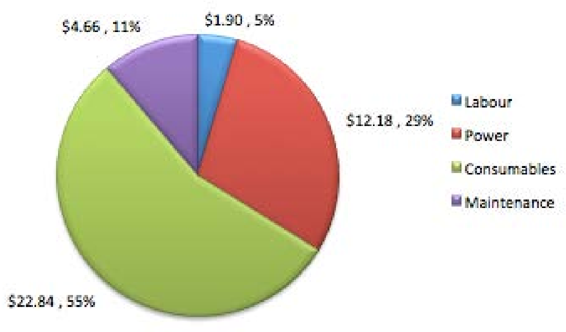
Operating costs are summarized in Table 21.7 and represented by cost centre in Figure 21.2 and are considered to have an accuracy of +30 / -10%.

Table 21.7 Process OPEX Summary

Description	$/year	$/t (milled)
Labour	415,257	1.90
Power	2,666,880	12.1
Consumables - subtotal	5,001,263	22.84
Reagents	*4,561,989*	*20.83*
Liners	*323,867*	*1.48*
Media	*115,408*	*0.53*
Maintenance	1,021,576	4.66
Miscellaneous (e.g. Laboratory)	493,718	2.25
Total Process Operating Cost	**9,598,696**	**43.8**
G&A Labour	351,409	1.60
G&A Expenses	2,659,616	12.14
Total G&A Operating Cost	**3,011,025**	**13.75**
Total Operating Cost	**12,609,721**	**$57.58**

* Mobile equipment carried in G&A

Figure 21.2 Process OPEX by Cost Centre



* **Includes plant operating and maintenance labour but excludes G&A labour costs.**

The relatively high consumable costs are mainly associated with the conservative cyanide consumption, which is established on a single test. Based on the observation of relatively low cyanide consumptions from whole ore leach test work, it is suggested that additional metallurgical testing would most likely optimize this reagent dose, consequently decreasing the overall plant OPEX.

The operating cost estimate presented in this section is exclusive of the following:

- Head office costs, including withholding taxes, other taxes and government charges, any license fees or royalties or other fees associated with government monitoring and compliance.
- Import duty on consumables.
- Any impact of foreign exchange rate fluctuations or escalation from the date of the estimate.

- Any contingency allowance.
- Any rehabilitation or closure costs.
- Tailings storage costs, including future lifts and rehabilitations.
- Gold refining and transport of gold from site.
- All costs associated with areas beyond the battery limits of the study.

Labour

The labour rates for expatriate salaries by position and skill level were advised by Canenco based on similar Canadian operating positions on similar scale projects, while the national workforce labour rates were advised by Lion One and are in accordance to the Fijian Wages Regulation (Mining and Quarrying Industry) Order 2012. Staffing requirements for the operation were developed based on experience of similar scale projects with operations working a four-panel roster in eight-hour shifts.

In discussion with Lion one, the burden for expatriates will include accommodation contribution and meals, while national labour overheads will cover items including payroll taxes, worker's compensation, insurance and leave provisions.

Table 21.8 Process Personnel Summary

Area	Number
Process Staff	9
Plant Operations	19
Laboratory	4
Plant Maintenance	4
Power Plant	1
Total	**37**

General and administration (G&A) labour includes the General Manager, Health, Safety, Environment and Community (HSEC) staff, finance, payroll and purchasing staff to be located in the existing Lion One Nadi office and warehouse management, and HR. No full time IT staff will be employed, but rather a contract set up with a local service provider to service the facilities. Other contracts to be set up are security, bus transport and off site road maintenance.

The labour cost for the project excludes all corporate costs and mining personnel salaries and overheads have been included in the mining costs.

Power

Power supply costs (including maintenance factors) has beed based on delivered diesel fuel cost of USD$0.90/L for a total cost of USD$0.24/kWh. Power consumption for the comminution circuit has been developed from the crushing circuit modeling, mill power calculations and estimates from ore typical properties with costs based on 75th percentile of the ore hardness indices. The power consumption for the remainder of the plant was estimated from the load list for the process plant and with factors from experience with similar projects for a total estimated connected load of approximately 2.3MW. Power for the underground mine has been developed

by AMC with an estimated total connected load of just over 1.2MW.

Consumables

Consumption of reagents has been developed from testwork results, industry practice or from Canenco's database of similar projects. Reagent costs were obtained from budget quotations and current in-house data.

Table 21.9 Reagent Cost Summary

Reagents	Total Annual Cost (incl. Shipping)	Cost Per Tonne Milled
Description	**$/a**	**$/t**
Sodium Metabisulphite (SMBS)	76,141	0.35
PAX	59,349	0.27
Frother (e.g., I50 or MIBC)	1,257	0.01
Sodium Cyanide (NaCN)	4,267,215	19.49
Flocculant	18,440	0.08
Copper Sulphate ($CuSO_4$)	11,614	0.05
Lime (CaO)	111,690	0.51
Sodium Hydroxide (NaOH)	1,800	0.01
Antiscalant	2,723	0.01
Carbon	10,819	0.05
Hydrochloric Acid (HCl)	942	0.00
Total Reagent Costs	**4,561,989**	**$20.83**

Please note the high cyanide consumption is based on a single test and with optimization test work, it is suggested that this cost will be reduced. Grinding media consumption and cost is calculated based on an abrasion index of 0.2 kg/t. Crusher ball mill liner and grinding media consumption rates have been calculated based on in house data.

Maintenance

Process maintenance costs for the operation have been factored using factors from the Canenco database.

21.11 General and Administration Costs

The G&A costs for the supporting facilities and administration are estimated to be $3.01Mpa or $13.75/t milled. These costs are assumed to consist of both fixed and partially variable, changing to reflect the plant operations. The G&A costs are summarized below in Table 21.23.

Administration costs have been developed based on inputs from Lion One and historical in-house data.

Table 21.10 G&A Cost Breakdown

Description	$/year	$/t (milled)
G&A Labour	351,409	1.60
G&A Expenses	1,262,876	5.77
G&A Mobile Equipment	512,460	2.34
G&A Materials	884,281	4.04
Total G&A Operating Cost	**3,011,025**	**13.75**

*Totals may not sum due to rounding.

The G&A costs include provisions for office administration, potable water supply and treatment, non-process related and off-site power costs, insurance, HSEC equipment and related stakeholder engagement costs, constracts for IT, site security, bussing, off-site road maintenance, running costs associated with the site mobile equipment fleet, site fuel costs, PPE supplies and laboratory consumables based on processing approximately 119 samples per day. Costs associated with business travel and training have been allowed as well as for ongoing use of consultants.

Mobile Equipment Costs

Mobile equipment and light vehicles required to support the operation and administrative functions have been assessed and the vehicles allowed are presented in Table 21.9

Mobile equipment costs provide for the fuel and maintenance of the light vehicles, portable generators and other mobile equipment for the process plant. Costs related to the purchase of this equipment have been included in the capital cost estimate.

The estimated maintenance costs for mobile equipment have been included under maintenance materials costs and the fuel costs have been included in the consumables cost estimate.

