Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji



PRESENTED TO

Lion One Metals Limited

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ABBREVIATIONS

Abbreviations	Definition
.csv	Comma Separated File
2D	Two-dimensional
3D	Three-dimensional
AAS	Atomic Absorption Spectrometry
ADR	Absorption-Desorption-Recovery
Ag	Silver
AG	Autogenous Grinding
A_i	Abrasion Index
Amdel	Amdel Limited Mineral Services
AMMTEC	Australian Metallurgical and Mineral Testing Consultants Limited
AMCT	AG Media Competency Test
AMT	Audio Magnetotelluric Survey
ANCOLD	Australian National Committee on Large Dams
ANFO	Ammonium Nitrate and Fuel Oil
APSAR	Applied Petrologic Services & Research
ARD	Acid Rock Drainage
ARI	Average Recurrence Interval
As	Arsenic
ASX	Australian Stock Exchange
Au	Gold
Bc	Banana Creek
Bi	Bismuth
BV	Bureau Veritas Commodities Canada Ltd.
B_{Wi}	Bond Ball Mill Work Index
$Ca(OH)_2$	Hydrated Lime
Cambria	Cambria Geosciences
Carpenters	Carpenters Fiji Ltd.
cBLEG	Clay Bulk Leach Extractable Gold
CDA	Canadian Dam Association
CEMMP	Construction Environmental Management and Monitoring Plan

Abbreviations	Definition
CGR	Continuous Gravity Recovery Tabling
CIP	Carbon-in-pulp
CMS	Central Mineralogical Services
CRM	Certified Reference Material
CSAMT	Controlled Source Audio Magnetotelluric Survey
CSF	Core Shed Fault
Cu	Copper
$CuSO_4$	Copper Sulphate
$CuSO_4.5H_2O$	Copper Sulphate Pentahydrate
CV	Coefficient of Variation
C_{Wi}	Crushing Work Index
DCS	Distributed Control System
DE	Department of Environment
DD	Diamond Drill
DDH	Diamond Drill Hole
EDF	Environmental Design Flood
EFL	Energy Fiji Ltd.
EIA	Environmental Impact Assessment
En	East Namotomoto
Entec	Entec Limited
Entech	Entech Pty Ltd.
FMS	Fiji Meteorologic Service
FRA	Fiji Road Authority
g*m	Gram*Meter
G&A	General and Administrative
Gekko	Gekko Systems Inc.
Geopacific	Geopacific Ltd.
GeoSpy	GeoSpy Pty Ltd.
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
HCl	Hydrochloric Acid

Abbreviations	Definition
HDPE	High-density Polyethylene
HEC	Hydrologic Engineering Center
ID^2	Inverse Distance Squared
IDF	Intensity-Duration-Frequency
IP	Induced Polarization
IRR	Internal Rate of Return
IUCN	International Union for Conservation of Nature
J	Jomaki/Davui/Ura
Jinpeng Group	Metallurgical Company Yantai Jinpeng Group
K	Kingston
Lion One	Lion One Metals Limited
LOM	Life-of-Mine
LOU	Land Owning Unit
MA	Mining Associates Pty Ltd
Metcon	Metcon Laboratories
Met-Solve	Met-Solve Laboratories Inc.
MHG	Magnitude of the Horizontal Gradient
ML	Metal Leaching
MMZ	Medium-grained Monzonite
MO	Molybdenum
MRD	Mineral Resources Department, Fiji
MSO	Mineable Stope Optimization
MZ	Monzonite
NaCN	Sodium Cyanide
NaOH	Sodium Hydroxide
$Na_2S_2O_5$	Sodium Metabisulphite
NLTB	Native Land Trust Board
NML	Non-metal Leaching
NN	Nearest Neighbour
NPAG	Non-potentially Acid Generating
NPR	Neutralizing Potential Ratios

Abbreviations	Definition
NPV	Net Present Value
Nr	Nasiti Ridge
Ns	Naisala Creek
OH&S	Occupational Health and Safety
OIS	Operator Interface Stations
OK	Ordinary Kriging
OMC	Orway Mineral Consultants
PAG	Potentially Acid Generating
PAX	Potassium Amyl Xanthate
PC	Personal Computer
PEA	Preliminary Economic Assessment
PGA	Peak Ground Acceleration
PL	Prospecting License
PLC	Programmable Logic Controller
QA	Quality Assurance
QC	Quality Control
QP	Qualified Person
RC	Reverse Circulation
RAS	River Analysis System
RL	Reduced Level
ROM	Run-of-Mine
R_{Wi}	Bond Rod Mill Work Index
RQD	Rock Quality Designation
SAG	Semi-autogenous Grinding
SCRMPD	Seepage Collection and Water Quality Monitoring Pond
SCS	Sediment Control Structure
SG	Specific Gravity
SGS Lakefield	SGS Canada Inc. in Lakefield, Ontario
SMEC	SMEC Australia
SML	Special Mining Lease
SO_2	Sulphur Dioxide

Abbreviations	Definition
SPL	Special Prospecting License
Tetra Tech	Tetra Tech Canada Inc.
the Project	Tuvatu Gold Project
Te	Tellurium
TGM	Tuvatu Gold Mines
tkm	Tonne-Kilometer
TSF	Tailings Storage Facility
Tv	Tuvatu
TLTB	Taueki Land Trust Board
UB Freight	UB Freight Ltd.
UQ	Upper Qalibua
UCS	Unconfined Compressive Strength
UR	Upper Ridges
UV	Unmineralized Vein
VA	Vigar and Associates
VAT	Value Added Tax
VBX	Vein Breccia
VFD	Variable Frequency Drive
Vs	Vunisalata Creek
VSD	Variable-speed Drive
VSI	Vertical Shaft Impactor
WAD	Weak Acid Dissociable
WMF	Water Management Facility
Wood	Wood plc
WTP	Water Treatment Plant
Xinhai	Yantai Xinhai Mining Research & Design Co. Ltd.
Zn	Zinc
Zonge	Zonge Engineering and Research Organisation

1.0 SUMMARY

This Preliminary Economic Assessment (PEA) Update (Technical Report) is a summary compilation of additional geological exploration work completed on the Tuvatu Gold Project (the Project or the Property) since the 2020 PEA (Wang et al. 2020).

A PEA is preliminary in nature and includes Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the Technical Report will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Project covers a total of approximately 200 km^2 and is located in the upper Sabeto Valley on the main island of Viti Levu in Fiji, 17 km by road from the international airport in Nadi. The Property is a high-grade, narrow vein, alkaline-hosted gold deposit located in five tenements held by Lion One Limited, a company incorporated in Fiji. Lion One Limited is a 100% owned subsidiary of Lion One Metals Limited ("Lion One"). Lion One was incorporated on November 12, 1996, under the name X-Tal Minerals Corp. and under the laws of the Province of British Columbia, Canada. On January 28, 2011, the Company executed a reverse takeover of X-Tal by American Eagle Resources Inc. and changed its name to Lion One Metals Limited. Lion One is a reporting issuer in British Columbia with its common shares listed on the TSX Venture Exchange under the symbol "LIO" and a secondary listing of Chess Depository Interests "CDIs" on the Australian Stock Exchange "ASX" under the symbol "LLO".

In 2015, Lion One released the 2015 PEA (Freudigmann et al. 2015), which outlined resource estimates and preliminary project economics.

In 2020, Lion One commissioned Tetra Tech Canada Inc. ("Tetra Tech"), Mining Associates Pty Ltd. ("MA"), Entech Pty Ltd. ("Entech"), Wood plc ("Wood"), and GeoSpy Pty Ltd. ("GeoSpy") to prepare the 2020 Technical Report incorporating updated studies on the Project. Tetra Tech prepared this Technical Report based on work by the following consultants:

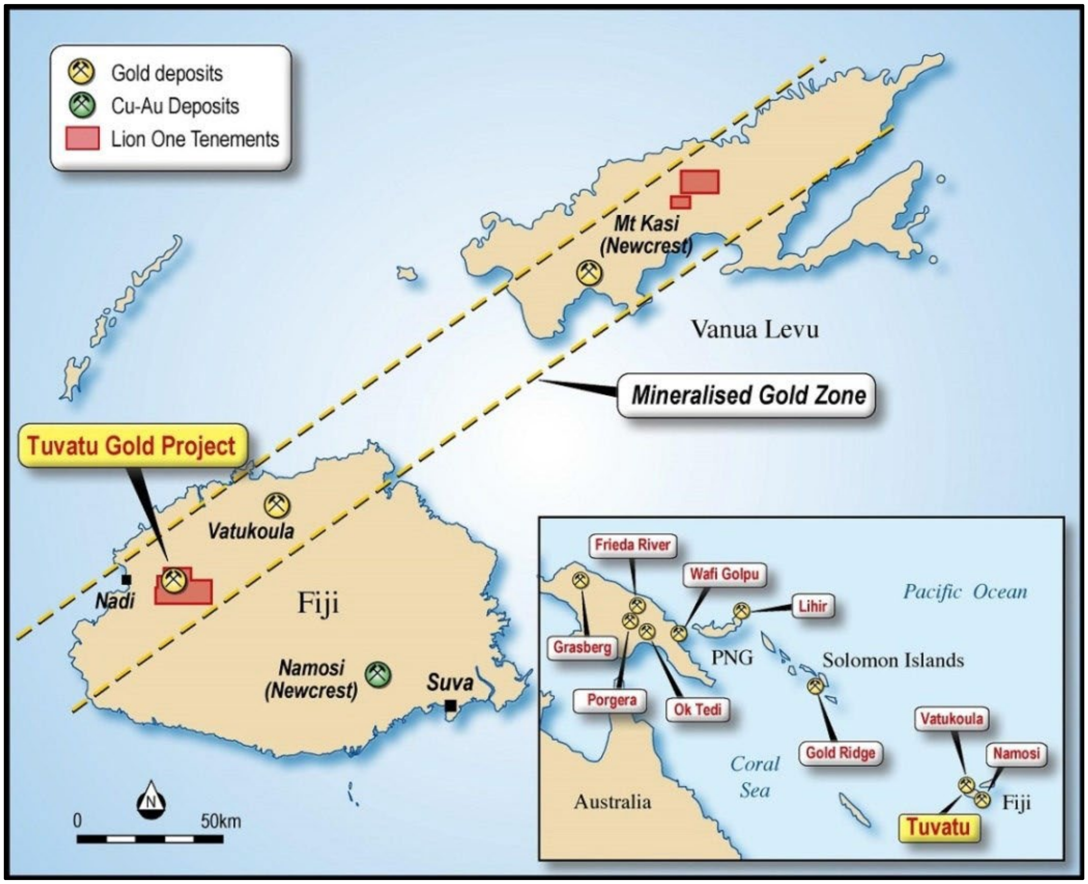
- MA – Geology and Mineral Resource estimate and related information
- GeoSpy – Geology, exploration
- Entech – Mining and mining-related operations, underground geotechnical investigations, mining-related capital and operating cost estimates
- Tetra Tech – Metallurgical test work review, process and process-related cost estimates, general and administrative (G&A) and surface service operating cost estimates, site infrastructures (excluding site geotechnical investigation and tailings storage facility [TSF]), and environment
- Wood – Site geotechnical investigation and TSF

This Technical Report is an independent technical report to update the 2020 PEA for the Project by incorporating geological exploration activities completed after September 25, 2020.

The effective date of the Mineral Resource estimates is January 8, 2018, and the effective date of this Technical Report is April 29, 2022.

1.1 Project Location

The Project is located on the west coast of the island of Viti Levu in the Republic Fiji, approximately 24 km northeast of the town of Nadi and 17 km from the Nadi International Airport (Figure 1-1). The Project area consists of four Special Prospecting Licenses (SPLs) held 100% by Lion One Limited (SPLs 1283, 1296, 1465, and 1512) and a 384 hectare Special Mining Lease (SML 62) hosting the Tuvatu Resource. The 200 km² project area is centered on the Navilawa Caldera, a 7 km diameter geological occurrence hosting a mineral system associated with widespread alkaline magmatism.



Source: Freudigmann et al. (2015)

Figure 1-1: Project location

1.2 Geology

Tuvatu is a high grade alkaline gold deposit forming a small part of the 7 km diameter Navilawa Caldera on a corridor of high grade alkaline gold systems in Fiji. Navilawa is located 45 km from the high grade Vatukoula gold mine, which is also hosted in an alkaline system of similar scale, and has produced over 7 million ounces of gold over the last 85 years.

The Project lies within the Fiji Gold Trend, which is a northeast trending extensional fault zone across northern Viti Levu and Vanua Levu (the two main Fijian islands). Virtually all significant gold occurrences in Fiji occur along the Fiji Gold Trend in clusters surrounding igneous craters such as the Tavua Caldera hosting the Vatukoula Gold Mine and the Navilawa Caldera hosting Tuvatu.

The main characteristics of these gold deposits/occurrences are:

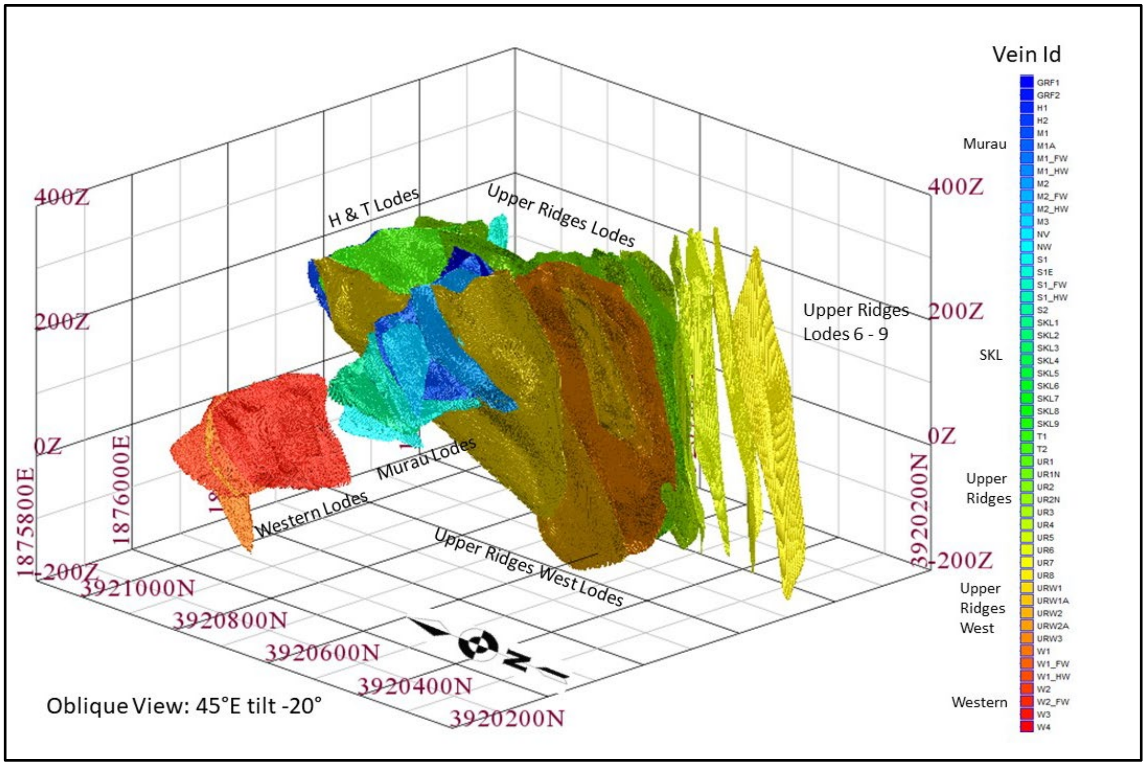
- The gold is igneous hydrothermal (250°C) in origin, introduced by the potassic, shoshonitic tertiary intrusives, which represent the feeders and latest phases of the volcanic rocks that dominate northern Fiji. Hydrous mineral phases (biotite and hornblende, particularly) in the intrusives develop whenever the hydrothermal mineralization systems have developed.
- Gold is generally vein-controlled and can be restricted to narrow bonanza-grade lodes within weakly altered host rocks. Primary gold is fine-grained and can be in the following forms:
  - Gold-silver tellurides
  - Electrum
  - Native gold
  - Gold-bearing pyrite
- Common minerals associated with mineralization are:
  - Quartz
  - Carbonates
  - Adularia - K-feldspars
  - Pyrite
  - Roscoelite (green vanadium-titanium micas)
  - Smectite clays
- Magnetite in the adjacent host rock has been converted to pyrite by the bisulphide complex, which carries gold in the alkaline-rich fluids, but alteration selvages are commonly very narrow (less than 0.5 m).

The Navilawa Caldera hosts a telescoped multi-phase mineralized system. In the eroded southeast end of the Caldera at Tuvatu, there is a high grade vein system present. Deeper level alkaline copper-gold mineralization is hosted by monzonitic intrusives in the center of the Caldera, while to the northeast at the Banana Creek Prospect, a high level, low temperature, less than 260°C, gold-bearing vein system (quartz-adularia-calcite) has been mapped over an area of 1,400 m x 400 m. The vein system is open to the northeast where it approaches the Caldera rim, which is a structural setting analogous to the Emperor (Vatukoula) Gold Mine where the vein system occurs in faulted volcanics within the Caldera margin.

The host lithologies of the project 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 5.00 m true width with a mean of 1.10 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 flotation 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 is clean with respect to deleterious elements such as arsenic, selenium, tellurium, and uranium.

The main mineralized zone (Upper Ridges) comprises eleven principal lodes with a strike length in excess of 500 m 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 47 different lode structures were identified in the resource area including 11 lodes in the Upper Ridges area, 7 lodes in the Murau area, 7 lodes in the West area, 7 lodes associated with Snake and Nasivi lodes, 4 lodes in the Tuvatu area, and 9 stockwork veins in the SKL area. A minimum of five 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 Mineral Resource estimates.



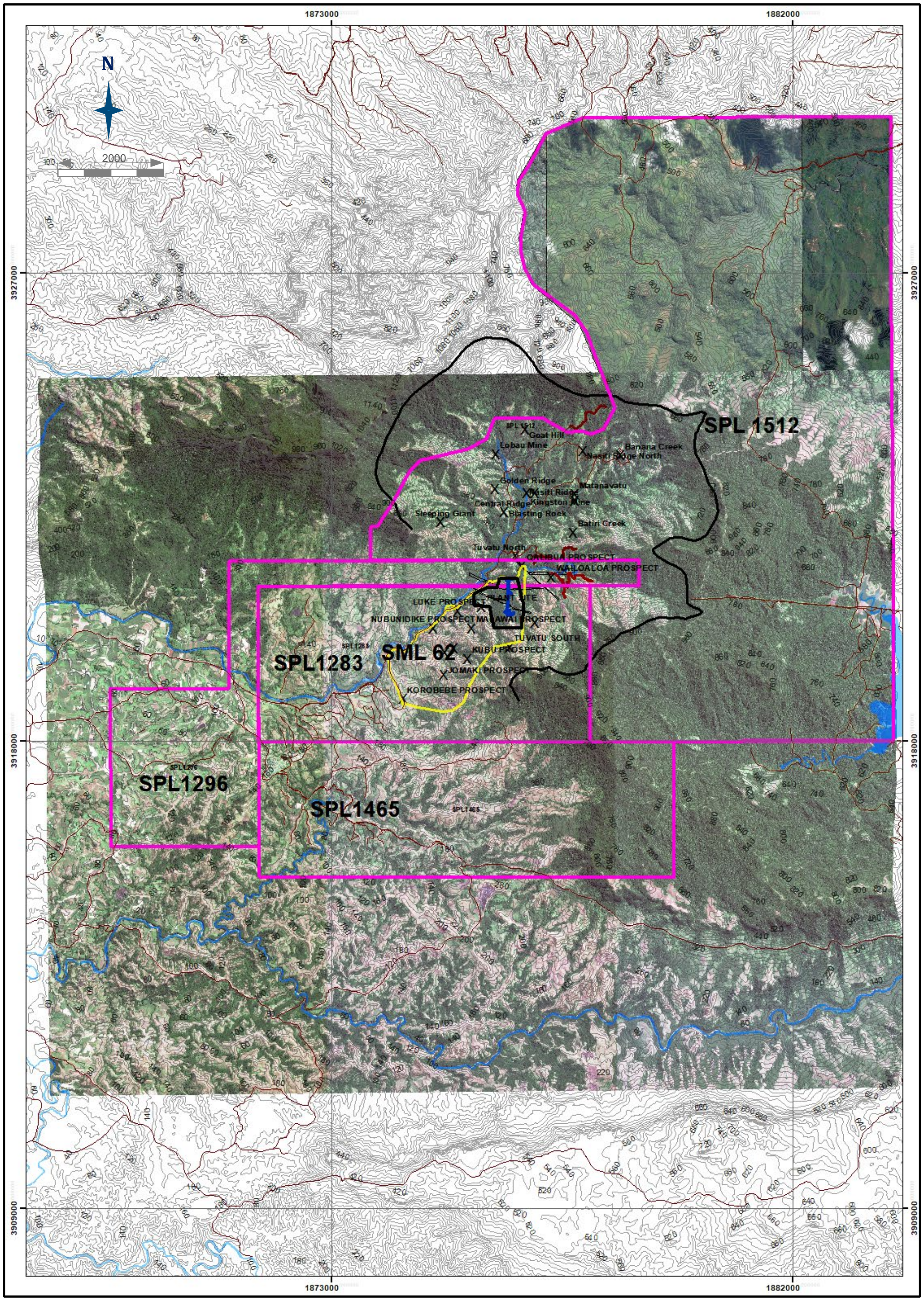
Source: MA (2018)

Figure 1-2: Tuvatu Lodes (oblique view [45°E -45°])

1.3 Property Description and Location

Lion One Limited is the registered holder of the following tenements (Figure 1-3) in Fiji, issued by the Government of Fiji's Mineral Resources Department in accordance with the Fiji Mining Act (1965). The Project has been fully permitted; however, MA has not undertaken any title search or due diligence to verify the Project permitting status. The tenements are listed below:

- Special Mining Lease 62 ("SML 62") effective until January 21, 2025
- Special Prospecting License 1283 ("SPL 1283") effective until August 23, 2025
- Special Prospecting License 1296 ("SPL 1296") effective until August 23, 2025
- Special Prospecting License 1465 ("SPL 1465") effective until March 4, 2025
- Special Prospecting License 1512 ("SPL 1512") effective until May 2024



Source: Lion One MRD Tenement File (2019)

Figure 1-3: Lion One Limited tenement map

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1.4 Mineral Resource

On January 2, 2018, Lion One engaged MA to prepare an updated Mineral Resource model suitable for mine design and scheduling of the resources for the Project.

Following the completion of the 2016/2017 diamond drilling program and field exploration, MA undertook this study to update the resources with the results of that drilling program and other work completed by Lion One to that date. In particular, drilling added significant additional information in the HT Corridor zone of mineralization (H and Tuvatu Lodes) and the Western Veins (which are interpreted to be the western extension of the Murau Lodes). Stricter parameters and tighter controls than those used for the 2015 estimate (which was put together for the 2015 PEA study [Freudigmann et al. 2015]) were used in this Technical Report. As a consequence of these tight controls, the Mineral Resource estimate related to some lodes was reduced in tonnes and/or grade.

The Mineral Resource has been estimated for each vein individually using Ordinary Kriging (OK) of width and grade, the latter using accumulations, into a three-dimensional (3D) block model.

The Project Mineral Resource is reported at one cut off (3 g/t Au) representing a Mineral Resource amenable to underground production. The total Indicated Resource is 1,007,000 t at 8.48 g/t Au for 274,600 oz of gold and an Inferred Resource of 1,325,000 t at 9.0 g/t Au for 384,000 oz of gold (Table 1-1).

Table 1-1: 2018 Tuvatu Mineral Resource estimate

Cut-off (g/t Au)	Resource Estimate Date	Resource Category					
		Indicated			Inferred		
		Material (t)	Grade (g/t Au)	Ounces (oz Au)	Material (t)	Grade (g/t Au)	Ounces (oz Au)
3	January 2018	1,007,000	8.48	274,600	1,325,000	9.0	384,000

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

The Mineral Resource has been reported at various cut-off grades to provide insight into the sensitivities of the resource (Table 1-2). The resource is reported at a 3 g/t Au cut off (Table 1-1). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under the parameters used.

Table 1-2: Tuvatu mineral reported above various cut-offs

Cut-off (g/t)	Indicated			Inferred		
	Material (t)	Grade (g/t Au)	Ounces (oz Au)	Material (t)	Grade (g/t Au)	Ounces (oz Au)
2.0	1,283,000	7.2	296,400	1,822,000	7.2	423,300
3.0	1,007,000	8.5	274,600	1,325,000	9.0	384,000
5.0	687,000	10.6	234,300	788,000	12.5	317,500

Notes: Numbers are rounded to reflect relative accuracy; numbers may not add due to rounding.

The effective date for the Mineral Resource estimate update is January 8, 2018.

The summary review of geology, Mineral Resource models and estimates, and the site visit were conducted by Mr. Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP) (Qualified Person [QP]) who visited the site from February 25 to 28, 2014; July 31 to August 5, 2017; and September 28 to October 3, 2017. Mr. Taylor viewed the geological setting, located some drill collars, and inspected drill core and sample storage.

Mr. Taylor has sufficient experience that is relevant to the Project's style of mineralization and deposits under consideration and to the activity that 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* (the JORC Code) (Australia) and as a QP as defined in National Instrument 43-101 (NI 43-101) (Canada). He is a member and certified professional of the Australasian Institute of Mining and Metallurgy (Melbourne). Mr. Taylor was employed by Mining Associates of Brisbane, Australia.

1.5 Mineral Processing and Metallurgical Testing

Extensive test work was conducted between 1997 and 2019, including mineralogy studies, comminution tests, gold recovery tests, and cyanide detoxification tests. In this Technical Report, test work, especially the test work conducted after the 2015 PEA (Freudigmann et al. 2015), has been reviewed in detail and summarized.

1.5.1 Mineralogy Studies

Previous mineralogical studies were conducted to identify major gold-bearing minerals, gangue minerals, and gold occurrences. The results indicate that the gold mainly occurs as free gold; gold-silver telluride; and associated gold with silicate, micas, pyrite, and other sulphides. The reported gold grain size is between 1 to 200 µm with significant gold grains between 1 to 30 µm.

Three mineralogy investigations were recently performed. The first was a petrologic study on core samples to generate geological information. The other two investigations, performed on head samples of gold recovery test work, confirmed the previous observations regarding gold/gangue mineral species. Additionally, a gold deportment study was completed on one of the blended samples, which shows a high presence of gold tellurides of 80% and varied gold grain sizes between 0.7 to 61.3 µm.

1.5.2 Comminution Tests and Simulations

Varied comminution testing programs were performed between 1997 and 2012 and mainly investigated the sample grinding competencies to the traditional tumbling mills and autogenous/semi-autogenous mills. Unconfined compressive strength (UCS), impact crushing work index (C_{Wi}), Bond rod mill work index (R_{Wi}), Bond ball mill work index (B_{Wi}), and abrasion index (A_i) were measured on various samples. The results indicate that tested samples possess medium to high impact strength, medium to high compressive/attrition strength, and low grindability. The samples tested can be considered as medium to low abrasive. Preliminary simulations of semi-autogenous grinding (SAG) and ball mill milling options were conducted and concluded that the two-stage ball mill milling appears more power efficient.

1.5.3 Gold Recovery Tests

Significant metallurgical test programs for gold recovery were completed via three major treatment routes by multiple laboratories:

- Route 1: Whole ore cyanidation
- Route 2: Gravity + cyanidation
- Route 3: Gravity + flotation + cyanidation.

These treatment routes also included various pre-treatments prior to cyanide leaching.

The Route 1 tests were conducted during the early-stage testing campaign. It was reported that gold recoveries varied from 56.5 to 98.3% on the samples grading between 1.17 to 136.9 g/t Au.

The Route 2 tests were performed during the early and recent testing campaigns, both of which confirmed the amenability of samples to this route. The overall gold recoveries from previous tests were between 53.9 and 97.2%. Preliminary optimization steps were conducted via the regrinding stage and longer leaching retention time process, but this can only marginally improve the overall gold recovery. In recent tests, Yantai Xinhai Mining Research & Design Co. Ltd. (Xinhai) studied a pre-oxidization treatment that appears to have a significant impact on increasing gold leaching recovery. At a grind size of 80% passing 75 µm and a 24-hour leach retention time, Xinhai reported that gold recovery increased to 87.2% from 76.9%. The highest gold recovery from Xinhai was 95.0% for the same feed size but a longer leach retention time of 72 hours. Bureau Veritas Commodities Canada Ltd. (BV)'s tests were based on a finer feed size and a longer leaching time than previous testing. At a primary size of 80% passing 20 µm and a 96-hour leaching process, the gold recovery can be as high as 94.3%.

The Route 3 tests were the most complicated and were conducted during early and recent testing campaigns. The overall gold recoveries ranged from 44.2 to 96.1% in previous tests and from 79.9 to 97.8% in recent tests. The recent higher gold recoveries were obtained using flotation followed by flotation concentrate regrinding to 80% passing approximately 20 µm and pre-oxidizing on the reground flotation concentrate and flotation tailings. The test results appear to show a longer leach retention time would improve gold extraction for the flotation concentrates.

A further test program (BV1803310) by BV using the Route 3 flowsheet on a composite representing the initial year mill feed, grading at 10.6 g/t Au. The test results show that with aeration pre-treatment with lime, or sodium hydroxide and peroxide, or ultra-fine regrinding on flotation concentrate, the overall gold recovery can be improved to greater than 90%.

1.5.4 Cyanide Detoxification Tests

Cyanide detoxification tests were conducted using sulphur dioxide (SO_2) / air technology by ALS Metallurgy and BV on varied leaching residue samples. Both studies indicate that the targeted weak acid dissociable (WAD) cyanide level of 1 mg/L can be achieved.

1.6 Mining

The Project is a planned underground mine, which is currently accessed through an exploration portal and decline that was initially mined in 1997. Currently, no mining is taking place, but access has been maintained for dewatering and follow-up geological sampling purposes. Lion One is currently completing site excavations to establish processing and mining facilities.

The Tuvatu deposit mineralization is primarily sub-vertical ranging from 70 to 80°, with less than 1% of stope tonnes contained in flat lying mineralization ranging from 0 to 30°. The veins are a series of parallel lodes with varying distance of separation of waste between lodes.

The proposed mining method is longhole stoping, with minor airleg stoping. Originally, the operation was envisioned as a handheld mining project due to the narrow veins; however, Lion One's production and financial targets have led to adopting mechanized longhole stoping as the primary stoping method. Stoping of flatter dipping mineralized areas (less than 1% of the stope tonnes), where longhole stoping is not viable, will be excavated via handheld airlegs.

The Tuvatu deposit is a high-grade, narrow vein deposit in competent ground. This mineralization style excludes many bulk mining techniques. Handheld airleg mining, mechanised longhole stoping, and mechanised cut-and-fill methods were considered as three viable methods for comparison. The high backfill cost associated with underhand cut-and-fill meant only overhand cut-and-fill was considered. The generally good geotechnical conditions allow for relatively small in situ pillars, reducing the benefits of a higher extraction, but more expensive backfill method.

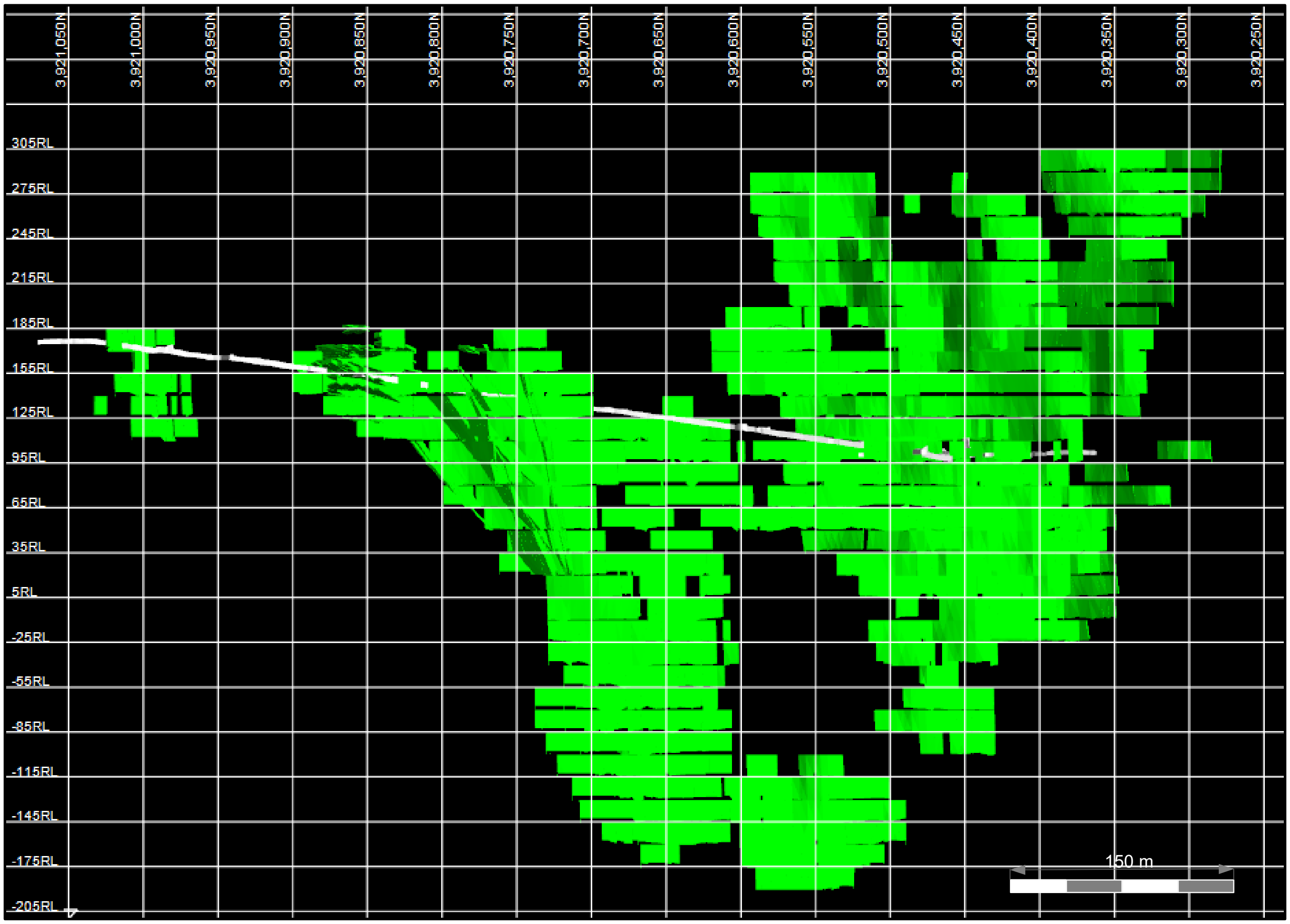
The three mining methods were assessed with a ranking system that took into account level spacing (lateral development cost), mineralized material extraction ratio, dilution control, productivity, production cost, geotechnical risk, and safety risk. The results of the quantitative analysis show longhole stoping to be the most suitable mining method.

Table 1-3 below highlights the input parameters used for the mineable stope optimization (MSO) used to generate stope shapes for the mine plan. The cut-off grade for the initial MSO software stope shape generation was derived from the 2015 PEA (Freudigmann et al. 2015) costs with updated revenue and recovery factors provided by Lion One.

Table 1-3: MSO software stoping parameters used for 15 m high stopes

Stoping Parameter	Unit	Value
Stoping Cut-off Grade	g/t Au	2.7
Minimum Mining Width	m	1
Vertical Level Interval	m	15
Section Length	m	5
Hanging Wall Dilution	m	0
Footwall Dilution	m	0
Minimum Parallel Waste Pillar Width	m	10
Minimum Footwall Dip Angle	degrees	40

Figure 1-4 shows the final stope shapes, which are the initial MSO software shapes trimmed to allow a sufficient crown pillar to the surface, pillars between parallel lodes, removal of uneconomic shapes, and removal of mining shapes that are impractical to mine.



Source: Entech (2019)

Figure 1-4: Final stope shapes after depletions

Table 1-4 shows the Tuvatu key annual physicals.

Table 1-4: Annual physicals

Physicals	Units	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Total Mill Feed Mined	t	32,150	311,553	361,192	337,138	297,533	44,501	1,384,067
Gold Grade Mined	g/t	7.77	7.42	8.50	10.40	8.09	7.31	8.57
Gold Mined	oz	8,032	74,356	98,596	112,718	77,364	10,466	381,532

1.7 Recovery Methods

The metallurgical test work as described in Section 13.0 was used as the basis for selecting the gold recovery methods. The test work indicates that the Tuvatu mineralization is amenable to gravity concentration and flotation followed by cyanidation processes. The process design criteria and process flowsheet have been developed for the process facility.

Lion One contracted Jinpeng Mining to provide process plant design and all process-related equipment quotations. The preliminary design work by Jinpeng Mining includes equipment sizing, flowsheet development, mass balance, general plant layouts, and circuit layouts.

1.7.1 Process Design Criteria

The average head grade of the plant feed is expected to be 8.6 g/t Au during the life-of-mine (LOM) of over five years. The overall gold recovery is estimated to be approximately 87.5%. The crushing circuit will operate during the day shift, while the milling and leaching circuits will operate 24 h/d and 330 d/a with an availability of 90.4%. Carbon stripping and gold electrowinning circuits will operate 16 h/d. The detailed key process design criteria are discussed in Section 17.0.

1.7.2 Process Description

The proposed process plant will process the mineralized material at a nominal rate of 1,000 t/d via sequential operations as shown in Figure 1-5.

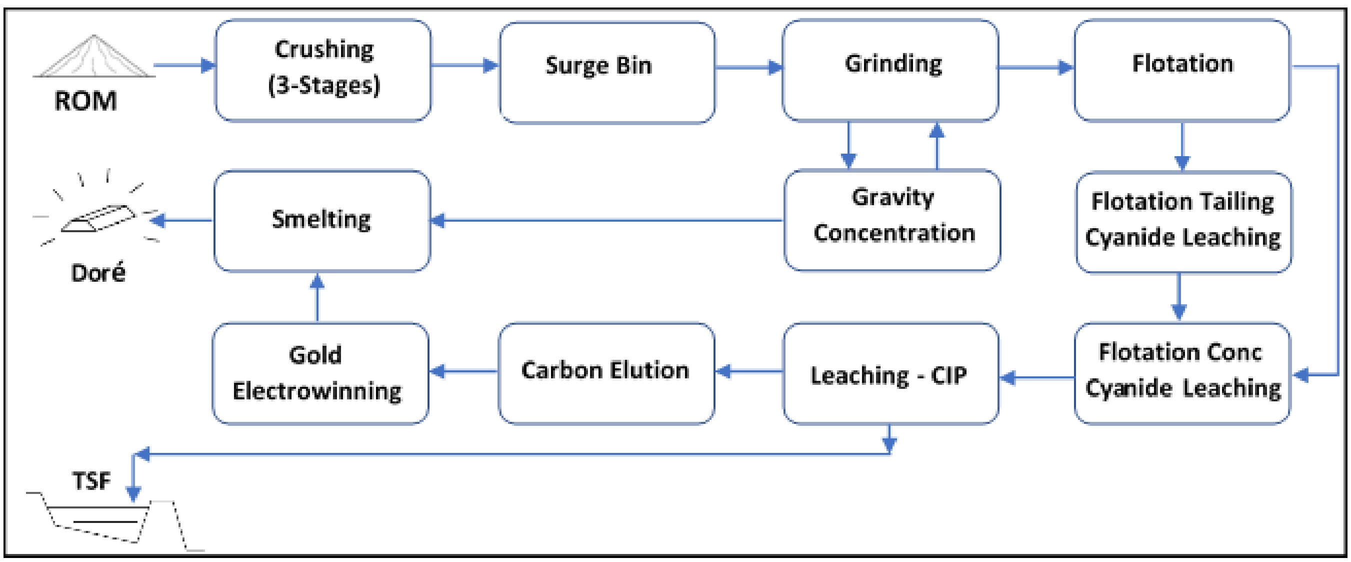


Figure 1-5: Simplified process flowsheet

The mill feed will be trucked from the underground mine and dumped onto mill feed surge stockpiles or directly into a primary feed dump pocket. The primary crushing circuit consists of three stages of crushing circuits where the mill feed will be crushed to a particle size of 80% passing 8 to10 mm. The crushed materials will be conveyed to a ball mill feed surge bin and then a two-stage grinding circuit to further reduce the particle size to approximately 80% passing 60 to 65 µm.

A gravity separation unit, installed in the primary grinding circuit, will treat approximately one third of the hydrocyclone underflow to recover the coarse-free gold grains. The recovered gold will be further treated by a tabling concentrator to upgrade the concentrate, which can be directly melted, while the tabling tailings is further processed by a centrifugal gravity concentrator to maximize free gold recovery.

The hydrocyclone overflow will be floated in rougher and scavenger circuits producing a gold-bearing concentrate and a tailings product. The flotation concentrate will be reground to 80% passing approximately 20 µm, thickened to approximately 45% w/w solids and then pumped to an aeration pre-treatment prior to a dedicated cyanide leaching process. The flotation tailings will be thickened to approximately 45% w/w solids and pumped to a separate tailing leaching circuit. Both the leached concentrate and tailings will report to a common gold carbon adsorption circuit.

The gold-loaded carbon will be sent to the on-site absorption-desorption-recovery (ADR) plant, which will use a non-cyanide stripping and electrowinning process in a pressurized and closed circuit. The gold-rich sludge will be washed from the steel cathodes and collected, then dried and smelted in an electrical furnace to produce gold doré. The gold doré will be stored in a safety vault within a secure and supervised area. The stripped carbon will be acid washed for reuse and periodically reactivated.

The leach residue from the carbon-in-pulp (CIP) circuit will be treated by cyanide destruction using the SO_2/air procedure prior to being pumped to the TSF.

1.7.3 Process Auxiliary Facilities

Plant air services, water supply systems, and reagent handling/storage system have been included in the process design. Low-pressure air will be provided by air blowers; high-pressure air will be provided from air compressors. One fresh water and two process water systems are proposed. Reagent handling and storage will be housed in a containment area.

A geochemical and metallurgical laboratory, which has been constructed at the Owner's office site in Nadi for supporting current exploration, will be used for assaying the routine samples for exploration, mining, processing, and environmental departments. The metallurgical laboratory is equipped with metallurgical test equipment and will perform metallurgical testing to optimize the process flowsheet and improve metallurgical performance.

1.7.4 Process Control and Instrumentation

The plant control system will consist of a distributed control system (DCS) with personal computer (PC) based operator interface stations (OIS) in the control room at the plant site. The plant control room will be staffed by trained personnel 24 h/d for monitoring the key process operations and key equipment within the plant.

1.7.5 Metallurgical Performance Projection

Based on the test work results and the proposed mine production schedule, gold recoveries to doré are projected on a yearly basis and shown in Table 1-5. Further metallurgical test work is recommended to better understand the metallurgical performance.

Table 1-5: Yearly gold production projection

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Totals
Total Mill Feed Mined	t	329,960	330,864	329,960	329,960	63,323	1,384,067
Gold Grade Milled	g/t	7.60	8.93	10.49	7.70	6.42	8.57
Gold Milled	oz	80,588	94,985	111,238	81,643	13,078	381,532
Gold Recovery	%	86.5	87.5	87.5	87.5	87.5	87.3
Gold Recovered	oz	69,708	83,112	97,334	71,438	11,443	333,035

1.8 Project Infrastructure

The Property is located 17 km by road from Nadi International Airport, Fiji with port facilities at Ba and Lautoka. It is accessed via Sabeto Road, which follows the Sabeto River Valley from its junction with Queen's Road, the primary access from the airport. Part of the access roads and bridges will need to be upgraded to allow for the future heavy freight.

1.8.1 Site Geotechnical Investigations

Wood reviewed the results of previous geotechnical investigations carried out by Entec Limited of Suva, Fiji (Entec); Knight Piésold, GBGMAPS of Australia; and Qingdao Geotechnical Investigation and Surveying Research Institute (Qingdao) of China between 2014 and 2018. The review by Wood is summarized below regarding geotechnical conditions at the process plant and TSF sites.

1.8.1.1 Process Plant Site and Adjacent Structures

The site of the process plant and crusher structures were selected and under active development at the time when the geotechnical review work by Wood was completed. The site preparation involved cut-and-fill earthworks with engineered perimeter slopes exceeding, at some locations, 10 m in height.

Excavation of test pits and borehole drilling were completed in the areas of the process plant, crusher, and screen structures, and in areas where structural foundations are proposed. Bedrock surface was found at various levels beneath the plant site, at relatively shallow depths in limited areas but sloping sharply toward the Sabeto River. Based on the results of recent geotechnical investigations, it appears that the proposed plant and crusher areas are underlain by deposits of weak (loose to soft) colluvial soils with varying thickness, eroded from higher elevations and carried / rolled down on the steep slopes of the area. These materials have been intermingled with alluvial soils deposited by the nearby creeks in the side valleys (Tuvatu and Murau Creeks, etc.), and at the toe of the main slope (Sabeto River). At the proposed screening plant, no bedrock was encountered within the investigated depth of 20 m below grade, while the thick overburden comprised very weak soils to significant depth, more than 10 m below grade. It is considered that the soil matrix is generally weak and compressible within large areas of the process plant complex. Similarly, the most recent geophysical tests found that weak soils are also encountered beneath the proposed conveyor belts, the screening plant, and beneath a large portion of the process plant, particularly at the thickener and storage tanks, which are located close to the Sabeto River.

In the relevant foundation recommendations, Entec has proposed a maximum allowable bearing capacity of 100 kPa in the 2014 geotechnical report and a wide range of bearing capacities in the 2017 report. Both reports recommended that the foundation must be constructed below all loose/soft soils and uncontrolled fills. A large quantity of weak soils was suggested to be removed and replaced by large and thick mass concrete, or alternatively, pile foundations in critical areas should be considered. Qingdao investigated the intercepted soils and rock strata based on their test pit and borehole results. They estimated a bearing capacity value between 150 and 260 kPa, that is equivalent to "allowable" bearing capacity for "serviceability limit state" according to Qingdao, based on the Chinese Building Code.

Based on the results of the geotechnical investigations, it is evident that the bearing capacity for foundation design is highly variable across the site. Wood recommends that the results of the geophysical survey program, together with the historical borehole results should be used to define the extent of soil improvement or soil removal areas, and the size (thickness and horizontal extent) of any engineered fill for the structures and foundation members individually. Recommendations by Wood to improve site-bearing capacities are presented in Section 18.3.1.

1.8.1.2 Tailings Storage Facility Site

Various geotechnical field-testing programs were conducted by Entec of Suva at the proposed TSF site, including bore drillings and test pits in 2013/2014 and 2018; as well as geophysical investigations by GBG MAPS of Australia in 2018. The results are summarized in this PEA Update design of the TSF by Wood and presented in this Technical Report.

The relevant geotechnical tests in 2013/2014 and 2018 indicated that, within the investigated depths, the overburden of the proposed TSF mainly comprises clayey silt and silt with silty clay lenses. The overburden thickness varies across the TSF site and can be up to more than 10 m in some areas. The soil materials were recorded as having moderate to high plasticity, and the consistency was reported as generally stiff, ranging from firm to hard. Beneath the overburden, weathered rock was encountered with various thickness. Further geotechnical tests in 2018 confirmed the previous observations and concluded the clay-rich soil is of very low permeability, which is suitable for use as a tailings basin and to use a dam core to control seepage. In addition, the geophysical survey programs were carried out to determine the in situ seismic shear and compressional wave velocities within the first 30 m of the ground. The results were used to help delineate the profile and removal of the soft overburden of the TSF site.

Based on the geotechnical and geophysical results, Wood determined that the proposed TSF is suitable for a wet, slurry tailings deposit, while additional information and details need to be obtained to develop the full TSF prefeasibility and detailed design. Wood's recommendations are discussed in Section 26.0.

1.8.2 On-site/Off-site Facilities

The Project site comprises steep topography coupled with multiple creek lines that flow into the Sabeto River. Currently on the site, there are core storage facilities and associated infrastructure for exploration activities, and a decline, which was built by Emperor Gold Mining Company Limited in 1999. In addition, Lion One maintains an operations office in Nadi. A geochemical and metallurgical laboratory has been constructed in Nadi to service current exploration activities and future site operations.

The following on-site services and facilities are considered for this Project (detailed information can be found in Section 18.0):

- Access roads and site roads
- Water supply system
- Power generation and distribution system
- Mine facilities
- Process facilities
- TSF
- Ancillary facilities including:
  - Administration, security, and emergency medical facilities
  - Maintenance shop and warehouse
  - Mine dry and mine truck shop
  - Site water management facility (WMF)

1.8.3 Tailings Storage Facility

The TSF will contain tailings from the process plant for a design capacity of 2,555 kt in dry solids mass. Higher capacity is achievable with substantially higher earthwork quantities. Wood designed the TSF according to the local government guidelines adopted from *Guidelines on Tailings Dams* established by the Australian National Committee on Large Dams (ANCOLD) in 2012 as well as the Canadian Dam Association (CDA) standards in 2014 since Lion One is a Canadian mining company.

Various test programs were conducted on the tailings samples to facilitate the preliminary design. The results indicate that the future tailings materials are potentially acid generating (PAG) and/or metal leaching (ML). Only, limited geochemical information is available. A deposition plan has been included in this preliminary design for acid rock drainage (ARD) management. Settling tests and drying tests were performed to determine tailings densities and specific gravity (SG) levels.

The TSF dam will be constructed primarily with geochemically acceptable mine rock and natural clay overburden and weathered rock materials borrowed from the TSF valley. The starter dam will be constructed for the Year 1 production, then raised in approximately 5 to 6 m following the first year of operation using centerline construction method. The yearly raise height will be decreased to 1 to 3 m for the final years of tailings disposal operation.

A tailings reclaim pond will be formed to the back of the TSF basin. A floating pump barge is proposed to send tailings water back to the process plant for reuse. A settling control pond will be constructed to collect water samples for monitoring contact water quality and to receive potential seepage from the TSF. Water will be released to the environment when it meets the required discharge criteria. Should the collected water not meet discharge quality, it will be pumped back to the TSF pond for reuse in the milling process.

1.8.4 Hydrology Study

Fiji has a warm tropical climate with maximum daily temperatures ranging from 28 to 32°C. The minimum daily temperature ranges from 18 to 23°C.

Mean monthly evaporation varies from a low of 130 mm in June to a maximum of 210 mm in December. Rainfall exceeds evaporation in the wet months from December to April, but evaporation exceeds rainfall in the dry months.

In 2017, Lion One retained SMEC Australia (SMEC), a member of the Surbana Jurong Group, in association with Entec, to undertake hydrologic investigations and preliminary drainage infrastructure design for the Project based on the local climate information, precipitation data, and the stream flow data of the rivers/creeks.

The study indicates that diversions are required upstream of the process plant area to direct hillside runoff around the process plant platform.

The culverts for two waterways (Tuvatu Creek and the Western Drain) that cross the mine access road adjacent to the process plant were designed. Tuvatu Creek has a catchment area of 82 ha and Western Drain has a catchment area of 6.4 ha. Both drains have very steep catchments with average slopes of 35 and 40%, respectively.

A hydraulic model was established using the Hydrologic Engineering Center (HEC)-River Analysis System (RAS) modelling software to determine the flood levels for channels adjacent to the site. The analysis showed that:

- The proposed finished levels for the process plant area are expected to be safe above the 100-year average reoccurrence interval (ARI) flood levels in Sabeto River.
- According to the current site layout design, the platform for the process plant along the Tuvatu Creek will be located at the 139 m level. The new crusher and screening plant pad at the west side of the Tuvatu Creek will be located at the 141 m level. The location of the previous crusher pad Platform 2 will be at the 148 m level. With this design, the process plant is expected to not to be inundated by the Tuvatu Creek 100-year flood.

1.8.5 Water Management

The site surface water management plan was developed for the Project by considering the high rainfall expected at the site and the requirement for full pond coverage of the tailings to mitigate acid generation. The plan incorporates the following aspects:

- TSF rainfall runoff and evaporation
- Predicted tailings supernatant release and achieved densities from subaqueous deposition
- Process plant site water demand
- External storm water runoff from the upstream catchment

Further details are discussed in Section 18.7.

1.9 Environmental

Environmental studies conducted for the Project include physical, biological, and social-cultural aspects. The greatest environmental concerns are water quality and fresh water flora and fauna, which can be mitigated by engineering design and management plans. Lion One has developed plans for waste management, ARD, and rehabilitation. Section 20.0 outlines the brief summaries of the environmental studies and management plans by Lion One.

The status and on-going activities of the permitting applications are shown as follows:

- On September 27, 2013, the approval of the initial environmental impact assessment (EIA) was granted for the Project. As part of the approvals process, Lion One agreed and prepared a Construction Environmental Management and Monitoring Plan (CEMMP).
- In April 2018, Lion One submitted a supplemental EIA to divert Tuvatu Creek, which is necessary for the rerouting of Navilawa Road and subsequent restriction of public access to the Tuvatu plant site.
- On May 29, 2018, the approval of the Tuvatu Creek diversion EIA was granted.

1.10 Capital and Operating Costs

A PEA is preliminary in nature, and includes Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the Technical Report will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The capital cost and operating estimates shown in this section for this Technical Report are based on the 2020 costing. No update for the cost estimates has been conducted since the 2020 PEA.

The capital and operating cost estimates for the Project are based on inputs from Lion One, Entech, and Wood, engineering design, quotations where applicable, and in-house data from Tetra Tech.

1.10.1 Capital Cost Estimates

The capital cost estimates were prepared with an accuracy range of +35%/-30%, including mining, processing, tailings management, overall site infrastructures, commissioning, and related indirect costs. The overall capital cost is estimated at USD$66.8 million (CDN$89.1 million) and is summarized in Table 1-6.

Table 1-6: Capital cost summary

Area		Cost (USD$ million)*	Cost (CDN$ million)*
Direct Costs			
10	Overall Site	3.2	4.3
30	Underground Mining	20.8	27.8
40	Process	13.7	18.2

table continues…

Area		Cost (USD$ million)*	Cost (CDN$ million)*
50	TSF**	4.1	5.4
70	On-site Infrastructures	1.8	2.4
Direct Cost Subtotal		**43.6**	**58.1**
Indirect Costs			
X	Project Indirect Costs	11.5	15.4
Y	Owner's Costs	4.8	6.4
Z	Contingencies	6.9	9.2
Indirect Cost Subtotal		**23.2**	31.0
Total		**66.8**	89.1

Notes: *Numbers may not total due to rounding.
**Estimate based on Wood's Material Take-off Revision F, dated January 23, 2019.

1.10.2 Operating Cost Estimates

The operating cost estimate was prepared with an accuracy range of +35%/-30%, including mining, processing, site servicing, and G&A costs with related freight costs. At a mill feed rate of 1,000 t/d, the total operating cost is estimated at USD$97.35/t (CDN$129.81/t) of mill feed processed as shown in Table 1-7. The cost distribution for each area is shown in Figure 1-6.

Table 1-7: Operating cost summary

Area	Operating Cost (USD$/t milled)*	Operating Cost (CDN$/t milled)*
Mining**	47.24	62.99
Process	41.49	55.33
Reclaim Water Handling	0.30	0.40
G&A	6.66	8.88
Site Services	1.66	2.21
Total Operating Cost	**97.35**	**129.81**

Notes: *Numbers may not total due to rounding.
**LOM average, excluding pre-production related costs.

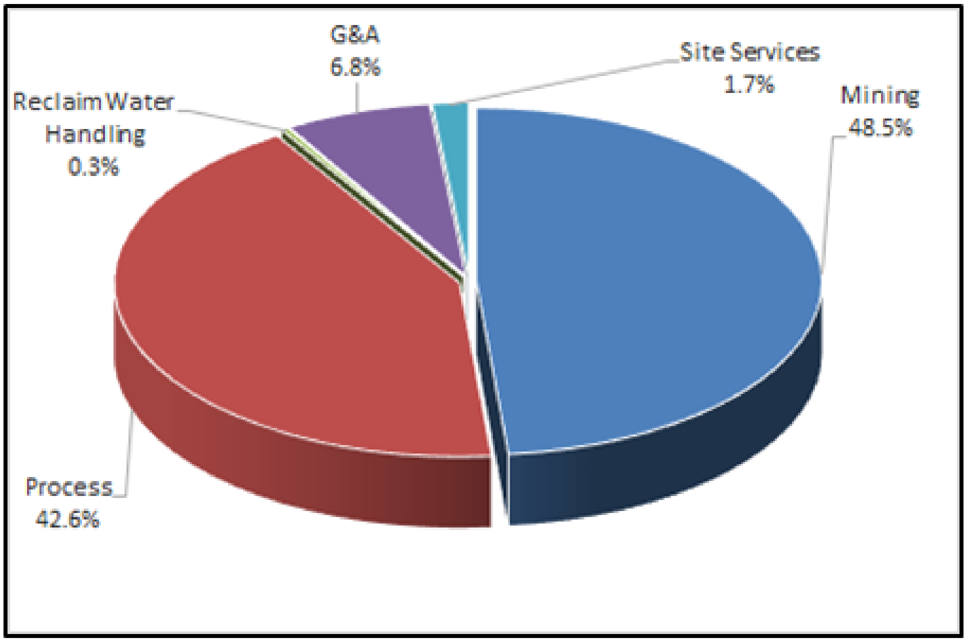


Figure 1-6: Cost distribution in various areas

1.10.3 Foreign Exchange

Table 1-8 shows the foreign exchange rates used for the capital and operating cost estimates.

Table 1-8: Foreign exchange rates

Base Currency (CDN$)	Other Currency
1.00	USD$0.75
1.00	RMB¥5.15
1.00	FJD$1.60

1.11 Economic Analysis

A PEA is preliminary in nature and includes Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the Technical Report will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The financial analysis data shown in this section for this Technical Report are based on the 2020 costing. No update for the financial analysis has been conducted after the 2020 PEA.

An economic model was developed for the Project, based on a 1,000 t/d mill feed from an underground mine of a five-year LOM plan. Material information in this report was compiled from data estimated by Lion One and external consultants based on the best available information at the time. Financial results are based on the following key assumptions (detailed in Section 22.0):

- Gold price of USD$1,400/oz
- Discount rate of 5%
- Mill feed rate of 1,000 t/d
- Preliminary mine plan

At a discount rate of 5%, the after-tax net present value (NPV) of the Project is estimated to be USD$121.7 million. The after-tax internal rate of return (IRR) is estimated to be 50.9%, and the payback period was estimated to be 1.7 years (after-tax). Section 22.0 presents the outcomes of the financial model and economic analysis.

The key financial results are shown in Table 1-9.

Table 1-9: Summary of financial results

Financial Summary	Units	Value
LOM	years	5
Annual Nominal Tonnage Processed	kt	330
Tonnes Processed	kt	1,384
Average Head Grade	g/t Au	8.57
Gold Recovered, including Refining Loss	'000 oz	331.4
Doré Refining Cost, including Assay and Insurance	USD$ millions	0.96
Net Revenue from Sales	USD$ millions	432.8
Capital Costs		
Pre-production Capital Costs	USD$ millions	66.8
LOM Sustaining Costs	USD$ millions	25.2

table continues…

Financial Summary	Units	Value
Cash Flow		
Royalty	USD$ millions	30.2
Pre-tax Cash Flow	USD$ millions	202.8
Taxes	USD$ millions	42.0
After-tax NPV at 5%	USD$ millions	121.7
After-tax IRR	%	50.9
Payback Period (after tax)	years	1.7

Note: Values may not sum perfectly due to rounding.

1.12 Recommendation

The Project is considered to be potentially economically viable based on this Technical Report. Lion One has received approvals for the initial and updated environmental assessments. It is recommended to advance the Project to the feasibility study. Recommendations are further detailed in Section 26.0.

2.0 INTRODUCTION

The Project is a high-grade, narrow-vein gold deposit located on hilly topography in the upper reaches of the Sabeto River Valley, which is approximately 24 km northeast of Nadi on the west coast of Viti Levu and 19 km by road from the Nadi International Airport.

This Technical Report was prepared for Lion One to summarize the project updates after the 2020 PEA. The updates incorporate geological exploration activities completed after September 25 2020. In 2020 Lion One commissioned a team of Mineral Resource estimate and engineering independent consultants to complete the 2020 Technical Report in accordance with NI 43-101 *Standards of Disclosure for Mineral Projects*.

The independent consultants are listed below:

- MA – Geology and Mineral Resource estimate and related information
- GeoSpy – Geology, exploration
- Entech – Mining and mining-related operations, underground geotechnical investigations, mining-related capital and operating cost estimates
- Tetra Tech – Metallurgical test work review, process and process-related cost estimates, G&A and surface service operating cost estimates, site infrastructures (excluding site geotechnical investigation and TSF), and environment
- Wood – Site geotechnical investigation and TSF

After September 25, 2020, Lion One conducted further geological exploration work which has been incorporated into this report.

2.1 Qualified Persons

A summary of the QPs responsible for this Technical Report is provided in Table 2-1.

Table 2-1: Summary of Qualified Person responsibilities

Technical Report Section		Company	Qualified Person
1.0	Summary	All	Sign-off by Subsection
2.0	Introduction	All	Sign-off by Subsection
3.0	Reliance on Other Experts	All	Sign-off by Subsection
4.0	Property Description and Location	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
6.0	History	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
7.0	Geological Setting and Mineralization	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)

table continues…

Technical Report Section		Company	Qualified Person
8.0	Deposit Types	GeoSpy	Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM (Geo)
9.0	Exploration	GeoSpy	Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM (Geo)
10.0	Drilling	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
11.0	Sample Preparation, Analyses, and Security	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
12.0	Data Verification	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
12.3	Site Visits	All	Sign-off by Subsection
12.4	Opinion of Qualified Persons	All	Sign-off by Subsection
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
14.0	Mineral Resource Estimates	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
15.0	Mineral Reserve Estimates	Entech	Shane McLeay, B.Eng. Mining (Hons), F.AusIMM
16.0	Mining Methods	Entech	Shane McLeay, B.Eng. Mining (Hons), F.AusIMM
17.0	Recovery Methods	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
18.0	Project Infrastructure		
18.1	Tuvatu Site Description	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.2	Site Development	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.3	Site Geotechnical Investigations	Wood	Laszlo Bodi, M.Sc., P.Eng.
18.4	Roads	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.5	Hydrological Study	Tetra Tech	Davood Hasanloo, M.Sc., M.A.Sc., P.Eng.
18.6	Water Supply	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.7	Site Water Management	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.7.1	Tailings Storage Facility Water Management	Wood	Norman Schwartz, M.Sc.Eng., P.Eng.
18.8	Tailings Storage Facility	Wood	Laszlo Bodi, M.Sc., P.Eng
18.8.4	Storm and Dry Year Events Data	Wood	Norman Schwartz, M.Sc.Eng., P.Eng.
18.9	Power Supply	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.10	Communications	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.11	Logistics and Other Infrastructure	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
18.12	Rehabilitation and Closure	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
19.0	Market Studies and Contracts	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.

table continues…

Technical Report Section		Company	Qualified Person
20.0	Environmental Studies, Permitting and Social or Community Impact	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
21.0	Capital and Operating Costs		
21.1	Capital Cost Estimate	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
21.1.2	Mining Capital Cost Estimate	Entech	Shane McLeay, B.Eng. Mining (Hons), F.AusIMM
21.2	Operating Cost Estimate	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
21.2.1	Mining Operating Cost Estimate	Entech	Shane McLeay, B.Eng. Mining (Hons), F.AusIMM
22.0	Economic Analysis	Tetra Tech	Maureen Phifer, P.Eng., B.Sc.
23.0	Adjacent Properties	MA	Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
24.0	Other Relevant Data and Information	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
25.0	Interpretation and Conclusions	All	Sign-off by Subsection
26.0	Recommendations	All	Sign-off by Subsection
27.0	References	All	Sign-off by Subsection

2.2 Site Visits

The following QPs conducted personal inspections of the Property and metallurgical test laboratories.

2.2.1 Site Visits by Mr. Ian Taylor

Mr. Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP), visited the Tuvatu deposit from February 25 to 28, 2014; July 31 to August 5, 2017; and September 28 to October 3, 2017. In the course of the site visits, Mr. Taylor viewed the mineralized drill core and examined the drill core processing and storage facilities. He also viewed and sampled the mineralized vein systems and outcrops and inspected decline dewatering and the development of the SKL Lodes. The UR2 and UR5 development drives were inaccessible in October 2017.

2.2.2 Site Visits by Dr. Darren Holden

Dr. Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM (Geo), visited the Property on February 16 to 23, 2020; December 1 to 7, 2019; October 27 to November 3, 2019; September 18 to 30, 2019; June 29 to July 7, 2019; April 7 to 14, 2019; March 7 to 15, 2019 and a total of 12 other times, for approximately 7 to 10 days each time, during 2017 and 2018. The purpose of these visits were to conduct exploration, mapping, data reviews and reporting, and develop sampling protocols.

2.2.3 Site Visit by Mr. Laszlo Bodi

Mr. Laszlo Bodi, M.Sc., P.Eng., visited the site from April 4 to 7, 2018. Mr. Bodi viewed the general process plant and TSF areas and reviewed available geotechnical information concerning the geotechnical conditions for various structures within the plant site.

2.2.4 Metallurgical Laboratory Visits by Dr. Jianhui (John) Huang

Dr. Jianhui (John) Huang, Ph.D., P.Eng., visited metallurgical testing laboratories including BV on August 8, 2018; Jinpeng Group on December 2, 2017; and Xinhai on December 4, 2017 to inspect test equipment, test procedures, and discuss test results with the technicians/engineers at the laboratories.

2.3 Effective Date

The effective date for this Technical Report is April 29, 2022. The effective date of the Mineral Resource Estimate is January 8, 2018.

2.4 Sources of Information

All sources of information for this Technical Report are noted in Section 27.0.

2.5 Units of Measurement and Currency

All measurements are reported in metric units, unless otherwise noted.

Canadian dollars (CDN$), US dollars (USD$), Fiji dollars (FJD$), and Chinese Renminbi (RMB¥) have been used in the cost estimates for preparing this Technical Report. All currencies have been converted to US dollars for financial analysis based on the currency exchange rates stated in this Technical Report, unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report followed standard professional procedures in preparing the contents of this Technical Report. Data used in this Technical Report have been verified where possible. The authors have no reasons to believe that the data was not collected in a professional manner.

Technical data provided by Lion One for use by the authors in this Technical Report is the result of work conducted, supervised, and/or verified by Lion One's professional staff or consultants retained by Lion One.

3.1 Mineral Tenure and Ownership

Mr. Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, M.AusIMM (CP), relied on Lion One for information regarding mineral tenure and ownership of surface rights in Section 4.0.

3.2 Environmental Studies

Dr. Jianhui (John) Huang, Ph.D., P.Eng., relied on Lion One and Ms. Tania Perzoff, M.Sc., R.P.Bio., a senior regulatory specialist of Tetra Tech, on matters relating to the environmental permitting plan and social or community impact in Section 20.0.

3.3 Economic Analysis

Ms. Maureen Phifer, P.Eng., B.Sc., relied on Mr. Tony Young, CPA, CA, CPA (Illinois), Chief Financial Officer of Lion One, for tax matters and calculations (the tax portion of the economic analysis) relevant to the Technical Report and detailed in Section 22.0.

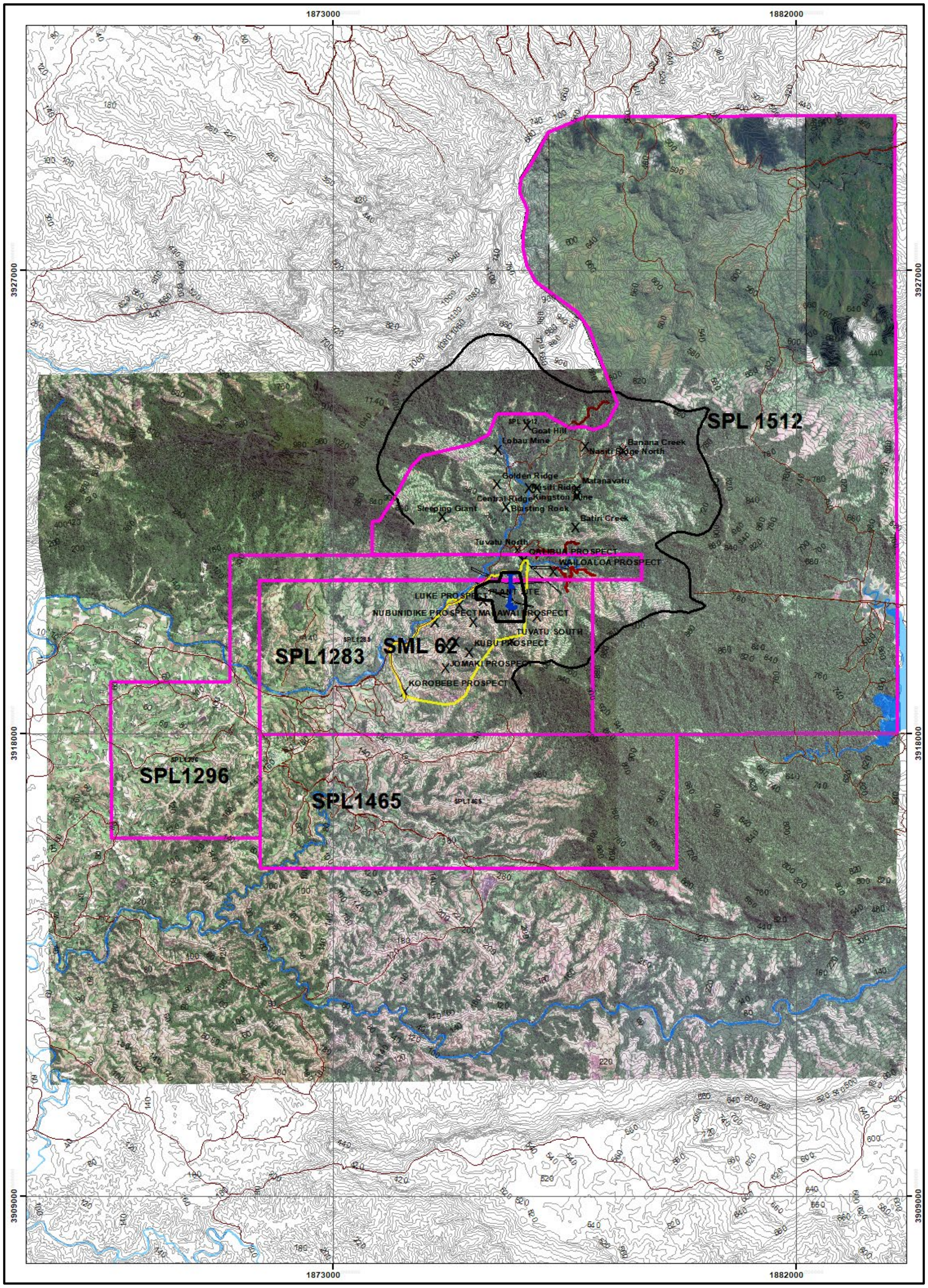
4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Area

Title to four SPLs (total area 20,786 ha) is held by Lion One Limited, a subsidiary company of Lion One Metals Limited (Lion One) (Table 4-1, Figure 4-1). A Special Mining Lease (384.5 ha) covering the current Mineral Resource for the Project was granted to Lion One on January 22, 2015. The status of the tenements has not been independently verified by any external source, but the license documents and other correspondence from the Mineral Resources Department, Fiji (MRD) has been supplied by Lion One for reference and confirmation. Letters from the MRD to Lion One have been viewed.

Table 4-1: Tenement details

Tenement	Area Size (ha)	Annual FJD Expenditure Requirement	Date of Grant	Term	Interest (%)
SPL 1283	1,951	$1,400,000	September 19, 2013	August 24, 2020 – August 23, 2025	100
SPL 1296	1,315	$1,600,000	September 19, 2013	August 24, 2020 – August 23, 2025	100
SPL 1465	8,900	$3,050,000	March 5, 2022	March 5, 2022 – March 4, 2025	100
SPL 1512	8,620	$15,333,305	May 14, 2019	May 14, 2019 – May 13, 2024	100
SML 62	384.5	N/A	January 22, 2015	January 22, 2015 – January 21, 2025	100



Source: Lion One MRD Tenement File (2019)

Figure 4-1: Tenement lease boundaries of Tuvatu Gold Project

4.2 Property Location

The tenements are located in the upper reaches of Sabeto Valley, approximately 24 km northeast of the town of Nadi on the west coast of Viti Levu and 17 km from the Nadi International Airport. The Tuvatu gold deposit is located within SML 62, which is surrounded by SPL 1283 and SPL 1296. SPL 1512 is a contiguous lease extending north, covering the majority of the crater of the Navilawa Caldera. SPL 1465 is a contiguous lease extending south, to cover additional prospective geology and to cover the area previously demarcated for a tailings dam by Tuvatu Gold Mines (TGM) in its 2000 mining study (TGM 2000).

4.3 Property Ownership, Rights, and Obligations

Title to the Property is held by Lion One Limited, a Fijian subsidiary of Lion One.

There are three classifications of land in Fiji: native land, crown land, and freehold land. The Project area lies mostly, 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. Approximately 5% of the SPLs are under cane lease through the *Agricultural Land and Tenants Act*.

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, (approximately 95%) of 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

There are no option agreements or joint venture terms in place for the Property. There are no known obligations on ground covered by claims comprising the Property.

In 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 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 Taueki 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 Agreement between Lion One and the TLTB is required prior to obtaining a mining lease from the MRD.

Under the terms of the Surface Lease Agreement, Lion One must make a one-time payment of FJD$1,000,000 of which FJD$730,337 (CDN$427,348) was paid upon acceptance of the Surface Lease Agreement (May 16, 2014) and the balance of FJD$300,000 (CDN$175,260) was due upon the first gold production from mining operations at Tuvatu. As a show of good faith, and taking into consideration the health of two of the landowners, Lion One paid FJD$251,000 of the balance of FJD$300,000 to the respective landowners on March 6, 2019. An additional lease payment of FJD$30,000 (CDN$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 SML 62 in June 2014 (Freudigmann et al. 2015). Compensation agreements between Lion One and the native title landowners in the Navilawa area (SPL 1512) have been signed in the period since June 2019.

4.5 Environmental Liabilities

Lion One 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 license implies a right to proceed to eventual project development. This is subject to the license 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

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

The Project lies on the west coast of Viti Levu, 24 km northeast of Nadi town and approximately 17 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 Hydroelectricity power line can be seen from the road and crosses over the Project area. Further along the Sabeto Road, the road forks, with the left fork going to Korobebe village, and onto Navilawa village.

SPL 1283 and SPL 1296 cover land areas in the upper catchment of the Sabeto River immediately south of Navilawa village. SPL 1512 adjoins the northern boundary of SPL 1296 and covers the Navilawa Caldera, including catchments on the north side of the Sabeto range. 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, Fiji Airways, and Air New Zealand. On several days in each week there are flights to other New Zealand cities, Los Angeles and San Francisco in USA, South Korea, Hong Kong, and China, in addition to other Pacific islands. The Project 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 east-southeast during all seasons. Maximum temperatures average 28 to 30°C for the cooler months (May to October) while November to April temperatures are higher (31 to 32°C) with heavy downpours. Monthly minimum temperatures vary between 18 and 23°C.

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 northwest tracks give some increased risk of damage in the outlying northwest island groups.

Viti Levu's climate is dominantly controlled by oceanic temperatures and winds, restricting the diurnal temperature range heavily; the average daily range is 8.5 to 10.3°C. Average minimum temperatures for Nadi range from 18 to 23°C, while average maximum temperatures range from 28 to 32°C; these temperatures can be expected to be 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 to 325 mm during the December to March wet season.

5.3 Local Resources

The Project 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 Project area are Lautoka, Nadi, and Ba (Table 5-1). Lautoka, Fiji's second-largest city, is located 30 km from the Project. The local economy 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 Project 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 the coastal Nadi-Lautoka-Ba region. There are also experienced former mine workers from the Vatukoula Gold Mine.

Table 5-1: Population centers (2017 Census)

Town	Population	Principal Economic Activity
Lautoka	52,500	Agriculture, tourism, fishing
Nadi	42,284	Tourism, manufacturing
Ba	14,596	

Source: Fiji Bureau of Statistics (2017 Census)

5.4 Infrastructure

Fiji has one of the most developed economies of the Pacific islands, although a large subsistence sector still exists. Sugar exports, remittances from Fijians working abroad, and a growing tourism industry (with approximately 750,000 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 Project other than a small exploration facility. Local villages utilize 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 2G/3G mobile-phone reception.

Energy Fiji Ltd. (EFL) holds the monopoly in all facets of the energy sector, including generation, transmission, and distribution. It was formerly called the Fiji Electrical Authority. Hydroelectric and diesel are the two sources of power generation for EFL. Its installed power generation capacity currently stands at 237 MW; however, rising use of electricity has prompted the government to call for submissions from independent power producers. EFL has an 11 kV line at Korobebe village, which could supply 2 MW of power. This line could be upgraded by EFL to 33 kV from the Sabeto turnoff to the mine site. The villages around the Project chiefly utilize fuel wood and small diesel generators.

5.5 Physiography

The upland areas of the Project area are grassland. Stream valleys and their perimeters are heavily vegetated. Several intermittent and perennial streams are located within the exploration tenements and mining lease area. Sabeto River is the largest perennial water feature in the Project area. Elevations of the Property range from 50 m to a maximum of 700 m above mean sea level. The area is hilly with slopes of 15 to 30% being common.

6.0 HISTORY

6.1 Tuvatu Area 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 Prospecting License (PL) 689. Later work in the area was undertaken by the Nadele Syndicate.

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

In 2007, following the closure of the Vatukoula Gold Mine, Emperor Gold Mining Company Limited (at the time a subsidiary of DRD) sold its Fijian assets, including the Tuvatu Property, to Westech Gold Pty Ltd and Red Lion Management Ltd. Licenses covering the Property were reissued in the name of Lion One by the Fijian Government. Subsequently, American Eagle Resources gained control of Lion One, the holder of the Project. Lion One 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.

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 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 three phases of exploration at the Property. The Phase 1 program, 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 induced polarization (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; however, the feasibility study was suspended in late 2000 as part of a general cost-cutting exercise by Emperor Gold Mining Company Limited 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, remapping 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, during this time there had been three programs of drilling at the Property 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 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 Tuvatu Mining Lease Area Regional Exploration

Only limited regional exploration had been carried out in the area by explorers (primarily Aquitaine Fiji) before TGM's work. During 2001 to 2003, a regional exploration program was carried out by TGM that involved regional mapping, trenching, stream sediment, and soil sampling. This work identified more than ten new prospect areas outside the Project 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 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 DD holes (DDHs) were completed in October 2008 at the Nubunidike Prospect to test the Nubunidike / Hornet Creek / 290 Vein system.

6.2.3 Navilawa Area Exploration

Prior to the grant of SPL 1512 to Lion One in May 2019, the Navilawa SPL 1412 enclosed the majority of the under-explored Navilawa Caldera, to the north and directly along strike from the Project. It was considered prospective for similar high-grade epithermal, porphyry, and alkaline gold mineralization. Previous exploration on this area is summarized in Table 6-1.

Table 6-1: Navilawa area previous exploration summary

Year	Company	Prospects	Geochemistry	Geophysics	Drilling
1906–1923	-	Kingston Mine, Central Ridge, Blasting Rock, Nasiti Ridge, Qalyalo, Vunatawa, Golden Ridge	Bulk Samples, Trial Mining	-	-
1943–1947	South Pacific Mining	Kingston Mine, Central Ridge	Rocks	-	3 DDH
1963	Higgs & Coulson	Kingston Mine, Central Ridge	Streams, Soils	-	-
1963–1964	Geological Survey of Fiji	Kingston Mine, Central Ridge, Blasting Rock	Streams, Soils	-	9 DDH (393 m)
1968–1969	Amad JV with Ah Koy Mining Syndicate	Central Ridge, Nasiti Ridge	Streams (36), Soils (378), Rock (16)	Ground: IP	-
1970–1976	Barringer/Amad	Kingston Mine, Central Ridge, Nasiti Ridge	Soils (804)	Ground: Magnetics & IP Airborne: Magnetics	5 DDH (731 m) No gold assays
1977–1979	Aquitaine/ Amoco	Central Ridge, Red Ridge, Nasiti Ridge, Vatume Hill	Soils (804), Rocks (66)	-	-
1979–1980	Aquitaine/Cluff	Central Ridge, Red Ridge	Rocks (281)	-	1 DDH
1985–1986	Venture Exploration	Kingston Mine, Central Ridge, Nasiti Ridge, Golden Ridge	Soils, Rocks (152)	-	-
1986–1993	Pan Continental/ Venture	Kingston Mine, Central Ridge, Blasting Rock, Golden Ridge, Red Ridge, Vunatawa, Ngalyalo, Vatume Hill, Banana Creek	Soils (+106), Rocks (+857), Streams	Airborne: Magnetics	14 RC (1902 m)
1994–1998	CRAE	Nasiti Ridge, Nasala, Banana Creek, Tuvatu North	Rocks (90)	Ground: Magnetics, IP	3 DDH (623 m); 1 RC (128 m)
1999–2002	Mincor	Banana Creek	Soils (215), Rocks (6)	-	5 DDH (520 m)
2002–2004	Alcaston/ Mincor	Banana Creek, Central Ridge, Tuvatu North	Rocks (181)	-	3 DDH (595 m)
2007–2008	Golden Rim/ Mincor	Central Ridge, Tuvatu North	Rocks (+368), Soil (+858), Streams (+132)	Airborne: Magnetics, IP	8 DDH (1670.5 m)
2007–2008	Golden Rim/ Mincor	No field work done. Reassessment of all exploration data.	-	-	-

6.2.3.1 Previous Exploration Details and Results

1906–1923

- Kingston mine shaft sunk to 14.6 m. Handpicked mineral materials from the shaft graded up to 176.27 g/t Au, 130.05 g/t Ag, and 40.6% Cu, and a mineral material parcel of 6 tons graded 84 g/t Au, 130 g/t Ag, and 33% Cu. The shaft was later flooded and filled with sediment from the nearby Sabeto River.
- Kingston mine adit driven 9.1 m. 1.4 tonnes of mineral materials were removed in 1915 containing 18.36 g/t Au and 7% Cu.
- Short adits dug into malachite-bearing rocks at Central Ridge, Blasting Rock, Nasiti Ridge, Qalyalo, Vunatawa, and benches cut at Golden Ridge.

1943–1947

- In 1946, the South Pacific Mining Company drilled three holes.
- The South Pacific Mining Company conducted a program of sampling of all known workings and analysed for gold. No follow-up work was done.

1963–1964

- In 1963, a stream sediment and residual soil (ridge and spur) sampling program was conducted by Higgs & Coulson, which defined four large regional copper anomalies. Samples were collected from 30 to 45 cm depths corresponding to the upper C soil horizon.
- The Geological Survey of Fiji conducted a program of geological mapping, stream sediment, soil sampling, and drilling to evaluate the copper potential of the Kingston mine area.
- The Geological Survey drilled nine DDHs (11 to 109 m depth) totalling 393 m, into regional geochemical anomalies and around the Kingston mine. The survey concluded from their results that no further work was warranted.

1968–1969

- Amad NL in a joint venture with Ah Koy Mining Syndicate completed stream sediment (36 samples), rock chip (16 samples), and ridge and spur soil sampling (378 samples) covering much of the present SPL 1412.
- They defined a 2.4 km x 450 m copper, gold, and silver anomaly centered around the Kingston mine workings. Work was then concentrated on a gridded area covering the Kingston mine, Central Ridge, and Nasiti Ridge.

1970–1976

- In 1970, Barringer Fiji Ltd farmed into the Amad license (PL1004) and commenced programs included soil sampling, ground IP, and ground magnetics. This was initiated by an airborne magnetic survey of Viti Levu flown at this time, which delineated the Navilawa monzonite stock at the intersection of a 2400 fault and displaced by a 150 fault and an adjacent magnetic high to the north.
- The soil grid was extended and filled in to 400 ft. x 200 ft. Several soil anomalies of >200 ppm Cu were defined.

- Five widely spaced DDHs (90 to 208 m depth) were drilled in the Central Ridge – Nasiti area, totalling 731 m. Three holes were sited on geochemical anomalies and two holes on IP anomalies. The best intercept was 85.4 m @ 0.12% Cu (Barringer Hole DDH No. 2) with peak values of 0.24% Cu drilled on Central Ridge. No gold assays were conducted. Work during this period concentrated on exploring for porphyry copper and little or no attention was paid to the gold potential. The geology of DDH 4 was summarized as "biotite andesite porphyry host, brecciated and shattered with numerous carbonate-quartz stringers with magnetite, pyrite and chalcopyrite". No assays are available.

1977–1979

- Aquitaine Fiji in a joint venture with Amoco conducted an extensive gridding and soil sampling program (samples on 50 m intervals on lines 200 m apart over a 2.8 km x 3.6 km area). An east–west oriented zone of copper anomalism was identified. Some broad gold anomalies were identified where values of 0.1 g/t Au were reported. Only every second soil sample was analysed for gold (i.e., 100 m x 200 m grid) using an atomic absorption spectrometry (AAS) with a 0.1 g/t detection limit. After this program, Amoco pulled out of Fiji.

1979–1980

- Aquitaine in a joint venture with Cluff Minerals investigated the alteration zones of Vatume Hill and Red Ridge for their gold potential. Track cuttings were mapped and channel sampled and one diamond hole (AC1) was drilled into Red Ridge. Although intense pyrite alteration and quartz veining was intersected in AC1, no significant gold values were reported. Aquitaine relinquished the ground in 1980.

1984–1986

- In 1985, Venture Exploration in association with geologist K. Glasson (SPL 1218) conducted a program of tape and compass surveying of all known workings and drill hole collars. All workings were resampled and analysed for a suite of elements. Glasson also conducted auger soil sampling and assayed for gold. This work established the gold-copper association in the workings at Nasiti, Central Ridge, Kingston, and Blasting Rock and that these workings line up in an east–west direction parallel to the intrusive contact. The workings at Golden Ridge, Red Ridge, Vunatawa, Vatume Hill, and Ngalyalo, however, contained gold with no anomalous copper.

- Previous geophysics was reviewed by P. Gunn in 1985. Gunn suggested that the Barringer IP concentric chargeable zones could be a response from a pyrite shell around the copper-gold mineralization in a porphyry type setting. The IP anomaly is 600 m southwest of the Kingston in a topographical low. The nearest drill holes are approximately 300 m to the north of the anomaly boundary, GD8 averaged 620 ppm Cu over 123 m, and in GD9, drilled to 109.9 m depth, copper assays ranged from 200 ppm to 0.55%.

1986–1993

- In 1986, Continental Resources (Fiji) Ltd, a subsidiary of Pan Continental Mining, took over management of SPL 1218 in joint venture with Venture Exploration. Pan Continental Mining undertook detailed mapping, extensive rock chip sampling, and ridge and spur soil sampling of the Kingston mine area, Central Ridge, Golden Ridge, Vatume Hill, and bulldozer cuttings at Banana Creek.

- Four RC holes were drilled to evaluate the quartz-alunite related alteration to the east of the main Kingston area at Vatume Hill, Vunatawa, and Golden Ridge. Ten RC holes (maximum 150 m depth) were also drilled into the quartz-sericite altered monzonites at Central Ridge. The best intercept was 60 m @ 0.7g/t Au to the southeast of the old Kingston mine. These mainly tested Venture Exploration geochemical anomalies.

- Regional mapping and sampling of all the main drainages in SPL 1218 delineated Au and As anomalies at Upper Nanganga Creek and Southern Ridge. Gold grades of up to 70 g/t were returned from surface samples at the apparent epithermal vein system at Banana Creek, approximately 4 km northeast of Kingston.

- A 300 line km airborne magnetic and radiometric survey (Austirex, 200 m spacing with a flying height of 60 m) was flown over the Kingston area in 1987.
- By the end of 1987, Pan Continental Mining concluded that the host rocks for mineralization at Central Ridge were late stage andesite porphyry dykes, the epithermal alunite+quartz+diaspore alteration at Vatume Hill is controlled by north-northeast-trending linears, the gold in soil samples at Vatume Hill are located lower than the argillic alteration and may represent a boiling zone, and the contact zone between monzonite and a 500 m wide monzonite porphyry body had been a locus for intense alteration, dyking, and Cu-Au mineralization.
- In 1990, a mapping/petrology honours thesis titled “The Geology of the Navilawa Area” was undertaken by M.B. Mills from the Australian National University.

1994–1998

- CRAE was granted SPL 1369 in 1994 and held it until surrendering the license in 1998.
- An IP survey defined six chargeability anomalies that were considered prospective for disseminated porphyry style Cu-Au and possibly high grade structurally controlled epithermal Au (Cu) mineralization. Four of these anomalies coincided with extensive zones of intermediate argillic and sericitic alteration.
- One RC and three diamond holes (totalling 751 m) were drilled over four chargeability anomalies. It appears that the source of the IP chargeability anomaly at the Southern (RC95KN1), Nasala (DD95KN2), and the Nasiti (DD95KN3) anomalies is 1 to 3% disseminated and veinlet pyrite related to the moderate propylitic or sericite-pyrite alteration of the monzonite and shoshonite. CRAE concluded that little scope exists for defining significant mineralization at reasonable depths.
- Hole DD95KN4 drilled over the Waloru chargeability anomaly intersected a weakly mineralized breccia hosted by strongly potassic altered monzonite with minor chalcopyrite and bornite. No mineralized material grade copper or gold intersections were obtained.
- Sampling at Banana Creek suggested that some of the high-grade gold values associated with quartz veins and shears may be attributable to localized secondary enrichment.

1999–2001

- SPL 1412 was granted to Oribi Resources in 1999.
- Reconnaissance mapping and rock chip sampling in the creeks between Tuvatu and Banana Creek were undertaken to see if there was a link between the two mineralized zones. Only very minor veins or shears were found and high grade alteration was lacking. Also many of the structural features had a more northerly strike than anticipated.
- A tape and compass grid was established over Banana Creek. This was oriented to the potassium radiometric anomaly at 48 degrees true north. The base line was 2.2 km long, with 200 m spaced lines. Pegs were spaced every 50 m along lines.
- Soil sampling (“C” horizon) was conducted at Banana Creek with a 50 mm diameter auger. A strong gold anomaly (>0.5 g/t Au) was outlined over an area of 1 km x 0.3 km. The gold-in-soil anomaly coincides with the strong potassium radiometric anomaly possibly indicating hydrothermal alteration.
- Geological mapping (1:2000 scale) and rock chip sampling (six samples) were conducted at Banana Creek. The best rock chip result was 6.05 g/t Au.

- Five DDHs (totalling 519.9 m) were completed at Banana Creek. Holes were started in PQ until competent rock was reached and the rest of the hole was drilled with HQ-sized core. The best intercept was 0.4 m @ 23.4 g/t Au (Hole BCDH4) from an epithermal quartz vein.

2002–2008

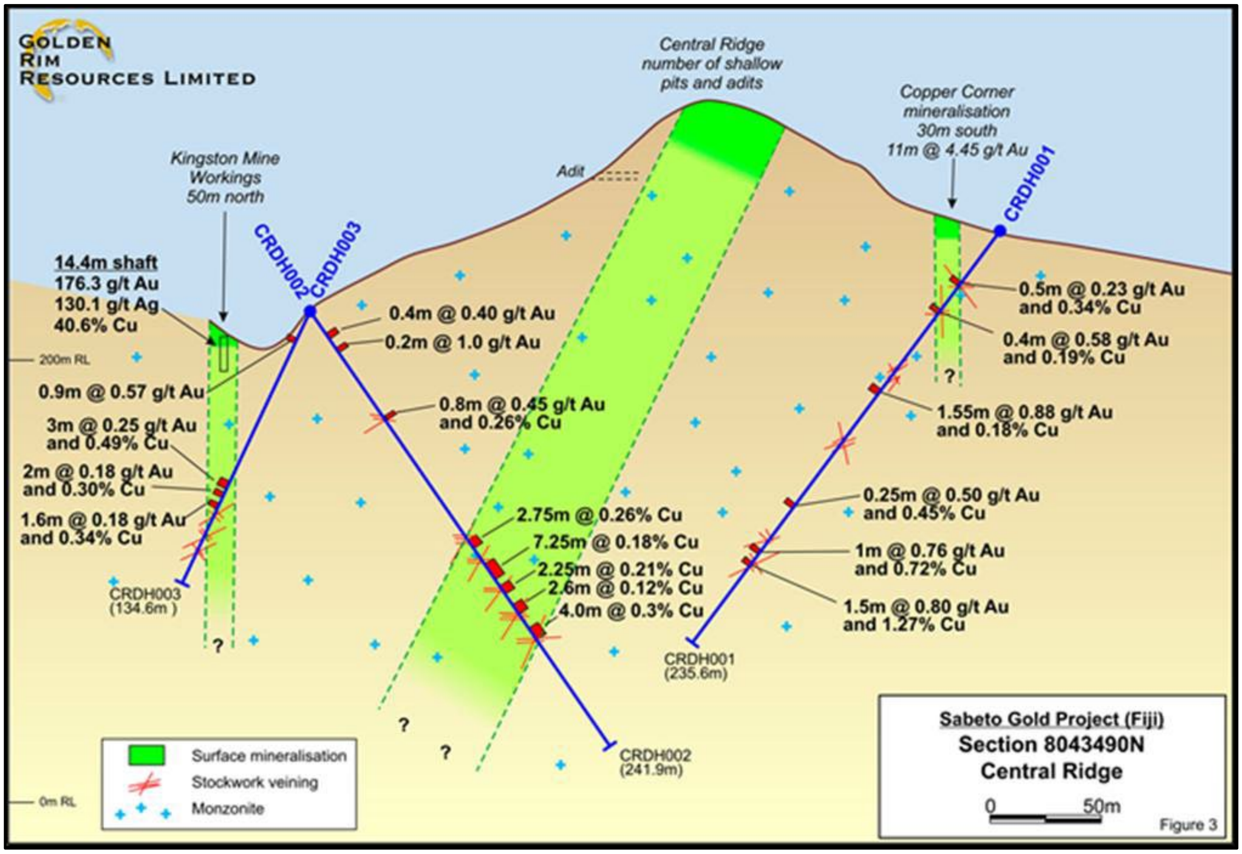
- A helicopter-supported magnetic and radiometric survey was flown over the entire tenement area.
- Central Ridge Prospect drilling

  Previous work in the area has located rock chips anomalous in gold and copper. Radiometric survey also defined anomalies in this area. Consequently, four diamond holes were drilled by Golden Rim.

  Mineralization intersected typically consisted of carbonate/pyrite veining with patchy silicification hosted in biotite monzonite. Much of the veining and silicification was associated with highly fractured zones within the main monzonite body. The most significant copper and gold mineralization was intersected in CRDH 001, which was collared near historical workings and drilled towards the old Kingston Mine. Significant intersections for holes CRDH 001 to 003 are presented in Figure 6-1. The fourth hole (CRDH 004) was drilled to test possible structural intersections to the south of historical workings.

- Tuvatu North Prospect Drilling

  Four diamond holes were completed for a total of 948.4 meters. This prospect area lies immediately north of the Tuvatu exploration adit. Major potassium radiometric anomalies were identified at this location from the survey conducted by GRM, and a number of historic IP anomalies were identified but not tested by CRAE in the mid-1980s. Mapping of tracks and creeks in the area have identified abundant silica/limonite veining, but gold tenors were low.



Source: Gold Rim (2008)

Figure 6-1: Drill hole cross-section showing significant intersection for holes drilled at Central Ridge

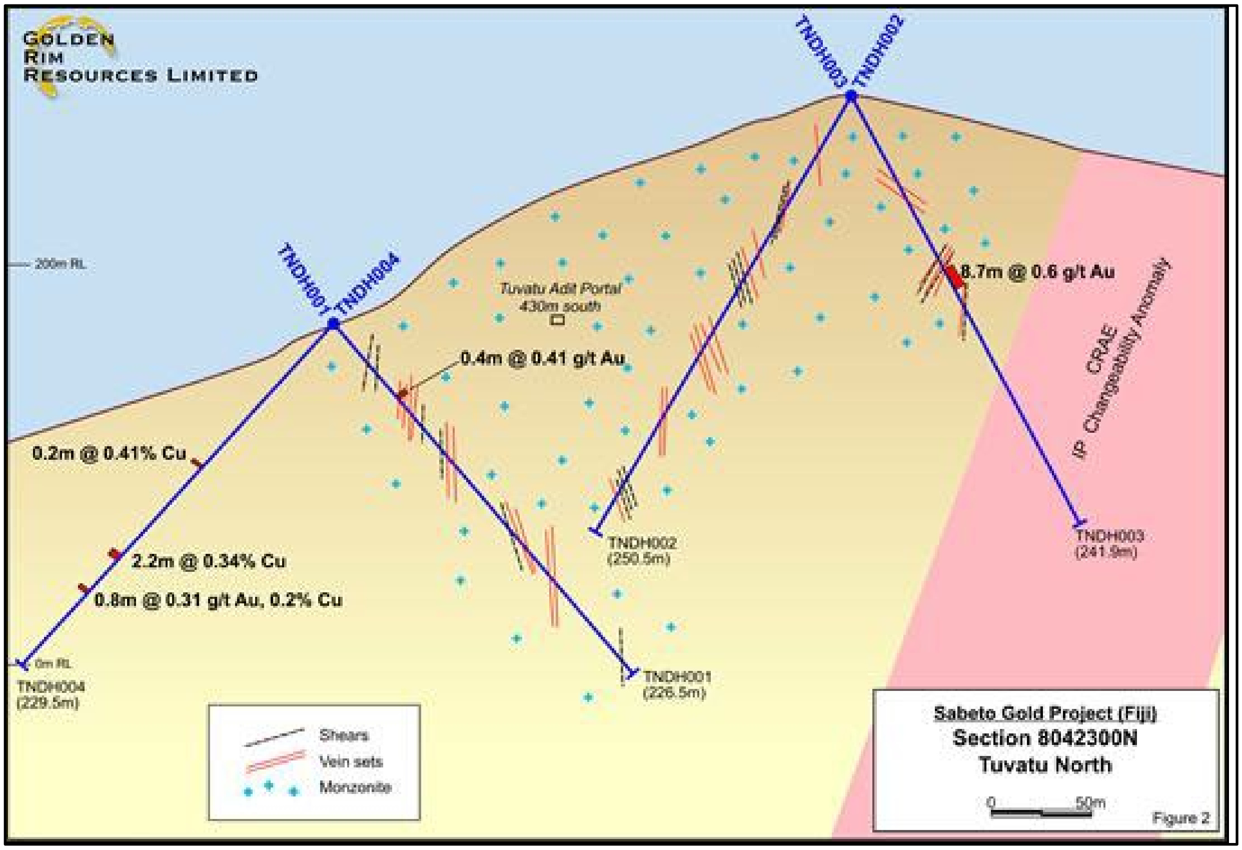
TNDH 001 intersected numerous zones of bleaching associated with silica/pyrite veins. The veins typically consist of fine-grained, dark grey pyritic silica, which may exhibit rhythmic banding parallel to the vein margins. Individual veins vary from thin (<1 cm) structures to over 30 cm in thickness. Zones of intense veining may reach over 8 m in thickness. Assay results from this hole were very disappointing considering the style and intensity of veining intersected. The best result obtained was 0.4 m at 0.41 g/t Au from 40.4 m.

Hole TNDH 002 intersected similar but less intense mineralization than hole TNDH 001.

TNDH 003 was drilled to test a strong IP chargeability anomaly. Numerous pyritic shears were intersected together with carbonate/pyrite veins and stockworks. A broad zone of anomalous gold, arsenic, and sulphur was intersected from 98.45 to 117.85 m. The best intercept within this zone was 8.7 m at 0.6 g/t Au from 106.5 m. The historic IP chargeability anomaly is believed to be caused by the abundance of pyrite intersected in the anomalous zone.

TNDH 004 was collared from the same location as TNDH 001 but drilled to the west, under the Sabeto River. This hole failed to intersect significant gold mineralization, with the best intercept of 0.8 m at 0.31 g/t Au from 169.5 m in a pyritic carbonate veined zone. A broad zone of quartz/pyrite +/- chalcopyrite veining was intersected between 138.65 to 171.6 m. Intersections within this zone included 0.9 m at 0.2% Cu from 143.85 m, 2.2 m at 0.34% Cu from 149.3 m, 0.7 m at 0.25% Cu from 164.7 m, and 1 m at 0.3% Cu from 168.5 m.

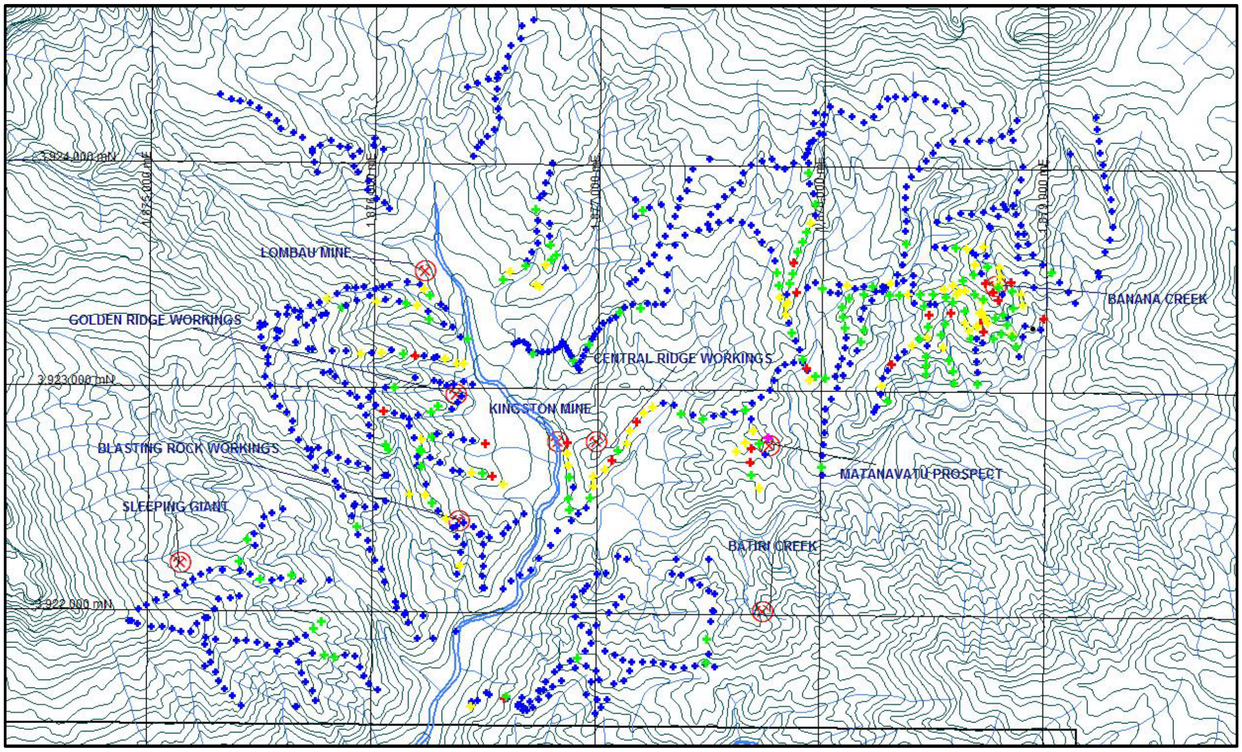
Significant intersections for holes TNDH 001 to 004 are presented in Figure 6-2.



Source: Gold Rim (2008)

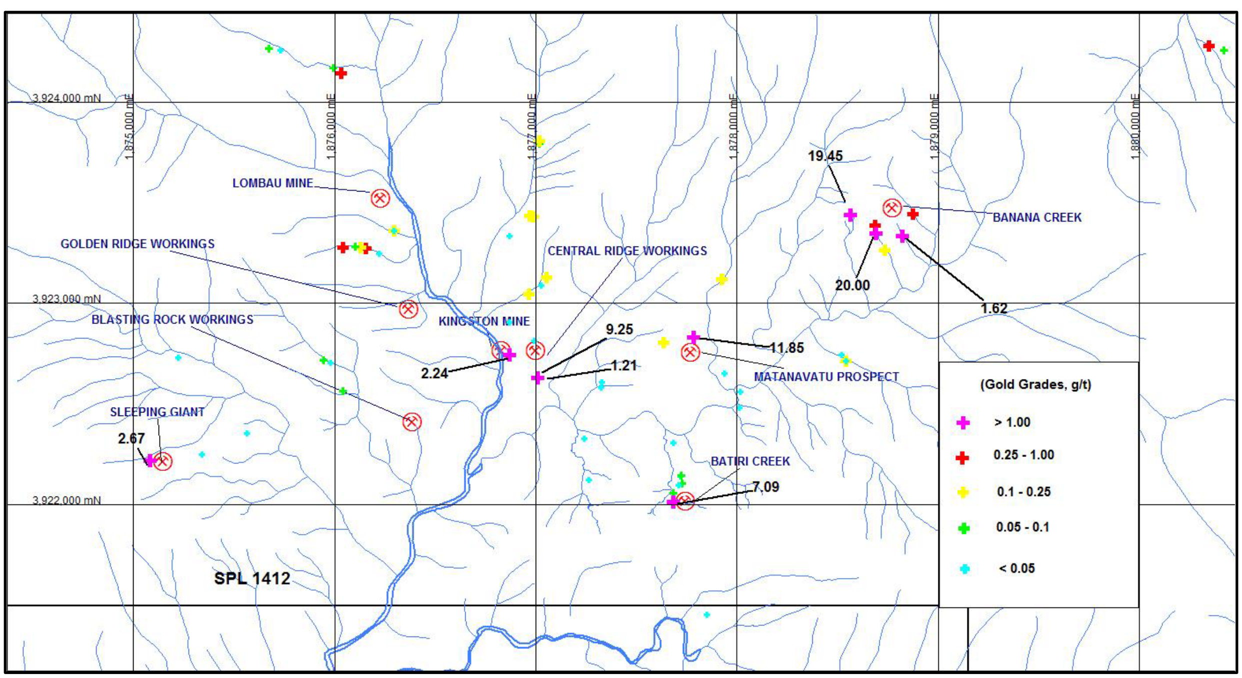
Figure 6-2: Drill hole cross-section showing significant intersection for holes drilled at the Tuvatu North Prospect

- Additional geological mapping and geochemical sampling was undertaken. Figure 6-3, Figure 6-4, and Figure 6-5 show the locations of geochemical sampling done with gold values.