Table 21.11 Site Mobile Equipment List

Vehicles	Number
Light Vehicles	3
Ambulance	1
Fire Tender	1
Flat Bed Truck	1
Front End Loader (FEL)	1
Rough Terrain Forklift	1
Bob Cat/Skid Steer	1
RoughTerrain Crane (20t)	1

Laboratory Assay Costs

Lion One has had discussions with a contract laboratory service provider in Fiji who would be willing to move their assaying operations to Nadi and expand the laboratory facilities to accommodate the 119 daily samples from the mine and process facility

.

The 119 samples are made up of, 72 mine grade control samples, 39 process samples and solutions, 3 environmental samples and 5 geological or special project samples on a daily basis.

22.0 ECONOMIC ANALYSIS

An engineering economic model was developed for the Tuvatu Project, to estimate cash flows and determine the economic sensitivities of the project.

This report includes forward-looking information regarding cash flow forecasts, as a result of the studies projected mine production rates, developed recoveries and associated process construction and mine development schedules. Such factors as the ability to obtain skilled labour or major construction equipment or long-lead items in a timely fashion, or such as the ability to obtain the many permits required in an appropriate timely basis in order to construct and/or operate the mine and process facilities, or to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis. The processing plant feed grades are based on adequate sampling that is reasonably expected to be representative of the realized grades from mining and processing operations.

Pre- and After-tax estimates of project values were prepared for comparative purposes and for approximating the true investment value, respectively. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during actual operations and, as such, the after-tax results are only approximations. It is recommended that in the next phase of study a tax consultant undertake a review of the tax regimes for the Tuvatu project.

22.1 Assumptions and Qualifications Basis of Estimate

The basis of the economic evaluation of the Tuvatu Project has been obtained from a variety of information sources including:

- The capital cost estimates and expenditure schedules prepared by AMC Consultants (AMC) and Canenco Canada Inc. (Canenco) in discussions with Lion One Metals Inc. (Lion One).
- The mine operating cost estimates and mine schedule prepared by AMC.
- Process operating and general and administration costs estimated by Canenco. Owner's capital costs, sustaining capital costs and closure costs were estimated based upon assumptions or on detail provided by AMC, Lion One, Canenco and Knight Piésold.
- Metal pricing, royalties and refining charges based upon guidelines provided by Lion One.
- Study gold recovery derived from variability testwork results.

The economic evaluation reflects one metal price scenario and considered only cash flows from the start of the construction period, being based on a 14-month design and construction period, assuming the expenditures for construction initiate in Month 10 of Year -2 and that gold production commences in Month 1 of Year 1.

Other assumptions used in the economic analysis include:

- Discount Rate of 5%.
- Costs, revenues, costs and taxes are calculated for the period in which they occur rather than actual.
- Corporate tax rate of 20%.
- Provision was made for closure and rehabilitation costs of US$900,000 payable in the last month of production and equally divided over the closure period of two years.
- Working capital has been excluded from the capital cost estimate, but implicitly included in the

cash flow model as the difference between the peak funding requirement and the initial capital expenditure.

- Capital depreciation has been considered based on a straight-line approach, however no provision has been made for escalation or inflation.
- Results are presented on a 100% equity basis, i.e., the cash flow model assumes full equity funding.
- No financing costs or management fees have been considered and no provision has been made for interest or cost of capital.
- Value Added Tax (VAT) has assumed to be recoverable while there has been no provision made for any additional taxation or costs related to the repatriation of funds from Fiji.
- No provision has been made for corporate head office general and administration costs during operations.
- No plant salvage value has been considered.
- Pre-development and sunk costs up to the start of detailed engineering are excluded.
- No contractual arrangements for refining exist at this time.

These assumptions are appropriate and typical for this level of study.

Gold Price

The reader is cautioned that the gold price used in this study is an estimate based on recent historical commodity performance in the markets and there is no guarantee it will be realized if the project proceeds into production. The gold price is based on complex factors and there are no reliable long-term predictive tools.

The economic evaluation has applied a gold price of US$1,200 per ounce, based on advice from Lion One and represents the approximate average spot gold price during May 2015 published by Kitco®. A refinery gold payable rate of 99.8% has been applied with a refining charge of $0.80 per payable ounce.

Royalties and Export Taxes

The royalty and export tax model inputs provided by Lion One are all based on gold revenue and total 9.5%. They include;

- Government Royalty: 5.0 % of gold revenue.
- Laimes Global Inc. Royalty: 1.5 % of gold revenue.
- Export Taxes: 3.0 % of gold revenue.

Total royalty and export tax payments amount to approximately $40.2M over the 6.16 year LOM.

22.2 Economic Results

The reader is cautioned that this study includes the use of inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have the appropriate economic considerations applied to them that would enable them to be categorized as mineral reserves and, as such, there is no certainty the economic results presented in this study will be realized. This study is preliminary in nature and uses 55% inferred mineralized material.

A discounted cash flow model was prepared based on the mining schedule and estimated capital and operating costs. The pre-production mine operating costs have been capitalized. The current mining schedule results in a period of low production towards the end of the mine life from the shrink stoping areas. Low-grade stockpiles are not accounted for in this study and will be used to supplement plant feed at the end of the mine life.

Table 22.1 presents a summary of the production information on which the cash flow model is based. A total of 1.13 Mt ore with an average head grade of 11.3 g Au/t will be processed at an average recovery of 86.3% to recover 352,931 ounces of gold.

The life of mine capital cost for the project is estimated at $74.6M, with a pre-production and peak capital expenditure of $48.6M and $55.8M respectively.

Table 22.1 Project Production Summary

Project Production Summary	Basis of Estimate	
Total ore mined and processed	1,125,548	t (dry)
Average head grade	11.30	g Au/t ore
Contained gold in mined ore	408,958	oz Au
Recovered gold	352,931	oz Au
Average gold recovery	86.3	%
Production mine life	6.16	years
Nominal production rate	219,000	t/y
Average annual production	182,802	t/y
	57,320	oz Au/y

Table 22.2 illustrates the project cash flow summary.

Table 22.2 Project Cash Flow Summary

Project Cash Flow Summary	Project US$ Million	US$/t ore*	US$/oz Au**
Mine operating cost	86.11	76.50	243.98
Processing cost	49.33	43.83	139.78
Exploration costs	1.73	1.53	4.89
General and administration cost	21.94	19.49	62.16
Smelting and refining cost	0.85	0.75	2.40
Subtotal cash operating cost	**159.95**	**142.11**	**453.21**
Royalties and export taxes	40.23	35.75	114.00
Total cash operating cost	**200.19**	**177.86**	**567.21**
Revenue	**423.52**	**376.28**	**1,200.00**
Total cash cost	200.19	177.86	567.21
Operating cash flow (EBITDA)	**223.33**	**198.42**	**632.79**

* Basis is LOM tonnes
** Basis is recovered not contained ounces

At a gold price of US$1,200 per ounce, the project is estimated to have a pre-tax IRR of 67.1% and a respective pay-back period of 1.25 years after the commencement of production. At a discount rate of 5%, the pre-tax NPV is estimated at US$116.99M. The project is estimated to have an after-tax IRR of 52.3 % and a respective pay-back period of 1.5 years after start of production. At a discount rate of 5%, the after-tax NPV is estimated at $86.5M. The project economics have been summarized in Table 22.3.