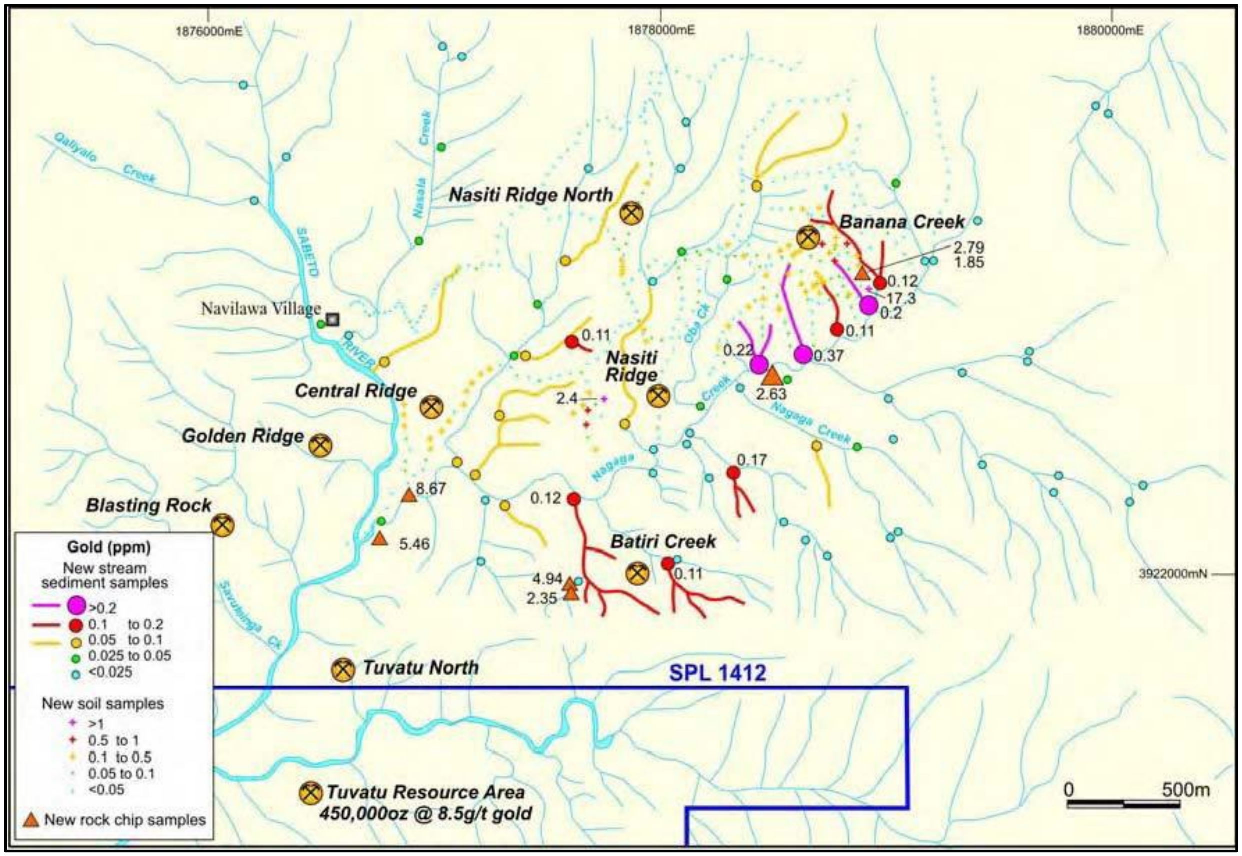
Source: Golden Rim (2008)

Figure 6-3: Soil samples coloured by gold content. Magenta >1 g/t Au, Red 0.25–1.0 g/t Au, Yellow 0.1–0.25 g/t Au, Green 0.05–0.1 g/t Au, Blue <0.05 g/t Au.



Source: Golden Rim (2008)

Figure 6-4: Rock chip samples with significant results



Source: Golden Rim (2008)

Figure 6-5: Summary of geochemical sampling with significant results highlighting some prospect locations

The soil sample results outlined two new gold prospect areas at Nasiti Ridge and Nasiti Ridge North. Ground checking of the Nasiti Ridge soil anomaly (up to 2.42 g/t Au) located oxide copper mineralization associated with a late-stage monzonite dyke. This area has been called the Matanavatu Prospect. Numerous thin silica/limonite veins occur with the dyke, and copper occurs as malachite stains on fractures and joint surfaces. Significant results obtained from limonite/malachite fractures at this prospect area include 9.54 g/t Au, 4.7 g/t Ag with 3.1% Cu, and 1.64 g/t Au with 1.19% Cu.

At Nasiti Ridge North, which is located approximately 800 m west of Banana Creek, a line of anomalous soil samples extends for approximately 500 m, with a maximum value of 0.29 g/t Au. Assay results from aplitic veins located at the Nasiti Ridge North soil anomaly returned low gold values (best result 0.11 g/t Au with 1.2 g/t Ag). This result suggests that the low level of gold anomalism obtained from soil samples is probably due to mineralization associated with siliceous aplitic dykes.

Samples of limonitic aplite veins located between the Matanavatu and Central Ridge Prospects returned values of 1.14 g/t Au and 1.6 g/t Ag. Another sample of similar material returned a lower value of 0.19 g/t Au. This further confirms that the aplite dykes in the area carry low-level gold mineralization. Additional sampling at the Matanavatu Prospects returned a result of 11.85 g/t Au, 7.0 g/t Ag, and 0.37% Cu. Mapping indicates that the main structural trend at this prospect is west-southwest, and it is probable that this trend continues at least 750 meters west to

Copper Corner Prospect at Central Ridge. A float sample of quartz vein material was located at Central Ridge, which returned a grade of 9.25 g/t Au and 14.9 g/t Ag. The true location of this sample is uncertain.

Ground checking of the 17.3 g/t Au composite soil sample from Banana Creek Prospect located a 30-cm-wide quartz vein. Rock chip sampling of this structure returned an average grade of 1.87 g/t Au. The gold anomaly over the Banana Creek Prospect is approximately 630 m in diameter. Additional sampling at the Banana Creek Prospect returned significant results of 20.0 g/t Au and 11.6 g/t Ag from a 0.5 m wide quartz vein, and 19.45 g/t Au and 3.8 g/t Ag from a structure previously considered to be low grade. The 20 g/t Au sample may be associated with a sinistral offset of the Iliesa vein; however, additional detailed mapping is required to confirm this. The 19.45 g/t Au sample was obtained from a narrow quartz vein, which previously had only returned maximum gold grades of 0.32 g/t Au. Gold tenors of these samples are encouraging as they are significantly higher than previously obtained surface samples in the Banana Creek area.

Sevekoro Creek, immediately north of the historical Golden Ridge workings, was identified as being anomalous from stream sediment results, which returned a result of 0.17 g/t Au. Mapping in this area located a broad zone of fracturing associated with major north-northeast-trending fault structures hosted in biotite monzonite. The fault zones occur as puggy pyritic cataclasites or as zones of very high jointing intensity. Clay/pyrite +/- quartz occurs along the joint planes, and the host rocks are generally weakly pyritic. A single sample from a shear zone in this area returned a value of 9.16 g/t Au. Other shears sampled in the area returned anomalous but lower gold values, typically ranging between 0.11 to 0.35 g/t Au.

Irregular zones of gold anomalism were located around the Golden Ridge Prospect. These results confirm the highly erratic nature of gold-bearing structures in this area.

A low-level anomaly was located at Goat Hill, immediately north of Navilawa village. Numerous limonitic shears and fracture zones coincide with elevated gold-in-soil values and are considered to be the source of the anomalous soil geochemistry.

Reconnaissance mapping at the Sleeping Giant Prospect in the western part of the tenement along the foothills of the Mt. Evans Range, 800 m west of the inferred contact of the Navilawa Monzonite, located a number of quartz vein stockwork zones associated with strong north–south-trending structures. The structures are probably hosted in rocks assigned to the Koroimavua Volcanic Group. The most significant result from this area returned 2.67 g/t Au and 1.9 g/t Ag.

Very low levels of gold anomalism were detected southeast of the Sleeping Giant Prospect and southwest of Batiri Creek

Mapping at Batiri Creek located a major series of northeast-trending faults and fracture zones with one sample returning 7.09 g/t Au, 4.4 g/t Ag, 0.32% Pb, and 0.8% Zn. More work is required in this area to attempt to locate additional vein structures.

A 1 m channel sample taken across a north-northwest-trending structure located east of the Kingston Mine portal returned a value of 2.24 g/t Au. The location of this surface sample suggests it was the same structure intersected in DDH CRDH002, which hit a fractured zone at 58 m depth, which returned 0.8 m at 0.45 g/t Au and 0.26% Cu. No significant copper values were recorded in the surface sample.

6.3 Historic Resource and Reserve Estimates

A number of historical Mineral Resource estimates were carried out for Tuvatu by previous operators over the period from 1997 to 2000. A QP has not done sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves. 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 Mineral Resource statement presented in Section 14.0 in this document supersedes all previous Mineral Resource figures.

6.3.1 TGM 1997 Upper Ridges Resource

A Mineral Resource figure was calculated internally by TGM in September 1997 for the Upper Ridges area as 904,000 t at 5.1 g/t Au (149,272 oz). 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 Mineral 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 Mineral Resource figure for the Upper Ridges lodes of 602,000 t at 8.2 g/t Au was calculated for a total of 159,362 oz (Table 6-2).

Table 6-2: Summary of Upper Ridges resources

Lode	True Width	Tonnes (t)	Grade (g/t)	Ounces (oz)
Upper Ridges West 1	1.00	16,462	24.74	13,093
Upper 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**

Source: TGM (1998)

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 to 60% reflecting the lower density of drilling in the southern part of Tuvatu. Additional drilling was considered to be required to infill this Mineral 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 estimate for the Tuvatu Lode and Nasivi-SKL stockwork area. The estimate used a 3D "service variable" block modelling technique using 2.0 m composites and a 1.0 g/t Au cut-off. A revised Mineral Resource figure for the Murau flatmake was calculated using the September 1997 Geoval block model after the area included in the February 1998 Geoval recalculation of the Nasivi-SKL stockwork was excluded. Global Mineral Resource figures for the various lodes are summarized in Table 6-3.

Table 6-3: Summarized Tuvatu resource (February 1998)

Lode	Lower Cut-Off	Tonnes (t)	Grade (g/t)	Ounces (oz)
Upper Ridges	2.00 m-gram	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 program by TGM in July 1999, an additional Mineral Resource estimation was completed for the Upper Ridges area based upon data gained from underground development and surface and underground drilling (Table 6-4). 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 estimates, 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 6-4: Upper Ridges resource (July 1999)

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

Two areas of Indicated Mineral Resource were classified within the UR1 South strike drive area and the UR2 North strike drive area based on geological and channel sampling data on 2 m centers in the underground development. 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 Mineral Resource figures estimated 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 channel sample grades of each of these areas was higher than indicated by drilling.

Upon completion of the Phase 3 drilling program, the geological model was updated, and a new polygonal Mineral Resource estimate was completed by TGM using the same parameters as for the July 1999 estimate. Figures previously estimated by Geoval for the Nasivi-SKL area were superseded by Mineral Resource figures for the GRF steep shear and the Murau Lodes. Updated resource figures were not estimated for the SKL flatmakes. Preliminary figures were also estimated 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 Mineral 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 Mineral 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 Mineral Resource estimate had been excluded from the model, and VA was commissioned to revise the resource estimate.

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 that fully enclose a volume and is referred to as a "solid".

Lodes were modelled as mutually exclusive wire-frames, one for each lode (Table 6-5). The lode widths were taken from the wire-frames. All drill holes intersecting the structure were used, whether mineralized or not.

Table 6-5: Lode wire frame – drill hole details

Wireframe	Number of Intercepts	Volume (m^3)
UR1	69	117,710.3
UR2	133	300,234.5
UR4	45	79,406.8
UR5	48	127,694.4
UR6	36	102,722.7
UR7	15	97,685.6
UR8	11	104,455.7
URW1	93	220,146.2
URW2	72	151,057.8
URW3	88	174,019.1
GRF1	24	38,271.1

Source: Vigar (2000)

A total of 41 lodes were identified of which 37 had sufficient intercepts to be modelled in the Mineral Resource estimate. True widths were used, and a mining width of 1.2 m was 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 lode 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. OK estimation was used for the grade of each block. Data used for the estimate were drill lode composites where an upper cut of 75.0 gold m-grams was applied on the raw drill data prior to lode compositing.

Lode blocks were filled with grades using the estimation of a grade*width accumulation (m-grams) using OK 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 and width multiplied by grade and the grade back-calculated.

The extent of the search ellipse used in the OK 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.

Mineral 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 1.2 m were applied.

Table 6-6 summarizes the total resources as reported by VA in August 2000 using a 3.0 g/t Au cut-off. The Mineral Resource was stated as being JORC compliant at the time. The Mineral 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 6-6: Historical resource figures

Lode	Indicated Resource			Inferred Resource		
	Tonnes (t)	Gold Grade (g/t)	Gold Ounces (oz)	Tonnes (t)	Gold Grade (g/t)	Gold Ounces (oz)
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**

Source: Vigar (2000)

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 Mineral Resource statement presented in Section 14.0 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 were carried out. A total of 588 samples were found to exceed 1.0 g/t Au and 214 samples were found to exceed 10.0 g/t 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 program was undertaken. In conjunction with the underground development, 17 underground DDHs (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 Mineral 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 DDHs (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 DDHs (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 program 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.

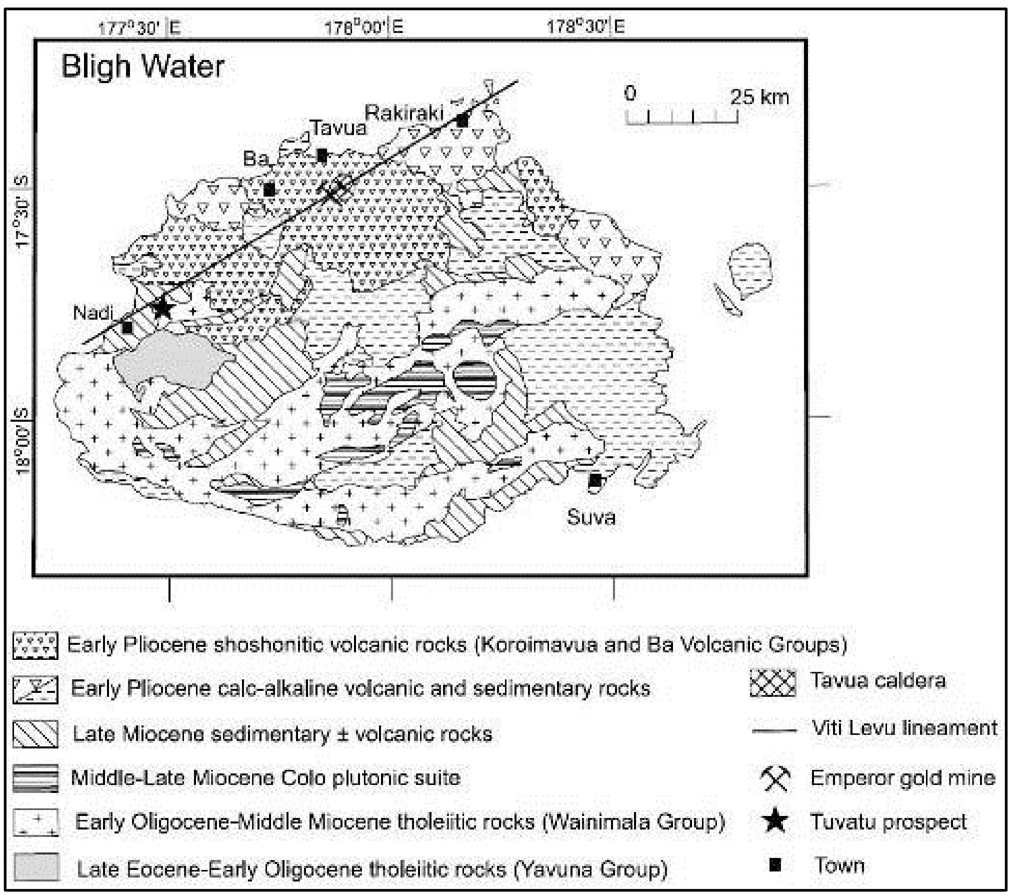
7.0 GEOLOGICAL SETTING AND MINERALIZATION

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 gold systems along the greater than 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 seven million ounces since 1937.



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

Figure 7-1: Regional geology. Location of Tuvatu Project with respect to Viti Levu lineament is indicated.

The oldest unit in the region is the Nadele Breccia (Late Oligocene-Middle Miocene, 26-12 Ma). Thin layers of sandstone and siltstone are interbedded with grits and often exhibit cross-bedding. 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 northeast trending band across the northwestern side of Viti Levu and host a number of gold mines and prospects, including 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 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 emphasize 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 at Tuvatu. 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 elongated in a northeast 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 Project area appears to have had 1 to 2 km 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.

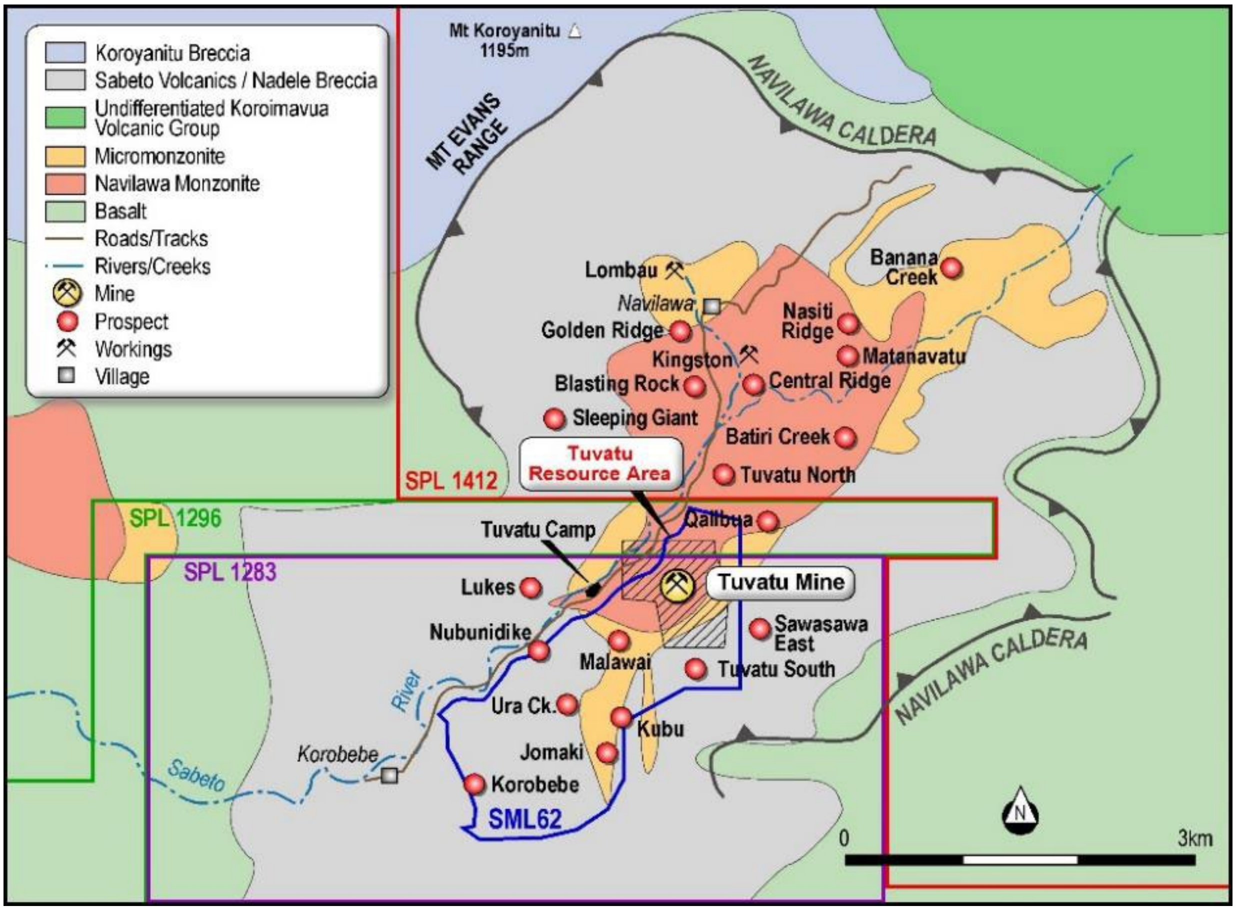
7.2 Local Geology

Tuvatu is one of several alkaline hosted gold prospects known from the Sabeto area of northwestern 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 centers 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.

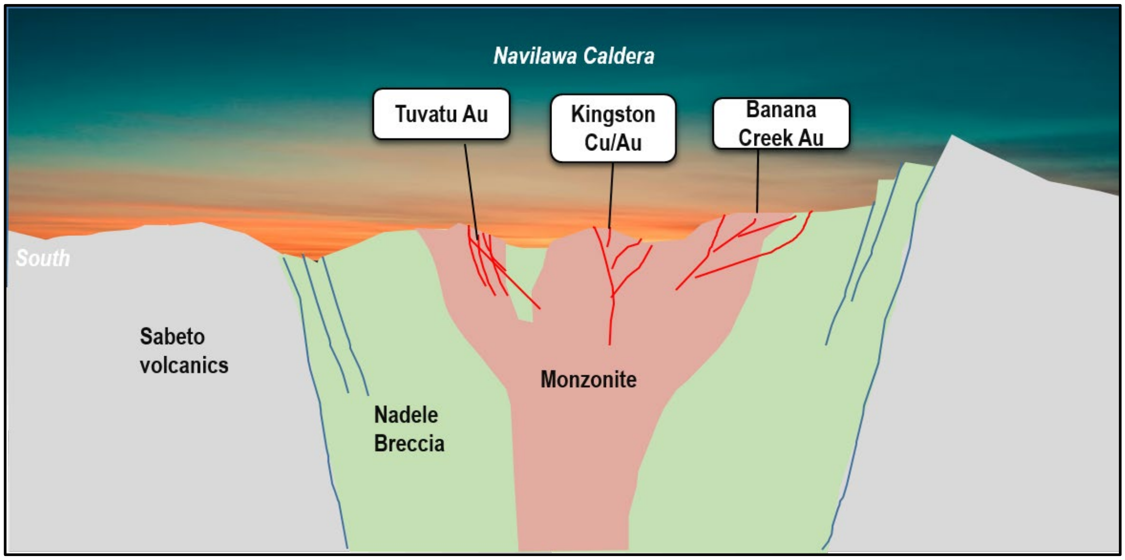
The volcaniclastic units were subsequently intruded by a monzonitic stock (Figure 7-2 to Figure 7-4). Earlier mapping by Emperor Gold Mining Company Limited geologists and latter mapping by Lion One indicated that it is a composite intrusive body with several different phases of intrusion associated 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 represents the root of a Caldera and is elongated 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.



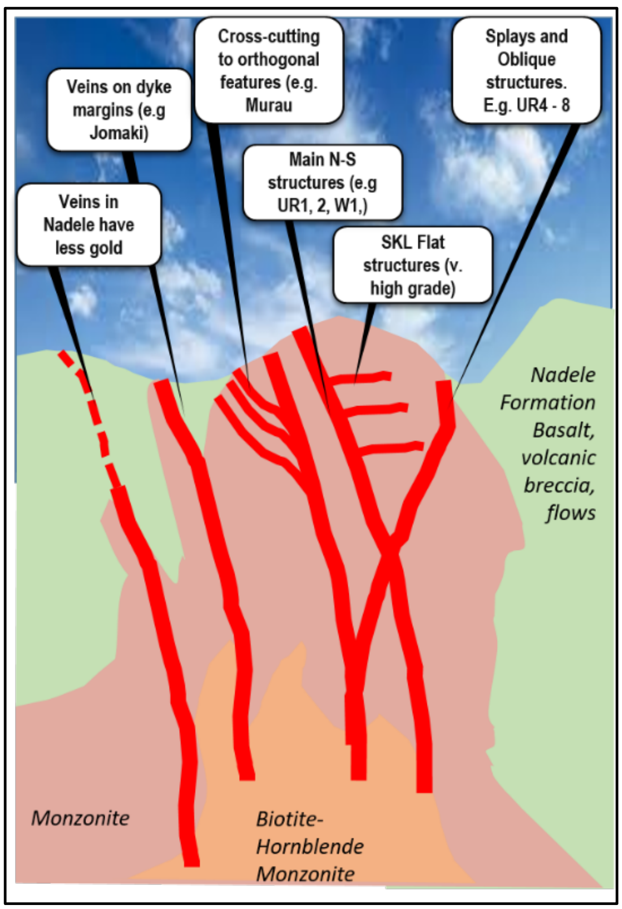
Source: Lion One Limited Internal File (2017)

Figure 7-2: Project geology



Source: Lion One Internal File (2019)

Figure 7-3: Schematic long section through Navilawa Caldera



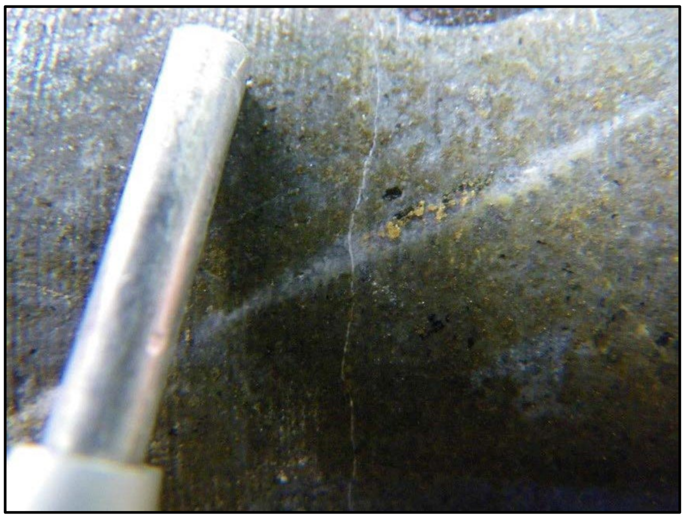
Source: Lion One Internal File (2019)

Figure 7-4: Schematic cross-section through the Tuvatu Vein system

7.3 Mineralization

Tuvatu is an alkaline hosted gold deposit. Mineralization is structurally controlled and occurs as sets and networks of narrow veins and cracks, with individual veins generally ranging from 1 to 200 mm wide (Figure 7-5 and Figure 7-6). 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 500 m 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 can contain significant gold grades. Lode mineralogy is varied, with most veins containing quartz, pyrite, and base metal sulphides.



Source: Taylor (2013)

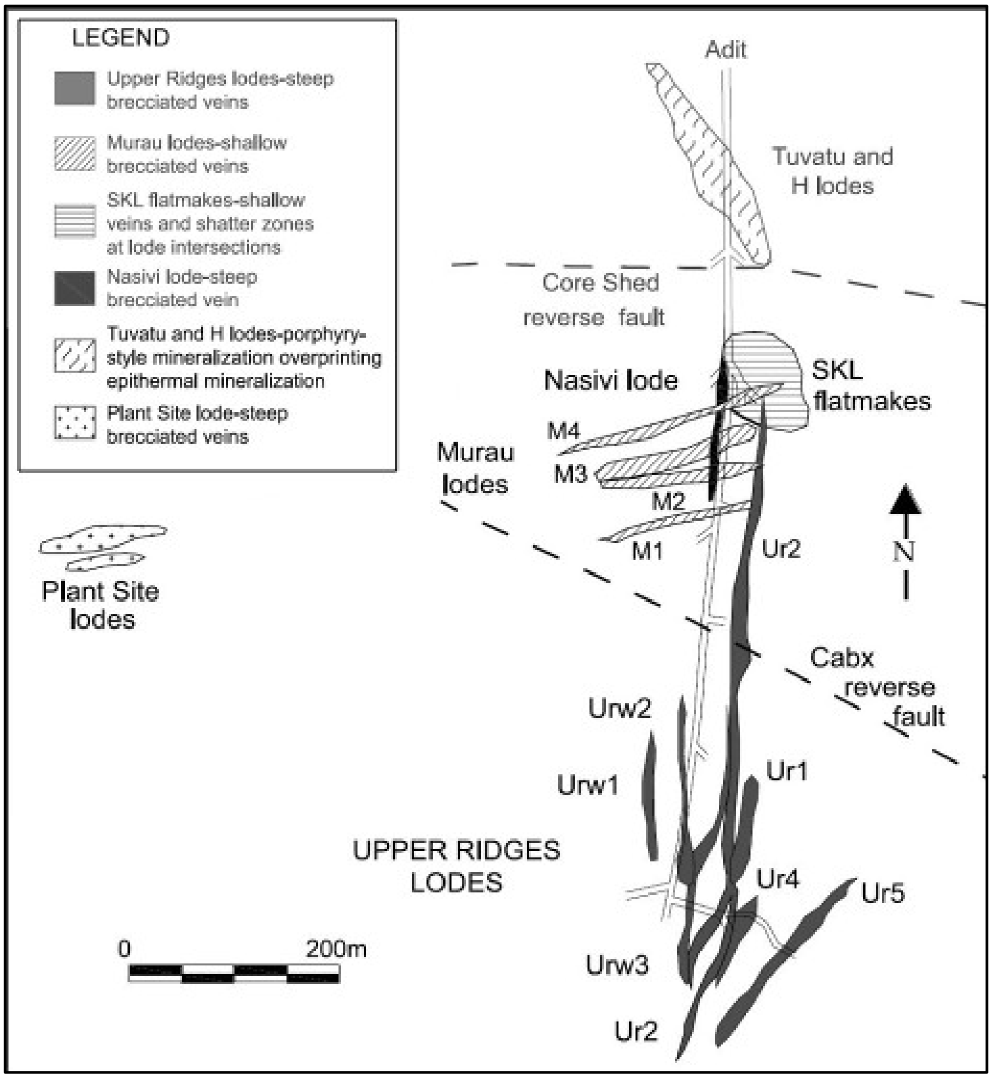
Figure 7-5: Visible gold in W3 Lode, TUDDH371 at 204.3 m

Source: Lion One (2019)

Figure 7-6: Bench exposure of UR5 Lode

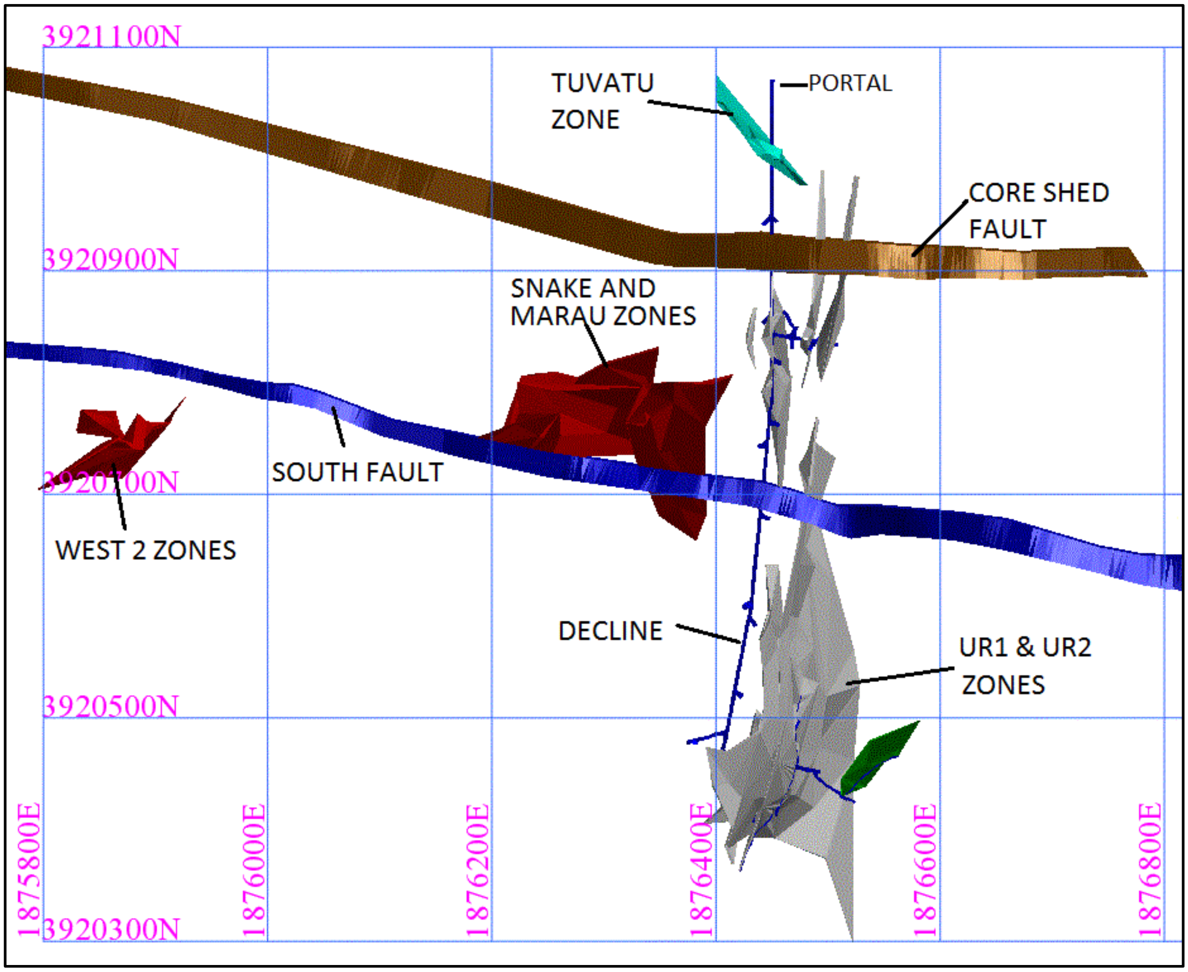
A very high proportion of the gold occurs as either free gold or is contained in quartz or pyrite composite particles that can be floated. Free gold present is both fine and coarse grained. Mineralization is clean with respect to deleterious elements such as arsenic, selenium, and uranium.

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 7-7 and comprise ten lodes in the Upper Ridges area, two lodes in the Murau area, three lodes in the West (Plant Site) area, two lodes in the Tuvatu area, and three lodes in the SKL area. Lodes were reinterpreted by Lion One's geologists following infill and resource extension drilling (Figure 7-8).



Source: Scherbarth and Spry (2006)

Figure 7-7: Plan view of Tuvatu Lode structures (historic interpretation)



Source: Lion One (2014)

Figure 7-8: 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 to 10.0 g/t. Gold mineralization tends to be quite coarse, and visible gold can be observed in mineralized sections of core (Figure 7-7).

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 100 m. The majority of lodes vary in width from 0.5 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 intrusive related gold mineralization with potassic alteration.

7.3.3 Paragenesis

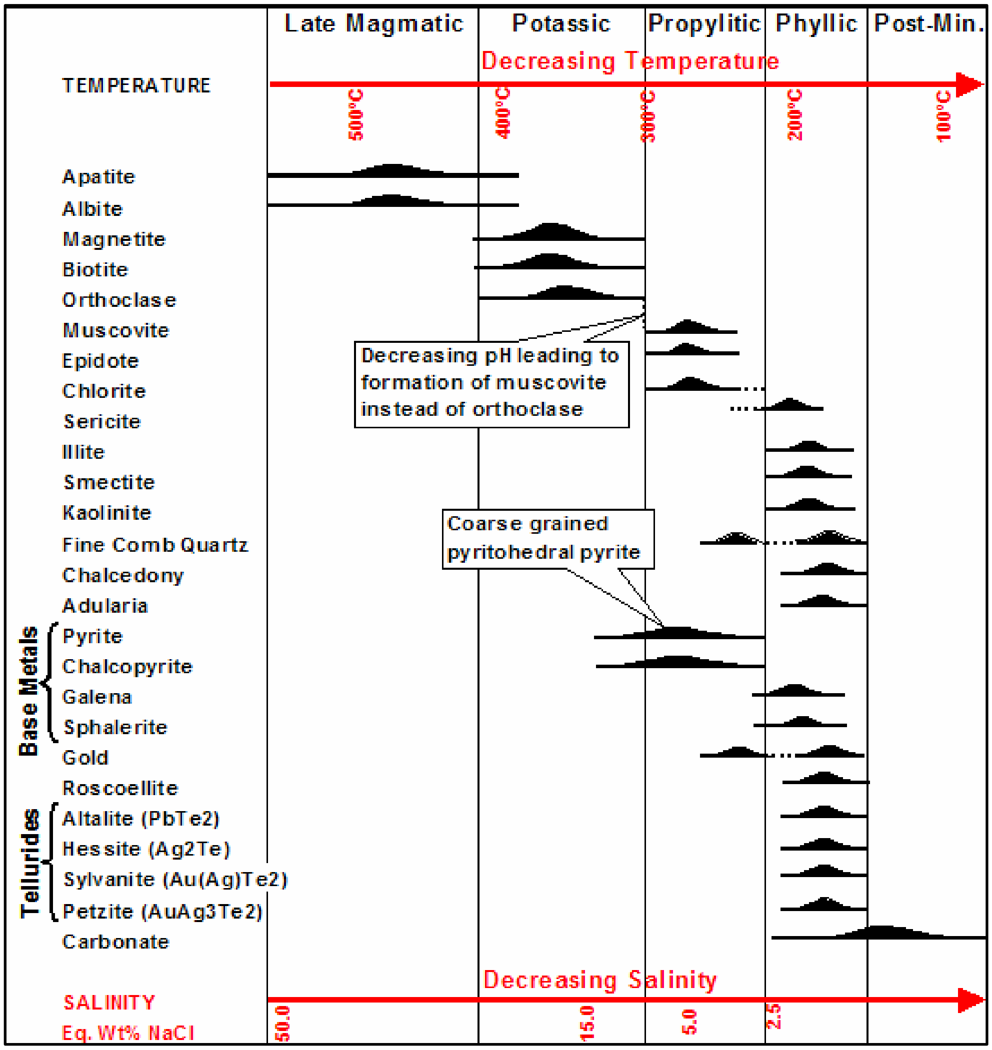
Scherbarth and Spry (2006) suggest that the mineralized zone at Tuvatu may have originally developed as a porphyry system, which was overprinted by later intermediate-sulphidation gold mineralization. This interpretation was updated by another consultant who visited the Project and stated the Tuvatu system was dominated by intrusion related gold mineralization with a late intermediate sulphidation. The style of mineralization is thought to have evolved as the local monzonite intrusives cooled and magmatic fluids mixed with the groundwater fluids, resulting in the gradational changing of the mineralization and alteration styles.

Mineralization associated with the intrusive related gold is characterized 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 crystallization and early stages of cooling. As the system cooled, it was overprinted by a phase of phyllic alteration, which was characterized by a quartz-sericite-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-sericite selvages and commonly exhibit banded textures. A summary of the mineral paragenesis at Tuvatu is shown in Figure 7-9.

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 (e.g., Porgera, Hishikari) and invariably has a close association with gold mineralization. The precipitation of roscoelite generally requires the reduction of a vanadium-bearing mineralizing fluid. Reduction of the mineralizing 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 mineralizing 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 intrusive related mineralization overprinted by late intermediate epithermal episode.
- 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 Au and 0.3 m at 1,130 g/t Au).



Source: A-Izzedin (2000)

Figure 7-9 Paragenesis of mineralization

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 1 to 2 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

Tuvatu is an alkaline gold system related to the intrusion and subsequent cooling of a local monzonite. Stress regimes within intrusion 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 grades of any model developed. The mineralization is typical of this style of deposit in being confined to narrow structures with little wall rock alteration, which are hence "blind" outside of the mineralization.

8.0 DEPOSIT TYPES

8.1 Metallogenic Model – The Alkaline Gold Model

The Tuvatu Gold Property is an alkaline gold system (Hennigh 2019; Holden 2019). An alkali or alkaline gold system, according to Eric Jensen and Mark Barton, is a particular class of epithermal gold deposit wherein the potential scale and grade is considerably larger than typical 'hot-spring' epithermal gold system (Jensen and Barton 2000). Hot-spring deposits are bonanza grade systems, due to the narrow vertical extent of boiling horizons, and do not show large vertical or lateral continuity. In addition, the mineral system forms principally in pipe-type deposits. Alkaline gold systems, as a class in the broadest definition of epithermal systems, can show large tonnages and vertical extents beyond 1,000 m (Hedenquist 2000).

According to Jensen and Barton (2000), Kelley and Luddington (2002), and a review from Hennigh (2019), alkaline gold systems are characterized with the following characteristics:

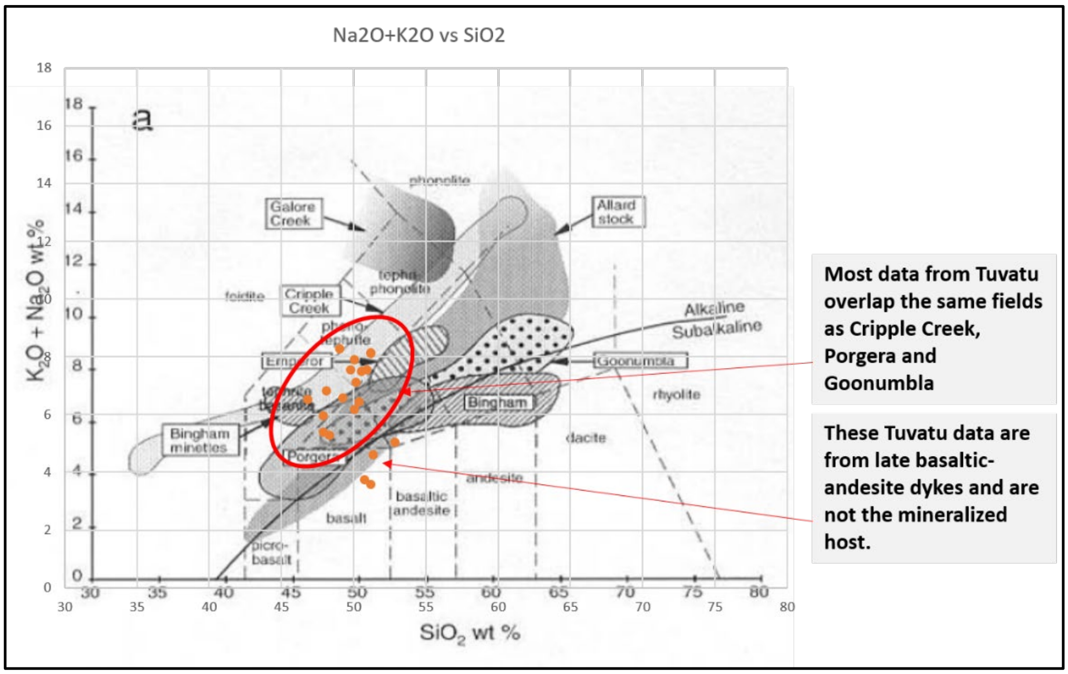
- Demonstrable connection to alkaline magmatism, both volcanic and plutonic.
- Large, often nebulous alteration systems dominated by potassium enrichment.
- General scarcity of silica within the alteration assemblage and in veins. While quartz is often present, veins are often dominated by potassium feldspar and carbonate minerals.
- Limited clay alteration.
- Overall low abundance of sulphide minerals.
- A large vertical profile, often in excess of 1,000 m.
- Gold deposition driven by fluid phase separation, hydrothermal "flashing", cooling, and fluid mixing.
- "Veins" that are commonly comprised of interconnected narrow veinlets. Individual veinlets commonly display extremely high gold grades such that a few small veinlets can carry economic mineralization over mineable widths.
- "Spider web" like networks of veins sometimes displaying a feeder structure at their core.
- Multiple mineralized centers within the great gold system.
- Gold occurring as native gold, gold tellurides, and gold-bearing pyrite.
- Enrichment in elements, including arsenic, tellurium, molybdenum, zinc, fluorine, and bismuth.
- Ag-to-Au ratios generally 1-to-1 or less.

Other examples of alkaline gold systems include Vatukoula (Fiji), Porgera (Papua New Guinea), and Cripple Creek (Colorado, USA) (Jensen and Barton 2000; Kelley and Luddington 2002; Richards, Bray, Channer, and Spooner 1997).

8.1.1 Lithochemistry

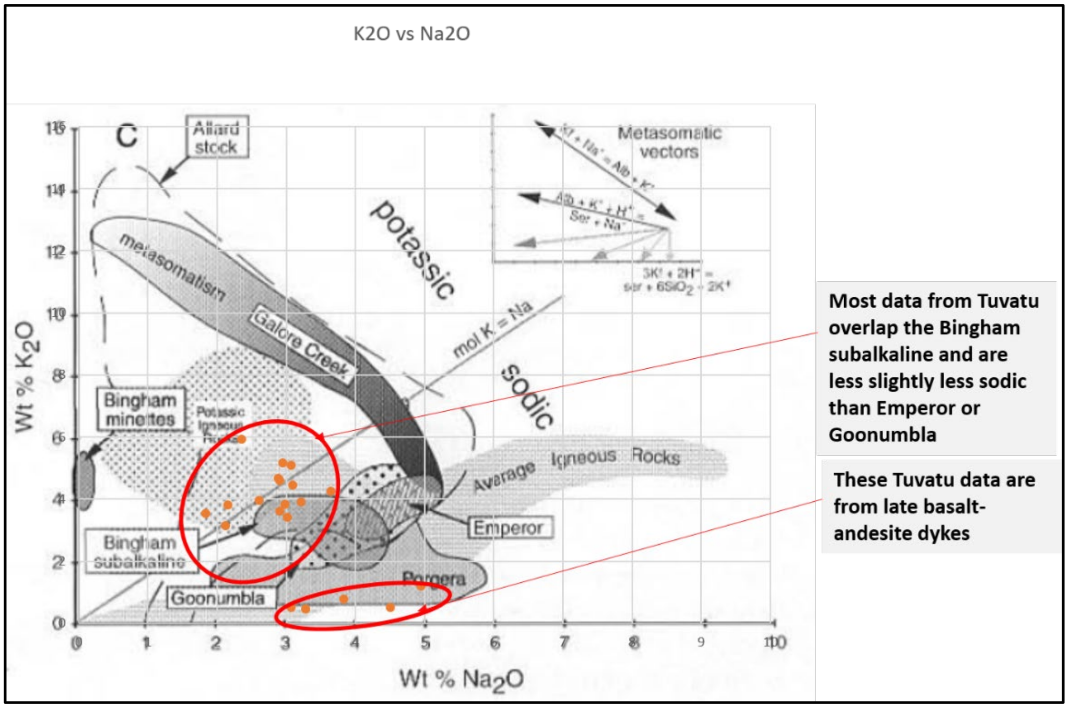
Alkaline magmatism relates to intrusions that are generally low in silica and are made up of alkali feldspars. The Tuvatu host is considered a monzonite (Scherbarth and Spry 2006). Monzonites are an alkaline lithology. Scherbarth and Spry present a series of multi-element lithochemistry tables, including the K_2O and Na_2O values for unaltered monzonite from Tuvatu. These data, when compared to graphs of Na_2O, K_2O, and SiO_2 presented by Jensen and Barton (2000), show clear similarities between Tuvatu and other world-class alkaline gold systems.

As shown in Figure 8-1, data from Tuvatu, when overlaid upon the Na_2O+K_2O versus SiO_2 plot from Jensen and Barton (2000), shows a favorable comparison with Porgera, Goonumbla (NSW), and Cripple Creek alkaline gold systems. As shown in Figure 8-2 and Figure 8-3, data from Tuvatu, when overlaid upon K2O versus Na2O plots, shows a favorable comparison with Bingham Canyon subalkaline gold deposit (Utah), Cripple Creek, and Golden Sunlight (Montana) and is slightly less sodic than both Emperor (Vatukoula, Fiji) and Porgera (Papua New Guinea). Scherbarth and Spry (2006) plotted lithochemistry, which also places the Project in this environment (Figure 8-4).



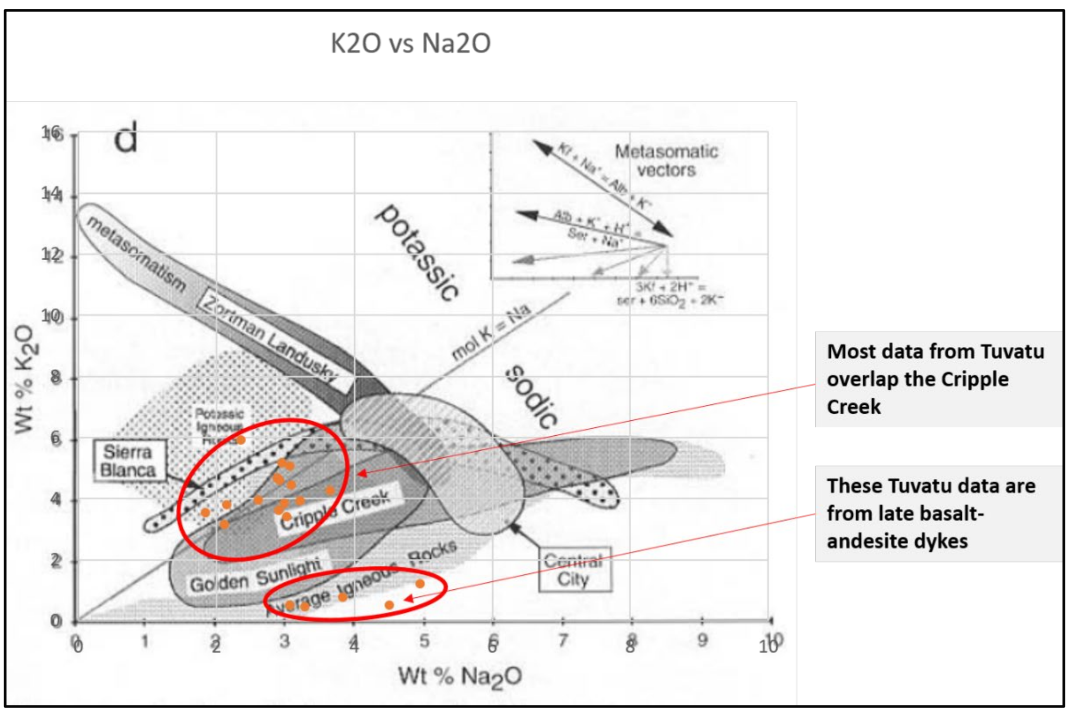
Source: Lion One modified from Jensen and Barton (2000)

Figure 8-1: Data from Tuvatu (orange dots) derived from Scherbarth and Spry (2006) compared to K_2O+Na_2O vs. SiO_2 plot of alkali gold systems from Jensen and Barton (2000)



Source: Lion One modified from Jensen and Barton (2000)

Figure 8-2: Data from Tuvatu (orange dots) derived from Scherbarth and Spry (2006) compared to K_2O vs. Na_2O of alkali gold systems from Jensen and Barton (2000)



Source: Lion One modified from Jensen and Barton (2000)

Figure 8-3: Data from Tuvatu (orange dots) derived from Scherbarth and Spry (2006) compared to K_2O vs. Na_2O of alkali gold systems from Jensen and Barton (2000)

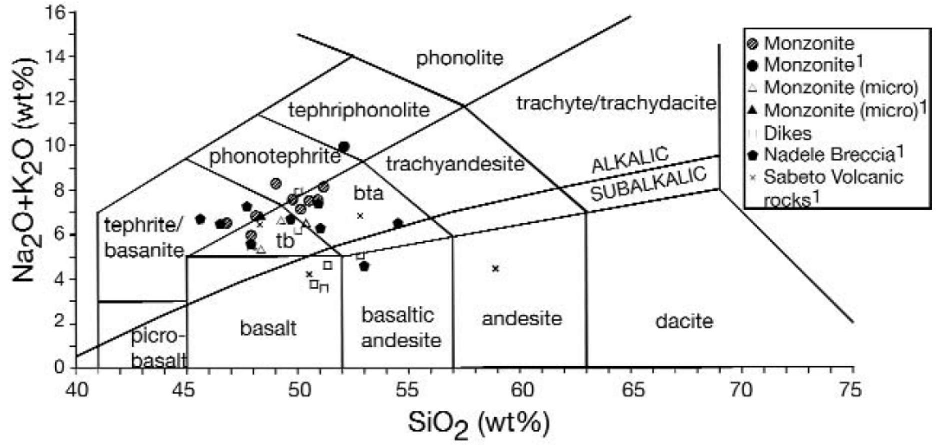


FIG. 6. Total alkali-silica diagram of Le Bas et al. (1986) for the Navilawa Monzonite, basaltic-andesite dikes, Nadele Breccia, and Sabeto Volcanics. The alkalic-subalkalic line is defined by Miyashiro (1978). Abbreviations: bta = basaltic-trachyandesite, tb = trachybasalt. [1]Data obtained from Hatcher (1998).

Source: Scherbarth and Spry (2006)

Figure 8-4: Plots of Na_2O + K_2O vs. SiO_2 for intrusions and volcanics from Tuvatu

This work presents supporting evidence that Tuvatu compares favorably with other alkali gold systems and in particular Porgera, Cripple Creek, Goonumbla, and Bingham Canyon subalkaline gold system.

8.1.2 Alteration

The alteration criteria for alkaline gold systems are:

- Large, often nebulous alteration systems dominated by potassium enrichment.
- General scarcity of silica within the alteration assemblage and in veins. While quartz is often present, veins are often dominated by potassium feldspar and carbonate minerals.
- Limited clay alteration.
- Overall low abundance of sulphide minerals.
- Low temperatures, even as low as sub-220°C.

At Tuvatu, potassium feldspar is dominant in many veins. The alteration into the wall-rock is normally restricted to within a few centimeters of the veins and is suggestive of a mineralizing fluid that is in equilibrium with the surrounding host. This fluid, in equilibrium, is not typical of a hot-spring epithermal deposit where meteoric waters dominate epithermal activity and react with the host rock producing intense alteration. Tuvatu is generally a low-silica system, and large/wide quartz veins are not observed. Quartz-carbonate-feldspar veins, with some sericite alteration (Figure 8-5 and Figure 8-6) dominate the main mineralized structures at Tuvatu. Roscoelite (a vanadium mica) is found within particularly high-grade, gold-bearing structures.

The HT-Lodes (Tuvatu Lodes) appear different to the main body of mineralization (such as the Upper Ridges Lodes). These HT-Lodes contain coarse, shreddy biotite (Figure 8-7), which is consistent with high-temperature potassic alteration (see also fluid inclusion work below). As Richards et al. (1997) described, the Porgera alkali system contains both an early magmatic phase (stage 1) and a later alkaline phase (stage 2) of mineralization. This is also consistent with multiple phases described at Cripple Creek and other Rocky Mountain alkaline systems (Kelley and Luddington 2002).

The low-sulphide content at Tuvatu is noted by Spry and Scherbarth (2006) and is consistent with the multiple element data. For gold assays >2 g/t from Tuvatu, the median sulphur content is 1.8%, suggesting total sulphides of less than 4%. Gypsum and anhydrite are present in some veins suggestive of sulphate as well as sulphide mineral phases.

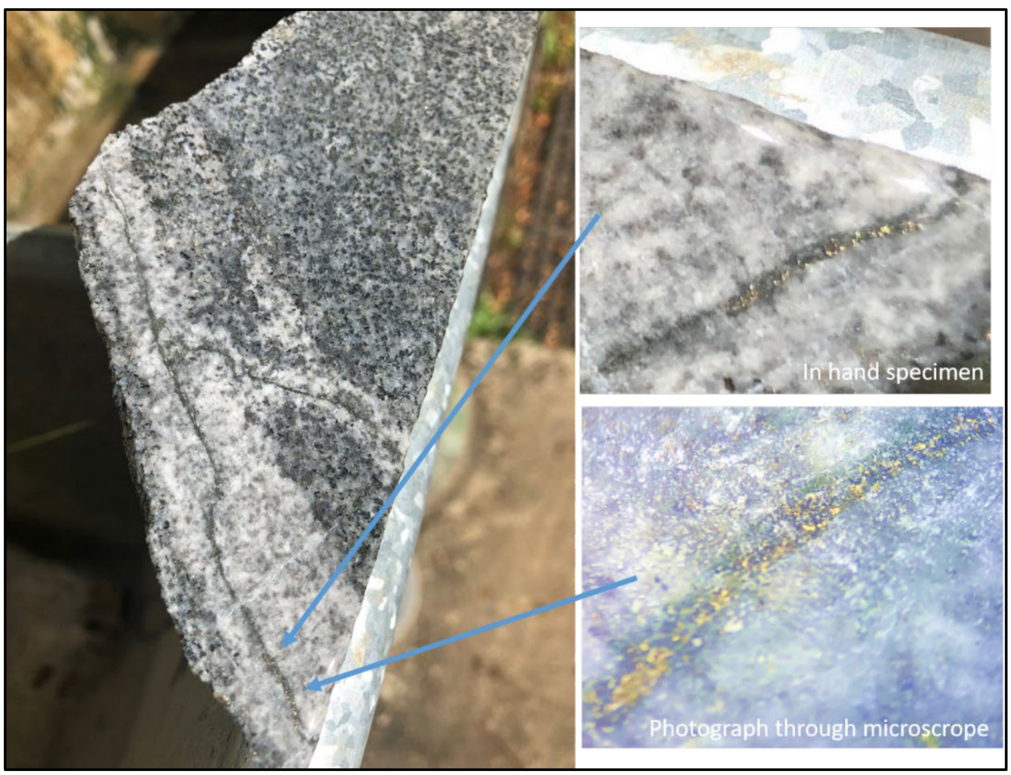
The sphalerite (Zn sulphide) present at Tuvatu is generally pale yellow; this is suggestive of a low-temperature mineralization (Figure 8-5), though in deeper parts of the Tuvatu system, bornite (copper iron sulphide) has been identified and indicates an increasing temperature of mineral deposition.

Source: Lion One (2019)

Figure 8-5: Some alteration styles at Tuvatu. Top left image is base-metal and silica dominant; bottom left image is potassium rich minerals, including secondary biotite and potassium feldspar. Right image is typical Upper Ridges style with sericite and carbonate with low-silica.

Source: Lion One (2019)

Figure 8-6: Feldspar + silica breccia fill (left) and high-temperature secondary biotite (right)



Source: Lion One (2019)

Figure 8-7: Very high-grade gold on a narrow vein with minimal wall-rock alteration

In assessing the temperature of mineralization, work by Emperor Gold Mining Company Limited on fluid inclusions from Tuvatu (Baker 1989) and subsequent work by Scherbarth and Spry (2006) suggest that the UR2 vein is generally deposited at low temperature, whereas the HT-Lodes are deposited at a higher temperature (Table 8-1).

Table 8-1: Fluid inclusion data compiled after Baker (1989) and Scherbarth and Spry (2006) showing the temperature of vein deposition

Lode Name	Lowest Temperature	Highest Temperature	Average Temperature
UR2	84	290	207
UR8	205	277	241
URW3	217	292	242
UR6	273	278	276
URW1	250	382	279
H-Lodes	176	413	285
UR9	210	382	291
T-Lodes	296	498	411

It is important to note that the UR2 vein, which is the most extensively modelled vein and is the most productive in the proposed mine plan, has the coolest depositional environment. This is consistent with the alkaline gold system model. Whereas the high-temperature T-Lodes are outside of the normal range for an alkali vein system but constitute only a small portion of the gold system.

8.1.3 Mineralization Associations

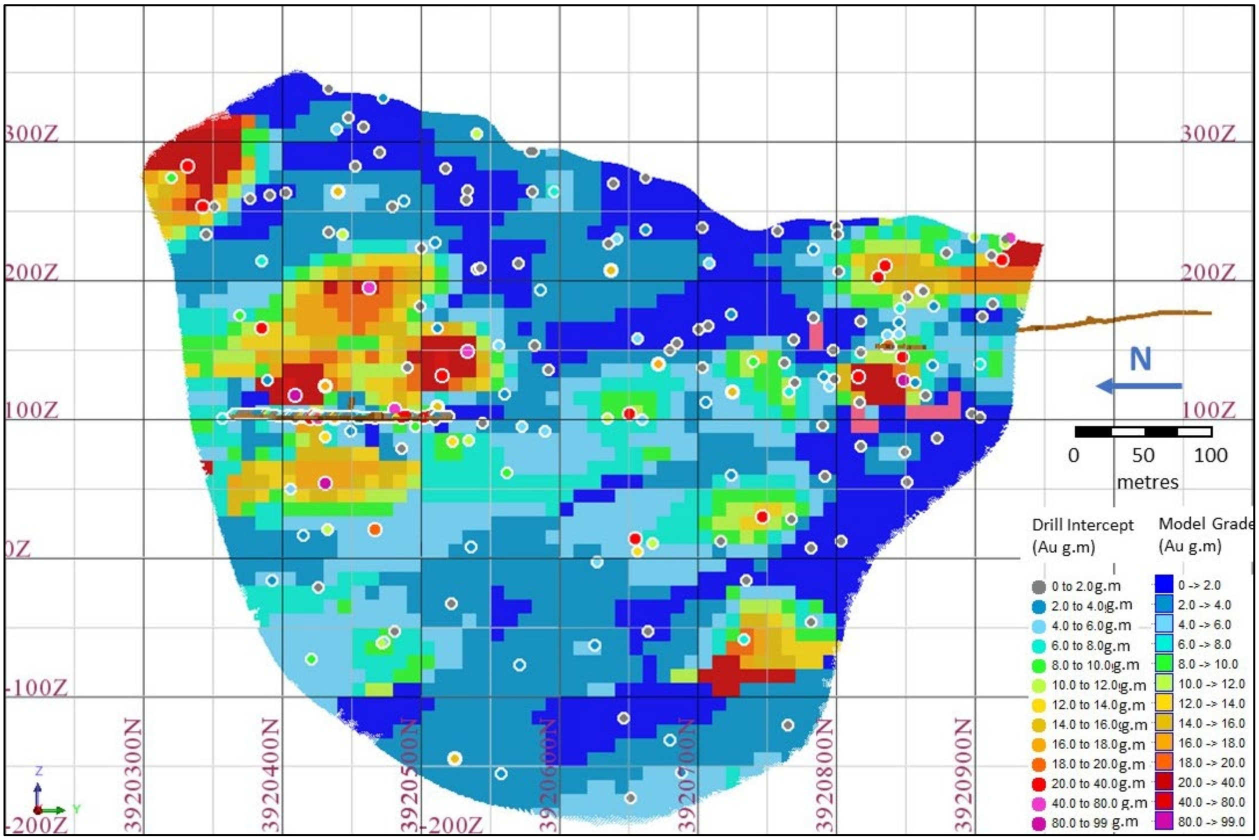
The criteria presented for mineralization associated with alkaline gold systems include:

- A large vertical profile, often in excess of 1,000 m.
- Gold deposition driven by fluid phase separation, hydrothermal “flashing,” cooling, and fluid mixing.
- “Veins” that are commonly comprised of interconnected narrow veinlets. Individual veinlets commonly display extremely high gold grades such that a few small veinlets can carry economic mineralization over mineable widths.
- “Spider web” like networks of veins sometimes displaying a feeder structure at their core.
- Multiple mineralized centers within the great gold system.
- Gold occurring as native gold, gold tellurides, and gold-bearing pyrite.
- Enrichment in elements, including arsenic, tellurium, molybdenum, zinc, fluorine, and bismuth.
- Ag-to-Au ratios generally 1-to-1 or less.

The following subsections address each of the criteria listed above.

A large vertical profile, often in excess of 1,000 m.

The Upper Ridge Lodes, in particular UR2, are confirmed currently over a vertical extent in excess of 500 m. This structure remains open at depth and may extend with vertical extent beyond 1,000 m. The mineralization appears to be not vertically constrained. Figure 8-8 shows large vertical extent of UR2 as illustrated with block-modeling.



Source: Mining Associates Ltd. (2014)

Figure 8-8: Large vertical extent of UR2 depicted in long-section of the block model. Labels indicate g/t*thickness.

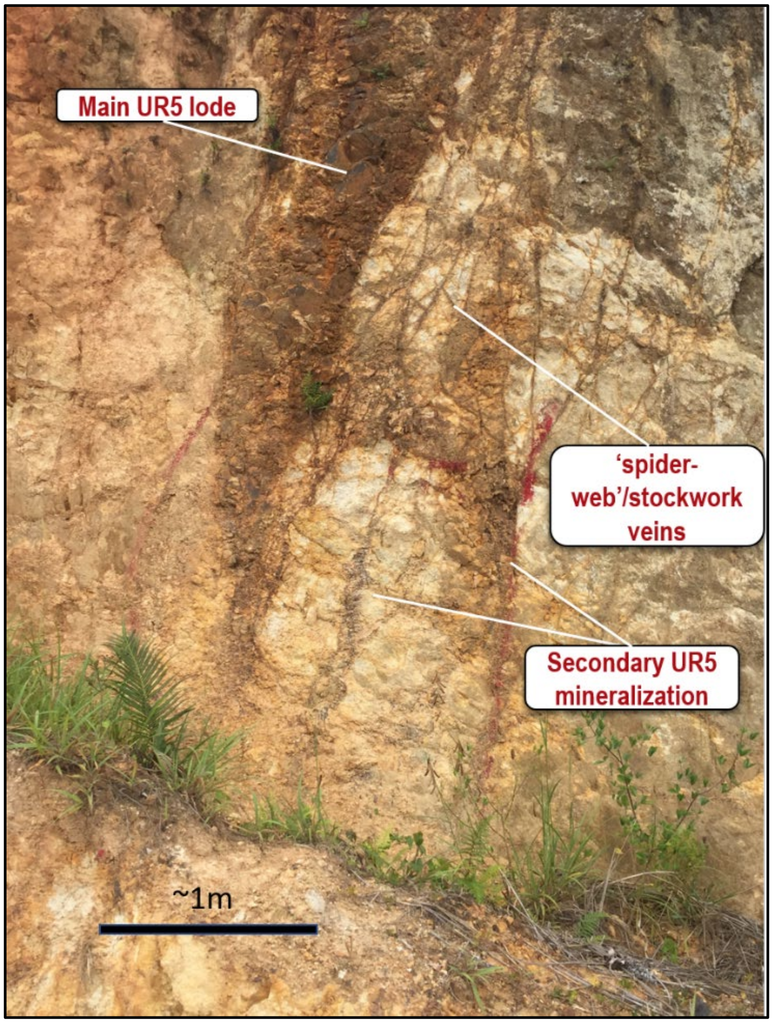
Gold deposition driven by fluid phase separation, hydrothermal "flashing", cooling, and fluid mixing.

This criteria, for alkaline systems, is difficult to objectively assess. However, many veins show multiple phases of mineralization highlighted by zoned veins, which is consistent with "flashing". Scherbarth and Spry (2006) also noted fluid mixing and multiple pulses of fluid flow exhibited in fluid inclusion work.

Veins at Tuvatu are commonly comprised of interconnected narrow veinlets. Individual veinlets commonly display extremely high gold grades such that a few small veinlets can carry economic mineralization over mineable widths.

"Spider web" like networks of veins sometimes displaying a feeder structure at their core.

Referencing Figure 8-5 and Figure 8-7 above, narrow veins can show very high-grade gold. The main mineralized structures, such as UR2, are a composite of anastomosing veins and veinlets. A feeder structure has not yet been identified and represents, potentially, a significant target at depth. Figure 8-9, as an example of spider-web' vein networks, shows a single major vein, several parallel structures, and "spider web" or stockwork veins accompanying on a surface outcrop of the UR5 (Upper Ridges style) Lode at Tuvatu.



Source: Lion One (2018)

Figure 8-9: Surface outcrop of UR5 (Upper Ridges style) Lode at Tuvatu, showing a single major vein, several parallel structures, and "spider web" or stockwork veins accompanying

Multiple mineralized centers within the greater gold system.

There is more than 7 km of strike of the mineral system in the Tuvatu district, including in the central part of the Navilawa Caldera. Other centers such as Banana Creek and the southern Jomaki-Ura-Kubu system suggest multiple mineralizing centers.

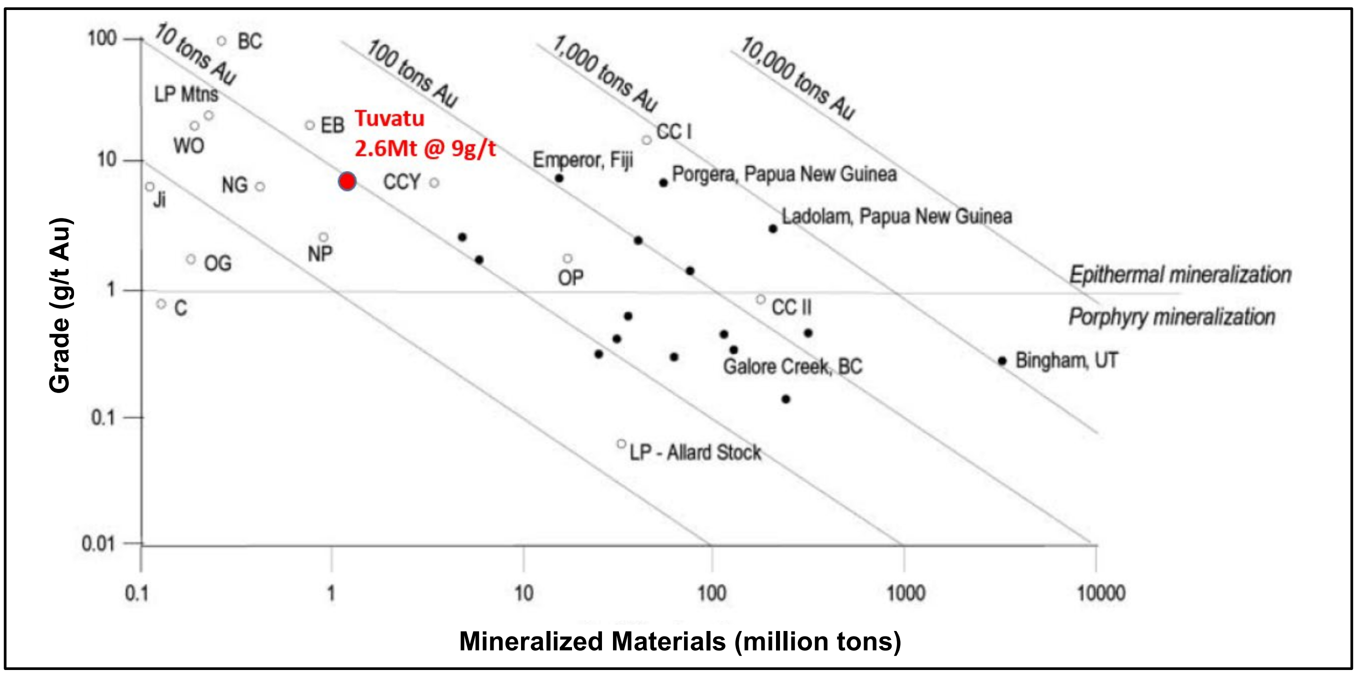
Gold occurring as native gold, gold tellurides, gold-bearing pyrite. Enrichment in elements, including arsenic, tellurium, molybdenum, zinc, fluorine, and bismuth. Silver-to-gold ratios generally 1-to-1 or less.

Gold at Tuvatu occurs as native gold (Figure 8-7) and is associated with tellurides and pyrite. Multi-element assay data is restricted to recent data only (historic Emperor data was assayed just for gold). Overall, the Ag-Au ratio is less than 1, and the other elements are enriched as indicated in Table 8-2. The only exception is a low bismuth content with an average of only 1.3 ppm.

In summary, Tuvatu displays many of the characteristics of typical alkaline gold systems. A comparison of identified tonnes and grade at Tuvatu, compared to other alkaline gold systems, is shown in Figure 8-10.

Table 8-2: Multi-element data for all drill assays >2 g/t Au (as of Holden, 2019)

Element	Mean	Comment
Gold (ppm)	16.7	2 g/t cut-off for samples with also multi-element data.
Silver (ppm)	7.2	Less than 1:1 (consistent with alkali model)
Arsenic (ppm)	688.3	Enriched arsenic (consistent with alkali model)
Bismuth (ppm)	1.3	Low
Molybdenum (ppm)	41.6	Enriched molybdenum (consistent with alkali model)
Tellurium (ppm)	14.7	Enriched tellurium (consistent with alkali model)
Zinc (ppm)	617.3	Enriched zinc (consistent with alkali model)



Source: Lion One modified from Jensen and Barton (2000)

Figure 8-10: Grade vs. tonnes (log) of alkaline gold systems from Kelley and Luddington (2002) after Jensen and Barton (2000)

9.0 EXPLORATION

Lion One has undertaken exploration activities for the Project in three main phases: surface work and limited exploration drilling from 2008 to 2010, more extensive drilling in 2012 to 2013 focused on extensional exploration areas, and drilling and intensive surface work including benching from 2016 to 2018.

9.1 2008 to 2010 Lion One Exploration

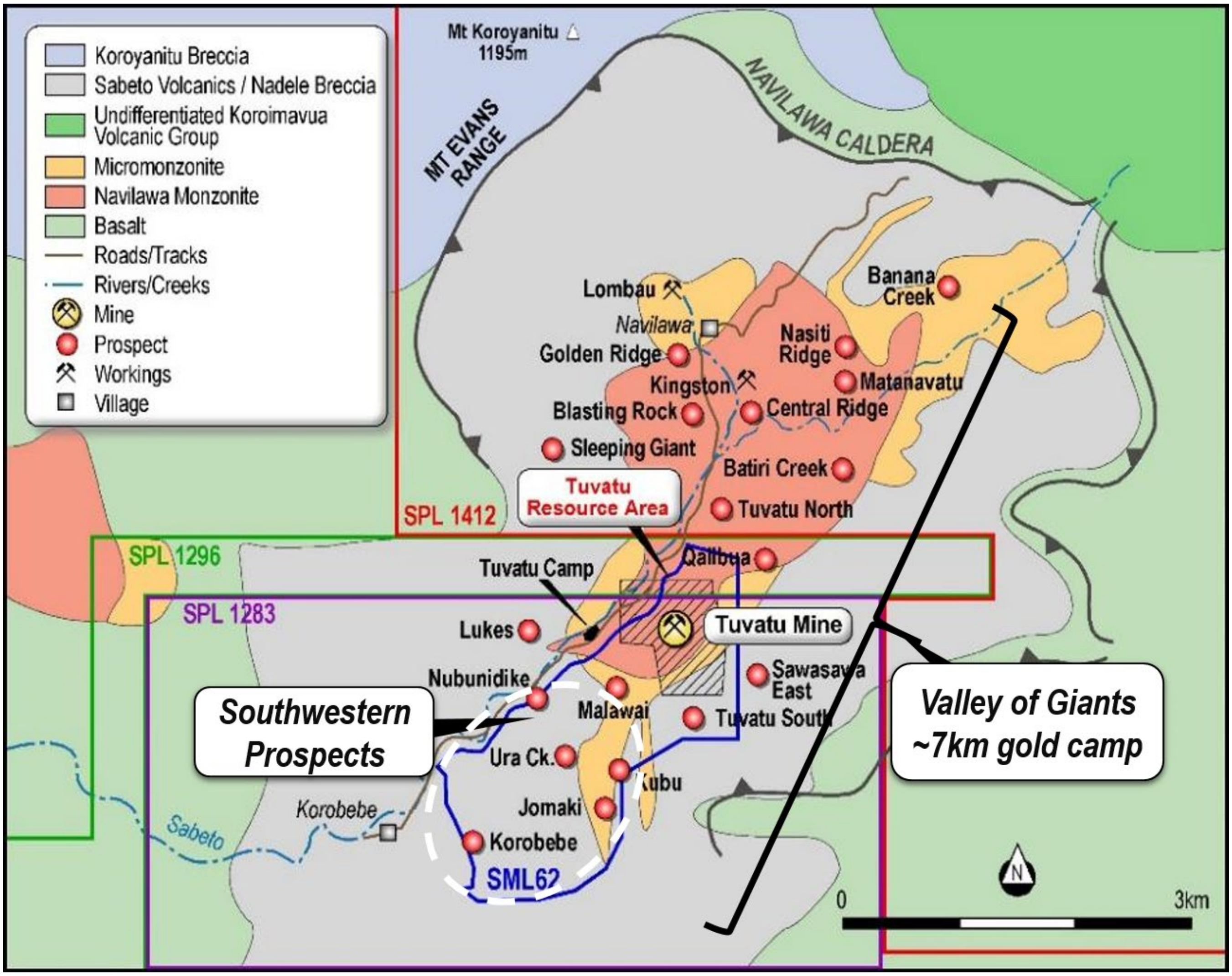
During 2008, Lion One completed extensive mapping and geochemical sampling. Two surface drill holes were also completed. Field work was carried out by Lion One staff (W. Kuruisaravi, R. Sulua, and S. Bulu) under the direction of various expatriate consulting geologists. The mapping, rock chip, and channel 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 to 2003 by TGM were followed up (Figure 9-1):

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

Detailed geological mapping and rock chip and channel sampling in the region south of the Tuvatu resource area and Qalibua Creek were 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 laboratories in Brisbane, Australia.

Two surface DDHs (TUDDH-338 and TUDDH-340) totaling 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.



Source: Lion One (2019)

Figure 9-1: 2008 prospect locations

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 relogging and resampling of approximately 10,000 m of the total 60,000 m of core. In addition to the ongoing program of mapping, core relogging and resampling, 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 CSF.

Initial sampling was between the 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 relogging and resampling 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 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.

9.3 2017 to 2018 Lion One Exploration

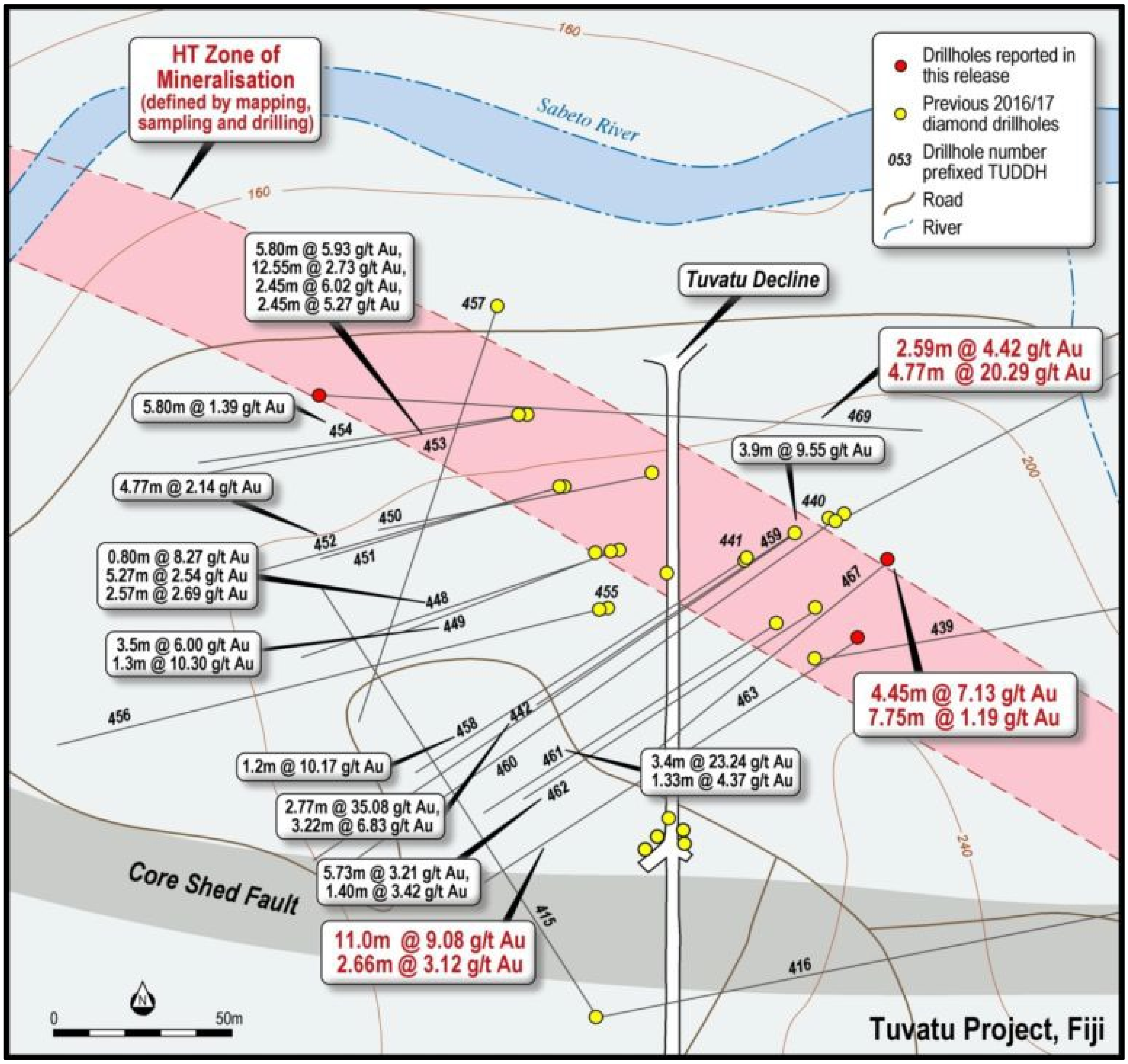
During 2017, Lion One conducted extensional exploration drilling in the vicinity of the H and Tuvatu Lodes at Tuvatu. This work concluded with encouraging results of wide and high-grade drill intersections. The result of this work has enabled a revaluation of the H and Tuvatu Lodes (now referred to as the HT Corridor (Figure 9-2) and the zone is considered to be geologically different to the main Tuvatu (Upper Ridges) Lodes. The HT Corridor characteristics include:

- A higher temperature alteration (potassic) characterized by shreddy biotite.
- Increased fracture density with vuggy clay zones and open-fill veins.
- A strong association of gold with base-metal sulphides, including galena and chalcopyrite.

Results from the 2017 program included:

- 11 m averaging 9.1 g/t Au from 80.8 m including:
  - 2 m averaging 15.3 g/t Au.
- 4.8 m averaging 20.3 g/t Au from 21.5 m including:
  - 1.8 m averaging 51.8 g/t Au.
- 4.5 m averaging 7.1 g/t Au from 214.5 m including:
  - 1.8 m averaging 15.0 g/t Au.

Based on the evaluation of the area, Lion One geologists believe that the HT Corridor lodes have a strong potential for additional extensional exploration discoveries. The alteration has been observed over 500 m along strike from the existing drilling with several extensions remaining to be tested.



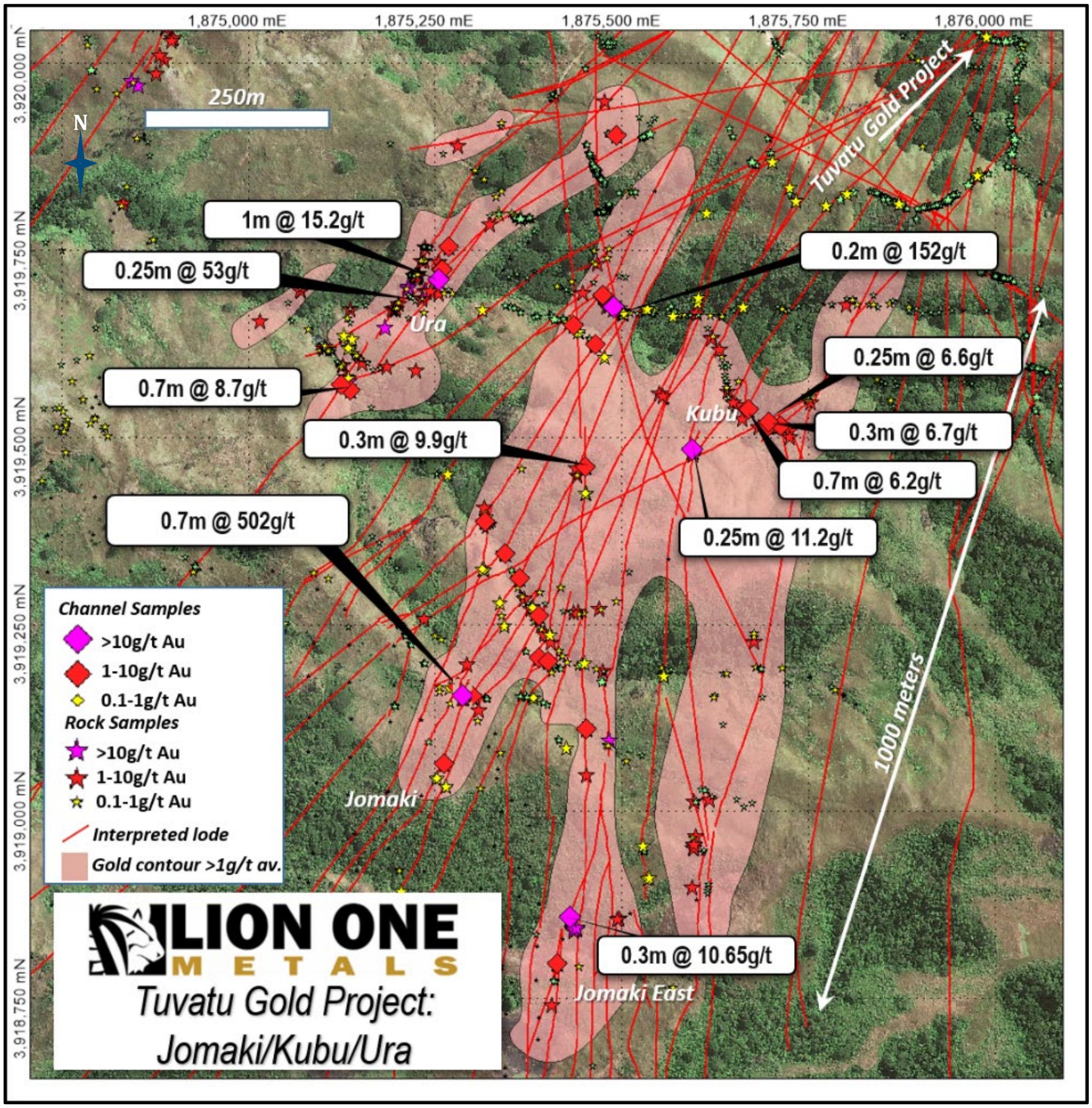
Source: Lion One (2018)

Figure 9-2: 2017 summary of drilling in the HT Corridor (plan view)

Starting in 2017 and continuing through 2018, Lion One has conducted more than 10 km of benching, mapping, and sampling principally on the prospects to the south of Tuvatu. This work involved the cutting (using an excavator) of tracks along the sides of ridges and hills to expose the geology. All areas are mapped, with sampling undertaken orthogonal to structures using rigorous channel sampling methods. This work has increased the geological understand of the southern prospects and has enabled the prioritization of several target areas. Highlights from the sampling have included (Figure 9-3):

- 502 g/t Au over 0.7 m width at the Jomaki West Prospect
- 152 g/t Au over 0.2 m at the Jomaki West

- 53.1 g/t Au over 0.25 m width at the Ura Creek Prospect
- 15.2 g/t over 1 m at the Ura Creek Prospect
- 10.65 g/t over 0.3 m width at Jomaki East Prospect
- 11.2 g/t Au over 0.5 m width at the Kubu Prospect



Source: Lion One (2018)

Figure 9-3: 2017 to 2018 benching program for the southern prospects

9.3.1 Results: Exploration Potential

The Project has clear exploration potential. Analysis of the drilling conducted to date indicates that the known Mineral Resource is open in several directions along strike and down dip. In particular, the HT Corridor shows a higher temperature mineralization and is a structural corridor that can be traced for at least a further 500 m long strike.

Prospects to the south of Tuvatu, namely Ura Creek, Jomaki West, Jomaki East, and Kubu have extensive surface anomalism over an area of at least 1,000 m x 1,000 m. In total, more than 30 mineralized structures have been identified throughout this area. While it is unlikely that all of these structures will contain economically extractable mineralization, the potential exists to find one or two systems that are equivalent to the main Tuvatu Mineral Resource. However, surface geology is dominated by Nadele Volcanics / Breccia, which is recognized as a poor host to mineralization. As such, targets are at depth in the younger monzonite rocks.

9.4 Tuvatu Gold Project Extensional Exploration

9.4.1 Deposit Style

The Tuvatu gold deposit consists of a series of 0.1 to >10 m wide zones of mineralization hosted within a monzonite. The monzonite has been unearthed in the floor of the collapsed Navilawa Caldera.

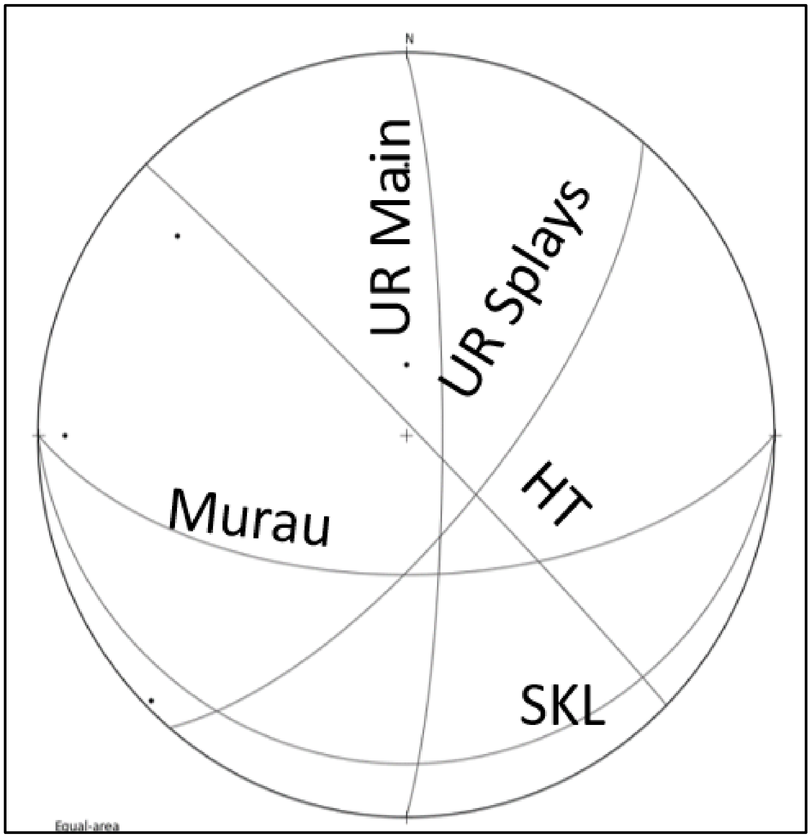
Mineralization at Tuvatu has been confirmed over a north–south strike length of over 900 m and from outcropping at surface to depths in excess of 800 m. The system has a complex structural control, with high-grade bonanza plunging shoots within individual zones. Mineralization is confined to structural zones consisting of veins and veinlets of quartz-potassium feldspar-carbonate. Sulphide content is generally low, but a typical association includes minor components of pyrite-arsenopyrite-galena-sphalerite-bornite-tellurides-roscoelite. Gold is generally fine free-gold, with occasional visible gold sighted in drilling. However, the 500 zones and M-lodes, commonly display coarse visible gold. The individual lodes and characteristics are listed in Table 9-1.

Table 9-1: Tuvatu Lode summary

Structural Mineralized Zones	Principal Veins	Orientation (Figure 9-4)	Comments
UR Zones	UR1, URW1, URW2, UR2, UR3, UR4, UR5, UR6, UR7, UR8	UR1 to UR3: North–south strike and dipping 70 to 85° east dip UR4 to UR8 (UR splays): Splaying from north–south to northeast–southwest with 70 to 85° east dip (UR4-8 have affiliations with 500 zones below)	UR is an abbreviation of “Upper Ridges”. Contributing >80% of the current mine plan, these are the most productive lodes. Lodes extend over strike lengths of >900 m, with 20 to 30 m long bonanza zones plunging steeply southward. Mineralization consists of zones of quartz-feldspar veins and veinlets with minor sulphide (Asp-Py) and roscoelite.

table continues…

Structural Mineralized Zones	Principal Veins	Orientation (Figure 9-4)	Comments
500 Zones	500prefix	These are at-depth continuations of the zone of the UR splays, though linking directly to individual splays has not yet been possible. These show orientations generally NE-SW strike and steep dips, with some evidence of a steep SW plunge to the high-grade core of the mineralisation	Discovered in hole TUDDH-500. This zone consists of multiple structures of high-grade gold mineralization, and is a potential feeder zone to the main Tuvatu UR lodes above. This zone does not yet have sufficient drilling to define a resource estimation, though mineralization is observed on a strike length of >200 m, with a vertical extent of 200 to 400 m, and is located close to the contact of monzonite and andesite.
HT Zone	H1, H2, T1, T2	Steep west-northwest–east-southeast to northwest–southeast strike with vertical to dipping steeply both northeast and southwest	These are the original outcropping discoveries with T = Tuvatu. Wide vuggy zones of variable gold mineralization. Alteration includes potassium feldspar and shreddy coarse biotite. Galena and sphalerite are common.
Murau-West	M1, M2, M3, M4, Snake (S1-3), W1, W2, W3, W4	East–west strike with 60 to 80° dip south	Narrow zones with the most abundant visible gold in the system. Minor sulphide (py) content and associated telluride
SKL Lodes	SKL1, SKL2, SKL3, SKL4, SKL5	East–west strike with shallow (10 to 30°) dip to the south	Flat zones with limited extent (20 to >50 m), but with very high (>30 g/t) bonanza grades. Equivalent to the “flatmakes” at Vatakoula.



Source: Lion One (2018)

Figure 9-4: Stereonet showing general orientation of lodes at Tuvatu (UR Splays include similar quadrant for 500 Zones)

9.4.2 Extensional and Infill Exploration Targeting

In 2019, Lion One commenced a deep exploration targeting and drill program. The aim of the program is target for deep feeder zones to the main lodes at Tuvatu. Based on the alkaline model, the Tuvatu system should have veins that coalesce into single or multiple feeder zones.

The principal outcome of this program, to date, has been the identification of the 500 zone mineralization. This zone is deep beneath the UR splay structures and consists of high-grade gold associated with pyrite-roscoelite-carbonate quartz veins, with minor telluride and base metal sulphides. Individual veins include vein breccia, and carbonate-silicification. The 500 zone contains regular visible gold in hand-specimen. Hole TUDDH-494 was extended in 2022, and intersected epidote-bornite-gold mineralization more than 200 m beneath the 500 zone drilling to date (refer to Figure 9-5, Figure 9-6, Figure 9-7, Figure 9-8).

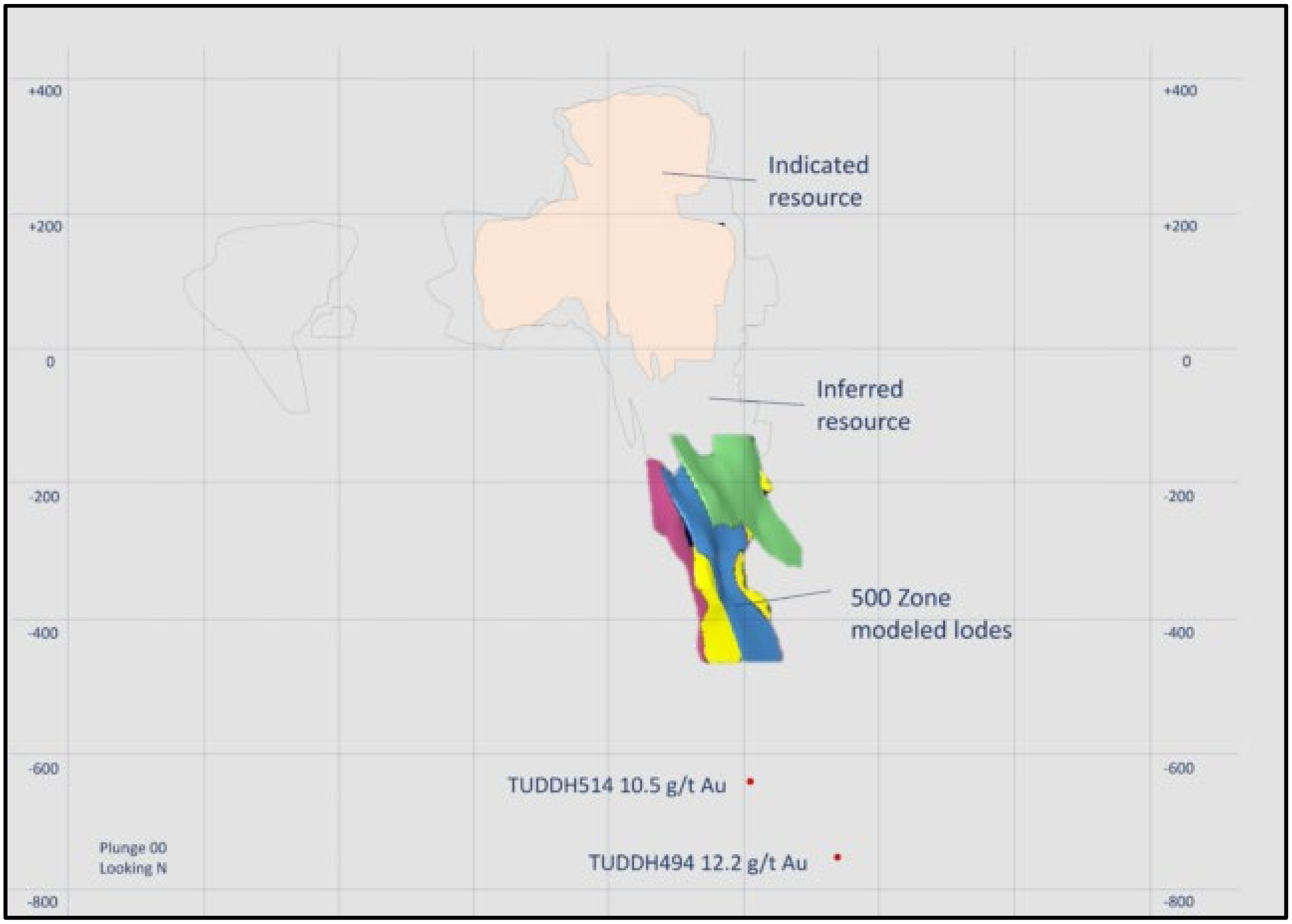


Figure 9-5: Schematics long Section View North: 500 Zone geological models beneath the main Tuvatu Resource

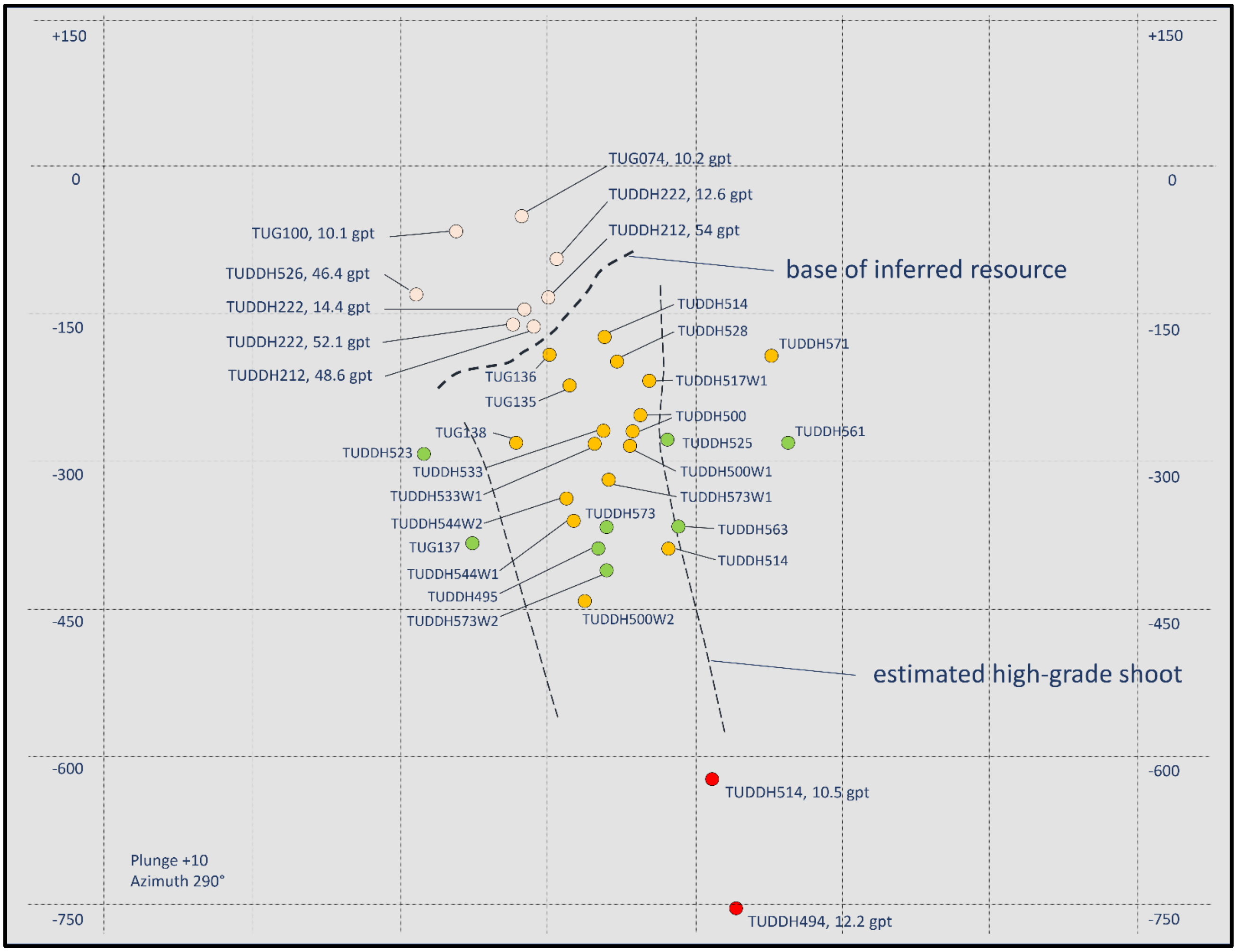


Figure 9-6: Long Section pierce points (view NW) of drilling in the 500 Zone.

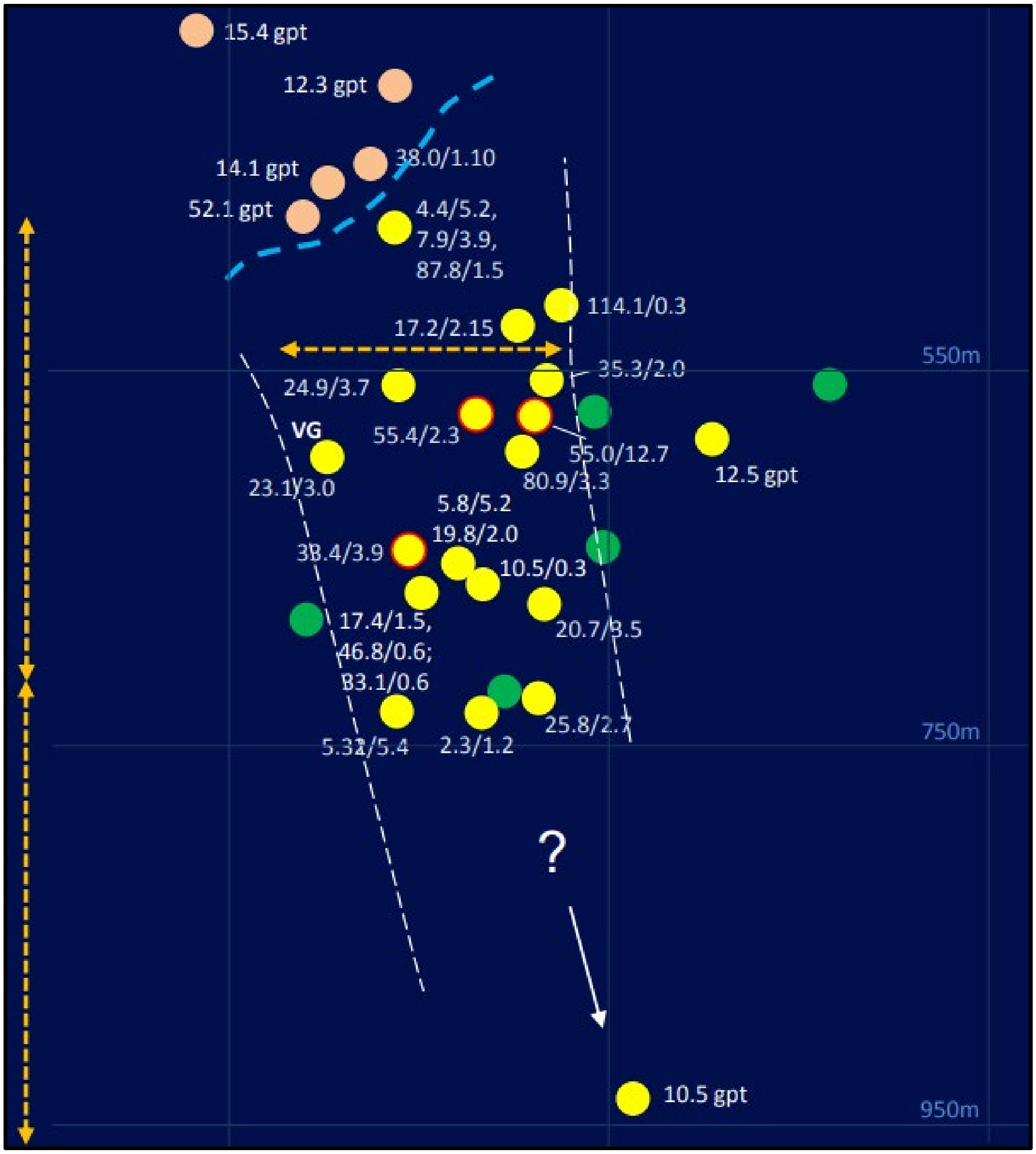


Figure 9-7: Long Section pierce points (view NW) of drilling in the 500 Zone, labelled with grade (g/t Au)/drilled width. Orange dots refers to holes in the previous resource estimate; yellow indicates new holes with appreciable mineralization; green indicates holes without appreciable mineralization.