Table 22.3 Project Financial Measures Summary

Project Financial Measures Summary	Basis of Estimate	
Revenue from gold (based on US$1,200/oz)	423.52	US$ M
Total cash cost excluding royalties	453.21	US$ / oz Au
Total cash cost (including royalties)	114.00	US$ / oz Au
All-in cost	778.60	US$ / oz Au
Capital expenditure (life-of-mine)	74.60	US$ M
Initial capital investment (excluding working capital)	48.60	US$ M
Peak funding	55.83	US$ M
Deferred and sustaining capital	25.10	US$ M
Closure cost	0.90	US$ M
Pre-tax Economics		
Free cash flow after cost allocation (undiscounted)	148.73	US$ M
Internal rate of return (IRR)	67.1	%
Project NPV (discounted at 5.0%)	116.99	US$ M
Payback period	1.25	years
After-tax Economics		
Free cash flow after cost allocation (undiscounted)	112.54	US$ M
Internal rate of return (IRR)	52.3	%
Project NPV (discounted at 5.0%)	86.54	US$ M
Payback period	1.50	years

* Total cash cost, including sustaining and deferred capital

22.3 Sensitivity Analysis

The sensitivity response of the calculated pre-tax IRR, NPV (5%), and payback period to variations in gold price, capital cost, mining and processing costs and gold recovery are illustrated in Figure 22.1, Figure 22.2 and Figure 22.3, respectively.

A 10% increase in gold price to US$1,320 per ounce would increase the project pre-tax IRR to 80%, the discounted pre-tax NPV to $149M and decrease the payback to 1.1 years. A 10% reduction in gold price to US$1080 per ounce would reduce the project pre-tax IRR to 53% and discounted pre-tax NPV to $85M and increase the payback to 1.6 years.

The analysis indicates that the project is most sensitive towards gold price.

Figure 22.1 Sensitivity of IRR to Variations in Project Inputs

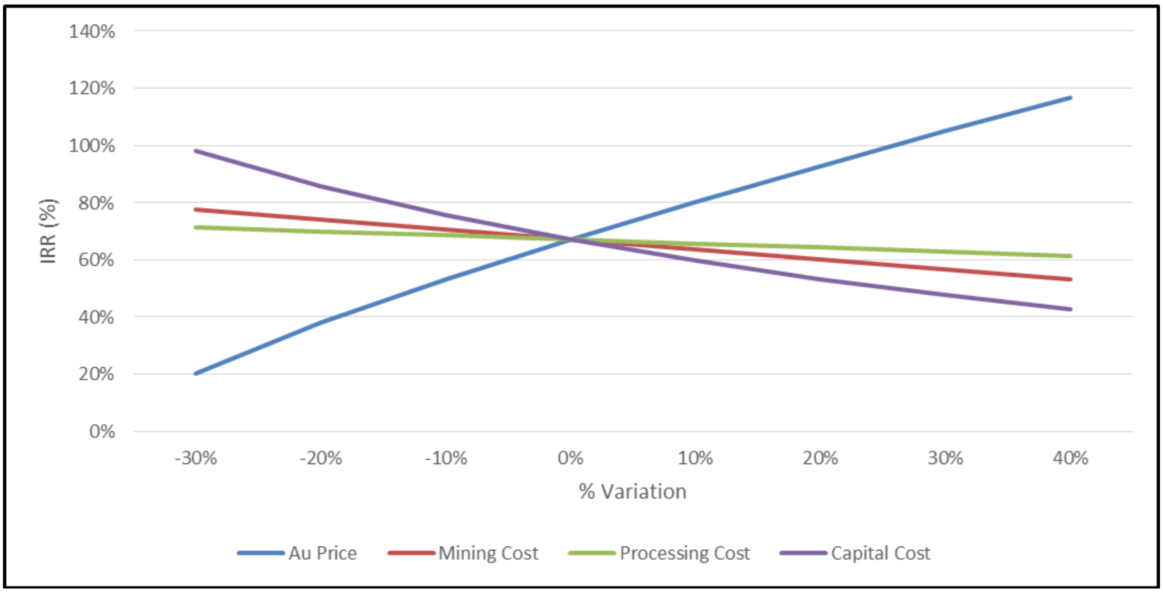


Figure 22.2 Sensitivity of NPV (5% Discount) to Variations in Project Inputs

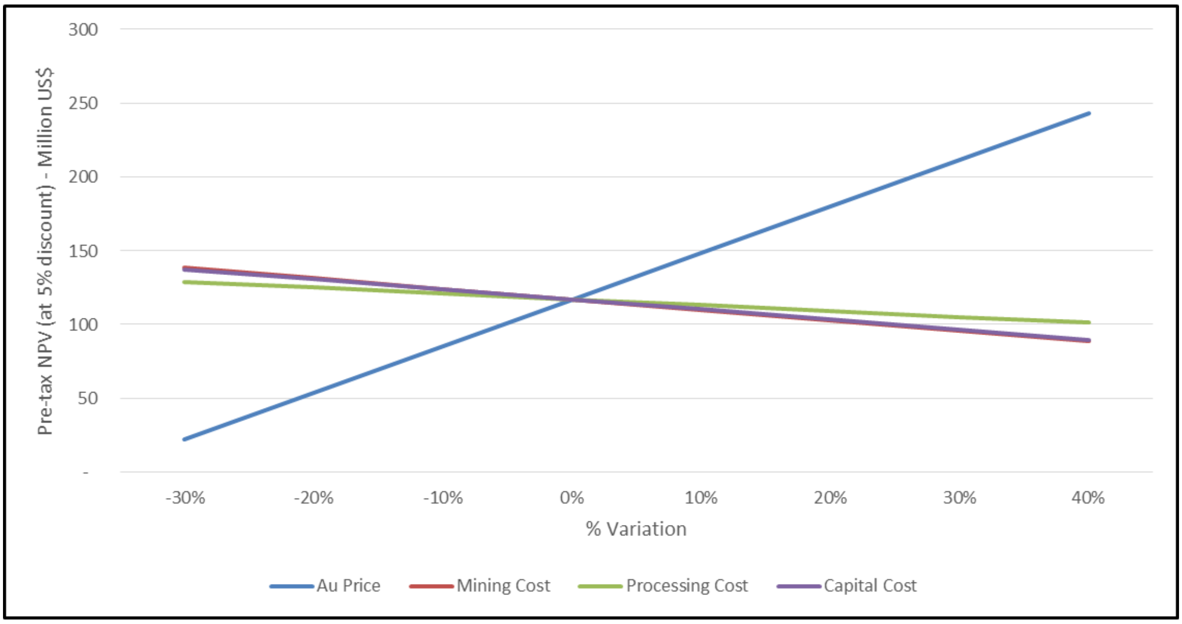
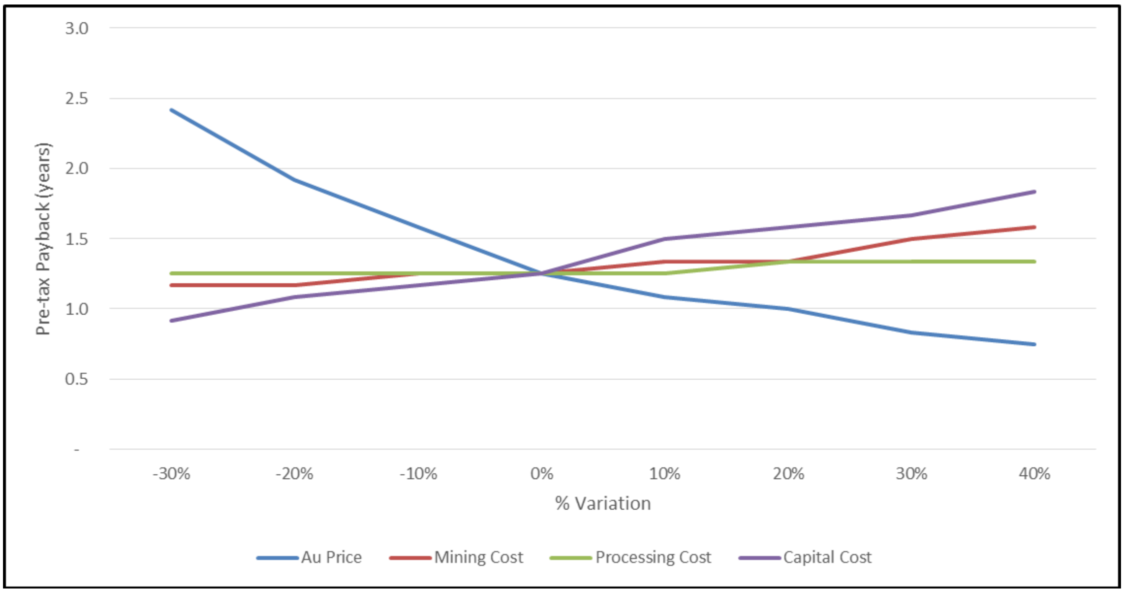