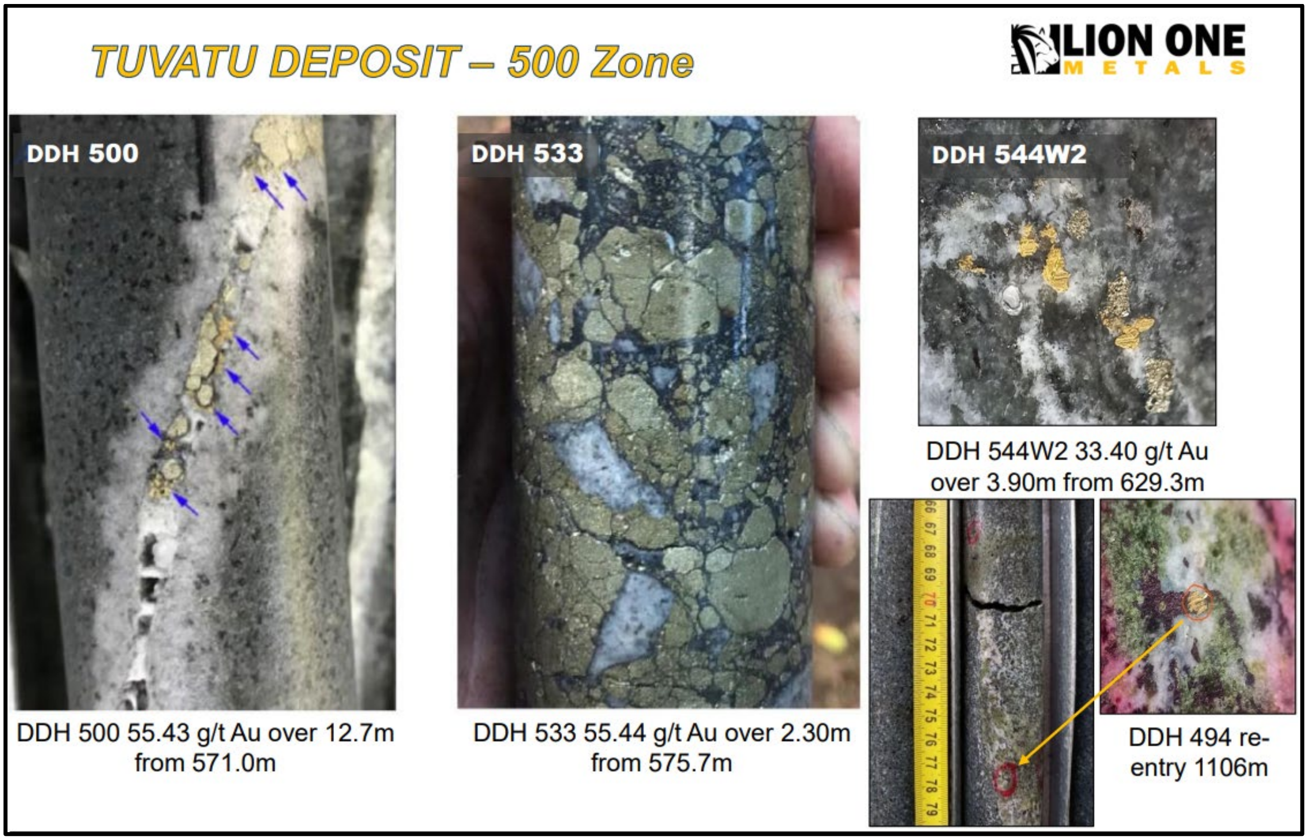


Figure 9-8: 500 zone mineralization in drill core.

The Company has also embarked on extensional infill drill program targeting areas for the initial stages of mining. This drill program has generally confirmed the known models in the main UR, M and SKL lodes and have added further intersections pierce points that will be incorporated into future mine-scheduling and resource updates. This work is on-going and the Company considers is not yet at a stage to warrant mineral resource estimate.

9.5 Navilawa Caldera District Exploration

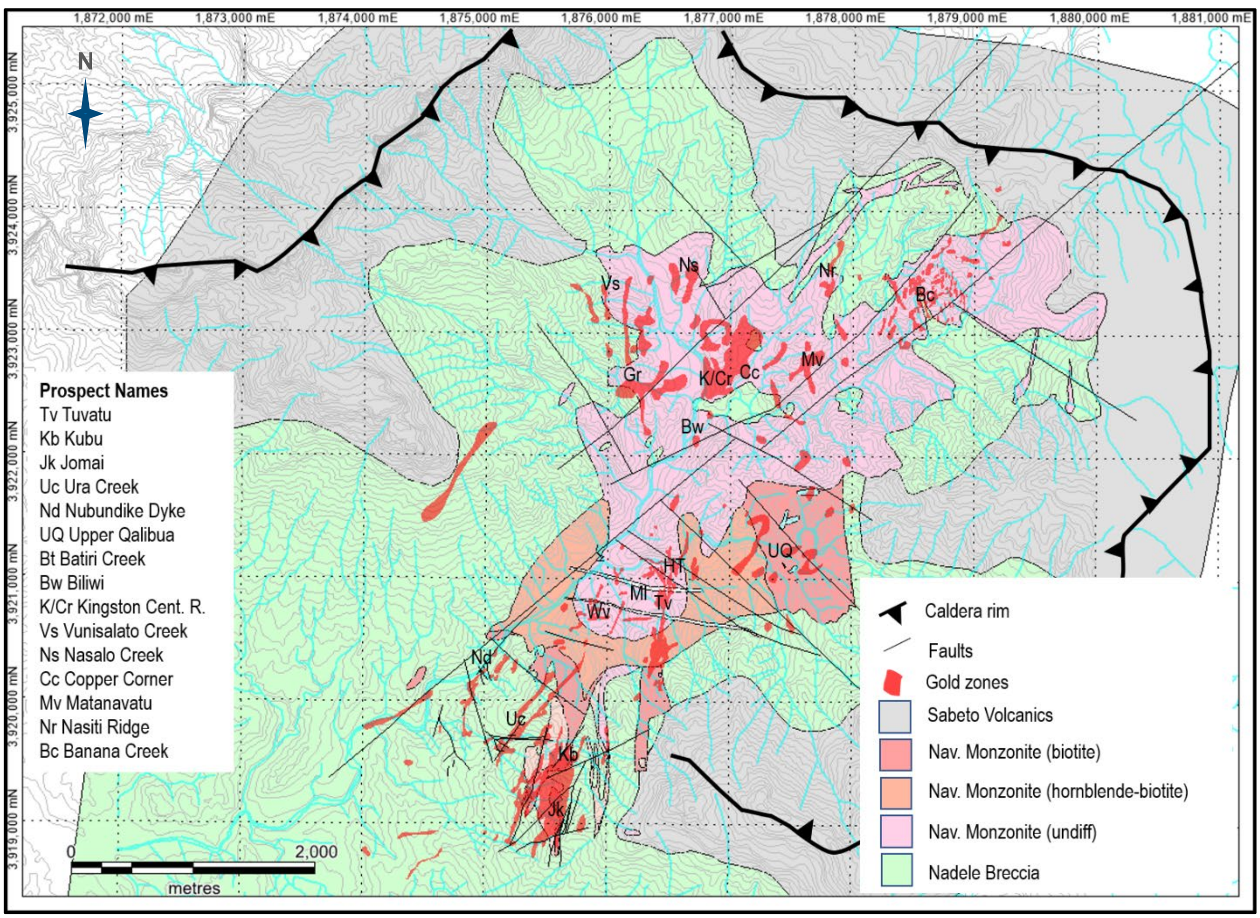
9.5.1 District Geological Setting

As described by Scherbarth and Spry (2006) (paraphrased/summarised):

The oldest geologic unit in the Tuvatu area is the 12 to 26 Ma Nadele Breccia, which is a member of the Wainimala Group. The Nadele Breccia consists of andesitic to basaltic reworked, polymict volcanic breccias, pillow lavas, and sedimentary rocks. The polymict volcanic breccias appear to be the dominant unit of the Nadele Breccia. Thin layers of volcanic sandstone and siltstone are commonly interbedded with the breccias and can exhibit crossbedding. Previous researchers identified zeolite, chlorite, and epidote within the groundmass of the Nadele Breccia, which formed in response to the low-grade metamorphism.

The Wainamala Group is overlain by members of the Sabeto Volcanics, the basal unit of the Koroimavua Volcanic Group, which consists of augite-biotite flows and breccia with a basal sequence of andesitic and dacitic lithic and crystal tuffs, grits, and agglomerates with minor flows. K-Ar dating of the Sabeto Volcanics yielded an age of 5.4 ± 0.1 Ma. These volcanic rocks constitute roughly 10 to 15% of the topographic high zone along the eastern margin of the deposit area. The Nadele Breccia was intruded by the 4.9 ± 0.1 Ma Navilawa Monzonite, interpreted as a late intrusive and is chemically similar to the Sabeto Volcanics, though it is approximately 500,000 years younger in age. The Navilawa Monzonite consists of multiple phases, including an early fine-grained micromonzonite, a medium-grained monzonite, basaltic-andesite dikes, and late-stage pegmatitic dikes. The medium-grained monzonite is surrounded by an envelope of micromonzonite and contains inclusions of micromonzonite. Together, the micromonzonite and medium-grained monzonite have an elliptical shape with a northeast–southwest axis 3 km long and 2 km wide. Copper mineralization occurs at the center of the Navilawa Monzonite near the transition zone between micromonzonite and medium-grained monzonite at the Kingston prospect, 1.8 km north of the Tuvatu deposit. Faults in the Tuvatu area strike north to northeast and northeast to southeast with near-vertical dips; however, in places, they dip shallowly to the west. Some of these shallow-dipping structures host epithermal gold mineralization near the margin of the Navilawa Monzonite.

The current interpretation of geology, by Lion One geologists, is shown in Figure 9-9. A schematic cross-section is displayed in Figure 9-10.



Source: Lion One (2020)

Figure 9-9: District geology map of the Navilawa Caldera

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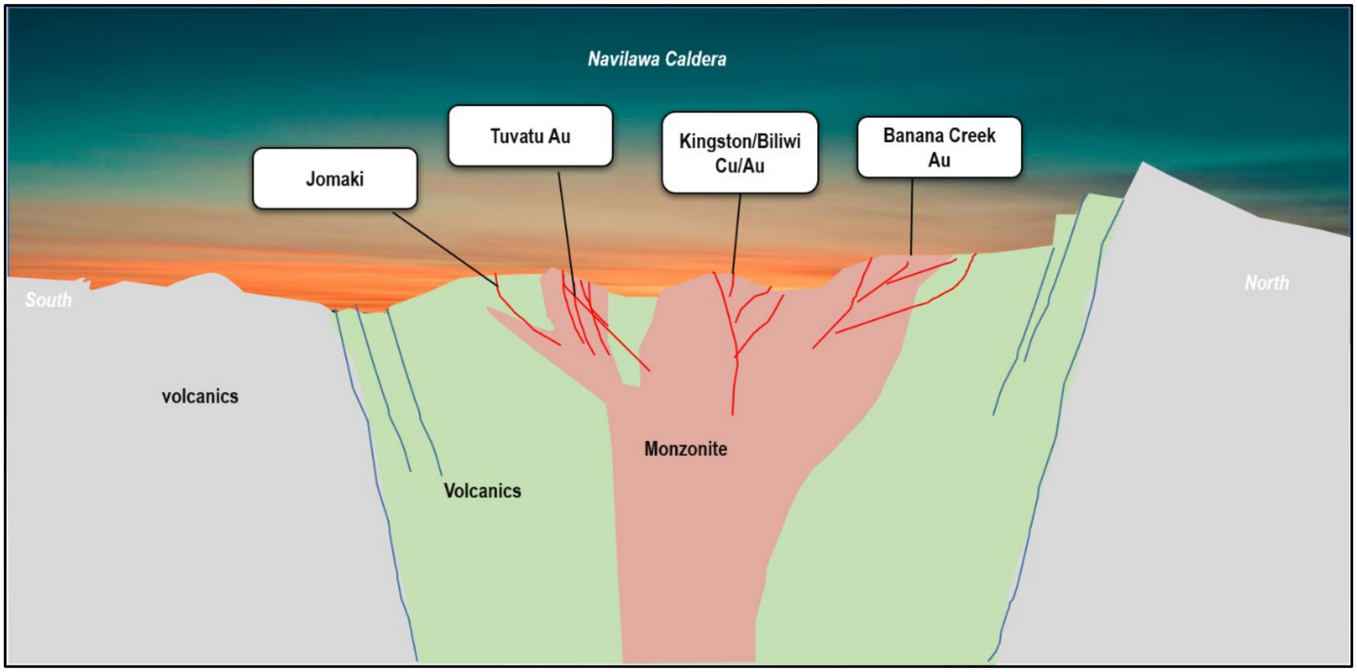


Source: Lion One (2019)

Figure 9-10: Schematic cross-section view west

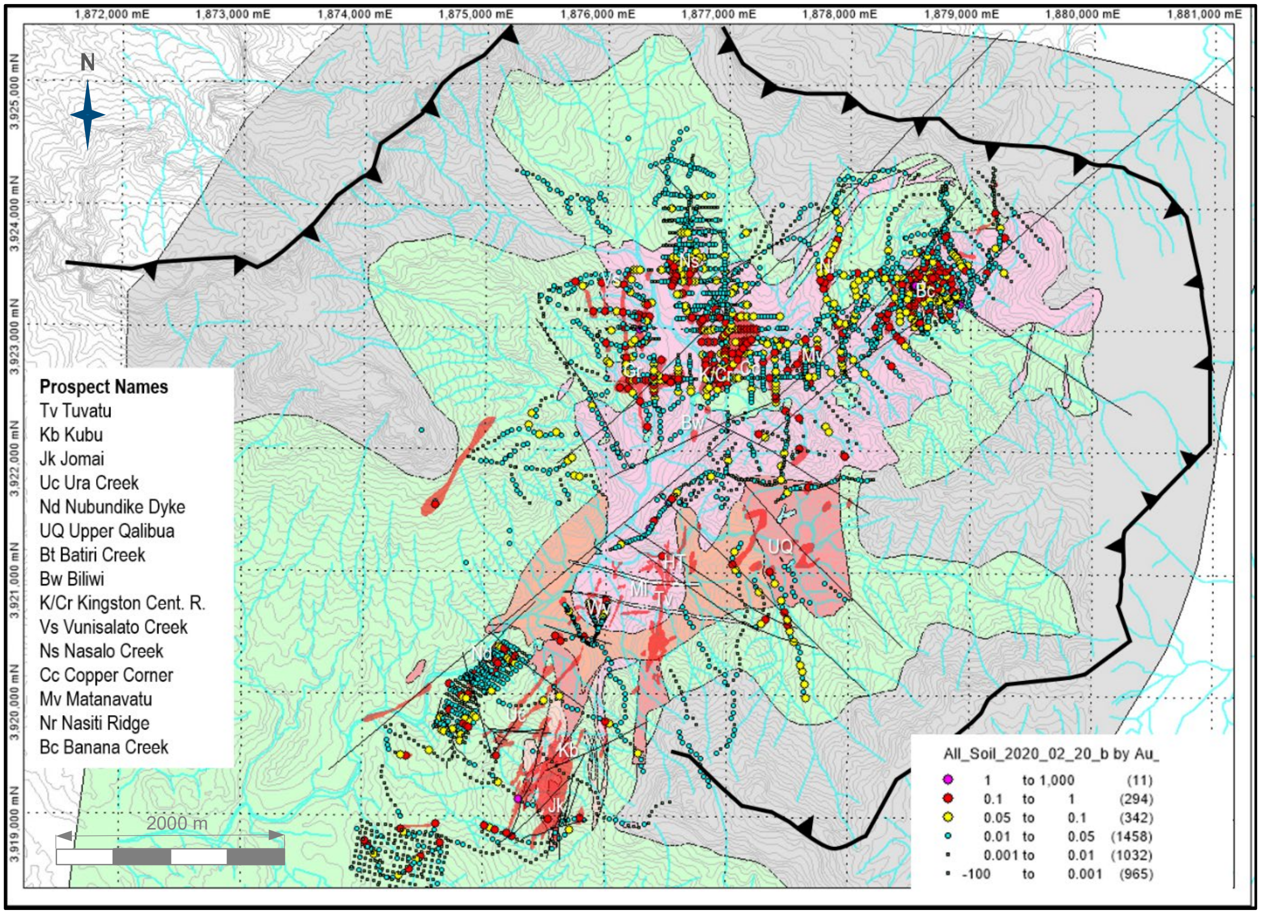
9.5.2 Surface Geochemistry and Mapping Techniques

9.5.2.1 Soil Sampling

Lion One has compiled a database, including soil sampling from previous explorers. The original sampling techniques of previous explorers have not been recorded in the database. In the case of Lion One, a sieved b-horizon sample is collected and processed in the field before being sent to ALS Laboratory in Townsville, Australia for Au-AA25 (fire assay) with a 0.02 ppm (20 ppb) detection limit and ME61S for 33 element aqua regia digest analysis.

There are 4,107 soil samples recorded in the database of which 2,092 have analysis for multi-element and the remainder for gold only.

Due to the variable nature of the soil sampling, including a mixing of historic and Lion One results, a combination of ridge and spur sampling, with gridded soils, data is not levelled and cannot be used reliably for prospect prioritisation. As such, the soil sampling data is considered to be used only as a guide for on-ground prospecting. Furthermore, there is no soil sampling data collected over the Tuvatu deposit; hence, there is no effective comparison for known mineralized zones compared to undrilled prospects. Nevertheless, large zones of >0.1 ppm (100 ppb) gold in soils have been identified at Banana Creek, Kingston / Central Ridges, and Golden Ridge areas; these are high tenor occurrences (Figure 9-11).



Source: Lion One (2020)

Figure 9-11: Soil samples at the Tuvatu Gold Project in ppm Au

9.5.2.2 Benching / Chip-Channel Sampling / Rock Chip Sampling

Lion One has compiled a database consisting of 3,675 samples recorded as rock or rock-chip samples. The sampling technique and assay method for these are widely variable, and in many cases, assay results may be biased to an unrecorded specific structure. As such, these selective rock-chip data are used sparingly by Lion One and used in identifying mineralized trends rather than specific targets (Figure 9-12).

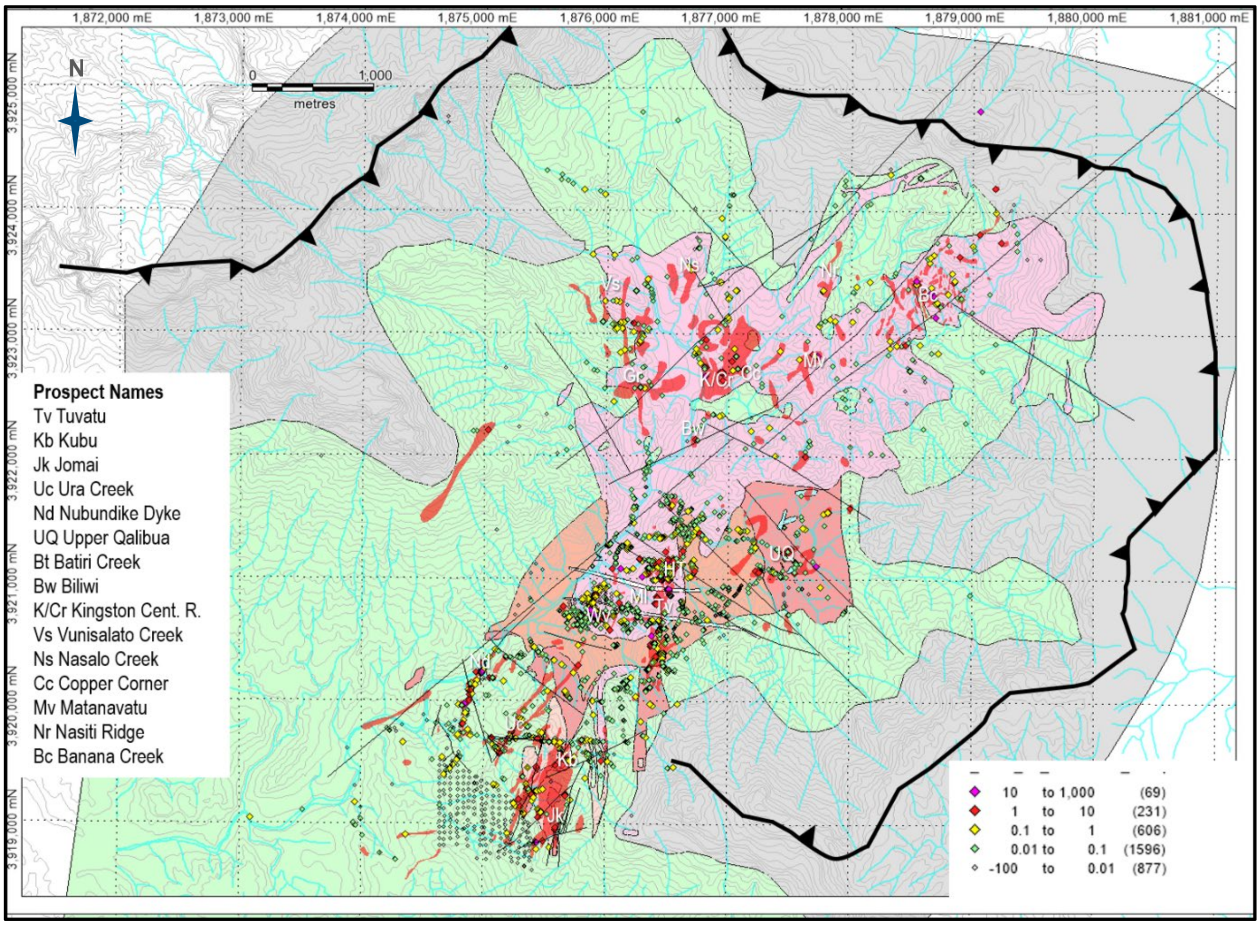
Recognising the uncertainty presented in selective rock-chip sampling, Lion One commenced a chip-channel and map-sampling program. As of 2017, this was fully migrated, and as such, all regional surface-rock sampling is now done with a recorded width and mapping of structures. These data are recorded in an equivalent database to the drilling data, with each channel having a "collar" recorded being the start of the channel, the survey being the dip and azimuth of the channel, and a sample table containing "from" and "to" as well as multi-element assay results.

Since 2016, Lion One has used ALS laboratories in Townsville for Au-AA25 (fire assay) with a 0.02 ppm (20 ppb) detection limit, and ME61S for 33 element aqua regia digest analysis. Since mid-2019, samples are being prioritised through Lion One's own laboratory, with assays greater than 0.5 ppm Au also being sent to ALS in Townsville for further check assay. Samples with significant base metal assays are also forwarded for check assays.

Certified reference material (CRM) / standards are inserted into the sample stream at a ratio of 1:20 to check laboratory performance.

The sampling locations include creek-channel samples, benching, and underground sampling in the existing adit. Creek-channel samples are from natural exposure in the various creeks. Benching involves excavation of a 1 to 5 m bench along a hillside to expose weathered rock. In the case of benching, the rock is extremely weathered and friable, and is subject to bias with either oxidized leaching of metal or enrichment.

The channel sampling database contains 15,643 samples (Figure 9-13). All data are located with either a handheld global positioning system (GPS) (accuracy +-5 m) or differential GPS by a qualified surveyor (+-0.01 m). Figure 9-14 shows a typical channel sampling survey, with white lines indicating the structure, and red paint marks showing the sampling marks. Similarly in creek-channel sampling, the process is repeated (Figure 9-15).



Source: Lion One (2020)

Figure 9-12: Rock sampling locations by gold on background geology

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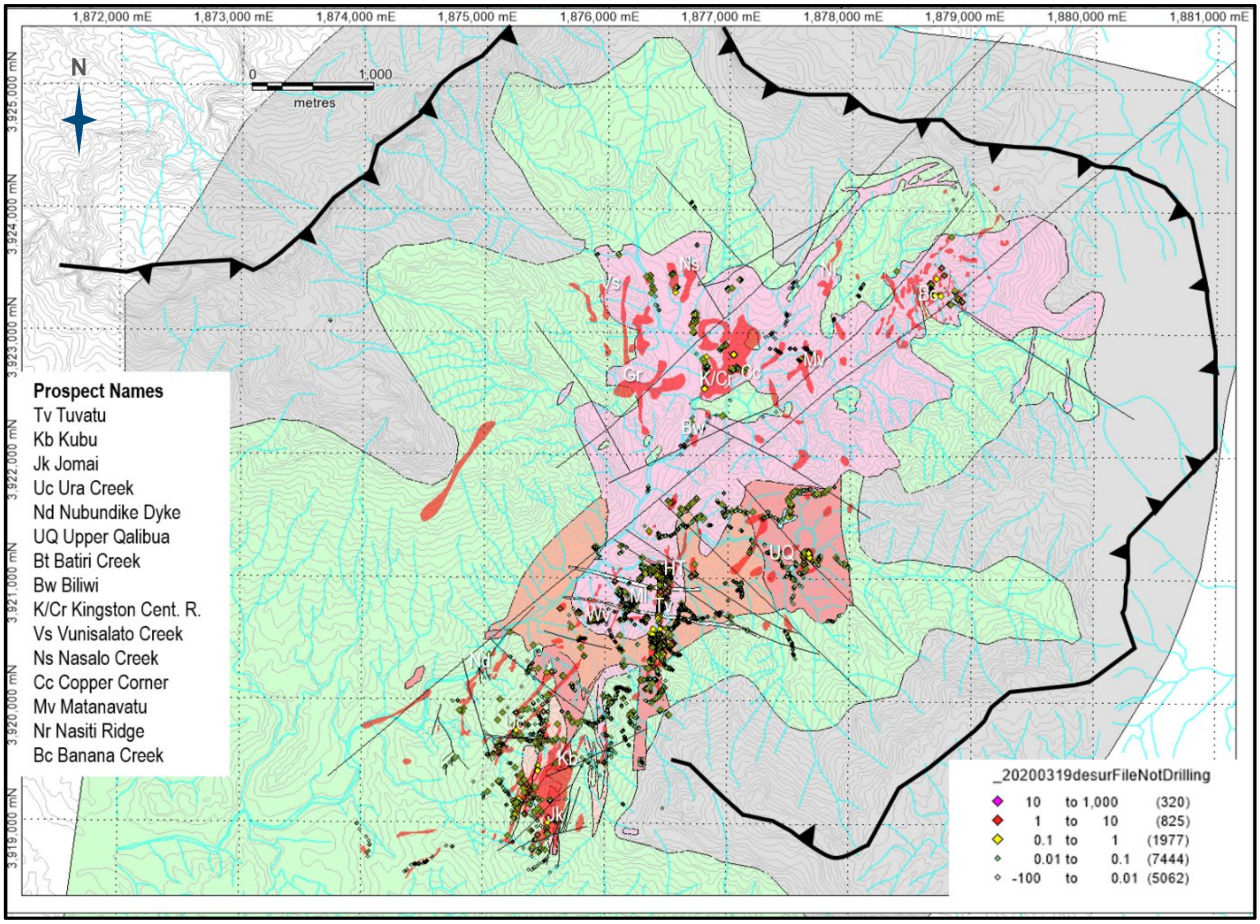


Source: Lion One (2020)

Figure 9-13: Chip channel sampling from benching and creek sampling on background geology

Source: Lion One (2020)

Figure 9-14: Example of chip channel sampling in a bench. The red marks indicate sample from and to locations, presented orthogonal to a structure (marked in white paint).

Source: Lion One (2019)

Figure 9-15: Chip channel sampling from a creek exposure (prior to sampling). The red marks represent the sampling intervals.

9.5.2.3 Database and Data Integration

All samples collected on site are done so by a qualified and experienced team of geologists and field assistants. Samples are secured in cable-tied polyweave bags before being transported to Lion One's Nadi office for staging. If samples are exported to an overseas laboratory, e.g., ALS in Townsville, they are inspected by a government official for the granting of a requisite export permit before being airfreighted. If samples are to be prepared for analysis at Lion One's own laboratory (since October 2019), they are checked by laboratory staff before being processed.

Assay results are sent to database consultant rOREdata Pty Ltd in Perth, Western Australia, where they are recorded in an aQuire SQL Server database and checked for sample consistency, quality assurance (QA) / quality control (QC), and duplicates.

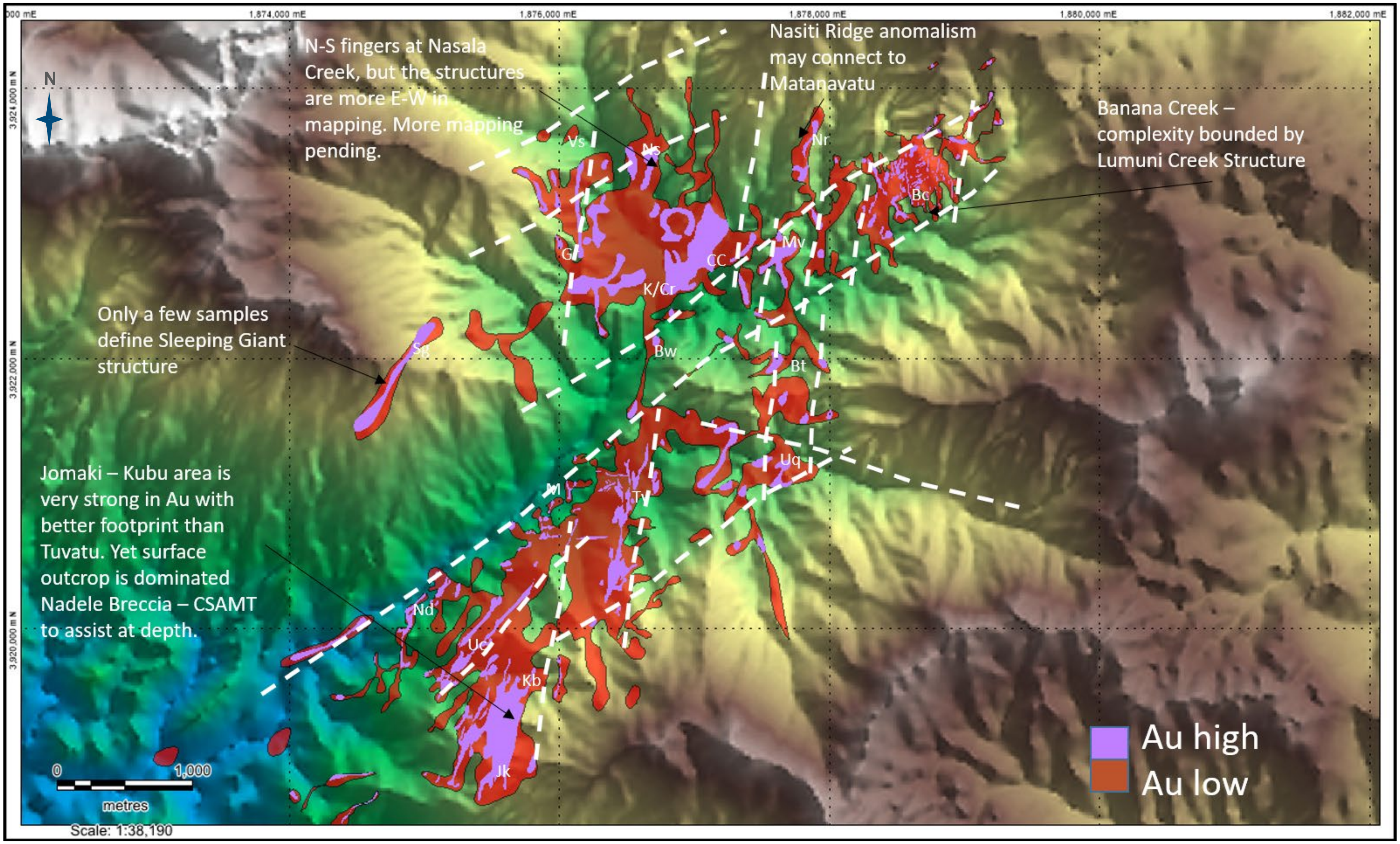
Exports from the database are routinely sent to Lion One's GIS officers, database administrators, and geoscientific staff for integration into digital data systems.

Use of Multiple Data Techniques in Building Maps of Mineralizing Cells for Prospectivity Analysis

Due to the variability in sample and assay techniques, along with topographic, tropical weathering, and stream dispersion, it is not possible to build an objective picture of the mineralized footprints for regional targeting. As an alternative, a review of all surface sampling data of various mediums has been integrated with hand contouring of either low-level, mid-level, or high-level results. In the case of gold, for example, low level was considered where clusters of surface sampling in areas that showed grades >0.5 ppm Au in rocks and channels, and >0.1 ppm in soil samples. Similarly, the mid-level gold contours encircle clusters of samples where Au is >1 ppm in rocks and channels, and >0.05 ppm in soils. Whilst absolute values were used in this way, null values or sub-grade values were largely ignored, and as such, this is to be considered as a spatial analysis of potential prospectivity and anomalism rather than an absolute enclosure of mineralized grade. Figure 9-16 to Figure 9-20 illustrate this regional contouring of multiple-elements. Table 9-2 summarizes the results of this work.

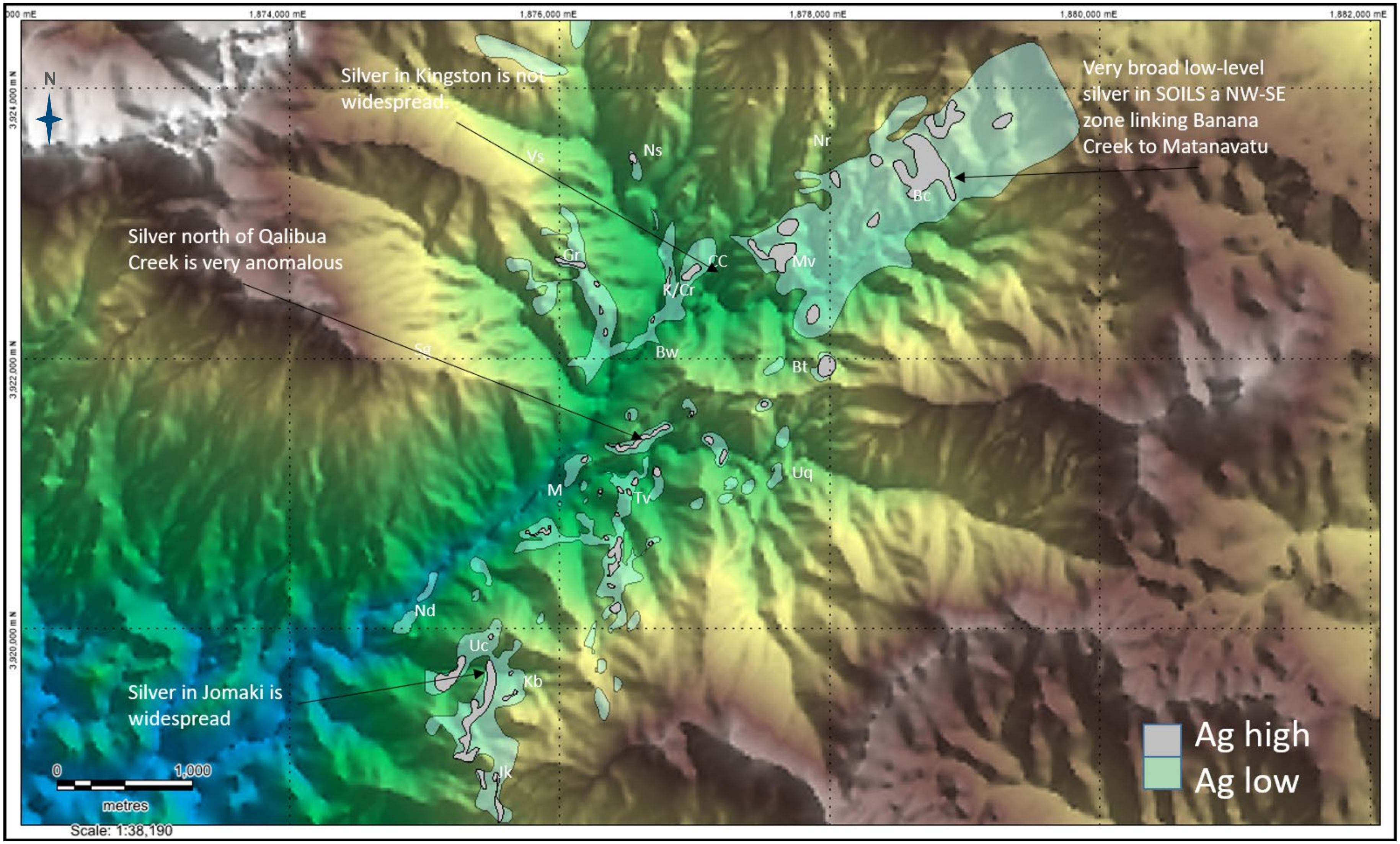
Table 9-2: Summary of multi-element pathfinders over various prospects

Prospect	Au	Cu	Ag	As	Te	Comments
Tuvatu	High	Low	Mod	Mod	Mod	Tellurium appears controlled by east–west structure.
Banana Creek	High	Low	High++	High++	Mod	Arsenic is far more extensive than the gold footprint. Continues to north and northeast.
Nasiti Ridge	High	Mod	Low	Mod	Low	Not much sampling; could be a continuation of Matanavatu with gold > copper increasing to the north.
Matanavatu	High	High++	High	Low	Low	Has affiliations with Banana Creek, Kingston. and Nasiti Ridge. A cross-roads in elements.
Kingston / Central Ridges / Copper Corner	High	High++	Low	Low	High	Tellurium driven by north–south structures.
Kingston/Biliwi	High	High++	Mod	Low	High	Tellurium driven by north–south structures.
Nasala Creek	Mod	Low	Low	High	Mod	Only sporadic sampling.
Vunisalato Creek	Mod	Low	Low	High	Low	Only sporadic sampling.
Upper Qalibua	Mod	High	Low	Low	Mod	Copper and low-level gold to follow up.
Batiri Creek	Mod	Low	Low	Low	Low	Only sporadic sampling.
Ura Creek	High++	Low	High	High	Mod	Very similar to Tuvatu, but in Nadele and no appreciable gold in shallow drill results.
Jomaki-Kubu	High	Low	Low	Low	Mod	Very similar to Tuvatu, but in Nadele and no appreciable gold in shallow results.
Golden Ridge	Low	High	Mod	High	Low	Copper and low-level gold to follow up.
Sleeping Giant	Low	Low	Mod	High	Low	Not much sampling.



Source: Lion One (2020)

Figure 9-16: Regional analysis of gold mineralizing cells from a combination of surface sampling techniques



Source: Lion One (2020)

Figure 9-17: Regional analysis of silver mineralizing cells from a combination of surface sampling techniques

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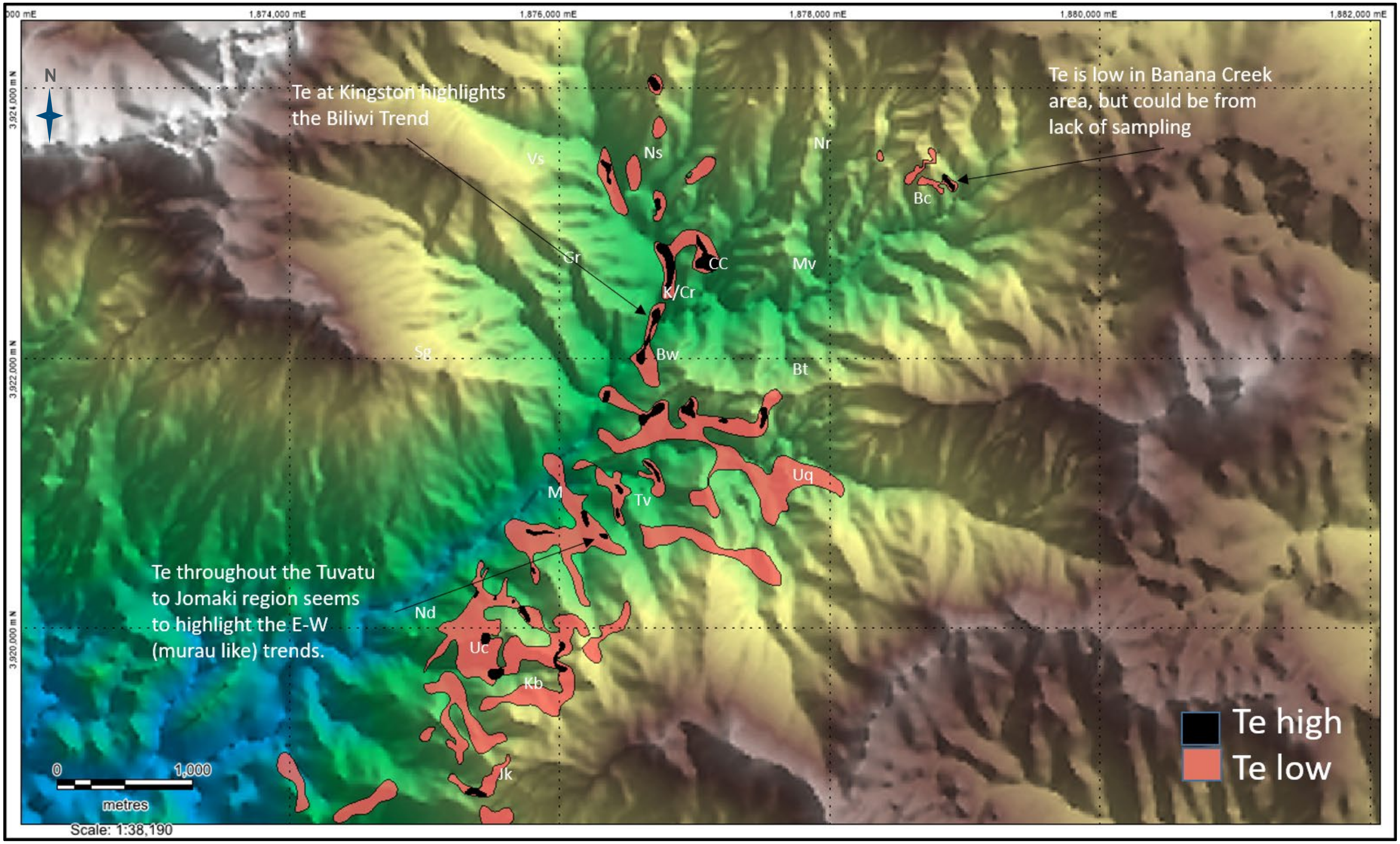

Source: Lion One (2020)

Figure 9-18: Regional analysis of tellurium mineralizing cells from a combination of surface sampling techniques

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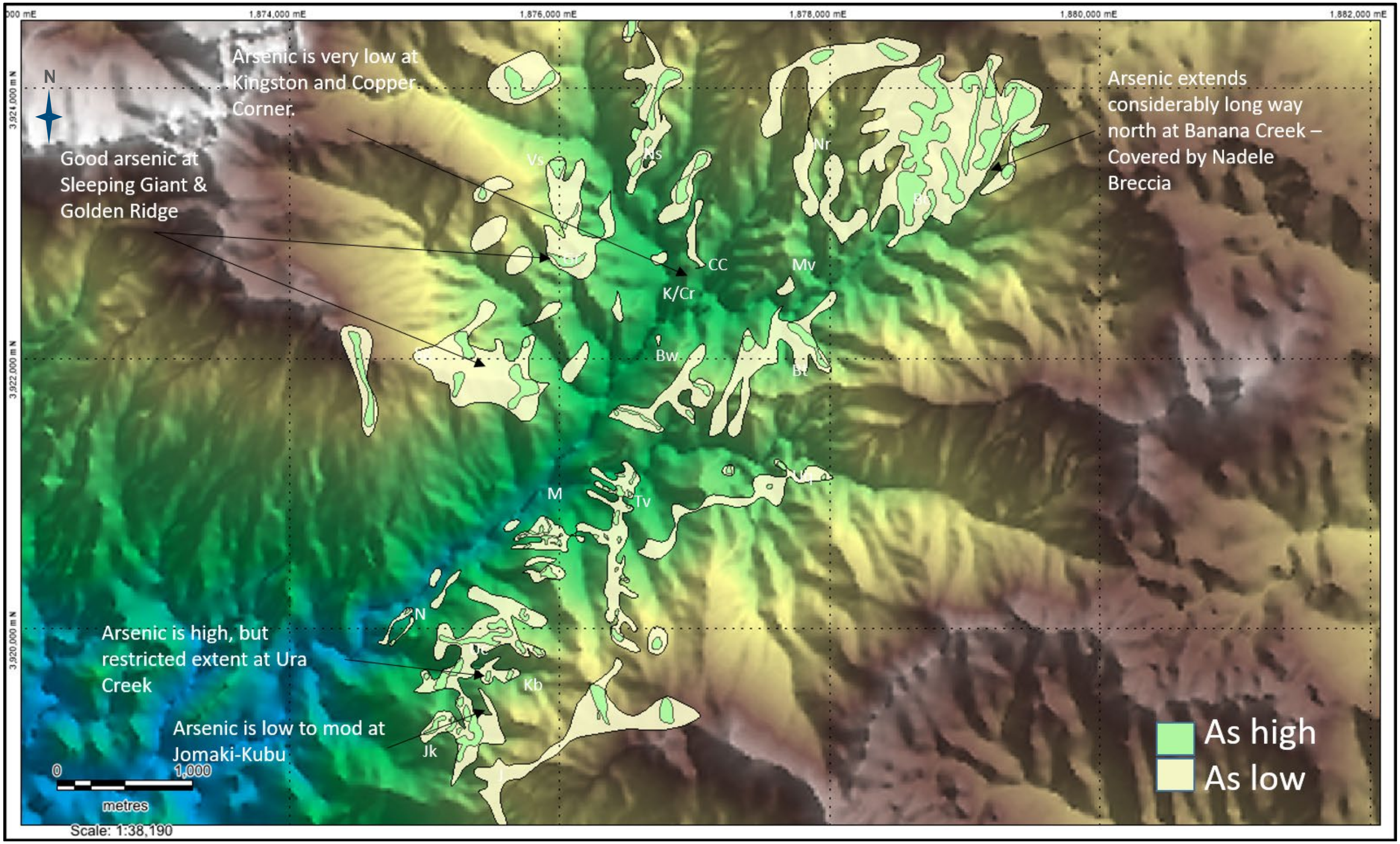
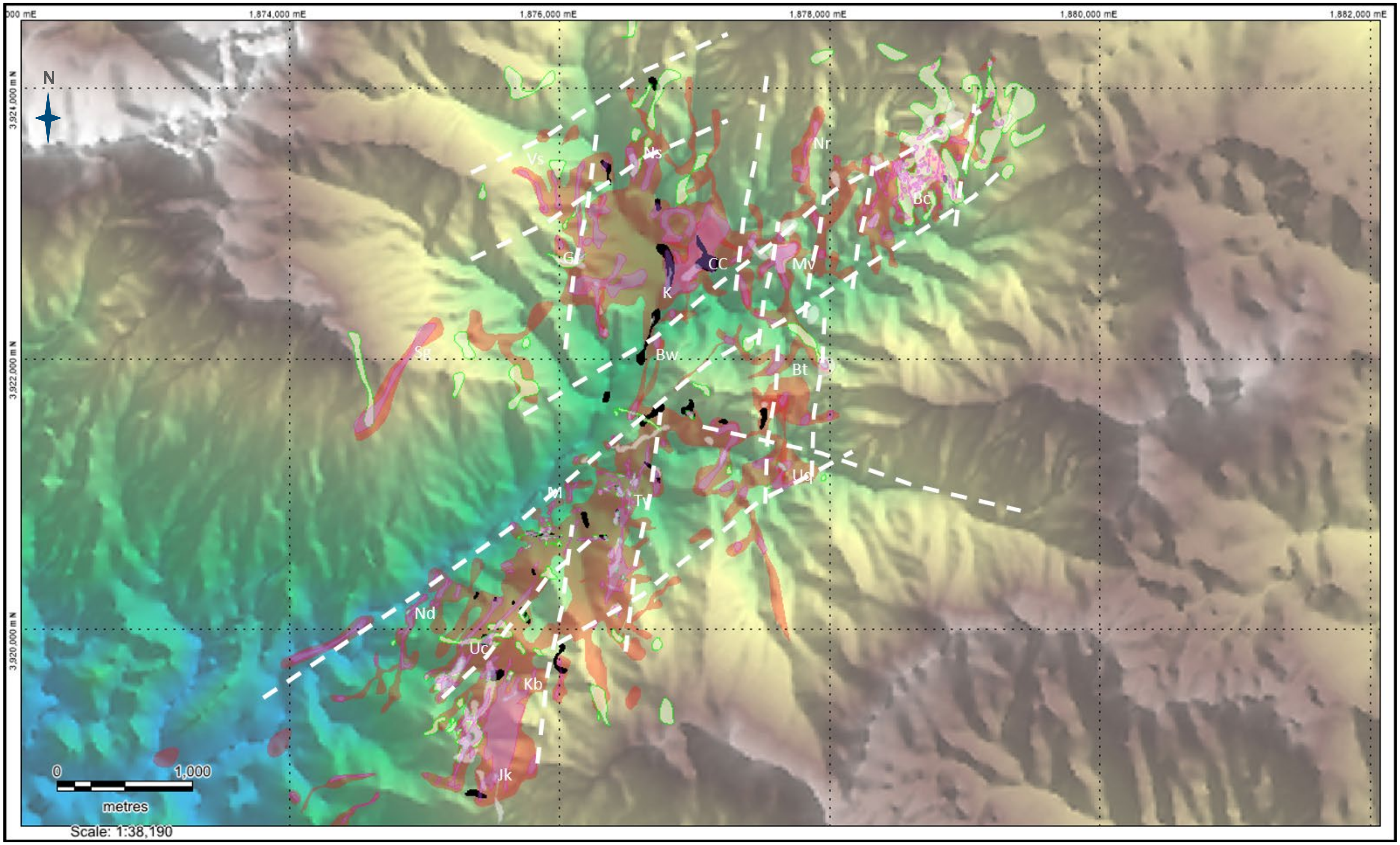


Source: Lion One (2020)

Figure 9-19: Regional analysis of arsenic mineralizing cells from a combination of surface sampling techniques



Source: Lion One (2020)

Figure 9-20: Regional analysis of Au-Ag-As-Te mineralizing cells from a combination of surface sampling techniques

9.5.3 Clay Bulk Leach Extractable Gold Stream Sediment Sampling

During 2019, Lion One instigated a project-wide clay bulk leach extractable gold (cBLEG) stream sediment program.

cBLEG is a technique developed by North American explorers, including pioneering work by Newmont, and is particularly useful in prioritising regional target areas. This technique was introduced to Lion One by Dr. Quinton Hennigh, an advisor to Lion One.

In short, cBLEG is a standardised technique that ensures the same fraction of material is taken from each stream catchment. This technique differs considerably from other stream-sediment techniques, including high-energy stream sampling, as the fraction is the finest possible material (clay size). Typically, stream sediment sampling is qualitative and subject to nugget effect, whereby a coarse gold grain caught in a high-energy environment, for example, can lead to sampling bias. However, cBLEG, through standardisation of sample collection to the clay fraction, removes the nugget effect bias, and as such, cBLEG catchment analysis can be considered as **quantitative**, whereby direct comparisons between catchments can be made.

This technique, cBLEG, is only useful for detecting catchments that contain mineral lodes that break the surface and are intersected by creeks and streams. However, as discussed below, Lion One has also conducted multi-element analysis whereby pathfinders are used to also ensure that other potentially concealed gold deposits are explored. In addition, cBLEG is only effective in the presence of weathered clay minerals. Fortunately at Tuvatu, the monzonite host is a feldspar-rich rock that readily weathers to clays.

9.5.3.1 Description of Technique

A clay is defined as particles less than 2 microns and form from chemical weathering of aluminium rich rocks to phyllosilicate minerals such as smectite, illite, and kaolinite. Due to their high-surface area, clay minerals become statically charged, and therefore, can attach very fine metal ions.

Compared to conventional stream sediment sampling (e.g., gravel or sand), the advantages of cBLEG sampling are:

1. Standardised method of the same fraction from every catchment.
2. Fine clay material only reducing the risk of “nugget effect” from larger gold particles.
3. Static leach of gold to ensure only fine gold particles are digested (thus further reducing the risk of nugget effect and outliers.

This technique has been used effectively in stream geochemistry throughout the world. The Tuvatu Regional Project is considered a particularly good location for the application of this method due to:

1. Feldspar-rich monzonites breaking down to clays.
2. High rates of weathering (10 to 20 m of weathered material on ridges).
3. Plentiful streams flowing year round.
4. Incised steep gullies with a high sediment load generating clay traps.
5. Seasonal variations in stream flows causing a dispersion of clay downstream from the source.

9.5.3.2 In Field Collection

The cBLEG technique involves multiple steps in the field materially including:

1. A stream site is identified that is upstream from a confluence. This allows for analysis of the whole catchment above the sample.
2. In a clean 20 L bucket, approximately 20 kg of fine sediment is collected, sieved to -2 mm, and taken from at least four sites along a 20 to 40 m stretch of creek.

9.5.3.3 In Camp Sample Processing

Once the sample has been received from the field, it undergoes further processing at the exploration camp:

1. Samples are gradually sieved to <0.4 mm.
2. Fresh clean water is added to the sample.
3. Samples are mixed to suspend fine sediment (clay) in the water column.
4. The suspended clay is poured off into a new clean bucket.
5. The sample is transferred to a micropore (breathable) sample bag and dried in an oven.

Final samples average 1 kg of dried clay. The samples are then transported to ALS Laboratories in Vancouver for analysis using Au-CN44a (static leach with cyanide) with multi-elements analysed with ME-MS41L (aqua regia ICP with multi-element analysis).

9.5.3.4 Visualization of Results and Catchment Analysis

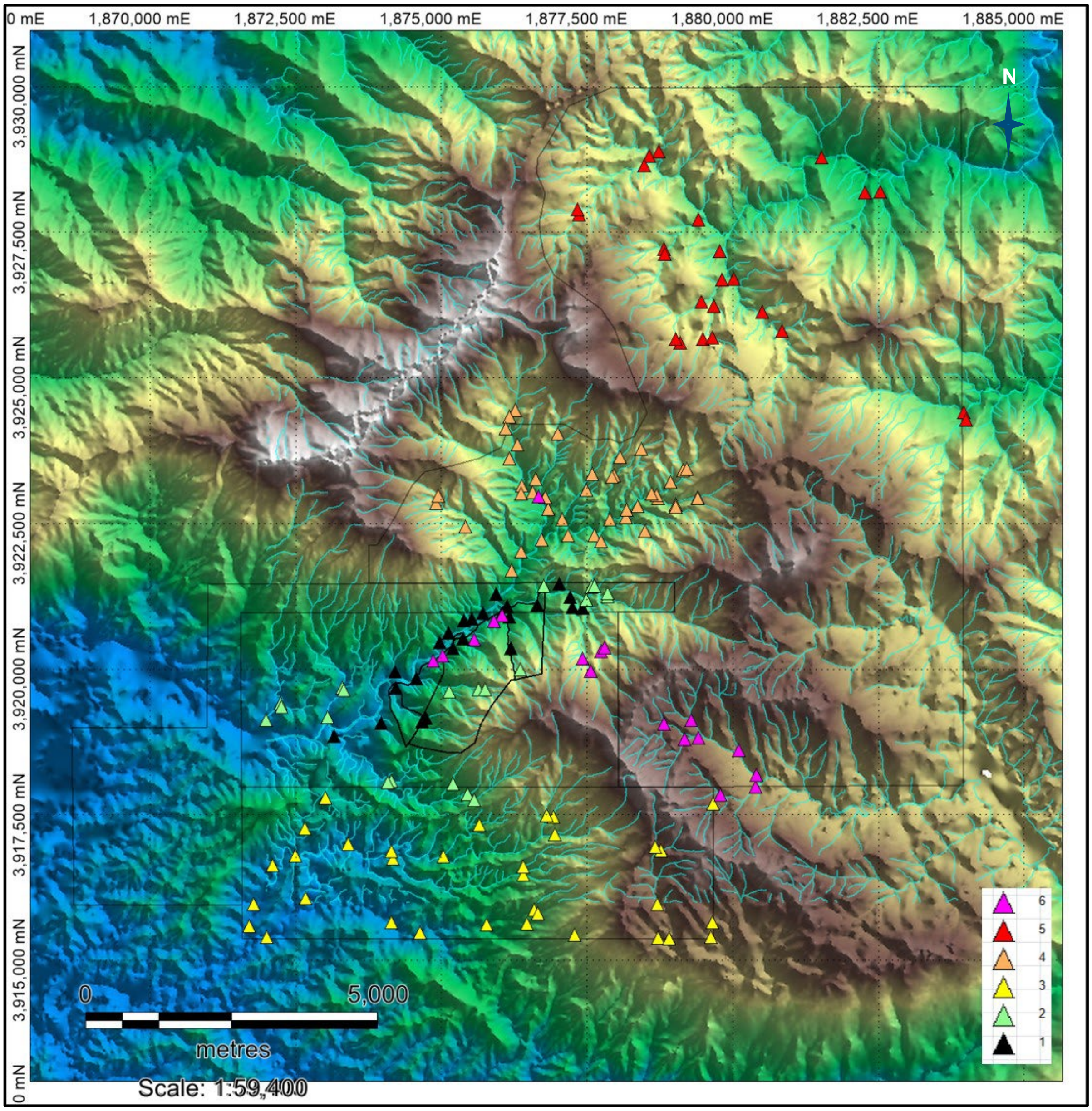
Fathom Geophysics was used to automatically generate a ridge and valley vectorization of the 20 m digital elevation model. This generates nested polygons based on stream order (first order rivers such as the Sabeto River to fourth order hanging valleys and small steep creeks above tributaries). Due to the resolution of the digital elevation model, there are some inaccuracies in this process. As such, after each batch of results, the catchment is manually digitized using the Fathom Geophysics analysis as a guide.

9.5.3.5 2019 cBLEG Program

From May to December 2019, Lion One collected a total of 163 cBLEG samples, which were submitted for analysis in 6 batches (Table 9-3).

Table 9-3: cBLEG samples 2019, by batch (refer also to Figure 9-21)

cBLEG Batch	Sample Submission Number	Number of Samples
1	375	24
2	378	22
3	385	33
4	390	42
5	395	24
6	399	18
Total	**-**	**163**



Source: Lion One (2020)

Figure 9-21: 2019 cBLEG sample locations coloured by batch number

9.5.3.6 Discussion of Results and Targeting

As a first pass, all data is presented with absolute values. However, each cBLEG sample represents a catchment size of variable area. If a high-grade vein is shedding gold into a catchment, it gets diluted by other material. As it stands, all the high-grade cBLEG samples (>20 ppb Au) are from relatively small catchments. It is possible that a high-grade zone still exists in larger catchments, but is diluted by more sediment load. In order to normalise the data to catchment area, the following procedure was implemented:

- The Tuvatu catchment has an area of **0.506 km^2** and a cBLEG result of 47 ppb Au; this sets a benchmark based on a known system.
- An "Area Factor" is calculated as the area of a catchment divided by **0.506 km^2**. Hence the area factor for the Tuvatu catchment is **1**, but the area factor for a catchment of twice the size is **2**.
- Each assay result is multiplied by the area factor. As a result, consider the following:
  - If a catchment is 0.5 km^2 and has a gold grade of 40 ppb, the normalised value will be 40.
    - (0.5/0.5)*40 = 40.
  - If a catchment is 1 km^2 and also has a gold grade of 20 ppb, the normalised value will also be 40.
    - (1/0.5)*20=40

9.5.3.7 Usefulness of Pathfinder Elements

The primary purpose of this work was to test for cBLEG in the clay fraction; however, conventional multi-element ICP-MS analysis was also carried out on the samples. The following elements have been analysed for these reasons:

a. **Silver** – There is a positive correlation between silver and gold in the Tuvatu system. Furthermore, silver may represent a higher/more distal part of the mineral system, and hence, catchments where silver is high, yet the gold is low, still warrant further investigation for a buried system.

b. **Arsenic** – Arsenic values associated with high-grade at Tuvatu range from 50 ppm to >1%. There is a zone variation and only a weak correlation between arsenic and gold at Tuvatu. However, there is a stronger correlation exhibited in the cBLEG data.

c. **Copper** – There is a weak positive correlation between gold and copper at Tuvatu; however, further into the Caldera, the Kingston, Biliwi, and Copper Corner copper occurrences are closely associated with high-grade gold.

d. **Lead** – There is a positive correlation between gold and lead at Tuvatu, though this is zone dependent with the HT corridor exhibiting the strongest response with coarse galena.

e. **Tellurium** – The presence of telluride at Tuvatu has been noted by several researchers. The assaying of telluride versus gold is biased due to variable detection limits applied. Nevertheless, there is a positive correlation, which is strongly exhibited in the cBLEG work.

f. **Vanadium** – Vanadium is found in the mica mineral roscoelite ($K(V^{3+},Al,Mg)_2AlSi_3O_{10}(OH)_2$, which is often closely associated with very high-grade gold at Tuvatu. However, vanadium can also be found in other minerals, including magnetite, and is common in intermediate to mafic rocks. As such, vanadium is of limited use as a pathfinder, but is nevertheless routinely reviewed in cBLEG data.

g. **Zinc** – Sphalerite is noted throughout the Tuvatu mineralization system but is particularly prevalent as coarse grained crystals in the HT zone. Due to its variability in zones, it is only partially useful as a pathfinder at Tuvatu. It is nevertheless indicative of hydrothermal activity, which may, however, prove to be distally zoned to alkaline gold systems.

The pathfinder elements above are listed as having an association with wider mineral system and alteration at the main Tuvatu deposit. On a localized scale, such as individual veins and assay results, a high-grade gold assay may not correlate directly with a high-grade pathfinder assay.

9.5.3.8 Description of Patterns and Target Catchments

The following images, Figure 9-22 to Figure 9-30, demonstrate the various patterns and catchments of the cBLEG results from the 2019 program at Tuvatu. A discussion on the effectiveness is presented beneath each figure.

The 2019 cBLEG program, in six batches, has been extremely effective in discerning the Tuvatu mineralized catchment, and hence, we can consider its effectiveness elsewhere. At Banana Creek, for example, it shows a small but very high-grade catchment that also lights up in pathfinders. The best pathfinders seem to be gold, silver, arsenic, and to some degree, tellurium.

The cBLEG generally shows that the area south of Tuvatu (e.g., the Jomaki Ridge areas) is perhaps less prospective (in the near surface at least) compared to other areas.

This work has confirmed the prospectivity of the Upper Qalibua and Banana Creek areas. New target areas have been defined with strong mineralized catchments at Naisalo Creek and Vunisalato Creek on the western part of Navilawa Caldera, as well as strong pathfinder responses near Matanavatu and Nasiti Ridge. These areas are will be mapped and sampled. In addition, there is a highly anomalous catchment to the east of the Namotomoto divide.

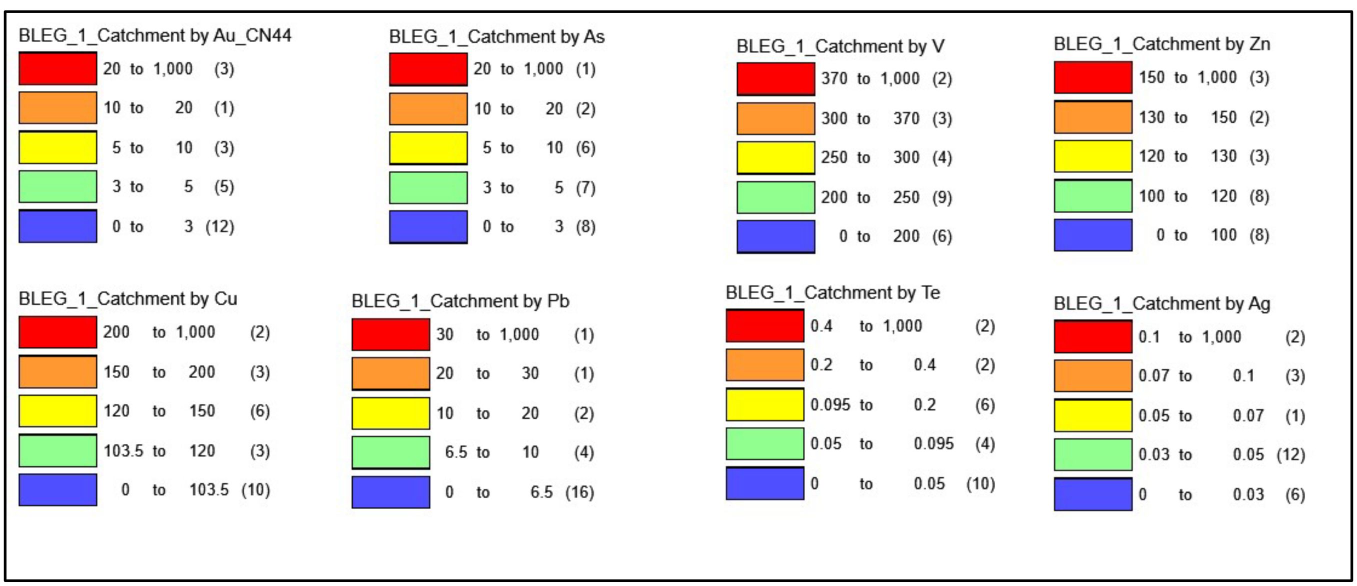


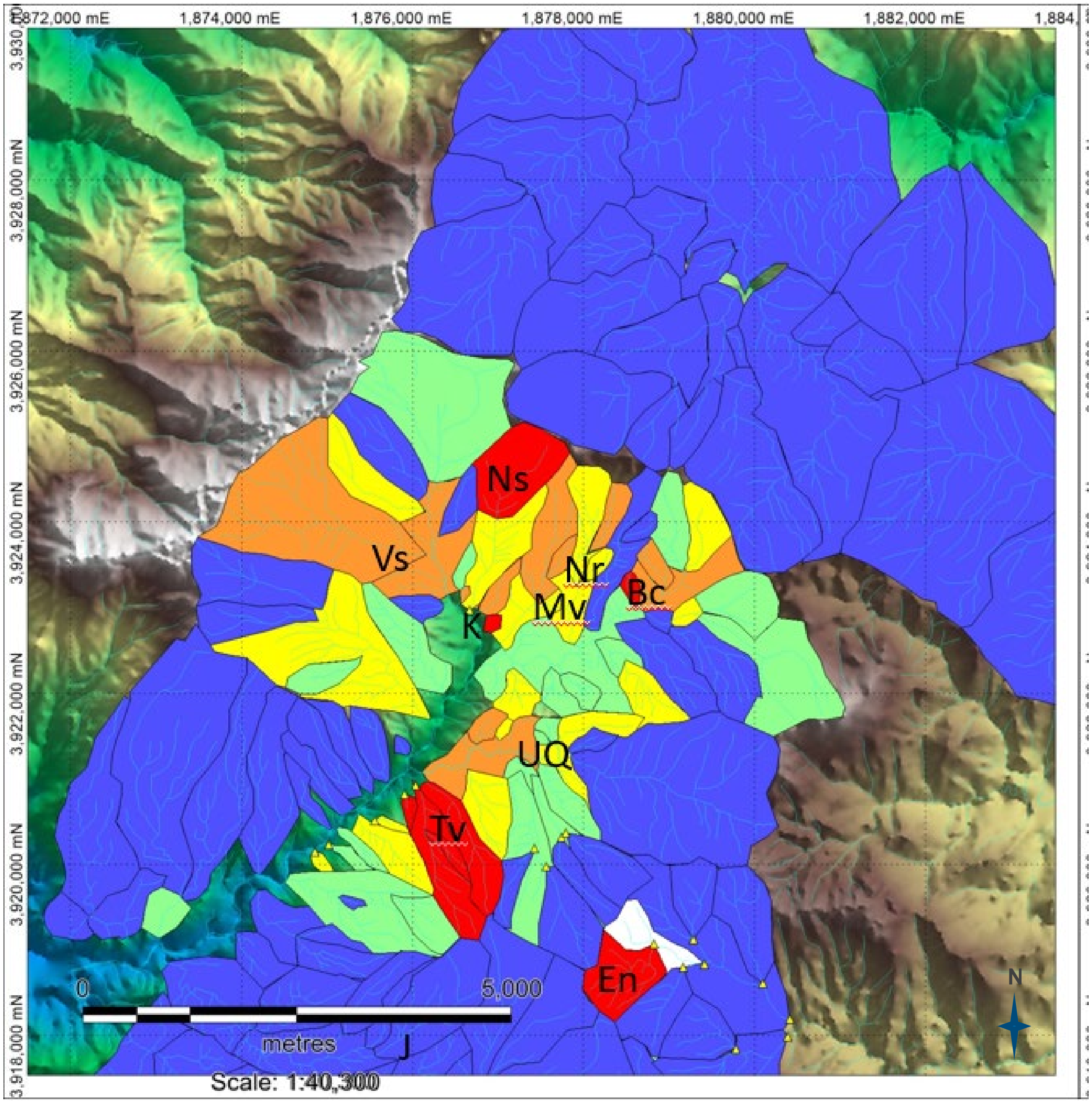
Figure 9-22: Legends for cBLEG multi-element images. Au = PPB, remaining elements in PPM.

The following prospect area catchment abbreviations are used in Figure 9-23 to Figure 9-30:

- Bc – Banana Creek

- En – East Namotomoto
- J – Jomaki/Davui/Ura, etc.
- K – Kingston
- Nr – Nasiti Ridge
- Ns– Naisala Creek
- Tv – Tuvatu
- UQ – Upper Qalibua
- Vs – Vunisalata Creek

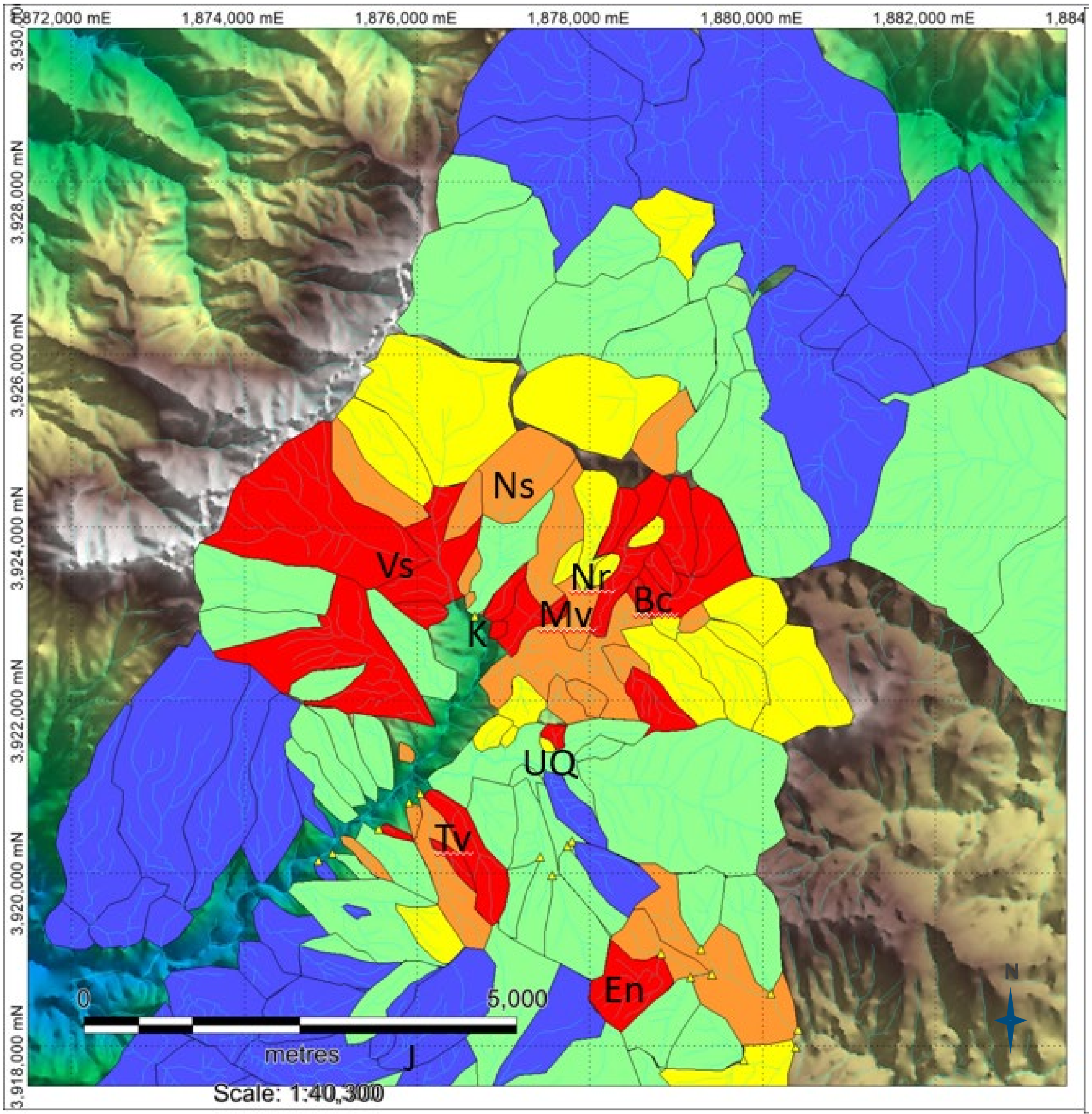
The Tuvatu catchment area, as a benchmark, is very strong in gold, with other high-gold cBLEG catchments located at Kingston, Naisalo Creek, and Banana Creek. The Vunisalato Creek catchment is registered as 19.9 ppb AuCN44a, and is hence only just below the cut-off of high-grade classification. The main Navilawa Caldera displays multiple clustered catchments indicative of potentially mineralized systems that cross several catchments.



Source: Lion One (2019)

Figure 9-23: Au CN44a, gold in cBLEG stream survey data

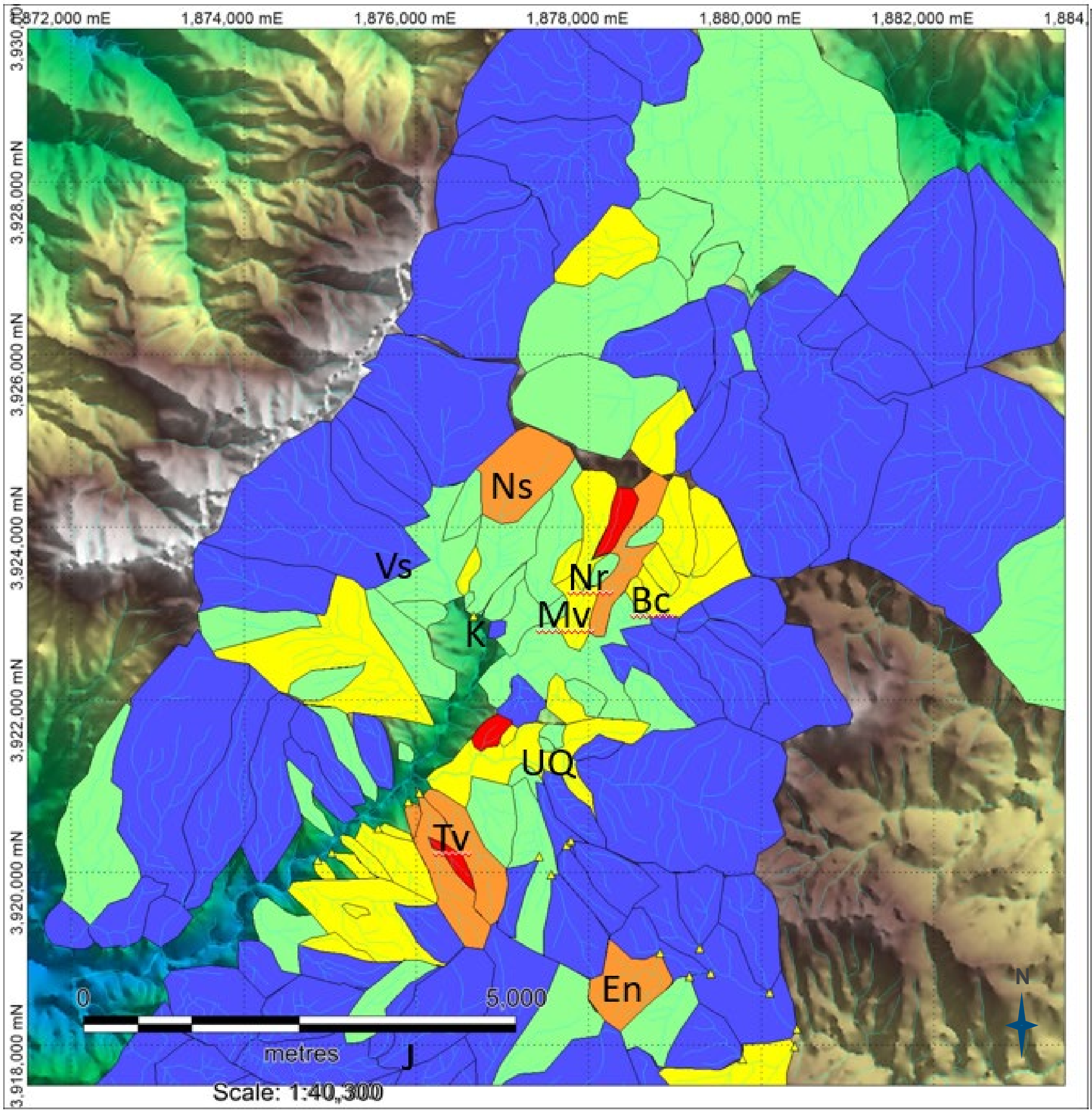
Silver shows a good correlation with the Tuvatu benchmark. Signatures within the Caldera also show good responses, particularly as a pathfinder at Vunisalato, Banana Creek, and Kingston to Matanavatu areas. Overall there are low, yet anomalous areas outside of the Caldera rim.



Source: Lion One (2019)

Figure 9-24: Silver in cBLEG stream survey samples

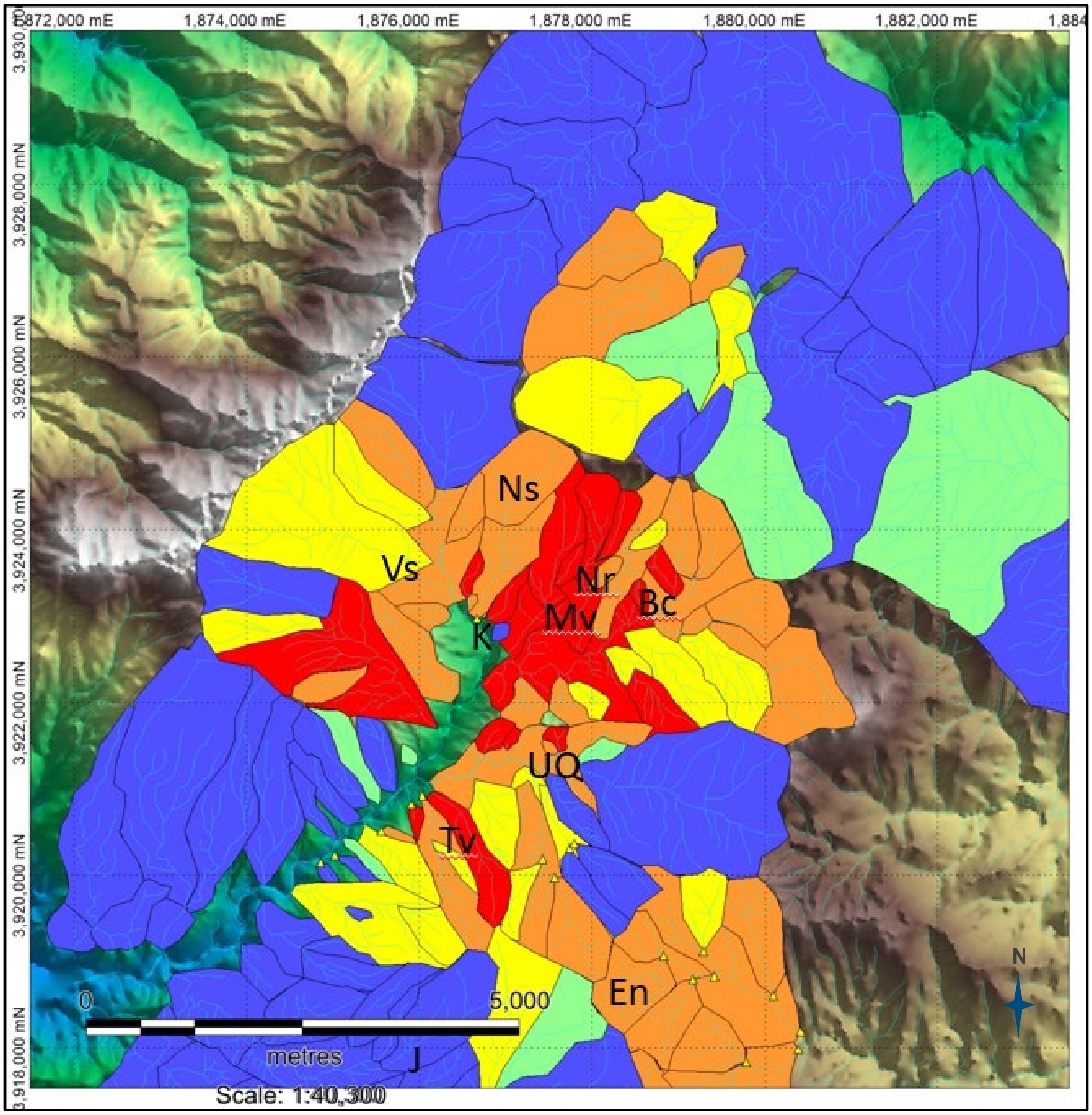
The arsenic correlation is good at Tuvatu. Overall, this appears to be highlighting a northeast–southwest striking zone, including over the north rim of the Navilawa Caldera. This is, perhaps, related to a deep underlying structure upon which the Tuvatu and Vatakoula Prospects lie upon.



Source: Lion One (2019)

Figure 9-25: Arsenic in cBLEG stream survey samples

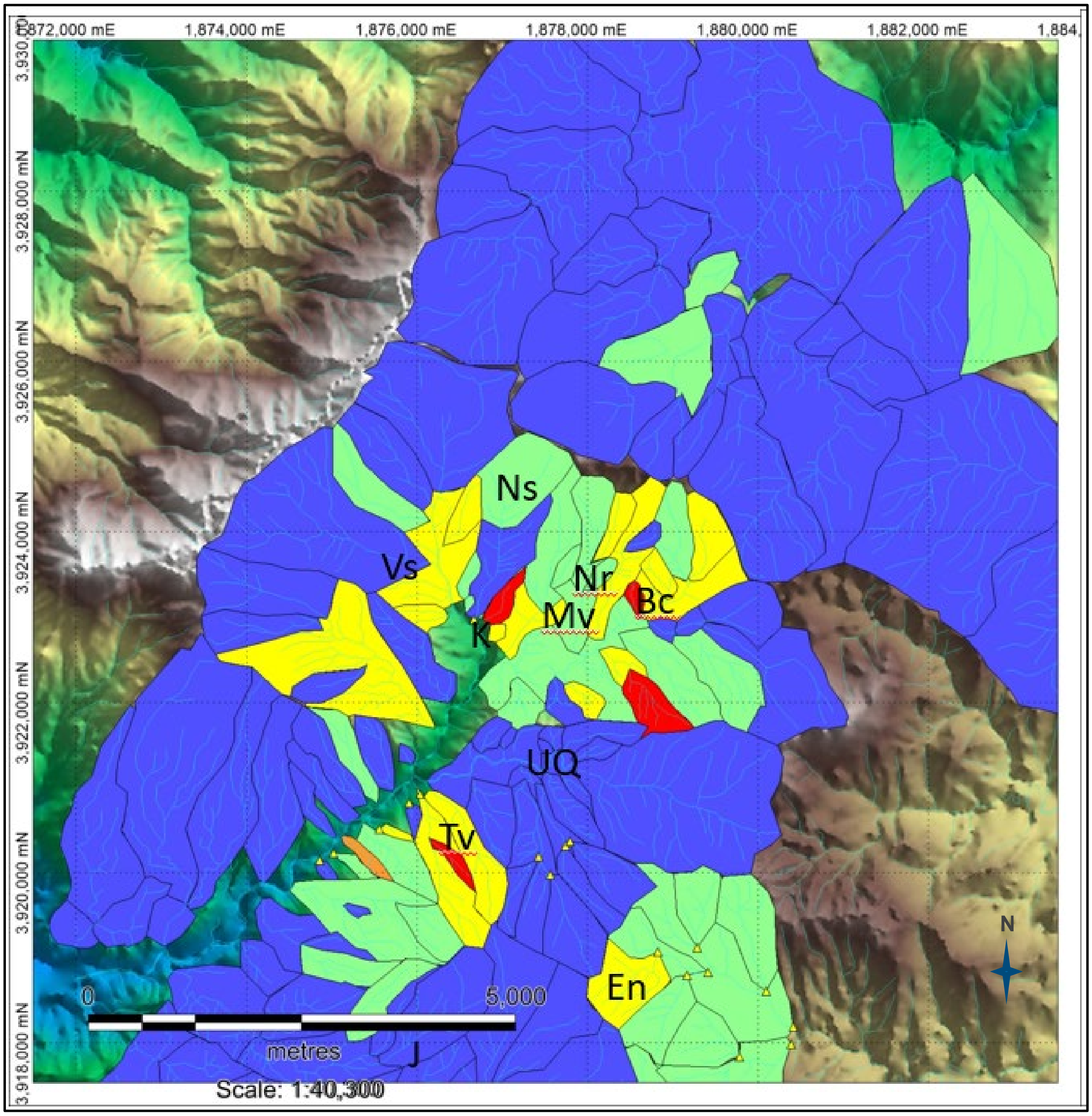
The most consistent copper catchments lies in the central Caldera, including between Kingston and Matanavatu, and Nasiti Ridge. A single catchment south of Vunisalato should also be investigated.



Source: Lion One (2019)

Figure 9-26: Copper in cBLEG stream survey samples

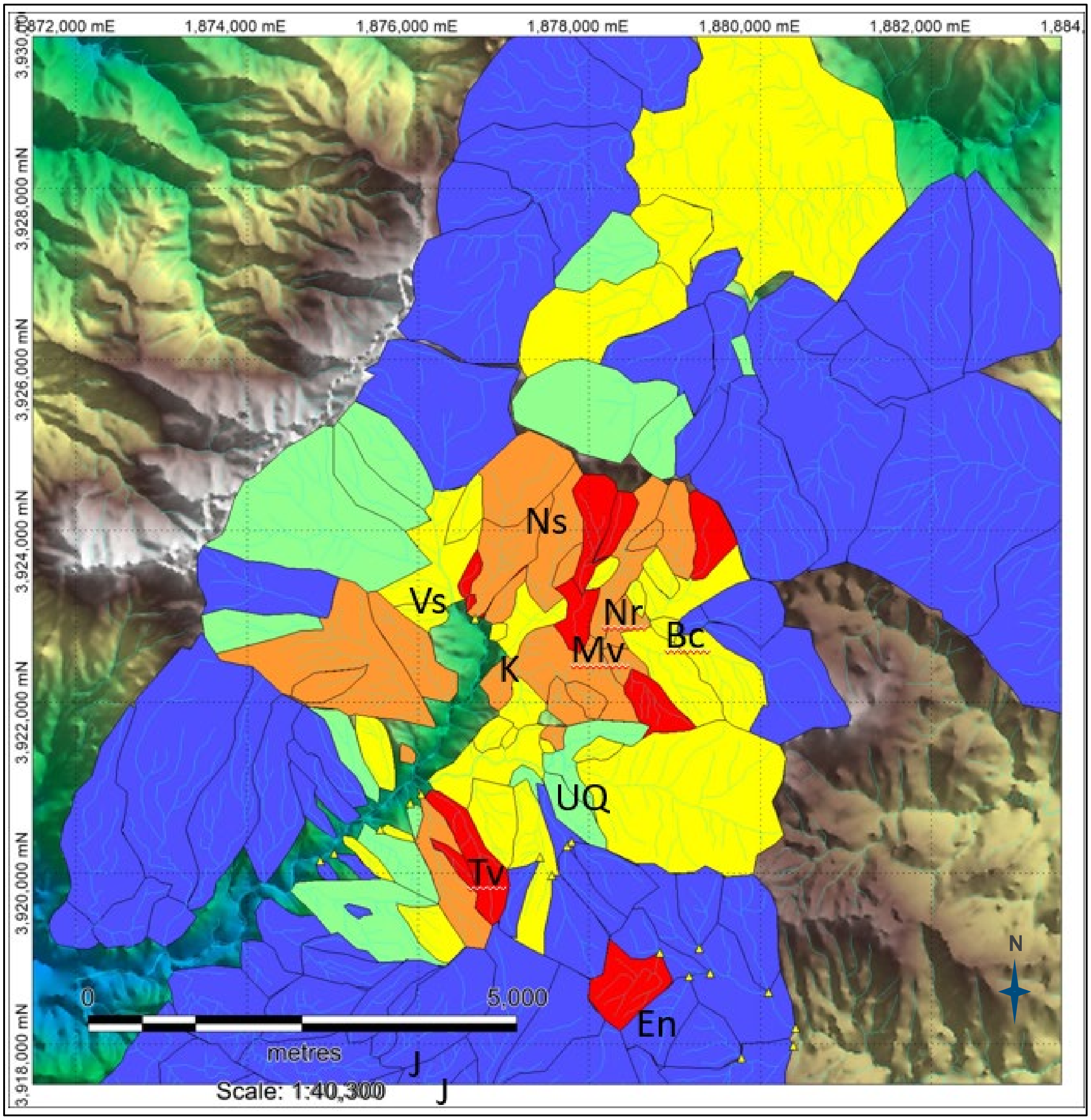
Lead is very localised to the central Caldera, the Tuvatu area, and also the East Namotomoto area. There appears to be a reasonable correlation with known mineralization at Kingston and Banana Creek, yet the Upper Qalibua catchments are devoid of lead association.



Source: Lion One (2019)

Figure 9-27: Lead in cBLEG stream survey samples

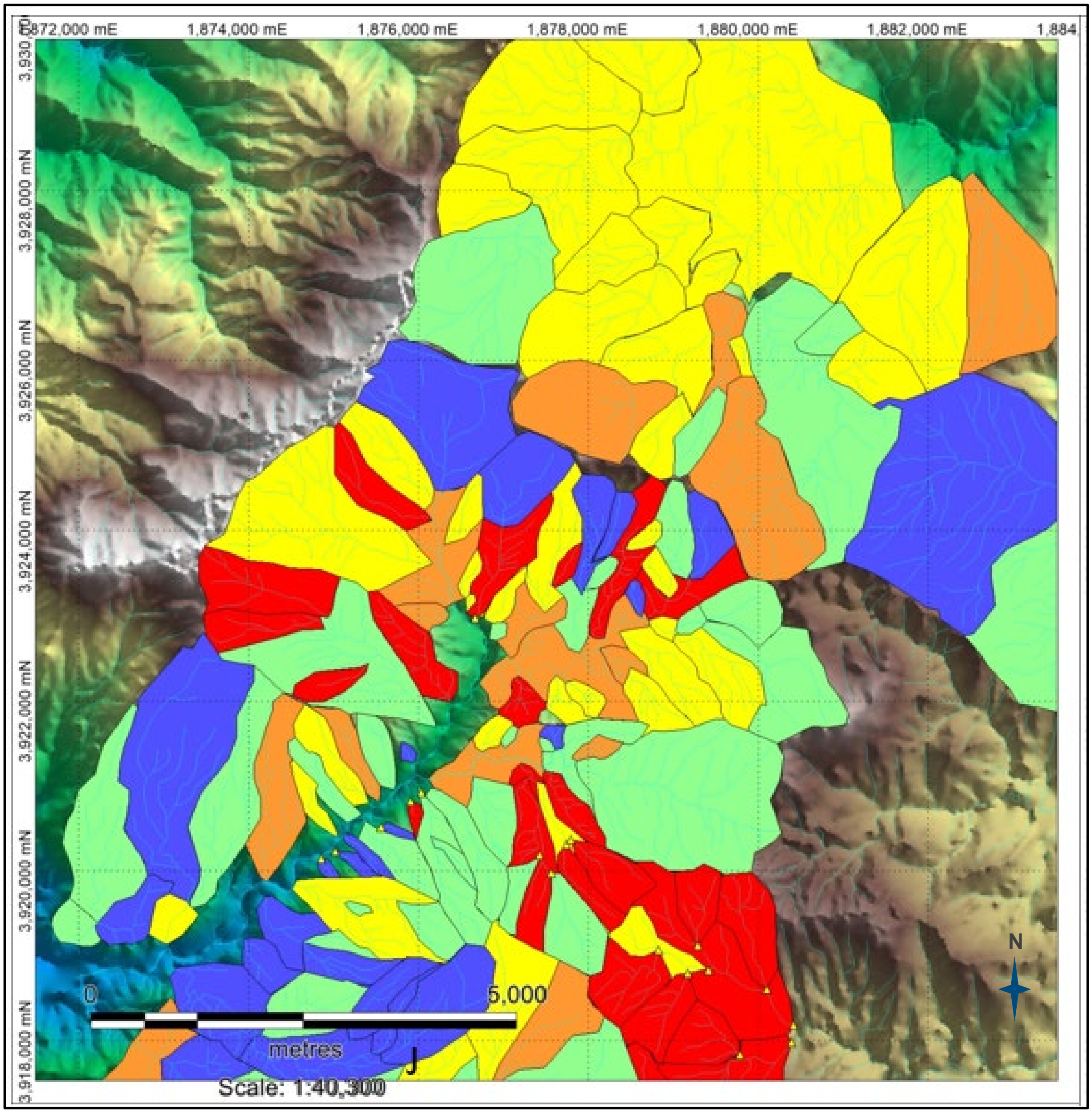
The tellurium anomalism in the cBLEG data is evident around the Tuvatu deposit. In addition, there is a large cluster of high-tellurium catchments central to the Caldera and include Naisalo Creek, and particularly to the west of the Nasiti Ridge and Matanavatu area. Given the strong correlation between gold and tellurium, these areas are prioritised for follow-up work.



Source: Lion One (2019)

Figure 9-28: Tellurium in cBLEG stream survey samples

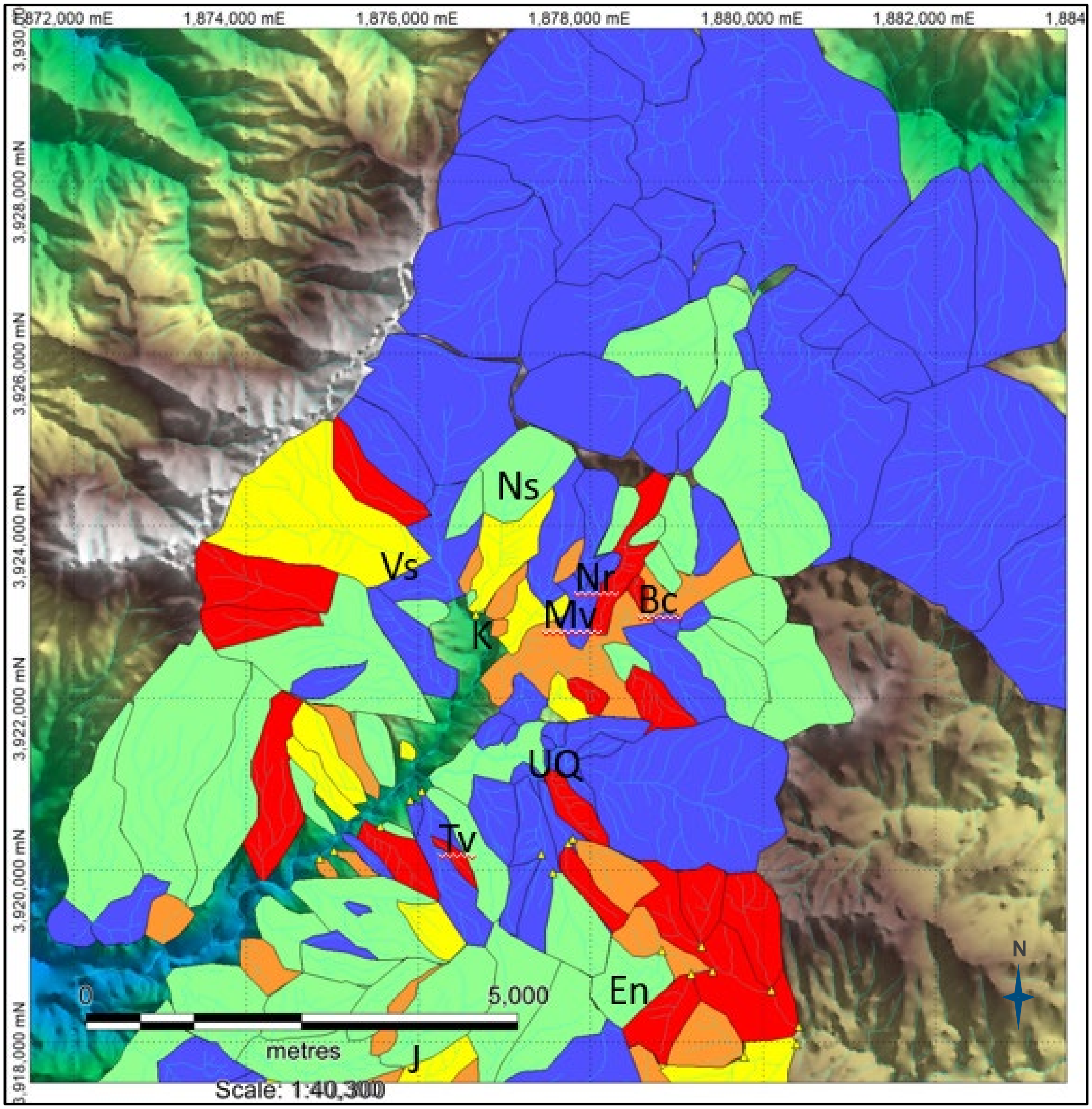
Vanadium in cBLEG shows no discernible pattern or relationship to known mineralization and is, perhaps, influenced primarily by the distribution V-bearing minerals in the mafic rocks of the Nadele Breccia.



Source: Lion One (2019)

Figure 9-29: Vanadium in cBLEG stream survey samples

Broadly, the zinc anomalism appears to be peripheral to the mineral system and is perhaps indicative of a distal hydrothermal signature.



Source: Lion One (2019)

Figure 9-30: Zinc in cBLEG stream survey samples

Based on the 2019 cBLEG sampling, and recognising that **gold and tellurium** are the strongest pathfinders from this analysis, the following areas, in addition to the active work areas at Banana Creek, Matanavatu, and Lumuni Creek, are recommended for follow-up reconnaissance and mapping:

1. Nasalo Creek – Based on high gold samples (catchment above cBLEG sample **TCBL19122**)
2. Vunisalato creek – Based on high gold samples (catchment above cBLEG sample **TCBL191200**)
3. The catchments (name) immediately west and south of Nasiti Ridge, based on high tellurium samples, and along strike from Matanavatu (catchment above cBLEG samples **TCBL19116, TCBL19105, TCBL19098**)
4. The catchment north (east) of Banana Creek, based on high tellurium samples (catchment above **TCBL19114**)
5. The catchment south of Lumuni Creek (south of Banana Creek), based on high tellurium samples (catchment above **TCBL19099**)

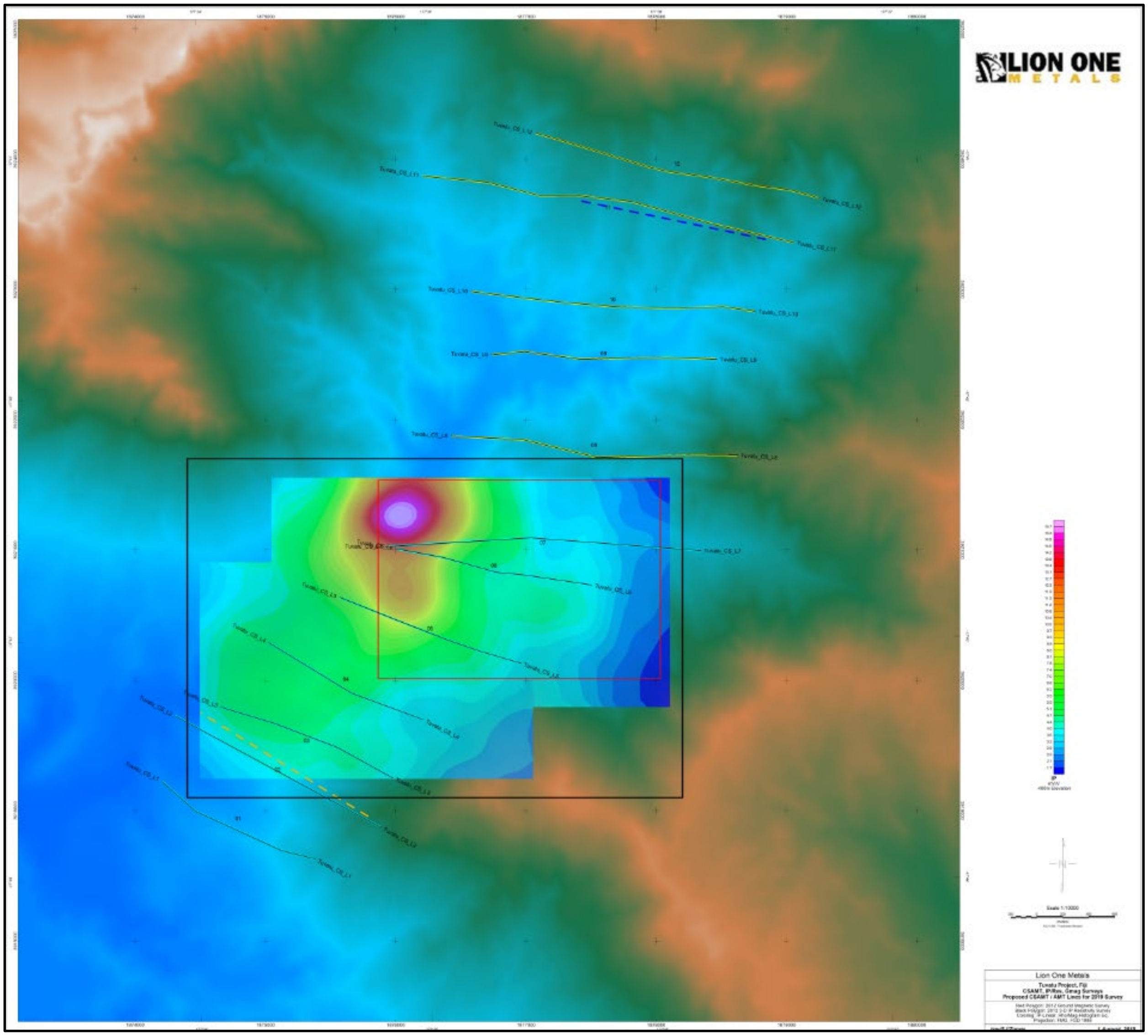
In addition, the outlier in the East Namotomoto area, where Au-Te-Pb-Zn was highlighted as associated with a single catchment (**TCBL19169**) should be followed up.

9.5.4 Geophysical Techniques

9.5.4.1 Induced Polarization Survey – 2012

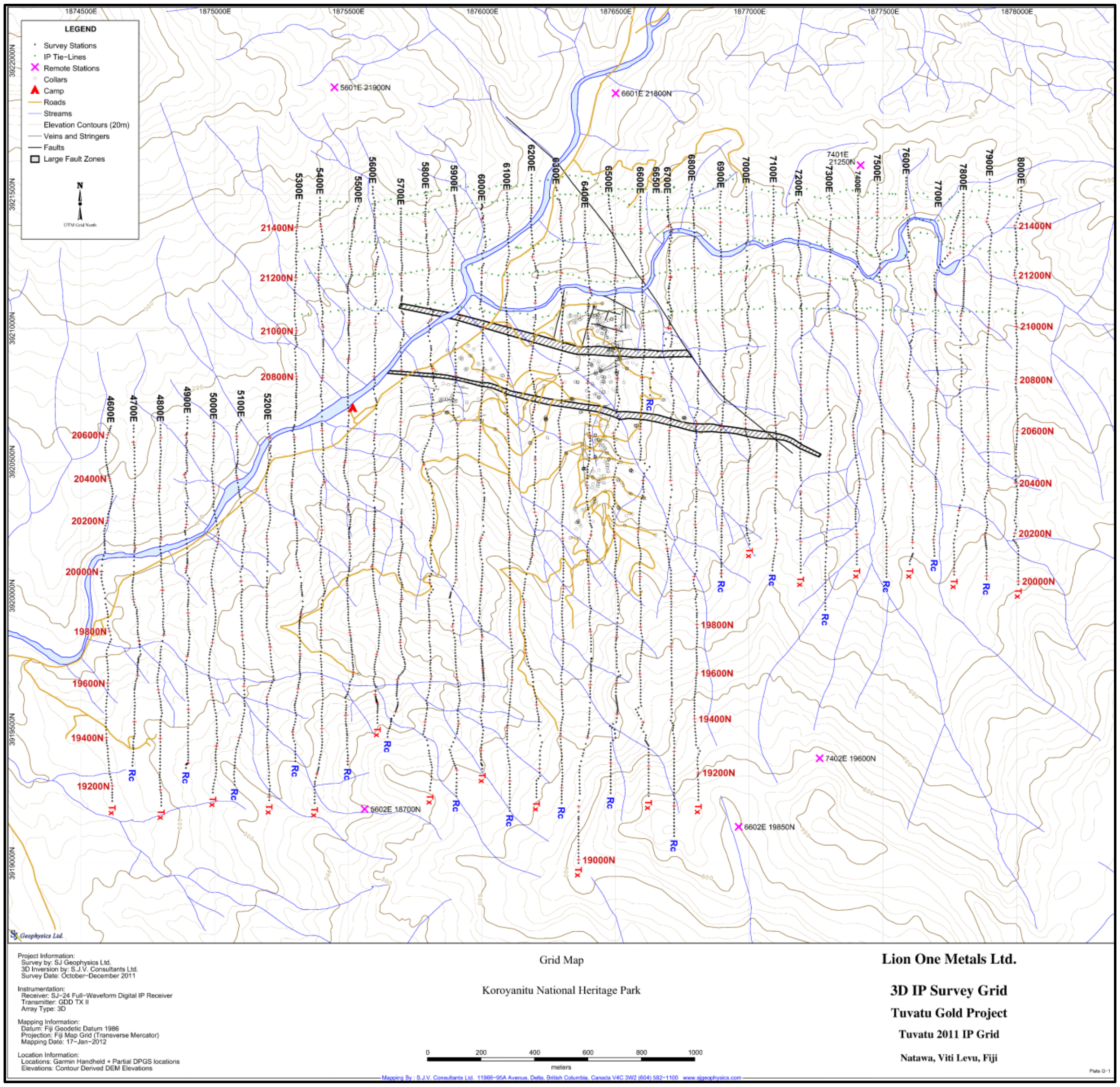
In 2012, SJ Geophysics of Vancouver conducted a ground based IP survey over the main Tuvatu Mine area (Figure 9-31 and Figure 9-32). These data were modelled and interpreted in 2019 by Thomas Weis, a geophysical consultant based in Denver, Colorado. Weis' remit was to not only consider the chargeability and conductivity zones revealed in the 2012 IP survey, but to also consider the electrical properties of the rock types and alteration with a view to conducting a controlled source audio magnetotelluric survey (CSAMT). This survey was subsequently completed in late 2019 (refer below).

Much of this section is paraphrased or summarised from Thomas Weis' 2019 report. Weis considered that the 2012 data from SJ Geophysics was of extremely high quality.



Source: Thomas V. Weis & Associates (2019)

Figure 9-31: An index map showing the planned 2019 CSAMT line locations, the 2012 3-D IP/resistivity polygon (black), and the 2012 ground magnetic survey polygon (red). Data is plotted on digital topographic grid.

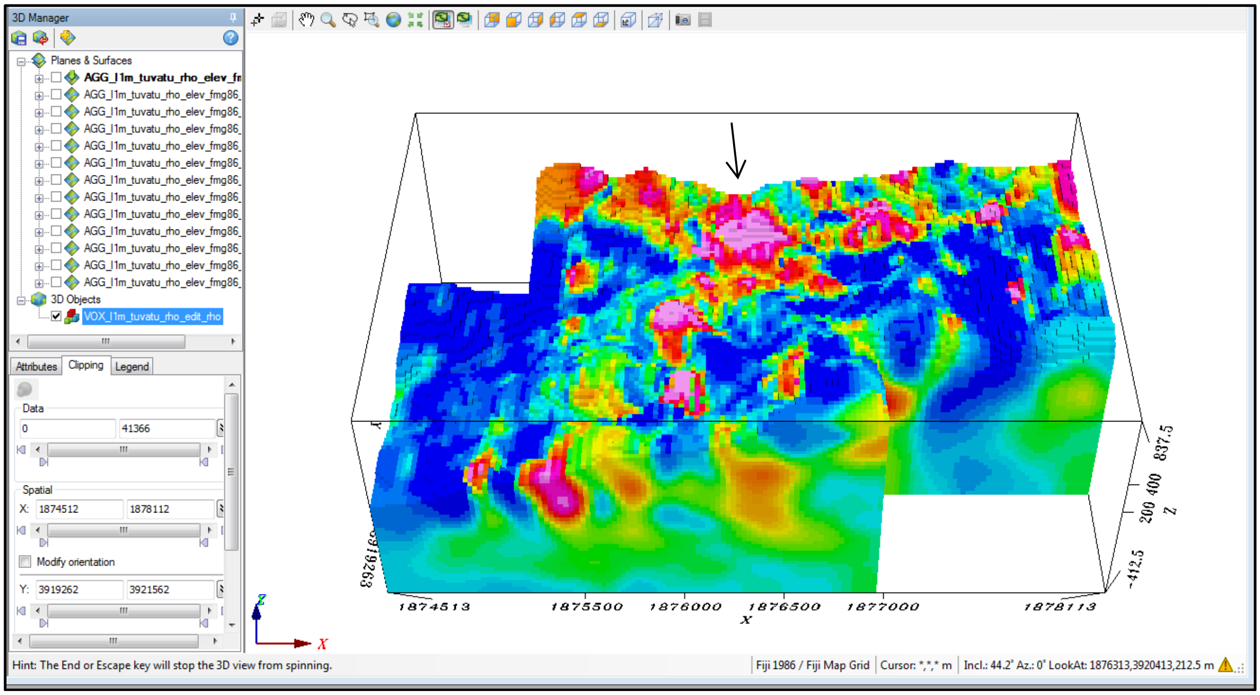


Source: Thomas V. Weis & Associates (2019)

Figure 9-32: The IP/resistivity grid plotted on the topography with drill pattern and main structures indicated. The line spacing is approximately 100 m and stations are marked out every 25 m on the ground with potential electric measurements every 50 m.

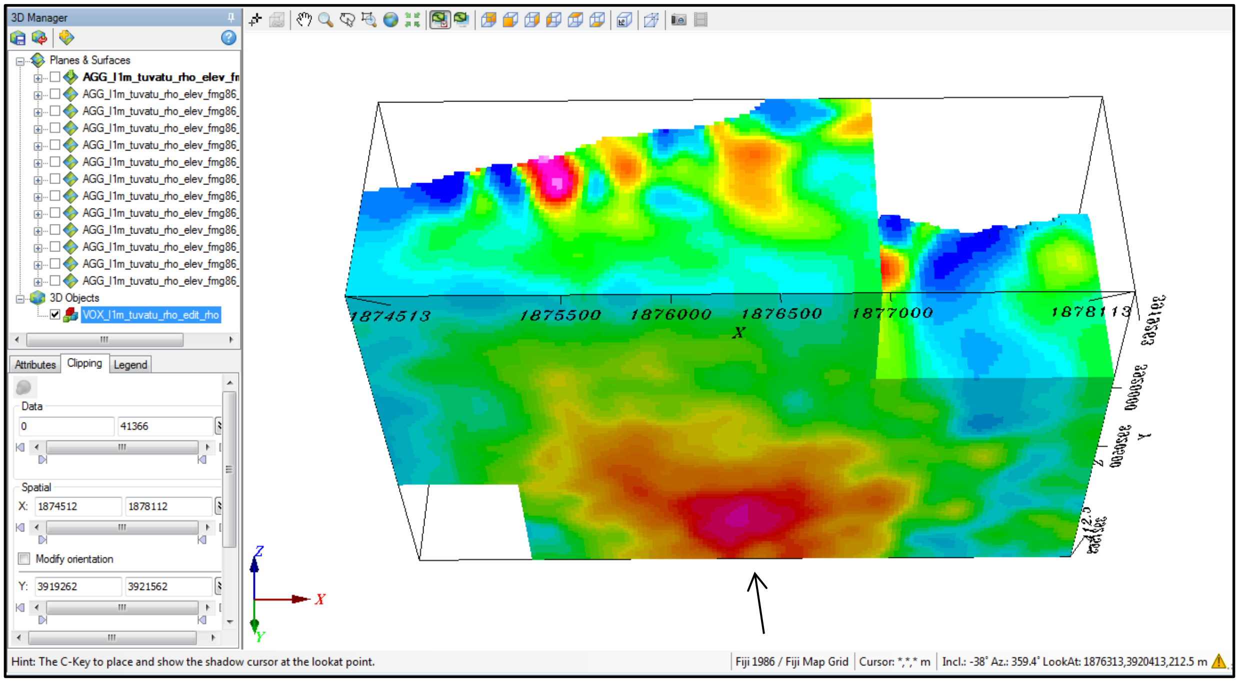
Resistivity

The resistivity data set appears to map lithology better than the IP or magnetic data sets. The resistivity highs are interpreted as intrusive bodies. Figure 9-33 and Figure 9-34 are images of the top and bottom of the resistivity voxels, respectively. A near surface resistor indicated by a black arrow is shown in Figure 9-33. A deep resistor is indicated in Figure 9-34. Both resistors are interpreted to be intrusive bodies; however, increased resistivity due to alteration or silicification is an equally plausible conclusion.



Source: Thomas V. Weis & Associates (2019)

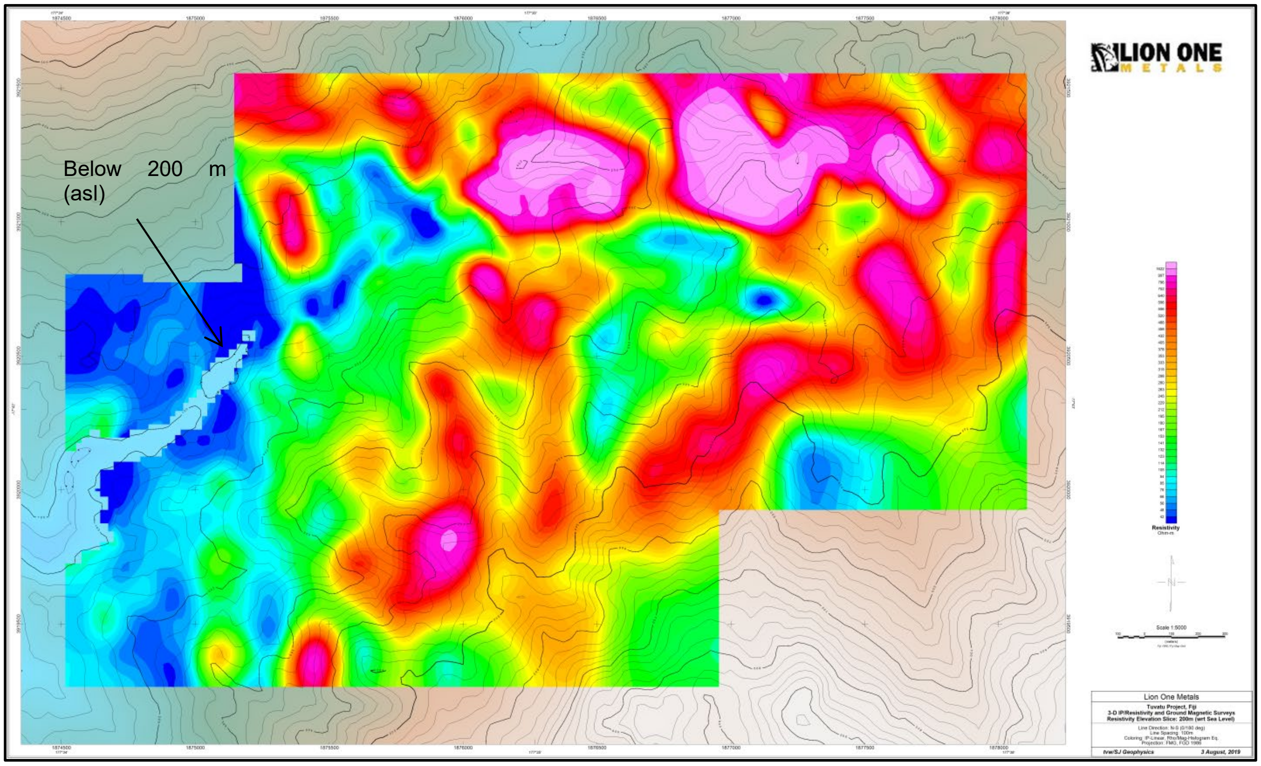
Figure 9-33: The 3-D resistivity voxel viewed down and to the north. Note the near surface resistivity highs located at the north end of the survey block (colored red). This 3D model is attached to this report as an ASCII file and can be entered into a 3D viewing package such as Vulcan or Leapfrog.



Source: Thomas V. Weis & Associates (2019)

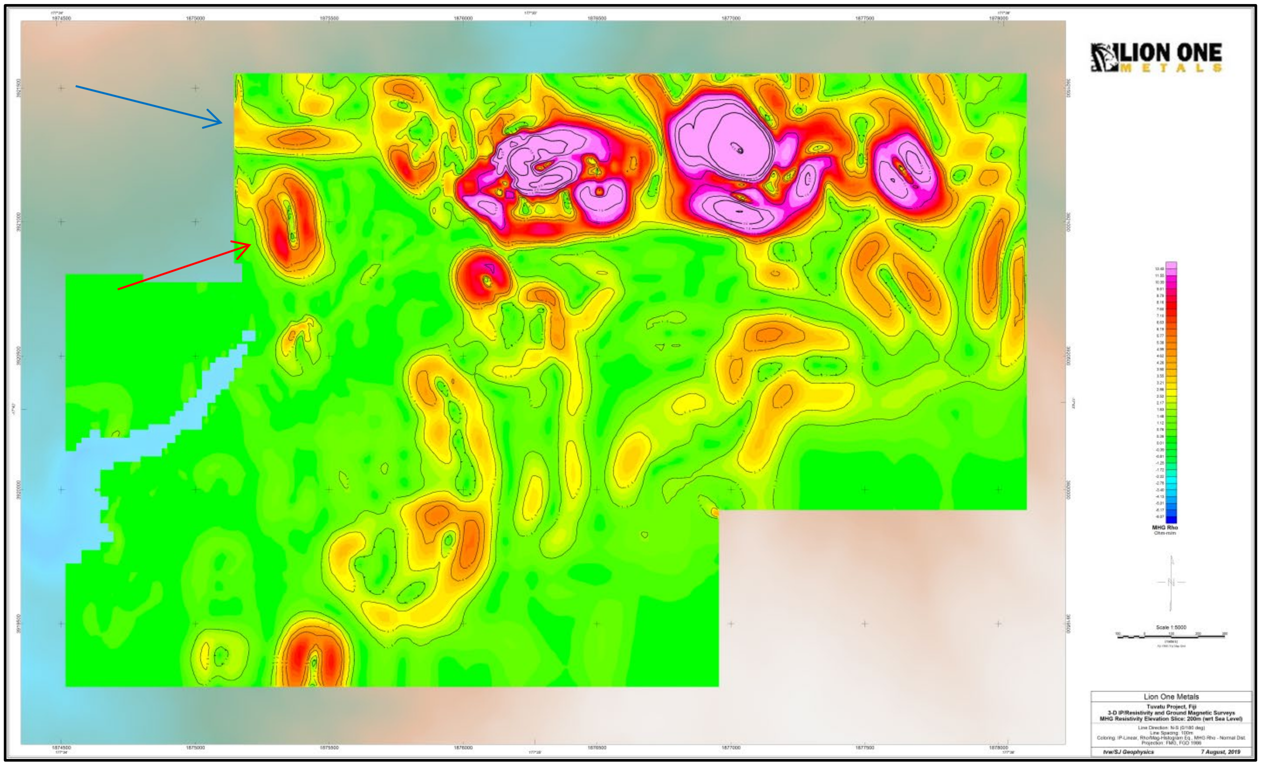
Figure 9-34: The 3-D resistivity voxel viewed up and to the north. Note the deep high resistivity zone located at the north end of the survey block (colored red). The source of this resistivity high is interpreted to be an intrusive body.

Three elevation slices from the resistivity and IP voxels are discussed in this interpretation review. The shallowest elevation slice at +200 m elevation is shown in the figures below. Figure 9-35 shows the resistivity slice at +200 m elevation. This slice was chosen as it is the shallowest slice that covers the majority of the survey block (not incised by creeks). Note the lower elevation river drainage where no resistivity data is presented in this elevation slice. The high resistivity zones (red/pink) are interpreted to be lithologic features, possibly intrusions. It is the edges of these features that are interpreted to be of exploration interest. Figure 9-36 shows the magnitude of the horizontal gradient (MHG) of this data set at +200 m elevation. The MHG is a mathematical way to locate the edges of bodies or the axis of structures. The MHG of resistivity is calculated for all depth slices from +600 m elevation to -400 m elevation. However, only the +200, 0, and -300 m MHG data sets are shown here. Figure 9-37 shows the interpreted edges of the magnetic bodies or axis of structures. The thickness of the black lines indicates the amplitude of the MHG in that area. They are divided into thick, moderate and thin lines in a qualitative way.



Source: Thomas V. Weis & Associates (2019)

Figure 9-35: The Tuvatu 3-D resistivity elevation slice at 200 m (asl). This is the shallowest elevation slice that covers the complete survey area. Note the gap in the data (black arrow) where the topographic surface is lower than 200 m (asl) in the river drainage. Numerous structurally controlled resistivity highs are shown here (red). These features are relatively near surface. Edges of these bodies are difficult to pick visually from this data set.

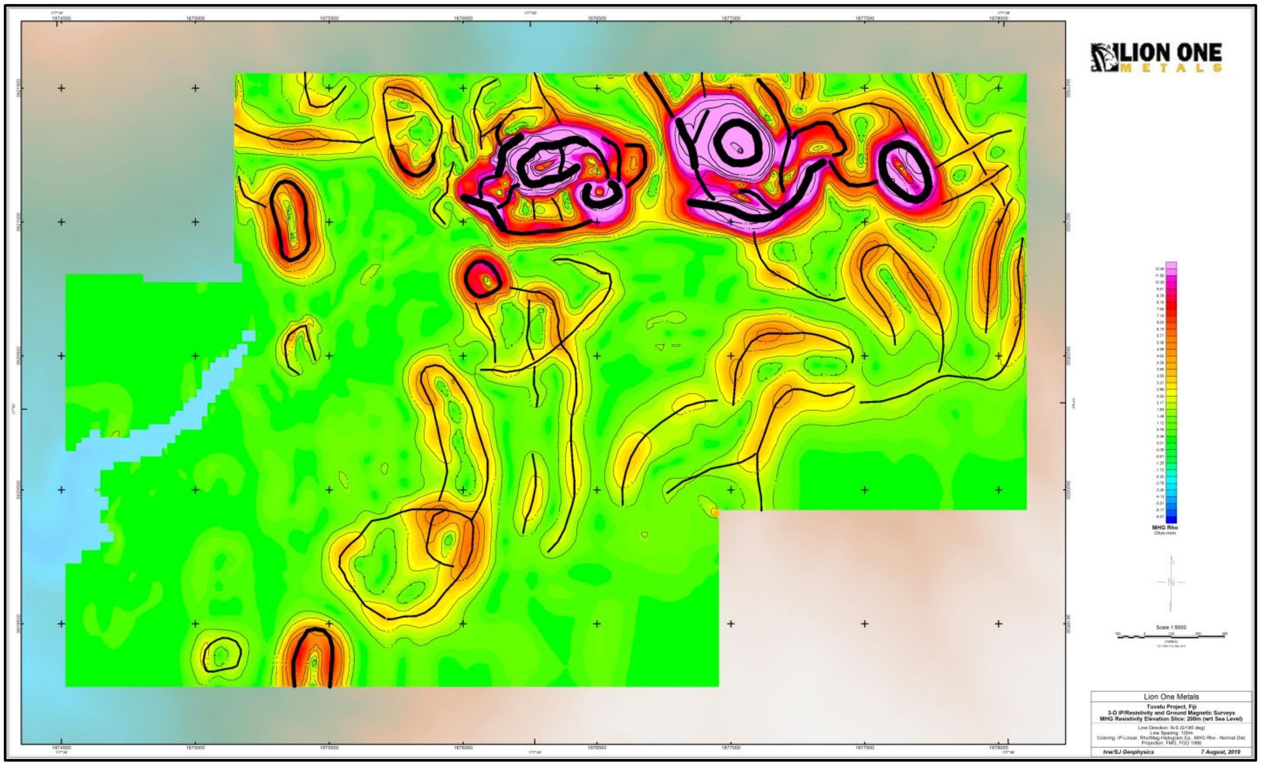


Source: Thomas V. Weis & Associates (2019)

Figure 9-36: The MHG for the 200 m elevation slice. This systematically identifies the edges of bodies. Single MHG high features are contacts or structures (example blue arrow). Circular, closed, donut-like features are the edges of individual intrusive-like bodies (example red arrow). There are numerous features that are somewhere in between.

Figure 9-36 is the calculated MHG of resistivity for the 200 m elevation depth slice. The yellow, red, and pink highs are the gradients at the edges of resistive bodies or at contacts/structures. It is proposed here that these features are drill targets.

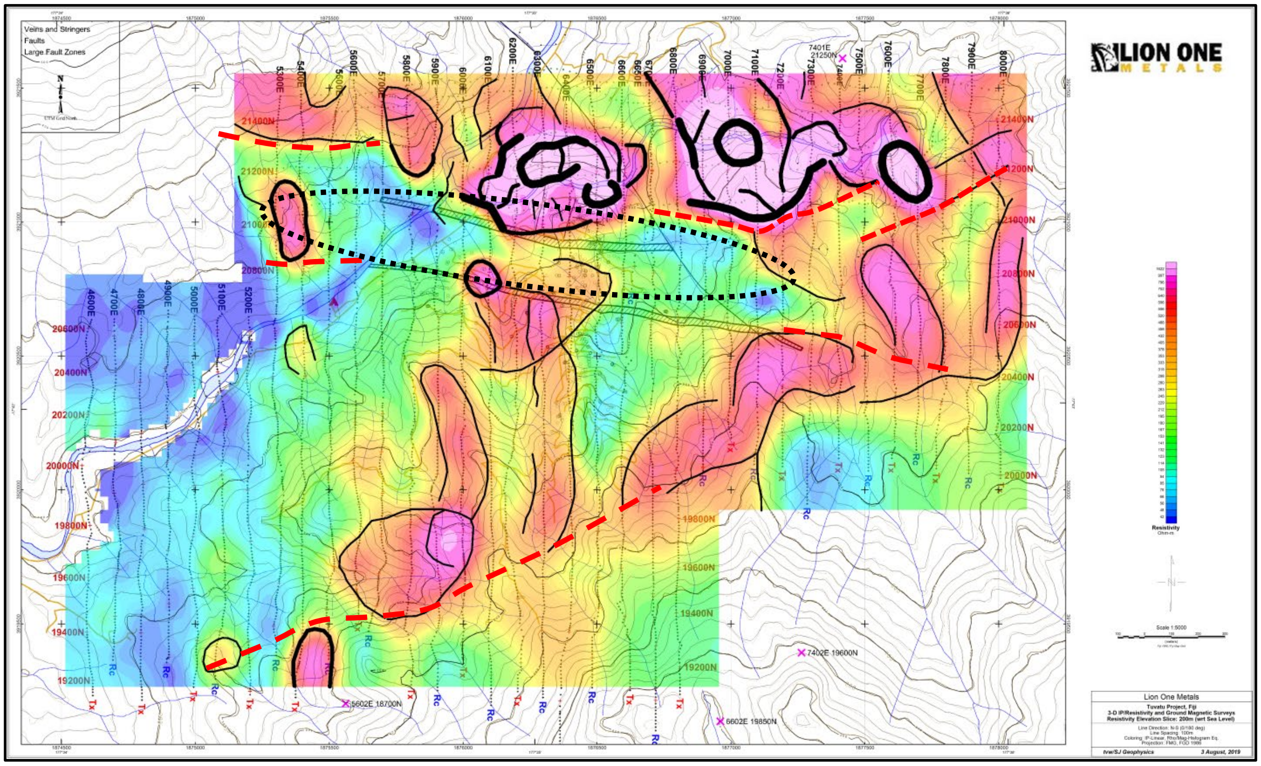
Figure 9-37 shows the MHG image with the interpreted contacts plotted as an overlay on the +200 m (asl) elevation slice. This overlay is then plotted on top of the other geophysical data sets, including the IP elevation slices and the ground magnetic data set.



Source: Thomas V. Weis & Associates (2019)

Figure 9-37: The MHG for the +200 m elevation slice with the interpreted/digitized resistivity edges plotted on top. This type of overlay is available for the +200, 0, and -300 m elevation slices shown here. They can be generated for the additional elevation data sets if or when required.

Figure 9-38 shows the +200 m resistivity slice with interpreted horizontal gradients (MHG) overlay plotted on top of the survey line / drill hole base map (as shown in Figure 9-32). The east–west structural zone (black dashed ellipse) is identified in this map. Additional interpreted structures are shown as red dashed lines. Both the gradient highs and the interpreted red dashed structures are exploration targets. Any geochemical anomalies associated with these features upgrade the targets. This is the beginning of an interpretation overlay that can be improved by field geologists with geologic and geochemical mapping.



Source: Thomas V. Weis & Associates (2019)

Figure 9-38: The MHG overlay plotted on top of the +200 m resistivity elevation slice and interpretation with the survey line / drill hole and topography as a base

In the integration of the resistivity data with known geology, the following observations are made:

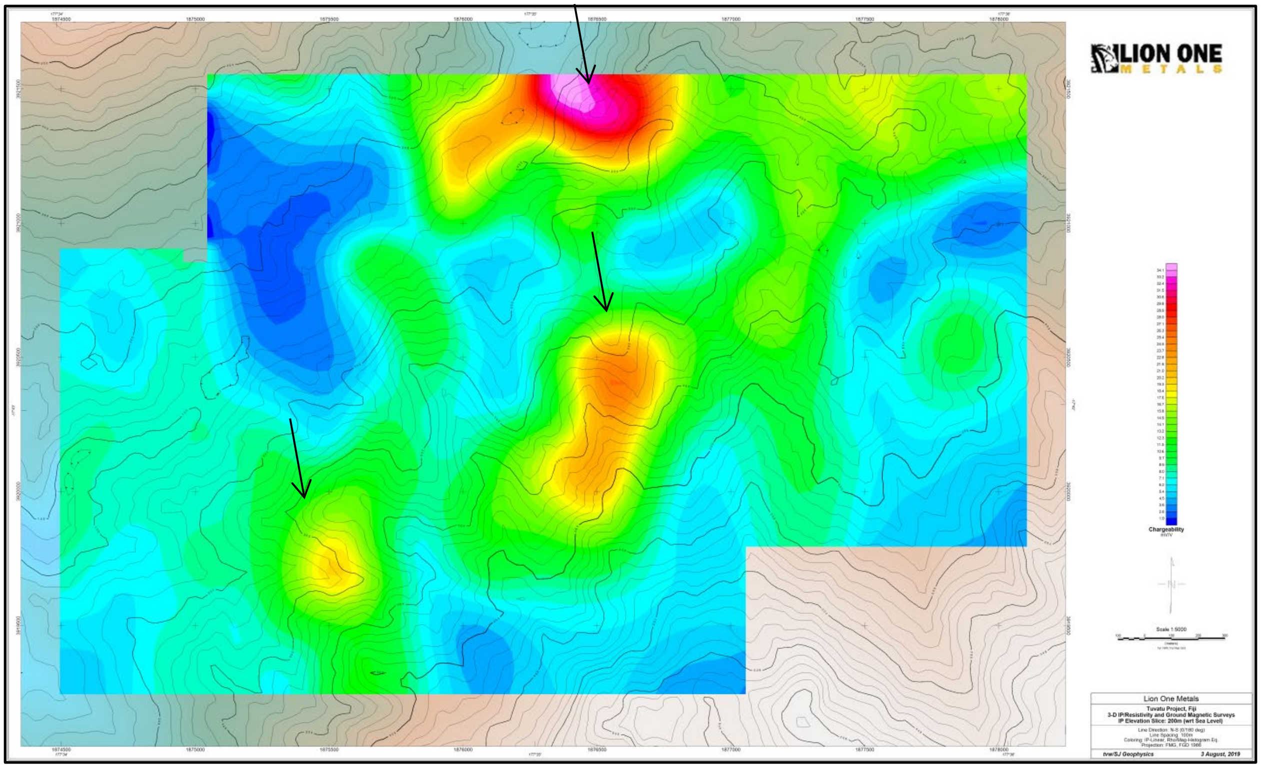
1. The east–west structure labelled as the dashed line in Figure 9-38 most likely relates to the zone of influence of the Core Shed and CABX faults. These faults, whilst containing some elevated mineralization, are not major mineralized zones. However, the southern margin of the dashed area, particularly west of the main areas of mineralization in the deposit, coincides with the Murau / West Vein areas.

2. The north–south features shown between IP lines 5900E and 6200E are coincident with the main Upper Ridges Lodes in Tuvatu.

3. The highly resistive bodies in the north of the survey area, between 6000E and 6600E are coincident with the high-sulphide and argillic alteration in the vicinity of the HT-Lodes.

IP Chargeability

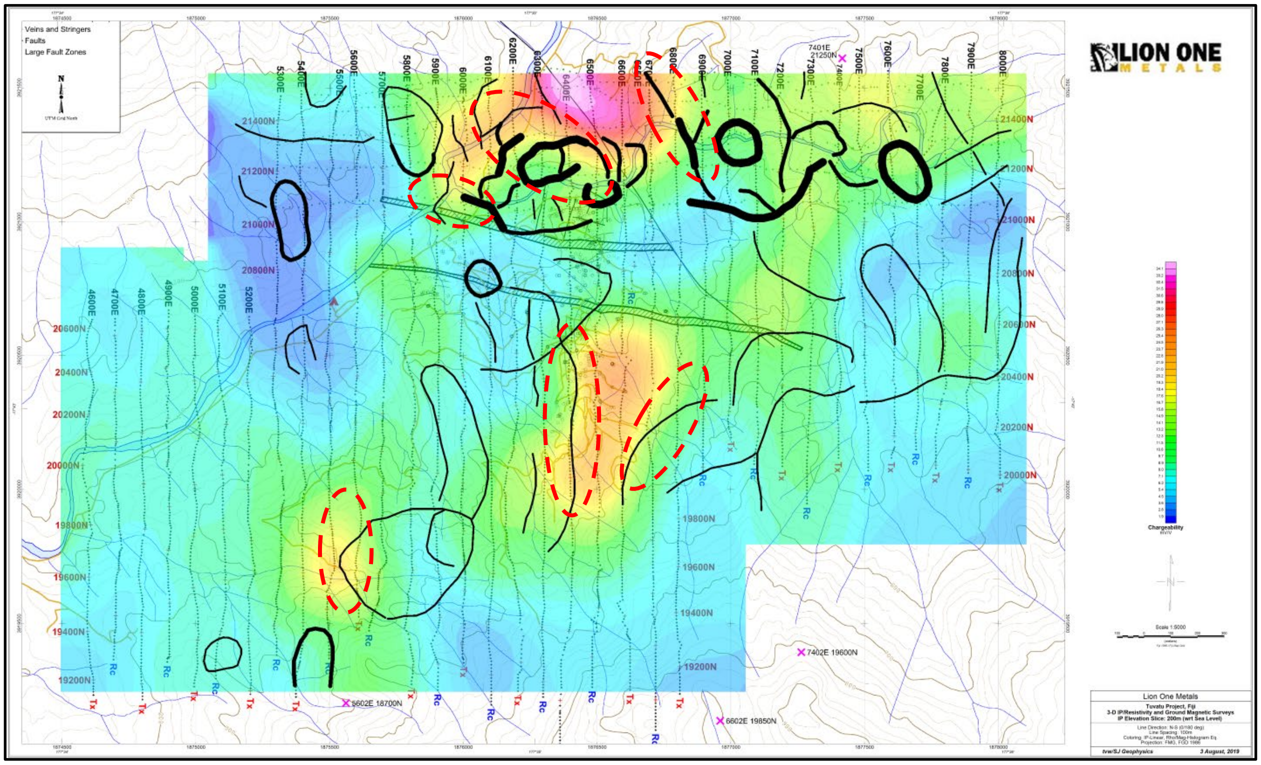
Figure 9-39 through to Figure 9-43 illustrate the IP model elevations slices at +200, 0, and -300 m elevations. At all depths, the largest IP responses are adjacent (not coincident) to the resistivity highs interpreted to be intrusive centers. The IP high responses are on the low side of resistivity gradients. The gradient features adjacent to the IP highs are interpreted to be exploration targets. They are indicated by red dashed ellipsoids in Figure 9-42, Figure 9-43, and Figure 9-44



Source: Thomas V. Weis & Associates (2019)

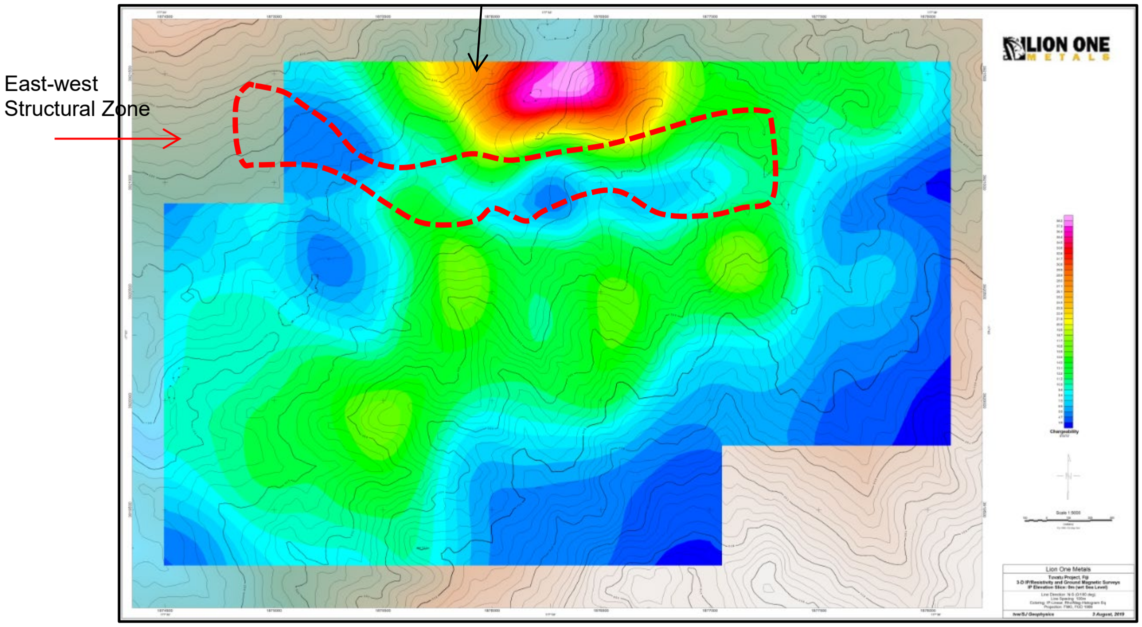
Figure 9-39: The Tuvatu IP +200 m elevation slice. The near surface high IP responses are indicated by the black arrows. The relationship of these anomalous IP responses to gold mineralization is unknown at this time. They could be related to a porphyry system. The important point is that the IP highs are shifted to the side of the resistivity highs and are interpreted to occur at the edges of high resistivity intrusions (see Figure 9-40).

The anomalous IP highs are interpreted to indicate sulphide rich alteration. These zones are possibly related to disseminated pyrite in the wider alteration cell, such as indicated by phyllic to argillic alteration.



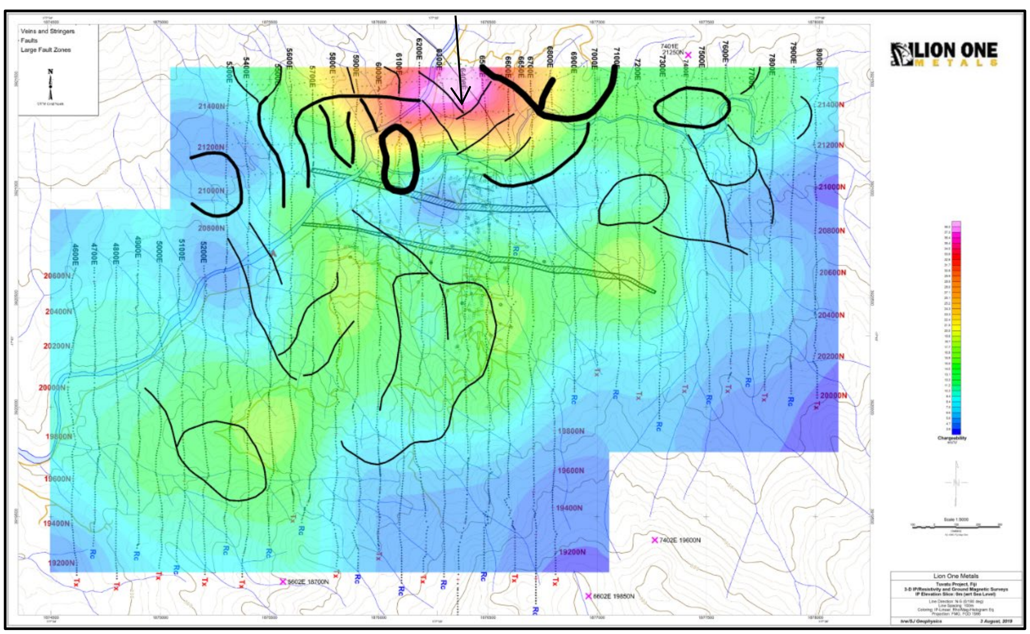
Source: Thomas V. Weis & Associates (2019)

Figure 9-40: The +200 m MHG resistivity overlay plotted on the +200 meter IP elevation slice. Note the IP anomalies are shifted to the side of the resistivity high zones. Structures and contacts near the edges of the Sulphide zones may be targets of interest. The Lion One geochemical database can be used to test this idea.



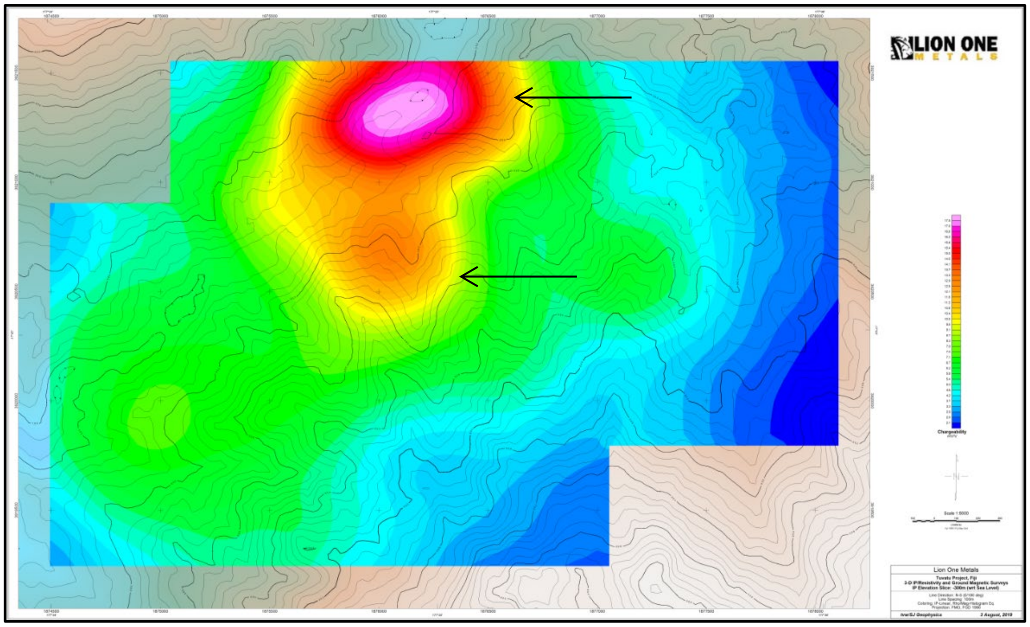
Source: Thomas V. Weis & Associates (2019)

Figure 9-41: The Tuvatu IP 0 m elevation slice. The high IP response of interest is indicated by the black arrow. The east–west trending IP low (red dashed shape) is coincident with the east–west trending structural zone shown in Figure 9-42 (below).



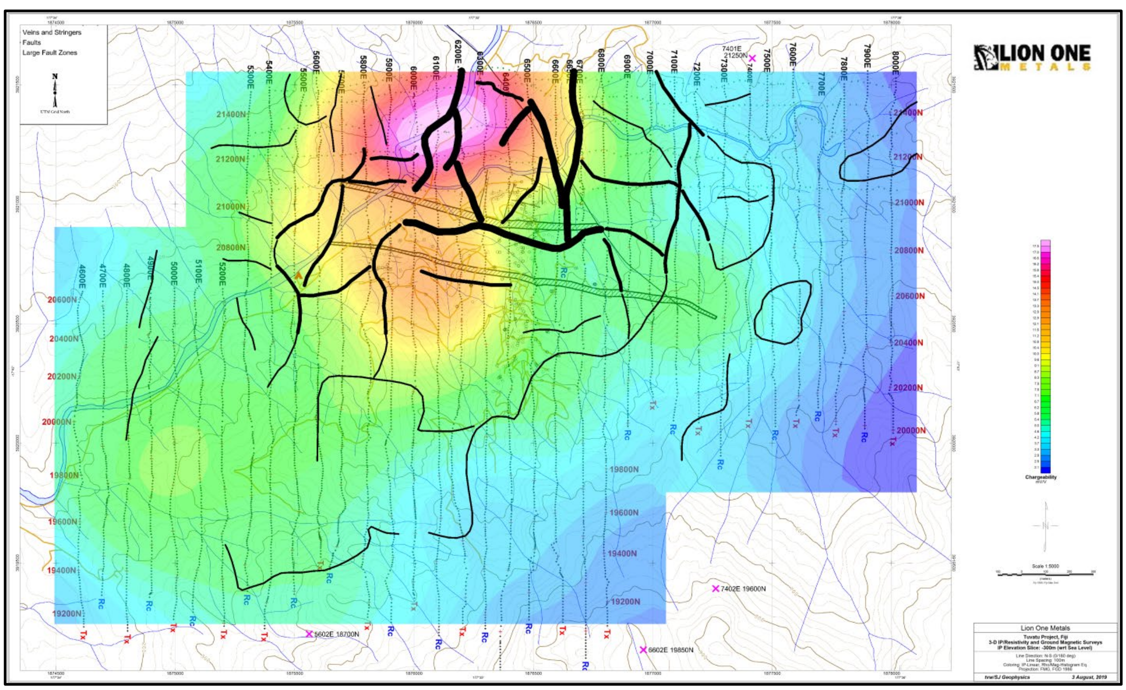
Source: Thomas V. Weis & Associates (2019)

Figure 9-42: The 0 m MHG resistivity overlay plotted on the 0 meter IP elevation slice. Note that once again the IP anomaly is shifted to the side of the resistivity high zones (see black arrow). In fact the high IP response is surrounded by the higher resistivity features in this elevation slice. Once again interpreted contacts and structures adjacent to the IP high are exploration targets.



Source: Thomas V. Weis & Associates (2019)

Figure 9-43: The Tuvatu IP -300 m elevation slice. The high IP responses of interest are indicated by the black arrows.



Source: Thomas V. Weis & Associates (2019)

Figure 9-44: -300 m MHG resistivity overlay plotted on -300 m IP elevation slice. At this depth, the relationship between the resistivity and IP highs is more poorly defined. A slight shift appears to exist.

In integration of the IP (chargeability) data with known geology, the following observations are made:

1. The east–west structure labelled as the dashed line in Figure 9-41, similar to the resistivity data, most likely relates to the zone of influence of the Core Shed and CABX faults. These faults, whilst containing some elevated mineralization, are not major mineralized zones. However, the southern margin of the dashed area, particularly west of the main mineralization deposit, coincides with the Murau / West Vein areas.
2. The IP generally shows a wide diffuse signature not directly related mineralization, though a body does sit to the hanging wall of the UR2 Lodes. This is, perhaps, an elevated, disseminated pyrite content in the overall monzonite.
3. The highly chargeable anomaly in north of the area is coincident with a zone of increased silver content in surface samples (north of Qalibua Creek) and is perhaps related to higher-epithermal level mineralization in an argillic alteration zone.

9.5.4.2 CSAMT – 2019

In late 2019, Lion One completed a CSAMT survey over the Project, including the Navilawa Caldera. The total survey was over 22,000 m of data across twelve individual lines. In combination with natural source, the data were able to be modelled from surface to depths of up to 1,400 m.

The survey was carried out by Zonge Engineering and Research Organisation (Zonge) based in Adelaide, Australia. The equipment specifications were:

- Zonge International GDP3224 receiver system (roving and remote reference).
- ANT-6 0.1-10000 Hz magnetic sensors.
- Zonge International GGT10 transmitter system.
- Non-polarisable copper sulphate electrodes.
- CSAMT data recorded at 32 kHz same rate.
- Audio Magnetotelluric Survey (AMT) data sampled at 1 kHz and 32 kHz.
- GPS locations recorded using hand-held Garmin GPS.

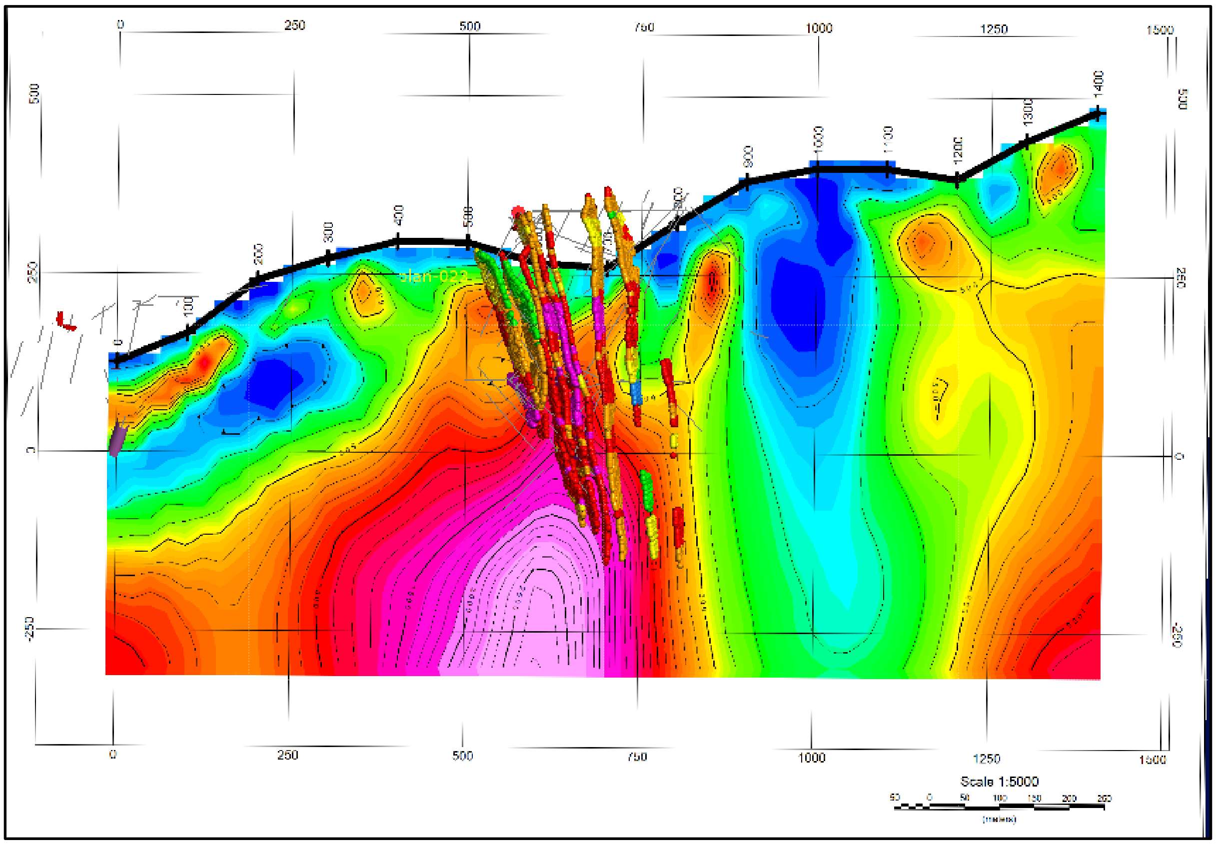
The survey specifications were as follows:

- Data recorded on twelve lines.
- AMT and CSAMT data recorded over the 8-8192 Hz frequency range.
- ExHy and EyHx components recorded (EyHx component recorded for every second site).
- 100 m station spacing.
- Two transmitter dipoles of 1.5 km in length were used, lines 8 to 12 read from tx electrodes at 560086E/8040526N and 560722E/8040121N.
- Lines 1 to 7 read from tx electrodes at 563050E/8044425N and 564099E/8044198N.
- Transmitter current ranged from approximately 14 A at 8 Hz to 3 A at 8 kHz.

The purpose of the survey was to develop a stronger 3D representation of the resistivity and conductivity of the major units. This included:

- The possible mapping of contacts between the monzonite and the overlying Nadele Breccia
- The identification of gradients in resistivity potentially locating significant structures
- The identification of major alteration zones or zones of increased sulphidation

The data (as illustrated with an example cross-section in Figure 9-45) was modelled by Thomas Weis of Denver, Colorado.



Source: Lion One (2020)

Figure 9-45: CSAMT section through the southern part of the Tuvatu mineralization-system (view north). The block model of veins is shown in the foreground, and the resistivity high is footwall to the main Tuvatu Lodes.

CSAMT/AMT Survey 2019

(This section is paraphrased from Thomas Weis' report [2020].)

Twelve lines of combined CSAMT/AMT were run over the greater Project area. Figure 9-46 shows the location of these lines. The survey was run in the scalar mode because of logistical terrain difficulties. The receiver (Rx) station spacing is 100 m, and both CSAMT and AMT are measured on those stations. The total combined frequency range is 4 Hz to 6,000 Hz.

After editing, the data is inverted in 1-D and 2-D, and it is the 2-D inversion models that are interpreted here. It is difficult to invert CSAMT in 3-D because of the multiple distant transmitter (Tx) arrays, so it is not done with the Tuvatu data set. However, the 2-D inversion models are combined, and a 3D voxel display is presented here. The main problem with this 3D display is that the lines are far apart (up to 800 m), so a 3D voxel cell spacing of 100 m is required. This is relatively coarse when considering the nature of the target. There is a lot of interpolation between lines.

The 3-D voxel display used by Lion One advisors is good for suggesting target areas to explore; however, these target areas should not be considered drill ready. Additional lines should be run to tighten up the resolution prior to drilling. If specific areas of interest are identified, the Rx spacing can be reduced to as small as 25 m for drill targeting purposes.

In comparison, the 2-D inversion cell size is based on distance between stations along each section line so they are smaller (25 m), which results in better lateral resolution. Targeting should be based on the individual 2-D line inversions where actual data exists. If detailed 25 m stations are surveyed, the grid cell size can be reduced to 5 m, which will improve drill hole placement dramatically.

The interpretation products presented here are the 3-D resistivity voxel display from the combined 2-D inversion models, the 2-D resistivity inversion models, and the MHG of Resistivity along the 2-D inversion models.

Targets selected in this report are based on the 2-D inversion models while keeping the 3-D voxel display in mind.

Interpretation Approach

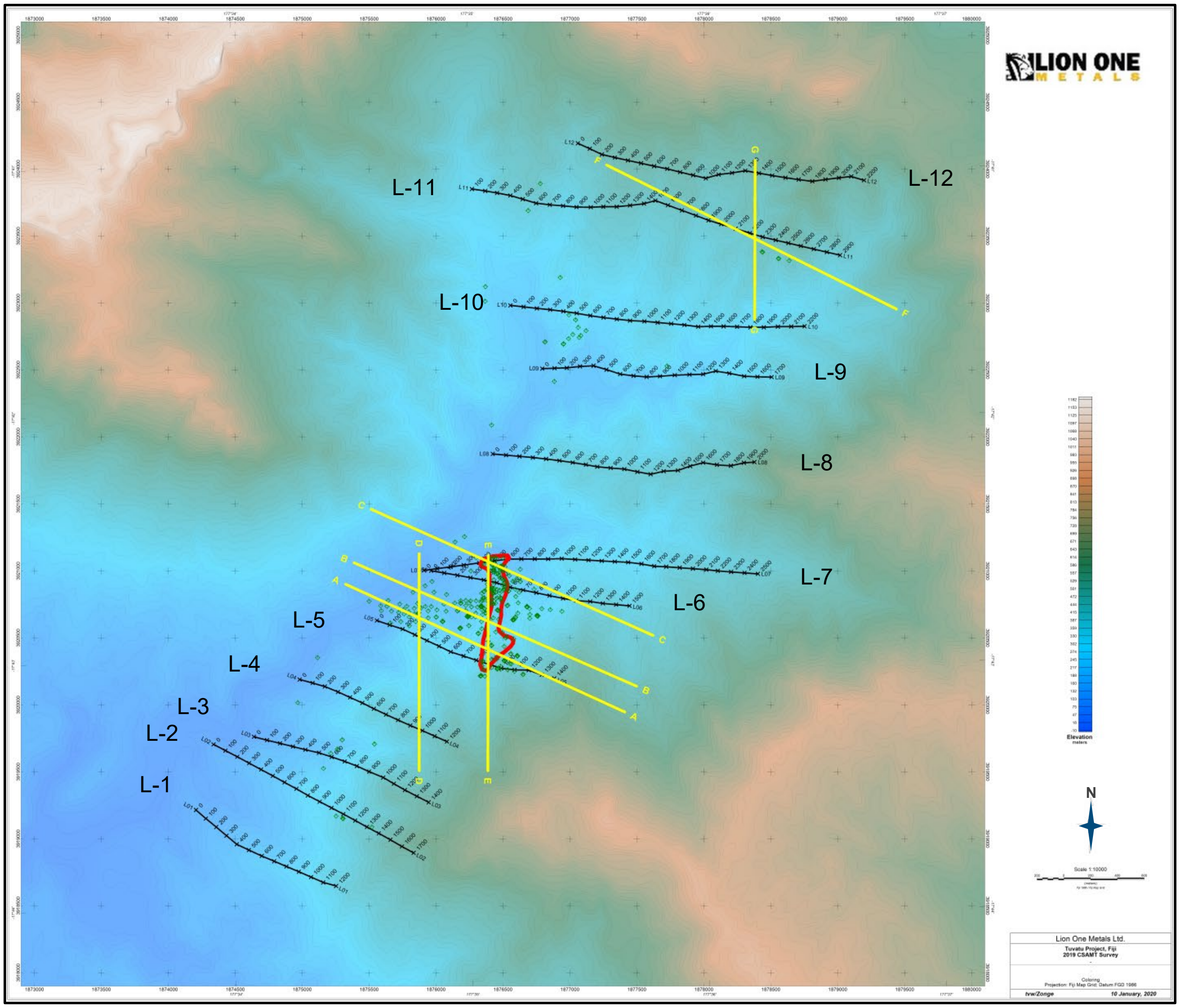
The interpretation approach utilized is to use resistivity to map the brittle monzonite intrusions, which can act as host rocks for the gold mineralization. The lode deposits can be relatively small, but the geologic environment hosting these lode deposits are large enough to be mappable. The boundary of the monzonite intrusives, which may control structures, and the structures themselves, which may cut through the monzonite intrusive bodies, are the resistivity targets. The surrounding Nadele Breccia volcanics are relatively ductile and not amenable to mineralizing fluids passing through or hosting gold mineralization.

The targets are:

1. The edges of highly resistive monzonite intrusive bodies
2. Structures cutting through the monzonite bodies
3. Several interesting vertical low resistivity zones that may indicate alteration along major deep seated structures. These zones could also be structures in the ductile Nadele Breccia so they need to be checked

Survey Location

The location of the Tuvatu CSAMT/AMT survey is shown in Figure 9-46 with lines plotted on top of topography. The important thing to note in this figure is the CSAMT line locations and how far apart they are located, from 300 m to greater than 800 m. The line spacing controls the 3-D voxel cell size and all model information between the lines must be recognized as interpolated. The receiver station spacing is 100 m, so the 2-D along line grid spacing is approximately one-quarter of that (25 m).

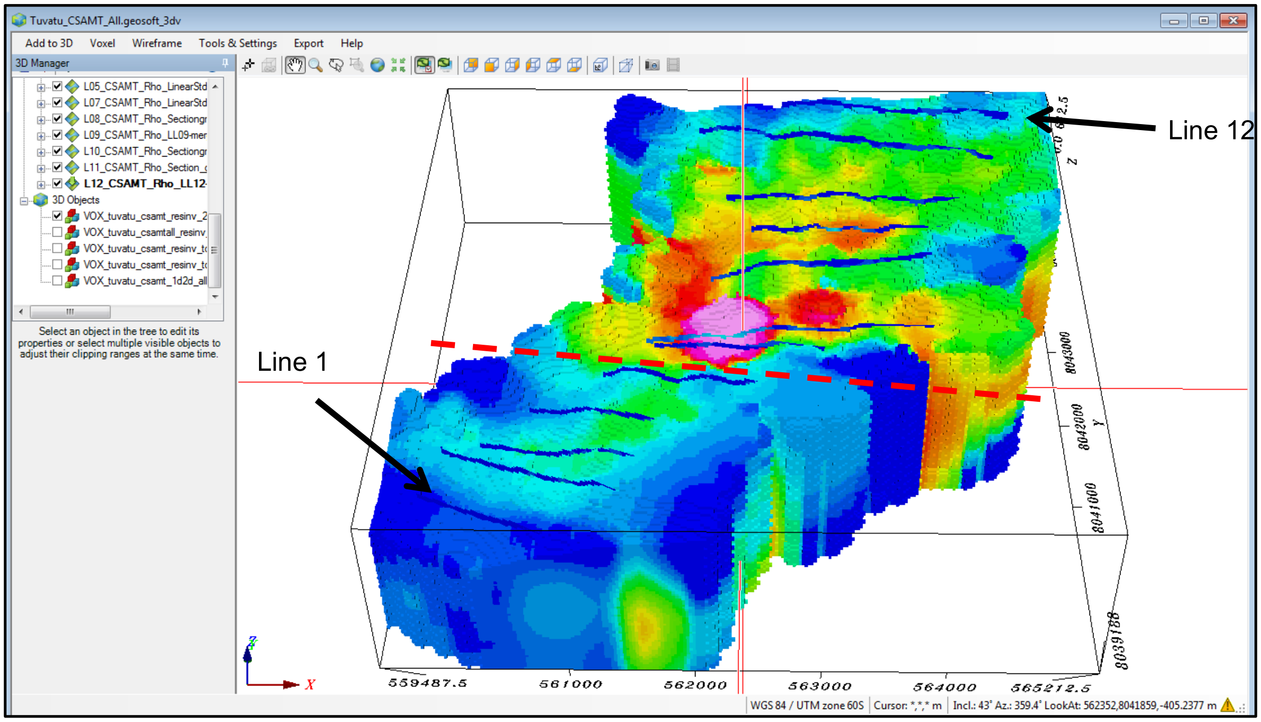


Source: Thomas V. Weis & Associates (2020)

Figure 9-46: The Tuvatu CSAMT/AMT survey lines plotted on topography (Fiji Map Grid, FGD1986). The black lines with stations posted are the CSAMT lines in local coordinates. The red polygon is the approximate location of the deposit projected to the surface. The green circles are drill hole collar positions. The yellow lines are interpreted section lines from a company advisor (Quinton Hennigh).

3-D Voxel Display of Combined 2-D Inversion Models

The 3-D voxel displays of the Tuvatu CSAMT/AMT resistivity data set are shown from Figure 9-47 through to Figure 9-58. Figure 9-47 shows the complete voxel with survey lines plotted on top in blue. Lines 12 to the north and 1 to the south are indicated by black arrows. Note that the model response located off the ends of the lines is extrapolated and of limited use.

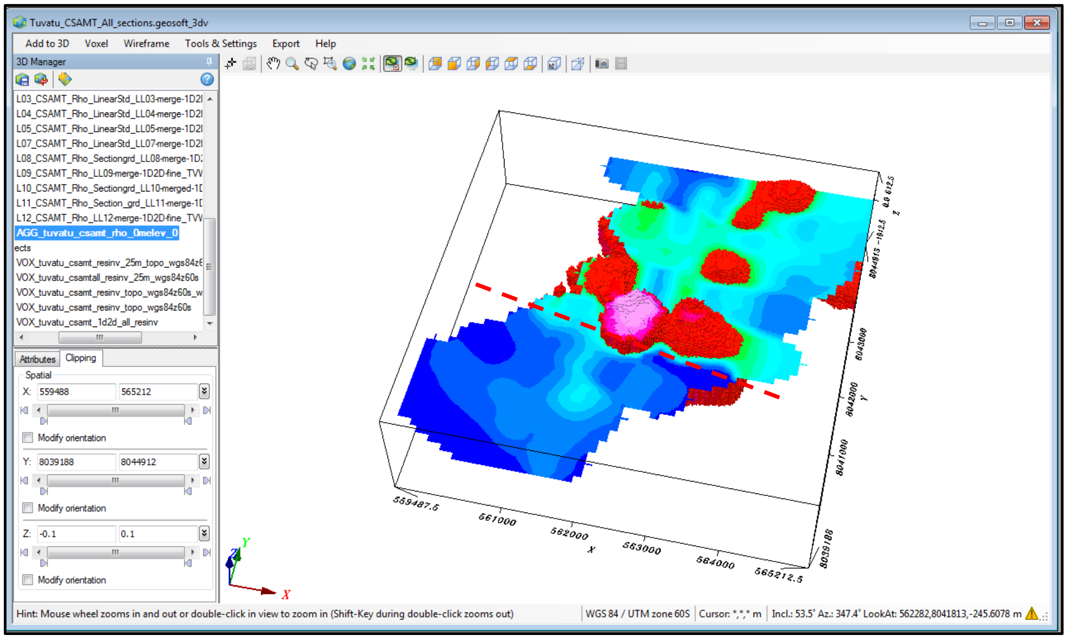


Source: Thomas V. Weis & Associates (2020)

Figure 9-47: The Tuvatu 3-D resistivity voxel display of the combined 2-D inversion models. Line 1 is located on the south end of the survey block. Line 12 is located on the north end of the survey block with the remaining lines numbered sequentially from south to north. The red dashed line separates a zone of higher resistivity to the north and lower resistivity to the south.

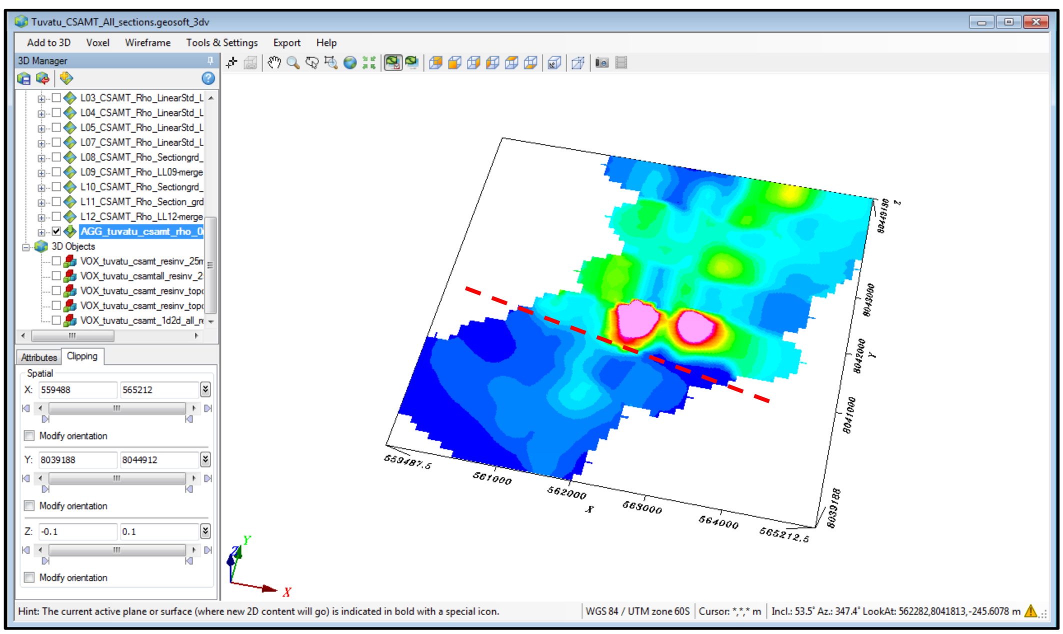
Figure 9-48 and Figure 9-49 show example elevation slice (0 meters asl) through the Tuvatu Resistivity voxel.

Figure 9-48 includes the high resistivity bodies (rho (ρ, electrical resistivity) >2500 ohm-m) as red shapes and Figure 9-49 has these removed. Note the break between high background resistivity to the north and low background resistivity to the south.



Source: Thomas V. Weis & Associates (2020)

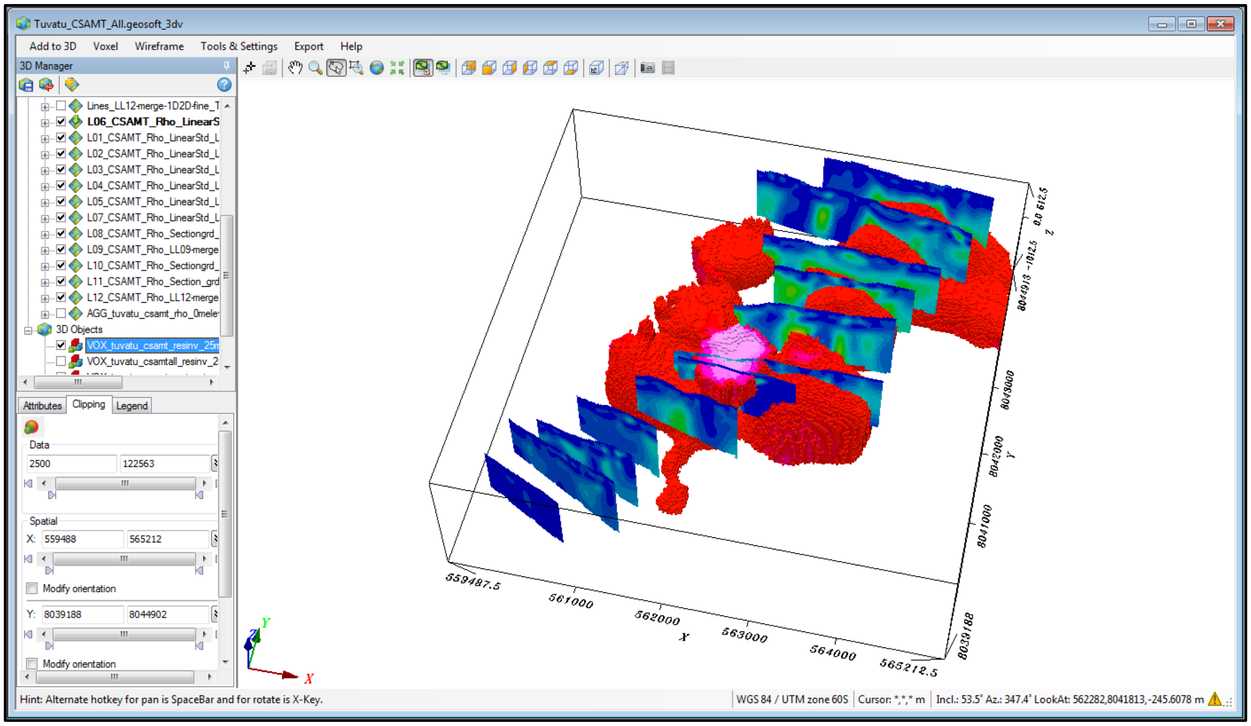
Figure 9-48: The 0 m resistivity elevation slice with the clipped (rho>2500 ohm-m) voxel superimposed on top



Source: Thomas V. Weis & Associates (2020)

Figure 9-49: The 0 m resistivity elevation slice with clipped voxel removed

Figure 9-50 shows twelve section slices for Lines 1 through 12 superimposed on the clipped (rho >2500 ohm-m) voxel display. The high resistivity background to the north (rho >2500 ohm-m) stands out when compared to the low resistivity background to the south (rho <2500 ohm-m) and is interpreted to have geologic significance. The higher resistivity values to the north are interpreted to reflect a greater quantity of resistive monzonite. The contact between high and low resistivity is structural. Based on previous observations, where the rheology of the monzonite allows for brittle fracture and is a permissible host for vein-related mineralization (compared to the Nadele Breccia, in which veins are generally not well formed), the northern zone of high resistivity monzonite intrusive rock is considered more prospective than the lower resistivity rocks (Nadele Breccia or Sabeto volcanic unit) to the south.



Source: Thomas V. Weis & Associates (2020)

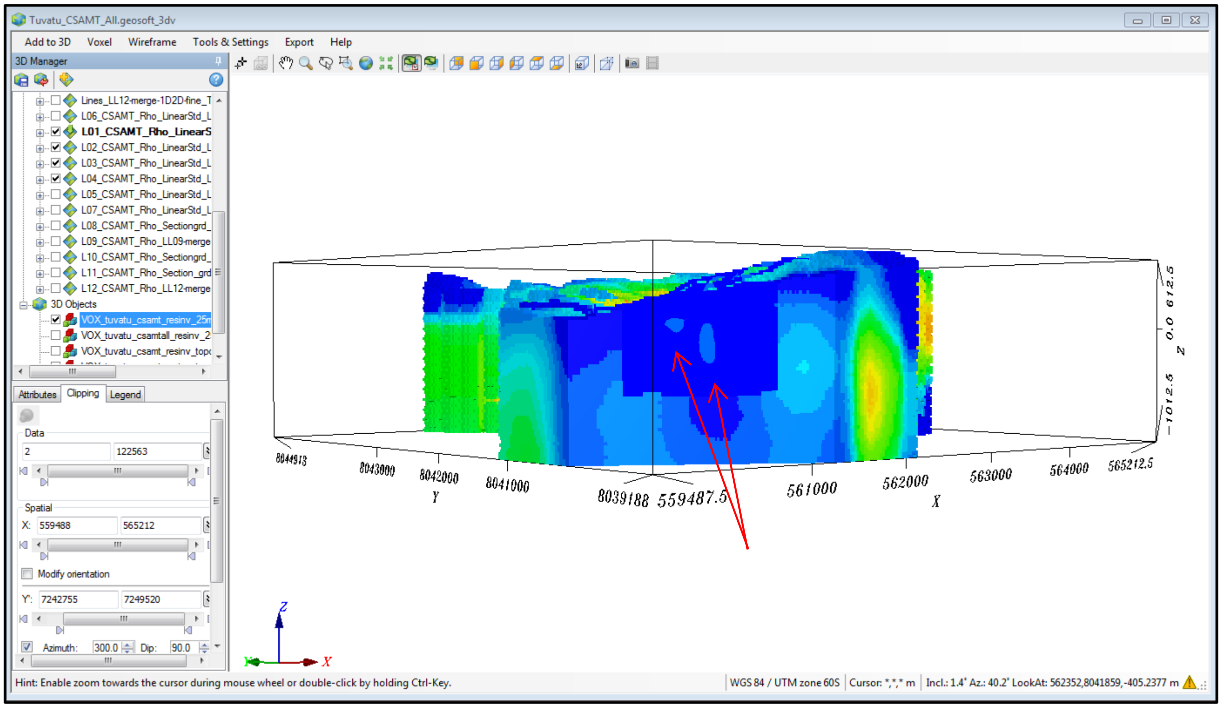
Figure 9-50: The section slices for Lines 1 through 12 superimposed on the clipped (rho>2500 ohm-m) voxel display of the combined 2-D inversion models

An alternative cause of this difference may be instrumental, for example transmitter location or operation, but this is unlikely.

2-D Inversion Models Compared with 3-D Section Slices

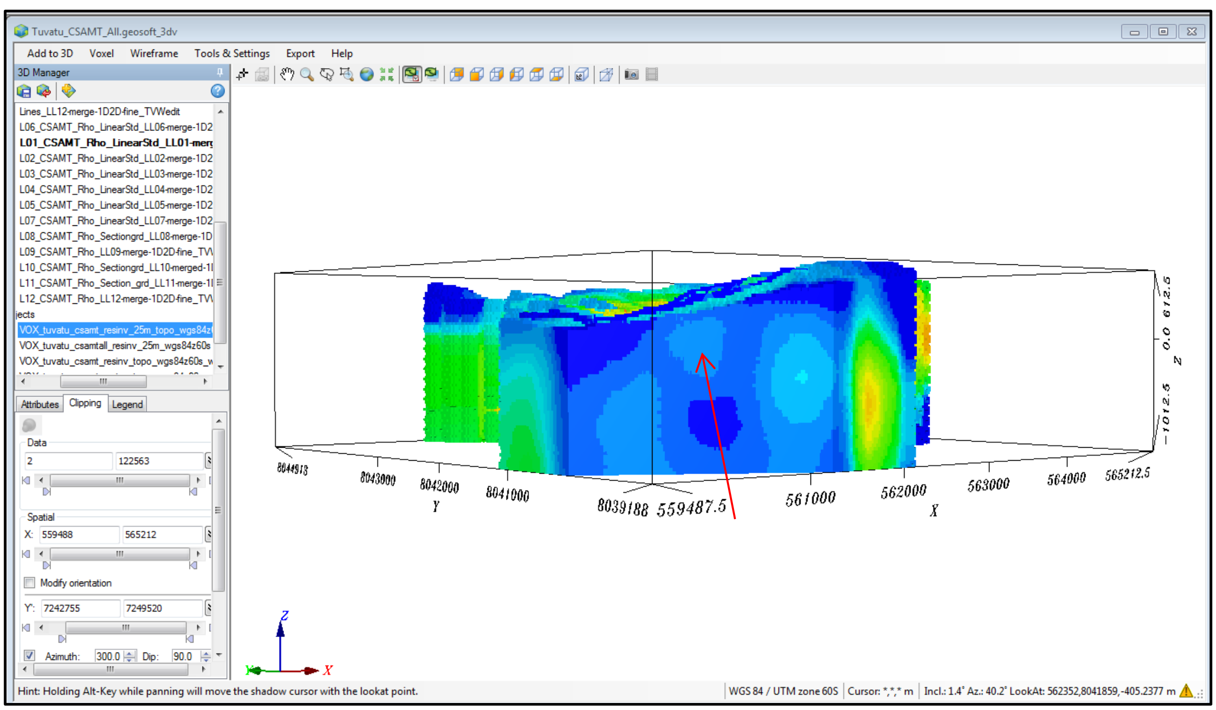
Figure 9-51 through Figure 9-58 show a comparison of selected 2-D line sections and the 3-D voxel response at that line location. The 2-D inversion sections are higher resolution than the 3-D voxel display. This confirms that the 2-D sections should be used for targeting purposes. Drill holes should be located on or immediately adjacent to the CSAMT/AMT survey lines.

Figure 9-51 and Figure 9-52 show the Line 1 comparison between the 2-D inversion and 3-D voxel display data. Similar examples are shown for Line 7 (Figure 9-53 and Figure 9-54), Line 9 (Figure 9-55 and Figure 9-56) and Line 11 (Figure 9-57 and Figure 9-58).



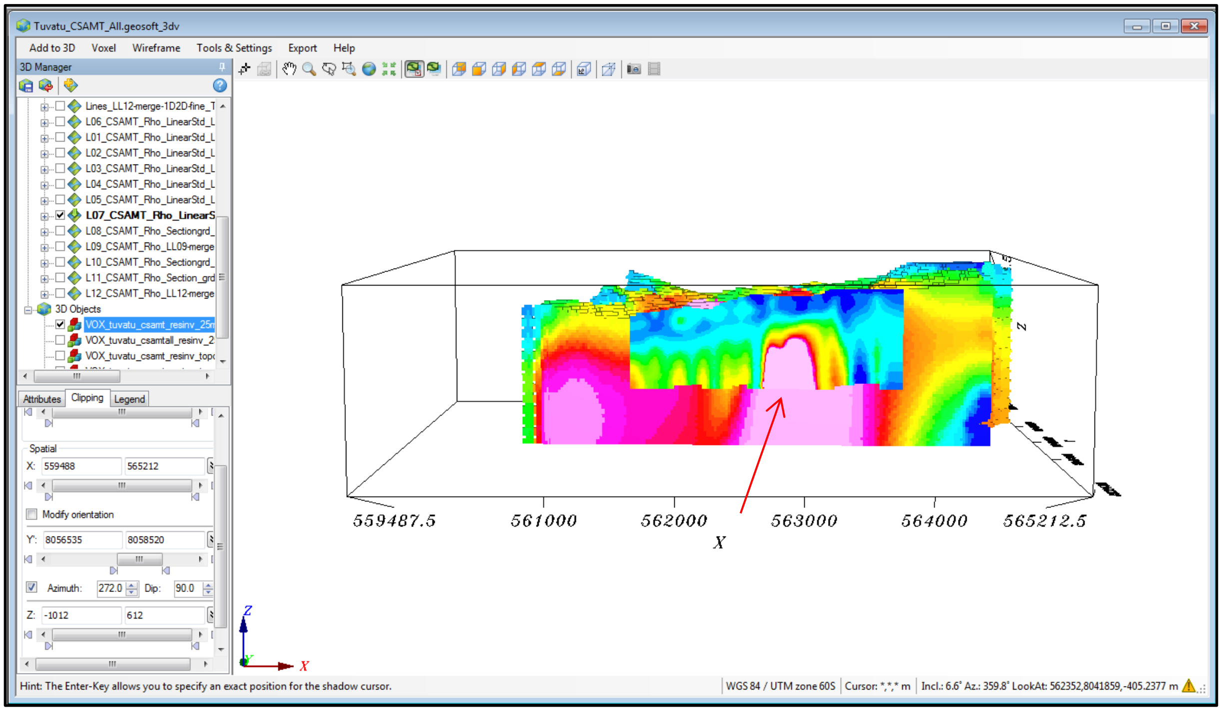
Source: Thomas V. Weis & Associates (2020)

Figure 9-51: The Line 1, 2-D section is turned on. Note the weak, detailed resistivity highs (red arrows).



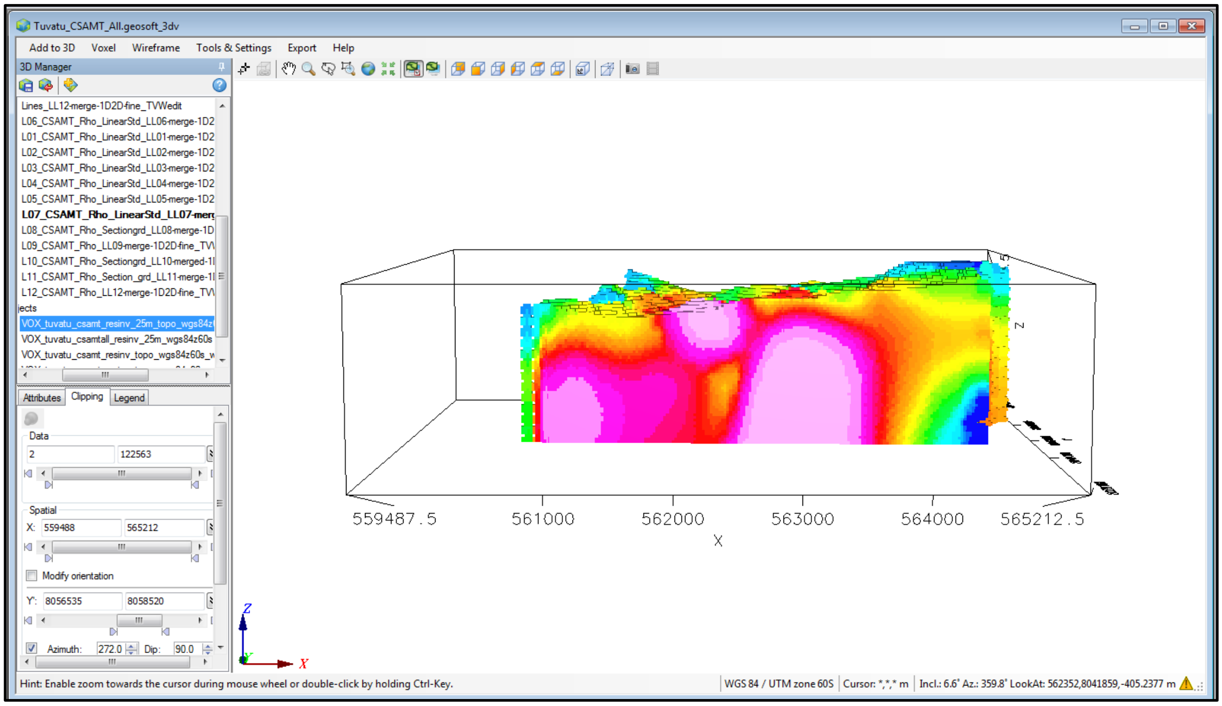
Source: Thomas V. Weis & Associates (2020)

Figure 9-52: The Line 1, 2-D section is turned off. Note the lack of weak, detailed resistivity highs as indicated by the red arrow. A broad resistivity high exists, but no detail is resolved.



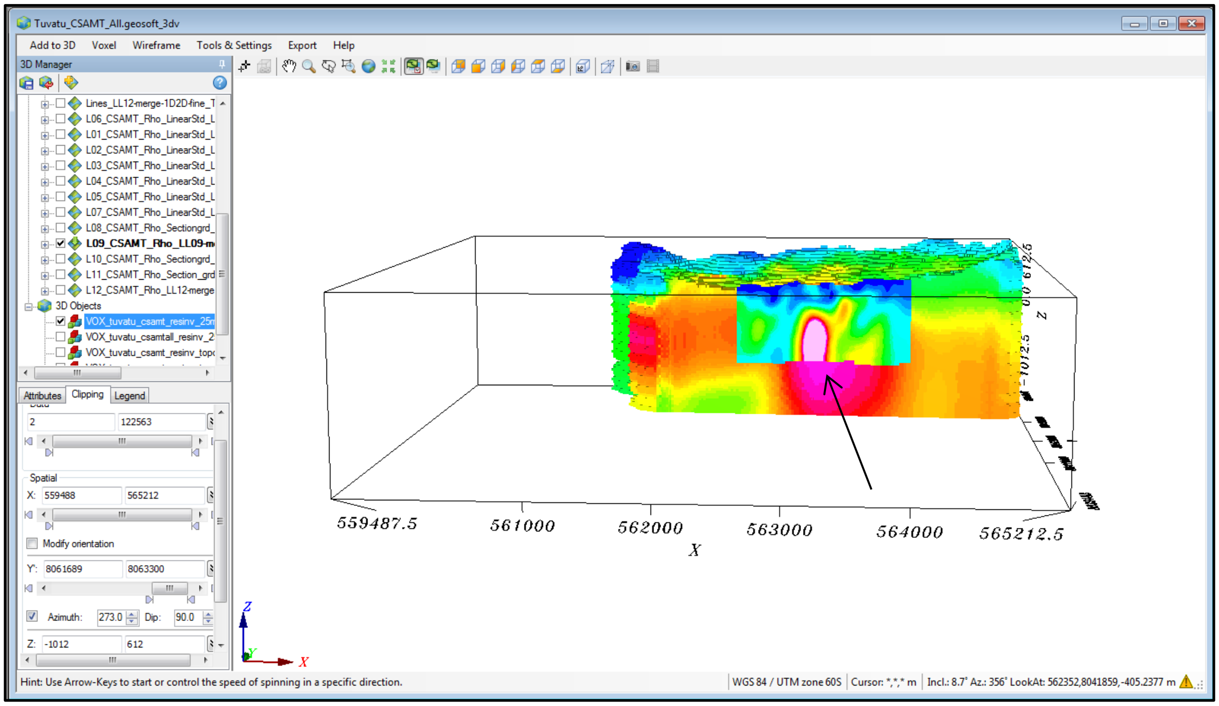
Source: Thomas V. Weis & Associates (2020)

Figure 9-53: The Line 7, with 2-D section shown. Note the detailed resistivity high (red arrow) not observed in the 3-D voxel view in Figure 9-54 with the 2-D section not shown. This illustrates that detailed structures can be interpreted from the 2-D inversion section.



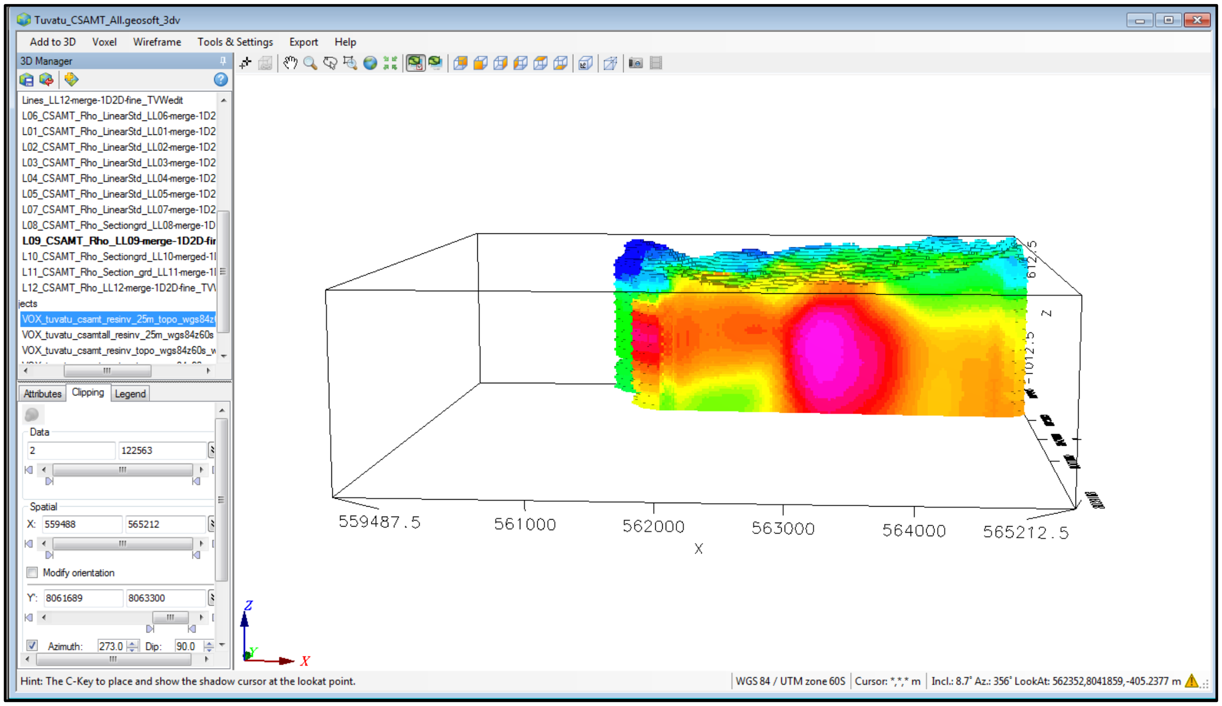
Source: Thomas V. Weis & Associates (2020)

Figure 9-54: The Line 7, 2-D section is not shown. Compared with Figure 9-63, the added detail observed in the 2-D line is replicated in the 3-D model. It should also be noted that on Line 7, a major power line goes through the area resulting in noisy data that is highly edited. Two distinctly different models have been generated, which are shown below.



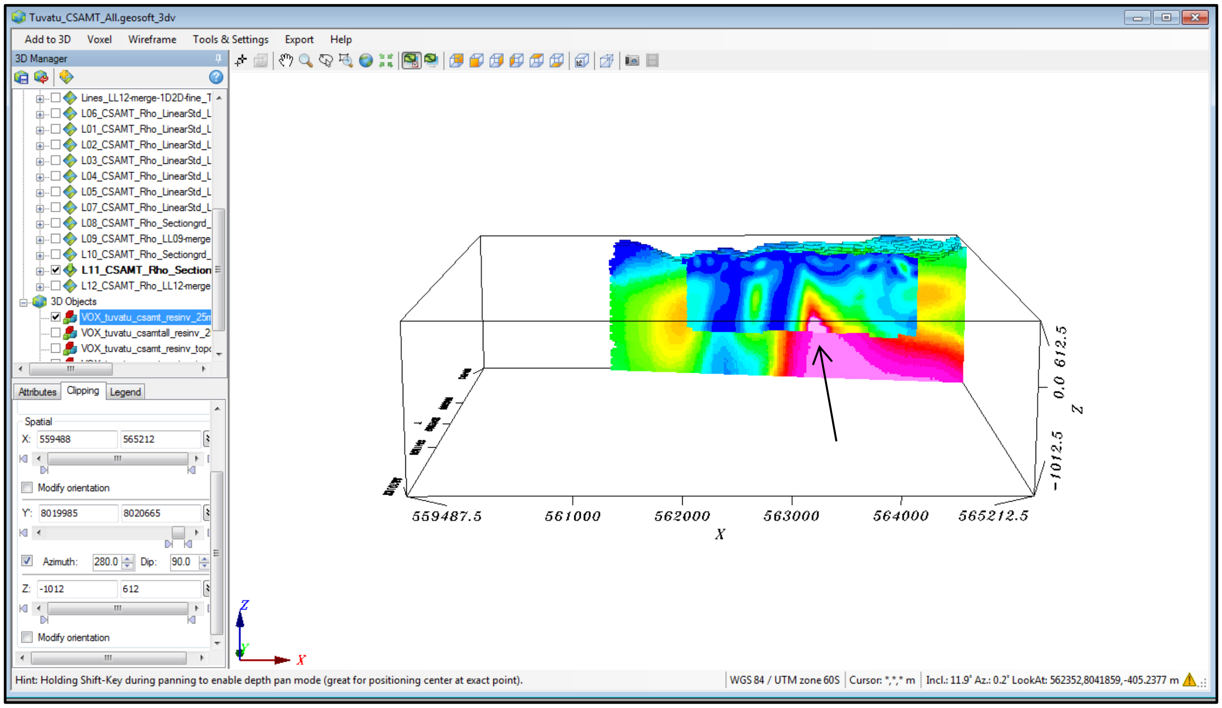
Source: Thomas V. Weis & Associates (2020)

Figure 9-55: The Line 9, 2-D section is shown. Note the high resolution detail in the 2-D inversion section (see black arrow).



Source: Thomas V. Weis & Associates (2020)

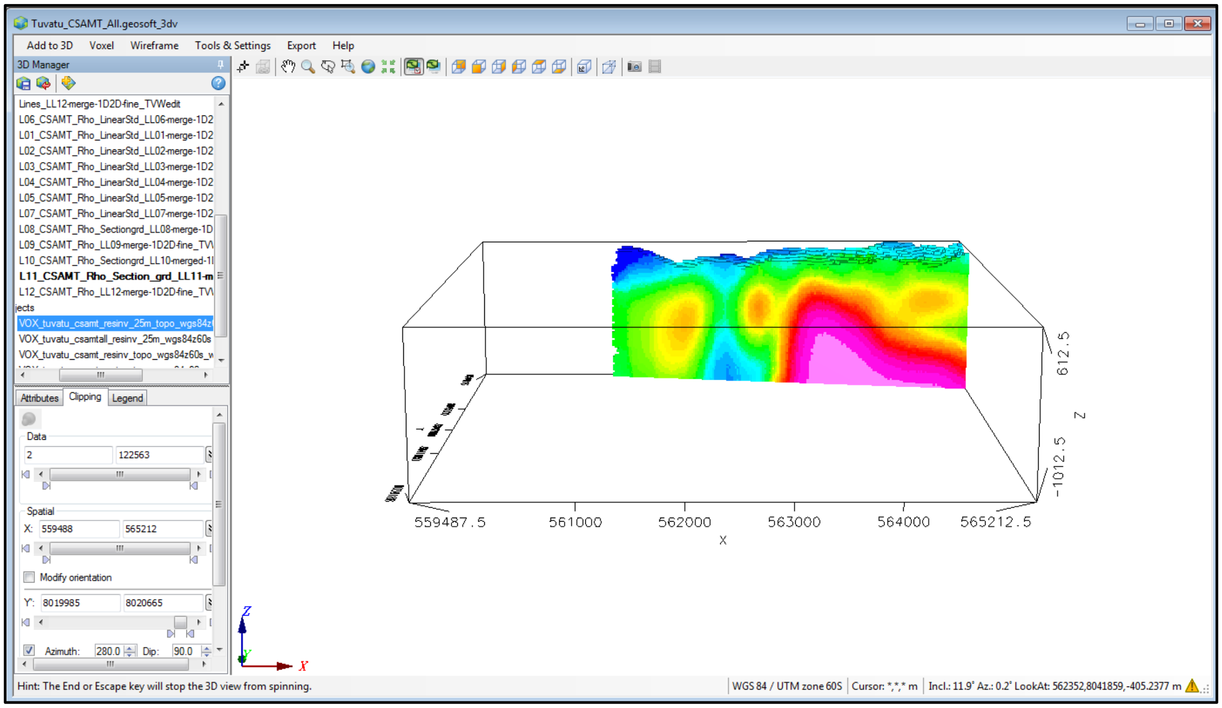
Figure 9-56: The Line 9, 2-D inversion section is not shown. The detailed high resistivity features are not present in the 3-D voxel display.



Source: Thomas V. Weis & Associates (2020)

Figure 9-57: The Line 11, 2-D resistivity inversion section is shown. Note detail indicated by black arrow.

TETRA TECH



Source: Thomas V. Weis & Associates (2020)

Figure 9-58: The Line 11, 2-D resistivity inversion section is not shown. The general shape of the 3-D voxel slice is similar to the 2-D inversion section, but the detail is missing.

CSAMT Conclusions

There is a relationship between structurally controlled edges of alkali monzonitic intrusives and mineralization at the Project in Fiji. This is similar to the situation at the Cripple Creek deposit in Colorado, USA. The CSAMT technique for resistivity mapping identifies these structures and intrusive boundaries which are related to gold mineralization. Furthermore, particularly at the southern prospects (Jomaki-Kubu) and northern prospects (north of Banana Creek), the CSAMT maps the contact boundary between the underlying intrusive complex and the Nadele Breccia/volcanics.

The CSAMT and AMT techniques are merged at Tuvatu to provide increased depth of exploration. This technique seems to be modelled down to approximately 1 km below surface of the combined technique using 2-D inversion modeling and deeper with 1-D inversion modeling.

Twelve lines of CSAMT/AMT were run by Zonge Australia and a total of 47 structural/contact targets have been identified in the data set. The Rx station spacing of 100 m is used, which is coarse for this type of mineral system. A closer spacing in areas of interest is required. If a 3-D combined voxel is used for presentation, the line spacing should be decreased as well. There appears to be a relationship between IP response (sulphides) and mineralization at Tuvatu.

CSAMT Recommendations

Since the line spacing is broad, the proposed targets should be drilled on or close to the survey lines, since the resistivity model from one line to the next varies considerably.

With 100 m Rx station spacing resolution, multiple drill holes may be required to test the 100 m gap between stations when testing the interpreted structure/contact targets.

It is recommended that the CSAMT/AMT survey line spacing be decreased to 200 m.

It is also recommended that the Rx station spacing be decreased to 25 meters in areas of interest. The relationship between the IP response and the mineralized system should be investigated. The 2011 IP response suggests the possibility of a porphyry system at depth.

9.5.5 Prospectivity Analysis

The Tuvatu district comprises outcropping and surface signatures of mineralization over a 7 km x 5 km area. With the exception of the main Project, previous drilling is restricted to only near surface testing of specific lodes. There is considerable untested potential both for feeder zones beneath the Tuvatu Mineral Resource and on the prospects on the wider area.

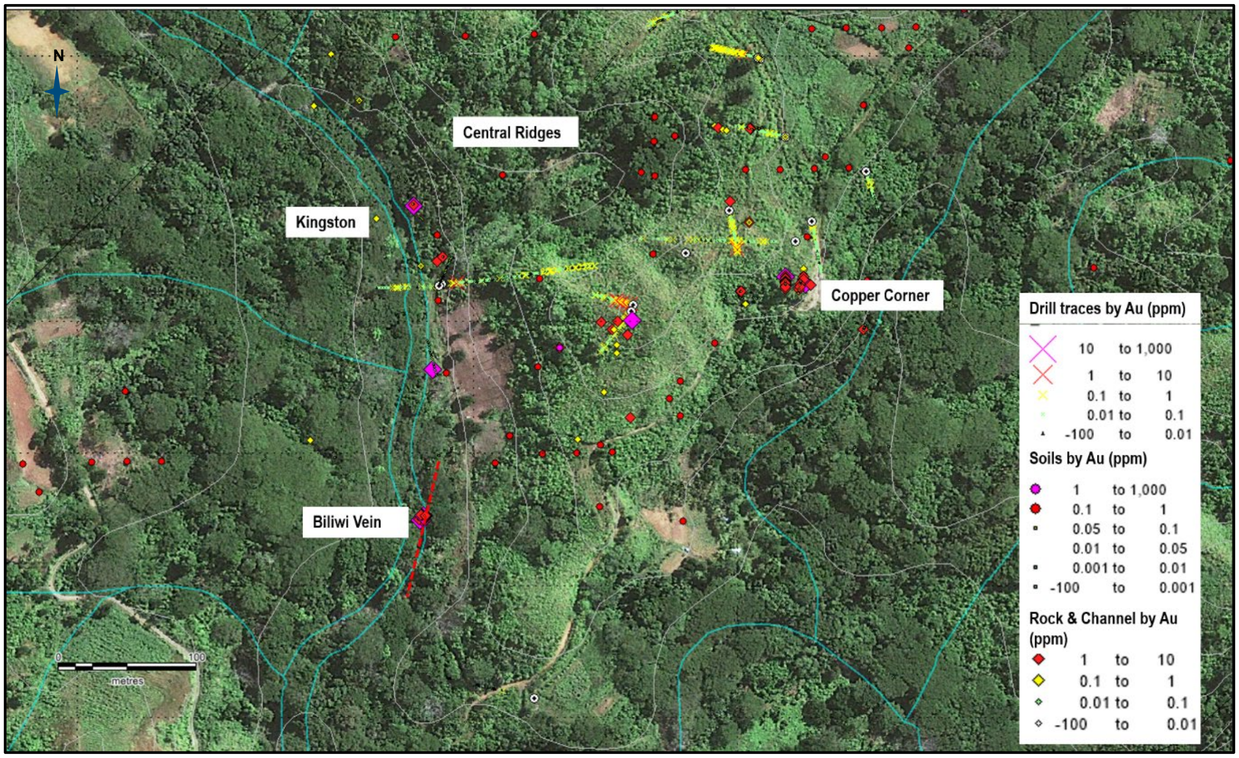
Lion One is well advanced in exploration targeting on this project. There is a considerable body of knowledge, including drill data (Tuvatu resource environs), geochemistry (rock, channels, soils, and stream-cBLEGs), and geophysics (induced polarisation and CSAMT). This is supplemented by local scale mapping of structures through the creek, outcrop, and benching mapping programs.

The first priority targets are directly extensional to the main deposit. The second priority targets are drawn from the extensive anomalism throughout the district and include the Biliwi/Kingston/Central Ridges area, Matanavatu, Banana Creek, Jomaki/Ura/Kubu, and Upper Qalibua areas.

9.5.5.1 Biliwi / Kingston / Central Ridges

This prospect area includes the historic Kingston Mine and is located approximately 1,600 m north of the Project. Between 1900 and 1923, a shaft was dug to 14.6 m and an adit driven 9.1 m in a supergene enriched malachite (copper oxide) deposit at Kingston. The copper at Kingston is hosted in monzonite porphyry and is suggestive that a substantial copper-gold porphyry may exist at depth. Lion One has developed a compilation of historic rock and soil sampling from the Kingston / Central Ridges Prospect area and currently has active programs designed to confirm historic results. The historic results are, however, as reported in previous company reports and cannot be immediately verified for the purposes of this Technical Report.

Rock sample results from Kingston / Central Ridges have returned high-grade gold up to 19.5 g/t, and the surrounding area is characterised with a large gold in soil anomaly (>100 ppb) over 700 m x 500 m (refer to Figure 9-59). The Central Ridges area has been subject to several drilling campaigns, with sixteen diamond core and reverse circulation holes recorded in previous company reports. Drill assay results record up to 1.59 g/t Au and 1.26% Cu. Previous drilling focussed principally on the copper showings, and Lion One considers that several of the gold mineralized structures have not been adequately tested.



Source: Lion One (2020)

Figure 9-59: Kingston/Biliwi soil samples and target area location

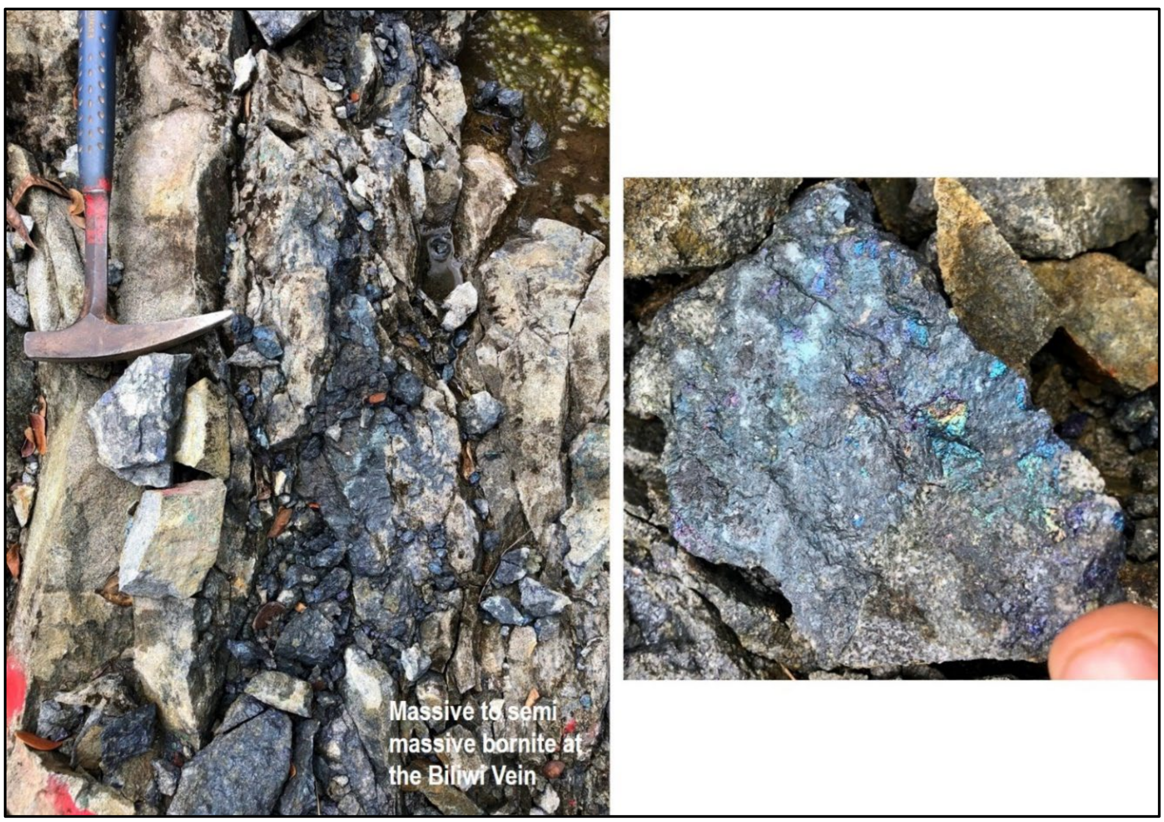
Sabeto floodwaters in 2017/2018 exposed a new outcrop of high-grade massive to semi-massive bornite (copper sulphide) (Figure 9-60) in a structure striking north-northeast and dipping steeply to south-southeast. This vein has been named the Biliwi Vein. The outcrop has been subject to two chip-channels revealing mineralized widths of up to 4 m with a high-grade central area of 0.8 m averaging 77.6 g/t Au and 35% Cu. The 2019/2020 Biliwi Channel sampling results are shown in Table 9-4.