Figure 22.3 Sensitivity of Payback Period to Variations in Project Inputs



23.0 ADJACENT PROPERTIES

23.1 Ex - SPL1412 Sabeto

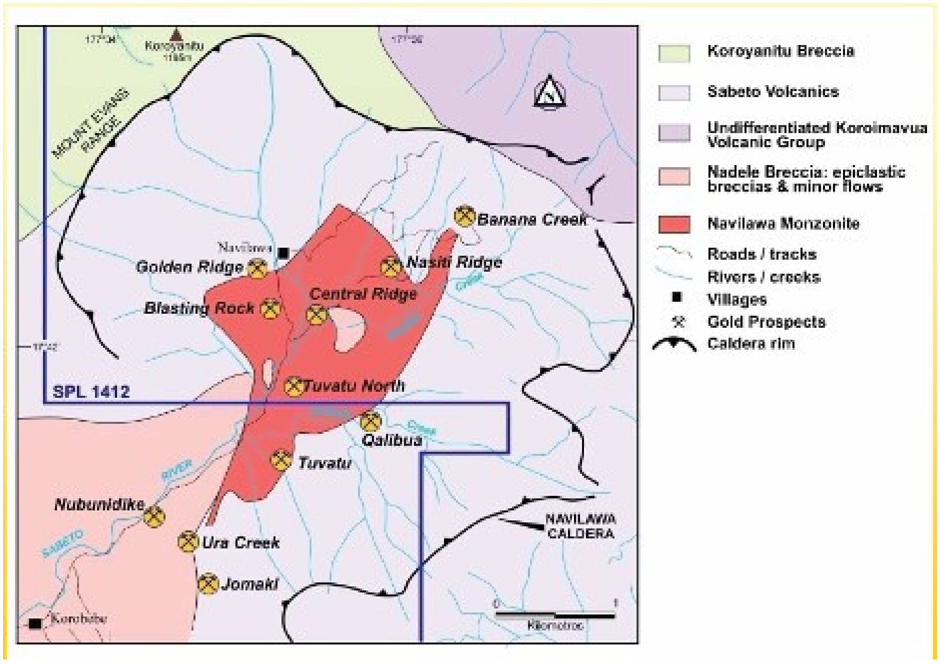
(previously Golden Rim 75%, Mincor Resources NL 25%)

Special Prospecting Licence 1412 (previously held by Golden Rim Resources (GRR) 75% Mincor Resources NL 25%) was located directly north of the Tuvatu tenement block covering an area of about 110 km² and covers the majority of the under-explored Navilawa Caldera (or Navilawa goldfield). The project which sits to the north and directly along strike from the Tuvatu gold project was considered prospective for similar high-grade epithermal gold mineralization.

Previous exploration by Mincor had established potential for higher grade epithermal deposits similar to the Tuvatu style deposit at the Banana Creek, Tuvatu North, Kingston and Vatume Hill prospects.

Figure 23.1 SPL1412 prospect locations

Showing areal relationship to main Tuvatu prospects (Source: GRR 2007 Presentation)



During the 2008 year Golden Rim completed an additional 382 m of diamond drilling in 2 holes to test for strike extensions of the high-grade Tuvatu gold resource at Tuvatu North. Relatively increased amounts of quartz, carbonate and pyrite mineralization was intercepted in both holes, however assay results were disappointing, with a best intercept of 8.7 m at 0.6 g/t gold obtained in drill hole TNDH 003.

Two major programs of geological mapping and geochemical sampling were completed at Sabeto. A total of 243 rock chips, 132 stream sediment samples and 394 soil samples were collected. High gold values were found in rock chips from two new prospect areas. A quartz vein float sample of 8.67 g/t gold was collected from Nagaga Creek to the south of the Central Ridge prospect. Significant rock chip results of 4.94 g/t gold and 2.35 g/t gold were obtained from float samples collected in Batiri Creek. These promising new areas lie midway between the Banana Creek prospect and the Tuvatu gold deposit. Follow-up work on these areas was planned for September 2008. Soil sampling produced a very high result of 17.3 g/t gold from the southern part of the Banana Creek prospect area. Ground checking of this sample located a 30 cm-wide quartz vein. Rock chip sampling of this structure returned an average grade of 1.87 g/t gold. The grade discrepancy between the soil and rock samples was not adequately explained.