Table 9-4: Biliwi Channel sampling results (2019/2020)

Channel ID	Sample ID	From (m)	To (m)	MidEast (FJM m)	MidNorth (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)	Cu (ppm)
CH1466	TUS011022	0	0.85	1876803	3922541	0.85	0.25	0.17	1,840
CH1466	TUS011023	0.85	1.6	1876802	3922542	0.75	0.6	0.41	2,870
CH1466	TUS011024	1.6	2.4	1876802	3922542	0.8	3.6	1.7	8,120
CH1466	TUS011025	2.4	2.8	1876801	3922542	0.4	105	60.4	263,000
CH1466	TUS011026	2.8	3.2	1876801	3922542	0.4	214	94.8	449,000
CH1466	TUS011027	3.2	3.9	1876800	3922542	0.7	29.5	17.9	51,100
CH1466	TUS011028	3.9	4.55	1876799	3922543	0.65	2.2	1.64	8,970

table continues…

Channel ID	Sample ID	From (m)	To (m)	MidEast (FJM m)	MidNorth (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)	Cu (ppm)
CH1466	TUS011029	4.55	4.85	1876799	3922543	0.3	1.5	0.94	5,290
CH1466	TUS011031	4.85	5.3	1876799	3922543	0.45	0.25	0.39	2,330
CH1467	TUS011032	0	0.6	1876804	3922545	0.6	5.4	3.79	13,050
CH1467	TUS011033	0.6	1	1876804	3922545	0.4	1.4	1.07	7,450
CH1467	TUS011034	1	2	1876803	3922546	1	0.6	0.79	4,620
CH1467	TUS011035	2	2.45	1876802	3922546	0.45	102	83.6	17,4000
CH1467	TUS011036	2.45	3.25	1876802	3922546	0.8	1.5	1.76	7,440
CH1467	TUS011037	3.25	4	1876801	3922546	0.75	1.6	1.25	5,880
CH1467	TUS011038	4	4.5	1876801	3922546	0.5	2.1	1.21	6,400



Source: Lion One (2019)

Figure 9-60: Massive to semi-massive bornite at the Biliwi Vein

Lion One completed four holes, for 421m at Biliwi in 2020-21. The best result received was 0.7 m at 0.95 g/t au and 0.95% Cu (hole TUDDH-501, from 56.64 m). All four of the holes intersected wide background levels of chalcopyrite (1000-2000ppm Cu). Even though the high-grade bornite observed at surface was not replicated in drilling, the

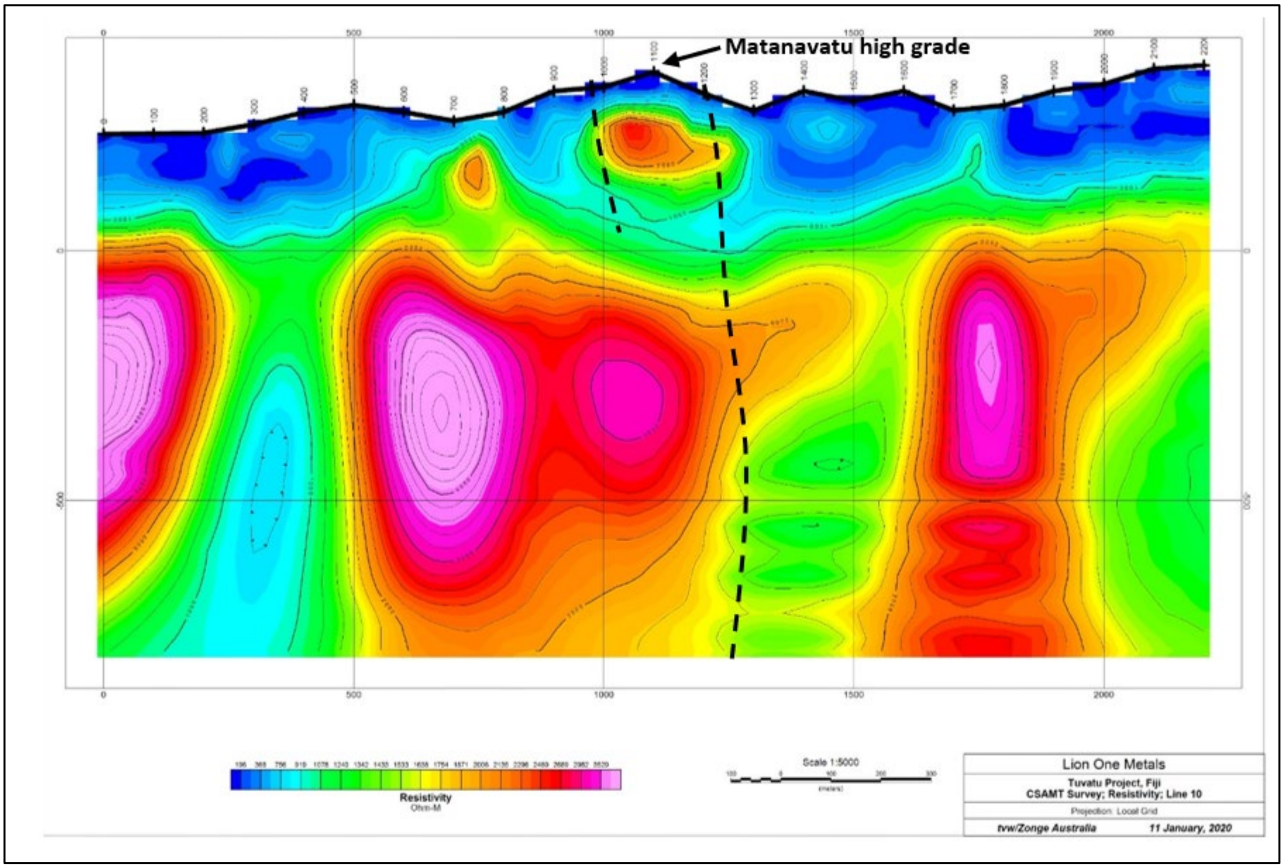
Company geologists believe that the plunge of the lensoidal bodies is yet to be fully understood. A ground based electromagnetic survey to better target massive sulphide bodies at depth has been considered.

9.5.5.2 Matanavatu

The Matanavatu Prospect is located central to the Navilawa Caldera, approximately 1,900 m northeast of the Project. A limited amount of soil sampling has defined a gold anomaly approximately 200 m x 200 m. At least two zones of structures have been exposed during Lion One's benching program with assay results up to 53.2 g/t Au over 0.2 m. Similar to Biliwi, both high-grade gold and copper occur in this project. Table 9-5 shows the Matanavatu channel sampling results (>1 g/t Au) and Figure 9-61 illustrates CSAMT section close to the Matanavatu high-grade structures exposed in benching.

Table 9-5: Matanavatu Channel sampling results >1 g/t Au

Channel ID	Sample ID	From (m)	To (m)	East (FJM m)	North (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)	Cu (ppm)
CH1844	TUS013203	1.1	2.1	1877698.1	3922821.9	1	-	2.86	-
CH1844	TUS013204	2.1	3.1	1877697.2	3922822.5	1	-	1.15	-
CH1793	TUS013041	0.6	1.6	1877583.8	3922772.8	1	-	1.6	-
CH1475	TUS011110	0	0.6	1877695.1	3922826.9	0.6	3.3	4.92	>10,000
CH1475	TUS011111	0.6	1	1877694.6	3922827	0.4	1.5	1.62	>10,000
CH1475	TUS011112	1	1.35	1877694.2	3922827.1	0.35	3.1	1.75	9,300
CH1475	TUS011114	2.1	2.6	1877693.1	3922827.4	0.5	1.3	2.21	4,730
CH1475	TUS011116	2.6	3.3	1877692.5	3922827.5	0.7	1.7	2.08	>10,000
CH1475	TUS011117	3.3	3.9	1877691.9	3922827.7	0.6	2.7	3.71	>10,000
CH1474	TUS011108	0.3	0.5	1877587.4	3922782	0.2	112	53.6	>10,000
CH1474	TUS011109	0.5	0.8	1877587.6	3922782	0.3	17.6	18.15	>10,000
CH1473	TUS011101	0	1	1877587.3	3922782.6	1	2.9	2.15	>10,000
CH1473	TUS011103	1.6	1.95	1877587.2	3922783	0.35	1.3	1.13	9,160
CH1473	TUS011104	1.95	2.6	1877587.2	3922783.2	0.65	42.2	32.3	>10,000
CH1473	TUS011105	2.6	3.05	1877587.1	3922783.4	0.45	0.5	2.35	7,280
CH1473	TUS011106	3.05	4.05	1877587.1	3922783.6	1	0.25	3.09	5,740



Source: Lion One (2020)

Figure 9-61: CSAMT section (view north) close to the Matanavatu high-grade structures exposed in benching

9.5.5.3 Banana Creek

The Banana Creek Prospect is located approximately 3 km northeast of the Project. This prospect has had a considerable amount of work by previous explorers, including soil sampling, rock-chip sampling, and mapping. Lion One has compiled this historic data from reports provided by the Mineral Resources Department of the government of Fiji. Based on a review of the historic data, it is apparent there is a survey error of up to 40 m on the historic data. As such, Lion One is currently in the processing of repeating sampling programs to verify the results of the project area.

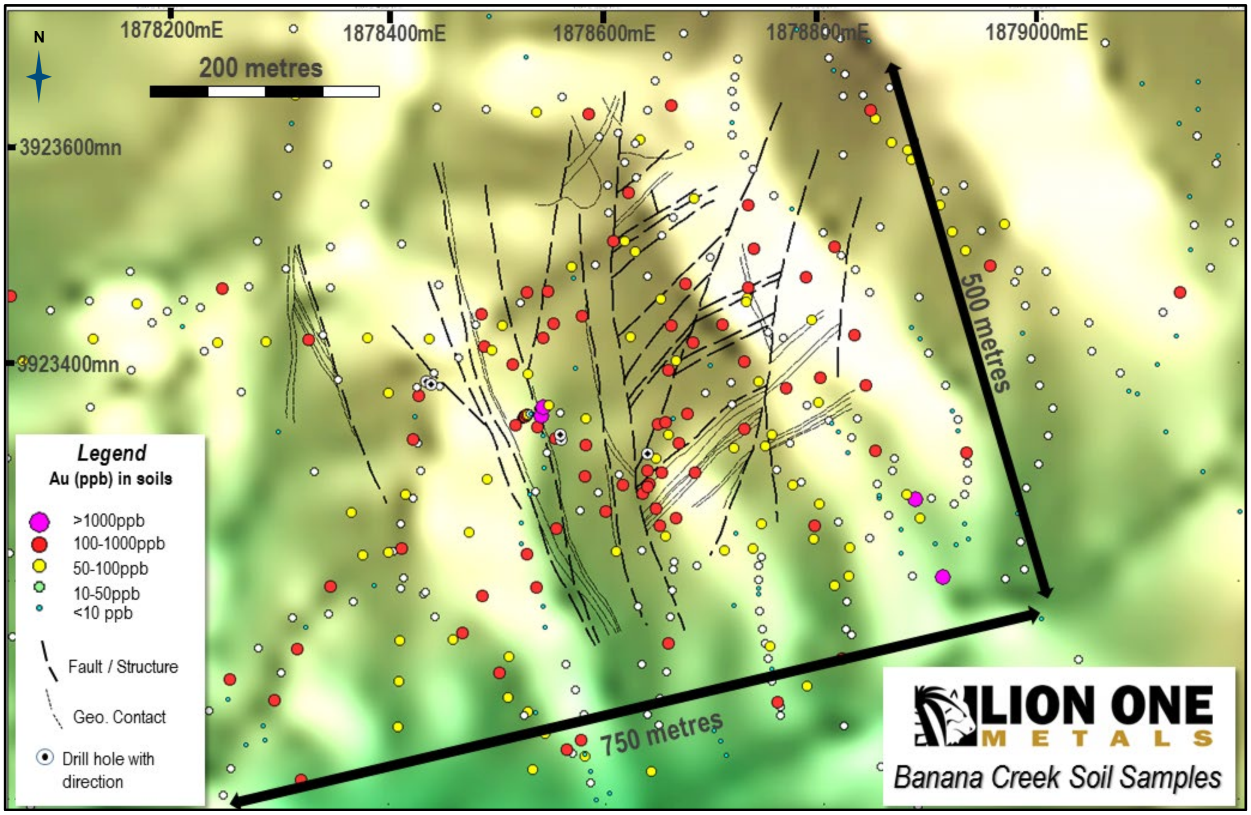
Historic Sampling and Drilling

The Banana Creek Prospect has a long exploration history as outlined in Table 9-6.

Table 9-6: Historic work at Banana Creek

Company	Report Title/Date	Sample Type/Detail	Laboratory/Assay Details
Pan Continental Mining	Reported as historic results compiled in database from Fiji Government **1986–1993**. Reported by Alcaston in 2003 annual report and compiled in databases from Golden Rim mining (2010).	Rock and soil samples. Techniques not disclosed.	Not disclosed in reports.
CRAE	Reported as historic results compiled in database from Fiji Government **1994–1998**. Reported by Alcaston in 2003 annual report and compiled in databases from Golden Rim mining (2010).	Rock and soil samples. Techniques not disclosed.	Not disclosed in reports.
Oribi Resources NL / Mincor Resources NL	Reported as historic results compiled in database from Fiji Government **1999–2002**. Reported by Alcaston in 2003 annual report and compiled in databases from Golden Rim mining (2010).	Five DDHs, HQ size core. Soil sampling using 50 mm augur samples. 'C' horizon soil. Rock channel sampling.	ALS Chemex Laboratory in Brisbane. Gold: Fire Assay 50 g charge (AA26); Multi-element: Aqua regia digestion / atomic emission spectroscopy MEICP-41.
Alcaston Mining NL / Mincor Resources NL	Sabeto Project Special Prospecting License 1412: Annual Reports **2003, 2004, 2005, 2006, 2007** (five reports). Compiled in databases from Golden Rim mining (2010).	Rock chips (2003, 2004) Radiometrics / magnetics 2007	ALS Chemex Laboratory in Brisbane. Gold: Fire Assay 50 g charge (AA26); Multi-element: Aqua regia digestion / atomic emission spectroscopy MEICP-41.
Golden Rim Ltd / Mincor Resources N:	Sabeto Project Special Prospecting License 1412: Annual Reports **2008, 2009, 2010** (three reports).	Rock chips (2008)	Not disclosed in reports but assumed equivalent to above 2003–2007 annual reports.

Reconnaissance has included several mapping, soil, and rock sampling programs. The prospect is defined as a broad soil anomaly over an area of 500 m x 750 m (Figure 9-62).

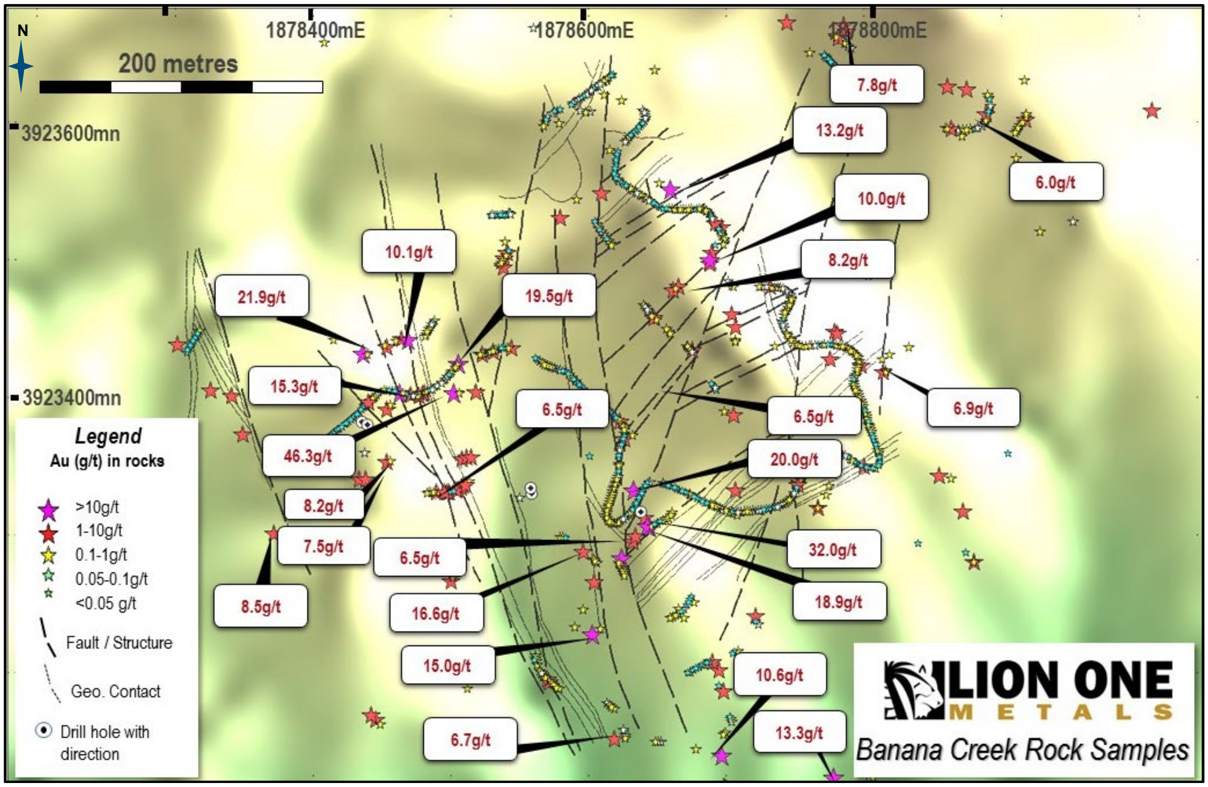


Source: Lion One (2019)

Figure 9-62: Historic soil sampling results from Banana Creek

As well as the soil sampling, rock samples with assays up to 46.6 g/t Au are presented in historic data, and these results are summarised in Table 9-7 and Figure 9-63.

A total of five diamond core holes, to a maximum depth of 116.5 m, have been completed by previous explorers. These have shown anomalous results, though upon assessment only BCDH4 and BCDH5 were drilled at an appropriate angle to structure (Table 9-8 and Figure 9-64).



Source: Lion One (2019)

Figure 9-63: Historic rock sampling results from Banana Creek

Table 9-7: Historic rock sampling at Banana Creek sorted by gold grade from various reports as compiled by Alcaston Mining and Golden Rim Mining

Easting (m)	Northing (m)	Sample Type	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
1878501.95	3923403.66	Rock	46.3	14.4	146	1,110	276
1878642.91	3923306.93	Rock	32	20.9	579	8,130	1,860
1878436.26	3923432.09	Rock unclassified	21.9	-	130	5,000	335
1878632.66	3923332.84	Rock	20	11.6	234	2,000	1,185
1878505.75	3923424.71	Rock	19.45	3.8	199	605	357
1878643.91	3923305.94	Channel	18.9	20.1	413	671	1,340
1878624.05	3923283.25	Rock unclassified	16.6	-	115	1,500	110
1878463.08	3923403.5	Rock unclassified	15.3	3	85	365	60
1878602.42	3923228.14	Rock unclassified	15	3	30	60	55
1878777.77	3923124.34	Rock unclassified	13.3	5	85	370	35

table continues…

Easting (m)	Northing (m)	Sample Type	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
1878659.21	3923551.47	Rock unclassified	13.2	-	125	550	55
1878696.47	3923140.36	Rock	10.6	1.5	65	930	17
1878468.64	3923442.12	Rock unclassified	10.05	1	45	30	15
1878688.04	3923499.74	Rock unclassified	10	2	215	1,800	390
1878371.68	3923301.29	Rock unclassified	8.55	-	25	135	15
1878596.3	3923287.85	Rock unclassified	8.3	2	165	1,980	120
1878661.37	3923478.24	Rock unclassified	8.2	-	115	320	65
1878453.76	3923353.73	Rock unclassified	8.2	-	75	40	60
1878785.37	3923670.21	Rock unclassified	7.75	110	50	120	20
1878626.05	3923370.31	Rock unclassified	7.6	-	80	390	240
1878438.07	3923340.62	Rock unclassified	7.5	-	45	220	75
1878813.94	3923418.56	Channel	6.86	2.5	166	49	49
1878618.66	3923152.15	Rock unclassified	6.7	-	570	205	360
1878503.9	3923332.64	Rock unclassified	6.5	5	145	1,650	**3,350**
1878634.25	3923299.73	Rock unclassified	6.5	-	395	3,850	340
1878813.94	3923418.56	Rock	6.23	1.3	52	53	21
1878453.56	3923436.63	Rock unclassified	6.2	bd	145	960	50
1878888.23	3923605.73	Rock unclassified	6	2	370	80	210
1878632.14	3923295.78	Rock unclassified	5.97	-	215	400	185
1878508.28	3923355.51	Rock unclassified	5.3	-	40	150	65
1878505.77	3923333.48	Rock unclassified	5.12	bd	240	790	475
1878476.59	3923401.37	Rock unclassified	5.09	bd	135	315	210
1878665.11	3923480.73	Rock unclassified	4.94	-	290	160	100
1878522.69	3923431.07	Rock unclassified	4.93	bd	170	890	695
1878699.66	3923116.19	Rock unclassified	4.81	1	280	275	355
1878301.32	3923438.79	Rock unclassified	4.78	-	280	430	145
1878603.86	3923266.32	Rock unclassified	4.73	bd	505	275	1,950
1878507.42	3923334.58	Rock unclassified	4.6	bd	255	55	355
1878743.85	3923672.77	Rock unclassified	4.36	60	65	45	15
1878721.54	3923241.63	Channel	4.31	1.1	203	180	213

table continues…

Easting (m)	Northing (m)	Sample Type	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
1878675.17	3923434.39	Rock unclassified	4.1	-	350	445	310
1878579.53	3923530.93	Rock unclassified	4.09	-	50	230	60
1878780.71	3923446.82	Rock unclassified	3.99	-	120	50	25
1878787.38	3923665.86	Rock unclassified	3.98	**160**	55	190	30
1878797.64	3923423.65	Rock unclassified	3.95	bd	125	2,400	105
1878872.19	3923317.57	Rock unclassified	3.9	-	90	130	50
1878647.69	3923458.12	Rock unclassified	3.9	-	380	370	205
1878464.31	3923442.91	Rock unclassified	3.57	-	40	350	40
1878698.29	3923186.61	Rock unclassified	3.46	-	720	**21,000**	400
1878494.3	3923049.58	Rock unclassified	3.35	-	195	2,900	515
1878689.28	3923505.2	Rock unclassified	3.33	bd	250	390	160
1878706.5	3923332.69	Rock unclassified	3.05	bd	190	1,450	145
1878860.85	3923626.02	Rock	2.93	bd	60	125	15
1878874.45	3923624.11	Rock unclassified	2.93	bd	60	125	15
1878722.42	3923692.65	Rock unclassified	2.89	-	290	1,500	280
1878471.07	3923401.95	Rock unclassified	2.87	bd	345	170	545
1878544.17	3923435.94	Rock unclassified	2.87	bd	315	50	330
1878481.04	3923403.29	Rock unclassified	2.86	bd	235	800	560
1878696.57	3923523.68	Rock unclassified	2.82	-	50	850	70
1878879.85	3923281.12	Rock	2.79	1.9	237	197	83
1878698.86	3923117.8	Rock unclassified	2.7	bd	560	11,400	520
1878509.07	3923335.78	Rock unclassified	2.68	3	200	480	375
1878514.95	3923357.08	Rock unclassified	2.66	-	85	1,350	570
1878705.63	3923387.25	Rock unclassified	2.64	bd	115	70	120
1878498.77	3923331.41	Rock unclassified	2.6	bd	115	80	275
1878571.55	3923193.23	Rock unclassified	2.35	3	630	15,500	535
1878512.94	3923071.1	Rock unclassified	2.26	2	**1,180**	5,400	1,200
1878704.26	3923460.9	Rock unclassified	2.17	bd	210	135	170
1878694.68	3923202.44	Rock unclassified	2.13	-	615	790	860
1878340.61	3923401.23	Rock unclassified	2.08	-	35	50	80

table continues…

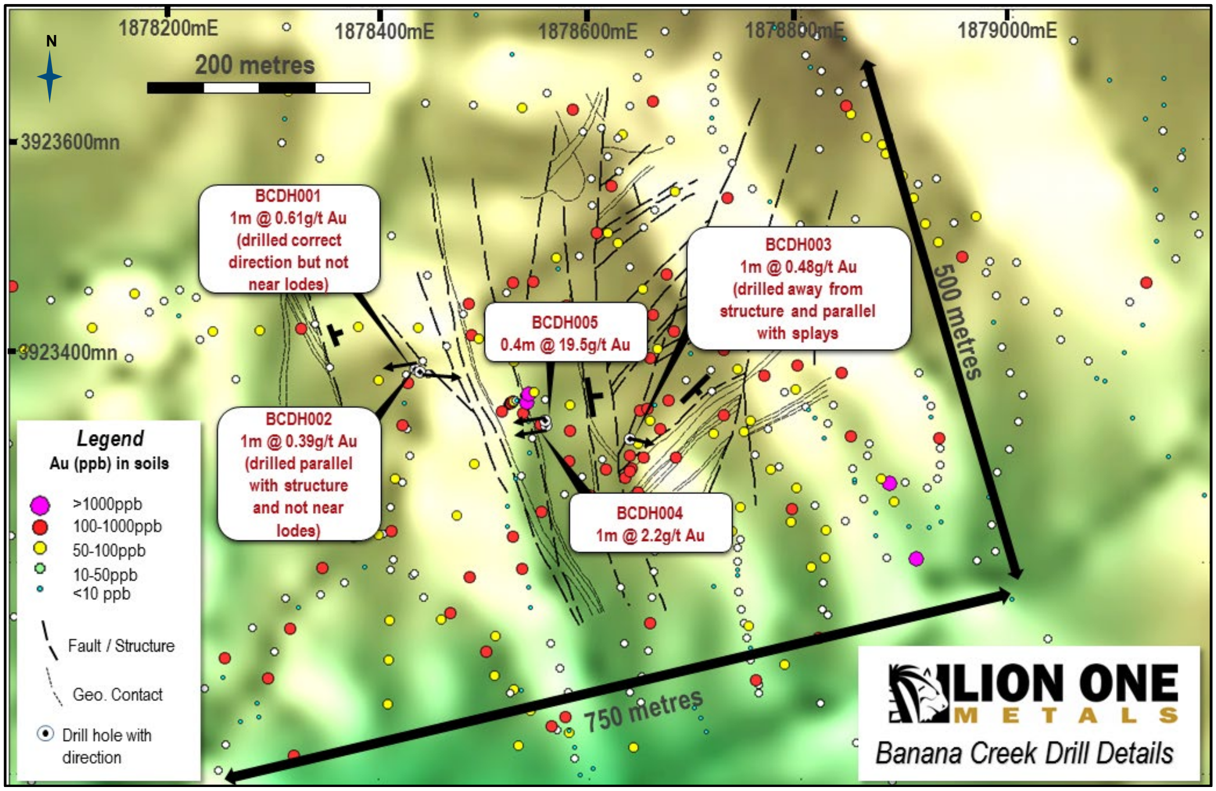
Easting (m)	Northing (m)	Sample Type	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
1878479.37	3923402.63	Rock unclassified	2.02	bd	215	200	420
1878489.76	3923331.25	Rock unclassified	1.9	3	270	920	515
1878530.54	3923433.47	Rock unclassified	1.89	bd	235	710	250
1878778.88	3923447.97	Rock unclassified	1.88	-	215	55	100
1878517.95	3923403.81	Rock	1.87	0.9	115	361	225
1878541.07	3923505	Rock unclassified	1.85	bd	310	20	160
1878879.85	3923281.12	Rock	1.85	1.3	155	89	53
1879008.91	3923608.39	Rock unclassified	1.85	4	85	50	125
1878570.39	3923195.49	Rock unclassified	1.81	bd	485	2,500	750
1878453.2	3923391.47	Rock unclassified	1.75	-	255	660	525
1878538.1	3923494.44	Rock unclassified	1.73	-	40	50	50
1878687.92	3923496.99	Rock unclassified	1.71	bd	160	130	40
1878445.39	3923166.18	Rock unclassified	1.69	-	30	1,500	190
1878499.91	3923267.09	Rock unclassified	1.65	1	215	35	200
1878325.29	3923405.65	Rock unclassified	1.63	-	50	450	235
1878766.84	3923320.08	Rock	1.62	1.4	178	36	115
1878916.97	3923601.92	Rock unclassified	1.6	-	125	1,450	115
1878348.93	3923373.51	Rock unclassified	1.6	-	35	140	60
1878757.63	3923350.21	Rock unclassified	1.54	bd	245	460	280
1878800.01	3923350.11	Rock unclassified	1.54	-	-	-	-
1878538.26	3923081.13	Rock unclassified	1.54		485	4,500	440
1878731.67	3923480.39	Rock unclassified	1.53	bd	270	260	290
1878816.27	3923133.93	Rock unclassified	1.53	2	250	65	65
1878511.76	3923356.5	Rock unclassified	1.53	-	120	890	595
1878609.12	3923548.51	Rock unclassified	1.52	-	140	340	90
1878775.38	3923428.16	Rock unclassified	1.5	-	285	1,550	275
1878623.05	3923382.72	Rock unclassified	1.49	1	265	45	650
1878515.57	3923068.33	Rock unclassified	1.468	5	640	7,200	665
1878538.74	3923501.57	Rock unclassified	1.45	1	220	-5	130
1878497.35	3923331.39	Rock unclassified	1.43	bd	185	90	290

table continues…

Easting (m)	Northing (m)	Sample Type	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
1878640.53	3923312.71	Rock unclassified	1.4	-	165	330	165
1878433.76	3923340.82	Rock unclassified	1.4	-	165	65	445
1878854.61	3923343.56	Rock unclassified	1.32	-	65	135	45
1878691.77	3923526.87	Rock unclassified	1.3	5	270	150	420
1878863.27	3923595.13	Rock unclassified	1.24	bd	300	180	495
1878495.29	3923331.19	Rock unclassified	1.24	1	195	175	450
1878689.84	3923208.32	Channel	1.18	0.5	213	88	193
1878752.06	3923336.53	Rock unclassified	1.119	bd	330	10	170
1878440.4	3923396.71	Rock unclassified	1.04	bd	340	350	630
1878781.55	3923660.96	Rock unclassified	1.03	6	185	295	50
1878442	3923169.5	Rock unclassified	1.029	-	170	1,250	375
1878706.5	3923451.55	Rock unclassified	1.012	-	360	35	215
1878522.17	3923064.1	Rock unclassified	1.004	-	150	1,750	405
1878510.49	3923336.91	Rock unclassified	1.002	bd	215	90	630

Table 9-8: Historic drilling at Banana Creek with peak gold values

Hole ID	East (FJM m)	North (FJM m)	Azimuth (deg from true north)	Dip (deg)	Hole Depth (m)	Intercept From (m)	Intercept To (m)	Peak Assay (g/t Au)
BCDH1	1878435.31	3923382.39	315	-60	101.1	10	11	0.61
BCDH2	1878439.45	3923379.97	135	-60	100.4	58	59	0.4
BCDH3	1878637.91	3923316.77	135	-60	100.6	1	2	0.5
BCDH4	1878559.04	3923329.88	337	-60	102.6	32	32.4	19.5
BCDH5	1878558.18	3923334.4	323	-75	116.5	115	116	2.16



Source: Lion One (2019)

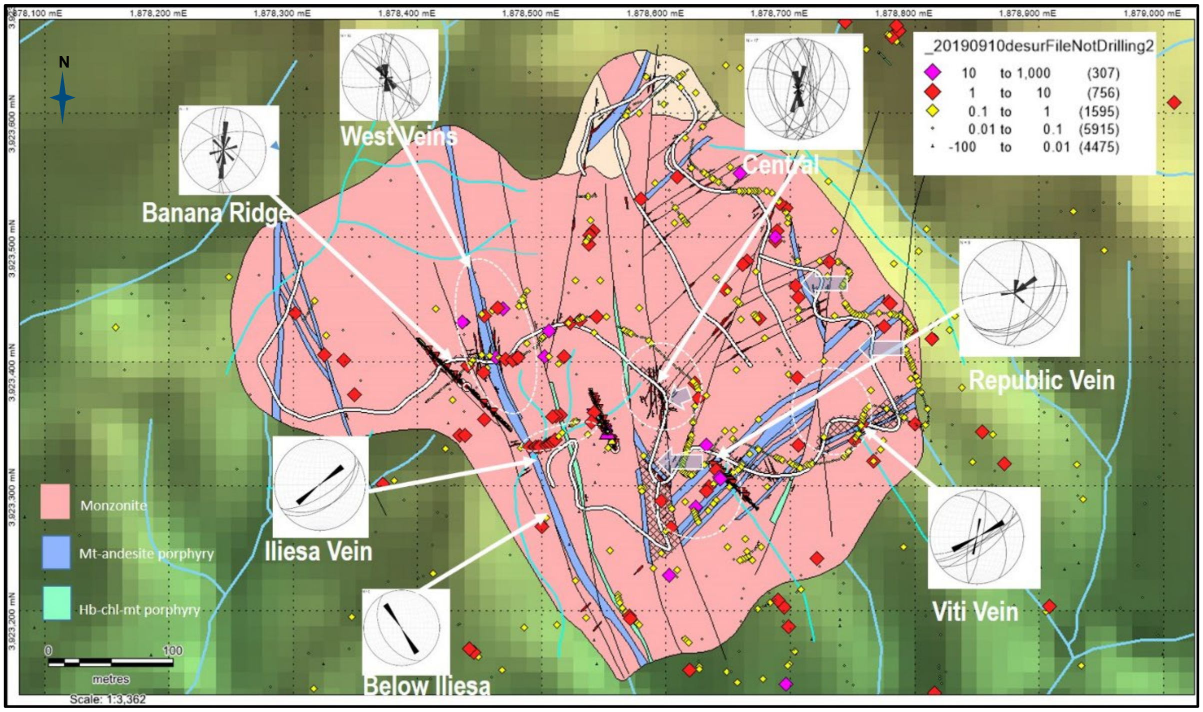
Figure 9-64: Banana Creek historic drilling. Holes BCDH001 to 003 were drilled sub-parallel to structure and hence true width is not determined. Holes BCDG004 to 005 were drilled across structure, and true width is estimated at 80% of drill widths.

Banana Creek – Lion One Work 2018/2019

Since gaining access to the Navilawa Caldera, Lion One has conducted its own reconnaissance and sampling verification program at the Banana Creek Prospect area. In addition, two CSAMT lines were completed in the Banana Creek area.

The Banana Creek Prospect is hosted in a monzonite, which is transected by north–south andesite dykes. Mineralization is oriented predominantly in steeply dipping north-northwest to south-southeast striking structures and in 40 to 70 degree southeast dipping, northeast–southwest striking structures (Figure 9-65). The prospect includes multiple named veins, including the Republic Vein, Banana Creek West Veins, Iliesa Vein (Figure 9-66), and the Viti Vein. On field reconnaissance, visible gold was located in the outcrop of the Iliesa Vein, and the Republic Vein has a strong association with manganese oxide.

The Republic Vein (Figure 9-67), Banana Creek West veins (Figure 9-68), Iliesa Vein (Figure 9-66) have all been exposed by Lion One geology team with check sampling verifying the historically reported results. A table of Lion One's verification bench / channel sampling from 2019 to 2020 is shown in Table 9-9.

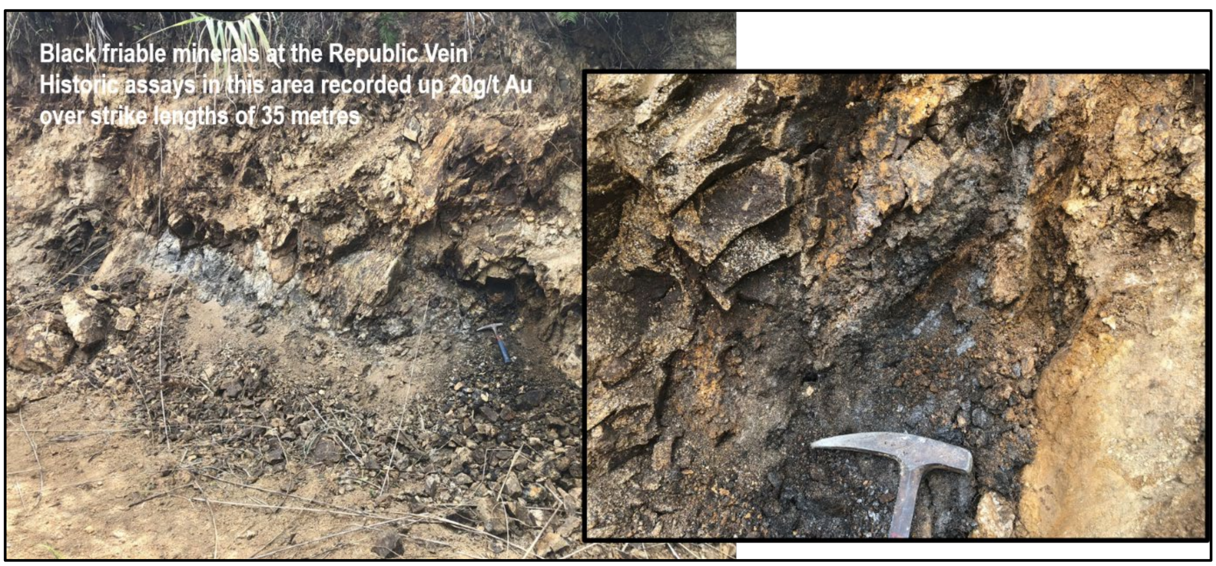


Source: Lion One (2019)

Figure 9-65: The Banana Creek Prospect with local scale geology map and major structures labeled. Historic rock and channel sampling grades show with diamond symbol insets are stereonets of Lion One mapping in the area.

Source: Lion One (2019)

Figure 9-66: Iliesa Vein at Banana Creek



Source: Lion One (2019)

Figure 9-67: Manganese alteration at the Republic Vein

Source: Lion One (2019)

Figure 9-68: Banana Creek West Vein, exposed in benching by Lion One

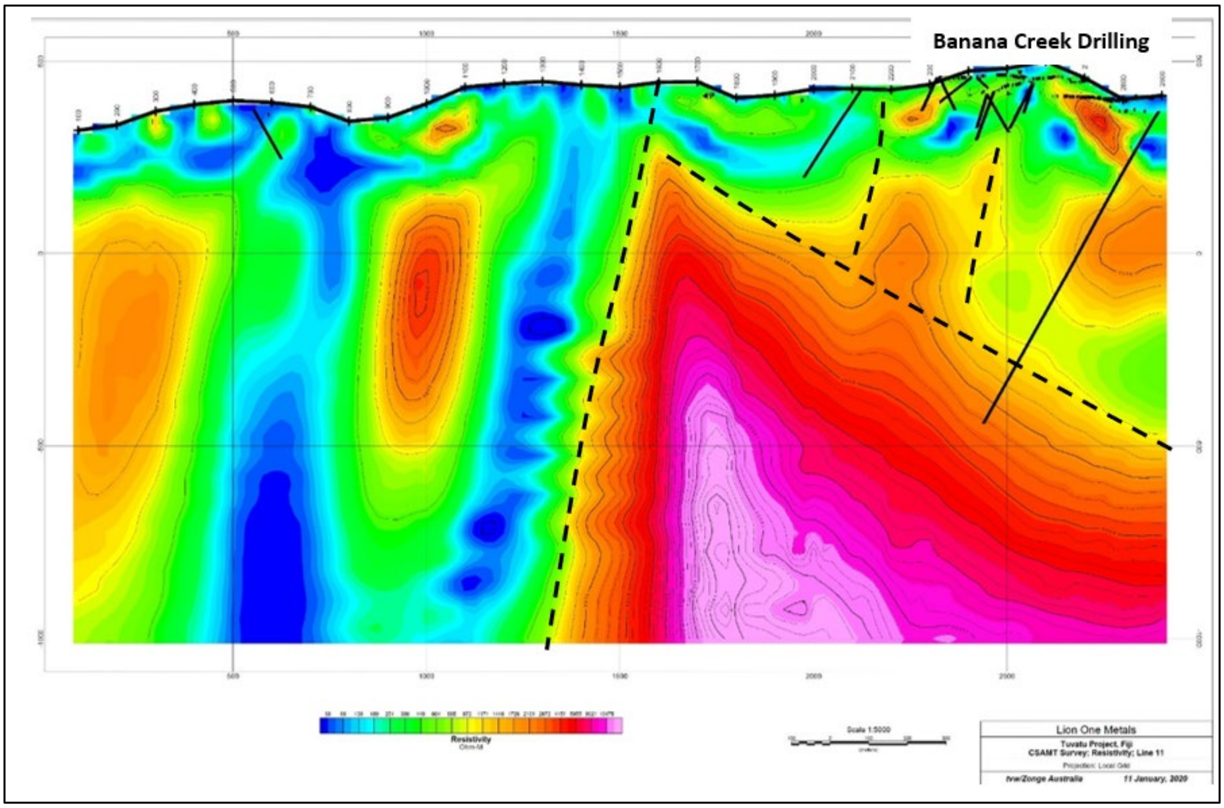
Table 9-9: Banana Creek Channel sampling (>1 g/t Au) by Lion One

Hole ID	Sample ID	From (m)	To (m)	East (FJM m)	North (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)	Cu (ppm)
CH1872	TUS013442	2.07	2.57	1878656.1	3923368.4	0.5	-	2.09	-
CH1871	TUS013437	2.16	2.46	1878655.8	3923371.9	0.3	-	1.07	-
CH1867	TUS013412	0.7	1.4	1878645.3	3923327.9	0.7	-	1.69	-
CH1867	TUS013416	2.6	3.5	1878645.8	3923327.8	0.9	-	5.37	-
CH1866	TUS013409	0.7	1.3	1878650.3	3923325	0.6	-	39.74	-
CH1865	TUS013402	0	0.8	1878656.8	3923328.9	0.8	-	2.43	-
CH1865	TUS013407	4.7	5.6	1878653.8	3923327.8	0.9	-	4.82	-
CH1586	TUS012003	1.6	1.85	1878817.5	3923242	0.25	-0.2	3.23	383.6
CH1584	TUS011842	0.65	0.95	1878909.9	3923260.8	0.3	6.16	50.94	610.4
CH1584	TUS011843	0.95	1.5	1878910.3	3923260.7	0.55	-0.2	2.2	207.6
CH1577	TUS011803	0.65	0.93	1878697.8	3923452.4	0.28	2.2	3.71	389
CH1576	TUS011749	0.8	1.2	1878740.7	3923536	0.4	5.4	1.64	591
CH1575	TUS011746	0.6	0.85	1878761	3923479.3	0.25	1.8	4.66	170
CH1574	TUS011743	1.45	1.9	1878849.4	3923344.9	0.45	30.7	5.00	146
CH1572	TUS011736	1	1.25	1878703.4	3923324.2	0.25	4.5	5.19	1025
CH1571	TUS011727	0	1	1878682.5	3923309.3	1	6.5	2.14	366
CH1571	TUS011728	1	2	1878683.1	3923310.1	1	9.5	10.83	295
CH1571	TUS011731	3	4	1878684.4	3923311.6	1	2	4.02	451
CH1571	TUS011732	4	4.7	1878684.9	3923312.3	0.7	4.4	5.72	600
CH1571	TUS011733	4.7	5.4	1878685.4	3923312.9	0.7	2.0	1.00	478
CH1571	TUS011734	5.4	6	1878685.8	3923313.4	0.6	0.25	1.02	293
CH1461	TUS010991	0.25	0.53	1878576.5	3923331.5	0.28	5.2	7.68	158
CH1460	TUS010988	0.5	0.95	1878676.5	3923303.3	0.45	19.4	35.3	616

Drill Targeting at Banana Creek

Banana Creek has a reasonably high elevation within the Caldera at 480 mASL (compared to Tuvatu surface at surface 200 to 240 m AWSL). There is also a strong silver association, along with observed manganese with gold mineralization at Banana Creek (Figure 9-67). These observations, perhaps, present Banana Creek at also a higher position (metallogenically) in the overall system. The previous drilling, whilst identifying a potential economic intersection in BCDH19 (0.4 m at 19.5 g/t Au) did not target beneath a 100 m depth and did not perhaps intersect an alkaline-gold system beneath the near surface. The CSAMT at Banana Creek (Figure 9-69) indicates a series of significant gradient changes in resistivity at between 200 and 500 m depth.

Banana Creek is clearly a significant mineralizing cell with a large footprint and multiple mineralized structures.



Source: Thomas V. Weis & Associates (2020)

Figure 9-69: Banana Creek CSAMT section (2019/Line11) view north

In 2021 the Company completed 5 holes for 1859 metres of drilling at Banana Creek (Table 9-12). The best intersection returned 0.5m at 4.17g/t Au from 463.5m (Table 9-13). Further work is required to understand structural intersections and to target Banana Creek at depth.

Table 9-10: Banana Creek drill collar table by Lion One

HOLEID	EAST	NORTH	RL	DEPTH	AZIMUTH_TN	DIP
TUDDH-503A	1878537	3923433	456.545	117.17	252	-35
TUDDH-507	1878548	3923436	456.873	83.54	61.2505	-34.5822
TUDDH-508	1878671	3923293	434.083	200	348	-35
TUDDH-536	1878247	3923443	422.351	519.9	299.48	-53.41
TUDDH-543	1878991	3923187	369.266	938.8	287.57	-58.16

Table 9-11: Assay results >1g/t Au from Banana Creek 2021 drilling

HOLEID	EAST	NORTH	RL	DEPTH	SAMPFROM	SAMPTO	INTERVAL_M	Au_ppm_BEST
TUDDH-536	1878247	3923443	422	519.9	463.5	464	0.5	4.17
TUDDH-503A	1878537	3923433	457	117.17	27.7	28.2	0.5	3.1
TUDDH-507	1878548	3923436	457	83.54	19.9	20.5	0.6	3.01
TUDDH-503	1878537	3923433	457	40.31	28.3	28.85	0.55	2.01
TUDDH-543	1878991	3923187	369	938.8	593.3	593.9	0.6	1.97
TUDDH-536	1878247	3923443	422	519.9	458	458.5	0.5	1.56
TUDDH-508	1878671	3923293	434	200	24.15	24.45	0.3	1.32
TUDDH-503A	1878537	3923433	457	117.17	66.12	67.03	0.91	1.3
TUDDH-543	1878991	3923187	369	938.8	904.3	904.6	0.3	1.26
TUDDH-543	1878991	3923187	369	938.8	593.9	594.2	0.3	1.23
TUDDH-508	1878671	3923293	434	200	108.08	108.45	0.37	1.09
TUDDH-503A	1878537	3923433	457	117.17	63.17	63.58	0.41	1.06

9.5.5.4 Jomaki/Kubu/Ura

The Jomaki/Kubu/Ura Creek target areas are located approximately 2 km southwest of the Project. The prospects cover steep terrain (for example see Figure 9-70) and consist of a series of surface mineralized structures hosted primarily in Nadele Breccia / volcanics and extend over an area approximately 1,000 m x 850 m (Figure 9-3, Figure 9-71). Surface results, in places, have been very high grade with results recorded up to 0.6 m at 502 g/t Au at Jomaki in weathered rock (Table 9-12).

Several monzonite dykes are mapped and interpreted throughout this prospect area. However, in places, the Nadele Breccia is highly bleached, and some of the interpretations of surface monzonite may be a result of mis-identifying monzonite from highly bleached mafic volcanics.

As well as considerable surface rock sampling anomalism, Lion One has conducted extensive benching and channel sampling. A total of eleven diamond core holes have been drilled to an average depth of 125 m, and a maximum depth of 269 m. Whilst the drilling did not intersect appreciable mineralization (Table 9-13), the drilling was conducted primarily through Nadele Breccia, with only narrow intersections of monzonite dykes. Indeed, the prevalence of narrow monzonite dykes (1 to 3 m wide) increases with depth, for example, in the Jomaki area.

CSAMT geophysics completed in 2019 suggests that the principal monzonite body lies beneath the volcanics at a depth of 250 to 300 m. Whilst it is geologically possible for the Nadele Breccia to host mineralization, it is without a doubt that the best mineralization in the district is hosted in monzonite. As such, the previous drilling did not intersect structures hosted in the monzonite pluton, and the surface results are considered partially enriched due to surface weathering, but indicative of structures at depth (refer to Figure 9-72).

Table 9-12: Jomaki-Ura-Kubu area channel sampling results >1 g/t Au

Channel ID	Sample ID	From (m)	To (m)	East (FJM m)	North (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)
CH519	TUS006863	0.6	1.3	1875277.4	3919162.5	0.7	92.9	502
CH855	TUS008185	0.7	0.9	1875488.5	3919674.5	0.2	27.1	152
CH630	TUS007402	0.65	0.9	1875254.2	3919711.1	0.25	100	53
CH335	TUS005775	1.6	2.6	1875267	3919763.1	1	46.5	15.7
CH798	TUS008003	0.95	1.2	1875594	3919483.5	0.25	17	11.6
CH225	TUS005112	3.74	4	1875530.3	3918911.3	0.26	11.1	10.65
CH713	TUS007785	1.2	1.5	1875429.8	3918858.6	0.3	20.6	10.65
CH694	TUS007699	0.7	1	1875388.9	3919262	0.3	9.1	9.86
CH859	TUS008190	1	1.71	1875474	3919689.9	0.71	7.7	8.72
CH671	TUS007572	0.5	1	1875261.1	3919063.9	0.5	26.4	7.43
CH636	TUS007422	0.5	0.8	1875123.8	3919570.6	0.3	75.9	6.75
CH804	TUS008025	0.7	0.95	1875669.8	3919537.1	0.25	16.1	6.62
CH633	TUS007412	0.7	1.4	1875268	3919754.4	0.7	60.5	6.22
CH1171	TUS009788	0.55	0.95	1875256.9	3919170	0.4	4.4	6.17
CH263	TUS005324	0.7	1.1	1875295.1	3919191	0.4	11.1	5.68
CH221	TUS005187	0.5	0.68	1875528.5	3918964.2	0.18	4.1	5.21
CH332	TUS005766	2	3	1875273.6	3919805.9	1	5.8	4.93
CH223	TUS005195	0.5	0.6	1875536.6	3919002	0.1	4.7	4.84

table continues…

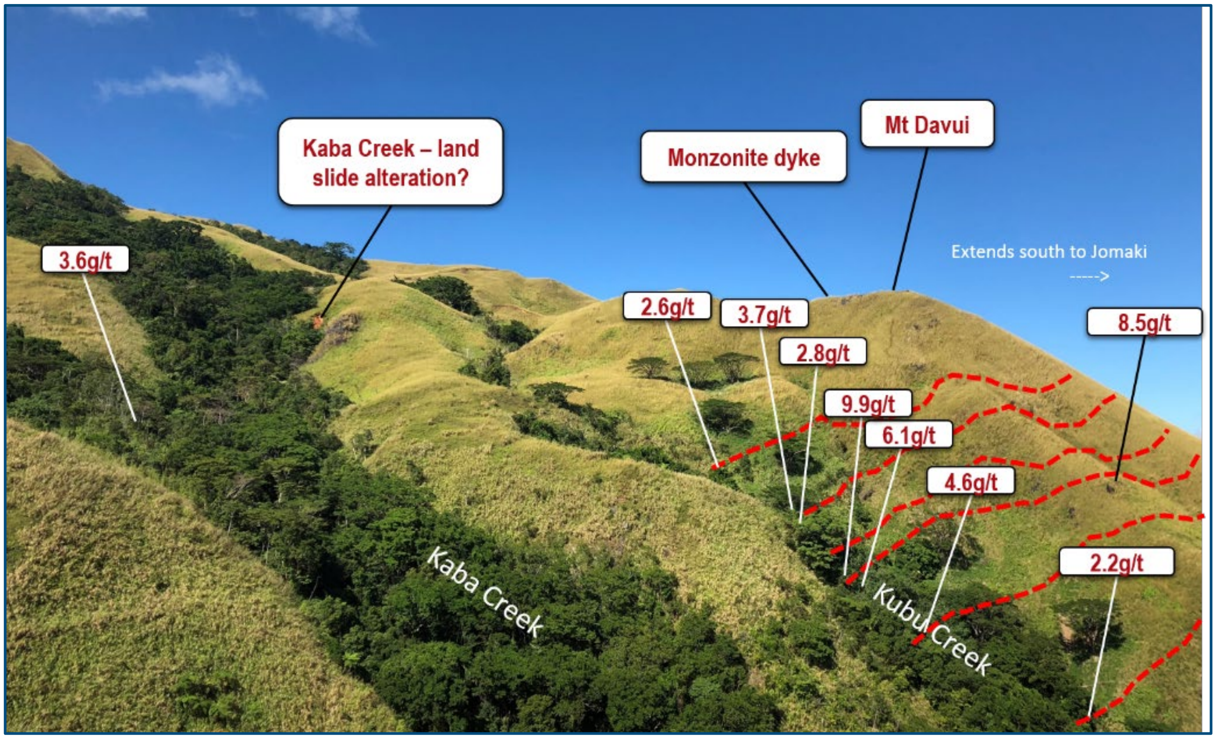
Channel ID	Sample ID	From (m)	To (m)	East (FJM m)	North (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)
CH225	TUS005108	1.15	1.43	1875532.8	3918911.1	0.28	7.4	4.81
CH631	TUS007405	0.6	0.9	1875259.4	3919724.1	0.3	100	4.53
CH802	TUS008019	0.7	1	1875695.7	3919520.3	0.3	7.8	3.28
CH552	TUS007033	12.4	13.3	1875400.8	3919200.4	0.9	0.8	2.36
CH859	TUS008189	0.5	1	1875474.4	3919689.5	0.5	7.7	2.23
CH634	TUS007416	0.7	1	1875136.5	3919563.6	0.3	100	2.1
CH800	TUS008012	0.65	0.85	1875451.4	3919460.9	0.2	39	2.1
CH1172	TUS009791	0.53	0.94	1875270	3919153.8	0.41	2.1	2.07
CH218	TUS005177	0.5	0.67	1875346.5	3919068.3	0.17	0.25	2
CH578	TUS007205	1.3	1.7	1875434.8	3919650.2	0.4	4.1	1.93
CH697	TUS007762	1.5	1.78	1875344	3919344.4	0.28	15.2	1.88
CH631	TUS007406	0.9	1.5	1875259	3919724.2	0.6	73.9	1.87
CH274	TUS005244	0.5	0.92	1875349	3919248	0.42	1.9	1.84
CH277	TUS005346	0.5	0.8	1875541.2	3919043.6	0.3	2.7	1.84
CH339	TUS005791	0.6	0.9	1875131.6	3919645.4	0.3	1.5	1.79
CH225	TUS005110	2.4	3.2	1875531.3	3918911.2	0.8	0.5	1.75
CH268	TUS005335	0.5	0.8	1875456.9	3919117.5	0.3	1.3	1.73
CH640	TUS007435	0	0.52	1875280.2	3919166.9	0.52	0.5	1.73
CH801	TUS008016	0.9	1.2	1875711.9	3919515.4	0.3	3.5	1.71
CH803	TUS008022	0.5	0.8	1875694.4	3919515	0.3	0.25	1.69
CH631	TUS007407	1.5	2	1875258.5	3919724.3	0.5	69.1	1.54
CH759	TUS007911	0.5	0.7	1875412.6	3918796.9	0.2	1.4	1.49
CH1003	TUS008969	0.7	0.95	1875475.7	3919742.7	0.25	29.9	1.47
CH1090	TUS009493	1.7	2	1875273.8	3919898.6	0.3	6.4	1.42
CH695	TUS007753	0.55	0.8	1875363.4	3919311.3	0.25	2.2	1.38
CH520	TUS006868	0.4	0.8	1875299.4	3919154.6	0.4	2.2	1.34
CH270	TUS005148	0.5	0.75	1875342.8	3919274.3	0.25	3.7	1.32
CH520	TUS006867	0	0.4	1875299.8	3919154.6	0.4	1.1	1.28
CH260	TUS005132	0.5	0.65	1875451.8	3918901.6	0.15	3.3	1.19

table continues…

Channel ID	Sample ID	From (m)	To (m)	East (FJM m)	North (FJM m)	Interval (m)	Ag (g/t)	Au (g/t)
CH258	TUS005312	0	0.5	1875274.4	3919079.2	0.5	1.4	1.18
CH700	TUS007726	2.3	3	1875491.8	3919904.3	0.7	0.5	1.08
CH641	TUS007440	0.46	0.7	1875452.6	3919110.9	0.24	2.3	1.07
CH622	TUS007240	2.65	3.43	1875464.3	3919624.3	0.78	3.3	1.04
CH559	TUS007039	0.7	1	1875389	3919205.1	0.3	0.9	1.00
CH679	TUS007653	0.7	1	1875316.3	3919387.5	0.3	7.9	1.00

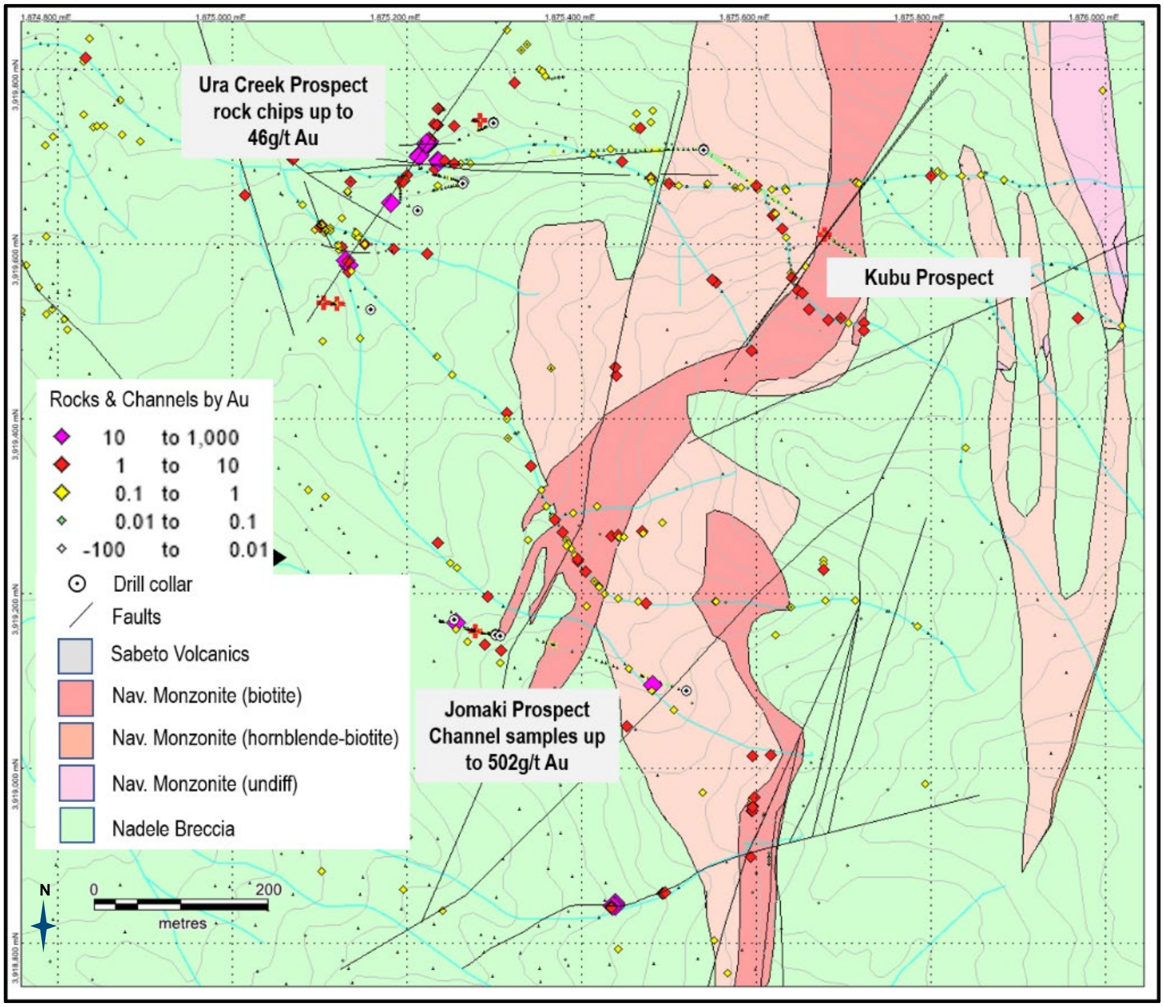
Table 9-13: Jomaki/Ura drill collars (no significant intersections)

Hole ID	East (FJM m)	North (FJM m)	RL (ASL)	Depth (m)	Target Zone
TUDDH-473	1875299.7	3919740.4	235.523	99	Ura Creek
TUDDH-474	1875299.5	3919739.1	235.662	91.5	Ura Creek
TUDDH-475	1875264.7	3919671.3	231.116	90	Ura Creek
TUDDH-476	1875264.2	3919669.5	231.27	91.5	Ura Creek
TUDDH-477	1875212.8	3919639.2	232.55	120	Ura Creek
TUDDH-478	1875159	3919526	232.484	132	Ura Creek
TUDDH-486	1875301	3919153.9	349.754	106.3	Jomaki
TUDDH-487	1875302	3919153.4	349.709	106.1	Jomaki
TUDDH-488	1875253.6	3919170.6	339.959	151.1	Jomaki
TUDDH-489	1875306.8	3919151.6	350.634	118.3	Jomaki
TUDDH-490	1875520.5	3919089.1	418.479	269.7	Jomaki



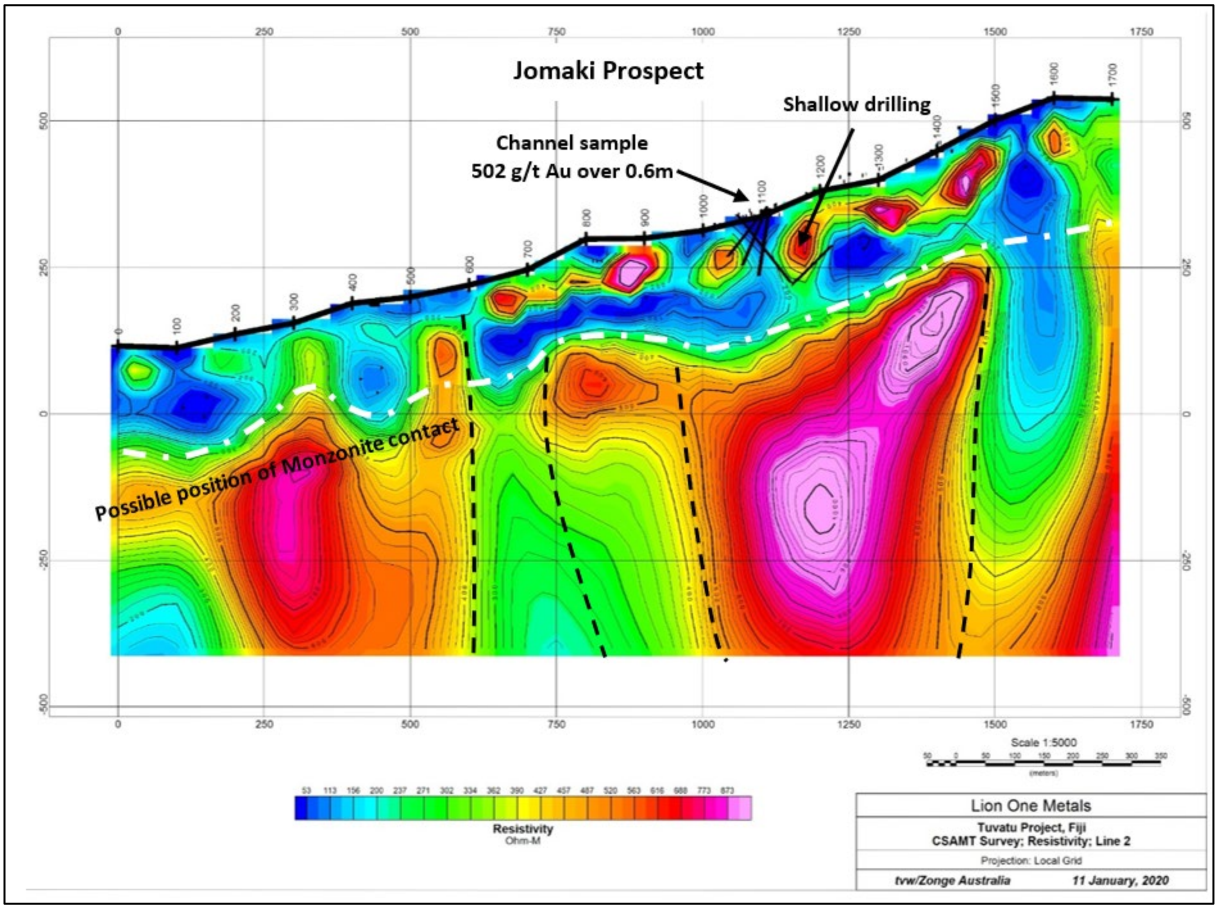
Source: Lion One (2018)

Figure 9-70: Illustration of terrain at the Kubu Prospect area with rock sampling



Source: Lion One (2019)

Figure 9-71: Jomaki-Ura-Kubu geology, with drill collars



Source: Lion One after Thomas V. Weis & Associates (2020)

Figure 9-72: Jomaki CSAMT showing targets at depth

9.5.5.5 Upper Qalibua and Tuvatu North

Following the strong cBLEG sampling results, Lion One has conducted reconnaissance and benching in the Upper Qalibua area, located between 200 and 800 m east of the main Project. This work also included potential northern extensions of the Tuvatu Upper Ridges Vein system. Benching results from this area include:

- Gold grades of 12.65 g/t, 4.29 g/t, and 2.8 g/t over narrow widths of 0.2 to 0.4 m
- A 380 m wide zone in highly leached rock with 57 samples between 0.1 and 1 g/t Au also highly anomalous in tellurium (5 to 10 ppm Te)
- A 130 m wide zone of anomalous pathfinder copper between 1,000 and 5,000 ppm in highly leached rock

Benching results from northern continuations of the Upper Ridges Vein system include:

- 230 m wide zone of very high tellurium (a pathfinder for alkaline style mineralization) (between 10 and 40 ppm Te), with Au assays up to 4 g/t.

In the eastern zone, the bench on the northern side of Qalibua Creek, follows a high ridge and the rock exposed is very highly leached. Despite copper anomalism of up to 5,000 ppm over wide widths, no malachite has been observed, and as such, it is thought that the dominant copper mineral is tenorite (CuO), a highly weathered and leached product of copper sulphides. Given that this area is also carrying multiple structures with between 0.1 and 1 g/t Au, it is currently inferred that gold has been leached in the surface weathering environment. Structures are dominated by north–south, northeast–southwest, northwest–southeast, and east–west striking features. The Upper Qalibua system, as zone marked by CSAMT work, strikes north and potentially links with the Batiri Creek area in the central Caldera, where previous reconnaissance sampling has yielded several structures with grades >2 g/t Au. The Upper Qalibua sampling areas are displayed in Figure 9-73 to Figure 9-75.

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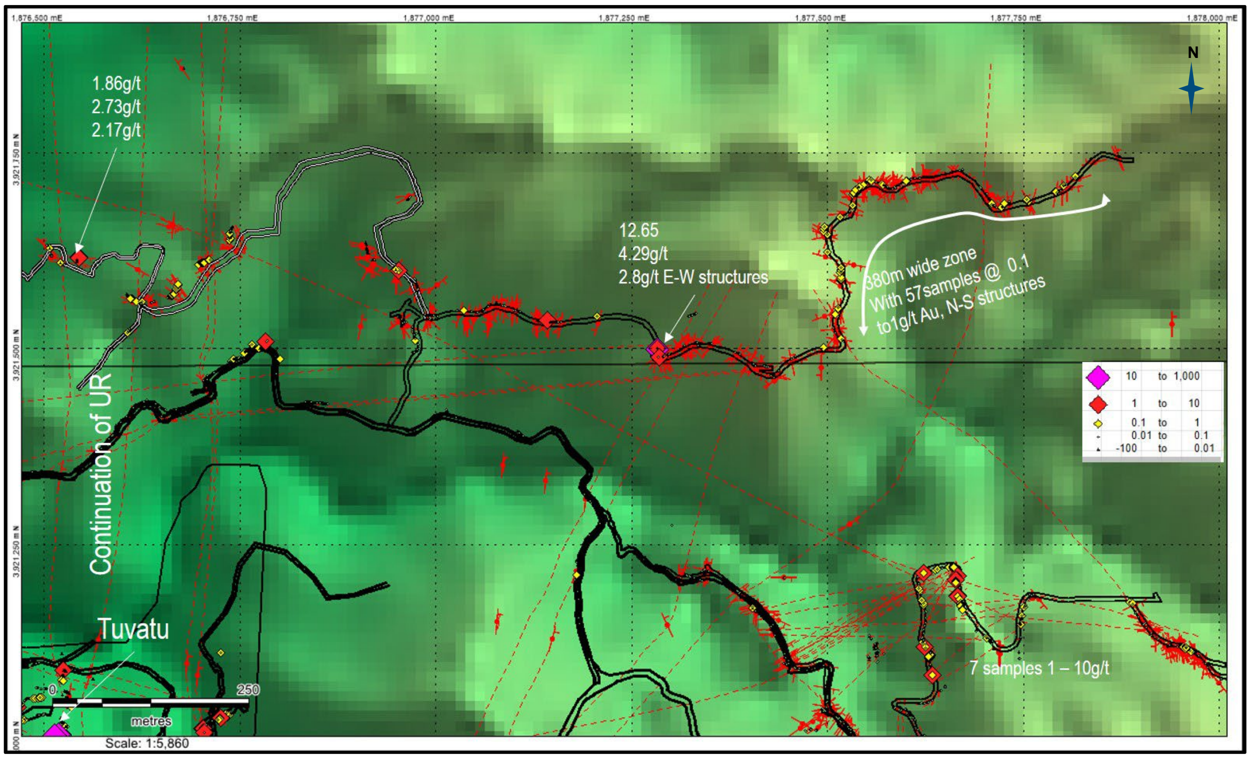
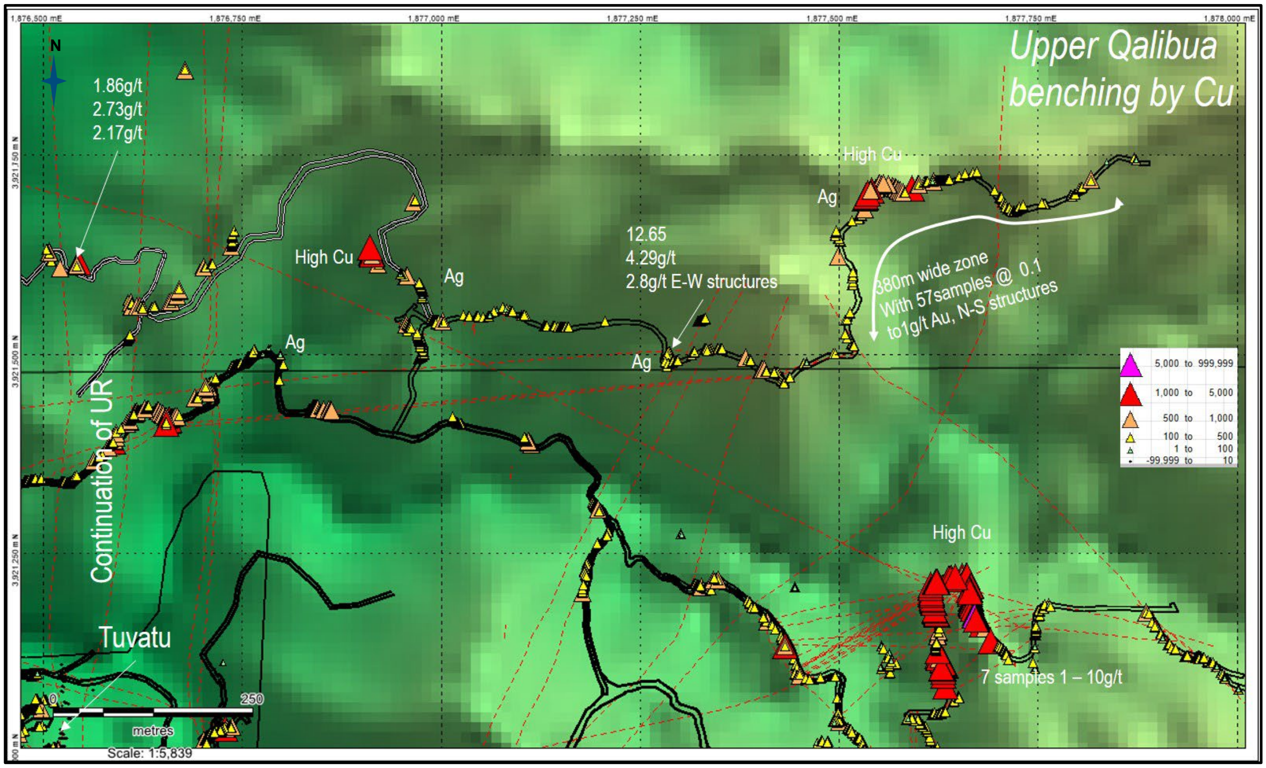


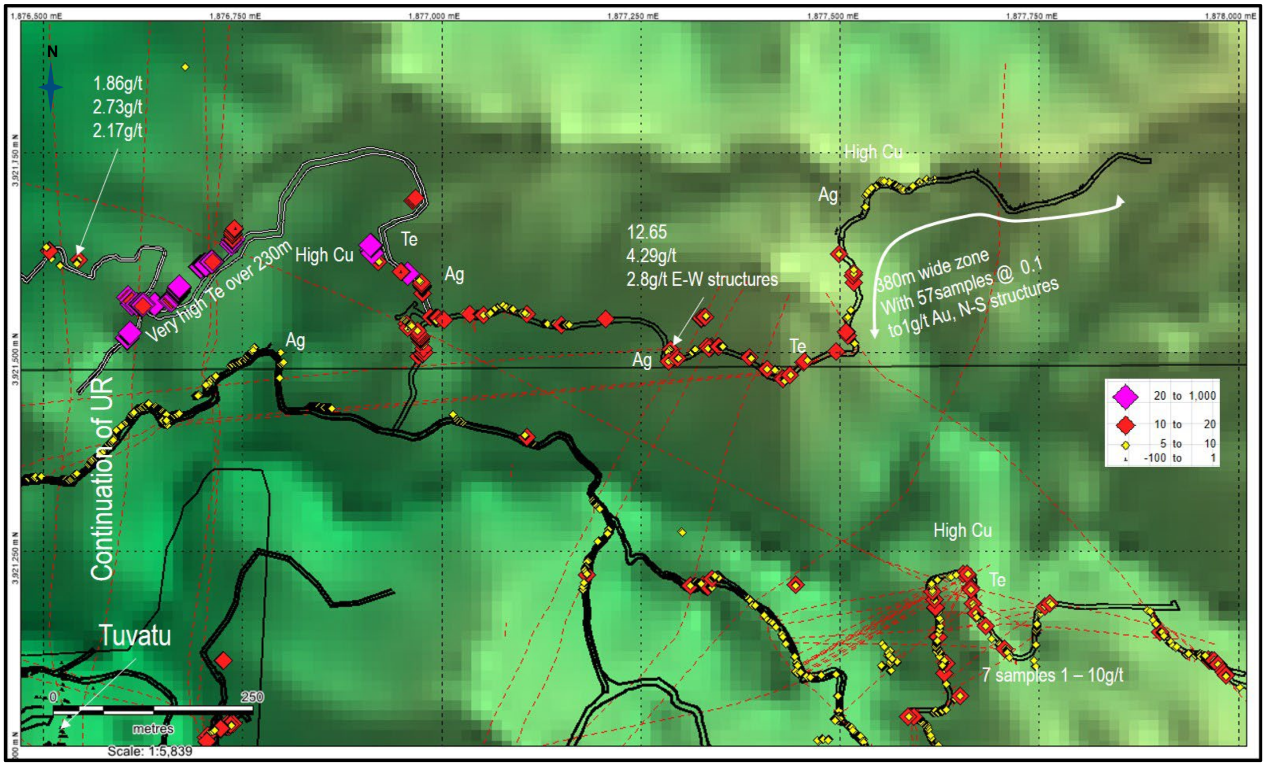
Source: Lion One (2020)

Figure 9-73: Upper Qalibua benching by gold with structures



Source: Lion One (2020)

Figure 9-74: Upper Qalibua benching by copper (with gold grades labelled)



Source: Lion One (2020)

Figure 9-75: Upper Qalibua benching by tellurium (with gold grades labelled)

9.5.5.6 Other Prospects

As well as the aforementioned district prospects, there are several other areas that warrant follow-up work. Refer to Figure 9-9 for location details. These include:

1. Vunisalato Creek – a high cBLEG anomaly with several mapped structures.
2. Nasalo Creek – a high cBLEG anomaly, with preliminary reconnaissance indicating considerable alteration.
3. Nasiti Ridge – historic soil anomaly, adjacent strong cBLEG results, and a potential continuation of the Matanavatu Project area.

10.0 DRILLING

Drilling campaigns were completed in several phases by TGM from 1995 to 2001 and by Lion One between 2008 and 2020. The drilling campaigns are summarized in Table 10-1.

Table 10-1: Summary of Tuvatu exploratory drilling

Company	Surface RC Drilling		Surface Diamond Drilling		Underground Diamond Drilling	
TGM 1995 to 2000 Total	9,265 m (81 holes)	-	51,484 m (217 holes)	-	13,408 m (112 holes)	-
TGM Phase 1	*5,225 m (44 holes)*	*TURC101 to 171*	*42,783 m (193 holes)*	*TUDDH013 to TUDDH205*	*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 TUDDH229*	*10,926 m (69 holes)*	*TUG45 to 113*
Lion One 2008	-	-	376 m (2 holes)	TUDDH338 and TUDDH340	-	-
Lion One 2012 to 2013	-	-	13,842 m (65 holes)	TUDDH341 to TUDDH405	-	-
2014	-	-	-	-	-	-
2015	-	-	-	-	-	-
2016	-	-	2,472.9 m (12 holes)	TUDDH-406 to TUDDH-418	0 holes	-
2017	-	-	8,619.6 m (54 holes)	TUDDH-419 to TUDDH-472	1684.2 m (16 holes)	TUG114 to TUG129
2018	-	-	624 m (6 holes)	TUDDH-473 to TUDDH-478	-	-
2019	-	-	3,733 m (15 holes completed)	TUDDH479-TUDDH493	-	-
2020-2022	-	-	32,368 (87 holes program ongoing in 2022	TUDDH494-TUDDH585 (including wedges)	3,977 m (11 holes)	TUG-130-140

10.1 TGM 1995 to 2001

The focus of the TGM historical drilling programs was to test the strike length of known mineralization mostly with the objective of producing a Mineral Resource estimate. 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 DD core and RC methods. Overall, TGM completed 51,484 m of DD 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 DD holes (TUDDH-013 to 205) and 44 RC holes (TURC-101 to 171) were completed. A total of 42,783 m of DD core (HQ and PQ diameter) and 5,225 m of 5¼ in. RC drilling were completed in the area. This program delineated an area of mineralization that extends over a distance of 800 m. In conjunction with the underground development, seventeen underground DD 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 second phase of work by Emperor Gold Mining Company Limited, 26 underground DD 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 RC drilling program 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 to 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 east–west and northwest–southeast. The lodes may be up to 5 m wide. A series of 69 underground DDHs (TUG045 to 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 program successfully extended the Upper Ridges Lodes (particularly UR2) and upgraded the Phase 2 resource.

A series of surface DDHs 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 DDHs (TUDDH-338 and TUDDH-340) totaling 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 to 2013

Since little shallow drilling had been completed in the resource area, and in response to the results from the trenching program, Lion One focused part of this drilling program to identify any broad zones of low-grade mineralization potentially exploitable by surface mining methods. Additionally, as little drilling had previously targeted the east–west striking Murau Lodes, Lion One commenced a systematic program to delineate the extent of gold mineralization to the west of the main Upper Ridges and Upper Ridges West Lodes. 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 was re-commenced in June 2012 with a combination of infill and step-out holes. The program had three objectives:

- Infill drilling to increase the confidence level of the existing resource
- Step-out drilling to expand the resource base
- Exploratory drilling to test additional targets

Initial infill drill holes were planned to test areas of the intersections of the east–west trending Murau Lodes with the north–south trending Upper Ridge Lodes west of the north–south trending Upper Ridges structural corridor and current resource.

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.

A total of 65 surface DDHs were completed for an advance of 13,842 m.

10.4 Lion One 2016 to 2022

Between 2016 and 2020, Lion One restarted drilling within the main deposit areas and also drilled a number of holes in regional targets. The 2016 to 2017 drilling (Table 10-2) in the Tuvatu deposit area was designed to confirm the drilling by Emperor Gold Mining Company Limited and to better differentiate between the various deposits and lodes within the mineralization system. In 2017 Q3 to Q4, drilling focused on the H and Tuvatu Lodes.

Although the majority of drilling was collared from the surface, 16 DDHs were also completed from underground within the dewatered workings of the existing underground exploration decline.

In total more than 13,000 m of diamond core was completed during this period.

Table 10-2: Summary of 2016 to 2017 diamond drilling at Tuvatu Deposit Area

2016 to 2017 Drilling	Number of Holes	Number of Meters
Surface DDHs	67	11,195.80
Underground DDHs	16	1,684.19
Geotechnical diamond holes	42	1,066.50
Total	**125**	**13,946.49**

Diamond drilling in 2018 targeted surface anomalies in the Ura Creek area to the southwest of Tuvatu. A total of six DDHs (TUDDH473 to 478) were completed for an advance of 625 m. Minor anomalies were identified.

Diamond drilling in 2019 targeted a number of anomalies. Initial drilling in 2019 (TUDDH479 to 485) was largely confirmatory and immediate near surface extensions in the Tuvatu resource area, specifically the HT zone, SKL's GRF, and UR2 Lodes. This was followed in 2019 by fiveDDHs (TUDDH486 to 490) that targeted the Jomaki surface trench anomaly to the south of Tuvatu, and further holes into the HT zone corridor (TUDDH491, 492, 493).

From 2020 to present, drilling resumed at the Tuvatu Project area and focused on deep extensions in search of feeder zones (Table 10-3). Hole TUDDH-500 intersected high-grade zones, with these new zones referred to as the '500 Zones' or '500 feeder zones'.

Table 10-3: Summary of 2020 to 2022 diamond drilling at Tuvatu Deposit Area (refer to Section 9.0 for significant intercepts)

2019-2022 Drilling	Number of Holes	Number of Meters
Surface DDHs (infill / extensional)	55	9911
Underground DDHs (infill / extensional)	6	962.3
Surface DDHs (500 Zones)	32	22,458
Underground DDHs (500 Zones)	5	3,015.5
Total	**98**	**36,347**

10.5 Drilling Procedures

10.5.1 TGM 1995 to 2001

With the exception of a scant few reverse circulation holes (not used), all of the drilling has been surface DD coring with HQ and NQ drilling, with underground drilling using NQ, NQ2, and BQ drilling.

All holes are surveyed and logged using repeatable criteria. Once identified, mineralized intersections are cut with a core-saw with a standardization of which half is sent to the laboratory. 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.
- For the drilling prior to 2008, selected drill core sections were halved with a core saw and samples were dispatched to the Emperor Gold Mining Company Limited laboratory at Vatukoula.
- Waste intervals were not assayed.
- TGM used the assay laboratory at the Vatukoula Mine operated by the Emperor Gold Mining Company Limited. 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.
- In most cases, the whole sample was pulverized in a 5 kg ring mill prior to splitting. A 50 g subsample was analyzed for gold by fire assay with an AAS finish.

- Samples within the interpreted lode structure were composited to obtain an overall grade for the lode.
- For the drilling since 2008, selected drill core sections were halved and sent to ALS Minerals laboratories, originally in Suva, Fiji, for sample preparation and then to Townsville, Australia for analyses. Since 2016, samples were couriered directly to Townsville, Australia for preparation and analysis.
  - Standards (CRMs) were inserted into the sample stream to check for laboratory bias.
  - 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, rock quality designation (RQD) values, alteration, gangue minerals, sulphide minerals, mineralization, sample numbers, intervals samples, analytical values, comments, date logged and by whom. SG of selected intervals and lithologies were measured.
  - A summary log was prepared after the hole was logged.

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 were also calculated by the surveyor to compare with the planned orientation and downhole survey data. The majority of DDHs were also surveyed 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.

During the 2016 to 2017 drilling program, some samples were sent to Vatukoula Gold Mines laboratory for analysis. The pulps of any samples analyzed at Vatukoula and returning gold results greater than 1 g/t Au were then sent to ALS Minerals in Australia for check analysis.

10.5.2 Lion One 2008 Onwards

Drilling by Lion One was diamond core drilling from surface. 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, sulfide minerals, mineralization, sample numbers, intervals samples, analytical values, comments, date logged and by whom. SG 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. Half-core samples prior to December 2019 were dispatched to the ALS sample preparation facility in Suva, Fiji. Samples were first crushed and pulverized at Suva, Fiji prior to analysis at ALS Minerals, an independent and qualified analytical laboratory in Brisbane, Australia. Gold was determined by fire assay and silver by aqua regia digestion and AAS finish. Consistent with industry standard practice, standard reference samples and blanks and additional control methods are used to ensure quality control.

From December 2019, drill samples were processed in the Lion One laboratory. Drill samples were crushed and pulverized at Nadi, Gold was determined by fire assay (30 g charge) and silver, arsenic, copper, iron, lead, selenium, tellurium, vanadium, and zinc were routinely analyzed by a three-acid digestion and inductively coupled plasma optical emission spectroscopy finish. Samples returning greater than 0.5 g/t Au or 0.5% Cu, Pb, or Zn were sent to ALS Minerals (Townsville) for check analysis. The high base metal samples were also fire assayed providing a check of gold analysis below 0.5 g/t Au.

10.6 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. One-meter composites were restricted to cover the same area (roughly corresponding to the 2,000 Mineral 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 (Figure 10-1 A)
    b. DD data restricted in z extent to cover the same depth range as RC data (Figure 10-1 B)
    c. DD and RC data restricted to depths greater than 20 m below surface (to reduce effects of shallow RC drilling near-surface) (Figure 10-1 C)
    d. Data plotted north (Figure 10-2 A) and south (Figure 10-2 B) 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 DDHs.

The resulting charts are shown in Figure 10-1 and Figure 10-2. In all cases, the red line indicates a 1:1 correlation.

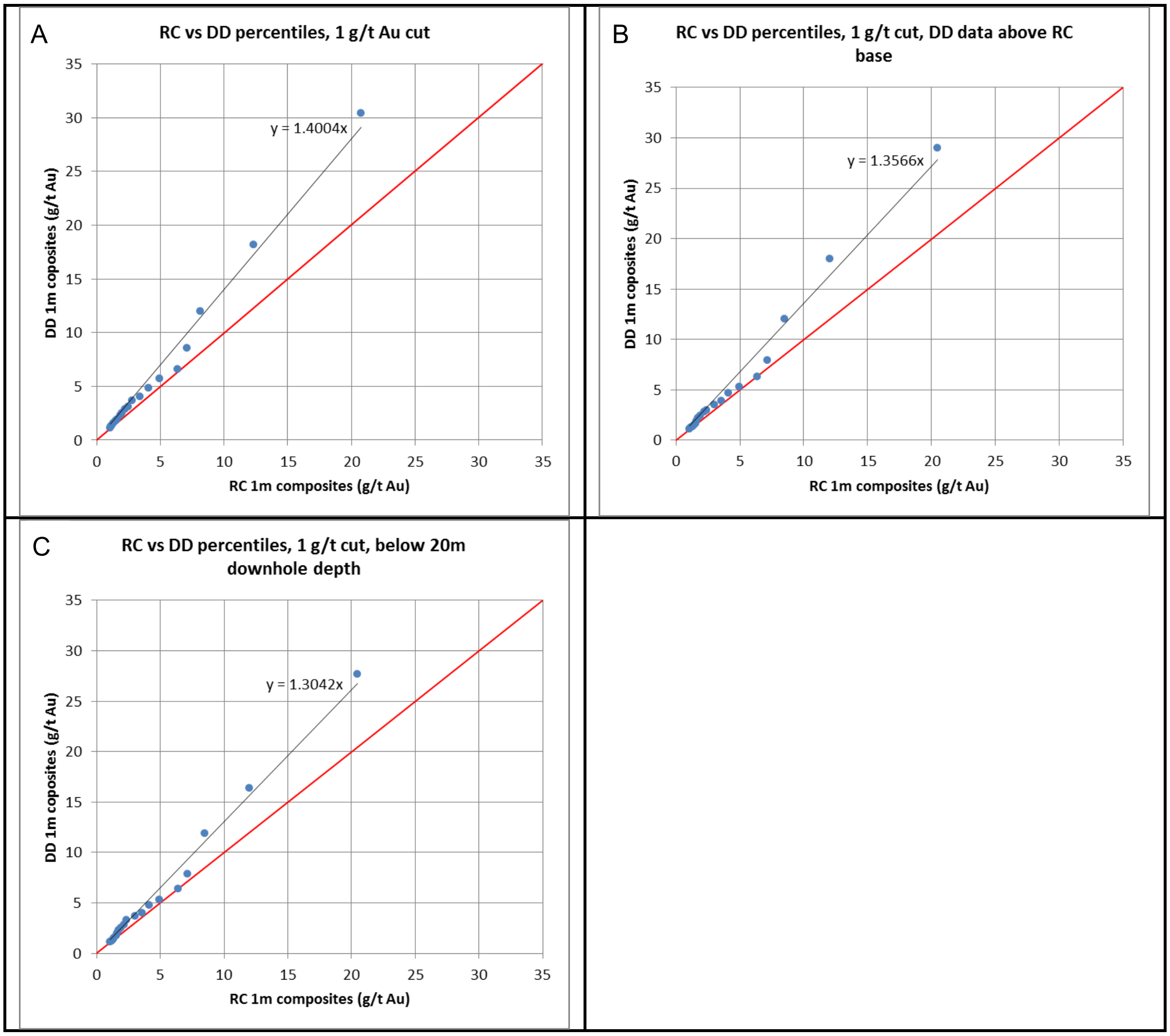


Figure 10-1: Q-Q plots RC vs. DDHs

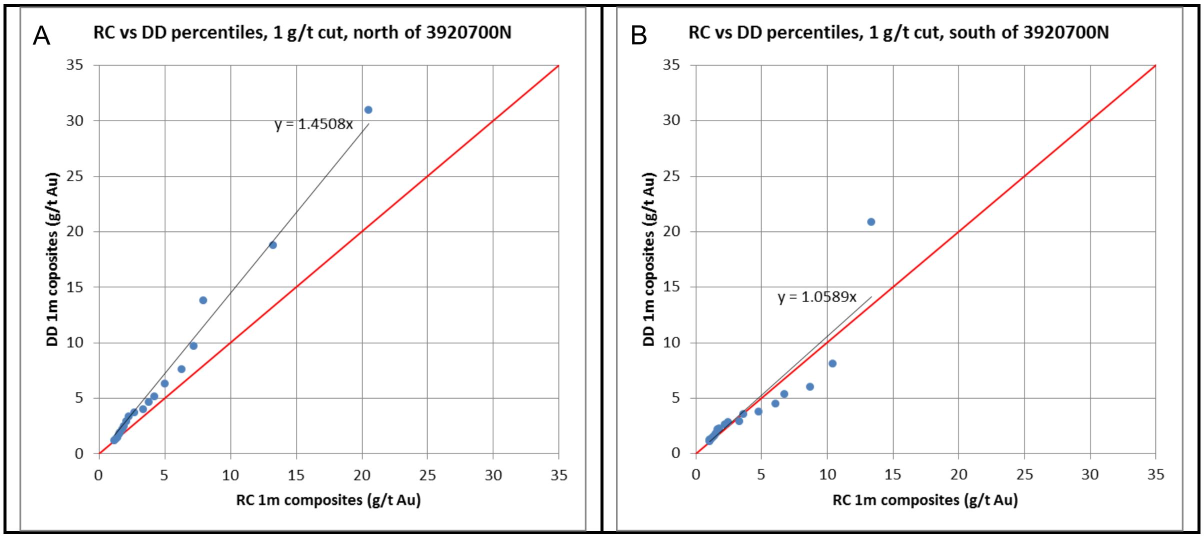


Figure 10-2: Q-Q percentile plots RC vs. DDHs

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 subsampling – The method and possible introduction of bias during subsampling 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 subsampling – Possible introduction of bias during core cutting, especially if the 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 where drill holes sample a vein in a perpendicular orientation.

- From the available data, MA concludes that there appears to be no major problem with utilizing RC samples as part of a Mineral 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 overestimated for thickness and underestimated 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 underestimated and grade possibly slightly overestimated.
  - Vein thickness in RC intersections can only be practically resolved to the nearest meter using grades.
  - The only way to compare DD and RC drill samples and assess potential risks to Mineral Resource estimation is to undertake a small program of drill hole twinning.

10.7 Discussion

Apart from the review completed as part of the 2014 Mineral Resource estimation, there has been no independent review of the drill hole sampling, geological logging, and geological interpretations undertaken for Lion One. Although it is expected that this work was completed to an industry-acceptable standard, there is always a risk involved with structural interpretations, grade, and geological continuity. However, it is believed that the information revealed in the exploration decline is consistent with the results from the drilling and thus mitigates the impact of this risk to a large extent.

10.7.1 Recovery and Quality

Core recovery is overall very high, although within sheared and broken intervals there are evidence of some core loss, particularly if associated with carbonate minerals. 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 pulverized in a 5 kg ring mill prior to splitting.

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

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

Following the closure of the ALS Minerals sample preparation laboratory in Suva in 2015, Lion One dispatched the entire sample to ALS Minerals in Australia for sample preparation and analysis. Samples were securely dispatched via DHL courier.

The same process of sample preparation was applied for the 2016 to 2017 diamond drilling program and the rock chip, trench, and costean sampling.

Following the commissioning of the Lion One Limited geochemical laboratory in Fiji in October 2019, Lion One has completed its own sample preparation to the same standard as that undertaken by ALS in Australia. Lion One sends check samples to ALS in Australia for appropriate QA/QC.

11.2 Sample Security

No particular security measures were used during the life of the 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.

The same philosophy of sample security has been applied for all subsequent assay work.

11.3 Sample Analyses

11.3.1 Laboratory Analysis Procedures

All pre-2000 assaying by Emperor Gold Mining Company Limited for TGM used a 50 g subsample that was analyzed via fire assay with an AAS finish at the mine laboratory at Vatukoula. All samples above 1 g/t Au were re-assayed.

All analyses in the exploration programs by Lion One in 2008 and 2012 to 2019 were carried out by ALS Minerals laboratories in Brisbane, Australia. Gold was analyzed by fire assay with a 30 g charge and AAS finish. Samples with higher grade gold (greater than 3 g/t Au) were re-assayed. Silver was analyzed by aqua regia digestion and AAS.

Both drilling, rock chip, and costean/trench exploration samples were analyzed for 33 elements using a four-acid digestion and inductively coupled plasma atomic emission spectrometry, as well as for gold.

Following the commissioning of the Lion One Limited geochemical laboratory in Fiji in October 2019, Lion One has completed its own sample analysis to the same standard as that undertaken by ALS in Australia. Lion One sends check samples to ALS in Australia for appropriate QA/QC. Samples are analyzed at Lion One's own geochemical laboratory in Fiji, whilst duplicates are sent to ALS laboratories in Australia. All samples are pulverized to 80% passing through 75 microns. Gold analysis is carried out using fire assay with an AA finish (ALS code Au-AA26). Samples that returned grades greater than 10 g/t Au by Au-AA26 are re-analyzed by gravimetric method (ALS code Au-GRA22). The results are all analyzed by ALS Townsville, Queensland, Australia and include Au-AA26, and also Au-GRA22 where applicable.

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 Limited.

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

ALS Minerals has been used by Lion One for all subsequent geochemical assay work.

11.4 Quality Control

11.4.1 Quality Control 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 short QA/QC program for the 2008 drilling by Lion One.

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

CRM, blanks, and field duplicates samples were inserted prior to shipment from site to monitor the quality control of the data. MA understands that three CRM samples were inserted every 100 samples and 2 field duplicates were inserted in every batch of 100 samples. MA received and reviewed QA/QC summary reports (for CRMs, field duplicates, and assay laboratory duplicates) from rOREdata Pty Ltd. database consultants for Lion One.

The same QA/QC program was applied for the 2016 to 2017 diamond drilling program and the rock chip, trench, and costean sampling.

11.4.2 Quality Control Program Results

11.4.2.1 Standards Results – Accuracy

Accuracy is identifying the true grade of a sample, often achieved by submitting CRM commonly referred to as standards.

Ten different gold CRM standards supplied by Rocklabs Ltd. of New Zealand were used by Lion One for QC in core sampling. Seven of the standards were submitted more than five times (Table 11-1). A total of 216 CRM gold standards and 26 silver standards were submitted during the Lion One drilling program.

The same process of addition of standards has been applied for the 2016 to 2017 diamond drilling program and all rock chip, trench, and costean sampling.

Table 11-1: Summary of CRM used by Lion One, 2012 to 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)	122.300	5.4751	26	2

11.4.2.2 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 drill core samples.

The same process of addition of field blanks has been applied for the 2016 to 2017 diamond drilling program and the rock chip, trench, and costean sampling.

11.4.2.3 Field Duplicates – Precision and Bias

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

Thirty-five 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 samples from TUDDH-384 assayed 0.64 g/t Au compared to the original sample, which returned an assay of 0.16 g/t Au.

The same process of addition of field duplicates has been applied for the 2016 to 2017 diamond drilling program and the rock chip, trench, and costean sampling.

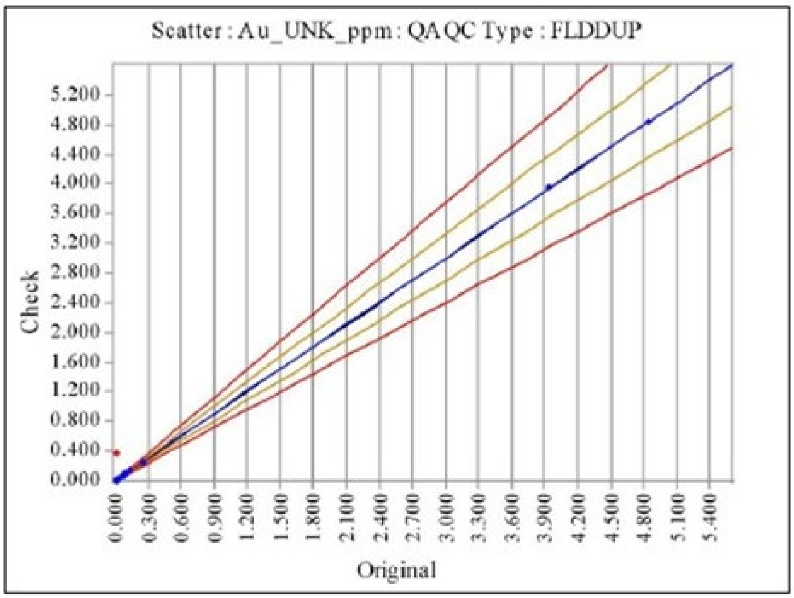


Figure 11-1: Field duplicate results for gold

11.4.2.4 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 have not seen these results.

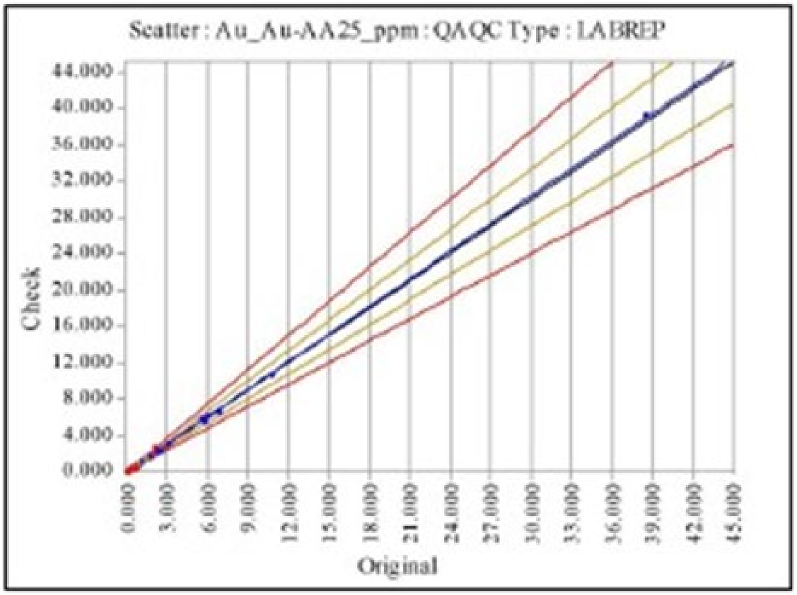


Figure 11-2: Laboratory duplicate results for gold

11.4.2.5 Inter-laboratory Checks

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

11.5 Adequacy Opinion

Apart from the brief review completed by the independent consultant, MA, there has been no other 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 followed 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 visits, 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 Mineral Resource estimation. MA also accessed archived data used for Mineral 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 Microsoft® Access format, named Database ExportDrillHoles.mdb. The database contained tables for collar details, collar metadata, downhole surveys, assays, weathering, lithology, alteration, geotechnical, specific gravity data, and lode tags.

12.2.1 Database Validation

Microsoft® Access queries were used to perform basic validation checks, and holes were then loaded into GEOVIA Surpac™ for a second round of validation. Table 12-1 summarizes 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 DDHs (1970s). Need to confirm locations and assign z coordinates from topography digital terrain model.
Missing downhole surveys	Four holes with no downhole surveys.	Lion One is checking the missing downhole surveys. One hole is 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.	QA/QC results mistakenly included in assay table.
Downhole survey validation check, Microsoft® Access query	Seventy-one drill holes with survey deviation. Greater than 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.

12.3 Site Visits

12.3.1 Site Visits by Mr. Ian Taylor

Mr. Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP), visited the Tuvatu deposit from February 25 to 28, 2014; July 31 to August 5, 2017; and September 28 to October 3, 2017. In the course of the site visits, Mr. Taylor viewed the mineralized drill core and examined the drill core processing and storage facilities. He also viewed and sampled the mineralized vein systems and outcrops and inspected decline dewatering and the development of the SKL Lodes. The UR2 and UR5 development drives were inaccessible in October 2017.

12.3.1.1 Independent Samples

For this Technical Report, Mr. Taylor collected two independent samples: one from outcropping Tuvatu Lode and one from drill core (Figure 12-1).

The selected samples were chosen by Mr. Taylor; at no time prior to the sampling were any employees or other associates of Lion One advised of sample locations or were any samples to be collected identified. The samples 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 Minerals Laboratories in Suva, Fiji for sample preparation. The prepared samples were sent to ALS Minerals in Brisbane, Australia for analysis for gold by 30 g fire assay (ALS Minerals method Au-AA25). Table 12-2 lists the samples and descriptions of the gold assay results.

Table 12-2: Mining Associates independent sample descriptions

Sample ID	Sample Description	Gold (g/t)
MA_TV_01	Rock chip from Tuvatu Lode outcrop	13.65
MA_TV_02	Drill core from TUDDH112; 282.4 m to 282.8 m	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.

Source: MA (2014)

Figure 12-1: Mining Associates sampling Tuvatu Lode outcrop

12.3.2 Site Visits by Dr. Darren Holden

Dr. Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM (Geo), visited the Property on February 16 to 23, 2020; December 1 to 7, 2019; October 27 to November 3, 2019; September 18 to 30, 2019; June 29 to July 7, 2019; April 7 to 14, 2019; March 7 to 15, 2019 and a total of 12 other times, for approximately 7 to 10 days each time, during 2017 and 2018. The purpose of these visits were to conduct exploration, mapping, data reviews, and reporting, and develop sampling protocols.

12.3.3 Site Visit by Mr. Laszlo Bodi

Mr. Laszlo Bodi, M.Sc., P.Eng., visited the site from April 4 to 7, 2018. Mr. Bodi viewed the general process plant and TSF areas and reviewed available geotechnical information concerning the geotechnical condition for various structures within the plant site. Mr. Bodi also specified and was present during a geophysical testing program carried out at the TSF and process plant areas by GBG Australia. The geophysical survey results provided a near continuous profile of the underlying bedrock and also the stiffness characteristics of the soil layers across critical sections of the process plant and TSF areas. Furthermore, shear wave velocity values were also provided to estimate the approximate allowable bearing capacity values of soils for the preliminary consideration of potential foundation alternatives and highlight weak soil areas that will require further analyses during the feasibility and detailed design stages of the project.

12.3.4 Laboratory Visits by Dr. Jianhui (John) Huang

Dr. Jianhui (John) Huang, Ph.D., P.Eng., visited the metallurgical testing laboratories including BV on August 8, 2018; Jinpeng Group on December 2, 2017; and Xinhai on December 4, 2017 to inspect test equipment, test procedures, and discuss test results with the technicians/engineers at the laboratories.

12.4 Opinion of Qualified Persons

The QPs' opinions of the data verification from the site and laboratory visits are summarized below.

12.4.1 Opinion of Qualified Person, Mr. Ian Taylor

Based on the data verification performed, it is Mr. Taylor's opinion that the data reviewed is adequate for the purposes used in this Technical Report.

12.4.2 Opinion of Qualified Person, Dr. Darren Holden

Based on the data verification performed, and in relation to field collected data (such as surface channel samples) whereby assay results were reasonable compared to observed mineralization, it is Dr. Holden's opinion that the data reviewed is adequate for the purposes used in this Technical Report.

12.4.3 Opinion of Qualified Person, Mr. Laszlo Bodi

It is Mr. Bodi's opinion that the up-to-date geotechnical drilling and testing program across the process plant area, (sampling and testing) are consistent with recognized industry practices and are considered adequate for this type of somewhat complex geotechnical conditions, at PEA-level of study. The obtained information will guide geotechnical and structural engineers to select the most appropriate and economical foundation alternatives for the structural components of buildings and machines across the plant site, during the upcoming feasibility and detailed design stages of the project.

12.4.4 Opinion of Qualified Person, Mr. Laszlo Bodi

It is Mr. Bodi's opinion that the geotechnical drilling and testing program across the TSF area (sampling and testing), are consistent with recognized industry practices and are considered adequate for this type of somewhat complex geotechnical conditions, at PEA-level of study. The obtained information will guide geotechnical dam engineers to select the appropriate and economical dam foundation and diversion design works. However, further geotechnical investigation and geochemical test works are required to optimize the TSF design during the upcoming feasibility and detailed design stages of the project.