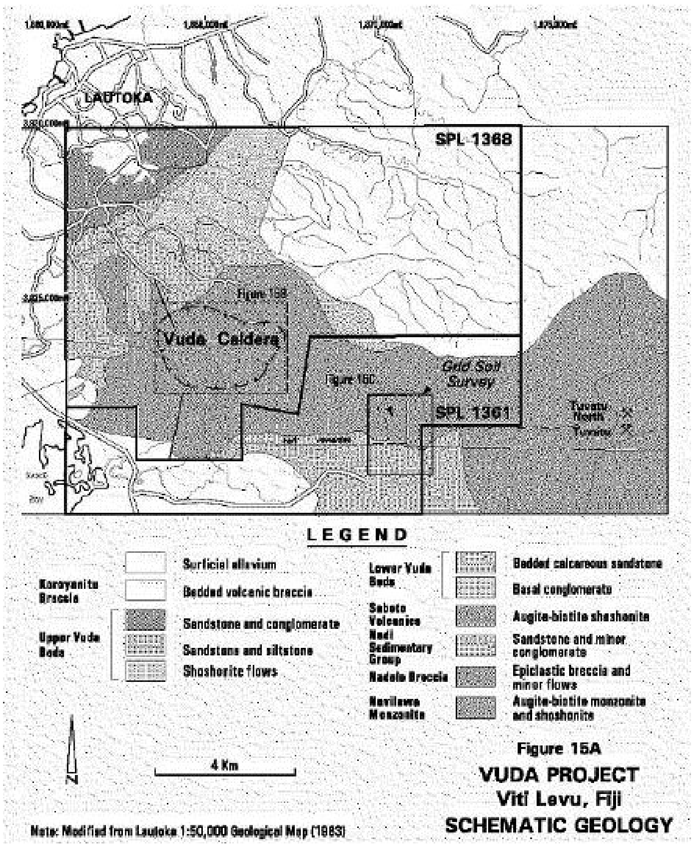
In its Annual Report for the 12 months to 30 June 2009 Golden Rim state they have decided to divest its share of the Pacific gold projects (in both Fiji and Vanuatu) to concentrate further on the gold projects in Mali. SPL 1412 is no longer shown as current on a map of Viti Levu mining tenements on the Fijian Mineral Resources Department website.

23.2 SPL1361 and SPL1368 (Vuda)

(Geopacific Ltd (subsidiary of GPR) 100% Sabeto and Option to Purchase 100% Vuda)

Geopacific Ltd (a subsidiary of Geopacific Resources Ltd (GPR) is exploring the Vuda Prospect targeting high grade vein style epithermal/mesothermal gold. The Vuda Project consists of two tenements SPL 1368 (100% Geopacific) and SPL 1361 (option to purchase 100% granted to Geopacific) located directly west of the Tuvatu tenement block and covering an area of about 85 km².

Figure 23.2 Vuda Project Location and Geology (Source: GPR Prospectus)



Geopacific describe the Vuda Project as including three defined gold targets; Natalau Mine Area, Vuda Alteration Area and Sabeto, plus a substantial area of alteration with potential for additional discoveries (Geopacific Resources, 2005). Tuvatu is located 3 km east of the boundary of SPL 1361.

Exploration on the Sabeto / Vuda project has focussed on the search for buried gold-copper porphyry mineralization and Geopacific believes that Sabeto and Vuda have potential to host porphyry related gold-copper mineralization.

In 2012, three deep (235-400m) diamond drill holes were completed on the Sabeto Porphyry Project. Geology within the drill holes has confirmed observations made from surface mapping and sampling programmes that the Sabeto geology comprises a multi-phase monzonite intrusive stock intruding volcaniclastic country rocks of the same magmatic source. The alteration and mineralization within the drill holes provide a vector toward potential porphyry-related gold-copper mineralization within an area around and to the south of SBDD001, with this drill hole displaying the most proximal alteration and mineralization assemblage. While mineralization in SBD001 is associated with syenite porphyry, it is thought that the actual mineralising porphyry phase remains undiscovered.

The area immediately south of drill hole SBD001 was to be the focus of exploration in 2013 with further detailed surface geochemistry being used to target several deep diamond drill holes to test the nature of the deeper mineralization.

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional relevant data or information.

25.0 INTERPRETATION AND CONCLUSIONS

In Canenco's opinion, there is adequate pertinent data available to produce a PEA on the Tuvatu Project, and based on the present understanding, knowledge, and using industry standard construction, mine design, process design and economic evaluation methods to assess the project, the Tuvatu project should be advanced to the next level of study as the results of this PEA indicate the project has positive economics.

25.1 Geology

Generally, the results of the QA/QC program implemented by Lion One are considered satisfactory for resource definition. It is MA's opinion that the sample preparation, security and analytical procedures were adequate and follows accepted industry standards for a mid-stage exploration property. Based on the data verification performed, it is MA's opinion that the data reviewed is adequate for the purposes used in this technical report.

25.2 Resource

No further resource drilling has been conducted since October 2013, however MA notes that the current drill hole database is reasonable and suitable for the use in the preparation of an independent NI 43- 101 compliant resource estimate which is the subject of this report.

MA agrees with the conceptual model of narrow mineralised veins as provided by site geologists, identified intercepts were considered and MA modified the vein names as appropriate to better define the vein orientations, thickness, grade and continuity.

The local geology is interpreted as a sequence of volcaniclastic units intruded by a monzonite intrusion complex with the narrow epithermal style vein mineralisation occurring as structurally controlled sets and networks of narrow veins and cracks within the monzonite host. Veins are narrow, generally less than 1 m up to a maximum of 7 m, and ore grades are erratic. This style of deposit is best developed as a selective (small mining equipment) open pit mine near surface and an underground mining operation targeting the high grade veins at depth. Mining will require significant geological control to direct development, controlling dilution and monitoring stope production.

The local geology is quite well understood from recent drilling and underground mapping, MA believes that structural complexities will be locally important and that tight geological control of production will be critical. A detailed understanding of the local structures and geometries will be required moving forward.

Risks and uncertainties which may reasonably affect reliability or confidence in future work at Tuvatu relate mainly to the reproducibility of exploration results (i.e. exploration risk) in a production environment. MA believes the exploration risk to be low due to the historical identification of mineralisation and decline development. There is a very low risk in terms of access, title or ability to perform future exploration and development work for statutory reasons

25.3 Mining

More than 50% of the mine plan is based on Inferred Mineral Resources. The Inferred Mineral Resources have a higher grade than the Indicated Mineral Resources. There is a risk that with increased geological confidence, the grade may decrease. The waste rock and ore are visually similar, making it difficult to differentiate based on visual recognition. It will require a high level of grade control definition to delineate the ore prior to mining and there is a risk that structures that were not identified in the geotechnical study will result in unfavourable underground stope wall conditions and lead to increased dilution.

To achieve the minimum mining widths good mining practices and a high level of supervision, will be required. Failure to do so may increase the level of dilution. The shrinkage stoping rates used in the schedule are aggressive in order not to result in a long tail with a low production rate. If the rates are not achieved, the processing schedule in the later years of the mine life may be impacted.
Due to the dispersed nature of many low tonnage lodes, development needs to remain well ahead of stoping areas to replace depleted levels. A risk exists that steady state production will not be maintained if development starts to lag behind the schedule.

The mining schedule and costs assume local personnel are reasonably experienced in mechanised mining techniques. If suitably experienced personnel are not available then the mining schedule may not be achieved and costs may be higher.

Limited hydrogeological inflow information is available. The study is based on groundwater inflows of 20 L/s per underground mine. If higher rates are encountered, there may be adverse impacts on the underground mine schedule, particularly the development rates. A water management strategy is required to ensure the inflow of water into the underground workings from the 1997 development, which intersects the Coreshed Fault is suitably controlled.

25.4 Metallurgy and Recovery Methods

The metallurgical data potentially indicates a bimodal gold distribution, with coarse gravity recoverable gold and a more finely disseminated fraction, being consistent with previous mineralogical and petrographic observations. Mineralogical studies have concluded that Tuvatu samples contain free gold, gold as electrum and gold-silver telluride particles.