12.4.5 Opinion of Qualified Person, Mr. Norman Schwartz

Meteorological and hydrologic data received from Fiji Meteorologic Service (FMS) for Nadi Airport were used in the current TMF water management design and water balance. Independent verification of this data was not carried out, as no site-specific data was available. It is the opinion of Mr. Norman Schwartz, M.Sc.Eng., P.Eng., that the data used should be verified for applicability to the project site in any future project design and development.

12.4.6 Opinion of Qualified Person, Dr. Jianhui (John) Huang

Dr. Huang visited the Jinpeng Group, Xinhai and BV metallurgical testing laboratories and reviewed all the test work that were available. Significant test work has been completed by various test programs from 1997 to 2020. In general, the test results show that the samples tested respond reasonably well to the process flowsheet proposed, although some variations in metallurgical performance were observed from some of the samples tested. There is a significant amount of nugget gold, which varies significantly, occurring in the mineralization. It is Dr. Huang's opinion that the metallurgical test work is considered adequate for this study and the processing flowsheet proposed is reasonable and expected to be adaptable to the mineralogical variation of the mill feed. Further test work is recommended to better understand the metallurgical responses of the mill feed, especially planned mill feed in Year 1 and Year 2, and to improve gold recovery, including pre-treatment methods and intensive extraction methods. Further flowsheet optimization is also recommended to simplify the process flowsheet and improve project economics.

12.4.7 Opinion of Qualified Person, Mr. Davood Hasanloo

It is opinion of Mr. Davood Hasanloo, M.Sc., M.A.Sc., P.Eng., that the hydrology and hydraulic analysis completed at this stage is adequate for a PEA-level of study. However, moving forward, more detailed analysis accompanied with the use of site-specific meteorological and hydrometric data would be needed. At a detailed design stage, a site visit by the water management engineer is necessary, and the design criteria shall be revisited.

12.4.8 Opinion of Qualified Person, Mr. Shane McLeay

Based on the data verification performed, it is the opinion of Mr. Shane McLeay, B.Eng. Mining (Hons), F.AusIMM, that the data reviewed is adequate for the purposes used in this Technical Report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Since 1997, extensive test work has been conducted on the samples collected from the Property. The test work includes mineralogy studies, comminution tests, gold and silver recovery tests, cyanide destruction tests, and process-related parameter determination tests. The gold and silver recovery methods tested include gravity concentration, flotation, and cyanide leach extraction.

Table 13-1 summarizes the main test work programs conducted. Test work performed between 1997 and 2015 were reviewed and summarised in the 2015 PEA (Freudigmann et al. 2015). Freudigmann et al. (2015) included a high-level review of these tests, excluding the comminution tests, which are discussed in greater detail.

Table 13-1: Metallurgical test programs and reports

Year	Test Programs/Reports	Company
1997	Processing Test Report	AMMTEC
1997	Tuvatu Comminution Test Work Report No. A5724	AMMTEC
1997	Pilot-Scale Processing Test Report	OMC
1997	Tuvatu Comminution Tests Report No. N8394	Amdel
1997	AG/SAG Amenability of Tuvatu Project Ore Types	OMC
1998	Tuvatu Comminution Tests Report No. N8448	Amdel
2000	Processing Test Reports	Metcon
2000	Petrological Study of Tuvatu Gold Project Samples	CMS
2012	Gravity Test Report	Gekko
2015	Processing Test Report	Jinpeng Group
2015	Tuvatu Gold Property PEA NI 43-101 Report	Lion One
2016	Metallurgical Test Work Report No. A16743	ALS Metallurgy
2018	Mineral Processing Report	Xinhai
2018	Mineralogical Assessment Report	BV
2018	Petrologic Studies Report	APSAR
2018	Metallurgical Test Report (BV1801004)	BV
2018/2019	Metallurgical Test Data Reports (BV1801807 & BV1803310)	BV
2019/2020	MS1944 Lion One Metals Report	Met-Solve

Notes: Amdel – Amdel Limited Mineral Services; AMMTEC – Australian Metallurgical and Mineral Testing Consultants Limited; APSAR – Applied Petrologic Services & Research; CMS – Central Mineralogical Services; Gekko – Gekko Systems Inc.; Jinpeng Group – Metallurgical Company Yantai Jinpeng Group; Metcon – Metcon Laboratories; Met-Solve – Met-Solve Laboratories Inc.; OMC – Orway Mineral Consultants; Xinhai – Yantai Xinhai Mining Research & Design Co. Ltd.

13.1 Mineralogy Analysis

Mineralogical investigations conducted between 1997 and 2003 are summarized in Freudigmann et al. (2015). Xinhai, Bureau Veritas Commodities Canada Ltd. (BV), and Applied Petrologic Services & Research (APSAR) completed the most recent mineralogical studies.

13.1.1 Previous Mineralogy Test Work 1997 to 2015

Five laboratories—Australian Metallurgical and Mineral Testing Consultants Limited (AMMTEC) (1997), Metcon Laboratories (Metcon) (2000), Central Mineralogical Services (CMS) (2000), and APSAR (2003)—undertook various mineralogical determinations. The major conclusions are summarized in the following subsections:

APSAR (2003)

"Native gold, as well as gold-silver telluride comprise precious metal mineralogy within the banded quartz veins and stock-work zones."

CMG (2000)

"The gold associated with enargite, and with bismuth/galena and argentite, occurred as inclusions in these minerals, (1-200 µm in size) and as fine intergrowths with them. Other gold occurrences were observed to be variable, but with significant amounts of fine-grained (1-30 µm) gold occurring in silicate and carbonate gangue minerals."

Metcon (2000)

"The main gangue minerals are quartz and micas with heavy mineral assemblages of siderite, apatite and diopside. The dominant sulphides are pyrite with minor to trace amounts of galena, sphalerite, chalcopyrite, marcasite and bornite and oxide minerals of magnetite and hematite. All the minor sulphides are associated with and as inclusions in pyrite. One 20 µm free gold was seen, alone with another composite grain partly liberated from pyrite. There was trace amounts of tellurides as altaite (PbTe) and petzite (Ag_3AuTe_2)."

Metcon's (2000) report also indicates that the gold appears to be present as:

- Discrete or free gold grains or flakes.
- Fine gold associated with gangue.
- Fine gold associated with pyrite.
- Fine gold associated with enargite.
- Fine gold rimming and intruding in other sulphides.
- Possibly, but most likely only a minor portion, associated with telluride minerals.

CMS (2000)

In 2000, CMS conducted a preliminary petrological study to determine sulphide and gangue occurrences. The study indicated that the sulphide minerals were introduced during at least three stages:

- Late-magmatic scattered pyrite with trace amounts of associated chalcopyrite
- Emplacement of veins containing pyrite (marcasite as well), low-iron sphalerite, chalcopyrite, and galena

- Late-stage sulphide-tellurite-gold emplacement in pre-existing veins, involving the deposition of such minerals as enargite, tetrahedrite-tennantite, native bismuth, bismuthinite, galena, argetite, cosalite, and tellurites

Gold is partly associated with enargite and argentite and partly occurs as very small grains in gangue rimming and veining in earlier-formed sulphides. Variable but significant amounts of gold occur in silicate and carbonate gangue minerals. Most of the gold is fine-grained (1 to 30 µm). Gold associated with enargite, bismuth/galena, and argentite may occur as inclusions. Tellurite and argentite may be over ground due to soft/brittle characteristics.

AMMTEC (1997)

"A relatively uniform gold grade across the size ranges examined, which may be due to the presence of fine gold. A bimodal gold distribution, with coarse gravity recoverable gold (GRG) and a more finely disseminated fraction, being consistent with previous mineralogical and petrographic observations."

13.1.2 Recent Mineralogy Test Work 2016 to 2018

Both Xinhai and BV investigated mineralogical properties of the samples in their testing programs. APSAR completed an independent petrologic study on core samples.

13.1.2.1 Mineralogical Work – Xinhai, March 2018

Xinhai conducted preliminary mineralogical work as part of its mineral processing test program to find out the basic information of mineral compositions and associations. The mineral composition was classified into two groups: metallic mineral group and non-metallic mineral group. Pyrite, chalcopyrite, and magnetite were identified as major metallic minerals, while silica and feldspar were the major non-metallic minerals.

One free gold grain was spotted in wheat grain shape with a grain size of approximately 9 µm. It is expected that the gold should have a high purity.

13.1.2.2 Mineralogical Work – BV, June 2018

A mineralogical assessment was conducted on the feed material used in a metallurgical test program by BV. The purpose of the assessment was to understand gold deportment as well as mineral composition and fragmentation on a size-by-size basis. Four size fractions, +74 µm, -74+53 µm, -53+25 µm, and -25 µm, were examined. The 80% passing size (P_{80}) of the composite sample was determined as 77 µm. The major observations made on the composite sample are summarized in the following subsections.

Gold Deportment

- The composite sample was graded at 16.9 g/t Au. Over 80% of the gold in the composite occurs as calaverite ($AuTe_2$), while the remaining gold mainly presents as native gold. Trace amounts of the gold were observed in the forms of electrum, other telluride minerals, and uytenbogaardtite (Ag_3AuS_2).

- For all the sized fractions, 173 gold particles were identified with the average circular diameter of 7.7 µm, ranging from 0.7 to 61.3 µm. Approximately 65% of the gold is distributed in the sized fractions larger than 25 µm.

- The gold liberation is approximately 62.2% by weight for the composite sample. The non-liberated gold particles occur as either the exposed surfaces by attaching to other minerals at a weight ratio of 34.4% or locked grains at a weight ratio of 3.4%.

- Most non-liberated gold grains are associated with non-sulphide gangue minerals, including silicates, carbonates, and altaite (PbTe), while only a small amount of the gold particles are associated with sulphide minerals.

Minerals Composition and Fragmentation

- The composite sample contains 4.4% sulphide minerals, which is mainly composed of pyrite (FeS_2) of approximately 3.8%, sphalerite ((Zn,Fe)S) of approximately 0.36%, chalcopyrite ($CuFeS_2$) of approximately 0.10%, and arsenopyrite (FeAsS) of approximately 0.07%.
- The non-sulphide minerals found in the composite sample are 95.6% by weight, which is mainly composed of silicates (83.6%), carbonates (4.5%), and iron oxides (2.8%).
- The iron sulphide liberation is approximately 57% by weight for the composite sample. The non-liberated iron sulphides are mainly interlocked with non-sulphide minerals in a binary form. Approximately 97% of this binary form contains over 50% iron minerals by weight.

13.1.2.3 Mineralogical Work – APSAR, October 2018

Petrologic studies were conducted on sixteen mineralized core samples from the low-sulphidation, epithermal precious metal deposit. The examination was to determine the spatial relation of base metal and gold mineralization to parenting rocks. Detailed petrographic/mineral-graphic studies were completed, including primary rock types, metamorphism/metasomatism, hydrothermal alteration, and any mineralization. The results are helpful to determine geologic framework and the hydrothermal processes forming the observed mineral paragenesis and distribution.

13.2 Comminution Tests and Simulations

In June 1997, AMMTEC conducted a preliminary comminution test program on three samples from drill holes DDH-44, 65, and 76 of the Tuvatu deposit. In October of the same year, Amdel Limited Mineral Services also completed comminution tests. Based on these results, Orway Mineral Consultants (OMC) simulated various grinding circuit options proposed for a Tuvatu feasibility study. In February 1998, Amdel completed another comminution test program with a major focus on the sample’s competency to autogenous grinding (AG) and SAG. In 2000, Metcon measured comminution work indices for a few of the samples. The results, however, were considered as indicative only. In 2012, Gekko Systems Inc. (Gekko) measured Bond crushing work index and abrasion index on samples from the Property and valuated the possibility of using a vertical shaft impactor (VSI) as a main comminution device to reduce the particle size.

13.2.1 Comminution Tests – AMMTEC, June 1997

Three drill core samples were tested for UCS determinations. A composite sample was then generated by blending the three drill cores for rod mill work index and ball mill work index determinations. Table 13-2 summarizes the test results.

Table 13-2: Average results of comminution tests by AMMTEC

Samples	UCS (MPa)	A_i (g)	R_{Wi} (kWh/t)	B_{Wi} (kWh/t)
DDH-44	126	n/a	n/a	n/a
DDH-65	114	n/a	n/a	n/a
DDH-76	37	n/a	n/a	n/a
Composite	n/a	0.0943	20.2	18.1

Source: AMMTEC (1997)

13.2.2 Comminution Tests – Amdel, October 1997

Two mineralization samples from the Nasivi and Upper Ridges zones were tested for comminution characteristics, including UCS, SG, impact crushing index, abrasion index, rod mill work index, and ball mill work index. Table 13-3 shows the main results.

Table 13-3: Average results of comminution tests by Amdel, October 1997

Description	Unit	Nasivi	Upper Ridges
UCS	MPa	118	190
Impact Work Index			
19 x 25 mm	kWh/t	10.6	8.9
25 x 38 mm	kWh/t	16.8	15.1
38 x 51 mm	kWh/t	25.6	14.6
51 x 76 mm	kWh/t	35.0	28.3
76 x 102 mm	kWh/t	48.5	-
SG	-	2.78	2.82
A_i	g	0.169	0.184
R_{Wi}	kWh/t	17.7	20.2
B_{Wi}	kWh/t	17.7	19.6

Source: Amdel (1997)

Both the samples possessed a high impact strength with a low grindability. The ball mill work index is high, averaging at 18.6 kWh/t, which is similar to the data from the composite tested by AMMTEC. The abrasion index values for both the samples seem to be at an average level.

13.2.3 Simulations – OMC, October 1997

OMC analyzed the comminution test results by Amdel, concluding that both the mineral samples were amenable to single stage SAG and SAG-ball milling processes. AG milling was also considered suitable but required more test data. Based on the test data, OMC used the HelpSAG power modelling program to develop a grinding circuit flowsheet, including calculating material balance, sizing grinding mills, and estimating power consumptions. The feed material used for the simulation was a blend of 40% Nasivi mineralization and 60% Upper Ridges mineralization. The grinding circuit capacity was 400,000 t/a at initial stage with an expansion to 600,000 t/a. The target product size was 80% passing 75 µm.

Table 13-4 and Table 13-5 summarize the major simulation results. The simulation results indicate that using a conventional milling method should be more power efficient as compared with SAG and AG milling options. However, the local wet weather could result in some difficulties in material handling at the preceding crushing stage and an increased consumption of grinding media and liners. Also, a SAG mill has more flexibility in terms of mill capacity considering its maximum ball load can be up to 12%.

Table 13-4: Grinding circuit simulation results at the initial capacity of 400,000 t/a

Parameter	Unit	Option 1 Single-stage SAG Mill	Option 2 Single-stage AG Mill	Option 3 Single-stage Overflow Ball Mill
Diameter	m	6.10	7.32	3.96
Effective Length	m	3.20	2.90	5.79
Ball Charge	% (nominal)	6	0	35
Pinion Power	kW	1,414	1,530	1,265
Motor Rating	kW	2,000	3,000	1,650
Total Grinding Circuit Specific Power	kW/t	28.41	30.60	25.29

Source: OMD (1997)

Table 13-5: Grinding circuit simulation results at the full capacity of 600,000 t/a

Parameter	Unit	Option 1 SAG Mill & Ball Mill	Option 2 Single-stage AG Mill	Option 3a Two-stage Ball Mills	Option 3b Rod Mill & Ball Mill
Primary Milling					
Diameter	m	6.10	7.32	3.96	3.20
Effective Length	m	3.20	2.90	5.79	4.27
Ball Charge	% (nominal)	3	6	35	35
Pinion Power	kW	1,088	2,121	528	427
Motor Rating	kW	2,000	3,000	650	600

table continues…

Parameter	Unit	Option 1 SAG Mill & Ball Mill	Option 2 Single-stage AG Mill	Option 3a Two-stage Ball Mills	Option 3b Rod Mill & Ball Mill
Secondary Milling					
Diameter	m	3.81	n/a	3.05	3.96
Effective Length	m	5.64	-	4.72	5.79
Ball Charge	% (nominal)	35	-	35	35
Pinion Power	kW	1,067	-	1,215	1,215
Motor Rating	kW	1,400	-	1,700	1,650
Total Grinding Circuit Specific Power	kW/t	27.57	28.41	23.43	22.20
Total Motor Rating	kW	3,400	3,000	2,350	2,250

Source: OMD (1997)

13.2.4 Comminution Tests – Amdel, February 1998

After the test work was completed in 1997, Amdel conducted another comminution testing program in 1998 to determine the sample's competency with AG and SAG milling through an AG Media Competency Test (AMCT). Other standard tests were also conducted, including UCS, impact crushing work index, abrasion index, rod mill work index, and ball mill work index. Table 13-6 shows the major results. Compared to the previous test results, the sample tested in 1998 showed less competencies to both rod mill work index and ball mill work index grinding.

Table 13-6: Average results of AMCT comminution tests by Amdel, February 1998

Description	Unit	Sample*
UCS	MPa	121
Impact Work Index		
19 x 25 mm	kWh/t	6.8
25 x 38 mm	kWh/t	9.8
38 x 51 mm	kWh/t	15.2
51 x 76 mm	kWh/t	23.2
76 x 102 mm	kWh/t	29.7
SG	-	2.71
A_i	g	0.145
R_{Wi}	kWh/t	17.1
B_{Wi}	kWh/t	16.3

Source: Amdel (1998)
Note: *The sample origin was not described in the report received.

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13.2.5 Comminution Tests – Metcon, 2000

Metcom's work in Program 00840 assessed ball mill grinding work index variations using the relative work index method. The tests showed the relative ball mill work index in a range between 12.1 to 19.2 kW/t. However, these results, based on the comments from the 2015 PEA (Freudigmann et al. 2015), are indicative only because of the poor testing methodologies.

13.2.6 Comminution Tests – Gekko, 2012

The average crushing work index determined on 20 mineral samples (-76+50 mm) was 12.5 kWh/t and ranged from 2.8 to 26.6 kWh/t. The abrasion index was measured as 0.0585 g. The VSI crushing amenability test was conducted on sample crushed to 100% passing 11.2 mm in both the open and closed circuits. The single pass test produced a modest product size reduction; however, the closed-circuit test results (Table 13-7) showed a low amenability of the tested samples to VSI.

Table 13-7: VSI cyclic amenability test results Gekko

Product Size (µm)	Product Size Produced (%)	Circulation Load at Product Size (%)	Amenability
1,180	26	379	Low
850	21	466	Low
600	17	585	Low

Source: Gekko (2012)

13.3 Gold Recovery – Metallurgical Tests

Extensive metallurgical test programs for gold recovery were completed on the Tuvatu gold samples between 1997 and 2018. Three major treatment routes were investigated:

- Route 1: Whole-Ore Cyanidation
  - Direct cyanide leaching on ground samples
- Route 2: Gravity Concentration + Cyanidation
  - Gravity concentration on head samples, with gravity tailings subject to cyanide leaching process
- Route 3: Gravity Concentration + Flotation +Cyanidation
  - Gravity concentration on head samples, with gravity tailings subject to flotation concentration followed by cyanide leaching processes on both flotation concentrates and tailings.

13.3.1 Previous Metallurgical Test Work 1997 to 2015

Five laboratories, including AMMTEC, Metcon, OMC, Gekko, ALS Metallurgy, and Metallurgical Company Yantai Jinpeng Group (Jinpeng Group), conducted the initial test work for preliminary processing flowsheet development test work and optimization.

13.3.1.1 Route 1 Test Work: Whole-Ore Cyanidation

Cyanidation on Ground Head Samples – Bench-Scale

Both bench-scale and large-scale whole-ore leaching test work on ground head samples were conducted since 1997. The batch-scale tests were first carried out by AMMTEC and OMC in 1997 and was further investigated by Gekko in 2012 and then the Jinpeng Group and ALS Metallurgy in 2015 and afterwards. Table 13-8 lists all the leaching test results obtained from the batch-scale testing.

Table 13-8: Route 1 bench-scale whole-ore cyanidation test results, 1997 to 2015

Description	Unit	AMMTEC 1997		OMC 1997	Gekko 2012	ALS Metallurgy 2016
Sample Source	-	Nasivi	Upper Ridges	n/a	n/a	n/a
Variability/Composite	-	Variability	Variability	n/a	Composite	Composite
Head P_{80}	µm	75	75	n/a	106	75
Head Grade						
Average	g/t Au	16.5	7.6	3.63	7.13	18.0
Minimum	g/t Au	1.35	1.17	n/a	n/a	n/a
Maximum	g/t Au	136.9	30.0	n/a	n/a	n/a
Leach Time	h	24	24	48	24	72
Residue Grade						
Average	g/t Au	1.15	0.70	tbc	2.29	1.84
Minimum	g/t Au	0.25	0.16	n/a	n/a	n/a
Maximum	g/t Au	3.28	3.47	n/a	n/a	n/a
Gold Recovery						
Average	%	82.2	87.5	82.7	68.7	89.8
Minimum	%	60.6	56.5	n/a	n/a	n/a
Maximum	%	98.3	96.7	n/a	n/a	n/a

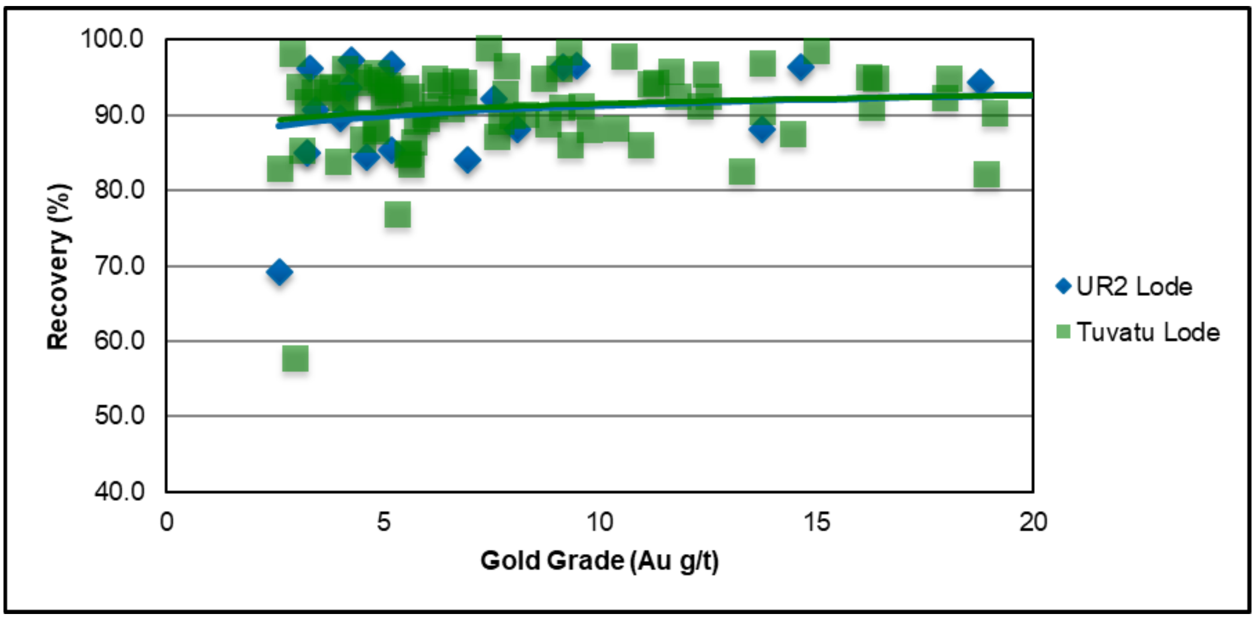
AMMTEC's leaching tests employed various core samples from Nasivi and Upper Ridges zones. The gold leaching recoveries varied significantly between 60.6 to 98.2% for Nasivi samples and 56.5 to 96.7% for Upper Ridges samples. The average gold leaching recovery was 82.2% for Nasivi samples and 87.5% for Upper Ridges samples.

OMC, Gekko, and ALS Metallurgy tests were completed on composite samples and produced gold leaching recoveries of 82.7%, 68.7%, and 89.8%, respectively. The variations in the gold leaching extractions may be related to multiple factors such as mineralogical characters, sample particle size distributions, and test conditions.

Jinpeng Group's tests were performed on two composite samples grading at 39.5 g/t Au and 7.5 g/t Au, respectively. For the 39.5 g/t Au sample, a high gold leaching extraction ranging from 91.5 to 92.9% was obtained. The 7.5 g/t Au sample gave a lower gold leaching extraction between 63.3 and 68.1%. The leaching residues, which contained 2.33 g/t Au, were further floated to improve gold recovery. The flotation concentrate contained 17.26 g/t Au and additionally recovered approximately 18.7% of the gold.

ALS Metallurgy conducted a cyanidation test on a bulk composite sample using direct cyanidation. The test results show that at a primary grind size of 80% passing 75 µm, 89.8% of the gold and 82.1% of the silver were extracted from a head sample containing 15.4 g/t Au and 9.0 g/t Ag.

The 2015 PEA (Freudigmann et al. 2015) shows that ALS Metallurgy undertook one-hour LeachWELL™ tests on 29 trench samples from the UR2 Lode and 98 samples from the Tuvatu Lode (Figure 13-1). The samples had a grade higher than 3 g/t Au. The average gold recovery from the UR2 and Tuvatu Lode trench samples was 91.8% and 91.5%, respectively, at a grind size of 80% passing approximately 75 µm.



Source: Freudigmann et al. (2015)

Figure 13-1: LeachWELL™ tests on UR2 and Tuvatu Lode trench samples – ALS 2015

Freudigmann et al. (2015) indicated that although the LeachWELL™ test results were considered only indicative, both the UR2 and Tuvatu Lode trench samples responded well, with only 1 sample returning a gold recovery less than 84% from the UR samples and only 7 of the 98 tests below 84% gold recovery from the Tuvatu Lode samples. The report points out that the variation in the results between this test work and the historical test work may be due to the difference in the level of oxidation of the trench samples.

Cyanidation on Ground Whole Ore Samples – Large-Scale

Large-scale, whole-ore leaching was investigated in the two batch industrial campaigns in 1998, which were conducted at Emperor mill on the samples from the Tuvatu adit. Both the campaigns tested the metallurgical responses of the materials to direct cyanidation and flotation followed by cyanidation. The test results are summarized below:

- Campaign 1: The sample tested was 968 t at a head grade of 3.63 g/t Au. Cyanide leaching recovered 82.7% of the gold with a leaching retention time of 48 hours. The test material was amenable to flotation concentration, which recovered approximately 88.5% of the gold. Gold leach extractions from the flotation concentrate and tailings were 92.8% and 36.9%, respectively. Overall gold recovery by the combined treatment of flotation and cyanidation was 86.3%.
- Campaign 2: The total sample used for the campaign was 1,095 t from the Tuvatu adit (consisting of two blocks of the Tuvatu samples). The average head grade of the campaign was 4.79 g/t. A high amount of gold recirculating in the cyclone underflow was noted. It was indicated that high recirculation load was due to occurrence of coarse free gold. Direct cyanidation extracted approximately 84.0% of the gold from the feed, while flotation followed by cyanidation on both the flotation concentrate and tailings recovered approximately 86.7% of the gold.

Cyanidation on Crushed Head Samples

In 1999, two cyanide heap leaching tests were tested using a column leaching procedure. Two columns were set up separately: one column was loaded with a 565 kg sample (as received) and leached for 31 days, and the other column was loaded with a 457 kg crushed sample (finer than 20 mm) for 37 days. The reconciled recovery was 56.4% for the as-received sample and 57.1% for the crushed sample. Both lime and cyanide consumptions were low to moderate.

13.3.1.2 Route 2 Test Work: Gravity + Cyanidation

Gravity concentration and cyanidation on gravity tailings tests were carried out by AMMTEC in 1997, Metcon in 2000, and Jinpeng Group in 2015 (Table 13-9 and Table 13-10).

AMMTEC's test work indicated that the overall gold recovery using Route 2 was between 89.8 and 90.5% for the Nasivi samples and between 90.5 and 93.2% for the Upper Ridges samples. The gravity concentration produced approximately 31 to 32% gold recoveries for both the samples that confirmed the existence of GRG. The leaching processes were carried out on the gravity tailings at various particle size for up to a 24-hour leach retention time. Finer particle size seems to be beneficial to the gold recovery by the leaching treatment. Table 13-9 lists the results produced by AMMTEC.

Table 13-9: Composite test results via Route 2 – AMMTEC 1997

Description	Unit	Value	
Head Sample	-	**Nasivi**	**Upper Ridges**
Particle Size, 80% Passing	µm	93	95
Grade	g/t Au	11.0	7.07
Gravity Concentration			
Tailings Grade	g/t Au	7.50	4.87
Gold Recovery	%	31.9	31.1

table continues…

Description	Unit	Value	
Gravity Tailings Cyanidation			
Leach Retention Time	h	24	24
Head Grade			
@P_{80} of 95 µm	g/t Au	7.96	5.38
@P_{80} of 75 µm	g/t Au	8.01	5.59
@P_{80} of 63 µm	g/t Au	7.94	5.37
@P_{80} of 45 µm	g/t Au	7.84	5.56
Leach Recovery			
@P_{80} of 95 µm	%	85.1	86.2
@P_{80} of 75 µm	%	85.3	88.4
@P_{80} of 63 µm	%	85.8	89.8
@P_{80} of 45 µm	%	86.1	90.1
Overall Gold Recovery			
@P_{80} of 95 µm	%	89.8	90.5
@P_{80} of 75 µm	%	90.0	92.0
@P_{80} of 63 µm	%	90.3	93.0
@P_{80} of 45 µm	%	90.5	93.2

Source: Freudigmann et al. (2015)

Metcon's work of gravity and cyanidation tests were performed in four separate programs, identified as Programs 00807, 00827, 00840, and 00867. Program 00807 was conducted on three composite samples, identified as Composites Upper Ridges 1, 2, and 3 (Table 13-10). The test results show significant variations in gold recovery, including the gold reporting to gravity concentrates. The gold recovery reduces slightly at a coarser grind size (80% passing 150 µm) and improved slightly at a finer grind size (80% passing 53 µm).

Table 13-10: Test results via Route 2 – Program 00807, Metcon 2000

Description	Unit	Value		
Head Sample	**-**	**Upper Ridges 1**	**Upper Ridges 2**	**Upper Ridges 3**
Particle Size	P_{80} µm	75	75	75
Grade	g/t Au	7.80	2.43	1.00
Gravity Concentration				
Gold Recovery	%	52.8	28.2	18.5
Gravity + Cyanidation				
Gold Recovery	%	89.2	91.7	81.7

Source: Freudigmann et al. (2015)

In the same year, Metcon conducted a separate test program (Program 00827) to assess the effect of primary grind size on gold recovery using a gravity and cyanidation combined treatment. The sample containing 7.8 g/t Au and 2.35% sulphur was from the Upper Ridges 1 zone. The test results (Table 13-11) show that there was a significant amount of the GRG occurring in the sample. The overall gold recovery improved with an increase in primary grind fineness. At the finest grind size of 80% passing 38 µm, the overall gold recovery reporting to gravity concentrate and leach solution was 94.4%.

Table 13-11: Test results via Route 2 – Program 00827, Metcon 2000

Grind Size (80% passing, µm)	Recovery (% Au)			Reagent Consumption (kg/t)	
	Gravity	Cyanidation	Overall	Lime	NaCN
150	45.6	42.5	88.1	0.91	0.76
75	42.8	46.4	89.2	0.99	0.66
53	61.1	32.5	93.6	1.23	0.48
38	59.7	34.7	94.4	1.18	0.66

Source: Freudigmann et al. (2015)

A variability test program was conducted by Metcon (Program 00840) on various samples from the Upper Ridges' mineralization zones 1, 2, and 5. The test work produced an average overall gold recovery of 82.3%, ranging from 55.8 to 97.2%. Table 13-12 shows the test results.

Table 13-12: Variability test results via Route 2 – Program 00840, Metcon 2000

Zone	Drill Hole	Depth (m)	Head Grade (g/t Au)	Recovery (% Au)			Reagent Consumption (kg/t)	
				Gravity	Leaching	Overall	Lime	NaCN
URI	67	225.4-227.6	7.30	51.9	45.3	97.2	1.10	0.76
URI	158	232.5-236 15	2.32	11.3	84.4	95.7	0.72	0.55
URI	53/67	66 3 & 83.35	10.90	18.1	73.5	91.6	0.94	0.76
UR2	79	123.3-125.3	5.45	34.1	56.9	91.0	1.16	0.62
UR2	95	16.5-18.25	26.40	53.3	15.2	68.5	2.73	0.76
UR2	122/148	268.8 & 159.8	4.70	57.9	17.1	75.0	1.41	0.76
UR2	123	308-310.95	6.95	34.4	21.4	55.8	0.53	0.62
UR2	160	469.55-472.55	0.09	68.0	16.2	84.2	0.66	0.61
UR2	183	419.9-422.6	2.60	22.7	62.0	84.7	0.66	0.56
UR2	207	298-310.3	3.00	29.1	31.6	60.7	0.58	1.06
UR2	212	519.3-526	17.80	67.1	21.0	88.1	0.40	0.91

table continues…

Zone	Drill Hole	Depth (m)	Head Grade (g/t Au)	Recovery (% Au)			Reagent Consumption (kg/t)	
				Gravity	Leaching	Overall	Lime	NaCN
UR2	60/62	139.0-148.7 & 125.0-127.2	4.00	62.7	29.1	91.8	0.39	0.79
UR2	68/75	63.4-65.6 & 190.9-194	2.65	22.1	51.9	74.0	0.50	0.73
UR5	152	192-195	0.36	70.2	23.5	93.7	0.40	0.67
Average			**6.55**	**43.1**	**39.2**	**82.3**	**0.87**	**0.73**

Source: Freudigmann et al. (2015)

A further variability test program by Metcon (Program 00867) were conducted on the samples from western zones of the Upper Ridges 1, 2, and 3 with two samples from the Murau area, including three Upper Ridges 3 samples showing a significant metallurgical performance variation. The overall average gold recovery from these samples was 72.0% and varied from 53.9 to 96.5%. Table 13-13 shows the Metcon test work results.

Table 13-13: Variability test results via Route 2 – Program 00867, Metcon 2000

Zone	Drill Hole	Head Grade (g/t Au)	Recovery (% Au)			Reagent Consumption (kg/t)	
			Gravity	Leaching	Overall	Lime	NaCN
URW1	DDH207	12.60	40.9	36.7	77.6	0.34	0.71
URW2	DDH176	14.40	39.5	22.3	61.8	0.38	0.89
URW1	DDH160	89.40	23.5	39.0	62.5	0.32	1.66
URW2	DDH076	13.30	74.1	20.8	94.9	0.91	0.76
URW3	DDH144	3.47	18.7	57.6	76.3	1.65	1.17
URW3	DDH223	5.55	50.2	9.7	59.9	1.16	0.99
Murau 1	DDH225	12.20	37.5	16.8	54.4	0.32	0.77
URW2	DDH228	6.29	39.5	14.4	53.9	0.57	0.91
URWI	TUG057	7.26	54.1	35.4	89.5	0.41	0.68
URW3	TUG051	42.90	9.9	78.4	88.4	1.06	1.21
URW3	TUG040	8.16	78.0	18.5	96.5	0.69	0.94
Murau I	TUG102	1.84	22.8	35.3	58.1	0.47	0.79
UR2	DDH176	1.47	26.3	36.7	63.0	0.36	0.67

table continues…

Zone	Drill Hole	Head Grade (g/t Au)	Recovery (% Au)			Reagent Consumption (kg/t)	
			Gravity	Leaching	Overall	Lime	NaCN
UR2	DDH092	4.61	18.3	66.7	85.0	1.40	2.01
UR2	TUG073	7.04	37.7	20.4	58.1	0.44	0.70
Average		**15.40**	**38.1**	**33.9**	**72.0**	**0.70**	**0.99**

Source: Freudigmann et al. (2015)

Four of the fifteen samples, which showed inferior metallurgical performances from the testing, were further leached for an additional 24 hours after regrinding. The target regrind size was 80% passing 38 µm. As shown in Table 13-14, the regrinding and extended leaching retention time have improved the overall gold recovery, but not significantly.

Table 13-14: Effect of regrind and extended leaching retention time on gold recovery – Route 2 – Program 00867, Metcon 2000

Zone	Drill Hole	Feed Grade (g/t Au)	No Regrinding		Regrinding	
			Extraction at 24 h (% Au)	Extraction at 48 h (% Au)	Regrind Size (80% Passing µm)	Extraction at 48 h (% Au)
URW2	DDH176	5.50	61.8	63.8	30	67.3
Murau1	DDH225	5.55	54.4	59.7	34	60.2
URW2	DDH228	2.90	53.9	54.7	30	56.3
Murau1	TUG102	0.77	58.1	62.8	29	65.9

Source: Freudigmann et al. (2015)

Jinpeng Group's test work also used the gravity and leaching combined flowsheet to investigate the metallurgical performance of the high-grade, composite sample grading at 39.5 g/t Au. The overall gold recovery achieved was 96.1%. However, it appears that the sample is not representative. Therefore, the test results are only indicative of the mineralized material response to the gravity followed by cyanidation treatment.

13.3.1.3 Route 3 Test Work: Gravity + Flotation + Cyanidation Test Work

Route 3 treatment includes gravity concentration, flotation on gravity tailings, followed by cyanidation on flotation products for gold recovery. This route was tested by AMMTEC and OMC in 1997, Metcon in 2000, Gekko in 2012, Jinpeng Group in 2015, and ALS in 2016.

AMMTEC's work indicated that the overall gold recovery was 93.3%. The gravity concentration produced more than 30% gold recovery. The flotation process was completed on a composite of gravity tailings produced from both Nasivi and Upper Ridges samples. Cyanidation was completed on the floatation concentrates reground to a particle size of 80% passing 20 µm and on the flotation tailings without regrinding. Table 13-15 summarizes the test results.

Table 13-15: Composite test results via Route 3 – AMMTEC 1997

Description	Unit	Value	
Head Sample	**-**	**Nasivi**	**UR**
Grade	g/t Au	11.00	7.07
Particle Size	P_{80} µm	93	95
Gravity Concentration			
Tailings Grade	g/t Au	7.50	4.87
Gold Recovery	%	31.9	31.1
Gravity Tailings Flotation			
Head Grade	g/t Au	6.61	
Head P_{80}	µm	75	
Gold Recovery to Concentrate	% Au	90.5	
Gold Recovery to Tailings	% Au	9.5	
Flotation Concentrate Leaching			
80% Passing	µm	75	
Leach Recovery	% Au	91.8	
Residue Grade	g/t	1.70	
Flotation Tailings Leaching			
80% Passing	µm	75	
Leach Recovery	% Au	75.3	
Overall Gold Recovery	% Au	93.4	

Source: Freudigmann et al. (2015)

In 2000, Metcon tested variability samples using the Route 3 procedure in Programs 00807 and 00840. Program 00807 used Composite Upper Ridges 1 at a primary grind size of 80% passing 150 µm. Two tests reground the flotation concentrate separately to 80% passing 50 µm and 10 µm, while one test kept the concentrate as produced. The test results (Table 13-16) show that approximately 40.7% of the gold was recovered into the gravity concentrate. The cyanidation extracted approximately 47 to 50% of the gold from the concentrates depending on the regrinding particle size and additional 4.2% of the gold from the flotation tailings. Regrinding of the flotation concentrate slightly improved gold extraction.

Table 13-16: Test results via Route 3 – Program 00807, Metcon, 2000

Description	Unit	Value		
Head Grade (assay)	g/t Au	7.8		
Gravity Recovery	% Au	40.7		
Flotation				
Recovery to Concentrate	% Au	53.7		
Recovery to Tailings	% Au	5.6		
Cyanidation				
Flotation Concentrate				
Regrind Size, 80% Passing	µm	n/a	50	10
Extraction	% Au	47.1	48.6	49.6
Flotation Tailings				
Regrind Size, 80% Passing	µm	n/a		
Extraction	% Au	4.2		
Overall Recovery	% Au	92.0	93.5	94.5

Source: Freudigmann et al. (2015)

Metcon also conducted a variability test program (Program 00840) using the combined gravity, flotation, and cyanidation flowsheet. The test results show that the average overall gold recovery was 73.5% in a range of 44.2 and 94.5%. Table 13-17 shows the test results. On average, the test program produced inferior test results, compared to Program 00807.

Table 13-17: Variability test results via Route 3 – Program 00840, Metcon 2000

Zone	Drill Hole	Depth (m)	Head Grade (g/t Au)	Recovery (% Au)				Reagent Consumption (kg/t)	
				Gravity	Flotation	Leaching	Overall	Lime	NaCN
UR1	67	225.4-227.6	7.30	41.1	55.8	53.4	94.5	0.23	0.23
UR1	158	232.5-236.15	2.32	6.4	89.7	80.8	87.2	0.08	0.10
UR1	53/67	66 3 & 83.35	10.9	26.9	70.2	62.0	88.9	0.08	0.18
UR2	79	123.3-125.3	5.45	15.5	60.5	47.1	62.6	0.25	0.23
UR2	95	16.5-18.25	26.4	40.9	50.7	6.7	47.6	0.23	0.23
UR2	122/148	268.8 & 159.8	4.70	56.3	40.5	11.5	67.8	0.21	0.28
UR2	123	308-310.95	6.95	31.5	65.1	13.9	45.4	0.12	0.17
UR2	160	469.55-472.55	0.09	48.3	46.9	24.2	72.5	0.09	0.08
UR2	183	419.9-422.6	2.60	22.0	69.0	51.5	73.5	0.13	0.21

table continues…

Zone	Drill Hole	Depth (m)	Head Grade (g/t Au)	Recovery (% Au)				Reagent Consumption (kg/t)	
				Gravity	Flotation	Leaching	Overall	Lime	NaCN
UR2	207	298-310.3	3.00	19.3	49.6	24.9	44.2	0.12	0.20
UR2	212	519.3-526	17.8	55.0	41.2	36.1	91.1	0.10	0.29
UR2	60/62	139.05 & 125	4.00	45.8	53.0	47.6	93.4	0.15	0.19
UR2	68/75	63.45 & 1909	2.65	11.0	60.0	43.1	54.1	0.23	0.29
UR5	152	192-195	3.75	43.2	50.1	51.3	94.5	0.05	0.08
UR5	100	229.3-301.4	0.36	6.9	90.3	77.9	84.8	0.15	0.18
Average			**6.55**	**31.3**	**59.5**	**42.1**	**73.5**	**0.15**	**0.20**

Source: Freudigmann et al. (2015)

Gravity concentration tests conducted by Gekko included a GRG test using a Knelson concentrator or a two-stage continuous gravity recovery tabling (CGR). Each gravity test was followed by a flotation concentration treatment. The average head gold grade was 8.04 g/t Au. The major test results are shown as follows:

- Both the GRG and CGR test results indicated that the sample is amenable to gravity concentration with a significant improvement in gold recovery at a finer grind size.
- Approximately 43% of the gold was recovered to the final GRG concentrate at a grade of 341 g/t Au. As a comparison, the CGR test on the finer than 1.18 mm samples generated six cleaner concentrates that gave a combined gold recovery of 57.4% at a gold grade of 21.9 g/t Au.
- The rougher flotation tests were performed on both GRG and CGR tailings to further improve gold recovery. The overall gold recovery from the combined GRG and flotation procedure was 74.1% at a grade of 159.9 g/t Au, while the overall gold recovery of the combined CGR and flotation flowsheet was 61.8% at a grade of 82.4 g/t Au.

Jinpeng Group's tests were performed on two composite samples with one sample grading at 39.5 g/t Au and the other sample grading at 7.5 g/t Au. However, the two samples may not be representative of the entire Mineral Resource as the test results are only indicative.

- For the 39.5 g/t Au head sample, the combined gold recovery of tabling, flotation concentrate, and flotation tailings leaching processes was 96.1%, with a gravity gold recovery of 44.1% at a grade of 50.8 g/t Au.
- For the 7.5 g/t Au head sample, the combined gold recovery of tabling and flotation concentration was 94%, including approximately 39% of the gold recovered by gravity concentration with a concentrate grade of 920.8 g/t Au. The flotation recovered 55.1% of the gold to the flotation concentrate with a grade of 14.9 g/t Au.

Jinpeng Group also conducted separate cyanidation tests on a 42.4 g/t Au flotation concentrate and the 7.5 g/t Au head sample. The leaching tests results show that at both the grind sizes of 92% passing 74 µm and 97% passing 74 µm, only approximately 68.1% of the gold was extracted from the head sample with a leach retention time of 48 hours. The gold extractions from the reground concentrates (93% passing 45 µm and 98% passing 45 µm) were approximately 73% with a leach retention time of 48 hours. With a further extension of the leach retention time to 60 hours, an increase in gold extraction by 2% from the flotation concentrate was reported.

In 2016, ALS Metallurgy also used the Route 3 procedure to investigate the metallurgical response of a composite mineral sample to the combined flowsheet. The test results are shown in Table 13-18. The flotation concentrates produced were reground to two particle sizes (80% passing 20 µm and 80% passing 10 µm) prior to being cyanide leached for 72 hours. Gold recoveries were monitored at 8 hours, 48 hours, and 72 hours. The gold recoveries produced at the two regrind sizes appear to be similar. This suggests that the residual gold in the leach residue may be closely associated with its bearing minerals. There was no regrinding performed on the flotation tailings.

Table 13-18: Composite test results via Route 3 – ALS Metallurgy 2016

Description	Unit	Value
Head Sample		
Grade	g/t Au	14.70
Particle Size	P_{80} µm	79
Gravity Concentration		
Concentrate Grade	g/t Au	533
Gold Recovery	%	39.1
Gravity Tailings Flotation		
Feed Grade	g/t Au	8.95
Particle Size	P_{80} µm	75
Gold Recovery to Flotation Concentrate	%	52.4
Flotation Concentrate Grade	g/t Au	101.4
Gold Recovery to Flotation Tailings	%	8.44
Flotation Tailings Grade	g/t Au	1.36
Flotation Concentrate Cyanidation		
Particle Size	P_{80} µm	17.4/10.3
Gold Extraction, 72-hour	%	74.1/74.1
Flotation Tailings Leaching		
Particle Size	P_{80} µm	75
Gold Extraction, 72-hour	%	71.3
Overall Gold Recovery	**%**	**83.4**

Source: ALS Metallurgy (2016)

Note: Cyanide consumption: 7.3 kg/t concentrate; 2.6 kg/t tailings; silver extractions were approximately 90.5% for the flotation concentrate and 45.8% for the flotation tailings; extractions are circuit extractions.

13.3.2 Recent Test Work 2016 to 2020

After 2016, Xinhai and BV carried out further gold recovery test work. Xinhai's testing used the Route 2 (gravity + cyanidation) and Route 3 (gravity + flotation + cyanidation) treatments. Chemical pre-treatments were also tested prior to cyanidation. BV's test programs (Program 1801004 and Program 1801807) also conducted preliminary tests using the procedures similar to Route 2 and Route 3. Met-Solve Laboratories Inc. (Met-Solve) also conducted a test program in 2019 on two samples: one sample was similar to the composite sample used by BV in the test program of BV1801807 and the other sample on a separate composite generated from new drill core samples. Met-Solve used intensive leaching procedure for the gravity concentrates produced from the second stage of gravity concentration.

13.3.2.1 Head Samples

Xinhai tested three composite samples constructed from eleven subsamples, named as Sample I, Sample II, and Sample III; BV test work tested three composite samples; and Met-Solve tested two composite samples. Table 13-19 lists gold grades and other major elements of interest of the samples tested. It was noted that a high gold grade in Sample I of 116 g/t was caused by blending one individual very high-grade core sample (GRF) grading at 546 g/t.

Table 13-19: Major head assay results – Xinhai 2018 and BV 2018

		Xinhai 2018			BV1801004	BV1801807		Met-Solve MS1944	
Description	Unit	Sample I	Sample II	Sample III	Tuvatu Composite	Master Composite	Early Mill Feed Comp A/B	Batch 1 Composite	Batch 2 Composite
Assayed Gold Grade	g/t	116	8.25	31.3	17.15	3.39	10.87 [2]	10.01	9.92
Calculated[1] Gold Grade	g/t	57.2	7.94	22.5	-	-	-	9.96	8.85
Silver	g/t	22.8	9.75	8.17	4	2.8	-	2.0	-
Total Sulphur	%	2.26	2.91	2.83	2.18	-	-	-	-
Total Carbon	%	0.94	1.53	1.31	0.58	-	-	-	0.32
Organic Carbon	%	n/a	n/a	n/a	0.06	-	-	-	-
Graphite Carbon	%	n/a	n/a	n/a	<0.01	-	-	-	-
Total Tellurium	g/t	n/a	n/a	n/a	18.0	4.0	-	-	-

Note: [1]Calculated values based on blending ratios of individual core samples.
[2]Average data of two Early Mill Feed Comp A and B samples by metallic gold assay.

13.3.2.2 Route 2: Gravity + Cyanidation Test Work

Xinhai conducted a comprehensive test work program, including tabling and cyanidation. Xinhai also investigated oxidation pre-treatment on the gravity tailings before cyanide leaching in an effort to improve gold recovery. In the preliminary test program undertaken by BV, instead of using tabling as gravity concentration, a centrifugal concentration followed by panning was used to confirm sample amenability to the gravity concentration equipment proposed for the Project.

Table 13-20 shows the test results generated by Xinhai and BV. Both of the tests confirm that using gravity concentration and cyanidation on gravity tailings can effectively recover the gold from the samples tested. The test results obtained are summarized below:

- Xinhai's test results: The highest overall gold recovery was 95%, obtained from Sample I using gravity concentration followed by an oxidation pre-treatment and a 72-hour cyanide leaching at a particle size of 92% passing 75 µm; the leach residue grade was 5.88 g/t Au.
- BV's test results: The highest overall gold recovery was 94%, generated with the finest primary grind size of 80% passing 20 µm. The test was conducted using a combined treatment of gravity concentration and 72-hour direct cyanidation; the residue contained 1.05 g/t Au.

Both the laboratories observed that finer grind size and longer leach retention time had positive impacts on overall gold recoveries. The Xinhai test results also indicate that chemical pre-treatment before cyanidation would improve the overall gold recovery. However, Xinhai did not report the chemicals used for the pre-treatments.

Table 13-20: Test results via Route 2 – Xinhai 2018 and BV 2018

<table>
<tr><th rowspan="2">Description</th><th rowspan="2">Unit</th><th colspan="2">Xinhai 2018[1]</th><th colspan="4">BV1801004[2,3]</th></tr>
<tr><th colspan="2">P_{92} of 75 µm</th><th>P_{80} of 75 µm</th><th colspan="2">P_{80} of 54 µm</th><th>P_{80} of 20 µm</th></tr>
<tr><td colspan="8">Head Sample</td></tr>
<tr><td>Head Grade</td><td>g/t</td><td colspan="2">113.9</td><td>16.8</td><td colspan="2">19.1</td><td>18.7</td></tr>
<tr><td colspan="8">Gravity Concentration</td></tr>
<tr><td>Concentrate Grade</td><td>g/t</td><td colspan="2">1,016</td><td>7,834</td><td colspan="2">16,749/4,535</td><td>5,223</td></tr>
<tr><td>Gold Recovery</td><td>%</td><td colspan="2">21.0</td><td>25.4</td><td>31.4</td><td>25.9</td><td>28.9</td></tr>
<tr><td colspan="8">Gravity Tails Cyanidation</td></tr>
<tr><td>Leach Method</td><td>n/a</td><td>Direct Leaching</td><td>Pre-oxidation + Leaching</td><td>Direct Leaching</td><td colspan="2">Direct Leaching</td><td>Direct Leaching</td></tr>
<tr><td colspan="8">Residue Grade</td></tr>
<tr><td>@24 h Leaching</td><td>g/t Au</td><td>27.0</td><td>15.0</td><td>n/a</td><td colspan="2">n/a</td><td>n/a</td></tr>
<tr><td>@48 h Leaching</td><td>g/t Au</td><td>19.6</td><td>11.8</td><td>n/a</td><td colspan="2">n/a</td><td>n/a</td></tr>
<tr><td>@72 h Leaching</td><td>g/t Au</td><td>11.3</td><td>5.9</td><td>2.28</td><td colspan="2">2.12</td><td>n/a</td></tr>
<tr><td>@96 h Leaching</td><td>g/t Au</td><td>n/a</td><td>n/a</td><td>n/a</td><td colspan="2">2.11</td><td>1.05</td></tr>
</table>

table continues…

Description	Unit	Xinhai 2018[1]		BV1801004[2,3]		
		P_{92} of 75 µm		P_{80} of 75 µm	P_{80} of 54 µm	P_{80} of 20 µm
Leach Recovery						
@24 h Leaching	%	70.7	83.8	n/a	n/a	n/a
@48 h Leaching	%	78.7	87.2	n/a	n/a	n/a
@72 h Leaching	%	87.7	93.6	61.0	51.5	n/a
@96 h Leaching	%	n/a	n/a	n/a	62.8	65.4
Overall Gold Recovery						
@24 h Leaching	%	76.9	87.2	n/a	n/a	n/a
@48 h Leaching	%	83.2	89.9	n/a	n/a	n/a
@72 h Leaching	%	90.3	95.0	86.4	88.9	n/a
@96 h Leaching	%	n/a	n/a	n/a	88.7	94.3

Notes: [1]Xinhai's leach recovery data are based on circuit recovery.
[2]BV's leach recovery data are based on overall recovery.
[3]BV conducted two gravity tests with the same size fraction samples producing 31.4% and 25.9% gold gravity recoveries. The higher-recovery sample was subject to a 72-hour leaching; the lower recovery sample was subject to a 96-hour leaching.

Xinhai performed similar tests to those used on Sample I, Sample II, and Sample III. Table 13-21 summarizes the test results. The pre-oxidation was conducted for 16 hours prior to the 48-hour cyanidation. The dosage for the oxidizing reagent was 20 kg/t for all the samples. The cyanide dosage was high at 6 kg/t for Samples II and III, while 8 kg/t was used for Sample I. To reach an overall gold recovery over 90%, a grind size of 95% passing 45 µm for the low-grade composite (Sample II) seems to be required.

Table 13-21: Variable test results via Route 2 – Xinhai, 2018

Description	Unit	Sample I				Sample II				Sample III			
		P_{92} 75 µm	P_{96} 75 µm	P_{95} 45 µm	P_{90} 38 µm	P_{95} 75 µm	P_{85} 45 µm	P_{95} 45 µm	P_{95} [2] 45 µm	P_{92} 75 µm	P_{85} 45 µm	P_{95} 45 µm	P_{95} [2] 45 µm
Head													
Gold Grade	g/t	113.9	113.6	106.1	115.1	8.2	8.2	8.3	8.3	31.5	31.3	31.3	31.3
Gravity Concentration													
Concentrate Gold Grade	g/t	1,015	1,660	2,781	3,056	440	515	584	584	916	969	933	933
Gold Recovery	%	21.0	19.3	12.6	9.6	18.7	18.8	19.1	19.1	22.1	17.7	13.7	13.7
Gravity Tailings Cyanidation													
Feed Gold Grade	g/t	92.1	92.9	93.2	104.5	6.7	6.7	6.7	6.7	24.7	25.9	27.1	27.1
Residue Gold Grade	g/t	10.8	4.20	3.90	3.90	1.83	1.24	0.63	1.71	3.89	1.36	1.03	3.16
Leach Recovery[1]	%	88.3	95.5	95.8	96.3	72.7	81.5	90.6	74.5	84.3	94.8	96.2	88.3
Overall Gold Recovery	%	90.8	96.4	96.3	96.6	77.8	85.0	92.4	79.4	87.8	95.7	96.7	89.9

Notes: [1]Circuit recovery.
[2]Gravity tailings is cyanide leached without pre-treatment.

Xinhai also conducted leaching optimization tests using gravity tailings produced from Sample I with a grind size of 95% passing 45 µm. Using the optimized leaching conditions, a confirmation test produced a total gold recovery of 97.1%. The increase of gold recovery was 0.8% compared with the baseline test results of 96.3%. However, the pre-oxidizing reagent consumptions increased from 20 kg/t used in the baseline tests to 40 kg/t.

13.3.2.3 Route 3 Test Work: Gravity + Flotation + Cyanidation

Both Xinhai and BV conducted the tests using Route 3 that involved cyanidation on both flotation tailings and reground flotation concentrates. The effect of pre-oxidization treatment was also studied by Xinhai and BV. The Xinhai test program included preliminary and baseline tests on the three different grade composite samples, while BV's testing was conducted on two different grade composite samples. Table 13-22 shows the results of the preliminary tests using the Route 3 procedure.

Table 13-22: Test results via Route 3 – Xinhai 2018 and BV 2018

Description	Unit	Xinhai 2018		BV1801004		BV1801807	
Head Sample							
Head Gold Grade	g/t Au	116.8		14.3	16.8	4.37/3.96	3.95
Primary Grind Particle Size		P_{87} 74 µm		P_{80} 74 µm	P_{80} 53 µm	P_{80} 61 µm	P_{80} 61 µm
Gravity Concentration							
Concentrate Gold Grade	g/t Au	1,633		7,834	16,749	1,631/1,613	1,242
Gold Recovery	% Au	7.7		29.7	35.5	36.4/35.6	30.6
Flotation on Gravity Tailings							
Flotation Concentrate Grade	g/t Au	299.8		42.63	39.14	12.3/11.7	11.9
Gold Recovery to Concentrate	% Au	89.4		65.9	61.7	60.3/61.4	53.5
Flotation Tailings Grade	g/t Au	5.2		0.8	0.63	0.18/0.15	0.18
Gold Recovery to Tailings	% Au	2.9		4.4	2.8	3.3/3.1	3.5
Leaching on Flotation Concentrate							
Pre-treatment (Pre-oxidation)		n/a	Yes	n/a	n/a	n/a	Yes
Regrind		yes	yes	yes	yes	yes	yes
Regrind Size	P_{80}, µm	P_{95} 45 µm	P_{95} 45 µm	24	21	20/10	20
Leach Recovery, 48-hour	% Au	78.9 [1]	97.9 [1]	53.8	53.5	46.2/48.1 [2]	53.5 [2]
Residue Gold Grade	g/t Au	63.2	5.6	7.8	5.2	2.89/2.53	2.24
Leaching on Flotation Tailings							
Pre-treatment (Pre-oxidation)		n/a	Yes	n/a	n/a	n/a	n/a
Regrind Size	P_{80} µm	n/a	n/a	n/a	n/a	n/a	n/a
Leach Recovery, 24-hour	% Au	57.7 [1]	90.4 [1]	2.9	1.9	2.3/1.7 [2]	2.7 [2]
Residue Gold Grade	g/t Au	2.2	0.42	0.26	0.19	0.06/0.07	0.04
Overall Gold Recovery	% Au	**79.9**	**97.8**	**86.5**	**91.0**	**84.9/85.3**	**86.8**

Notes: [1]Circuit recovery.
[2]96-hour leach retention time

BV's testing on the high-grade sample (BV1801004) gave an overall gold recovery of 86.5% at a grind size of 80% passing 74 µm and 91.0% at a grind size of 80% passing 53 µm. High cyanide consumption rates were observed in the concentrate leaching as 5.76 kg/t and 5.41 kg/t for the two different tests with different primary grind sizes.

BV's testing on the low-grade sample (BV1801807) produced an overall gold recovery of approximately 85% without pre-treatment and 86.8% with pre-treatment. A further regrinding on the concentrate appears to slightly improve gold extraction.

Xinhai conducted pre-treatment and leach condition optimization tests on the flotation concentrates and tailings generated from three different samples to improve gold recovery. The process conditions tested include regrind size, pre-oxidization reagent dosage and pre-treatment retention time, cyanide dosage, and cyanidation retention time. The optimized process conditions developed by Xinhai are:

- Flotation concentrates:
  - Regrind size: 95% passing 45 µm or 90% passing 38 µm
  - Pre-oxidizing retention time: 16 hours
  - Leaching retention time: 48 hours
- Flotation tailings:
  - No regrinding
  - Pre-oxidizing retention time: 5 hours
  - Leaching retention time: 24 hours.

Sample II and Sample III were tested using the test parameters. Table 13-23 compares the test results achieved by Xinhai on the three different grade samples.

The overall gold recovery generated from Sample I increased from 79.9 to 97.8%. This was mainly caused by the improved gold recovery in the concentrate leach process, which increased from 81.2 to 98.1%. The overall gold recovery was 91.1% for Sample II and 95.9% for Sample III.

Table 13-23: Variability test results via Route 3 (with oxidation pretreatment) – Xinhai 2018

Description	Unit	Sample I	Sample II	Sample III
Head Sample				
Head Grade	g/t Au	116.8	8.2	31.3
Particle Size	-	87% – 74 µm	87% – 74 µm	87% – 74 µm
Gravity Concentration				
Concentrate Grade	g/t Au	1,633	327.7	627.8
Gold Recovery	% Au	7.7	16.7	13.4

table continues…

Description	Unit	Sample I	Sample II	Sample III
Flotation on Gravity Tailings				
Gold Recovery to Concentrate	%	89.4	71.8	76.1
Flotation Concentrate Grade	g/t Au	299.8	41.9	223.2
Gold Recovery to Flotation Tailings	% Au	2.9	11.4	10.5
Flotation Tailings Grade	g/t Au	5.2	1.1	3.7
Leaching on Flotation Concentrate				
Regrind Size	-	95% – 43 µm	95% – 43 µm	90% – 38 µm
Leach Recovery[2]	% Au	98.1	92.4	97.4
Residue Gold Grade	g/t Au	5.6	3.2	5.9
Leaching on Flotation Tailings				
Regrind Size	n/a	n/a	n/a	n/a
Leach Recovery[2]	% Au	90.4	71.8	82.7
Residue Gold Grade	g/t Au	0.5	0.31	0.64
Overall Gold Recovery[1]	**% Au**	**97.8**	**91.1**	**95.9**

Notes: [1]Efficiency of zinc precipitation as 99.65% was applied.
[2]Circuit recovery.

A separate test program (BV1803310) conducted by BV used a composite representing the initial year mill feed, grading at 10.6 g/t Au. The tests used Route 3 flowsheet. The test results are summarized in Table 13-24. The test results show that with concentrate pre-treatments by sodium hydroxide and peroxide, the overall gold recovery can be improved to approximately 91 to 92%. A finer grinding on the flotation concentrate also improved the gold recovery to approximately 92%. The test results of Test GFC9 may suggest that lowering sodium cyanide concentration from 2 to 1 g/L may have a slightly inferior impact on gold extraction from the concentrate. The large-scale gravity concentration test, which used 10 kg test charge to generate flotation concentrate for the cyanidation tests (Test GFC6 to GFC9), produced a gravity concentrate with 17,326 g/t Au at a gold recovery of 46.8%. This result is much better compared to Tests GFC4/GFC5, which were conducted using a 2 kg test charge.

Table 13-24: Test results via Route 3 – BV 2018/2019

Description	Unit	BV1803310				
Test ID		GFC4/GFC5	GFC6 [1]	GFC7 [1]	GFC8 [1]	GFC9 [1,2]
Calculated Head Grade	g/t Au	9.50	-	-	-	-
Primary Grind Particle Size	P_{80}, µm	61	60			
Gravity Concentration						
Concentrate Gold Grade	g/t Au	5,001	17,326			
Gold Recovery	% Au	40.0	46.8			

table continues…

Description	Unit	BV1803310				
Flotation on Gravity Tailings						
Flotation Concentrate Grade	g/t Au	19.4	26.8			
Gold Recovery to Concentrate	% Au	56.2	48.5			
Flotation Tailings Grade	g/t Au	0.5	0.55			
Gold Recovery to Tailings	% Au	3.8	4.7			
Leaching on Flotation Concentrate						
Pre-treatment (Pre-oxidation)	n/a	Aeration with Lime	Aeration with $NaOH+H_2O_2$	Aeration with $NaOH+H_2O_2$ followed by adding $Pb(NO_3)_2$	Aeration with Lime	Aeration with Lime
Regrind Size	P_{80}, µm	16	19	19	9	19
Leach Recovery, 96-hour	% Au	85.2	87.1	85.4	87.3	79.9
Leaching on Flotation Tailings						
Pre-treatment (Pre-oxidation)	n/a	Aeration with Lime	Aeration with Lime			
Regrind Size	P_{80} µm	n/a	n/a	n/a	n/a	n/a
Leach Recovery, 96-hour	% Au	64.0	61.8			
Overall Gold Recovery	% Au	**90.1**	**91.9**	**91.1**	**92.1**	**88.4**

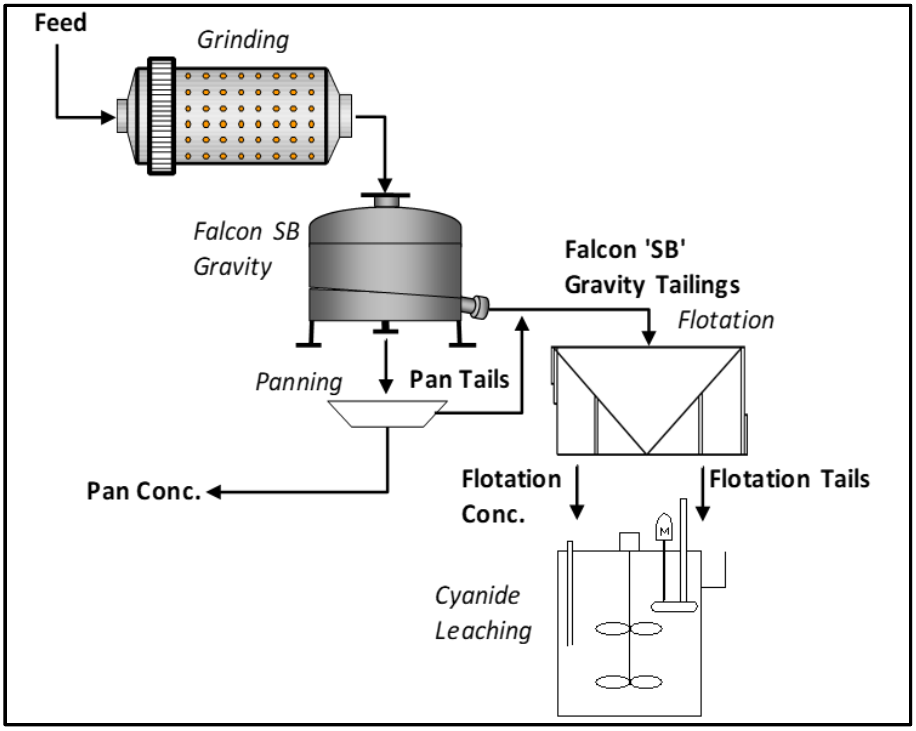
Notes: [1] 10 kg sample to produce gravity concentrate, flotation concentrate, and flotation tailings for leaching tests.
[2] 1 g/L NaCN dosage; other tests 2 g/L NaCN.

13.3.2.4 Gravity + Intensive and Conventional Cyanidation Test Work

In 2019, Met-Solve conducted a separate test program (MS1944) with the testing report submitted in 2020. Two composite samples were investigated. Batch 1 composite sample was constructed from the samples used by BV and Batch 2 composite sample was constructed from separate drill core samples.

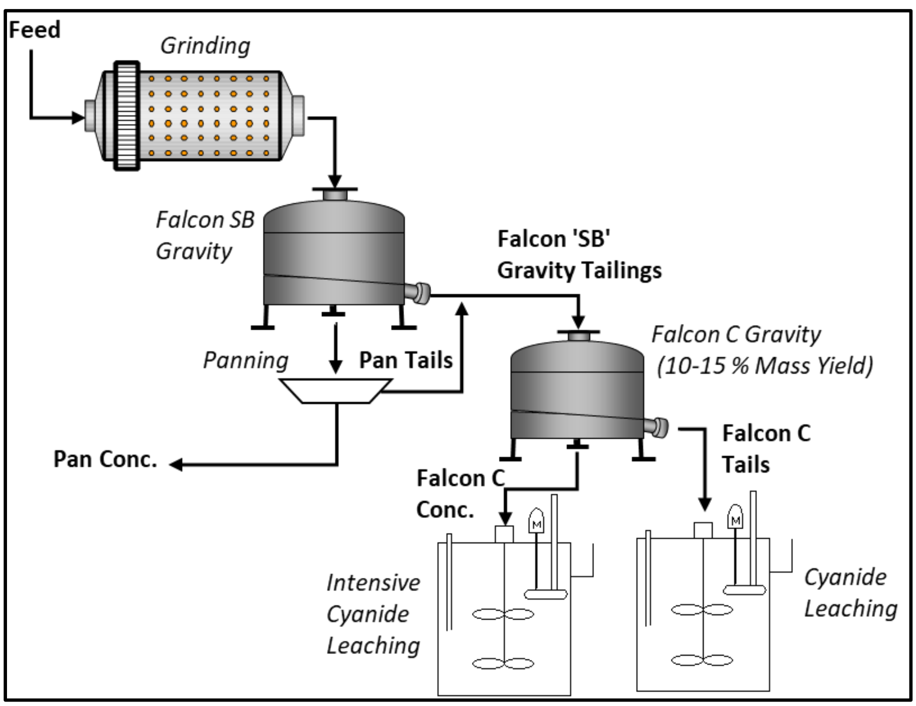
As shown in Figure 13-2 and Figure 13-3, two flowsheets were tested:

- Gravity concentration followed by flotation and then by cyanidation on the flotation concentrate and tailings (Flowsheet I; Figure 13.2);
- Gravity concentration followed by secondary gravity concentration on the tailings of the first stage of gravity concentration and then by intensive cyanidation on the concentrate of the secondary gravity concentration and conventional cyanidation on the tailings of the secondary gravity concentration separately (Flowsheet II, Figure 13-2).



Source: Met-Solve

Figure 13-2: Gravity concentration + flotation + cyanidation (Flowsheet I) – Met-Solve 2019



Source: Met-Solve

Figure 13-3: 1st Gravity concentration + 2nd gravity concentration + cyanidation (Flowsheet II) – Met-Solve 2019

The test conditions and the test results produced from the Batch 1 composite sample are summarized in Table 13-25. The test results appear to indicate that intensive leach can improve gold extraction from the high-grade gravity concentrate, which results in an overall gold recovery improvement, compared to conventional cyanidation only on flotation concentrate and tailings. Regrinding on the gravity concentrate prior to cyanide leach does not seem to improve the overall gold recovery. A finer primary grins size of 80% passing approximately 60 µm can achieve a better overall gold recovery, compared to a primary grind size of 80% passing coarser than 100 µm.

Table 13-25: Test results via Route 3 – Met-Solve 2019

Test ID	QL101a	QL101b	QL111a	QL111b	QL121a	QL121b
Test Conditions						
Flowsheet	I	II	II	II	II	II
Leach Retention Time, hours	96	96	24	24	24	24
Primary Grind Size, 80% passing, µm	60	60	63	63	103	103
Gravity Concentrate Regrind size, 80% passing, µm	n/a	n/a	n/a	24	n/a	33
Sodium Cyanide Consumption, kg/t	0.83	1.20	0.69	0.66	0.61	0.67
Lime Consumption, kg/t	0.69	0.64	0.25	0.25	0.20	0.20
Test Results – Gold Recovery						
Recovery – Gravity Concentrate, % Au	50.8	45.0	54.4	54.7	49.1	47.2
Recovery – Cyanide Leach, % Au	36.1	46.2	35.8	35.4	38.3	41.6
Recovery – Overall, % Au	86.9	91.2	90.2	90.2	87.4	88.7
Head Grade (Calculated), g/t Au	11.2	12.6	11.7	11.6	10.8	11.3
Residue Grade, g/t Au	1.48	1.11	1.15	1.14	1.37	1.27

Using Flowsheet II, Met-Solve conducted similar tests on the Batch 2 sample at primary grind sizes of 80% passing 70 and 80 µm, respectively. The tests produced an overall gold recovery in a range of 86.1 to 86.7%, which are lower than the recovery produced from the Batch 1 sample. The reasons for the inferior metallurgical response are not clear.

In general, it appears that the intensive cyanidation procedure produces promising results. Further test work is required to verify this finding and optimize intensive cyanidation conditions.

13.4 Cyanide Detoxification Tests

ALS Metallurgy and BV carried out cyanide detoxification tests using sulphur dioxide / air technology. Both studies indicated that the targeted WAD (CN_{WAD}) level of 1 mg/L was achieved.

13.4.1 ALS Metallurgy 2016 Test Work

ALS Metallurgy tested four leaching residue samples from the flotation concentrate (IS1702, IS1703), flotation tailings (IS1704), and bulk composite (IS1705) leaching tests. The pH level was maintained at approximately 8.5. Table 13-26 shows the test results, upon which the following conclusions can be made:

- The test results indicate that the target CN_{WAD} of less than 1 mg/L was achieved for all the samples tested (IS1702 to IS1705).
- For the flotation tailings leach residue, Test D3 was determined to be optimal using the lowest sulphur dioxide:CN_{WAD} ratio, the treated effluent contained a CN_{WAD} level lower than 0.5 mg/L. Copper sulphate, a catalyst of this process, was required to treat tailings leach residues to achieve the target CN_{WAD} level.
- For the flotation concentrate leach residues, a longer retention time and a higher sulphur dioxide:CN_{WAD} ratio were required due to a higher CN_{WAD} level. The add rate of copper sulphate can be optimized considering this reagent was already added during the preceding flotation stage.

Table 13-26: Cyanide detoxification test results – ALS Metallurgy 2016

Sample	Test ID	Time (min)	SO_2 (g/g CN_{WAD})	$CuSO_4.5H_2O$ (mg/L)	Lime (g/g SO_2)	Feed CN_{WAD} (mg/L)*	Treated CN_{WAD} (mg/L)*
IS1702	Bulk	210 (batch)	4.71	79	2.14	1,044	<0.20
IS1703	Bulk	210 (batch)	4.66	54	3.07	1,011	<0.2
IS1704	D1	82	4.77	77	0.87	370	0.67
	D2	83	3.90	78	0.82	370	0.99
	D3	84	3.00	78	0.76	370	0.46
	D4	84	3.00	0	1.07	370	1.22
	D5	57	2.95	78	0.72	370	0.98
	Bulk	87	2.76	75	0.44	370	1.37
IS1705	D1	57	4.03	79	0.54	316	0.32
	Bulk	87	3.73	78	0.43	316	<0.20

Note: *Picric acid method
$CuSO_4.5H_2O$ – Copper Sulphate Pentahydrate

13.4.2 BV 2018 Test

BV conducted one batch cyanide detoxification test on a flotation concentrate and tailings combined leaching residue sample. The pH level was maintained at approximately 8.7. Table 13-27 summarizes the test results and shows that target CN_{WAD} was achievable using the sulphur dioxide / air method. The required sulphur dioxide:CN_{WAD} ratio was 6.74 while copper sulphate:CN_{WAD} ratio was 0.21 g/g CN_{WAD}. Further test results are required to optimize the treatment conditions.

Table 13-27: Cyanide detoxification test results – BV 2018

Test ID	Time (h)	SO_2 (g/g CN_{WAD})	$CuSO_4$ (g/g CN_{WAD})	Lime (g/g CN_{WAD})	Feed CN_{WAD} (mg/L)*	Treated CN_{WAD} (mg/L)*
CD-1	4 (batch)	6.74	0.21	4.2	966	0.17

Note *Picric acid method

13.5 Conclusions

Since 1997, extensive metallurgical test work has been conducted on the samples collected from the Tuvatu Property. The test work covers mineralogy, comminution, gold and silver recovery, cyanide destruction, and process related parameter determination tests. The major conclusions are made as follows.

- The mineralogical determination shows that significant amount of the gold occurs in form of fine nugget gold grains. Some of the gold is closely associated with tellurium.
- The gold recovery tests indicate that the gold in the Tuvatu mineralization responded reasonably well to the process consisting of gravity concentration followed by further gold recovery by flotation and cyanidation. However, some metallurgical performance variations were observed from the test programs. Further test work should be conducted to optimize process flowsheet and conditions. Also it appears that the intensive cyanidation procedure produces promising results. Further test work is required to verify this finding and optimize intensive cyanidation conditions. The recommended test work is detailed in Section 26 of this Technical Report.

The test results appear to show that the gold recovery is more closely related to mineralogical characteristics, compared to feed grade. According to the test results produced, the average gold recovery reporting to the final doré is expected to be approximately 87.5%, including gold loss during gold-loaded carbon stripping and melting treatments.

14.0 MINERAL RESOURCE ESTIMATES

A number of historical Mineral Resource studies have been carried out for the Project by previous operators over the period from 1997 to 2010 (see Section 6.3).

Previous Mineral Resources were developed with classic techniques suited to broad zones of mineralization of relatively homogenous mineralization. Specifically, the compositing of individual samples to 1 m downhole and utilizing Inverse Distance Cubed linear weighting techniques of the capped data.

MA considers that a two-dimensional (2D) 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 Mineral 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 refolded model provides a more realistic vein model ideal for underground design or open pit design where veins come to surface.

Following the completion of the 2016/2017 diamond drilling program, MA undertook a study to update the Mineral Resources with the results of that drilling program and other work completed by Lion One to December 2017. In January 2018, MA was commissioned by Lion One to review the geology and create a Mineral Resource estimate for the Project. The Mineral Resource was estimated for each vein individually using OK of width and grade, the latter using accumulations, into a 3D block model.

MA updated the Mineral Resource Estimate in January 2018. The extrapolation of the vein extents is generally less than 30 m. The parent block size of the model is 10 m^3 with sub-blocks down to 0.3125 m^3. The small sub-block size allows more accurate design of stopes and dilution during the mine design phase.

In particular, drilling added significant additional information in the HT Corridor zone of mineralization (H and Tuvatu Lodes), whilst reinterpretation and surface work in the area of the Western Veins (which are interpreted to be the western extension of the Murau Lodes) also added significant information to Lion One's knowledge. Stricter parameters and tighter controls than those used for the 2014 estimate (which was put together for the 2015 PEA study) were used in this study. As a consequence of these tight controls, the Mineral Resource estimate related to some lodes was reduced in tonnes and/or grade. The study has not dramatically changed the interpretation or mineral inventory of that part of the mineralized body since the 2015 Mineral Resource report, Lion One's understanding of the geological controls within the area has increased considerably.

14.1 Approach

The main considerations as to whether a 2D or 3D estimate should be performed are as follows:

- Are there differences in mineral domains horizontally within the vein and can separating the vein improve grade estimation and control of feed grades?
- Does the mining equipment have the ability to mine smaller or multiple cuts across the vein? If not, and 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 mineralization across the vein, which are very narrow (less than 0.5 m in places), and no mining selectivity across the vein is possible. Thus, a 2D estimate of grade and thickness across the narrow vein is a better method to apply to the 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 (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	83,324
Collar	Collar information associated with drill type and location	1,841
Lithology	Logged lithological units	10,616
Specific Gravity Data	Bulk density data from drill core samples	2,079
Survey	Downhole survey data	6,915
Weathering	Logged oxidation codes	50,190

MA created a new table to store the vein intercepts in; veins were checked in three dimensions, cross section, and long section. Microsoft® Access queries were run to ensure mineralization was not excluded adjacent to composites and unnecessary waste samples were not included. There are examples of material below 0.5 g/t being included; however, these tags are required to constrain and ensure vein continuity.

There were mineralized samples outside vein tags (Table 14-2). The one exception to the high grade being excluded is in SKL8 where a 6.51 g/t Au mineralized sample is very improbable. Two additional "long" samples are recorded in the database 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: Mineralized samples outside vein tags

Hole_id	depth_from	depth_to	Au_ppm_BEST	Zonecode	Position
TUDDH-078	82.3	86.0	6.51	SKL8	Not included
TUDDH-42	32.5	37.5	1.58	M1	Included
TURC-21	7.0	10.0	0.63	M1	Included

14.3 Dimensions

Database extents (Table 14-3) are more extensive than the mineralized resource described in this report.

Table 14-3: Database extents

Database	Min (m)	Max (m)	Extents (m)
Northing	3917051.0	3923892.0	6841.0
Easting	1874687.4	1878637.9	3950.5
RL	99.75	547.00	447.3
Hole Depth	0.88	600.60	NA

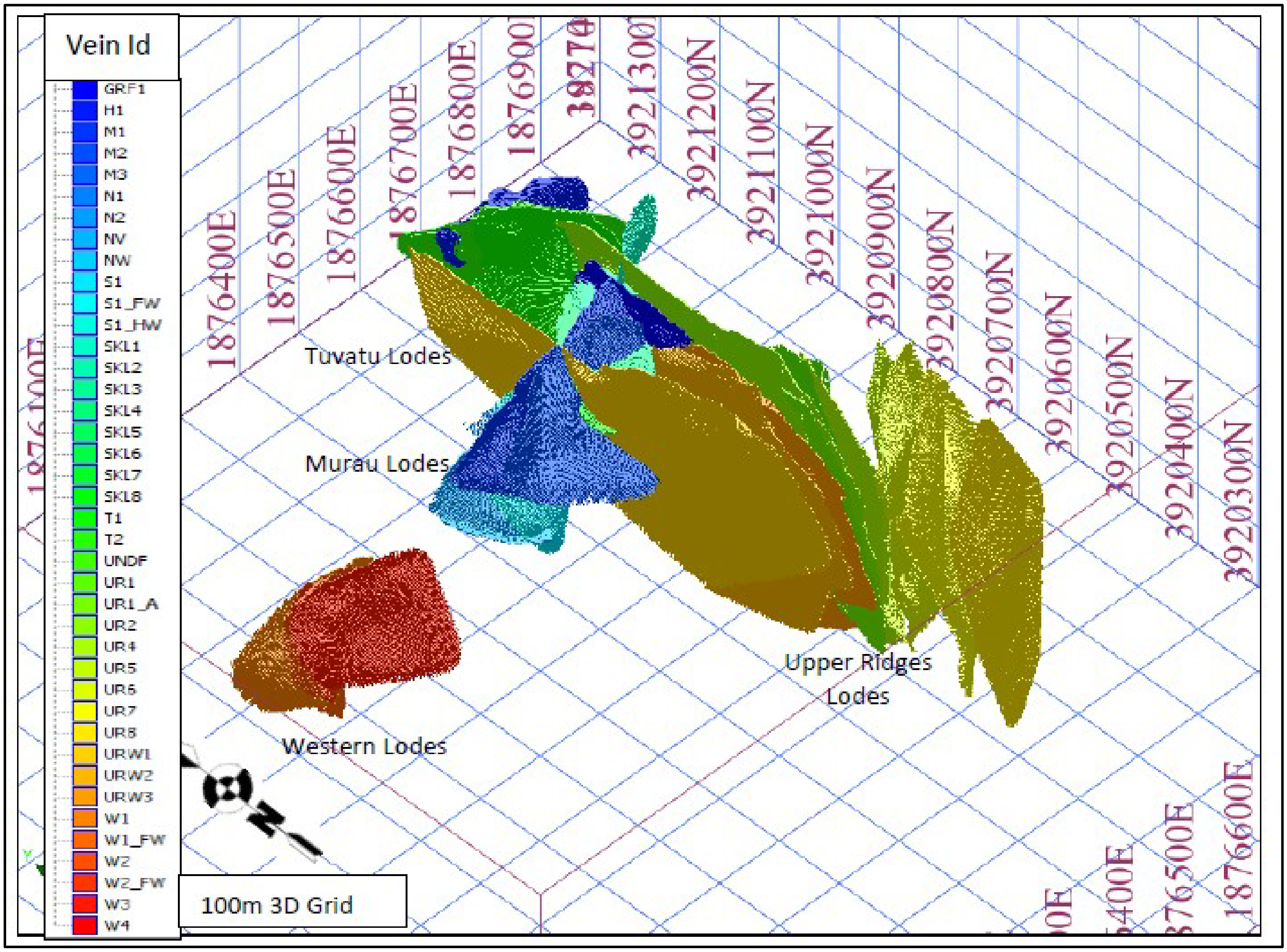
Although the Upper Ridges and associated north–south veins cover a strike extent of 900 m (3920200 m north to 3921100 m north), individual veins have shorter strike lengths. The Western Veins and Murau Veins strike approximately 800 m (1875750 m east to 1876650 m east) with a break between the Western Veins and Murau of approximately 150 m where the CABX fault (and associated andesite dyke) crosses mineralisation.

14.4 Geologic Interpretation

Tuvatu is the largest gold prospects known from the Sabeto area of northwestern Viti Levu, but forms only a small part of the Navilawa Caldera, the dominant geologic feature in the area. Navilawa itself is one of the major Fijian mineral systems aligned along the Viti Levu Lineament, located 45 km from the Vatukoula gold deposit, another large alkaline gold system, and one which has produced over 7 million ounces over the last 85 years.

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 rarely occur in volcanic units. Lodes are narrow, generally less than 1 m up to a maximum of 7 m, and metal grades are erratic. Lode mineralogy is varied, with most veins containing quartz, pyrite, and base metal sulphides. A total of 47 different lode structures were identified in the resource area including 11 lodes in the Upper Ridges area, 7 lodes in the Murau area, 7 lodes in the West area, 7 lodes associated with Snake and Nasivi lodes, 4 lodes in the Tuvatu area, and 9 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 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 0.5 m 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 0.5 g/t Au. These low grade intercepts are required to constrain and ensure vein continuity or pre-define the dilution grade.



Source: MA (2015)

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 mineralization. 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 semi-variogram analysis.

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

Statistical analysis of the grade data was principally carried out using the Surpac™ software package. Surpac™ was used to export composite drill hole data as a comma separated file (.csv) for importation into Supervisor™. More detailed spatial analysis (variograms) was conducted within Supervisor™. The Supervisor™ package is an internationally recognized 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 Data Spacing

Drill hole data spacing is variable within each domain. Above 50 m reduced level (RL), the drill spacing in Upper Ridges is reasonably tight on a 20 m grid, and below 50 m RL, the drill spacing increases to approximately 50 m grid. Upper Ridges Western Lodes are less well drilled. Development exists on UR2, UR5, and GRF Veins, and short drifts have been developed on the SKL Lodes. Murau Veins are shallower and are generally drilled at 20 m spacing. The 2016 and 2017 drilling has focused on the high grade proportions of the UR Veins and the SKL Lodes. Surface holes focused on the Tuvatu (T) and H Lodes.

14.5.2 Domains and Stationarity

A domain is a 3D volume that delineates the spatial limits of a single grade population, has a single orientation of grade continuity, is geologically 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 varies 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 2D technique used by MA to estimate Mineral Resources at Tuvatu uses a single downhole (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. The grade of each block is effectively the average of the estimated grade weighted by the length of the intervals. Scatter plots showed no correlation between grade and thickness; the variables (grade and thickness) are considered additive and were estimated in conjunction as gram*meter (g*m) and thickness (m). Gold grade was derived from these two variables by back calculation using g*m/m.

14.5.3.1 Channel Sample

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.

Generally, the underground channel samples return higher average grades than the accompanying drilling. Maximum sample grades in each data set are similar, confirming the underground channel data is collected from the high grade portions of the veins. While this data is higher in tenor, it is from the same domain, albeit highly clustered. Caution is required when considering areas of the model that have underground channel data, because at present, the orientation of high grade shoots is not known and there is a danger of over-estimation of grade in the wrong direction.

14.5.4 Summary Statistics

Summary statistics for vein gold, thickness, and grams multiplied by meters by area are presented in Table 14-4 and Table 14-5. Informing sample grades (uncapped) for the Upper Ridges Veins range from 2.66 g/t Au and 0.42 m thickness for UR7 and 10.12 g/t Au and 0.79 m thickness for URW1 (Table 14-4). In the Murau Area, veins range from 3.06 g/t Au and 0.89 m thickness for M1 to 7.60 g/t Au and 1.68 m thickness for Snake Vein (Table 14-5). SKL Veins have a very high vein at 12.45 g/t Au (Table 14-5).

Table 14-4: Summary statistics for Upper Ridges Veins

	Vein	UR1	UR2	UR3	UR4	UR5	UR 6,7,8	URW1	URW2	URW3	GRF1	GRF2
Gold	Number of Samples	47	184	65	88	85	75	83	57	110	68	20
	Minimum (g/t)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
	Maximum (g/t)	121.0	241.3	107.5	57.4	174.0	22.7	252.6	26.4	75.4	47.5	155.0
	Mean (g/t)	6.99	9.23	7.24	4.87	7.51	2.66	10.12	3.42	6.16	7.46	17.80
	Median (g/t)	18.40	24.60	14.35	10.14	21.12	4.15	30.91	5.86	12.28	9.41	36.19
	Standard Deviation	2.14	2.46	3.04	1.76	2.04	0.80	1.51	1.36	1.99	2.94	4.52
	Coefficient of Variation	2.63	2.66	1.98	2.08	2.81	1.56	3.05	1.71	1.99	1.26	2.03
True Thickness	Minimum (m)	0.09	0.09	0.13	0.02	0.07	0.02	0	0.14	0.08	0.04	0.06
	Maximum (m)	6.35	6.22	4.65	4.88	2.48	1.47	5.44	4.49	6.24	3.45	1.98
	Mean (m)	1.11	1.14	1.47	0.71	0.68	0.42	0.79	0.83	1.02	1.50	0.56
	Median (m)	1.19	1.04	1.09	0.76	0.58	0.28	0.94	0.82	1.02	0.99	0.47
	Standard Deviation	0.68	0.78	1.23	0.51	0.47	0.35	0.50	0.50	0.64	1.49	0.37
	Coefficient of Variation	1.07	0.91	0.75	1.06	0.85	0.67	1.19	0.99	1.00	0.66	0.84
Gram Meters	Minimum (g.m)	0.00	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Maximum (g.m)	35.48	248.49	174.11	49.93	67.32	8.21	401.71	50.65	98.37	108.68	49.60
	Mean (g.m)	5.49	9.61	13.13	3.23	3.70	1.04	11.35	3.96	6.58	13.84	7.53
	Median (g.m)	8.69	25.14	25.54	6.96	8.87	1.74	47.23	8.82	13.99	22.51	13.55
	Standard Deviation	1.30	2.24	3.72	0.77	1.19	0.25	0.51	0.67	1.34	5.51	2.51
	Coefficient of Variation	1.58	2.62	1.94	2.15	2.40	1.68	4.16	2.23	2.13	1.63	1.80

Table 14-5: Summary statistics for SKL, Murau, and Western Veins

	Vein	SKL 1,2,3,9	SKL4,5,6,7,8	H	T	M1	M2	M3 M1_FW&HW, M2_HW&HW, M1A	S1	W 1 2 3 4	W1 FW W2 HW	S FW HW
Gold	Number of Samples	75	178	60	103	45	41	68	24	75	21	21
	Minimum (g/t)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02	0.02	0.01
	Maximum (g/t)	37.0	176.0	34.2	11.9	15.4	109.0	57.9	51.0	45.6	9.5	36.8
	Mean (g/t)	4.86	12.45	3.88	2.57	3.06	5.50	4.32	7.60	6.72	2.18	5.41
	Median (g/t)	6.92	27.63	6.05	2.50	3.42	17.95	8.01	11.67	10.06	2.64	6.83
	Standard Deviation	3.10	2.55	1.56	1.79	1.74	0.01	2.44	3.35	2.26	1.14	2.85
	Coefficient of Variation	1.42	2.22	1.56	0.97	1.12	3.26	1.85	1.54	1.50	1.21	1.26
True Thickness	Minimum (m)	0.02	0	0.01	0.01	0.01	0	0.04	0.12	0.04	0.1	0.06
	Maximum (m)	4.87	3.71	6.54	7.68	4.34	1.83	5.34	5	6.16	3.68	3.68
	Mean (m)	0.89	0.78	1.09	1.94	0.89	0.54	0.88	1.68	1.64	1.15	1.03
	Median (m)	0.89	0.72	1.14	1.87	1.10	0.57	0.92	1.47	1.61	1.04	0.94
	Standard Deviation	0.61	0.50	0.84	1.37	0.50	0.01	0.53	1.07	0.86	0.66	0.72
	Coefficient of Variation	1.00	0.92	1.05	0.96	1.24	1.06	1.05	0.88	0.98	0.90	0.91
Gram Meters	Minimum (g.m)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00
	Maximum (g.m)	27.61	409.05	73.01	44.54	21.89	43.15	15.82	104.80	216.61	14.01	52.29
	Mean (g.m)	4.29	12.19	5.04	6.38	2.89	2.42	2.86	13.40	15.96	2.95	5.89
	Median (g.m)	5.89	36.77	11.75	9.84	5.40	7.12	3.77	24.19	36.60	4.07	10.37
	Standard Deviation	2.08	1.73	0.97	2.37	0.77	0.00	1.41	3.10	2.95	1.44	1.47
	Coefficient of Variation	1.37	3.02	2.33	1.54	1.87	2.94	1.32	1.81	2.29	1.38	1.76

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 minimizes 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 histogram log probability plots and metal loss. Uncapped and capped summary statistics are presented in Table 14-6. 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 14-7 and Table 14-8.

Table 14-6: Un-capped and capped summary statistics for gold per vein group

	Uncapped Composite Data				Capped Composite Data					Grade	
Domain	Count	Mean g/t Au	Maximum g/t Au	CV	Count	No. Capped	Mean g/t Au	Cap g/t Au	CV	% Cap	% ∆
GRF1	68	7.46	47.46	1.26	2	7.27	**35.6**	1.2	2.94%	-3%	GRF1
GRF2	20	17.80	155.00	2.03	1	15.66	**112.3**	1.8	5.00%	-12%	GRF2
H	60	3.88	34.15	1.56	2	3.59	**19.9**	1.4	3.33%	-7%	H
M1	45	3.06	15.42	1.12	2	2.95	**10.6**	1.0	4.44%	-4%	M1
M2	41	5.50	109.00	3.26	1	4.46	**66.3**	2.7	2.44%	-19%	M2
M Other	68	4.32	57.90	1.85	2	3.53	**18.7**	1.1	2.94%	-18%	M other
S1	24	7.60	51.04	1.54	1	7.15	**40.1**	1.4	4.17%	-6%	S1
SKL 1,2,3,9	75	4.86	37.00	1.42	2	4.78	**31.6**	1.4	2.67%	-2%	SKL 1,2,3,9
SKL4,5, 6,7,8	178	12.45	176.00	2.22	5	10.93	**97.9**	1.8	2.81%	-12%	SKL4,5, 6,7,8
T	103	2.57	11.86	0.97	3	2.53	**8.9**	0.9	2.91%	-2%	T
UR1	47	6.99	121.00	2.63	2	5.11	**35.3**	1.6	4.26%	-27%	UR1

Notes: CV – Coefficient of Variation; ∆ – Change in average metal

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Table 14-7: Un-capped and capped summary statistics for thickness per vein group

Domain	Uncapped Composite Data				Capped Composite Data				Grade	
	Count	Mean (m)	Maximum (m)	CV	#.Capped	Mean (m)	Cap (m)	CV	% Cap	% ∆
GRF1	68	1.50	3.45	0.66	1	1.49	3.4	0.7	1.47%	0%
GRF2	20	0.56	1.98	0.84	1	0.55	1.9	0.8	5.00%	-1%
H	60	1.09	6.54	1.05	1	1.08	5.9	1.0	1.67%	-1%
M1	45	0.89	4.34	1.24	1	0.88	4.2	1.2	2.22%	0%
M2	41	0.54	1.83	1.06	1	0.54	1.8	1.1	2.44%	0%
M other	68	0.88	5.34	1.05	1	0.87	5.0	1.0	1.47%	-1%
S1	24	1.68	5.00	0.88	1	1.68	4.9	0.9	4.17%	0%
SKL 1,2,3,9	75	0.89	4.87	1.00	1	0.89	4.6	1.0	1.33%	0%
SKL 4,5,6,7,8	178	0.78	3.71	0.92	1	0.78	3.7	0.9	0.56%	0%
T	103	1.94	7.68	0.96	3	1.94	7.7	1.0	2.91%	0%
UR1	47	1.11	6.35	1.07	1	1.10	5.8	1.0	2.13%	1%
UR2	184	1.14	6.22	0.91	1	1.13	5.2	0.9	0.54%	0%
UR3	65	1.47	4.65	0.75	1	1.47	4.6	0.7	1.54%	0%
UR4	88	0.71	4.88	1.06	1	0.71	4.8	1.1	1.14%	0%
UR5	85	0.68	2.48	0.85	1	0.67	2.4	0.9	1.18%	0%
UR 6,7,8	75	0.42	1.47	0.67	1	0.42	1.4	0.7	1.33%	0%
URW1	83	0.79	5.44	1.19	1	0.79	5.0	1.2	1.20%	1%
URW2	57	0.83	4.49	0.99	1	0.82	4.2	1.0	1.75%	1%
URW3	110	1.02	6.24	1.00	1	1.01	5.3	1.0	0.91%	1%
W 1 2 3 4	75	1.64	6.16	0.98	1	1.64	6.1	1.0	1.33%	0%
W1_FW W2_HW	21	1.15	3.68	0.90	1	1.15	3.6	0.0	4.76%	0%
S1_FW S1_HW, S2	47	1.03	3.68	0.91	2	1.02	2.9	0.0	4.26%	2%

Notes: CV – Coefficient of Variation; ∆ – Change in average metal

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Table 14-8: Un-capped and capped summary statistics for grams x thickness per vein group

Domain	Uncapped Composite Data				Capped Composite Data				Grade	
	Count	Mean (g*m)	Maximum (g*m)	CV	#.Capped	Mean (g*m)	Cap (g*m)	CV	% Cap	% Δ
GRF1	68	13.84	108.68	1.63	2	13.45	**83.0**	1.6	2.94%	-3%
GRF2	20	7.53	49.60	1.80	1	7.28	**44.5**	1.8	5.00%	-3%
H	60	5.04	73.01	2.33	2	4.34	**37.9**	1.9	3.33%	-14%
M1	45	2.89	21.89	1.87	2	2.86	**20.7**	1.9	4.44%	-1%
M2	41	2.42	43.15	2.94	1	1.99	**25.7**	2.4	2.44%	-18%
M other	68	2.86	15.82	1.32	2	2.84	**14.2**	1.3	2.94%	-1%
S1	24	13.40	104.80	1.81	2	12.41	**81.2**	1.7	8.33%	-7%
SKL 1,2,3,9	75	4.29	27.61	1.37	2	4.06	**18.3**	1.3	2.67%	-5%
SKL4,5,6,7,8	178	12.19	409.05	3.02	3	10.45	**107.8**	2.1	1.69%	-14%
T1, T2	103	6.38	44.54	1.54	3	6.21	**37.2**	1.5	2.91%	-3%
UR1	47	5.49	35.48	1.58	2	5.35	**29.2**	1.5	4.26%	-3%
UR2	184	9.61	248.49	2.62	3	8.19	**82.2**	1.8	1.63%	-15%
UR3	65	13.13	174.11	1.94	1	11.85	**90.5**	1.6	1.54%	-10%
UR4	88	3.23	49.93	2.15	2	2.82	**21.3**	1.7	2.27%	-13%
UR5	85	3.70	67.32	2.40	2	3.20	**28.7**	1.9	2.35%	-14%
UR 6,7,8	75	1.04	8.21	1.68	2	1.00	**6.6**	1.6	2.67%	-4%
URW1	83	11.35	401.71	4.16	2	7.07	**98.2**	2.5	2.41%	-38%
URW2	57	3.96	50.65	2.23	2	3.55	**28.5**	2.0	3.51%	-10%
URW3	110	6.58	98.37	2.13	3	6.01	**50.3**	1.8	2.73%	-9%
W 1 2 3 4	75	15.96	216.61	2.29	2	15.00	**155.2**	2.1	2.67%	-6%
W1_FW W2_HW	21	2.95	14.01	1.38	1	2.90	**12.9**	0.0	4.76%	-2%
S FW HW	47	5.89	52.29	1.76	2	5.57	**41.0**	0.0	4.26%	-5%

Notes: CV – Coefficient of Variation; Δ – Change in average metal

14.6 Variography

The most important bivariate statistic used in geostatistics is the semi-variogram. The experimental semi-variogram is estimated as half the average of squared differences between data separated exactly by a distance vector 'h'. Semi-variogram 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 semi-variogram analysis was undertaken in Surpac™ within each major vein; both gold and thickness were considered as separate variables. Variogram analysis was undertaken in unfolded space; thus, only the plunge was defined through variography analysis. Two-dimensional 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 mineralization 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.

The gram.meters variable is the primary variable used in defining spatial relationship of thickness and gold grades. The gram meter variable is used for estimating gram meters and thickness and the same variogram and search parameters are used for each variable to reduce the change of order relation issues.

Gram*meter variogram sills were standardized to 1. Nugget effects for gold were generally low to moderate, ranging from 0.11 to 0.69 and the range (A2) of the variograms varied from 12 to 95 m for gold variogram models.

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 (less than 90°) or below (greater than 90°) the horizontal.

Variogram parameters are summarized in Table 14-9 (gram.meters).

Table 14-9: Semi-variogram parameters for gram meters by vein group

Vein Set	Variogram Model	Plunge (Degrees from Vertical)	Max Range (m)	C0	C1	A1 (m)	C2	A2 (m)	Ratio1
GRF1,2	SPHERICAL	80	0.47	0.21	9	0.32	16	1.42	80
H1, 2	SPHERICAL	170	0.34	0.66	66.5	-	-	2	170
Murau 1 (HW FW)	SPHERICAL	70	0.11	0.90	85	-	-	2.15	70
Murau 2 & 3 (HW FW)	GAUSSIAN	70	0.29	0.71	73	-	-	1.88	70
Snake & Nasivi	GAUSSIAN	70	0.29	0.71	73	-	-	1.88	70
SKL 1-9	SPHERICAL	100	0.69	0.31	29.5	-	-	1.43	100
Tuvatu	GAUSSIAN	170	0.65	0.15	12.5	0.2	95	2	170
UR1 & 1N	SPHERICAL	110	0.13	0.87	45	-	-	2.4	110
UR2 & 2N	SPHERICAL	40	0.15	0.85	74	-	-	2	40
UR3	SPHERICAL	110	0.13	0.87	45	-	-	2.4	110
UR4, 6,7 & 8	SPHERICAL	100	0.31	0.32	12	0.38	36	1.5	100
UR5	SPHERICAL	100	0.32	0.12	12	0.56	34	1.5	100
URW1 & 1A	SPHERICAL	40	0.1	0.9	33	-	-	2.3	40
URW2, 2a & 3	SPHERICAL	40	0.28	0.72	40	-	-	1.64	40
Western Lodes	SPHERICAL	70	0.26	0.74	73	-	-	1.88	70

14.7 Grade Estimation

The drilling and channel data were examined using Surpac™ software package using the MA Proprietary System for Narrow Vein Modelling.

The MA Proprietary System for Narrow Vein Modelling estimates the grades and true widths of veins. This is done in unfolded space using 10 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 five different methods so that the results can be compared; these are Nearest Neighbour (NN) (capped), Inverse Distance Squared (ID^2) (capped), OK (uncapped) and OK (capped), and metal content (g*m). True widths are estimated directly using OK (capped).

One block model was created covering the entire Project. The final 3D block model utilized 10 m cubic blocks sub-blocked down to 0.3125 m cubes.

Table 14-10: Vein expansion distances

Vein name	Projection Plane	Extrapolation (m)	Vein name	Projection Plane	Extrapolation (m)
GRF1, 2	NS	20	T1,T2	EW	20
H1,H2	EW	20	UR1 & 1N	NS	20
M1	EW	20	UR2	NS	25
M1_FW, HW	EW	15	UR2N	NS	20
M1A	EW	20	UR3	NS	25
M2, HW, FW	EW	20	UR4 - 8	NS	30
M3, NV NW	EW	15	URW1	NS	22
S1	EW	15	URW1A	NS	20
S1_FW	EW	15	URW2 & 2a	NS	20
S1_HW	EW	12	URW3	NS	20
S1E	EW	20	W1, 2 & 3	EW	20
S2	EW	15	W1_FW, W1_HW	EW	20
SKL1,2	Z	10	W2_FW	EW	15
SKL3-9	Z	12	W4	EW	10

14.7.1 Methodology

The 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 downhole composites lengths for each vein.

2. Intercept Selection – 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). Internal waste of less than 0.5 g/t Au intervals and/or geologically continuous intervals just below cut-off may be included.

   b. Continuity – Waste (less than 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 g*m (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 is examined in 3D space. If more than one population is considered possible, the total population is decomposed into its component populations; 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 north–south striking veins are projected to the X axis, east–west veins are projected to the Y axis, and flat veins are 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 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 or variograms from similar veins sets where utilized.

5. Unfolded Grid Model and Extension – Generates a model of the vein centre using the coded intercepts, and estimates grades, vein true widths, and g*m. 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 five different methods so that the results can be compared. These are NN (capped), ID^2 (capped), OK (uncapped), and OK Upper (capped and g*m) estimates. The true widths are estimated directly using OK.

6. Minimum Width Application and Consequent Grade Dilution – Every 10 m x 10 m block in unfolded space with a vein width (in the perpendicular direction to strike) less than 0.3125 m is set to a width of 0.3125 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 0.3125 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 approximately 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. Capped Values Influence – Inverse Distance Squared method within a 30 m search radius was used to estimate a binary (between 0 and 1) indicator value where the indicator cut-off was the capping value. A "hybrid" estimate for all blocks was calculated by combining capped and uncapped estimated weighted by the indicator value. This effectively preserves very high grades within the final model, but limits their spatial influence.

8. Refolding and True Width Correction – The grid is refolded 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).

9. 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 in the respective direction to create footwall and hanging wall surfaces. 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".
10. 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.
11. Bulk Densities – The bulk densities for each block below the topographical surface are set to a constant value, using 2.57 t/m^3 for vein material.
12. Null Blocks – Blocks that are not present are flagged as air (above the original topography), pit (mined out in an open pit), or stoped (removed by underground mining).
13. Mineral Resource Categories – The Mineral 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.
14. 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.
15. Reporting – The Mineral Resource can be reported by Mineral Resource category, by vein, by cut-off grades, by different methods (sensitivity to method and upper cuts), by elevation (tonnes per vertical m), and by X and Y dimensions.