The design criteria inputs, including the comminution energy and gold recovery, are based on limited data and test results, and have been observed to be highly variable throughout the different lodes. The process flowsheet and design has been developed in an attempt to account for this, however without additional confirmatory testwork, there is a risk that these data may vary having either an potential adverse or opportunistic effect on the process capital and operating costs and metallurgical recoveries.

In Canenco’s opinion, there is enough pertinent data at this preliminary level to determine the flowsheet for the processing facility for the Tuvatu project, being based on testwork described in Section 13, and consists of a flotation and carbon-in-leach (CIL) flowsheet comprising a two-stage crushing and screening circuit, two-stage grinding, gravity concentration, rougher flotation, cyanide leaching and carbon adsorption of both the reground concentrate and flotation tailings, cyanide detoxification, carbon elution and regeneration, gold refining, and tailings disposal.

25.5 Tailings

The TSF is designed as a cross-valley impoundment located south west of the proposed plant site. The facility has been designed to store a total of 1,200,000 t of tailings with capacity to contain all supernatant and runoff from wet years.

The tailings have been classified in this study as potentially acid forming therefore the facility will be operated as a subaqueous facility with the embankment constructed as a water retaining structure. A surface water diversion will be constructed around the facility to control the amount of surface runoff discharging into the TSF basin area. A water conditioning pond will be constructed to store water for monitoring and treatment prior to release into the surface water diversion system.

Tailings will be pumped from the processing plant to the TSF via the tailings delivery line. The tailings pipeline will be encapsulated within a separate pipeline in a pipeline to prevent any spillage.

The stability of the embankment was analysed under a range of conditions representing various time periods during operation. The results of the analysis indicate that the embankment is stable under the range of conditions analysed.

The TSF will be designed to be decommissioned as a wet facility. A pond will be maintained on the facility on decommissioning by increasing the size of the TSF catchment by removal of the surface water diversion channel located upstream of the TSF. A closure spillway will be constructed to carry flows resulting from the probable maximum precipitation (PMP) event.

25.6 Risks and Opportunities

There are numerous risks and opportunities that influence any mining venture and as such are also risks and opportunities for the Tuvatu Project. The external factors such as fluctuations in metal prices and exchange rates are obviously not in the control of the project, while other risks are usually associated with insufficient technical information. The following summarises the main, yet also common risks and opportunities presently identified and typically associated with the early stage of development of the project.

25.6.1 Risks

Table 25.1 summarizes the potential impacts and possible mitigation approaches for each internal project risk. Deleterious modifications to these items from the assumptions made in the PEA, could potentially reduce the economic viability of the mineral resource estimates and the overall project.

Table 25.1: Summary of Potential Project Risks and Mitigations

Risk Description	Comment	Potential Project Impact	Potential Improvement
Resource Upgrading Issues	The Inferred Resource included in the mine plan account for 55% of the mine tonnage, which are unable to be converted for use in future more detailed levels of engineering	If only a limited amount of the inferred resource are able to be upgraded then at the next level of engineering the mineable tonnage would be truncated, which would negatively impact the economic viability of the project.	Undertake a renewed analysis and estimation on results from a well managed drilling program, to determine the amount of inferred resource that might be able to be upgraded.
Mining Dilution and Advancement	To achieve the minimum mining widths good mining practices and a high level of supervision, will be required. Shrinkage stoppage rates used in the study are aggressive and need to be achieved to maintain production in the latter years of the mine life.	Failure to achieve the mining widths may increase the level of dilution, which will negatively impact the process plant feed grade and the project economics. If the mining advancement rates are not reached, the processing schedule in the later years of the mine life may be impacted, reducing the cash flow and project economics.	Maintain a well managed, disciplined mining team to reach and exceed operational targets, maintaining good mining practices.
Underground Geotech and Hydrology	Underground designs were prepared based on limited availability of structural, geotechnical and hydrological data.	Presence of unfavorably oriented structures, weak rock masses or hydraulic gradients may result in shallower slope angles being required.	Conduct geotechnical site investigation program and produce 3D structural model at the next level of study.
Process Recoveries	Flotation and leaching recoveries are largely based on a limited number of samples and tests, and the flowsheet as described in the PEA has yet to be confirmed with testwork.	If the LOM gold recovery is reduced from predictions, it may potentially adversely affect the project economic viability.	Undertake a well managed mineralogical and metallurgical test program to confirm the flowsheet and design criteria assumptions.
Permit Acquisition	The ability to secure all the permits required for construction and operations of the project. Government replacements causing changes to mining law and royalty regime.	Inability to obtain the required permits could potentially cause delays in the project development schedule while changes in the royalty scheme may possibly negatively impact the project economic viability.	The ongoing maintenance and deepening development of close relationship with the government and communities in conjunction with a project design giving appropriate consideration to the environment and local population is essential.
Project Schedule	Project development may be delayed for a variety of issues, which may adversely impact project economics.	Project schedule delays will unfavorably impact the projected cash flows, which could potentially alter project economics through decreased revenue.	Maintain a well-managed project schedule upon initiation of construction.
Power Supply	Diesel power generation has been assumed as the source for site power.	If power is not available, or if the diesel price surges, this will likely either cause production losses or increased operating costs to the project, which adversely affect the project economics.	Undertake a more thorough power study to obtain a more detailed understanding of power availability and future power generation development in Fiji.

25.6.2 Opportunities

Table 25.2: Summary of Major Project Opportunities

Potential Opportunity	Explanation	Possible Benefit
Cost of Power	The project power cost is at \$0.24/kWhr and a long-term contract at a reduced price would decrease the operating cost of the project. If Fiji was to implement power generation projects in the next period, this could potentially put sufficient supply in the grid and again decrease the project operating cost.	Process power costs are 28% of the overall process operating costs. Such that a 20% reduction in the overall process power cost may potentially achieve as much as a 6% reduction in overall processing operating cost.
Silver Credits	Silver has been assayed in a number of the metallurgical samples. There is potential to add to the project economics with the recovery of silver.	Including silver recoveries may provide another stream of revenue and improve project economics.
Gold Price	Gold price has the largest impact on the project economics.	The impact is illustrated in the economic sensitivities.
Metallurgical Recoveries	The potential for optimized recoveries with the proposed PEA flowsheet may potentially improve the project economics.	The NPV of the project may be improved with optimization of metallurgical recoveries
Optimizing Reagent Consumptions	Cyanide consumption has bee conservatively estimated based on a single test.	Optimizing cyanide dosage and reducing the consumption by half will result in a reduction ior the processing operating cost by over 20%.

26.0 RECOMMENDATIONS

The following recommendations have been made as a result of this study. Individual recommendations have been made by each consultant based on the technical data on the Tuvatu Gold Project.

General

- Exploration work to be ongoing, both surface channel sampling and mapping, particularly in areas where veins outcrop, and areas which may be amenable to open pit mining..
- As Lion One develops a mine at the Tuvatu Project, exploration drilling would continue to expand the current identified resources present.

Underground

The Inferred Mineral Resources included in the mine plan should be upgraded in confidence to allow potential conversion to Mineral Reserves.

The use of shrinkage stoping should be reviewed at the next project stage. Whilst it provides lower dilution compared to mechanized methods, it also has a slower production rate, and the majority of the production material is locked up in the stope until the stope is completed. Shrinkage stoping must also be assessed from the personnel safety point of view, and particularly for the potential hazard associated with swelling clays, which can also lead to broken material locked up in stopes.

The method of developing the longer vertical raises should be further considered. Two methods are available: raise-boring or Alimak, with Alimak assumed for the study. Raise-boring has advantages of safety and speed of excavation but would require the regular mobilization of a raise-boring machine from Australia or New Zealand, which would incur a significant mobilization cost.

The diamond drill grade control component of the underground mine plan should be addressed in more detail, through planning of drill-holes in the operating phase.

Detailed engineering design should be completed for underground infrastructure, electrical, dewatering and compressed air systems.

Geotechnical

AMC recommends the following design and operating parameters, and additional work for input into further assessment and as potential mining progresses:

- Shrinkage stope panel height - 60 m.
- Sill pillar height between stoping panels - 6 m (or three times the stope width).
- Maximum strike length of stoping panel - 60 m.
- If strike length exceeds 60 m, pillars should be left closer to the centre of the stope.
- For budgeting and projected stability purposes, it should be assumed that the central one third (by height) of the stoping panel would have rock bolts installed in both walls on a regular pattern (nominal 1.5 m spacing) along the total stope length.
- Assume minimum dilution per shrinkage stope panel of 15%.

- Extract broken material in a controlled manner evenly from all draw-points to ensure that any waste rock sloughing from the stope walls stays on top of the planned production material, so dilution at the end of the draw process is minimized.
- Update the Coreshed Fault model to ensure the planned stopes do not intersect the fault.
- Complete two geotechnical diamond drill-holes (total length 440 m) to gather additional information about the rock mass that will form the potential underground mine. These holes should be oriented and geotechnically logged, including for structural measurements.
- Diamond drill-holes should also be completed to investigate suitability of rock mass conditions of the sites of any proposed raises and additional portals. For raises to surface, these should be located at the proposed collar and drilled to full planned depth.
- Arrange training in geotechnical logging for geotechnical and resource drill-holes and ongoing data collection ahead of mining.
- Conduct stress measurements when suitable sites become available. Typically the first measurement would be at a depth of about 200 m below surface.
- Conduct further investigations to determine the extent of the swelling clays in the potential areas of shrinkage stoping:
  - Undertake re-logging of the available core for the purpose of identification of zones with presence of smectite.
  - Obtain representative core runs with smectite within mineralized zone. Separate the core into a) free clayey materials, to be sent to a laboratory for a chemical analysis to determine the percentage of smectite, and b) solid rocks, to be sent to a laboratory for slake-durability testing. In the material lost during the slake-durability test assume percentage of smectite based on the chemical analysis of the free clayey material. Having analysed the test results estimate percentage of smectite in the rock mass. If smectite constitutes less than 5%, it is probably insufficient to cause any notable problems in the material draw. If smectite constitutes more than 5%, identify spatially the potentially affected areas and undertake further research into the issue.

Mine closure

A detailed mine closure schedule will need to be established prior to operations ceasing. The schedule will include the major mine closure activities Including demolition of the process plant, rehabilitation of the TSF, pit closures and removal of contaminated material to be undertaken progressively both during and beyond the final year of operations.

Recovery methods

Metallurgy and Processing

In the next phase of study, metallurgical samples from the veins in the first 3 years of operation should be included in any testwork programs to better define the precious metal recoveries through this period. The flowsheet proposed in the current study will also need to be confirmed.

Engineering work should include:

- Updated design criteria
- Detailed mass and process water balance calculations
- Equipment sizing and specifications
- Detailed operational and capital cost estimates
- Updated flow sheets for each unit operation

Through each test work campaign, a number of recovery improvement programs were conducted in an effort to explain the variable recoveries, however no conclusive reason was evident.

The samples with lower gold recoveries (higher leach residue grades), generally have relatively higher sulphur grades, but a relatively higher sulphur grade does not always result in lower recoveries.

With this in mind, the following metallurgical testwork programs are recommended moving forward:

- A more comprehensive mineralogical study, including gold deportment, in an attempt to characterize the gold association variability.
- Additional crushing and grinding work indices through the different lodes,
    - BWi, CWi, abrasion testing
- More detailed GRG work with vendors,
- Flotation testwork, (reagent selection, variable optimization, lock-cycle),
- Flotation Tailings and Concentrate regrind / leach characterizations,
- Aeration and leaching optimization,
- Thickening testwork,
- Cyanide detoxification testwork based on optimized process,

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28.0 CERTIFICATE OF AUTHOR

CERTIFICATE OF AUTHOR

I, Stacy Freudigmann do hereby certify that:

1. I am currently contracted as a Project Manager with Canenco Canada Inc. who has an office at 602 East 4th Street, North Vancouver, BC, V7L 1J8;

2. This certificate applies to the technical report titled "Preliminary Economic Assessment Technical Report Tuvatu Project, Fiji", (the "Technical Report"), with an effective date of June 1, 2015, prepared for Lion One Metals Inc. ("the Issuer");

3. I am a Professional Engineer (P.Eng. License #33972) registered with the Association of Professional Engineers, Geologists of British Columbia. I am a Member of the Canadian Institute of Mining and Metallurgy and the Australasian Institute of Mining and Metallurgy.

   I am a graduate of James Cook University with a B.Sc.(Hons) in Industrial Chemistry (1996) and Curtin University, Western Australia School of Mines with a Grad.Dip. Metallurgy (1999). I have been involved in mining since 1996 and have practiced my profession continuously since 1996. I have held senior process and metallurgical production and technical positions in mining operations in Canada and Australia. I have worked as a consultant for over five years and have performed process managment, project management, cost estimation, scheduling and economic analysis work for a number of engineering studies and technical reports located in Latin America, Europe, USA and Canada. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4. I visited the Tuvatu Project site from the 8th to the 14th of April 2014;

5. I am responsible for section numbers 1 to 3, 13, 17, 18 (except 18.3, 18.8, 18.9), 21 (except 21.1, 21.3), 19, 20, 22, 24, 25, 26, 27, 28, and 29 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had prior involvement with the property that is the subject of the Technical Report.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: June 1, 2015
Signing Date: July 10, 2015

Stacy Freudigmann, P.Eng

CERTIFICATE OF QUALIFIED PERSON

ANTHONY JAMES WOODWARD

I, Anthony James Woodward hereby certify that:

I am a Consulting Geologist and Professional Geoscientist residing at 14 Carlia Street, Wynnum West, Queensland 4178, Australia (Telephone +61-7-3396 9584). I am independent of the issuer as independence is described in Section 1.5 of NI 43-101.

I graduated from the University of Nottingham, UK in 1968 with a B.Sc. (Hons) in Geology and from James Cook University, Townsville, Australia in 1976 with a M.Sc in Exploration and Mining Geology.