14.7.2 Block Model

The Tuvatu block model (tuvatu_20180104_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 mineralization, 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 0.3125 m x 0.3125 m x 0.3125 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 (m)	3920000.0	1875500.0	-250.0
Maximum Coordinates (m)	3921305.6	1876901.6	508.4
User Block Size (m x m x m)	10	10	10
Min. Block Size (m x m x m)	0.3125	0.3125	0.3125

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

Vein Set	Max Search	Ratio	Maximum Samples
GRF1	30	1.42	6
GRF2	30	1.42	4
H1, H2	60	2.00	6
M1	60	2.15	8
M1_FW, M1_HW, M1A	50	2.15	4
M2	60	1.88	8
M2_FW, M2_HW	50	1.88	4
M3, NV & NW	50	1.88	4
S1, S1_FW	50	1.88	6
S1_HW, S1E	50	1.88	4
S2	50	1.88	4
SKL1, 2	30	1.43	4
SKL3, 4	30	1.43	8
SKL5, 6	40	1.46	8
SKL7	40	1.46	6
SKL8, 9	40	1.46	4
T1, T2	50	2.00	6

table continues…

Vein Set	Max Search	Ratio	Maximum Samples
UR1	50	2.40	8
UR1N	50	2.40	4
UR2, 2N	70	2.00	8
UR3	60	2.40	8
UR4	60	1.75	8
UR5, 6	60	1.51	8
UR7	60	1.51	6
UR8	60	1.51	4
URW1, 1A	70	2.30	6
URW2, 2A	70	1.64	6
URW3	70	1.64	8
W1, W1_HW, W1_FW	50	1.88	4
W2	50	1.88	6
W3, W4 W2_FW	50	1.88	4

14.7.4 Discretization

The Krige estimate used a 5 x 5 x 1 discretization (X, Y, Z)., providing discretization nodes spaced evenly within the block, the projection plane direction has no thickness (2D unfolded space); thus, one discretization 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, Mineral 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_20180104_veins.mdl) attributes are defined in Table 14-14.

Table 14-13: Block model attributes

Attribute Name	Type	Background	Description
dist_near	Float	0	Distance to nearest sample
gold	Float	0	Estimated gold, hybrid of capped and uncapped, diluted for mining thickness
gold_hyun	Float	0	Gold hybrid uncapped undiluted
gold_id	Float	0	ID value for gold uncut, diluted for min_thk

table continues…

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Attribute Name	Type	Background	Description
gold_nn	Float	0	Gold value of nearest sample, diluted for min_thk
gold_ok	Float	0	Hybrid of capped and uncapped, diluted for min_thk
hg_ind	Float	0	Proportion of block influenced by uncapped sample
hole_id	Character	UNDF	Name of nearest drill hole
krig_var	Float	0	Kriging variance
min_thk	Float	0	Horizontal width across whole vein set to a mining width 1.2 m
num_samp	Integer	0	Number informing samples
silver	Float	0	Value for silver capped, diluted for min_thk
true_width	Float	0	True width across whole vein
vein_name	Character	UNDF	Name of vein
x_width	Float	0	Horizontal width across whole vein in x direction
xau_gm_capped	Float	0	Capped g*m estimate
xau_gm_uncapped	Float	0	Uncapped g*m estimate
xau_krig_capped	Float	0	Krige value for gold top cut according to vein
xau_krig_uncapped	Float	0	Krige value uncapped
z_width	Float	0	Vertical width across whole vein in z direction

Table 14-14: Final block model attributes (exported)

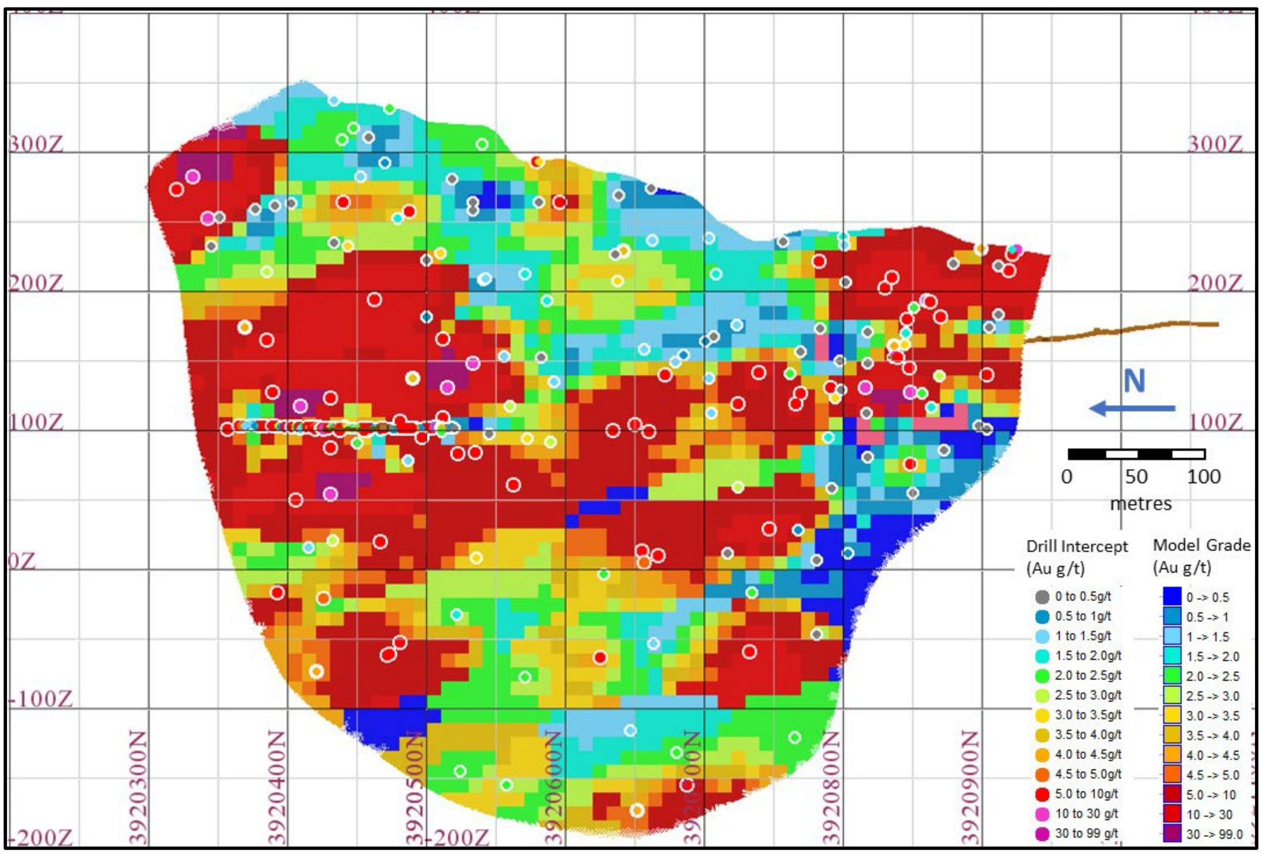
Attribute Name	Type	Background	Description
density	Real	2.61	Bulk density
dist_near	Real	0	Distance to nearest sample
gold	Float	0	Estimated gold, hybrid of capped and uncapped, diluted for mining thickness
gold_hyun	Float	0	Gold hybrid uncapped undiluted
hg_ind	Float	0	Proportion of block influenced by uncapped sample
hole_id	Character	UNDF	Name of nearest drill hole
krig_var	Float	0	Kriging variance
min_thk	Float	0	Horizontal width across whole vein set to a mining thickness 1.2 m
mined	Integer	0	0 insitu, 1 mined
num_samp	Integer	0	Number informing samples

table continues…

Attribute Name	Type	Background	Description
res_cat	Integer	3	1 measured, 2 indicated, and 3 inferred
silver	Float	0	Value for silver capped, diluted for mining thickness
true_width	Float	0	True width across vein
vein_name	Character	UNDF	Name of vein
x_width	Float	0	Horizontal width across vein in x direction
z_width	Float	0	Vertical width across vein in z direction

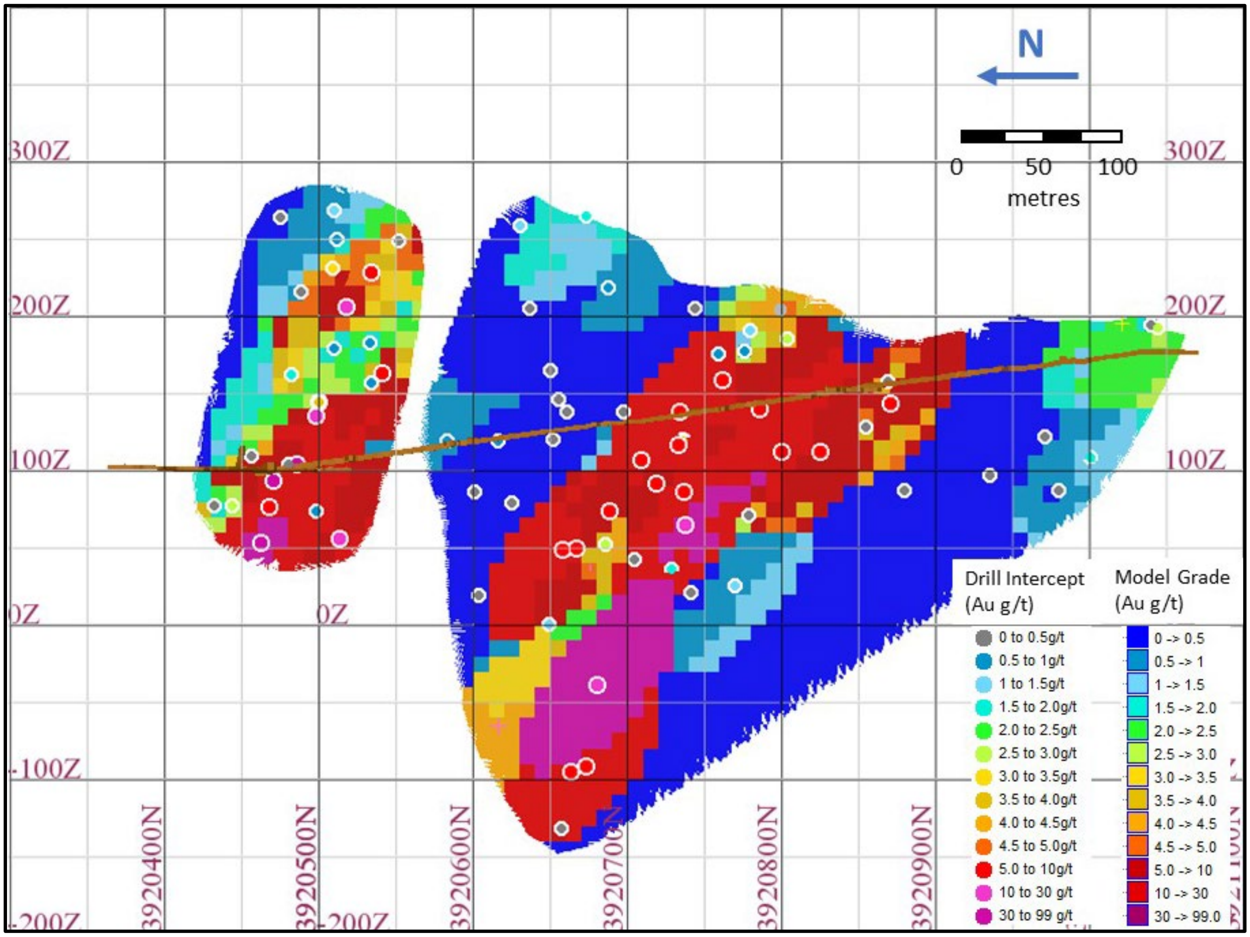
14.7.6 Validation

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. MA prepared vein long section views; two of the major veins are presented in Figure 14-2 and Figure 14-3.



Source: MA (2018)

Figure 14-2: UR2 estimated gold (long section view)



Source: MA (2018)

Figure 14-3: URW1 and URW1a estimated gold (long section view)

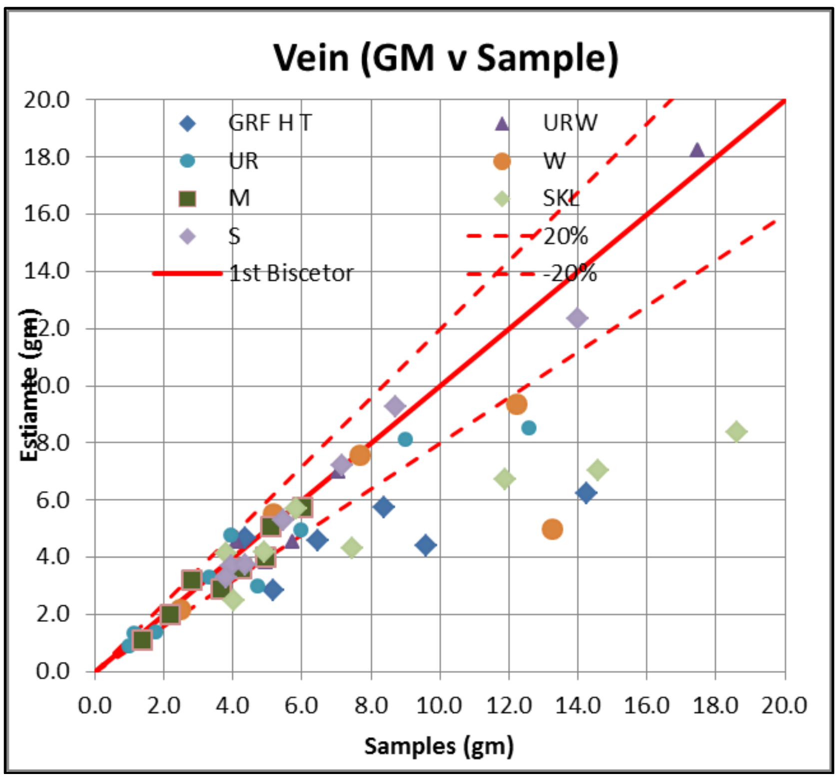
Alternative estimation methods (Table 14-15) were utilized to ensure the back-calculated grades from the g*m estimate was not reporting a global bias. The alternate estimates provided expected correlations. NN shows less tonnes and higher grade as it does not employ averaging techniques to assign the block grade. The ID^2 estimate is closer to kriging as it uses averaging weighted by distance but cannot assign anisotropy nor have the ability to decluster the input data. Using the kriging algorithm provides a reliable estimate due to the ability of kriging to decluster data and weight the samples based on a variogram (which incorporates anisotropy). Both the accumulations (g*m) and thickness estimates utilise the OK algorithm.

Table 14-15: Alternate estimation results at nominated cut-offs

Cut-off Grade	OK		Using g*m to Back Calculate Grade		ID^2		NN	
(g/t Au)	(Mt)	(g/t Au)	(Mt)	(g/t Au)	(Mt)	(g/t Au)	(Mt)	(g/t Au)
3.0	2.30	8.7	2.33	8.8	2.21	8.6	1.79	11.0

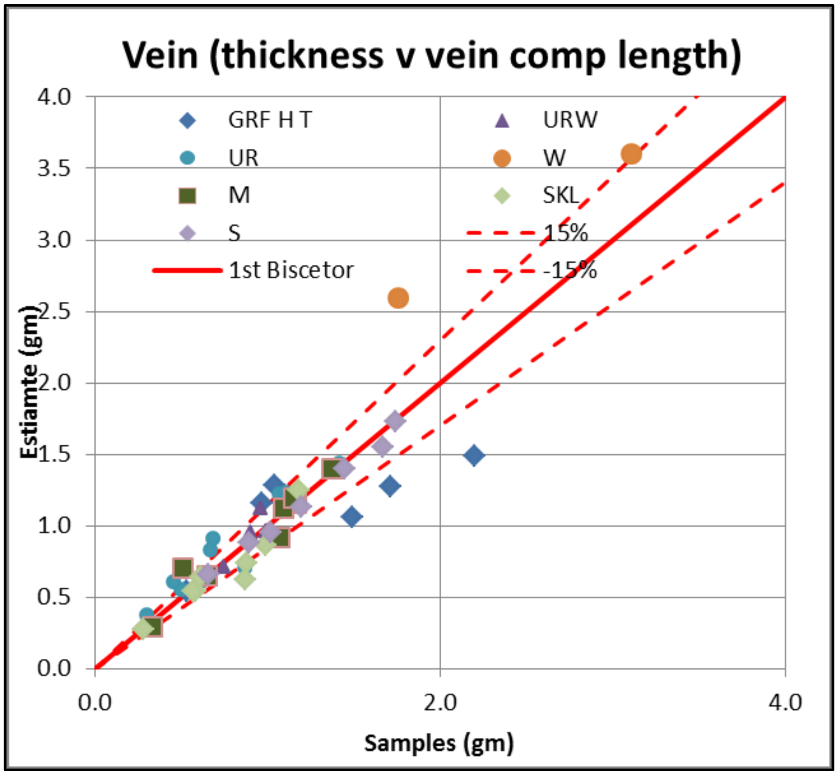
During validation, the grade estimates were assessed within the scope of the definition of Inferred Resources; this does not preclude higher confidence categories, and it is considered the majority of the estimate will convert to Indicated with further work. The guidelines applied to grades are ±20% (Figure 14-4 and Figure 14-5). SK Lodes (4, 5, 6, and 7) require additional work to identify which drifts have intercepted which lodes. These lodes have extreme outliers in the raw data, and a simple solution is further capping. SKL Veins are often thinner than at 0.3 m. GRF 1 and 2 show greater variance between samples and estimates due to the clustered nature of the back samples compared to drill results. W3 Lode is troublesome with four intercepts above 10 g/t, of which two are replicated with waste values. M3 has two high grade intercepts drilled down the vein negating a true width calculation and a default true width is assigned (5 cm) to the intercepts. UR5 and UR8 have estimated grades lower than the informing samples would indicate. UR5 informing composites include an isolated 0.18 m at 174 g/t sample, skewing the mean of the composites. UR8 has two close spaced high grade assays (~10 g/t), one at around 300 mRL and the other at -100 mRL with two low grade intercepts informing the intervening material (data clustering issues).

Reflecting the uncertainty in estimation (Nasivi and Snake), the majority of SKL and Western Lodes are classified as Inferred.



Source: MA (2018)

Figure 14-4: Vein accumulations, estimate vs. informing composites



Source: MA (2018)

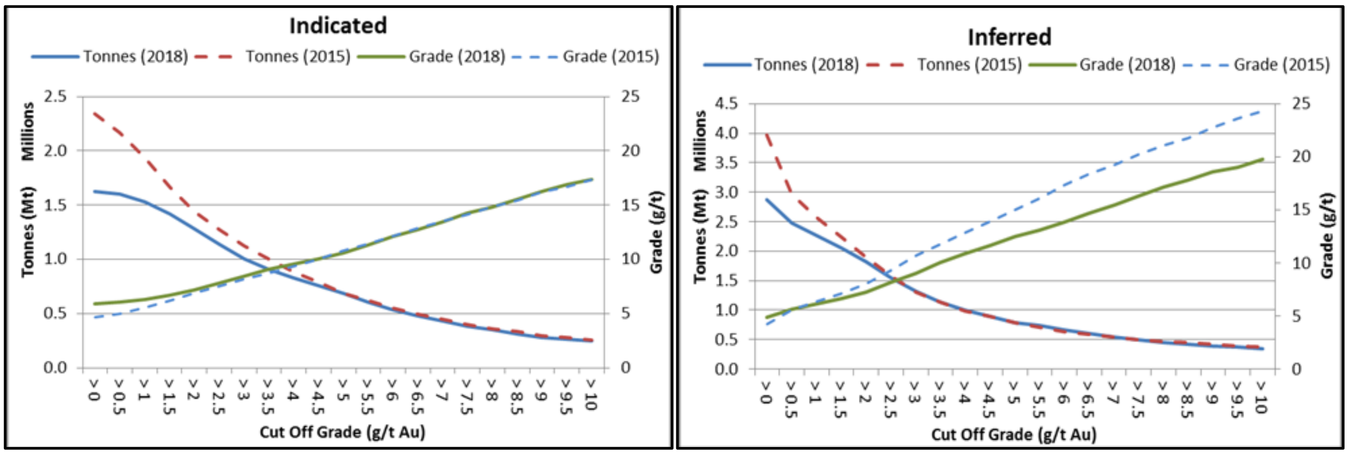
Figure 14-5: Vein thickness, estimate vs. informing composites

MA recommends further work (grade control drilling) targeting specific veins critical to the mine schedule. 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 14-3). There is evidence for shoots within this lode but not as high grade as the two intercepts (100 and 252 g/t Au) might suggest. A further two holes have targeted the 252 g/t intercept, but results were not received at the time of this report. The recent drilling has increased the average vein thickness in URW1 from 0.73 m in 2015 to 0.84 m, but dilution is still a major factor in the final grade. Currently a minimum mining width of 0.3125 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.

14.8 Economic Cut-Off Parameters

Mineral Resources have been reported above a 3 g/t Au cut-off assuming potential underground mining of veins with narrow widths from 0.3 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 assumed average stope parameters are 60 m long x 60 m tall x vein thickness. The assumed required head grade is 5.0 g/t Au and as is shown in Figure 14-6, the average head grade above a 3 g/t cut-off is 8.48 g/t in the Indicated Resource. It is assumed low grade material (less than 5 g/t) mined during development will be stockpiled.



Source: MA (2018)

Figure 14-6: Tuvatu grade tonnage charts compared to January 2015 resource (dotted lines)

The global Mineral Resource reported above various cut-offs is presented in Table 14-16. At the higher cut-off for all material of 3 g/t Au, there is an Indicated Resource of 1,007,000 t at 8.48 g/t Au for 274,600 oz and an Inferred Resource of 1,325,000 t at 9.0 g/t Au for 384,000 oz.

Table 14-16: Tuvatu resource reported at 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	1,530,000	6.27	308,400	2,265,000	6.1	444,600
2.0	1,283,000	7.19	296,400	1,822,000	7.2	423,300
3.0	1,007,000	8.48	274,600	1,325,000	9.0	384,000
5.0	687,000	10.60	234,300	788,000	12.5	317,500

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.

The resource by 20 m vertical increments is presented in Figure 14-7, with the majority of the mineralisation lying between 170 and 90 m RL with the highest grades deeper between -30 and -70 mRL.

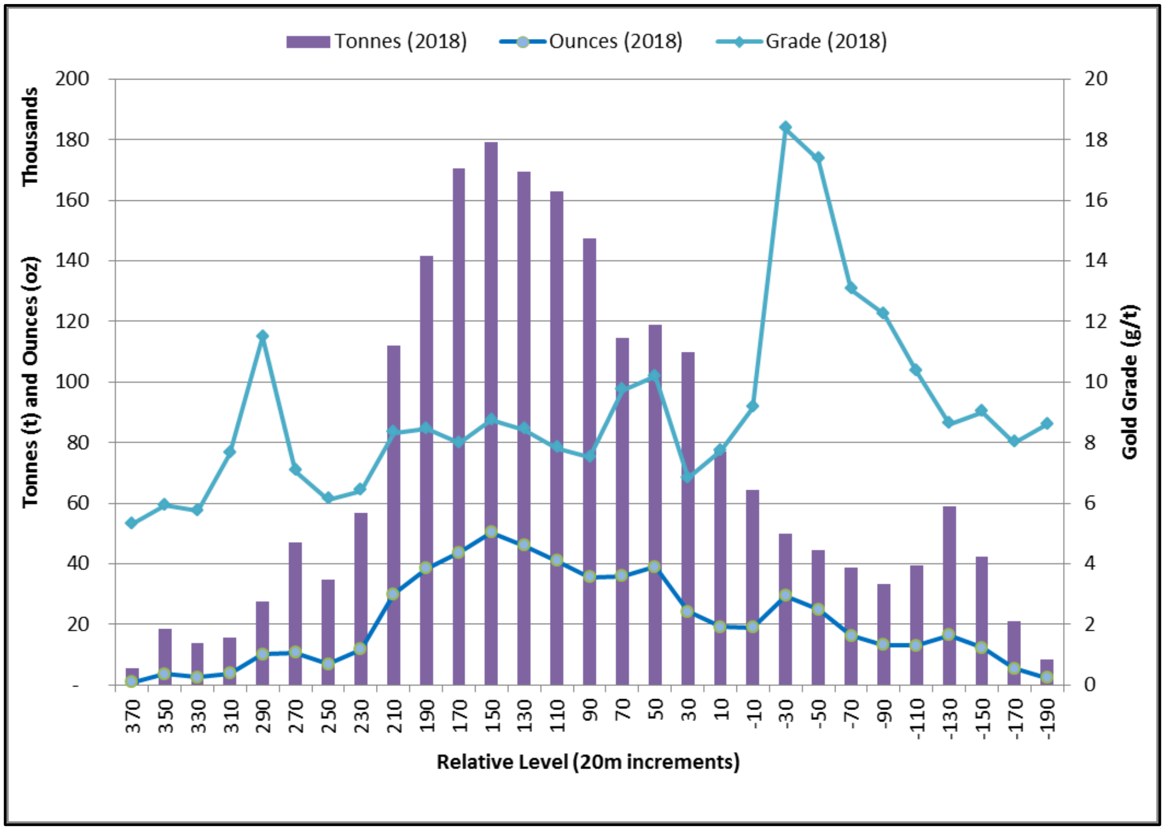


Figure 14-7: Tuvatu resource by 20 m RL intervals

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 (see Section 16.0):

- Gold pricing at USD$1,350
- Assumed underground mining costs of USD$76.50/t
- Assumed processing costs of USD$43.83/t
- Assumed G&A costs of USD$19.49/t

The deposit 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 2,079 bulk density measurements were reported from the drill hole cores at Tuvatu, with an average reported bulk density of 2.61 t/m³ (Table 14-17). The statistical average of the bulk density measurements is assigned to all lithologies for this Mineral Resource estimate.

Table 14-17: Bulk density statistics

Basic Statistics	All Density Readings (t/m³)	Vein Density (Subset) (t/m³)
Count	2,079	228
Minimum	1.86	1.86
Maximum	3.22	3.02
Average	2.61	2.57
Median	2.63	2.59
Mode	2.67	2.56
Standard Deviation	0.19	0.20

Bulk density data is stored in the drill hole database with a rock type code associated with each reading (Table 14-18). The majority of material is logged as either monzonite (MZ) or medium-grained monzonite (MMZ), each reporting average densities of 2.61 t/m³ and 2.62 t/m³, respectively. Mineralized samples are likely to be from vein breccia (VBX) or unmineralized veins (UV) with a density of 2.58 t/m³ and 2.50 t/m³, respectively.

Table 14-18: Bulk density by rock type

Rock Code	Count	Average (t/m³)
AN	223	2.63
BL	31	2.58
CV	1	2.80
MMZ	1,022	2.60
MZ	693	2.62
PG	1	2.30
TF	1	2.66
UV	53	2.52
VBX	46	2.61
Total	**2,079**	**2.61**

Notes: AN – Andesite; BL – Basalt; CV – Calcite Vein; MMZ – Medium-grained Monzonite, MZ – Monzonite, PG – Pegmatite; TF – Tuff; UV – Unmineralized Vein, VBX – Vein Breccia

Density values for mineralization were extracted from the drill hole database and 228 samples were found to be within defined mineralization. The average of mineralized samples used in definition of wireframe interpretations is 2.57 t/m^3 (Table 14-18). This is a similar result to the average of 46 VBX and 53 UV samples. The bulk density assigned to mineralization is 2.57 t/m^3.

Waste material is assigned the average bulk density of monzonite (2.61 t/m^3). 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

The resource is diluted to a minimum width of 0.3125 m, where the estimated vein thickness is less than 0.3125 m. 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 Mineral Resource is classified as a Mineral Resource according to the definitions from Canadian Institute for Mining, Metallurgy and Petroleum Definition Standards (2014):

> *A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*

A breakdown of the Project Mineral Resource estimate by category is provided in Table 14-19.

Table 14-19: 2018 Tuvatu resource estimates

	Resource Category					
	Indicated			Inferred		
Cut-off (g/t Au)	Tonnes (t)	Grade (g/t Au)	Ounces (Au oz)	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
3	1,007,000	8.48	274,600	1,325,000	9.0	384,000

The reporting of tonnages and grade figures reflects the relative uncertainty of the estimate, and rounding to the appropriately significant figures have been reported. Some discrepancy in the addition of rounded figures may occur.

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 20 m x 20 m and can demonstrate a high level of confidence in the geological continuity of the mineralization. Estimation statistics were used to guide the decision. OK variances of blocks within the Indicated category fall within the range of 0.15 to 0.4 and must not exceed 0.6. 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 informed by a minimum of six vein composites.

14.13.3 Inferred Mineral Resource

Defined as those portions of the deposit that 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 Inferred Resource is defined with a drill spacing of greater than 20 m x 20 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 of 30 m to the nearest sample and are informed by an average of four vein intercepts.

14.14 Comparison with Previous Estimates

The Tuvatu January 2015 Mineral Resource was reported as an Indicated Resource of 1,120,000 t at 8.17 g/t Au for 297,000 oz of gold and an Inferred Resource of 1,300,000 t at 10.7 g/t Au for 445,000 oz of gold (Table 14-20).

Table 14-20: Prior 2015 Tuvatu resource estimate (this estimate is superseded by the current resource estimate)

	Resource Category					
	Indicated			Inferred		
Cut-off (g/t Au)	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
3	1,120,000	8.17	297,000	1,300,000	10.7	445,000

Significant differences between the January 2015 and January 2018 Mineral Resources include reducing the extrapolation of veins in densely drilled areas, and splitting several veins where vein width and grade are well below cut-off. As shown in the comparative grade-tonnage charts in Figure 14-6, the main impact at the 3 g/t Au cut-off is an increase in tonnes and decrease in grade of Inferred Resources, with little change in tonnes and minor increase in grade of Indicated 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 Mineral 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 Project, and these structures are considered to have minor off-sets.
- An assumed mining width of 0.3125 m has been applied to the Mineral Resource; mining dilution has been incorporated at a 0 g/t grade. The final mining width achieved will have an impact on the narrow-veined mineralization 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), NLTB and the MRD.

- Lion One has completed an EIA and has obtained approval by the DE in Fiji. The DE has recommended mining to the MRD.
- Lion One has finalized negotiations with the NLTB, who represent the landowners regarding the surface license; the NLTB have also notified MRD.
- The proposed grant of the mining license 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 stakeholders supporting mining at Tuvatu.
- The EIA and surface license were prerequisites 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.

14.15.1 Mineral Resource Estimate Statement

The categorised Mineral Resources for the Project have been classified as Indicated and Inferred confidence categories on a spatial, areal, and zone basis and are listed in Table 14-21.

Table 14-21 Mineral Resources of the Tuvatu Gold Project (January 2018)

	Resource Category					
	Indicated			Inferred		
Cut-off (g/t Au)	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)	Tonnes (t)	Grade (g/t Au)	Ounces (oz Au)
3	1,007,000	8.48	274,600	1,325,000	9.0	384,000

14.15.2 Notes to Accompany Resource Statement

- The Project comprises four SPLs (SPL 1283, 1296, 1465 and 1512), which have total area of 20,786 ha, and for which Lion One has a 100% interest. The Tuvatu deposit itself is situated on SML 62, which has an area of 384.5 ha.
- MA was provided with an export of Lion One's current drill hole database in Microsoft® Access format.
- Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP), of MA visited the property in February of 2014 and in July, August, September, and October of 2017.
- Field exposures and numerous drill holes were examined during these visits, 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 mineralization lies within two zones of veining: the Upper Ridges zone striking north–south, which has extents of 700 m, and the Murau Corridor striking east–west, which has extents of 400 m. Additional to these areas is the Western Lodes, striking east–west for 200 m. At the intersection of the Upper Ridges and Murau Lodes occur the SKL veins, which are smaller, shallow, southerly dipping veins.
- Estimation was undertaken in Surpac™ using OK of g*m and thickness with the resulting gold grade back calculated as gram*thickness divided by thickness. NN and ID^2 of the gram meter estimates were used as validation estimates, A direct estimate of gold using OK was also run as a check estimate.
- Kriging as undertaken on 10 m x 10 m x 1 m blocks (a 2D grid). The 3D block model uses a 10 m x 10 m x 10 m block that considers vein orientations and drill pattern. (Approximately 1/3 in the drill spacing, in well-drilled areas). Sub-blocking of the 3D model to 0.3125 m x 0.3125 m x 0.3125 m, provides suitable resolution, allowing for very selective mining methods. Cubic blocks were required to accommodate all vein orientations sufficiently. Some dilution has been applied to the veins; a minimum vein width of 0.3125 m is applied (@ 0 g/t Au).
- Experimental variograms were generated in Surpac™. Nuggets were generally moderate to low, ranging from 0.1 to 0.69, and the range of the variogram ranged from 12 to 95 m. Geometric anisotropy was adopted in the plane of the vein and ellipsoid ratios applied to reflect directional variograms.

- Estimation parameters: Veins used a maximum of between 4 and 8 samples; the minimum sample number was set to one. Search distances reflect variogram ranges (30 to 70 m).
- Average dry bulk density is 2.61 t/m^3 for all rocks; 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 Mineral 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 g*m were estimated in 2D space; the resultant grids were refolded 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 groups (similar statistics) were assessed for outliers; grade caps were applied as appropriate and ranged from the 97.5th percentile to the 99th percentile. Thickness (m) and gram meter accumulations (g*m) were also capped.
- Global mean grades for estimated blocks and drill hole samples compared well.
- Gold back-calculated from the g*m estimates compared well with NN ID and OK estimates.
- No reconciliation data is available for the Project as no production records are preserved.

15.0 MINERAL RESERVE ESTIMATES

No Mineral Reserves have been calculated for this Technical Report.

16.0 MINING METHODS

The Project is a planned underground gold mine. The underground is currently accessed through an exploration portal and decline initially developed in 1997. Currently, no mining is taking place, but access has been maintained for dewatering and follow-up geological and sampling purposes. Lion One is currently completing site excavations in preparation for future construction of the processing plant and other surface infrastructure and mine development.

Recent studies assume a longhole stope mining method with only minor airleg stoping to be undertaken. Several criteria, including Lion One's production and financial targets, led Lion One to adopt mechanized longhole stoping as the primary stoping method rather than the entire operation being handheld mining as envisioned in the 2015 PEA. The mechanized longhole stoping method is also well suited to the geological and geotechnical aspects of the mineralized material body, allows for an increased production rate compared to airleg stoping, and is lower risk as a non-entry mining method. Stoping of flatter dipping mineralized material body areas (less than 1% of stope tonnes), where longhole stoping is not viable, will be excavated via handheld airleg mining techniques.

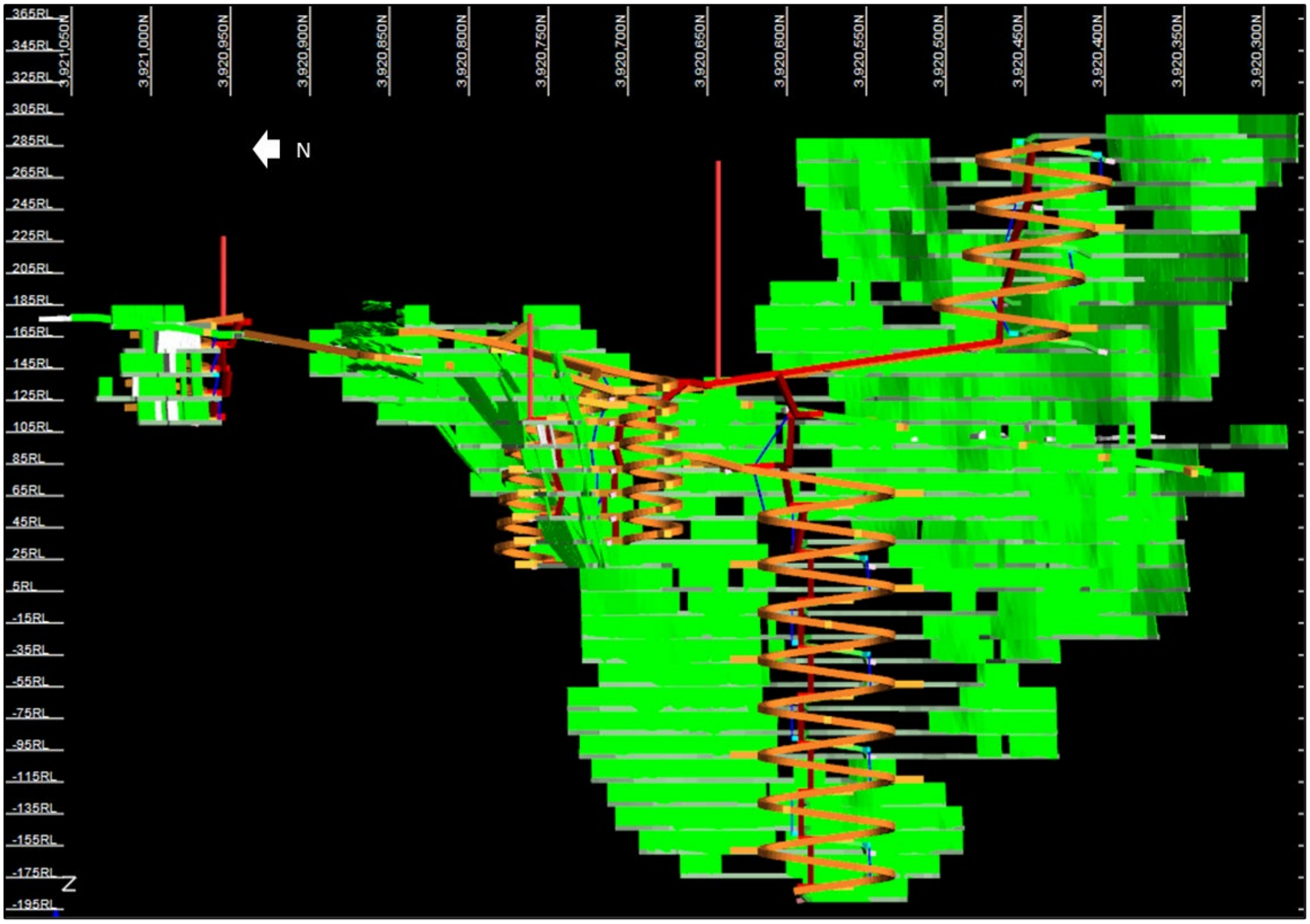
The Tuvatu deposit has a north–south strike of approximately 800 m. The proposed layout shows mine access from two portals: the existing exploration portal and a new portal, which will access the main haulage decline. The existing exploration decline was previously mined to approximately 3.0 m wide x 3.0 m high. The new design proposes stripping this decline to a 4.5 m wide x 4.5 m high drive size (the same profile as the main decline) to allow a second means of access at a sufficient size for the selected truck fleet. The exploration decline will be the primary haulage route for the initial part of the Project until the link drive is established to the new main decline. At this point, the main decline will become the main haulage and travel point into the mine, with the exploration decline serving as a second means of egress and additional fresh air intake into the mine.

The proposed ventilation layout uses the exploration and main declines as fresh air intakes, with the air then exhausting through the return air network and to surface via raise bores.

Level spacing of 15 m was selected to suit the proposed longhole stoping method, although shorter "blind" stopes can be taken as required. Stopes are proposed to remain open for the life of the Project.

The Tuvatu deposit mineralization is primarily sub-vertical ranging from 70 to 80°, with less than 1% of stope tonnes contained in flat-lying mineralization ranging from 0 to 30°. The veins are a series of parallel lodes with varying distance of separation of waste between lodes.

Perth-based mining consultant, Entech, produced a mine design and schedule for the Project using Deswik mining software. Figure 16-1 shows the proposed mine layout.



Source: Entech (2019)

Figure 16-1: Mine layout of the Tuvatu Gold Mine

The mine design and schedule work estimated a mining inventory of 1.4 Mt at 8.6 g/t for 381.5 koz of gold with a five-year LOM. The mining inventory is made up of 540 kt of Indicated material at a grade of 9.90 g/t for 171 koz and 850 kt of Inferred material at a grade of 7.7 g/t for 210,500 oz.

Lion One provided estimated mining physicals to an underground mining contractor and engaged them to provide cost estimation services.

16.1 Geology

Lion One provided Entech with the 2018 revised geological block model, which was used to produce and evaluate the mine design and schedule. The grade fields "gold" and "res_cat" were used to assess the Mineral Resource category (Measured, Indicated, and Inferred). The block model, produced by consultant MA, only contains Indicated and Inferred material. Table 16-1 shows the Mineral Resource information provided by MA.

Table 16-1: Tuvatu resource tonnes, grade, and ounces at specified cut-off grades

Cut-off (g/t Au)	Indicated			Inferred		
	Tonnes (t)	Grade (g/t Au)	Metal (oz Au)	Tonnes (t)	Grade (g/t Au)	Metal (oz Au)
3	1,007,000	8.48	274,600	1,325,000	9.00	384,000
5	687,000	10.60	234,300	788,000	12.50	317,500

16.2 Geotechnical

16.2.1 Previous Work

Several geotechnical studies previously undertaken for the Tuvatu deposit are summarized in the following subsections.

16.2.1.1 Tuvatu Gold Project Geotechnical Assessment (AMC 2000)

AMC completed the first geotechnical assessment for the Project. The report covered all normal aspects of a geotechnical assessment at approximately a PEA level, including:

- Rock mass classification
- Rock properties testing
- Stope spans, dilution, and extraction sequence
- Pillar design
- Backfilling
- Ground support design

At the time of the report, the long hole stoping method utilizing hydraulic mining fill was being considered. A level spacing of 15 m vertical with 3.0 m x 3.0 m mineralized material drives was being used to access the mineralized material. Stope spans of approximately 19 m down dip (i.e., a single level) and a strike span of more than 50 m were reported as likely achievable.

Rib pillars 3.0 m in strike length were recommended when applied and staggered from level to level.

Rock mass data had been collected from limited observations of DD core, the DDH database, and limited underground mapping. The sufficiency of rock mass data for this study is considered low, but commensurate with the level of study at the time.

16.2.1.2 Tuvatu Geotechnical Assistance – Lion One Metals Limited (AMC 2017)

AMC conducted a review and update of the previously completed geotechnical assessments (AMC 2000; 2014) for the Project. Additional information was collected from recent resource diamond drilling and observed during a site visit. The report provided geotechnical guidelines for when underground mining begins.

The geotechnical drill hole database was significantly built upon from previous studies. A total of 27 holes were logged geotechnically from drill core photos to supplement knowledge of ground conditions on the various lodes. Logging was conducted on and immediately adjacent to the main mineralized lodes.

Furthermore, the report discussed the site visit, an underground tour, and the limited viewing of drill core to confirm parameters such as joint planarity and alteration, which cannot be accurately estimated from photos.

At the time of the report, handheld shrinkage stoping and breast stoping mining methods were being considered as the main methods of extracting the mineralized material bodies.

Table 16-2 summarizes the stable stope spans recommended for each of the mining areas. These stable stope spans are broadly in line with the earlier AMC (2000) report.

Table 16-2: Summary stope spans

Lode	Average Dip (°)	Hanging Wall Hydraulic Radius (m)	Approximate Equivalent Stope Hanging Wall Dimensions	
			Dip Length (m)	Strike Length (m)
Upper Ridges	85	12	40	60
Upper Ridges West	70	9.6	40	40
Upper Ridge Southeast	89	9.6	40	40
Murau and Snake	67	9.6	40	40
West	-	7.5	20	35

Source: AMC (2000)

Entech reviewed the planned stoping dimensions in Table 16-2 and found these fit for purpose for the mining method considered in AMC (2017).

The report also provided recommendations for ground support in decline, access, and mineralized material drive development. The designs utilized empirical ground support charts to first estimate ground support requirements, followed by detailed kinematic analysis utilizing Unwedge software package. AMC (2017) did not specify the use of surface support in mineralized material drives; however, the report did specify the use of surface support (mesh) in all decline development and other long-term access development.

The ground support designs were found to be sound and fit for purpose; however, Entech recommends the use of mesh surface support in all mineralized material drives.

16.2.2 Geotechnical Scope and Findings

Lion One engaged Entech to complete a geotechnical gap analysis to review the sufficiency of geotechnical work undertaken to date. After reviewing the available reports, performing an underground site visit, and check-logging over 20 DDHs for verification, Entech concluded the following:

- The geotechnical drill hole data is sufficient in spread to account for spatial variability in ground conditions and accounts for the different mineralized material lodes present.

- Entech verified rock mass conditions by check logging and found that in some cases the condition of joints has been underestimated (only slightly). This has been factored by Entech.
- Ground support designs meet the demand requirements and serviceability of the mine; however, Entech recommends that all tunnels are meshed on the back to the shoulder as a minimum surface support, in line with Australian mining standards. This is important as the proposed mining method in this Technical Report is considering mechanized mining methods as opposed to the hand-held mining methods considered in previous studies.

In this Technical Report, Entech recommends the employment of top-down open stoping with rib pillars. This mining method is widely used in narrow vein mineralized material bodies, is well suited to good ground conditions, is mechanized and therefore speeds up the mining process, and is operationally one of the lowest cost narrow vein mining methods.

Entech reviewed the stable stoping spans (Table 16-2), with a strategy of leaving in situ rib pillars for local and global stability. Table 16-3 shows the recommended stoping spans.

Table 16-3: Stable stope spans recommended by Entech

Lode	Average Dip (°)	Approximate Equivalent Stope Hanging Wall Dimensions	
		Dip Length (m)	Strike Length (m)
Upper Ridges	85	40	60
Upper Ridges West	70	40	40
Upper Ridge SE	89	40	40
Murau and Snake	67	40	40
West	-	20	35

Pillar dimensions are recommended as follows:

- Rib pillars are to be 3 m along strike and the full stope height. Where the mineralized material body exceeds 3 m in thickness, pillar length should be increased to be equal to the stope width to maintain a 1:1 ratio of width to length. Rib pillars must be staggered from level to level.
- Sill pillars are to be left for the length of the full level, with a height of 5 m, where the down dip span of the individual mineralized material body exceeds 100 m.
- If stopes are 7.5 m apart or closer (i.e., a footwall and hangingwall lode), then the stopes should either be combined, or the highest value stope mined only. If parallel stopes are greater than 7.5 m apart, then the footwall lode should be mined first, and both stopes should be extracted on the same level before proceeding to the next level below.
- Stopes separated 30 m or greater can be considered independent of one another.

16.3 Mining Method Selection

Previous studies by Lion One have assumed a shrinkage mining method with in situ pillars, solely utilizing airleg stoping methods in an effort to reduce mining dilution as much as possible. Entech began the mining method selection process with a comprehensive review of the current proposed mining method and then identified other mining methods that warranted further consideration.

The Tuvatu deposit is a high-grade, narrow vein deposit in competent ground. This mineralization style excludes many bulk mining techniques. Handheld airleg mining, mechanized longhole stoping, and mechanized cut-and-fill methods were considered as three viable methods for comparison. The high backfill cost associated with underhand cut-and-fill meant only overhand cut-and-fill was considered. The generally good geotechnical conditions allow for relatively small in situ pillars, reducing the benefits of a higher extraction, but more expensive backfill method.

A list of criteria was compiled to rank the mining methods, including level spacing (lateral development cost), mineralized material body extraction ratio, dilution control, productivity, production cost, geotechnical risk, and safety risk. There are other considerations relevant to selecting a mining method; however, these were deemed the most comparable in creating an appropriate comparison.

Each of the mining methods was ranked from one to three for each criterion. The criteria were weighted based on their importance. The lower the total ranking score, the more applicable the mining method.

A detailed explanation of each criterion follows:

- Level spacing – A smaller level spacing between main level accesses and mineralized material drives (not including airleg size sub-level development) means an increased lateral development requirement, increasing the overall cost.
- Extraction ratio – This refers to the total mineralized material body extraction where the aim is to have total extraction as high as possible.
- Dilution control – Different mining methods may incur varying levels of dilution with the lowest dilution being preferred to maximize head grade.
- Productivity – This refers to the stoping productivity of the mining method and how the mining method impacts the maximum production rates.
- Production cost – This is a high-level comparison of production cost based on typical mining operations.
- Geotechnical risk – This refers to the overall geotechnical risk of the operation for a given mining method, in particular, global stability (e.g., a backfill method would generally score better as less voids remain post mining)
- Safety risk – This refers to the safety risk associated with the mining method (i.e., non-entry methods are generally considered safer).

The ranking scores are based on the assumptions shown in Table 16-4 regarding each mining method when comparing them to the extraction of a 1 m mineralized material body width (typical width at Tuvatu).

Table 16-4: Mining method modifying factor comparison

Modifying Factor	Unit	Mining Method: Airleg Mining	Mining Method: Longhole Stoping	Mining Method: Overhand Cut-and-fill
Mineralized Material Body Width	m	1	1	1
Level Spacing	m	20	15	15
Minimum Mining Width	m	1.2	1	1.5
Extraction Ratio	%	90	90	95
Dilution	m	0.20	0.30	0.30
Total Mining Width	m	1.4	1.30	1.80
Relative Dilution	%	40	30	80
Recovery	%	97	97	97

Table 16-5 shows the overall ranking score of each mining method.

Table 16-5: Mining method ranking score comparison

Selection Criteria	Mining Method: Airleg Mining	Mining Method: Longhole Stoping	Mining Method: Overhand Cut-and-fill	Weighting
Level Spacing	1	2	2	15.00
Extraction Ratio	2	2	1	15.00
Dilution	2	1	3	20.00
Productivity	3	1	2	20.00
Production Cost	1	1	3	15.00
Geotechnical Risk	2	2	1	5.00
Safety Risk	3	1	2	10.00
Total Ranking Score	**200**	**135**	**215**	**100.00**

The results of the quantitative analysis show longhole stoping to be the most suitable mining method. The Tuvatu mineralized material body characteristics (relatively steeply dipping with competent ground conditions) are well suited to longhole stoping. Mechanized mining, and particularly longhole stoping, would be the most commonly applied mining method for similar style mineralized material bodies mined in Australia. Lion One requires a high assurance of meeting production targets for financing repayment, and this was a key point in selecting a higher productivity mechanized mining approach. A small amount of stoping (less than 1% of stope tonnes) is planned to utilize handheld airleg mining techniques, as the mineralized material body is too flat dipping in these areas to use a longhole stoping method.

Given that Tuvatu is a narrow vein, high-grade deposit, emphasis was placed on selecting a mining method that will deliver a high-grade product to the mill through minimal dilution and with maximum recovery. To deliver this high-grade product, drill hole accuracy and successful drill and blast techniques have been identified as fundamental to the success of the recommended mining method. For this reason, a 15 m level spacing has been adopted with 64 mm diameter production drill holes to ensure drill hole lengths are appropriate, and the hole diameter is large enough to maintain drill hole accuracy. These parameters tie in with equipment fleet selection, lateral development profile sizes, and considers what has been successfully implemented in other mining operations of similar deposition style.

The stopes are planned to utilize airleg rises as the initial void in slot opening, with 64 mm blastholes then firing into the slot. Narrower stopes are planned to employ a modified dice five drill pattern where holes are offset, with wider stopes adopting a more standard ring design with holes aligning in single rings.

Lateral development is proposed to be mined via mechanized methods using twin and single boom jumbos.

16.4 Stope Design

The stope design process involved an initial generation of 5 m stope sections using Datamine's MSO mining software. These 5 m sections were then combined into stoping blocks and reviewed to ensure they were practically mineable stoping blocks and followed the geotechnical guidelines.

16.4.1 Stope Optimization

Mineable stopes were designed using the following process:

1. Apply the stope cut-off grade.
2. Create optimized stope shapes using Datamine's MSO mining software.
3. Using the 5 m sections produced by the software, design stopes to adhere to the geotechnical parameters.

MSO software utilizes several parameters to produce an optimised stope shape. These parameters include, but are not limited to cut-off grade, minimum mining width, and amount of unplanned dilution. Dilution and recovery factors were applied post optimization.

16.4.2 Underground Cut-off Grade Estimation

The cut-off grade for initial MSO software stope shape generation was derived from the preliminary economic assessment costs with updated revenue and recovery factors provided by Lion One. Table 16-6 shows the resulting fully costed, incremental, and development cut-off grades.

Table 16-6: Fully costed, incremental, and development cut-off grades

Cut-off Calculation	Units	Fully Costed Cut-off	Incremental Cut-off	Development Cut-off
Mining Cost	USD$/t mined mill feed	76.50	50.95*	-
Processing Cost	USD$/t mined mill feed	43.83	43.83	43.83
Exploration Cost	USD$/t mined mill feed	1.53	-	-
G&A Cost	USD$/t mined mill feed	19.49	-	-
Smelting and Refining Cost	USD$/t mined mill feed	0.75	0.75	0.75
Total Cost	USD$/t mined mill feed	142.10	95.53	44.6
Gold Price	USD$/oz	1350	1350	1350
Recovery	%	86%	86%	86%
Royalties	%	4.0%	4.0%	4.0%
Revenue	USD$/g	35.96	35.96	36.0
Cut-off Grade	**g/t**	**3.95**	**2.66**	**1.24**

Note: *Excludes mineralized material development cost.

MSO software shapes were generated based on the incremental cut-off grade, and then lateral extents were trimmed to meet the fully costed cut-off grade.

16.4.3 MSO Software Input Parameters and Results

MSO software runs were completed to produce stope shapes at 15 m vertical extents. Table 16-7 shows the MSO software stoping input parameters.

Table 16-7: MSO software stoping parameters used for 15 m high stopes

Stoping Parameter	Unit	Value
Stoping Cut-off Grade	g/t Au	2.7
Minimum Mining Width	m	1
Vertical Level Interval	m	15
Section Length	m	5
Hanging Wall Dilution	m	0
Footwall Dilution	m	0
Minimum Parallel Waste Pillar Width	m	10
Minimum Footwall Dip Angle	degrees	40

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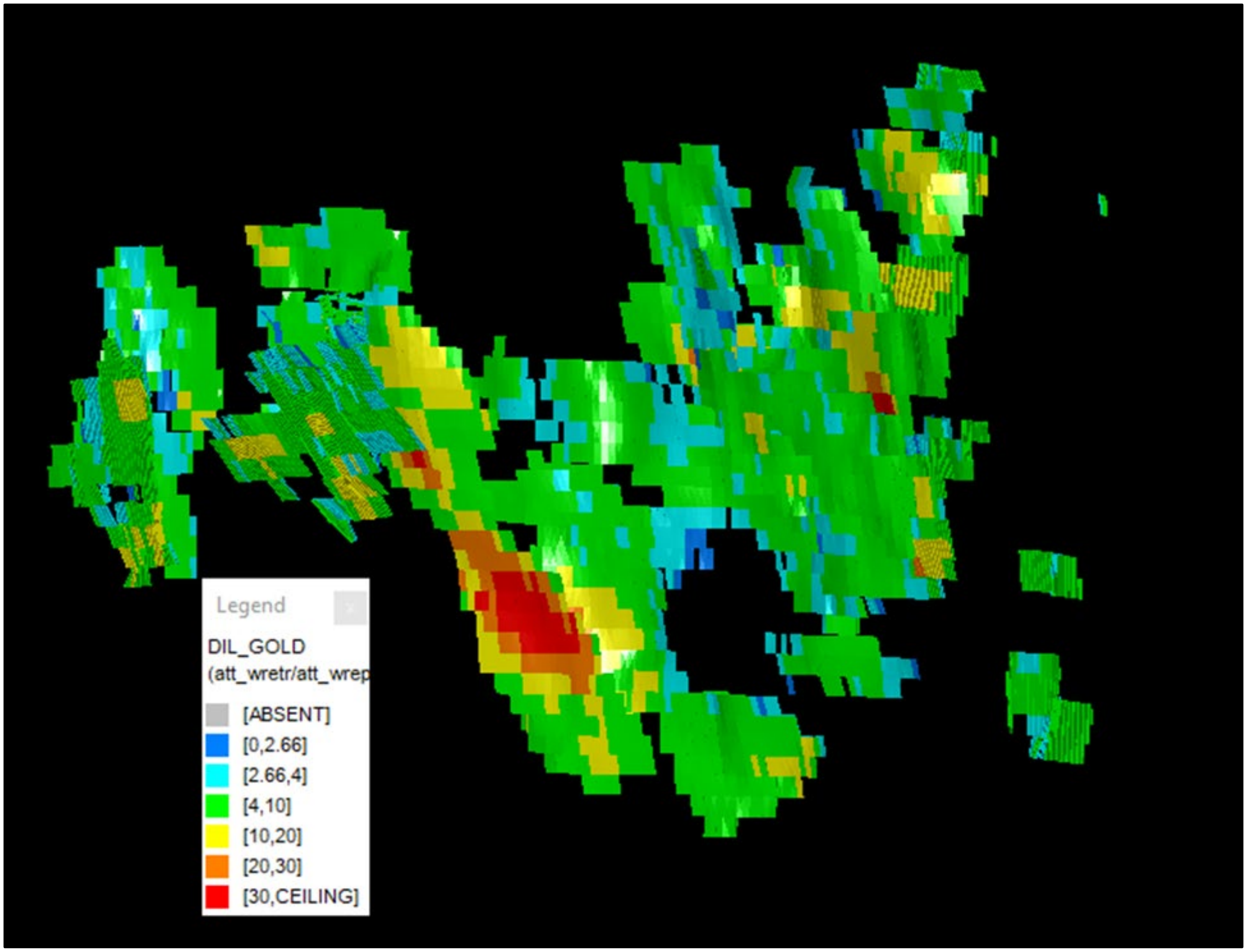
Figure 16-2 shows the resulting MSO software shapes.

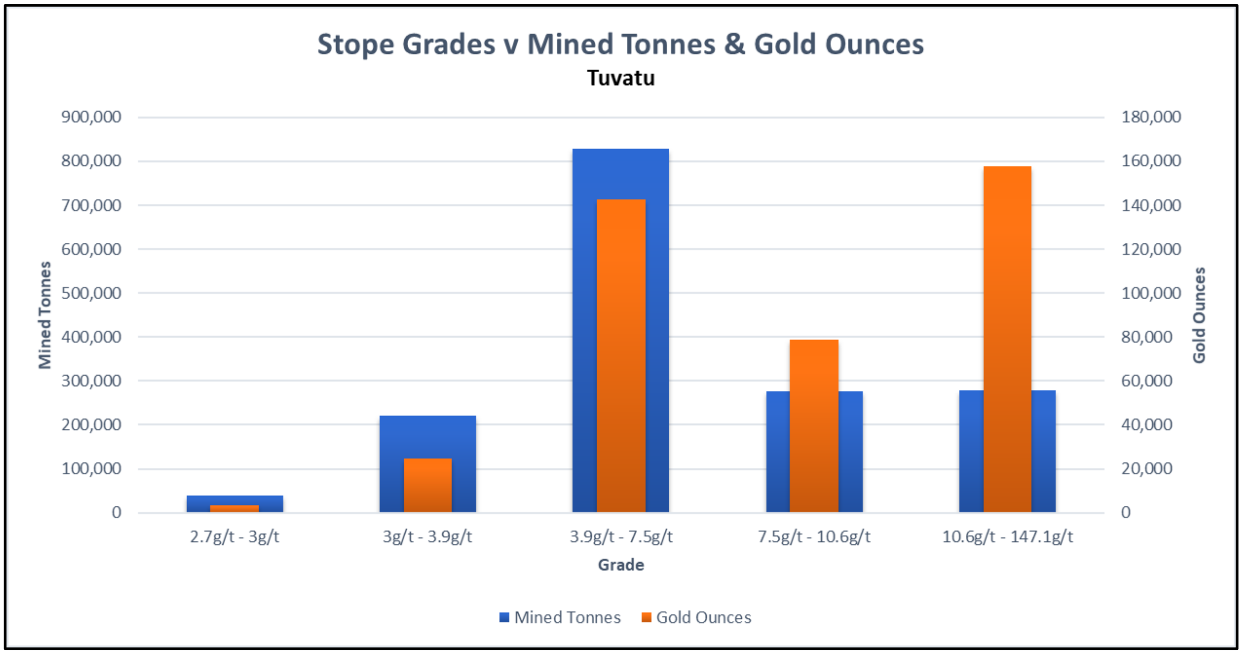


Source: Entech (2019)

Figure 16-2: MSO software shape results

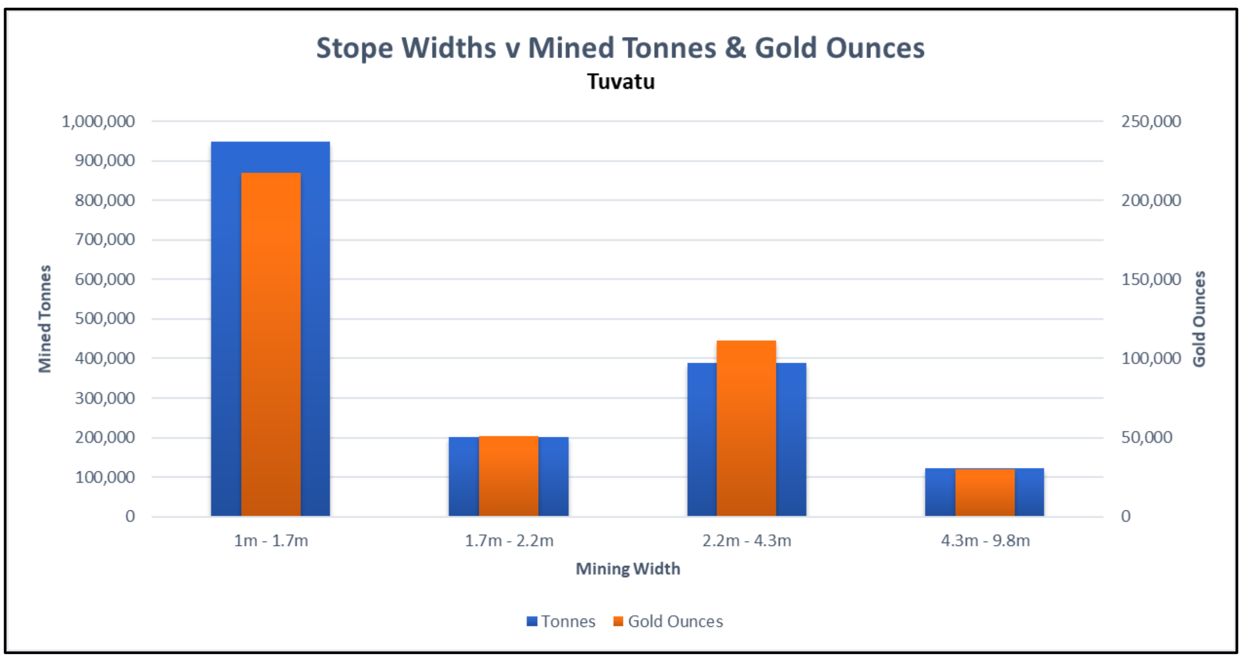
An initial dilution factor of 0.3 m of dilution per stope and an 80% recovery factor were then applied to the resulting MSO software inventory post evaluation. The resulting inventory was 1.64 Mt at 7.7 g/t for 408 koz. These shapes were then interrogated, removing uneconomic and impractical mining shapes, to arrive at the final stope shapes.

Figure 16-3 shows the grade distribution of the resulting MSO software shapes. Figure 16-4 shows the tonne and ounce distribution by stope width.



Source: Entech (2019)

Figure 16-3: Tonne and ounce distribution of MSO software shapes by grade bin

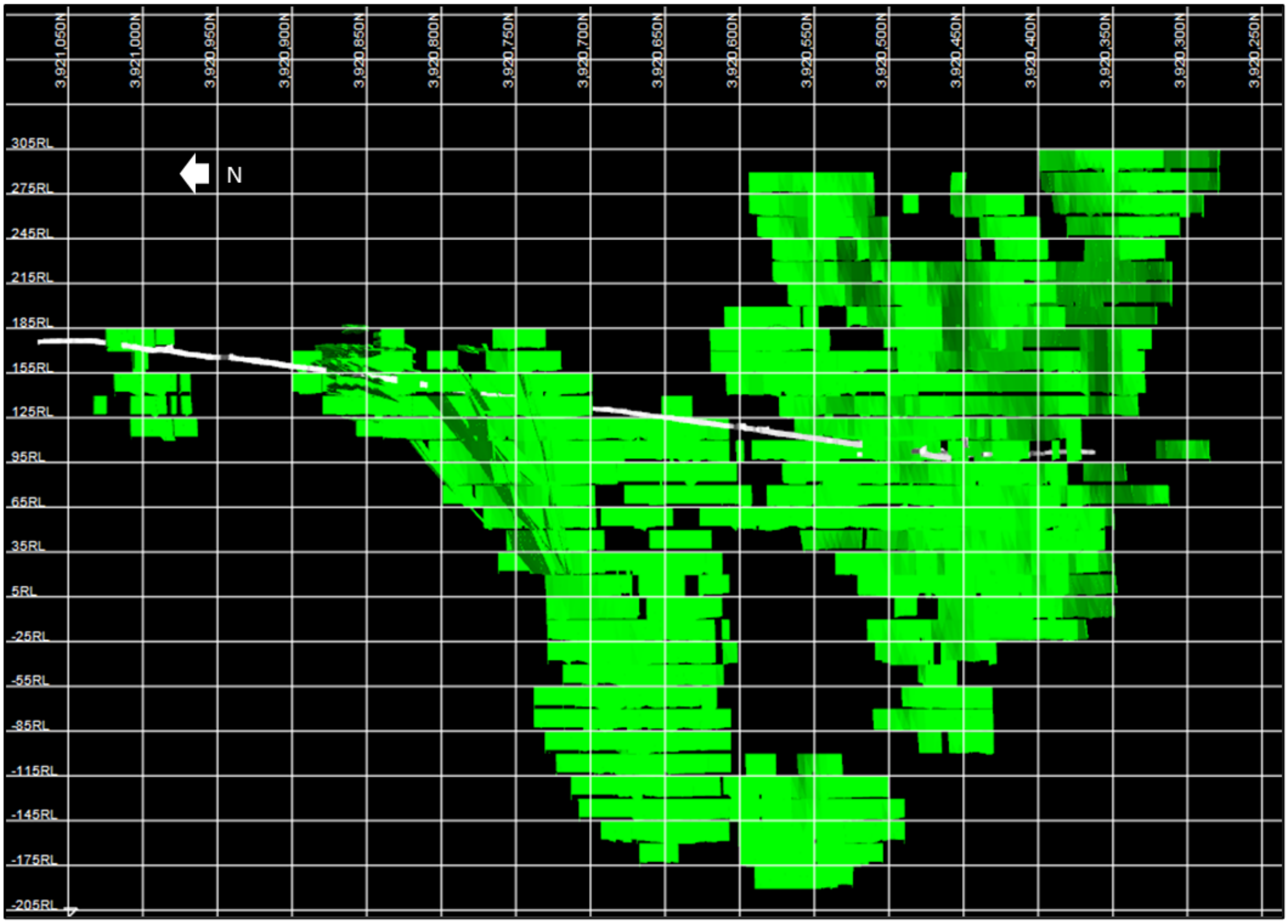


Source: Entech (2019)

Figure 16-4: Tonne and ounce distribution of MSO software shapes by width bin

16.4.4 Final Stope Shapes

Figure 16-5 shows the final stope shapes, which are the initial MSO software shapes trimmed to allow a sufficient crown pillar to the surface, pillars between parallel lodes, removal of uneconomic shapes, and removal of mining shapes that are impractical to mine.



Source: Entech (2019)

Figure 16-5: Final stope shapes after depletions

16.4.5 Mine Material Dilution and Recovery

The mined material dilution and recovery factors shown in Table 16-8 were applied to the mining schedule produced using Deswik software.

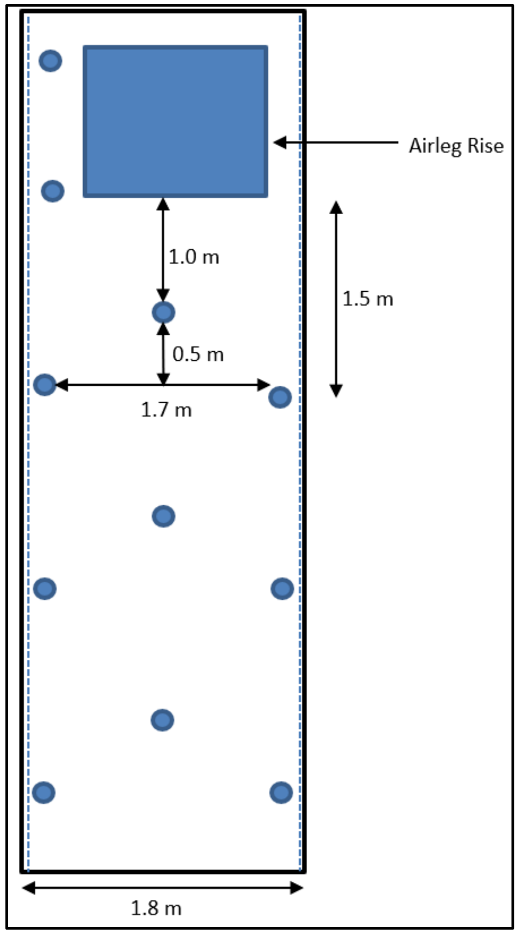
Table 16-8: Dilution and recovery factors

Parameter	Dilution (m)	Recovery (%)
Lateral Mineralized Material Development	0	100
Stopes	0.30	88

The dilution and recovery factors in Table 16-8 are based on the geotechnical parameters specified, with additional bogging recovery factors applied. The recovery factor allows for a 10% pillar factor and 2% in losses from bogging recovery. These factors have been applied post-stope interrogation with the geological block model.

16.4.6 Stope Drill and Blast

Drill and blast designs are based on the use of 64 mm blastholes drilled by a production drill rig utilizing speed rods. The stope slot is planned to open around a 1.2 m x 1.2 m airleg rise, which will serve as an initial void for blastholes to fire into. Figure 16-6 shows an example of a typical modified dice five drill pattern commonly used for narrow stoping. The example shown is for a 1.7 m wide stope, which is the average stope width for Tuvatu.



Source: Entech (2019)

Figure 16-6: Plan view of example drill and blast pattern (average stope width of 1.7 m). The blue dotted line shows the stope outline.

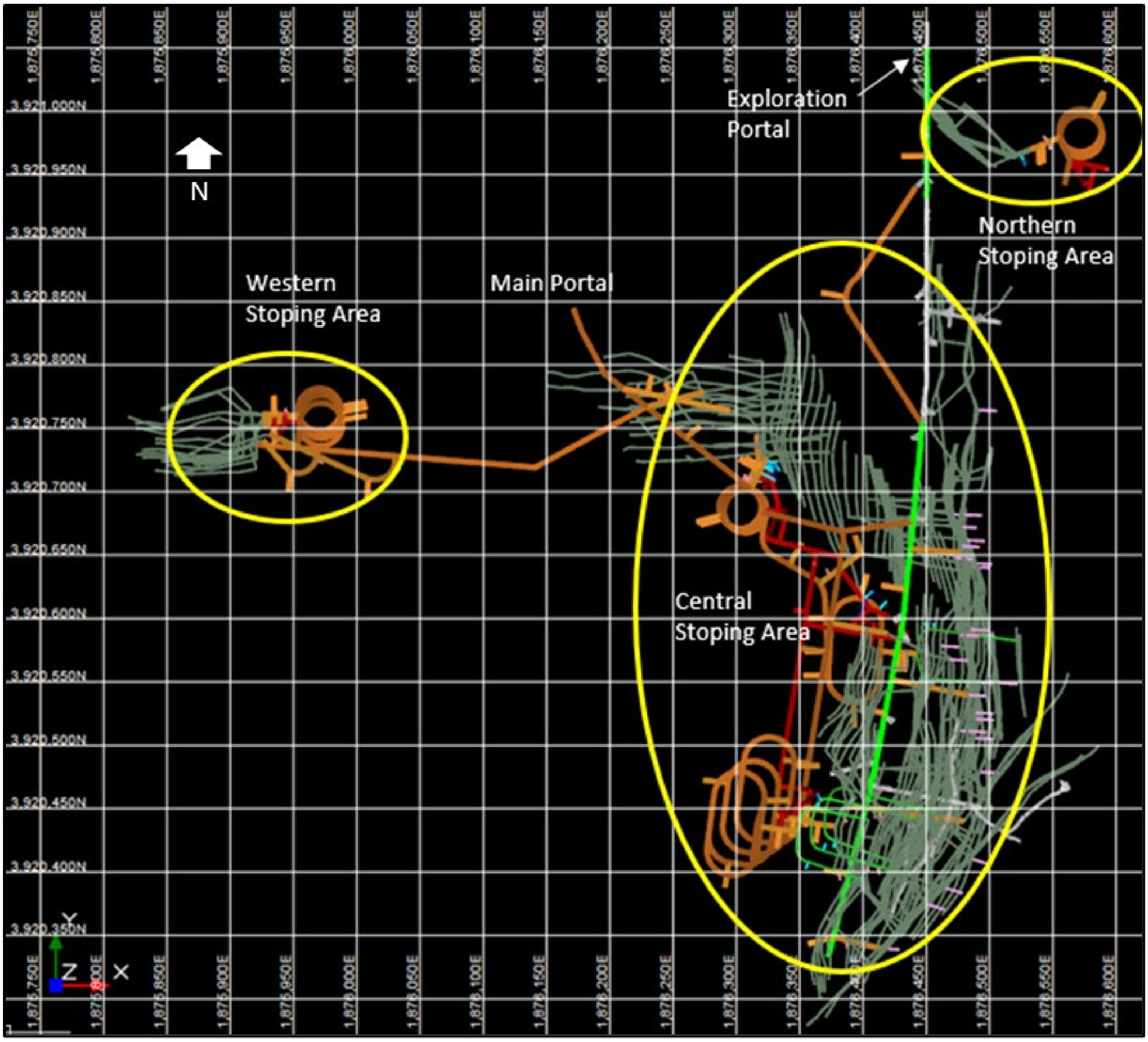
A drill yield of 2.85 tonnes per drill meter has been used, which is based on the width distribution shown in Figure 16-6, with the results shown in Table 16-9.

Table 16-9: Drill yields by width bin

Width (m)	Average Stope Tonnes (t)	Estimated Drill Meters (dm)	Drill Yield (t/dm)	Tonnes in Width Bin (t)
1.3-1.7	3,672	1,440	2.55	950,000
1.7-2.2	4,320	1,680	2.57	200,000
2.2-4.3	7,020	2,160	3.25	380,000
4.3-9.8	15,336	3,360	4.56	110,000
Weighted Average Drill Yield	-	-	2.85	-

16.5 Development Design

The proposed Tuvatu development layout shows the mine will be accessed via two declines as shown in Figure 16-7. A long section of the mine is shown in Figure 16-8.



Source: Entech (2019)

Figure 16-7: Plan view of the Tuvatu underground development layout

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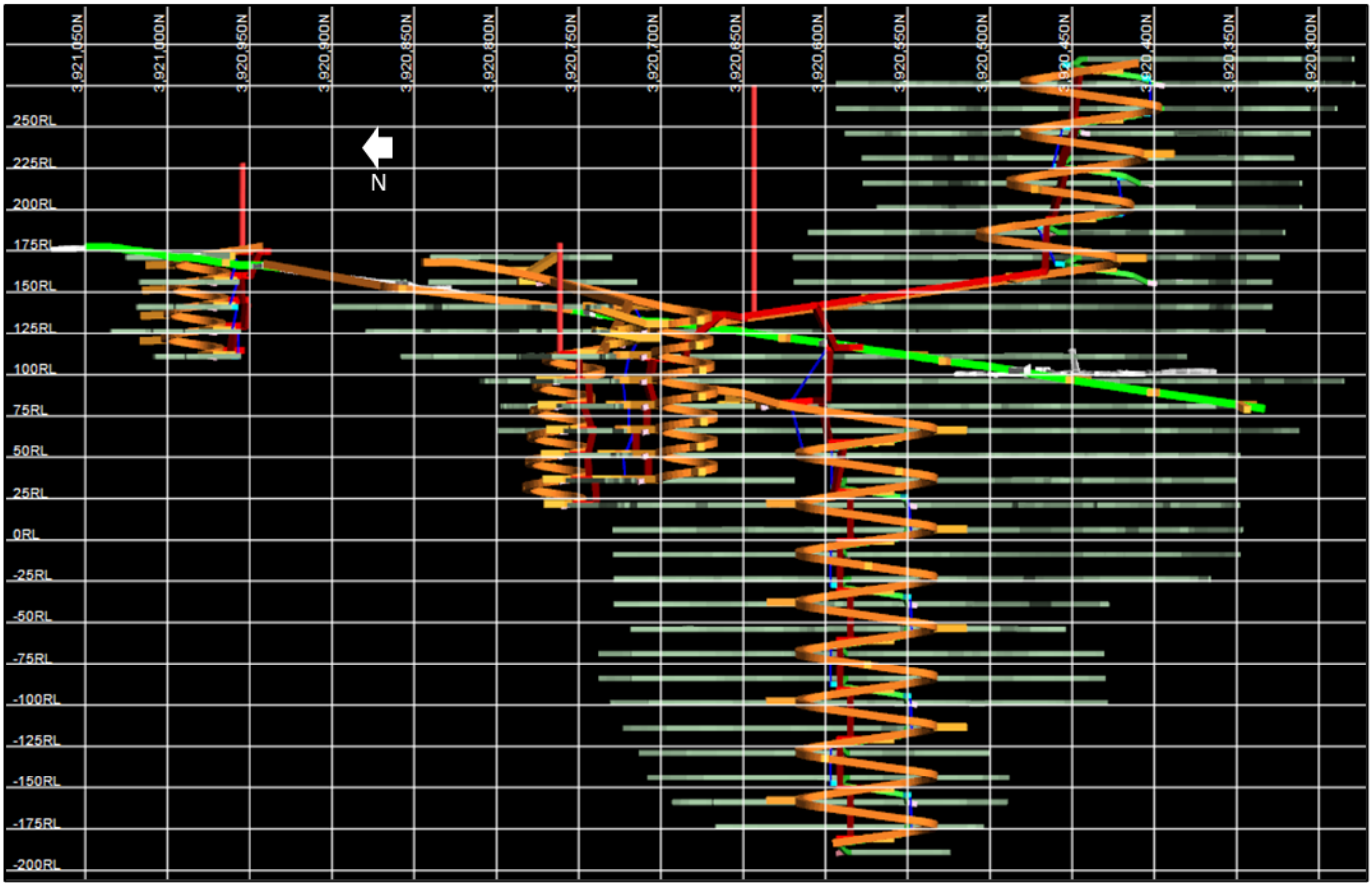


Source: Entech (2019)

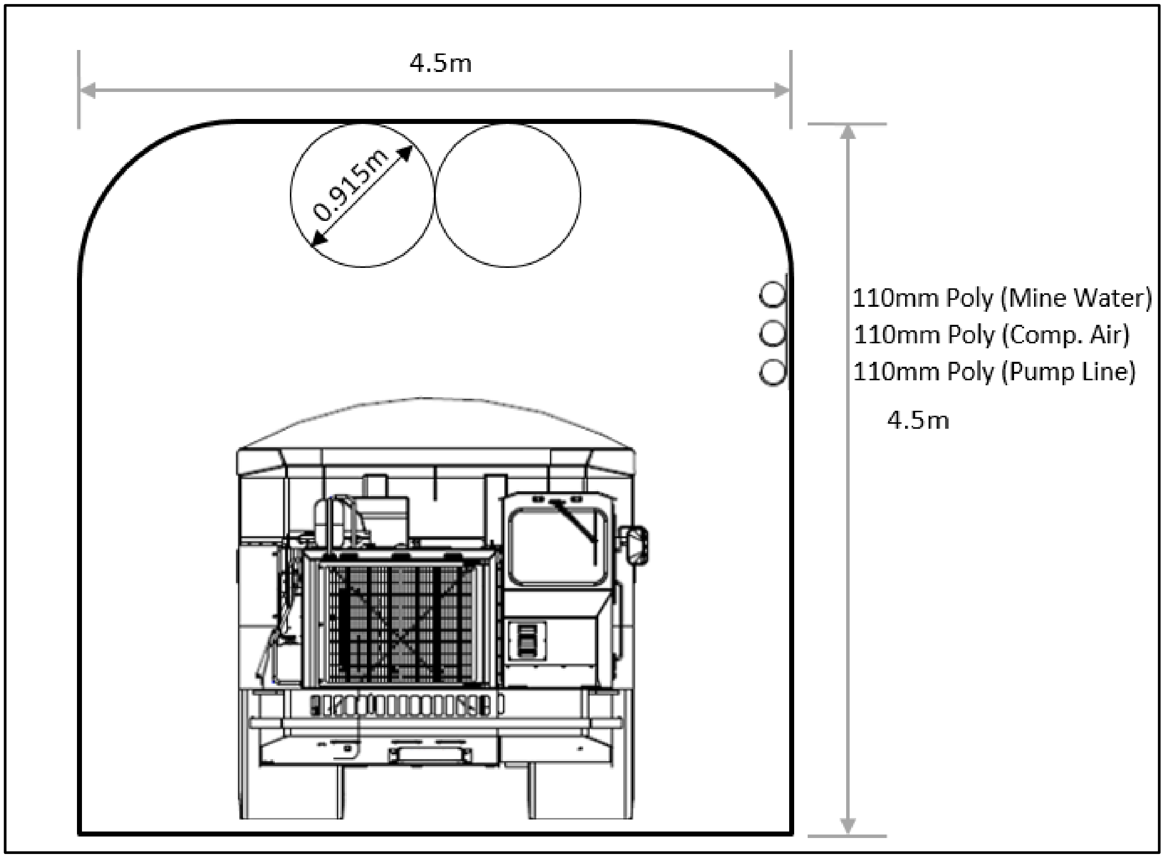
Figure 16-8: Long section of the Tuvatu underground development layout (looking east)

16.5.1 Decline

Two decline portals will be used to access the underground mine: the existing exploration decline and the proposed main decline to the west. The main decline will become the primary haulage route, and the exploration decline will be used as a haulage route at the start of the Project. Once a link is established between the declines, the exploration decline will be used as a second means of egress and a secondary haulage route. Both declines will serve as intake positions for ventilation purposes.

The exploration decline will be stripped from the current size of 3.0 m wide x 3.0 m high to match the main decline size of 4.5 m wide x 4.5 m high. This will accommodate the 16 t trucks and negate the necessity to have two separate trucking fleets for different size excavations. Stripping the exploration decline to a larger size also allows for productivity gains throughout the LOM; more material can be hauled more quickly while it is the solitary mine access, it offers a secondary haulage route, and it minimizes congestion for the remainder of the LOM.

The decline profile has a shoulder radius of 1.0 m as shown in Figure 16-9. This profile allows access for a standard 16 t truck and allows sufficient room for ventilation and other mine services. The majority of the decline has been designed at a 1 in 7 gradient.



Source: Entech (2019)

Figure 16-9: Decline profile with 16 t haul truck

The new decline will be situated in the footwall of the main mineralization zone. The minimum standoff distance from decline to stoping fronts is 40 m.

Where practical, a figure-eight and race-track style decline design was preferred; however, this was not always suitable in order for the decline to follow the mineralized material body and keep the access lengths appropriate. A 20 m decline radius was selected, with a minimum 18 m radius used where required. A maximum of 140 m between stockpiles has been maintained to allow for high development rates in the decline and provide adequate room for other infrastructure that will need to be positioned underground such as substations, mono pumps, and consumables storage.

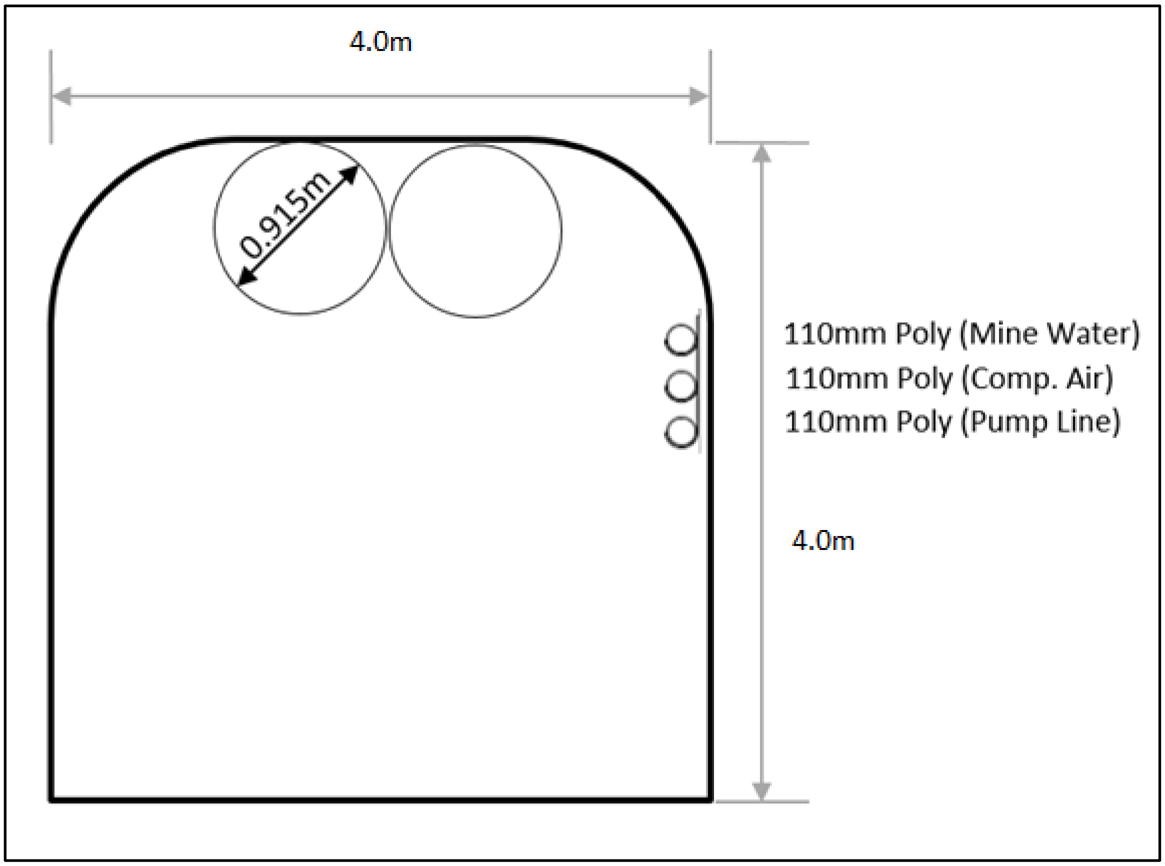
16.5.2 Level Access

The minimum access to each level will be 40 m long, allowing room for a level sump and stockpile. The first 25 m length of each level access will be at the decline profile to allow truck access to the level stockpile for loading purposes. Any level access after this point will be 2.0 m wide x 3.0 m high, the same profile used in the mineralized material drives.

A square profile stockpile has been designed on each level at 4.5 m wide x 4.5 m high and 20 m in length. The level stockpile will be used to stockpile development and stope mineralized material produced from the level until it can be loaded into a truck for haulage to the surface. The square profile allows for maximum material capacity by allowing the loader bucket to reach up into the corners to dump its load into the stockpile.

16.5.3 Return Air Drives

The return air drives are designed to be mined by the single boom jumbo and have a drive profile of 4.0 m wide x 4.0 m high. Figure 16-10 shows an example profile.

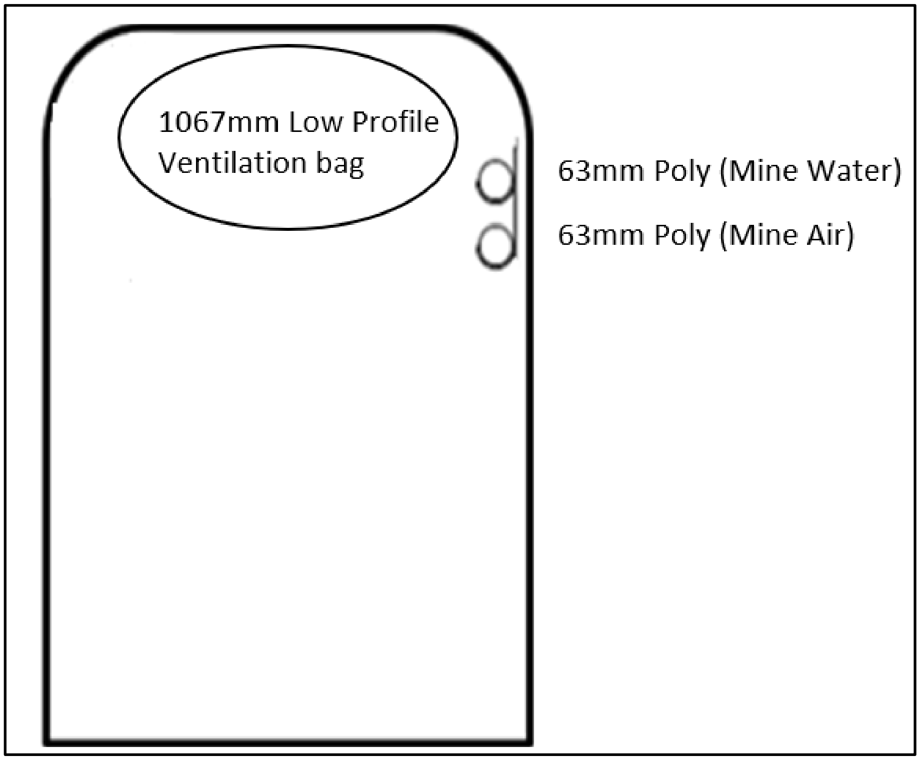


Source: Entech (2019)

Figure 16-10: Return air drive profile

16.5.4 Mineralized Material Drives

The mineralized material drives on each level have been designed to follow the mineralized strike of the veins. The drive profile is 2.0 m wide x 3.0 m high, which is the appropriate size for the LH203 loader. Figure 16-11 shows the mineralized material drive profile.



Source: Entech (2019)

Figure 16-11: Mineralized material drive profile

16.5.5 Vertical Development

Vertical development is planned to be excavated via airleg, longhole, and raisebore methods. Airleg rises of 1.2 m x 1.2 m will be utilised for production slotting, as well as escapeway rises. The return air rises between levels are planned to be mined via 3.0 m x 3.0 m longhole rises. The three surface legs that return exhaust air from the Northern, Central, and Western mining areas are planned to be excavated via raisebore, with a 3.0 m raisebore diameter used in the Northern and Western mining areas, and a 3.5 m diameter raisebore for the Central exhaust.

16.6 Productivity Assumptions

The productivity driver assumptions are based on Entech's benchmark database of productivity rates using similar equipment types and mining methods.

16.6.1 Jumbo Development

The development productivities are based on an Altlas Copco Boomer 2D twin boom jumbo drill rig and Resemin Muki single boom jumbo. The Atlas Boomer electric over hydraulic boom drills are capable of drilling 45 mm holes and reamers up to 125 mm. The single boom electric over hydraulic boom drills are capable of drilling 33 to 64 mm holes.

Jumbo productivities have been outlined as maximum rates achievable for different headings. A maximum total of 250 m/mo has been allowed for the twin boom jumbo, and 150 m/mo for the single boom jumbo. Table 16-10 and Table 16-11 detail the development rates.

Table 16-10: Maximum twin boom jumbo productivities for different development types (m/mo)

Units	Exploration Decline Stripping	Decline Development	Other Capital Development
Twin Boom	240	180	180

Table 16-11: Maximum single boom jumbo productivities for different development types (m/mo)

Units	Operating Access	Mineralized Material Drive
Single Boom	90	120

Consideration has been taken during scheduling to ensure that there are enough available headings to support the overall development advance.

16.6.2 Vertical Development

Vertical development will be done by a combination of mechanical and handheld excavation. Slot rises for stoping and escapeway rising are planned to be excavated via handheld airleg mining techniques. All airleg rising have an advance rate of 4 m/d. Ventilation return airway rises are planned to be excavated utilising mechanical raisebore machines. An advance rate of 3 m/d has been used for raisebore rising, which includes setup, drilling, and de-rigging time.

16.6.3 Production Drilling

A drilling rate of 150 m/d has been applied to the schedules based on drilling rates from Entech's database of comparable mines. These drilling rates are assumed to include all activities and delays related to production drilling, including drill rig up, drill rig down, slot drilling, production drilling, shift change and meetings, meal breaks, breakdowns, maintenance, services installation, and geology/survey control delays. An Atlas Copco H104 single boom jumbo modified for production drilling, as well as a fleet of Tamrock Long Hole Drill Rig to be supplied by the contractor complements the production drilling equipment.

16.6.4 Haulage

The monthly underground tonne-kilometer (tkm) haulage requirements have been provided to the mining contractor, who in turn has determined the appropriate truck fleet requirements. Monthly tonne-kilometer rates will reach a peak of 70,000 tkm, which would be within the expected performance of the four Tamrock trucks the mining contractor has proposed based on a conservative 20,000 tkm per month.

16.7 Mine Scheduling

Entech completed the mine scheduling using Deswik planning and scheduling software. Multiple iterations of the schedule were completed, considering alterations to the mining sequence, flexing of productivity rates, as well as detailed review of linking and interaction between activities in the schedule.

A focus on producing a practical, realistic, and executable mining plan is of key importance at Tuvatu. Several iterations of the schedule were provided to a mining contractor to provide cost estimates, and consideration was given to their feedback, which has been integrated into the mining schedule.

16.8 Underground Infrastructure and Services

16.8.1 Ventilation

As part of their study work, Entech conducted ventilation modelling; this Technical Report section provides details of the ventilation requirements, strategy, and fan selection.

16.8.1.1 Ventsim™ Model

Entech utilized Ventsim™ software to create a ventilation model based on the planned mine design.

Entech estimated the following geothermal properties, which were used for modelling, based on general information for the district (these values are indicative only):

- Geothermal gradient: 2.0°C/100 m
- Surface rock temperature: 25°C
- Rock density: 2,879 kg/m³
- Rock-specific heat: 790 J/kg °C

Geothermal properties specific to the mining area should be determined by local testing to provide further accuracy to the modelling of the expected heat load from the strata.

A reasonable rock wetness fraction was used in the model to account for damp-to-wet conditions underground.

The long-term annual weather statistics of the Nadi area were assessed and the 95th percentile wet- and corresponding dry-bulb temperatures were selected with the local barometric pressure. The values used in the model include:

- Wet-bulb temperature: 27.4°C
- Dry-bulb temperature: 30.4°C
- Barometric pressure: 97.5 kPa
- Reject wet-bulb temperature: 30°C

The model considered the mining fleet as well as the mine at depth.

16.8.1.2 Airflow Calculations

The parameters listed in Table 16-12 were used to determine the ventilation requirements for development and production activities. The primary circuit has been modelled on maximum fleet requirements.

Table 16-12: Design parameters

Primary Diesel Fleet	Unit	kW Rating	Airflow Requirement @ 0.05 m³/s/kW
Tamrock EJC 416 – 16 t Truck x 4	-	960	48
LH203 Loader x 5	-	360	18
R1700 Loader x 1	-	263	13
Ancillary Equipment (ITs, Charge-up, LVs)	-	200	10
Mean Density of Surface Air	kg/m³	1.11	
Mean Density of Underground Air	kg/m³	1.14	
Minimum Air Speed in Airways	m/s	0.5	
Maximum Air Speed in Populated Airways	m/s	6.0	
Air Velocities in Upcast Shafts	m/s	6.0–15.0	
System Leakage	%	25	

From a ventilation perspective, the mine will be split into three main areas: Central, Western, and Northern stoping areas. The mine will be accessed from the west through the main decline portal, and from the north through the exploration decline.

The main decline heads east and has a take-off after approximately 100 m to access the Western Lodes. The Western Lodes will have a standalone return air network. The main decline will continue past the take-off and link to the exploration decline.

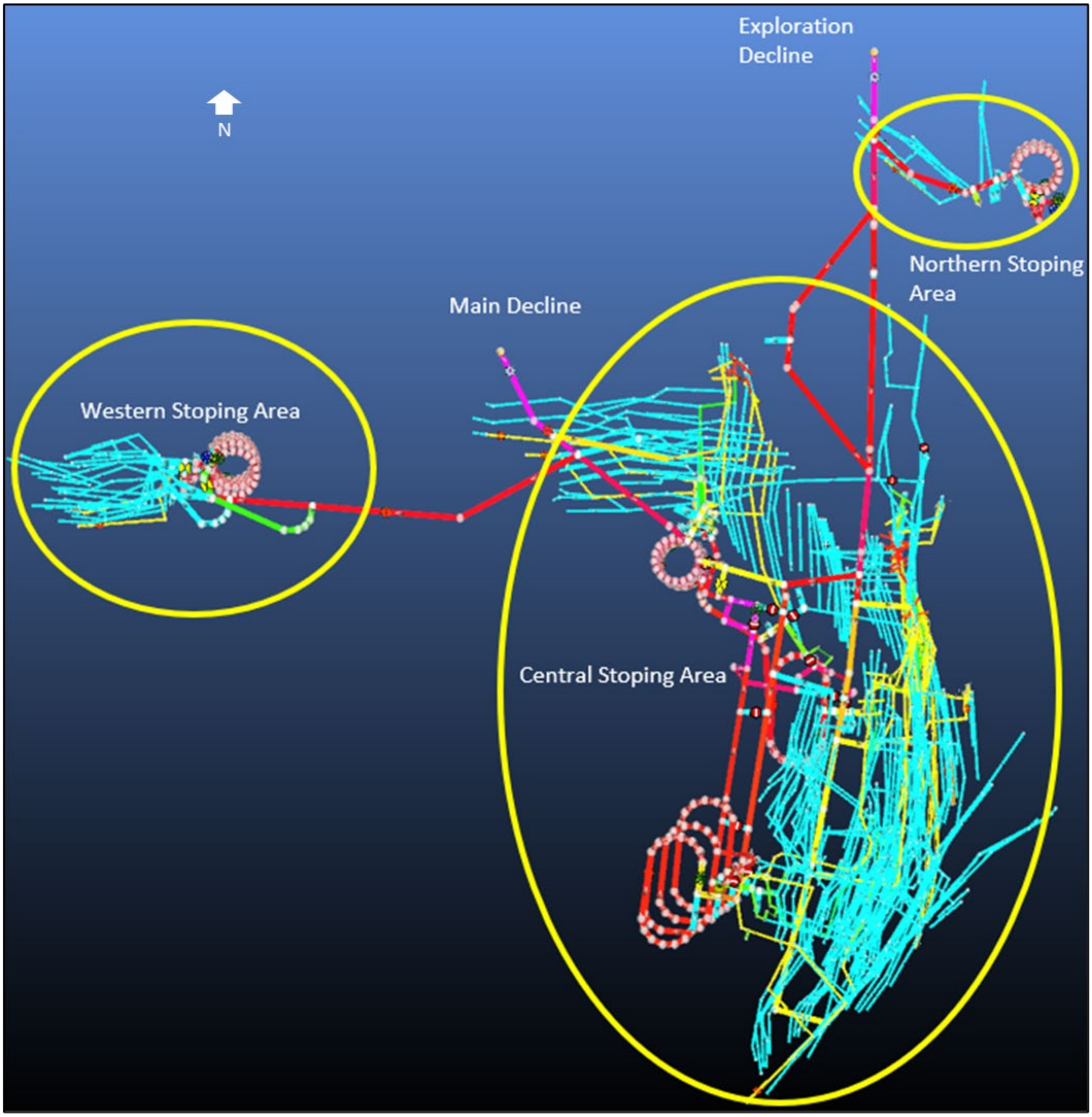
One hundred and twenty meters west of the link between the two declines, the main decline will continue to depth and an incline take-off will be used to access the upper areas of the mineralized material body. The main decline and incline will make up the Central stoping area. This Central area will have a centrally located return air rise to surface where most of the air ventilating the underground will be exhausted.

The Northern access through the exploration decline will split to the Tuvatu and H Lodes 65 m after entering the portal and will have a standalone return air network.

The used air in the Western and Northern stoping areas is planned to be exhausted through independent 3.0 m diameter raisebore shafts to surface. The Central stoping area is planned to be exhausted via a 3.5 m diameter raisebore shaft to surface.

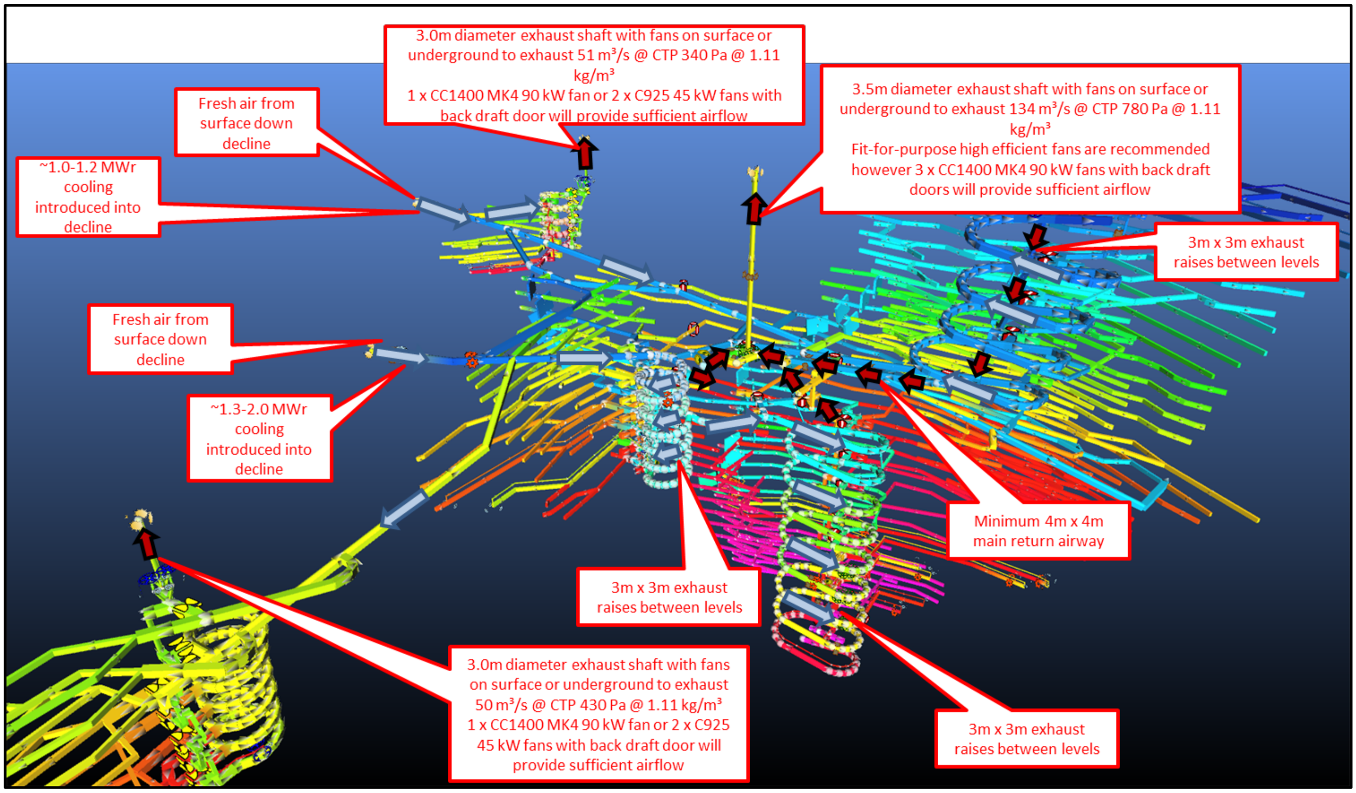
Escapeway rises on levels that link to the respective raisebore shafts that exhaust to surface will be used as a second means of egress for the Western and Northern stope areas. Escapeway rises on levels that link to the exploration decline will be used as a second means of egress for the Central stoping area. In the event of an emergency, the Central stoping area will allow personnel to exit the mine via the main decline or the exploration decline.

Figure 16-12 and Figure 16-13 show the ventilation layout.



Source: Entech (2019)

Figure 16-12: Tuvatu ventilation layout (plan view)



Source: Entech (2019)

Figure 16-13: Tuvatu ventilation layout (isometric view)

The ventilation analysis is based on requirements for equipment, infrastructure (underground fuel bay and dewatering station), and leakage.

Table 16-13 summarizes the airflow requirements.