I have over 35 years' experience in the minerals industry as a Geologist in the fields of mineral exploration, mine geology and mineral resource estimation. I have had senior exploration roles with Buka Gold, Niugini Mining, Eltin Minerals and Oakbridge Ltd. I have conducted evaluation of advanced exploration and mining projects in Australia, Brazil, Fiji, Indonesia, Kazakhstan, New Zealand, and Turkey. I worked as Technical Services Manager and Chief Geologist at the Vatukoula Gold Mine in Fiji (Emperor Mines Ltd) from 1995 to 2005 and as Technical Services Manager for Anvil Mining Congo at the Kinsevere copper mine, DRC from 2007 to 2008. Most recently, I have been an exploration consultant in the Philippines involved with total exploration program management on tenements prospective for both epithermal gold-molybdenum and porphyry copper-gold deposits including regional exploration targeting through to deposit resource drilling.

Applicable to the Tuvatu Project is my extensive experience in mineral deposits in volcanic terrains, specifically the Vatukoula and Tuvatu epithermal gold deposits in Fiji. I have also worked on epithermal/hydrothermal and porphyry-style mineralization in similar environments in Papua New Guinea, Fiji, New Zealand, Philippines, Indonesia, Brazil and Turkey as well as Australia.

I am a Member of the Australian Institute of Geoscientists (Member No. 2668).

For the purposes of the Technical Report entitled: "TUVATU GOLD PROJECT PRELIMINARY ECONOMIC ASSESSMENT" Effective Date 01 June 2015, of which I am a part author and responsible person, I am a Qualified Person as defined in National Instrument 43-101 ("the Rule").

I am responsible for the preparation of Sections 4 to 10 and 23 of the technical report.

I visited the Tuvatu project site several times in the period between 1995 and 2001 while employed by Emperor Gold Mining Company Ltd.

I have read the Rule and this technical report is prepared in compliance with its provisions. I have read the definition of "qualified person" set out in the Rule and certify that by reason of my education, affiliation with a professional association (as defined in the Rule) and past relevant work experience, I fulfil the requirement to be a "qualified person" for the purposes of the Rule.

To the best of my knowledge, information and belief those sections of this report for which I am responsible contain all scientific and technical information that is required to be disclosed in order to make this report not misleading.

I have no direct or indirect interest in the properties which are the subject of this report. I do not hold, directly or indirectly, any shares in Lion One Metals or other companies with interests in the exploration assets of Lion One Metals. I am independent of the Issuer, Lion One Metals, as independence is described by Section 1.5 of NI 43-101.

I will receive only normal consulting fees for the preparation of this report.

Dated at Brisbane this 8th day of July 2015.

Respectfully submitted

(signed) "*Anthony James Woodward"*

Anthony James Woodward, BSc Hons, M.Sc., MAIG
Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Ian A Taylor

As part Author of the Independent Technical Report entitled: "TUVATU GOLD PROJECT PRELIMINARY ECONOMIC ASSESSMENT" effective date June 1 2015, I, Ian A Taylor do herby certify:

I am a Principal Resource Geologist of Mining Assoicates Ltd, Level 4, St Pauls Terrace Spring Hill, 4004, Queensland Australia. (Telephone +61 (0)7 3831 9154)

I hold the following academic qualifications:

BSc (Hons) James Cook University, 1993

I am a member and certified professional geologist of the Australian Institute of Geoscientists and the Australian Institute of Mining and Metallurgy (# 110090) as well as a member of the Australian institute of Geoscientists. (#3069)

I have worked as a geologist in the minerals industry for 20 years.

I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience in the minerals industry includes resource geology, production geology in open pit and underground mines and exploration roles. I have experience in a range of commodity styles including orogenic gold, epithermal gold and silver, porphyry copper-gold-molybdenum, komatiitic nickel sulphide and intrusion related gold.

Those sections of this report for which I am responsible have been compiled in accordance with NI 43-101

I am responsible for the preparation of sections 11, 12 and 14 of the technical report.

I visited the Tuvatu project site during the period 25th to 28th February 2014

I am independent of Lion One Metals Pty Ltd as defined by NI43-101 and have had no prior involvement with Tuvatu Gold Project.

As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

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

Dated this 8th day of July 2015

<u>"signed" Ian Taylor</u>

Respectfully submitted

Ian Taylor BSc (Hons) MAIG

CERTIFICATE OF QUALIFIED PERSON

I, David John Toomey Morgan, MAusImm (CP) as an author of this report entitled '*Tuvatu Gold Project Preliminary Economic Assessment*', do hereby certify that:

1) I am a civil engineer with Knight Piésold Pty Ltd. My office address is Level 1, 184 Adelaide Terrace, East Perth, Western Australia 6004.
2) I am a graduate of the University of Manchester, (BSc, Civil Engineering, 1980) and the University of Southampton (MSc, Irrigation Engineering, 1981).
3) I am a Member of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and registered as a Chartered Professional. I have worked as a civil engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - review and report as a consultant on numerous tailings storage facilities and mining projects around the world for due diligence and regulatory requirements.
   - Project director on a number of feasibility studies and detailed designs in the gold industry in Africa, Australia and Asia.
   - Consulting engineer at a number of gold mines in Africa, Australia and Asia.
4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.
5) I have visited the Tuvatu Gold Project site on 8 – 10 September 2014.
6) I am responsible for all preparation of Item Numbers: 18.3, 18.8, 18.9 and 18.10 with contributions to item 26.0 of the Technical Report.
7) I am independent of the Issuer applying the test set out in Section 1.5. (4) of NI 43-101.
8) I have not been involved in any previous Technical Report on the Tuvatu Gold Project.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 1st day of June 2015

The Institution of Engineers, Australia
David J T Morgan
MIEAust CPEng
Chartered Professional Engineer
Membership No. 974219

David John Toomey Morgan, MAusImm (CP)

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CERTIFICATE OF QUALIFIED PERSON
David M Lee, BEng (Mining) (Hons.), Graduate Diploma of Business, FAusIMM
AMC Consultants Pty Ltd
9 Havelock Street, West Perth, WA, Australia

I, David M Lee, BEng (Mining) (Hons.), Graduate Diploma of Business, FAusIMM, am employed as a Principal Mining Engineer with AMC Consultants Pty Ltd in Perth, Australia.

This certificate applies to the Technical Report entitled "Tuvatu Gold Project Preliminary Economics Assessment (PEA) NI 43-101 Technical Report" dated 1 June 2015.

I graduated from the Curtin University of Technology, Australia, (Graduate Diploma in Business) in 2005, and the University of Sydney, Australia, (BEng (Mining) (Hons) in 1988.

I am a professional mining engineer. I am a Fellow of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 106796).

I have practiced my profession for 27 years since completing my honours degree in 1988 in the field of mining. I have been directly involved in the mining aspects related to both open pit and underground mining for the last 27 years in Australia, Africa and South America.

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 the NI 43-101.

I most recently personally inspected the property on 9 September 2014.

I am responsible for the technical content of Section 15 and 16 (Mining Study) and the mining component of Section 21 (Capital and Operating Costs) of the Technical Report, and those portions of the Summary, Conclusions and Recommendations that pertain to the referenced Sections.

I am independent of Lion One Metals Limited, as per independence described by Section 1.4 of NI 43-101.

I have had no prior involvement with the property.

I have read NI 43-101 and certify that the sections of the report I am responsible for have been prepared in compliance with that instrument.

At the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

“Original signed and sealed by”

David M Lee
BEng (Mining) (Hons), FAusIMM
Dated at Perth, Australia, this 13th day of July 2015