Table 16-13: Air requirement summary

Area	Unit	Requirement
Air Quantity for Equipment	m^3/s	89
Air Mass Flow for Equipment	kg/s	102
Air Mass Flow for Infrastructure	kg/s	34
Leakage @ 25%	kg/s	34
Total Air Requirement	kg/s	170
Total Air Requirement	m^3/s	155

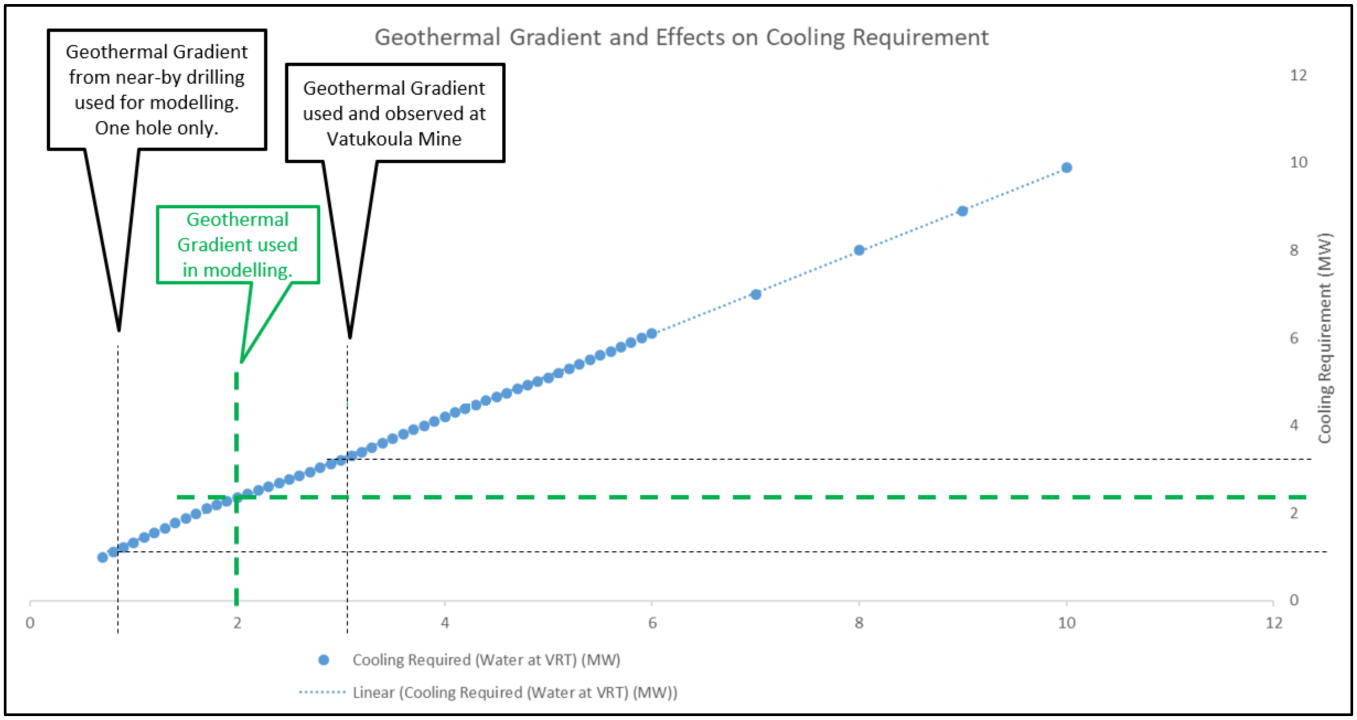
16.8.1.3 Heat

A high-level heat balance was conducted for the mine design; however, the lack of geothermal data means a full analysis cannot be done until geothermal drilling and analysis are completed. Table 16-14 provides a summary of the heat load components and subsequent heat balance.

Table 16-14: Heat balance summary

Heat Summary	Amount (MW)
Heat Summary	
Heat from Autocompression	0.6
Heat from Diesel Equipment	1.8
Heat from Strata	0.1
Heat from Fans	0.5
Heat from Ground Water	0.3
Other Heat Sources	0.0
Total Heat load	3.3
Cooling Summary	
Natural Cooling	1.1
Artificial Cooling	0.0
Total Available Cooling	1.1
Heat Balance	
Heating	3.3
Cooling	1.1
Balance	**2.2**

Table 16-14 shows that at a temperature gradient of 2.0°C/100 m, 2.2 MW of artificial cooling would be required. Figure 16-14 shows that based on current input parameters, artificial cooling will be required at a certain mining depth. Geothermal drilling is necessary to estimate the amount of artificial cooling and the timing of when that cooling is required.



Source: Entech (2019)

Figure 16-14: Geothermal gradient vs. artificial cooling requirements

16.8.1.4 Primary Fans

The primary fan requirements were modelled in Ventsim™, which suggests there is flexibility in fan selection. Entech recommends one, CC1400 MK4 90 kW fan at each of the Northern and Western return air rises, and a fit-for-purpose primary fan at the Central exhaust. The primary fan at the Central exhaust is recommended to be a custom, variable-speed drive (VSD) fan to deliver maximum efficiency and minimize power costs.

Three different fans were assessed with a total of six fan configurations. The assessment comprised testing the different fan configurations in the Ventsim™ model, as well as assessing the capital and operating cost components.

Table 16-15 provides a summary of the assessment.

Table 16-15: Primary fan cost estimation

	1 x Howden AFSO 2450/1200 with 50% Solidity at 55°	1 x Howden AFSO 2450/1200 with 100% Solidity at 55°	1 x Howden AFSO 2450/1200 with 100% Solidity at 50°	1 x MTV KGL 280/(d) AL28 fan at Blade setting 4	1 x MTV KGL 280/(d) AL28 fan at Blade setting 5	1 x AirEng Series 244-165-900-A-1 Axial Flow fan on setting 6
Fan Speed Setting (%)	100%	100%	100%	100%	92%	100%
VSD Included (Yes/No)	No	No	No	Yes	Yes	Yes
Required Airflow (kg/s)	170	170	170	170	170	170
Achieved Airflow (kg/s)	166	184	164	166	168	170
Required Quantity (m³/s)	158	158	158	158	158	158
Achieved Quantity (m³/s)	156	174	154	157	158	161
Simulated Fan Total Pressure (Pa)	2290	2810	2240	2530	2580	2910
Fan Power (kW)	357	489	345	397	408	469
Fan Efficiency (%)	***72%***	***72%***	***72%***	77%	67%	73%
Absorbed Power (kW)	496	679	479	516	608	642
Fan Capital Cost ($USD)	$ 1,342,500	$ 1,402,500	$ 1,402,500	$ 853,379	$ 853,379	$ 1,402,844
Cost per kWh ($USD)	***$ 0.24*** ***LOM***		***5*** ***count Ra***	***8%***		
			an	***3.9927***		
Input Power Cost per Annum ($USD)	$ 1,043,141	$ 1,427,705	$ 1,007,283	$ 1,084,538	$ 1,279,138	$ 1,349,309
Total OPEX LOM ($USD)	$ 20,824,741.36	$ 28,501,984.77	$ 20,108,898	$ 21,651,176	$ 25,536,068	$ 26,936,926
TOC ($USD)	***$ 22,167,241***	***$ 29,904,485***	***$ 21,511,398***	***$ 22,504,555***	***$ 26,389,447***	***$ 28,339,770***
Rating	**350**	**230**	**360**	**370**	**330**	**280**
Maximum Rating Achievable	**540**	**540**	**540**	**540**	**540**	**540**
Score (%)	**65%**	**43%**	**67%**	**69%**	**61%**	**52%**
Cost Weighted Rating Legend						
Points						
10	**Least Favorable**					
20						
30						
40						
50						
60	**Most Favorable**					

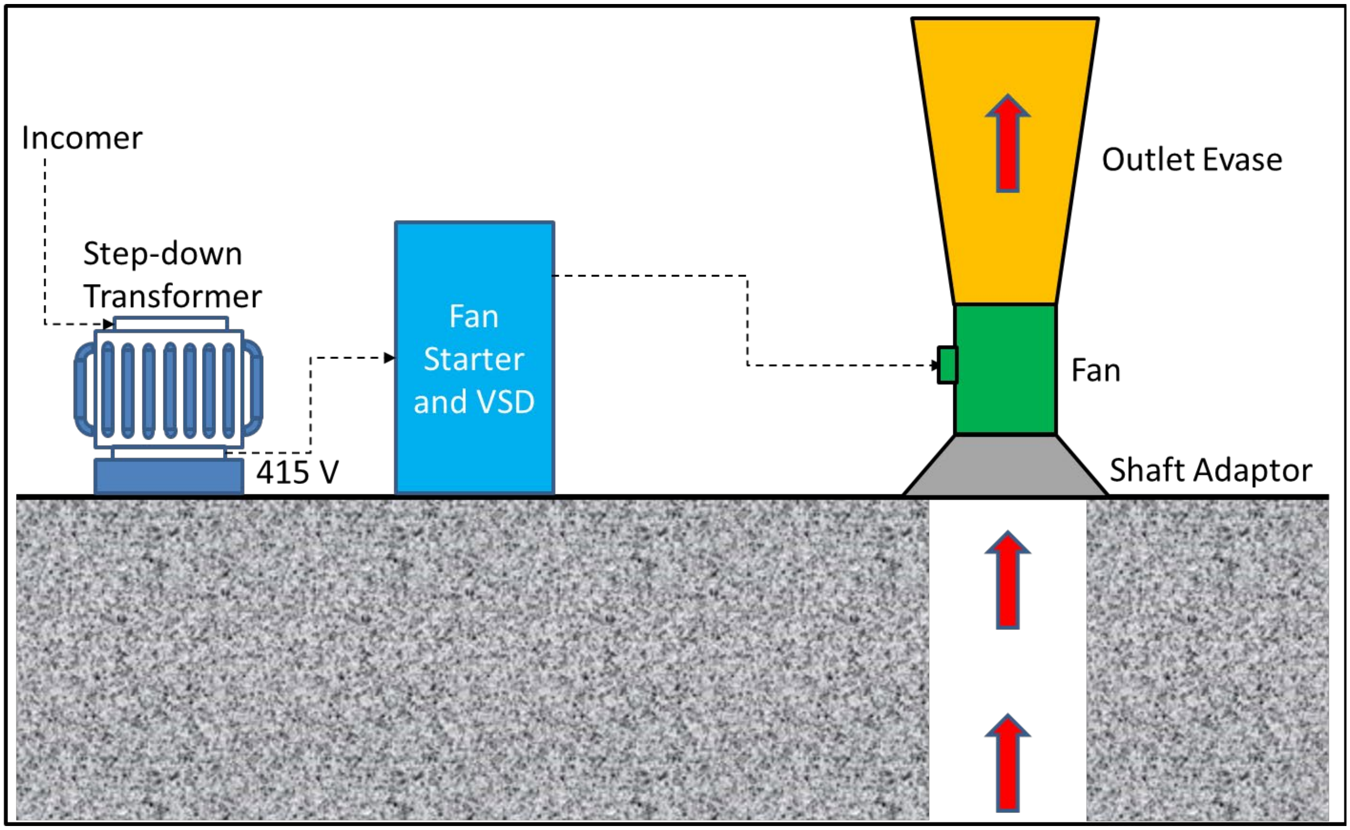
The assessment in Table 16-15 shows that all the fan configurations would be adequate. The 415 V MTV KGL 280/(d) AL28 fan option provides a low capital cost and slightly higher operating cost solution, with a VSD to help reduce power consumption, whereas the 415 V Howden AFSO 2450/1200 fan with soft starter option provides a higher capital cost and lower operating cost solution.

Table 16-16 shows the fan duty points issued to source the appropriate fans.

Table 16-16: Fan duty points

Duty Point	Quantity (m³/s)	Collar Total Pressure (Pa)	Density (kg/m³)	Resistance (N.s²/m⁸)
1	158	820	1.10	0.0329
2	158	1,290	1.10	0.0517
3	159	2,280	1.09	0.0902

The fan will be vertically mounted on top of the Central exhaust, as illustrated in Figure 16-15.

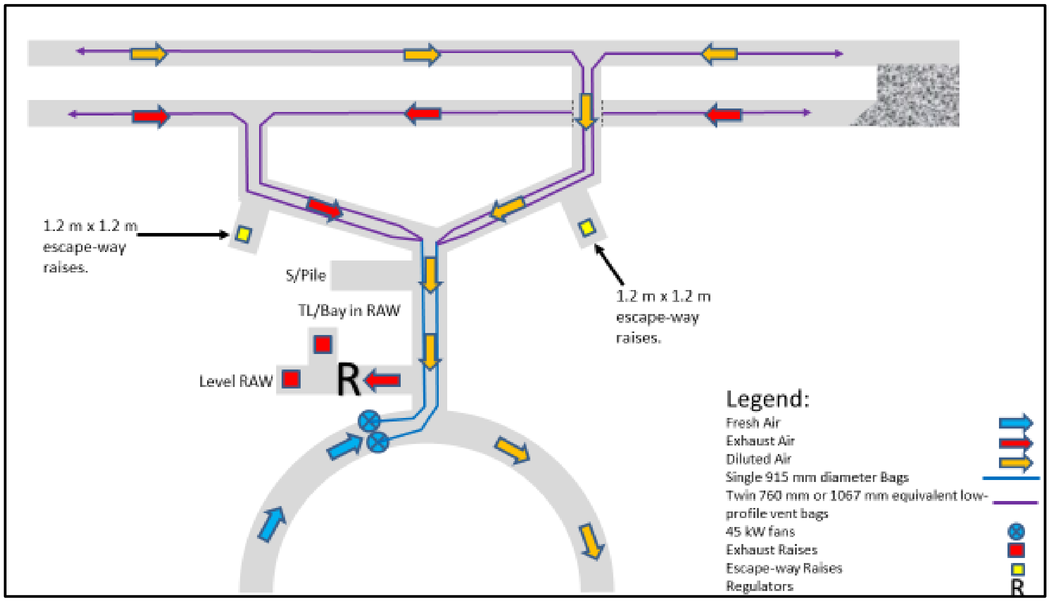


Source: Entech (2019)

Figure 16-15: Vertically mounted primary fan

16.8.1.5 Production Ventilation Layout

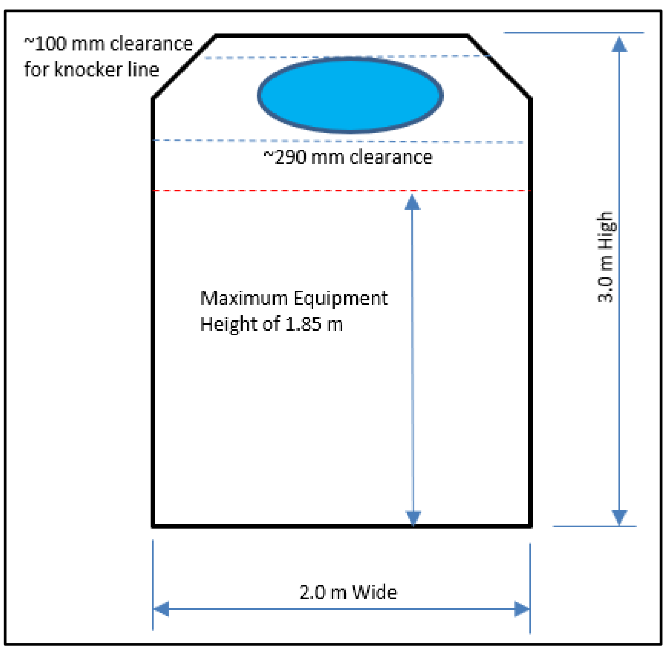
Figure 16-16 shows a typical secondary level layout that depicts 45 kW fans used as secondary fans to supply air to a given level, and a single 915 mm diameter bag, reducing to 1,067 mm equivalent low-profile ventilation bags, to service the mineralized material drives.



Source: Entech (2019)

Figure 16-16: Production level layout

Figure 16-17 shows a typical mineralized material drive, which displays the clearance from the 1,067-low-profile ventilation bag compared to the maximum equipment height.

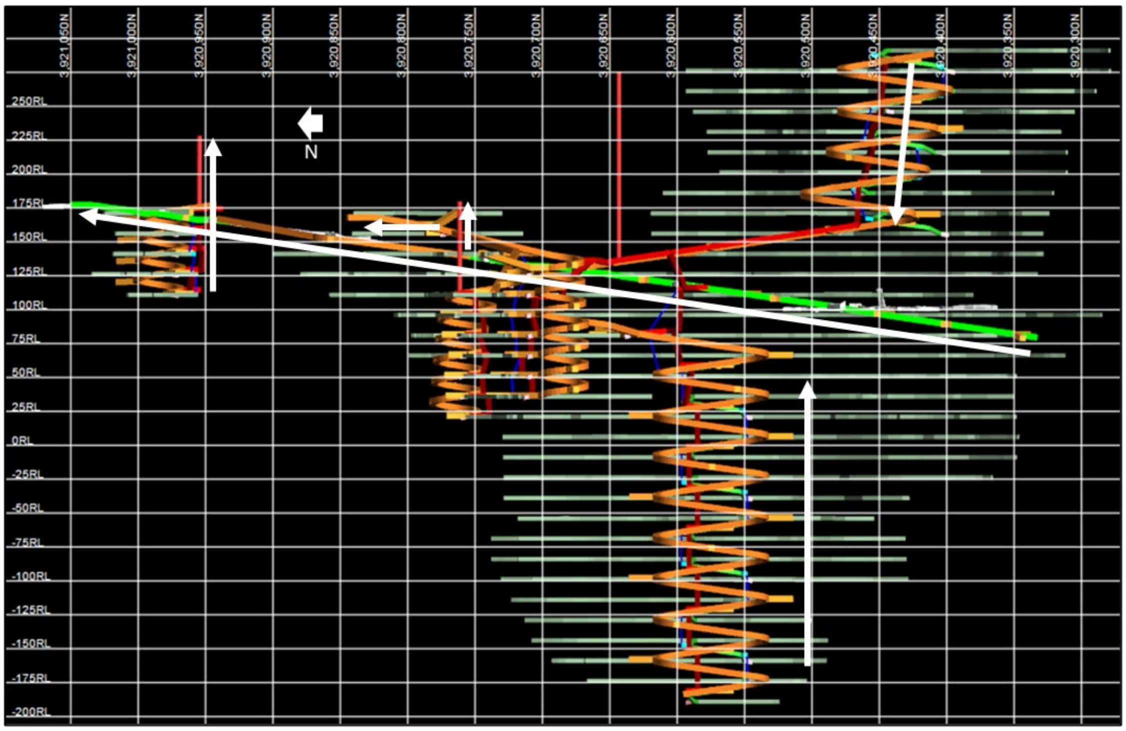


Source: Entech (2019)

Figure 16-17: Typical mineralized material drive ventilation layout

16.8.2 Second Means of Egress

Escapeway rises between levels with a final return to surface via the return air raise will be the secondary means of egress for the Northern and Western mining areas. The Central mining area will have escapeway rises between levels, which link to both the main and exploration declines, allowing two different routes to escape the mine in the event of emergency. Figure 16-18 shows the lateral and vertical development layout with the escapeway network.



Source: Entech (2019)

Figure 16-18: Lateral and vertical development layout showing the escapeway network. White arrows show routes to escape for different areas of the mine.

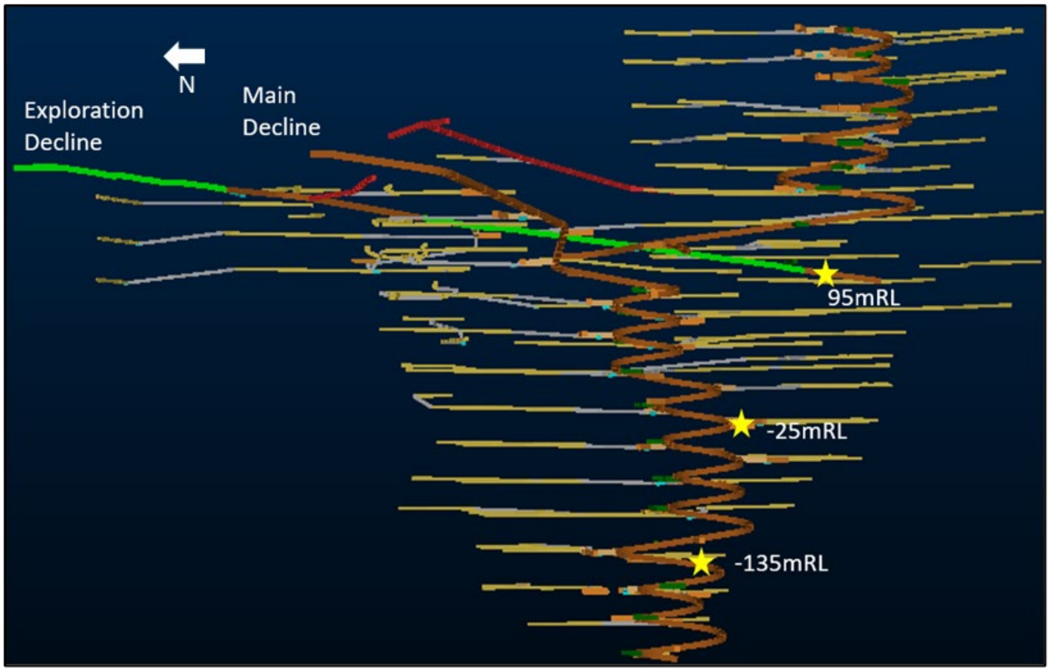
16.8.3 Underground Dewatering

16.8.3.1 Dewatering System Capacity

A conceptual pumping layout for the underground mine has been outlined and has the capacity to dewater the mine at a rate of up to 40 L/s. Previous studies have highlighted that this will provide enough capacity to dewater the mine.

16.8.3.2 Primary Pumping System

The conceptual primary pumping system comprises a combination of H104 and H103 mono pumps. A bank of two, 104 mono pumps would be placed in the stockpile located near the bottom of the exploration decline (sump positioned at the bottom of the exploration decline). A further two, 103 mono pumps would be positioned at approximately 120 m vertical positions in decline stockpiles, as shown in Figure 16-19.



Source: Entech (2019)

Figure 16-19: Conceptual dewatering layout with 103 mono pumps (yellow)

16.8.3.3 Secondary Pumping System

The secondary dewatering system will include 3.5 and 8 kW face pumps and 20 kW electric submersible pumps (e.g., Flygt pumps). The secondary pumps will transfer water from sumps located in the active headings to the primary pumping system.

16.8.4 Underground Power

Power is planned to be supplied via 415 V diesel powered gensets with a 415 V to 1,000 V step up transformer at each portal. The 1,000 V power will supply three underground substations, which will distribute power to the work areas. There is an allowance for three underground substations: 1 MVA, 1.5 MVA, and 2 MVA.

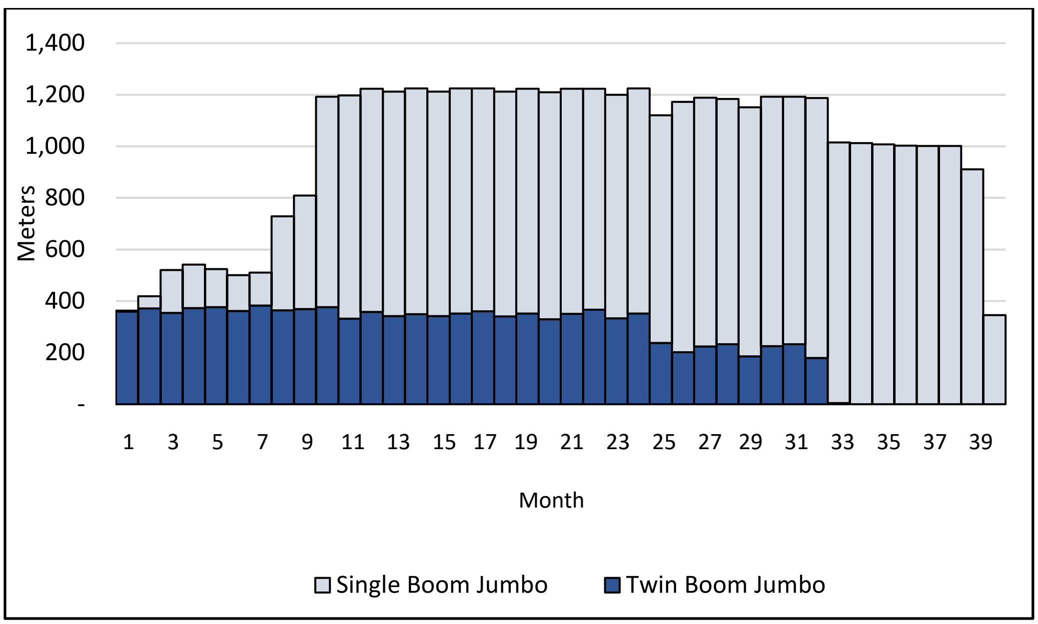
16.9 Life-of-Mine Production

16.9.1 Lateral Development

Table 16-17 shows the lateral development physicals total and Figure 16-20 shows the lateral development advance by month.

Table 16-17: Lateral development

Lateral Development	Units	Total
Capital		
Decline (4.5 m width x 4.5 m height SA)	m	5,689
Decline Strip (4.5 m width x 4.5 m height SA)	m	551
Lateral (4.5 m width x 4.5 m height SA)	m	2,851
Lateral (4.0 m width x 4.0 m height SA)	m	1,181
Lateral (3.0 m width x 3.0 m height SA)	m	229
Lateral (2.0 m width x 3.0 m height SA)	m	229
Operating		
Lateral (2.0 m width x 3.0 m height SA)	m	2,394
Mineralized Material Driving (2.0 m width x 3.0 m height SA)	m	26,699
Total Lateral Development	**m**	**39,823**



Source: Entech (2019)

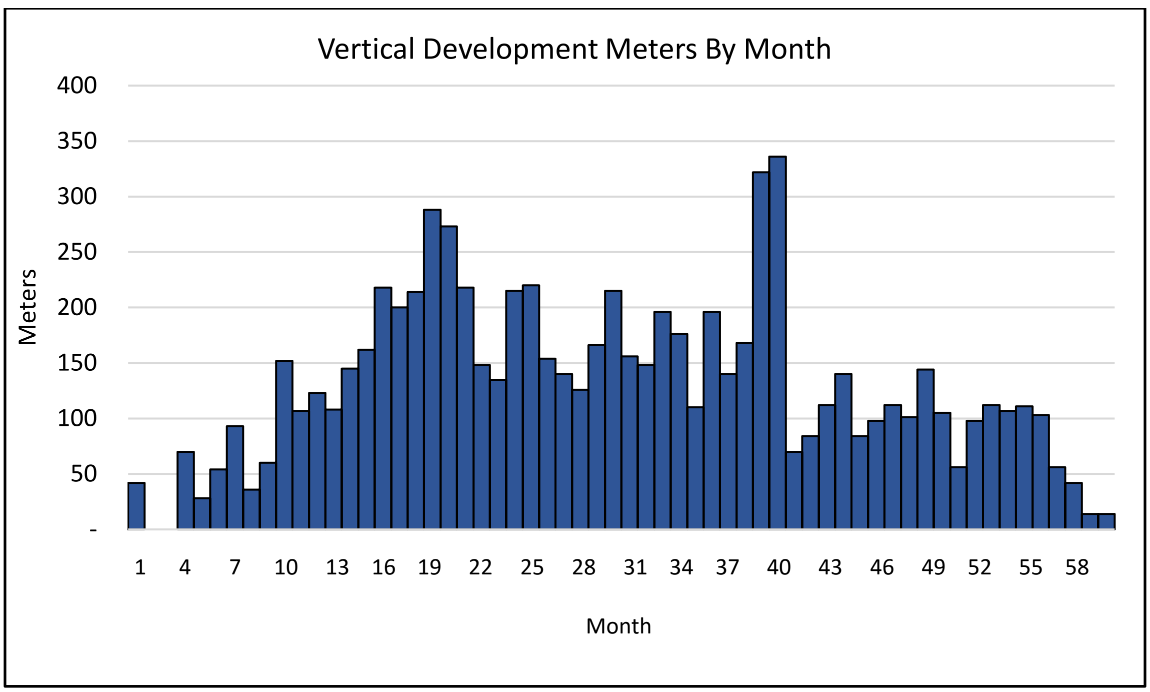
Figure 16-20: Lateral development meters by month

16.9.2 Vertical Development

Table 16-18 shows the total vertical development physicals and Figure 16-21 shows the vertical development by month.

Table 16-18: Vertical development

Vertical Development	Units	Total
Capital		
Surface Raisebore (3.5 m diameter)	m	267
Return Air Raise (3.0 m x 3.0 m)	m	651
Airleg Rise Escapeway (1.2 m x 1.2 m)	m	537
Operating		
Airleg Rise Slot (1.2 m x 1.2 m)	m	6,369
Total Vertical Development	**m**	**7,824**



Source: Entech (2019)

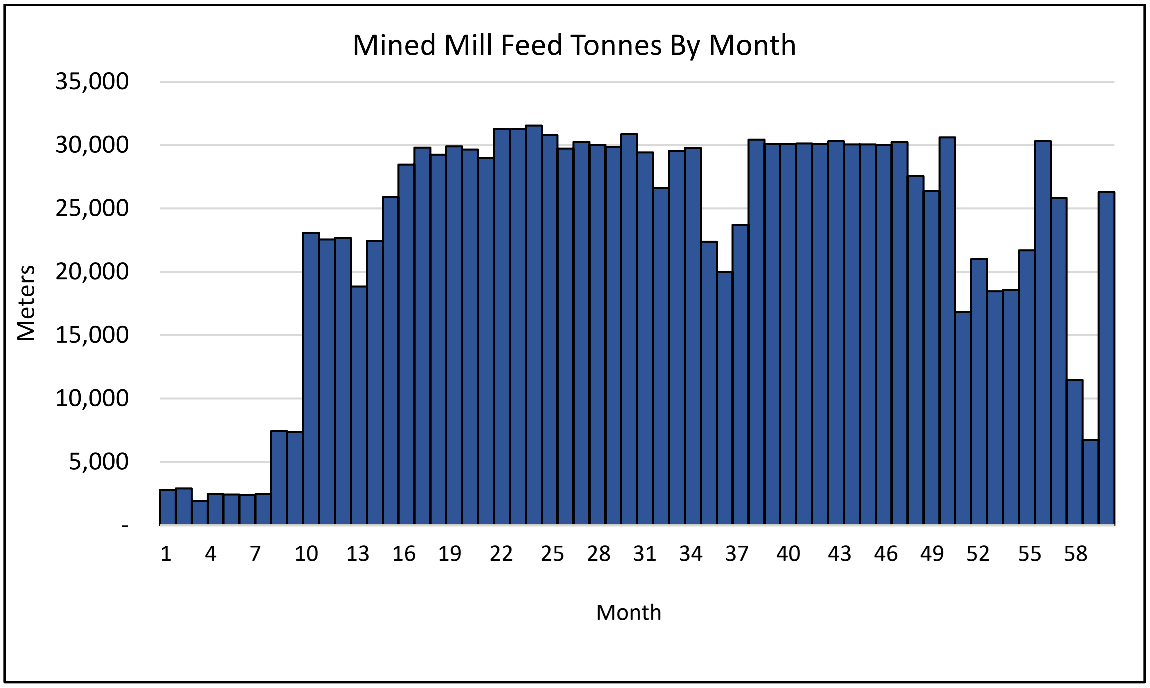
Figure 16-21: Vertical development meters by month

16.9.3 Material Movement

Table 16-19 shows the total mined mill feed and waste physicals and Figure 16-22 shows the mined mill feed tonnes by month.

Table 16-19: Material movement

Material Movement	Units	Total
Development		
Waste	t	681,000
Mined Mill Feed	t	308,000
Gold Grade	g/t	6.90
Gold Ounces	oz	68,500
Stoping		
Mined Mill Feed	t	1,076,000
Gold Grade	g/t	9.10
Gold Ounces	oz	313,000
Mined Mill feed Production – Total		
Total Mined Mill Feed	t	1,384,000
Gold Grade	g/t	8.60
Gold Ounces	oz	381,500



Source: Entech (2019)

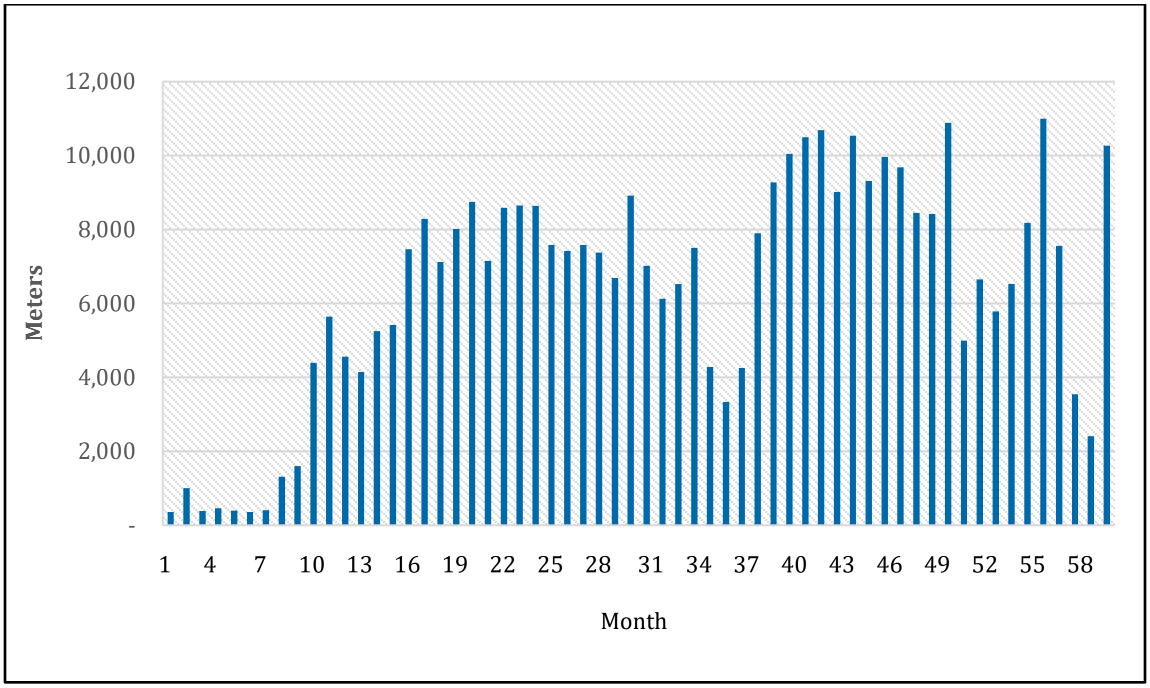
Figure 16-22: Mined mill feed tonnes by month

16.9.4 Production Drill Physicals

Table 16-20 shows the total production drill meters and Figure 16-23 shows the production drill meters by month.

Table 16-20: Production drill physicals

Drilling	Units	Total
Production Drill (64 mm)	m	378,000
Airleg Pilot Hole (127 mm)	m	6,900



Source: Entech (2019)

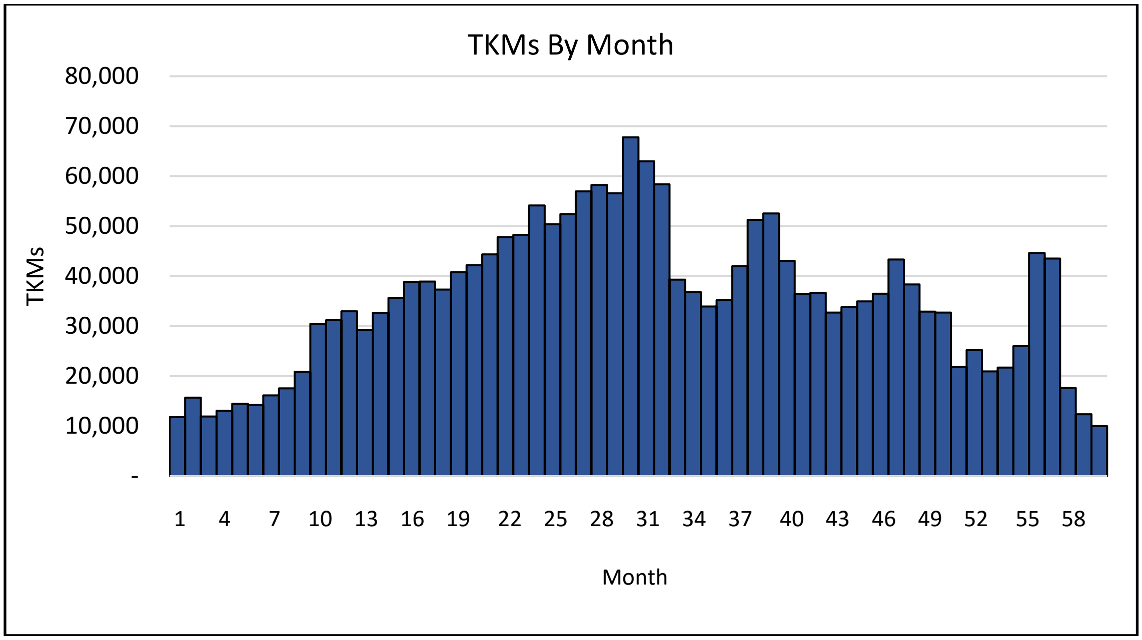
Figure 16-23: Production drill meters by month

16.9.5 Haulage Physicals

Table 16-21 shows the total haulage physicals and Figure 16-24 shows the LOM monthly tonne-kilometers.

Table 16-21: Haulage physicals

Material Movement	Units	Total
Total Movement	tkm	2,120,000



Source: Entech (2019)

Figure 16-24: Tonne-kilometers by month

16.9.6 Annual Physicals

Table 16-22 shows the Tuvatu key annual physicals.

Table 16-22: Annual physicals

Physicals	Units	Y-1	Y1	Y2	Y3	Y4	Y5	Totals
Total Mined Mill Feed	t	32,150	311,553	361,192	337,138	297,533	44,501	1,384,067
Gold Grade Mined	g/t	7.77	7.42	8.50	10.40	8.09	7.31	8.57
Gold Mined	oz	8,032	74,356	98,596	112,718	77,364	10,466	381,532
Lateral Development	m	4,917	14,575	14,049	6,282	0	0	39,823
Vertical Development	m	385	2,209	2,019	1,938	1,203	70	7,824
Production Drilling	m	6,315	76,215	91,125	96,660	98,107	16,221	384,643
Haulage	tkm	135,703	434,447	653,107	469,505	387,502	40,007	2,120,271

16.9.7 Mining Fleet and Manning Requirements

Mobile fleet equipment requirements are based on the specified equipment list the mining contractor provided in their cost submission. The fleet estimates outlined in this section represent the equipment necessary to perform the following duties:

- Excavate the lateral and decline development in both mined mill feed material and waste.
- Install all ground support, including rock bolting and surface support.

- Maintain the underground road surfaces.
- Drill, charge, and bog (including remote bogging) all stoping mineralized material.
- Haul all material out of the mine to Run-of-Mine (ROM) pad or waste dump.
- Drill slot raises for production stoping and other miscellaneous longhole drilling.

Twin-boom jumbos are planned to be used for most capital development. Single-boom jumbos have been assumed for smaller profile mineralized material driving and small amounts of capital, with booms of a suitable length to allow adequate installation of ground support.

A 5 m^3 sized loader (CAT 1700 or equivalent) has been assumed for the majority of capital development bogging and truck loading. 1.5 m^3 sized loaders (LH203 or equivalent) have been assumed for smaller profile mineralized material development and stope bogging.

Production drilling will be achieved using a drill capable of accurately drilling the required 64 mm diameter hole lengths such as the Resemin Muki drill or equivalent.

Haulage will be via 16 t trucks. Vertical development is planned to be excavated with a combination of raisebore drills, airleg rising, and longhole rising.

Table 16-23 shows the maximum fleet estimates.

Table 16-23: Tuvatu equipment list

Equipment	Maximum Quantity
Twin Boom Jumbo Drill	1
Single Boom Jumbo Drill	5
Production Drill	2
Raisebore Unit	1
Development and Stope Loader – LH203	5
Development Loader – 1700	2
Underground Truck – 16 t	4
Integrated Tool Carrier	1
Telehandler	1

The mining contractor estimated the mining manpower requirements and Entech estimated Lion One’s staff requirements. Table 16-24 shows the maximum manpower requirements.

Table 16-24: Lion One manning list

Role	# Personnel	Roster (Days On: Days Off)
Lion One Mining Staff		
Underground Manager	1	5:2
Senior Mining Engineer	1	5:2
Senior Mine Geologist	1	5:2
Geologist	1	5:2
Senior Mine Surveyor	1	5:2
Geology Technician / Underground Sampler	2	5:2
Total Lion One Mining Staff Personnel	**7**	**-**
Underground Mining Contractor		
Project Management and Administration Expatriate Manning (Underground Mining Contractor)		
Project Manager	1	12:1
Mine Foreman	1	12:1
Mining Engineer	1	12:1
Maintenance Superintendent	1	12:1
Electrical Superintendent	1	12:1
Underground Supervisor	3	12:1
Maintenance Planner	1	12:1
OH&S/Training Manager	1	12:1
Jumbo Coach	6	12:1
Loader Coach	3	12:1
Production Driller Coach	3	12:1
Charge Up Coach	3	12:1
Raise Drill Coach	2	12:1
Underground Electrical Supervisor	1	12:1
Fitter Coach Expat	6	12:1
Electrical Coach Expat	3	12:1
Procurement Manager	1	5:2

table continues…

TETRA TECH

Role	# Personnel	Roster (Days On: Days Off)
National Manning		
Engineers	2	6:1
Underground Supervisor	9	6:1
Maintenance Planner	1	6:1
OH&S Trainer	3	6:1
Site Clerk	3	6:1
Administration Coordinator	1	6:1
Clerk/Stores	6	6:1
Fitter/Mechanic	9	6:1
Electrical Tradesperson	6	6:1
Auto – Electrician	3	6:1
National Manning Operators		
Twin Boom Jumbo Operator	5	6:1
Single Boom Jumbo Operator	10	6:1
Production Drill Operator	8	6:1
Development and Stope Loader Operator	9	6:1
1700 Development Loader Operator	6	6:1
Underground Truck Operator	9	6:1
Integrated Tool Carrier Operator	3	6:1
Telehandler Operator	3	6:1
Charge up Operator / Blaster / Shotfirer	6	6:1
Service Truck Operator	6	6:1
Stores Truck Operator	2	6:1

Note: OH&S – Occupational Health and Safety

17.0 RECOVERY METHODS

Lion One contracted Jinpeng Mining to complete a process design for gold recovery from the Tuvatu mineralization. Jinpeng Mining is a process equipment manufacturer who has a technical support team for process design. Jinpeng Mining completed the flowsheet development, mass balance, equipment sizing, general plant layouts, and circuit layouts. Tetra Tech was also involved in the process design and reviewed the design. The general process design and process description are discussed in the following sections.

The metallurgical test work results described in Section 13.0 were used to select the recovery method for the Project and to develop the process design criteria. The metallurgical test results indicate that the Tuvatu mineralization is amenable to a combined process of gravity concentration and flotation followed by cyanidation. The process facility, together with the process flowsheet, was designed based on the process design criteria. Design factors, where applicable, are included in the equipment sizing and circuit design.

The proposed process plant will process the mineralized material at a rate of 1,000 t/d with an average LOM head grade of 8.56 g/t Au. The average gold recovery is estimated to be approximately 87.5%. The comminution circuits, including two-stage grinding circuit, will grind the mill feed to a grind size of 80% passing (P_{80}) 60 to 65 µm. A gravity separation circuit will be integrated with the primary grinding circuit to recover the coarse-free gold grains. The hydrocyclone overflow from the primary grinding circuit will be concentrated by flotation to separate sulphide minerals from non-sulphide minerals. The resulting flotation concentrate will be reground to 80% passing approximately 20 µm, followed by aeration pre-treatment prior to cyanide leaching. The flotation tailings will be cyanide leached as well. CIP treatment is proposed for extracting the gold and associated silver from the mill feed. The loaded carbon will be stripped, and the pregnant solution will be treated by a heated and pressured electrowinning unit to recover the gold from the solution. The carbon stripping and gold electrowinning will be operated in a closed circuit. Gold doré will be produced from an electric furnace located on site. The leach residue will be treated by cyanide destruction using the SO_2/air procedure prior to being pumped to the TSF.

The crushing circuit will operate during the day shift, while the milling and leaching circuits will operate 24 h/d and 330 d/a or 365 d/a with an availability of 90.4%. Carbon stripping and gold electrowinning circuits will operate 16 h/d.

17.1 Introduction

The proposed process plant will include the following unit operations:

- Primary Crushing – A truck dump hopper with a fixed grizzly, a vibrating grizzly, and a jaw crusher in open circuit producing a final product of 80% passing approximately 70 mm.
- Secondary/Tertiary Crushing – Two cone crushers in closed circuit with a vibrating double deck screen to further reduce the particle size of the primary crushing discharge to approximately 80% passing 8 to 10 mm.
- Primary Grinding – Two ball mills in series in closed circuit, with hydrocyclones to further reduce the crushed materials to a product of 80% passing approximately 60 to 65 µm.
- Gravity Separation – Integrated with the primary grinding circuit, a gravity separation circuit receiving approximately 33% of the hydrocyclone underflow to recover the coarse-free gold grains using two centrifugal concentrators and one shaking table.
- Flotation – Sulphide flotation of the hydrocyclone overflow to produce a gold-bearing sulphide concentrate.

- Flotation Concentrate Regrinding – A regrind vertical mill together with hydrocyclones in open circuit to regrind the flotation concentrate to a particle size of 80% passing approximately 20 µm.
- Cyanide Leaching – Gold leaching of the flotation tailings and the reground concentrate through the two separate leaching circuits followed by one common CIP circuit. The leach circuit will be aerated with air.
- Loaded Carbon Acid Washing, Desorption, and Refining – The loaded carbon from the CIP circuit to be treated by elution to produce a gold rich solution for electrowinning and then melting to produce gold doré. The stripped carbon will be reused in the CIP circuit either after acid washing to remove inorganic contaminants or treated by thermal regeneration.
- Carbon Handling – Thermal regeneration of barren carbon to remove organic foulants and preparation of make-up new carbon by attrition and sizing.
- Cyanide Detoxification – Detoxification of cyanide leach residue slurry using the SO_2/air process to destruct WAD cyanide to less than 1 ppm level prior to disposal of the detoxified tailings in the conventional TSF.
- Final Tailings Disposal – The detoxified leach residue slurry to be pumped to the TSF where a reclaimed water system will pump the collected water back to the mill site for process use.

17.2 Process Design Criteria

17.2.1 Process Design Criteria

Key process design criteria are shown in Table 17-1. The designed life for the Tuvatu process plant will be approximately six years.

Table 17-1: Major process design criteria

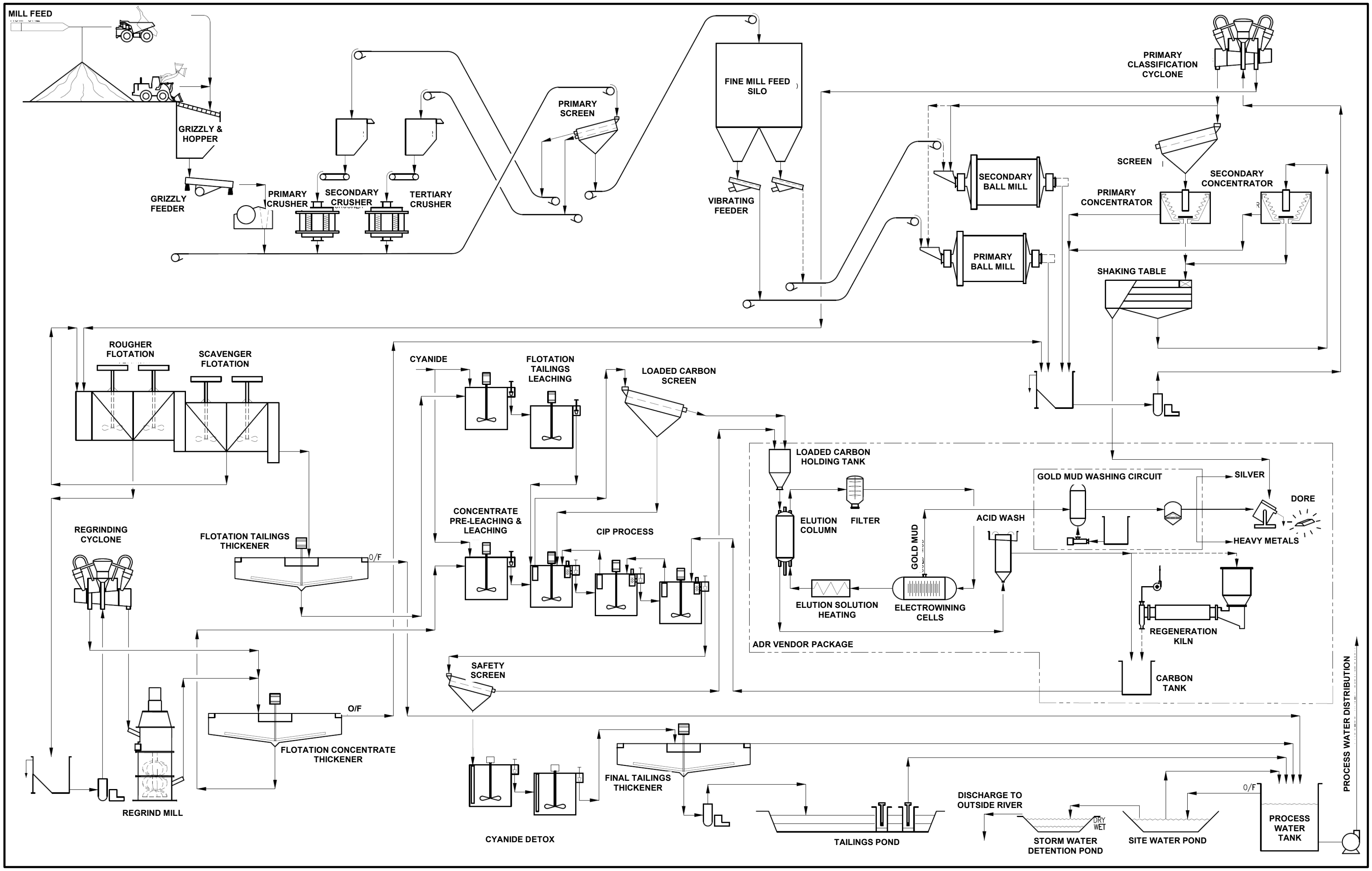
Criteria	Unit	Nominal Value
Mill Feed Characteristics		
Specific Gravity	g/cm^3	2.7
ROM Moisture	%	5
Bulk Density	t/m^3	1.6
C_{Wi}	kWh/t	12.5
A_i	kWh/t	0.184
R_{Wi} (75th)	kWh/t	19.8
B_{Wi} (75th)	kWh/t	18.6
LOM Average Gold Grade	g/t	8.6
Operation Schedule		
Operating Days per Year	d/a	330

table continues…

Criteria	Unit	Nominal Value
Daily Shifts		
Crushing	shift/d	1
Grinding/Flotation/Leaching	shift/d	3
ADR	shift/d	2
Operating Hours per Shift		
Crushing	h/shift	10
Grinding/Flotation/Leaching	h/shift	8
Process Plant Throughput	t/d	1,000
Process Plant Throughput – Crushing	t/h	125
Process Plan Throughput – Grinding/Flotation/Leaching	t/h	41.7
Main Process Plant Availability	%	90.4
Process Method & Gold Recovery		
Gold Recovery Method	-	Gravity + Flotation + Cyanidation
Primary Grind Size, 80% Passing	µm	60 - 65
Flotation Concentrate Regrind Size, 80% Passing	µm	20
Leach Residue Treatment	-	SO_2/Air
Tailings Storage	-	Conventional Wet Storage
Gold Recovery – Average	%	87.5
Gravity Separation	%	30 - 40

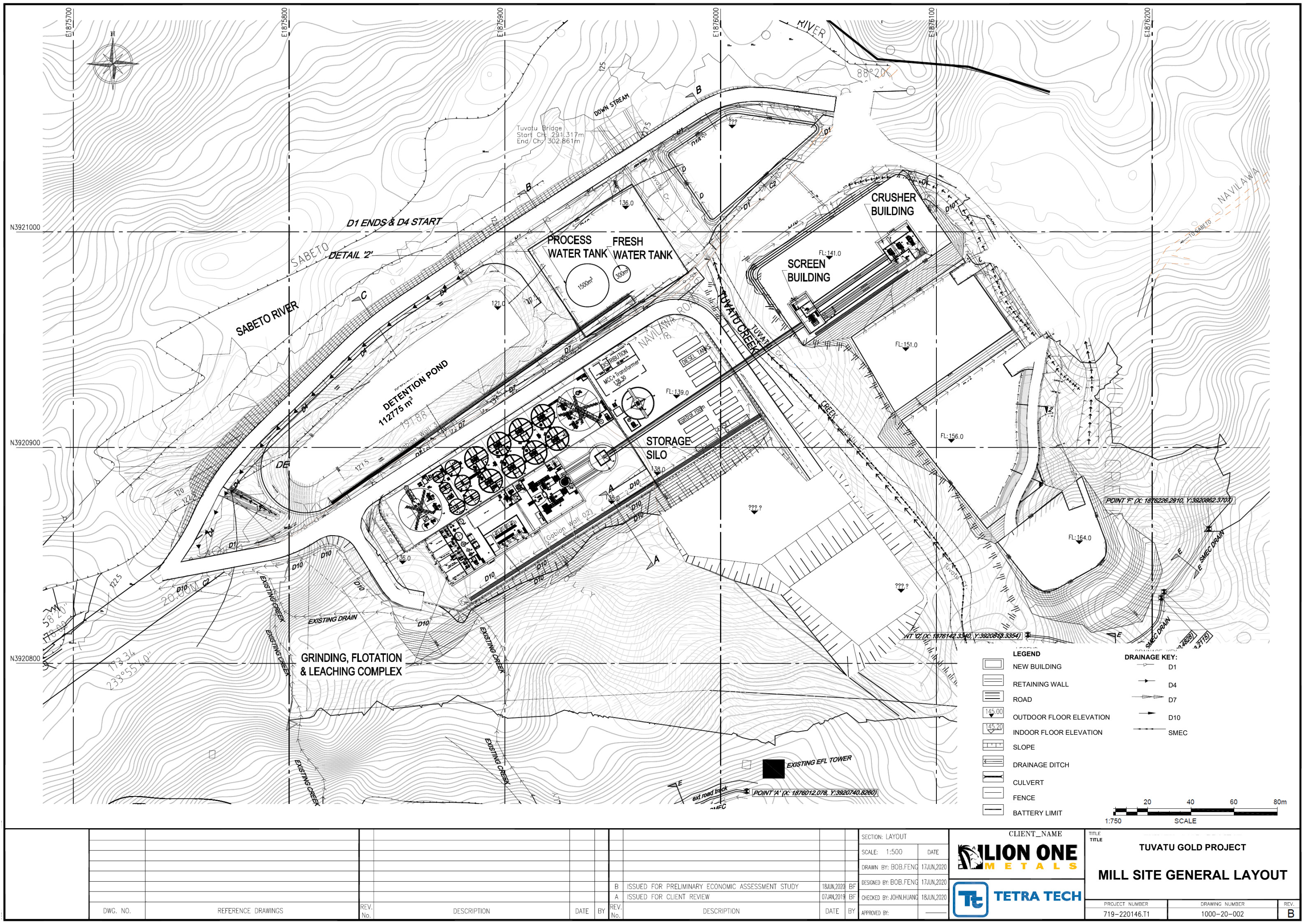
17.2.2 Plant Design

The proposed process flowsheet is presented in Figure 17-1. The general process plant arrangement is illustrated in Figure 17-2.



Source: Tetra Tech (2019)

Figure 17-1: Simplified process flow diagram



Source: Tetra Tech (2020)

Figure 17-2: Mill layout

17.3 Process Plant Description

17.3.1 Crushing

ROM from the underground mining operation will be hauled to the crushing area that will consist of a ROM receiving/storage pad and a three-stages crushing circuit. The plant will process 125 t/h of material, operate 10 h/d, and produce a final product of 80% passing 8 to 10 mm.

17.3.1.1 Primary Crushing

ROM material will either be stockpiled on the ROM receiving pad and reclaimed by a front-end loader to a jaw crusher feed hopper located at the northeast edge of the pad or directly dumped into the dump hopper. The jaw crusher feed hopper, with a 20 t live capacity, will be equipped with a static grizzly with 500 mm spacing. Oversize material from the static grizzly will be removed for particle size reduction using a rock breaker. The undersize material reporting to the dump hopper will be reclaimed by a vibrating grizzly feeder at a rate of 125 t/h. All the vibrating grizzly discharge will feed directly into a C106 or equivalent jaw crusher with an installed power of 110 kW. The jaw crusher discharge will report onto a screen feed conveyor. The primary crushing will reduce the feed particle size to 80% passing approximately 70 mm. The screen feed conveyor will transfer the primarily crushed product, together with the products from the secondary and tertiary crushers, to a double-deck vibrating screen.

17.3.1.2 Secondary and Tertiary Crushing and Classification

The secondary and tertiary crushers will be in closed circuit with a double-deck screen. The materials from the upper deck and lower deck of the screen will be separately conveyed to the secondary crusher and tertiary crusher feed surge bins. The undersize fraction, with a particle size of 80% passing approximately 8 to 10 mm from the screen lower deck, will be the final product of the crushing circuit and will be discharged onto the fine mill feed surge bin feed conveyor and then transferred to the surge bin.

Material from the secondary and tertiary crusher feed surge bins will be separately reclaimed by a belt feeder and separately feed into two HP200 or equivalent cone crushers, each with an installed power of 160 kW. The crushed products from the two crushers will return to the screen feed conveyor and then to the double-deck screen.

17.3.2 Fine Mill Feed Surge Bin

The double-deck screen undersize, with a particle size of 80% passing approximately 8 to 10 mm, will be conveyed to the fine mill feed surge bin, which can provide a live capacity of 1,000 t of the mill feed, or the equivalent of 24 hours of mill operation. Four vibrating feeders, together with two ball mill feed conveyors, will be installed underneath the surge bin in parallel forming two grinding mill feed systems. Each of the feeders can provide the full feed rate if one of the feeders will require unplanned maintenance. All the feeders will be equipped with variable frequency drive (VFD) control to adjust the reclaim rate. As designed, normally only one feeding system will be in operation.

17.3.3 Primary Grinding Circuit

Two ball mills will be installed in series (two stages of grinding) to grind the crushed mill feed to 80% passing approximately 60 to 65 µm. The grinding circuit can be converted into a one-stage grinding circuit with both the ball mills running in parallel. The grinding circuit design will provide better flexibility for the grinding circuit. For the two stages of grinding, one ball mill is the primary ball mill, which will be operated in open circuit, and the other will be the secondary ball mill, which will be operated in closed circuit with a hydrocyclone cluster. The nominal feed rate to the two-stage grinding circuit will be 41.7 t/h (fresh feed), and the final grind size of the hydrocyclone overflow will be 80% passing approximately 60 to 65 µm. A gravity separation circuit will be incorporated in the grinding circuit to recover coarse-free gold grains from the hydrocyclone underflow.

The reclaimed material from the fine mill feed surge bin will be conveyed and fed to a 3.2 m diameter x 5.4 m long overflow-type ball mill with an installed power of 800 kW. A belt scale on the mill feed conveyor will monitor/record the feed rate, which can be adjusted through the VFD control on the vibrating feeders. Water will be added to the mill to maintain the slurry solid density to approximately 75%. The slurry will overflow from the ball mill to a trommel screen attached to the discharge end of the mill. The trommel screen oversize will discharge into a trash bin. The trommel screen undersize slurry will flow by gravity into the hydrocyclone feed pump box where the discharge from the secondary ball mill and the gravity separation tailings will report to.

The combined slurry from the hydrocyclone feed pump box will be pumped up to a hydrocyclone cluster consisting of eight, 300 mm hydrocyclones (five operating and three standby) for size classification. The overflow of the hydrocyclones will flow to the flotation feed conditioning tank by gravity. Two-thirds of the hydrocyclone underflow will flow by gravity to the secondary ball mill, which is a 3.2 m diameter x 5.4 m long overflow ball mill for further grinding. The secondary ball mill discharge will report to the hydrocyclone feed pump box. One-third of the hydrocyclone underflow will report to the gravity separation circuit.

17.3.4 Gravity Separation

One-third of the hydrocyclone underflow will be directed to a gravity concentration circuit to recover the coarse-free gold grains. The gravity separation circuit will include:

- A gravity feed pumping system.
- A trash screen.
- A primary centrifugal concentrator.
- A shaking table to upgrade the concentrate from the primary gravity concentrator.
- A secondary centrifugal concentrator to recover the gold from the table tailings.

The hydrocyclone underflow will flow by gravity to the gravity separation circuit feed pump box where the slurry will be diluted with process water and pumped to the gravity concentrator feed trash screen. The screen oversize will discharge into the hydrocyclone feed pump box, the screen undersize will report to the primary centrifugal concentrator. The concentrate of the primary concentrator will feed to a shaking table feed surge bin and then be further upgraded by a shaking table while the gravity separation tailings will report to the hydrocyclone feed pump box. The final table concentrate, which is expected to contain approximately 23,500 g/t Au, will be treated directly by smelting on the site to generate gold-silver doré, while the table tailings will be sent to the secondary centrifugal concentrator. The concentrate of the secondary gravity concentrator will be recirculated back to the tabling treatment, while its tailings will flow together with the primary centrifugal tailings back to the hydrocyclone feed pump box.

17.3.5 Free Gold and Gold-Bearing Sulphide Flotation

The hydrocyclone overflow will report to the flotation circuit, consisting of rougher and scavenger flotation, to recover free gold and gold-bearing sulphide minerals. The flotation circuit consists of seven, 16 m^3 tank-type flotation cells equipped with froth paddle assemblies. Four of the flotation cells will be used for rougher flotation and the rest for scavenger flotation.

The flotation separation will be conducted at natural slurry pH. The hydrocyclone overflow will report to a conditioning tank where the collectors, potassium amyl xanthate (PAX) and A208 or equivalent, will be added to condition the slurry. The slurry will then flow by gravity into the head of the rougher flotation bank. DF250 or equivalent will be added as frother. The resulting flotation concentrate will be pumped to a concentrate regrind circuit. The rougher flotation tailings will flow to the scavenger flotation bank where a scavenger concentrate will be produced and be recycled back to the first flotation cell in the preceding rougher flotation bank. The scavenger flotation tailings will be pumped to the flotation tailings thickener prior to the tailings cyanide leach circuit.

17.3.6 Flotation Concentrate Regrinding

The flotation concentrate will be reground to further reduce the particle size down to 80% passing approximately 20 to 25 µm in an open regrind circuit. The flotation concentrate will be pumped to the hydrocyclone cluster consisting of six (four operating two standby) hydrocyclones, each with a diameter of 250 mm. The hydrocyclone overflow will report to the concentrate leach thickening feed pump box where the regrind mill will discharge to. The hydrocyclone underflow will report to a vertical regrinding mill with an installed power of approximately 180 kW. The product of the regrinding mill will also discharge into the concentrate leach thickener feed pump box. Together with the hydrocyclone overflow, the regrinding circuit product will be pumped to the flotation concentrate thickener feed well.

17.3.7 Flotation Concentrate Thickening

The reground flotation concentrate from the flotation regrinding circuit will be pumped onto a single deck trash screen for removal of any trash material. The oversize material will discharge into a trash bin, while the screen undersize will flow by gravity to a 15 m diameter high-rate thickener feed well. Flocculant solution will be added to the thickener feed to promote the settling of fine solids. The thickener will increase the slurry solids content to approximately 45 to 50% w/w solids. The thickener underflow will be pumped to the concentrate cyanide leach circuit. The thickener overflow will flow by gravity into the process water recycle tank and be used as make-up water in the grinding/gravity and flotation circuits.

17.3.8 Flotation Tailings Thickening

The flotation tailings will be pumped onto a single deck trash screen located above the tailings thickener for removal of any trash material. The oversize material will discharge into a trash bin, while the screen undersize will flow by gravity into a 20 m diameter tailings thickener. Similar to the concentrate thickener, flocculant will be added to the thickener to assist the thickening process. The target tailings thickener underflow is approximately 45 to 50% w/w solids. The thickened slurry will report to the tailings leach circuit. The overflow of the tailings thickener will flow by gravity to the process water tank to be used in the mill. Depending on the gold grade of the flotation tailings, the flotation tailings may by-pass the cyanidation circuit and directed into the final cyanide leach residue thickener where the thickened slurry will be pumped to the TSF.

17.3.9 Cyanide Leaching

The underflows from both the concentrate and tailings thickeners will be pumped separately to the concentrate and tailings cyanide leaching circuits, respectively.

The thickened flotation concentrate will be pumped to one, 10 m diameter x 10.5 m high pre-leach tank where slurry will be aerated with air at a pH of approximately 12 adjusted using hydrated lime. The pretreated slurry will flow by gravity into one, 10 m diameter x 10.5 m high leach tank and then to three additional tanks with the same tank sizing for leaching by sodium cyanide prior to reporting to a CIP circuit. The arrangement will provide sufficient pretreatment and leaching retention times for the concentrate leaching.

The thickened tailings will be pumped into two, 10 m diameter x 10.5 m high leach tanks for cyanide leaching before the slurry enters the common CIP circuit where the tailings and concentrate slurries will report to.

Air will be sparged into the bottom of all the aeration and leaching tanks. As designed, the total leaching retention time for the concentrate is approximately 96 hours or longer. With minor modifications to the leaching piping system, the leach circuit retention time can be adjusted to adapt to the mineralogical character or other requirements.

The cyanide leached concentrate and tailings slurries will report to the common CIP circuit consisting of six, 7 m diameter x 7.5 m high CIP tanks. Activated carbon granules will be added into the CIP tanks. The average carbon concentration in the CIP tanks will be approximately 20 g/L. The six-stage CIP circuit is expected to provide approximately 20 hours of adsorption retention time. Each CIP tank will be equipped with an inter-stage screen and an air lift device to advance the loaded carbon into the preceding CIP tank. The loaded carbon will leave the first CIP tank and report to a 1.2 m wide x 1.8 m long loaded carbon screen. The loaded carbon will be retained on top of the screen panel and will be transferred into a loaded carbon storage tank in the ADR circuit, while the slurry will pass through the screen apertures and return to the first CIP tank. Lime slurry will be added to the leach tanks to maintain protective alkalinity at a pH greater than 10.5, preventing the generation of hydrogen cyanide gas. Sodium cyanide will be added to the cyanide leach to extract gold and silver from the leach feed.

The tailings slurry from the last CIP tank will flow onto a 1.2 m wide x 1.8 m long safety screen to capture any carbon particles that may have escaped the CIP circuit. Captured carbon particles will be collected and transferred to the loaded carbon storage bin. The safety screen undersize will then be pumped to the cyanide destruction circuit.

Personal hydrogen cyanide gas detection devices will be provided to operators who work in the area and fixed hydrogen cyanide gas detection stations will also be provided in the area to monitor and alarm any higher than unsafe hydrogen cyanide gas level in the atmosphere.

17.3.10 ADR Plant

The Project will employ an alkaline, non-cyanide stripping/electrowinning process. The elution vessel and the electrowinning cell will operate in a closed loop. The circuit will operate at approximately 150°C under a pressured system of approximately 0.5 MPa. As expected, the elution system will be more efficient than a conventional elution process. The estimated elution retention time is approximately 12 hours or less.

The ADR circuit will be located within a secure and supervised area.

The loaded carbon from the loaded carbon storage bin will be transferred into a closed elution vessel. The stripping circuit will use sodium hydroxide as a stripping media. The barren solution conditioned with sodium hydroxide will be heated and circulated in the elution vessel. After the stripping solution is heated to approximately 110°C or higher, the pregnant solution from the elution vessel will be sent to the electrowinning circuit. The barren solution from the electrowinning circuit will be recycled back to the elution column in closed circuit with an electrowinning cell. The system temperature will be maintained at approximately 150°C using electrical heating. The stripping will finish in approximately 6 to 8 hours after the system reaches the designed temperature. Sampling outlets will be installed to collect representative pregnant and barren solutions for monitoring the elution and electrowinning circuit performance.

The electrowinning circuit will operate in a pressure vessel at a cell voltage of 2 to 4 V and a current density of 350 to 480 A. After the ADR system is shut down and the system pressure is reduced to atmospheric pressure, the gold-rich sludge will be washed from the steel cathodes and collected. The gold sludge will be dried and mixed with melting flux prior to melting in an electric furnace at approximately 1,200 to 1,300°C to produce gold doré. A similar process will be used to melt the gravity concentrate produced from the gravity concentration circuit. The gold doré will be stored in a safety vault within a secure and supervised area.

The gold sludge may be further treated by acid leaching to dissolve the heavy metals and silver that are plated onto cathodes together with the gold. The leach treatment will significantly improve the gold content of the gold doré. The wet treatment will also recover silver as metallic silver powder as a final product.

As required, the eluted carbon will be pumped to the carbon acid wash tank where diluted hydrochloric acid solution will be added to remove calcium and other inorganic impurities picked up during the CIP process. The carbon will initially be rinsed with fresh water. A diluted hydrochloric acid solution will be pumped from the acid washing circulation tank, upward through the acid washing vessel, and overflow back to the acid washing circulation tank. The carbon will then be rinsed with fresh water.

After the acid wash, the stripped carbon will be pumped back to the CIP circuit for reuse. However, the barren carbon will be periodically sent to a carbon thermal regeneration system for reactivation treatment. As required, new carbon will be added to the CIP circuit to supplement for the lost carbon. The new carbon will be treated by attrition and sizing treatment prior to being pumped to the CIP circuit.

Personal hydrogen cyanide gas detection devices and fixed hydrogen cyanide gas detection will also be provided in the ADR area to protect operators.

17.3.11 Carbon Reactivation

When the barren carbon loses its activity, the carbon will be transferred by a venturi pump to a carbon thermal regeneration system. The acid washed carbon will be pumped to a carbon surge tank located at the head of the carbon regeneration kiln. The carbon will be conveyed from the surge tank via a screw conveyor. The kiln, with a capacity of approximately 80 kg/h, will be heated by electricity and operated at approximately 650°C in an inert atmosphere. The hot and reactivated carbon will leave the kiln and be quenched in a conical bottomed quench tank flooded with water. The regenerated carbon will be sized and circulated back into the CIP circuit. The total installed power, including feeding, heating, and shell rotating systems, will be approximately 108 kW.

17.3.12 Leach Residue Cyanide Detoxification

The leach residue from the carbon safety screen in the CIP circuit will flow by gravity to a residual cyanide detoxification system where WAD cyanide will be destructed using the SO_2/air process. The circuit will consist of two, 5 m diameter x 5.5 m high mechanically-agitated tanks, each with a capacity to handle the full slurry flow for a retention time of approximately 75 minutes. The arrangement will provide sufficient detoxification capacity if one of the two tanks require unplanned maintenance. The reagents used will include hydrated lime, sodium metabisulphite, and copper sulphate. The reagent storage, preparation, and dosing systems for these reagents will be provided. After detoxification, the tailings slurry will be pumped to the TSF for storage.

17.3.13 Tailings Management

The cyanide detoxified leach residue will flow from the detoxification tank to a high-rate thickener. The residue will then be thickened to approximately 50 to 55% w/w solids. Diluted flocculant solution will be added to the thickener to assist the thickening process. The thickener underflow will be pumped to the TSF for subaqueous storage in a conventional TSF. Water from the TSF will be reclaimed by reclaim water pumps to the process tank located at the plant site as make-up water. Excessive water from the process water tank will overflow to a pond adjacent to the process water tank at the mill site. The tailings management is detailed in Section 18.8.

17.3.14 Reagents Handling and Storage

The reagents used in the process will include:

- Flotation – PAX, A208, and DF250.
- Cyanide Leaching and Gold Recovery – Hydrated lime ($Ca(OH)_2$), sodium-cyanide (NaCN), activated carbon, sodium hydroxide (NaOH), hydrochloric acid (HCl), nitric acid, and flux.
- Cyanide Destruction – Sodium metabisulphite ($Na_2S_2O_5$), copper sulphate ($CuSO_4$), and hydrated lime $Ca(OH)_2$.
- Others – flocculant and anti-scalant.

All reagents will be prepared in a separate reagent preparation and storage facility located in a containment area.

Liquid reagents, including A208, DF250, and anti-scalant, will be added in undiluted form at controlled rates via reagent addition devices and metering pumps. Other liquid reagents, including hydrochloride acid and nitric acid, will be diluted to approximately 10 to 20% prior to being added at controlled rates to the required process circuits via metering pumps.

All solid type reagents (calcium hydroxide, PAX, sodium cyanide, copper sulphate, and sodium metabisulphite) will be mixed with fresh water, in respective mixing tanks, to 10 to 25% solution strength and stored in separate holding tanks before being added to various addition points at controlled rates by reagent addition devices and metering pumps.

Cyanide monitoring/alarm systems will be installed at the cyanide preparation and leaching areas. Emergency medical stations and emergency cyanide detoxification chemicals will be provided at the areas as well.

Flocculant will be received in solid form and prepared in a packaged preparation system, including a screw feeder, a flocculant eductor, and mixing devices. The mixed solution will be transferred and stored in an agitated flocculant holding tank. Flocculant will be added at controlled rates via metering pumps to the leach feed thickeners and the leach residue thickener.

New activated carbon required to maintain the carbon inventory in the CIP circuit will be treated by attrition and sizing prior to being pumped to the CIP circuit.

The reagent preparation areas will be equipped with sump pumps to clean up any spillages and forced ventilation systems to control dusts and gases that may generate during reagent preparations.

17.3.15 Air Supply

Plant air service systems will supply air to the following areas:

- Flotation Circuit – Low-pressure air from two dedicated 45 kW blowers (one operating and one standby).
- Leach/Cyanide Detoxification Circuits – Low-pressure air from four dedicated blowers; two, 185 kW blowers (one operating and one standby) and two, 315 kW blowers (one operating and one standby).
- Crushing Circuit – High-pressure air from an air compressor for the dust suppression system and other services.
- Plant Services – High-pressure air from two dedicated air compressors for various services.
- Instrumentation Services – Instrument air from plant air compressors will be dried and stored in a dedicated air receiver.

17.3.16 Water Supply and Consumption

Three separate water supply systems will be provided to support the process operations: one fresh water system and two process water systems for various process circuits.

- General Site Process Water – Water reclaimed from the TSF and the water recovered from the tailings leach feed thickener and leach residue thickener overflows will be used as overall site process make-up water. The water will be pumped or flow by gravity to the process water tank located adjacent to the storm water detention pond, which will collect the site run-off water and the process water tank overflow. The water in the pond will be pumped back to the process tank as required or will be sand filtered and discharged into the environment. In dry seasons, approximately 50% of the water reporting to the TSF is expected to be reclaimed and reused in the process plant
- Gravity Separation Circuit Water – This water will be pumped through a pipe filter prior to being used as fluidizing water and back washing water in the centrifugal concentrators.
- Grinding and Flotation Process Water – Overflow water from the concentrate thickener will flow by gravity to a process water tank. The water will be used as process water in grinding, gravity separation, and flotation circuits with supplemental water from the general site process water tank.
- Fresh Water – Fresh water for the process plant will be pumped from the storm water detention pond through a water treatment unit or from wells to the fresh water tank from where the water will be distributed in the mill by the fresh water distribution pumps. Fresh water will be used as reagent preparation water, gland seal water, process make-up water, and cooling water services in the strip circuit. The estimated process plant fresh water consumption will be approximately 8 m^3/h in wet seasons and 15 m^3/h in dry seasons.

17.4 Geochemical and Metallurgical Laboratory

A geochemical and metallurgical laboratory has been constructed at the Nadi office site to assay samples from exploration activities. The laboratory is equipped with necessary sample preparation equipment and analytical instruments (atomic absorption and Inductively coupled plasma – optical emission spectrometers), which can be used to provide routine assays for exploration, mine, process, and environmental departments. The assay laboratory has functioned to provide the assays required for geological exploration, including gold fire assay. The metallurgical units are also planned, and some metallurgical test devices, including one flotation machine and leaching cells, have been installed.

As planned, the laboratory will be upgraded in the future to provide routine assays and metallurgical testing to support the planned operation. The data obtained from the assay and testing will be used for routine process optimization for metallurgical performance improvement and metallurgical balance accounting.

The samples from various process streams will be manually collected and assayed for daily metallurgical balance and process optimization.

17.5 Process Control and Instrumentation

The plant control system will consist of a DCS with PC-based OIS located in the plant-site control room. The plant-site control room will be staffed by trained personnel 24 h/d.

Jinpeng proposed an overall site process control system, which is illustrated in Figure 17-3. The objectives of the process control system are to reduce the labor requirement and improve process control and production efficiency to provide stable operation.

Operator workstations will be capable of monitoring the key process operations in the plant site and viewing alarms controlling key equipment within the plant.

17.5.1 Crushing Circuit

A central control room will be used to operate start, stop, and start-stop sequences for equipment in the crushing circuit. A radar level detector will be installed in the ROM dump bin and in the fine mill feed surge bin, which will be used for monitoring and alarming when the materials in the bins are out of the set values.

17.5.2 Grinding Circuit

The programmable logic controller (PLC) system will monitor and control the ball mill’s feed rate, water supply, slurry level of the hydrocyclone feed pump box, and feed pressure of the hydrocyclone system.

The system will control operation of the surge bin reclaim feeders by monitoring feed rates via the belt scale and controlling the conveyor feed rate by adjusting the vibrating frequency of the feeders.

The water supply will be controlled via electromagnetic flowmeters, which will detect the flow rates of water supply to ensure the optimum grinding solids density. The water supply quantity will be controlled by adjusting electrically operated valves.

The hydrocyclone overflow solids density will be automatically controlled by adjusting the dilution water to the hydrocyclone feed pump box to ensure the solids density of the hydrocyclone is meeting the flotation requirement.

Slurry levels of the hydrocyclone feed pump box will be controlled by recycling a portion of the hydrocyclone overflow back to the pump box. This control is expected to mitigate any hydrocyclone pump box slurry levels that are out of the range of the set values, which may cause pump air cavitation and spillage due to overflow from the pump box.

The pressure of the hydrocyclone feed system will be monitored and controlled to ensure the most stable feed pressure. With monitoring feed pressure of the hydrocyclone feed system, the control system will be able to adjust the frequency of the slurry pump’s frequency converter to maintain a relatively constant slurry feed pressure. An online particle size analyzer will be installed to monitor the hydrocylone overflow particle size distribution.

17.5.3 Flotation and Regrinding

The slurry levels of the flotation cells will be monitored using ultrasonic slurry level devices. The output data of the level detectors will feed to the flotation cell discharge gate control devices to automatically control the slurry levels.

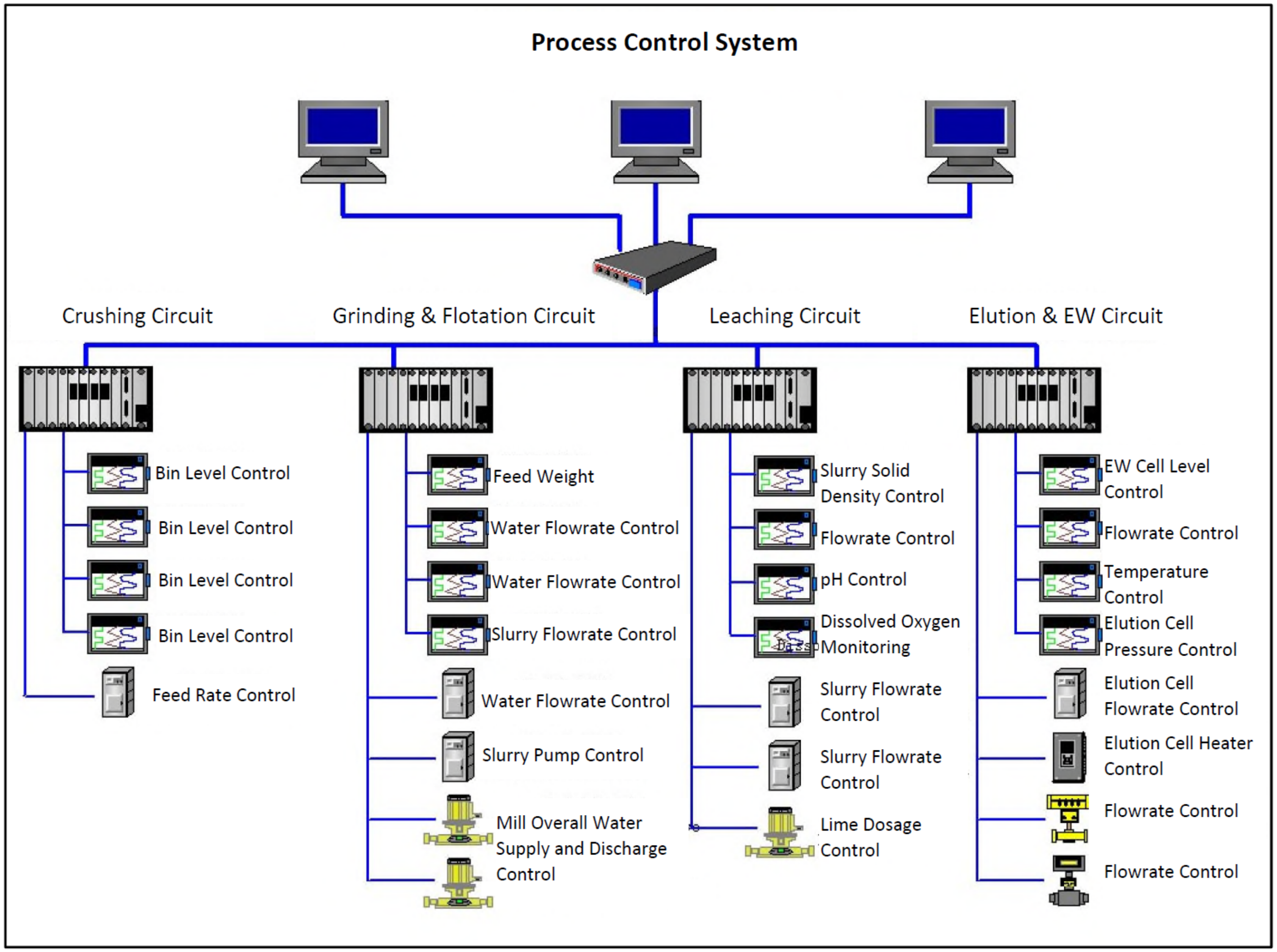
The feed solid density of the hydrocyclones in the regrinding circuit will be monitored. An electrically controlled valve interlocked with an electromagnetic flowmeter will control quantity of the dilution water added. The particle size distribution of the regrinding hydrocylone overflow will be monitored by the particle size analyzer, which is in the primary grinding circuit.

17.5.4 Thickening and Cyanide Leaching

The underflows of the flotation concentrate and flotation tailings thickeners will be equipped with nuclear densimeters and electromagnetic flowmeters, respectively. The meters will be used to automatically control the slurry solid densities and flow rates of the thickener underflows to the cyanide leach circuits.

Slurry pH in the leaching circuit will be monitored using pH meters. The signals from the pH meters will feed to electrically controlled pinch valves to control quantities of lime milk slurry added to the leach circuit.

The dissolved oxygen concentrations in the leach tanks will be automatically monitored.



Source: Tetra Tech After Jinpeng (2020)

Figure 17-3: Overall process plant control system (after Jinpeng)

17.5.5 Loaded Carbon Elution and Gold Recovery by Electrowinning

The elution and electrowinning system will work at approximately 150°C under pressure. An automatic control system has been designed to allow the system to operate automatically and be controlled from the center control room. The valves used in the system will be pneumatically operated ball valves. The key monitoring and control items include system pressure, temperature, electrowinning cell liquid level, and elution solution flow rate. As designed, the elution and electrowinning system have an extra capacity to handle a high-grade mill feed and a higher mill feed rate.

17.5.6 Closed-circuit Television Monitoring System

In addition to the plant control system, a high definition closed-circuit television system will be installed at various locations throughout the process plant site, including the crushing facility, the grinding facility, the gravity concentration area, and the leach and gold recovery and refining facilities. Cameras will be monitored from the central control room.

17.6 Yearly Metallurgical Performance Projection

According to the test work results described in Section 13.0 and the proposed mine production schedule, gold recoveries to doré are projected on a yearly basis and shown in Table 17-2. Further metallurgical test work is recommended to better estimate and improve metallurgical performance.

Table 17-2: Yearly gold production projection

Description	Units	Y1	Y2	Y3	Y4	Y5	Totals
Total Mill Feed	t	329,960	330,864	329,960	329,960	63,323	1,384,067
Mill Feed Gold Grade	g/t	7.60	8.93	10.49	7.70	6.42	8.57
Gold Milled	oz	80,588	94,985	111,238	81,643	13,078	381,532
Gold Recovery	%	86.5	87.5	87.5	87.5	87.5	87.3
Gold Recovered	oz	69,708	83,112	97,334	71,438	11,443	333,035

18.0 PROJECT INFRASTRUCTURE

18.1 Tuvatu Site Description

The Project is located 17 km by road from Nadi International Airport. The region is well serviced with port facilities at Ba and Lautoka. Lion One maintains an operations office in Nadi, including a geochemical and metallurgical laboratory to service site operations.

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

The core storage facility and associated infrastructure are maintained on site to service exploration activities. Emperor Gold Mining Company Limited developed a decline in 1997; however, the current mine development design proposes using an additional new access to the west as the primary access.

EFL's transmission line crosses the site, but no dependable surplus power is available in the grid for use by the Project.

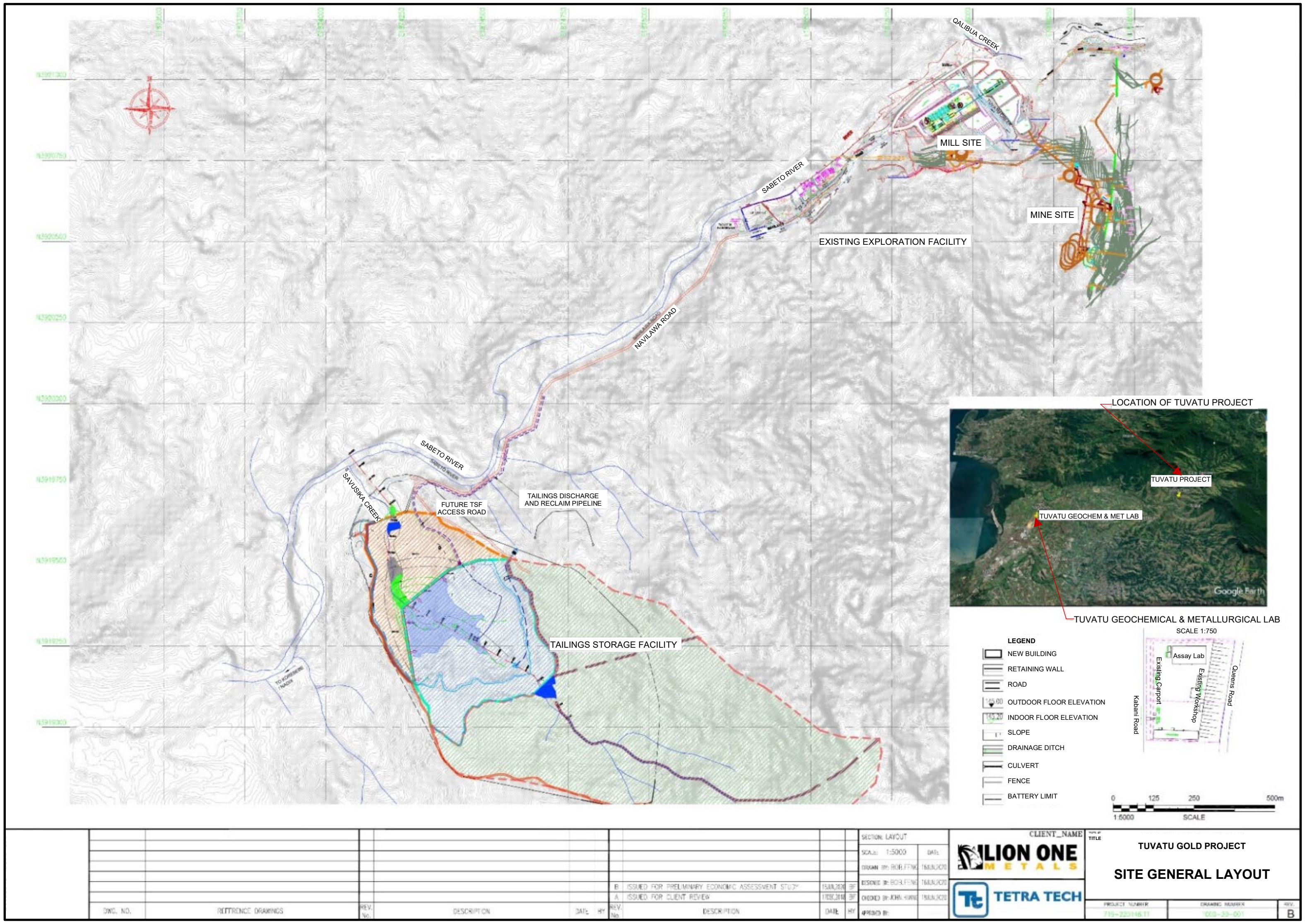
18.2 Site Development

Primary site development will consist of multiple platform construction to minimize cut-and-fill activities due to building/equipment foundation considerations and the steepness of the terrain. Process facilities, including the crushing plant, will be located on separate platforms as will the ROM stockpiles and ancillary facilities, such as storm water detention pond, power generation system, mine truck shop, and mine dry. There is an existing exploration decline that will be enlarged and reused for the mining operation. The overall site general arrangement is shown in Figure 18-1.

The proposed process plant is planned to be located northwest of the existing exploration decline. It is proposed to construct most of the process plant facilities in cut area, in an effort to establish the concrete foundations on competent soils. Numerous boulders on the site will generate a significant portion of the bulk aggregate for construction activities. The larger boulders will be crushed for additional rock fill across weak foundation areas. Soil within the final TSF impoundment footprint will be used to provide materials for the embankment construction. Removal of these soils from the TSF area will create more tailings storage capacity at the same time.

The mined material from the proposed mine pre-production will be stockpiled on ROM pads. The extracted waste rock expected to be non-acid generating from the mineralized material extraction will be crushed and used as road subbase at plant site and underground development areas and retaining walls throughout the operational area. The surplus waste rock is also expected to be sold to quarry operators in the area.

Site preparation, including ground clearing and grubbing, stormwater drainage, sediment control structures, and bulk earthworks, to develop multiple platforms, will be constructed prior to the start of process plant construction and mine development. Slope stability and site water management will be key considerations in the Project design. Lion One has conducted preliminary site preparation, including diverting some water courses that may impact future construction of facilities. The site preparation has been initiated as shown in Figure 18-2. The process plant site general arrangement is shown in Figure 18-3.

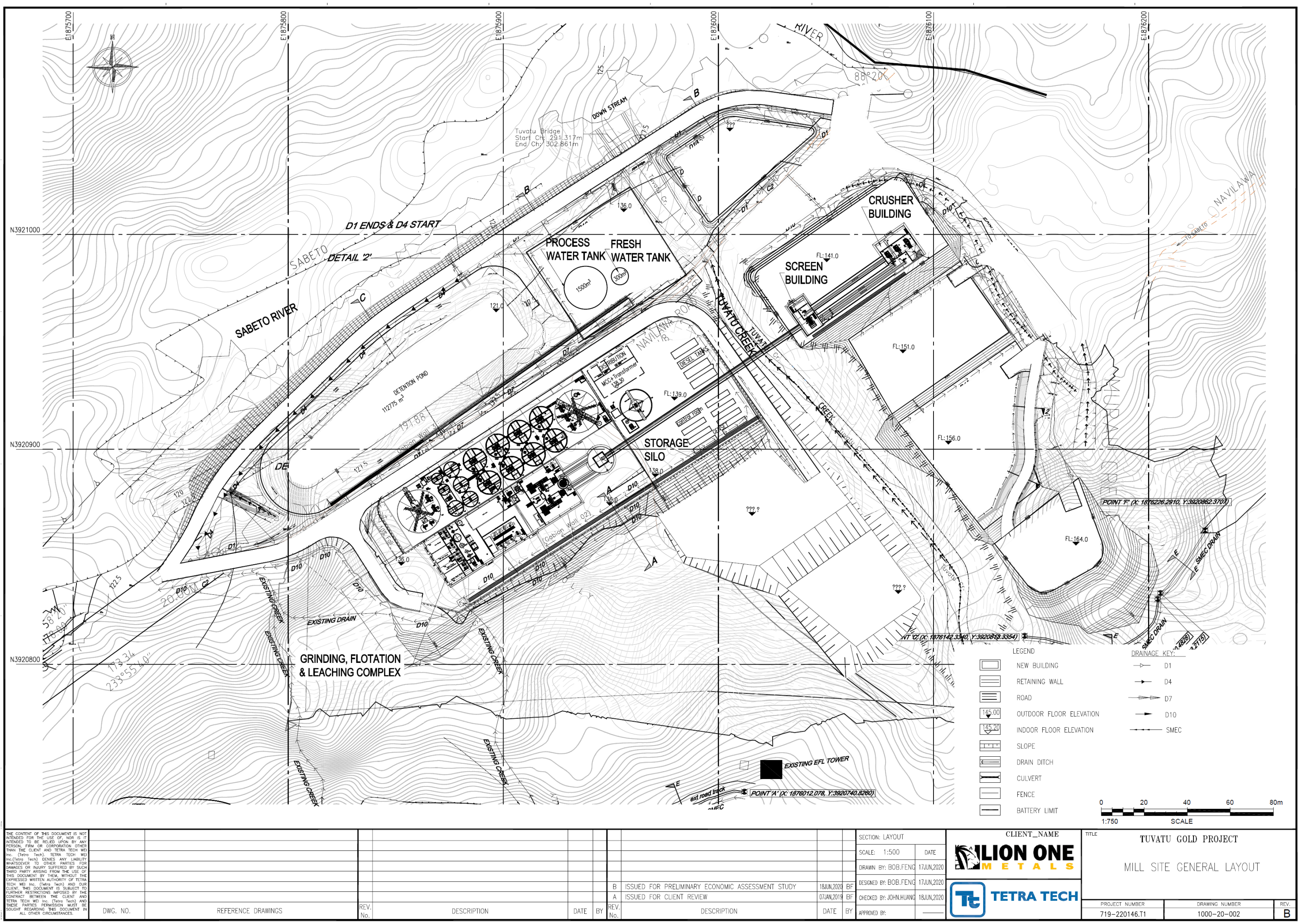


Source: Tetra Tech (2020)

Figure 18-1: Overall Project site – plan view

Source: Lion One (2020)

Figure 18-2: Overview of preliminary site grading at the plant site



Source: Tetra Tech (2020)

Figure 18-3: General plant site layout – plan view

18.3 Site Geotechnical Investigations

18.3.1 Process Plant Site

As shown in Figure 18-4, the process plant and adjacent structures are proposed for construction near the junction of Tuvatu Creek and Sabeto River. The existing ground elevation in the areas of the process plant building, primary crusher, screens, and fine ore bin, varies between 134 and 150 m above sea level. With the consideration of the significant elevation difference across the site, several platforms are proposed to accommodate the process plant building and the crusher structures involving cut-and-fill earthworks with engineered perimeter slopes and retaining walls exceeding, at some locations, 10 m in height. It is assumed that the run-off water and groundwater seepage within and around these structures will be controlled and collected by an engineered drainage system and conveyed to the nearby creeks and the Sabeto River. Lion One has conducted site preparation work for the process plant complex, and the areas are currently under active development. Figure 18-4 to Figure 18-6 show the clearing, grubbing, and earthwork site preparation activities across the site.

Source: Wood (2018)

Figure 18-4: Satellite image – proposed process plant / crusher areas

Source: Wood (2018)

Figure 18-5: Site preparation – process plant area – August 2018

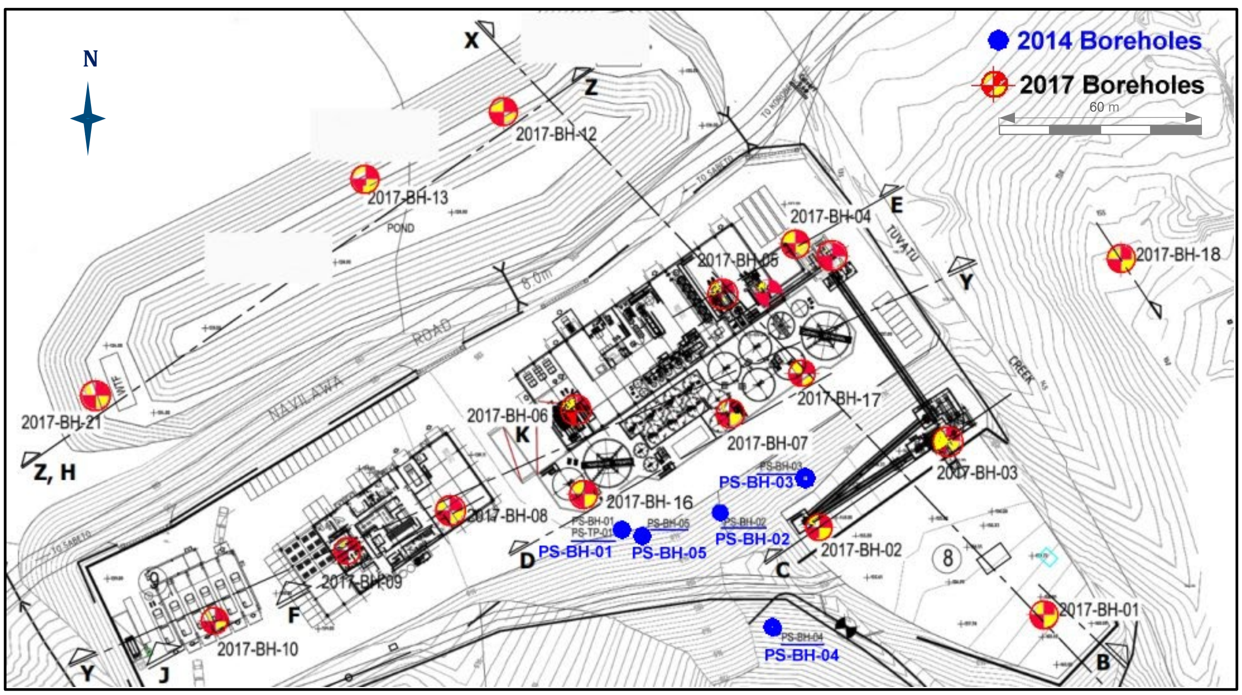
Source: Wood (2018)

Figure 18-6: Site preparation – crushing/screening plant area – August 2018

The following is a brief summary of the ground conditions, described by various site investigators (other than Wood) at the site of the proposed process plant, including crusher and screen structures. Details of the said conditions are included in the individual relevant reports.

Entec has carried out a test pit investigation program first, in August 2014, in the general area of the then proposed locations of the process plant and adjacent structures. The investigation included the excavation of 15 test pits, and the completion of 14 shear vane tests and 25 dynamic cone penetration tests, carried out near and between the test pits. The test pits were advanced by a backhoe, to various depths of between 0.8 and 5.2 m below ground surface, while the cone penetration tests had been advanced to depths of between 0.25 and 2.70 m below grade only. All trial pits were terminated due to refusal of the backhoe bucket on boulders, except TP12, which reached its maximum possible depth with the available backhoe. According to the relevant test pit logs, the overburden across the process plant site generally comprises soft to very soft silt and clayey silt soils near the surface, changing to firm to stiff clayey silt, and generally loose to compact sandy silt / silty sand with random cobbles and boulders (colluvial deposit) with depth. No groundwater seepage was encountered in the test pits, excavated during a period of dry and sunny weather.

Subsequent to the fifteen test pits, five boreholes were drilled in October 2014 near the then proposed process plant footprint, as reported by Knight Piésold Ltd. The locations of the five boreholes (PS-BH-01 to PS-BH-05) are shown in Figure 18-7 (blue dots).

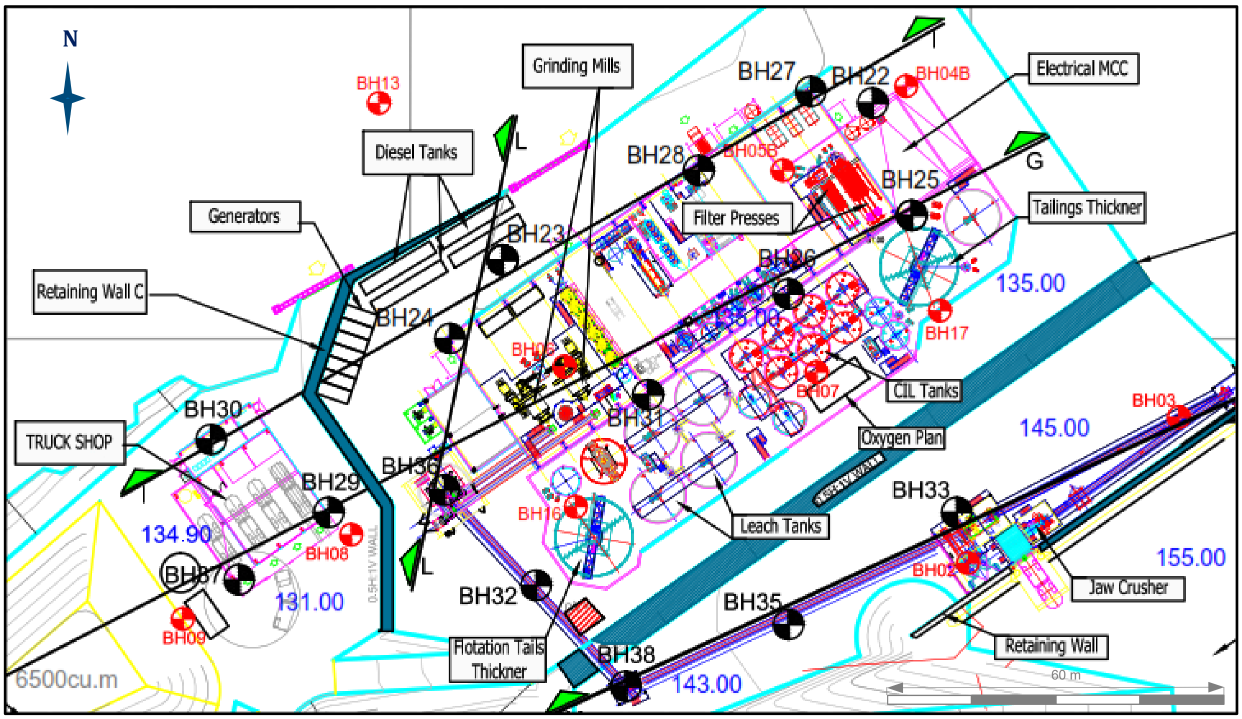


Source: Wood (2020)

Figure 18-7: Historical borehole locations across the plant area

The findings in the five boreholes were consistent with the overburden materials encountered in the earlier test pits, generally comprising clayey silt, sandy silt, and silty sand, with random cobbles and boulders. Entec has completed a follow-up geotechnical investigation in two stages in 2017. Eighteen boreholes were drilled initially between February 28 and March 23, 2017, at the locations marked with red/yellow circles in Figure 18-7 and Figure 18-8. Eight of the 18 boreholes (BH-04 to BH-09, BH-16, and BH-17) were drilled within the area of the then proposed process plant. Loss of soils during drilling and sampling had been reported in the 2017 geotechnical report, probably related to weak soil condition and poor sampling technique since the drilling and sampling program in 2017 was carried out with the use of an exploration rig and not by a traditional geotechnical drill rig.

During the second stage of the program between May 29 and June 15, 2017, 20 additional boreholes were drilled at the locations marked by black/white circles in Figure 18-7. Thirteen (boreholes BH-22 to BH-32, BH-36, and BH-37) of the 20 boreholes were drilled within the area of the process plant footprint.



Source: Wood (2020)

Figure 18-8: Borehole locations at the process plant area

The above summarized previous geotechnical investigations had confirmed that the areas of the proposed process plant and adjacent structures are underlain with heterogeneous mix of relatively weak soils, comprising random cobbles and boulders with various sizes. To augment the information about the ground conditions across the site, particularly for the areas where structural foundations are proposed within the process plant complex, large tanks, and heavy machines, a geophysical testing program was carried out in the spring of 2018. During the field program, GBG Maps of Australia had carried out the seismic geophysical testing program along seven strategically selected transects (lines/alignments), shown in Table 18-1.

Table 18-1: Coordinates of the transect lines in the process plant and crusher areas

Line	Site	Start		End		Length (m)
		Easting	Northing	Easting	Northing	
1	Processing Plant	1875942	3920844	1875896	3920950	115
2	Processing Plant	1876030	3920832	1875946	3920965	160
3	Processing Plant	1876052	3920938	1875994	3921006	90
4	Processing Plant	1876077	3921040	1876017	3920957	105
5	Processing Plant	1876008	3920941	1875800	3920836	235
6	Processing Plant	1876008	3920912	1875805	3920813	230
7	Processing Plant	1876080	3920985	1875987	3921041	115

Source: GBG Maps

Detailed results of the geophysical testing program were provided in a report prepared by GBG Australia in 2018. The geophysical survey program has confirmed that the competent load bearing rock surface across the project site slopes sharply toward the Sabeto River, and toward Tuvatu Creek. In addition, the sloping rock is generally weathered to various degrees, and is covered with heterogeneous overburden of varying thickness, comprising cobbles and boulders, embedded in a generally weak clayey and sandy silt soil matrix.

Qingdao Geotechnical Investigation and Surveying Research Institute (Qingdao), China, has carried out the most recent geotechnical investigation program in the general area of the currently proposed locations of the process plant and adjacent structures (crusher and screen) between July 15 and August 10, 2018. The drilling program was carried out by Derwent Geoscience and Geodrill Ltd. (Fiji), while the laboratory tests on the retrieved samples were carried out by Entec.

Seven boreholes were drilled within the area of the proposed crusher, screen, and connecting conveyors, where the geophysical survey program indicated particularly weak soils to considerable depth. The boreholes indicated that the thickness of the weak overburden was increasing not only toward the Sabeto River, but toward the Tuvatu Creek as well, as reported earlier in the geophysical survey report.

Based on the results of previous geotechnical investigations, it appears that the bedrock surface beneath parts of the proposed process plant complex is located at relatively shallow depth; however, the rock surface slopes sharply toward the Sabeto River, and toward the west-end of the process plant, and also toward the Tuvatu and Murau Creeks. At the proposed screening facility, for example, no bedrock was encountered within the investigated depth of 20 m below grade, and the thick overburden comprised very weak soils. Based on limited standard penetration tests and on recent shear wave velocity readings, it is evident that the soil matrix is generally weak and compressible to varying depths below the floor levels of the proposed structures within the entire process plant complex.

Prior to the construction of the main process plant and adjacent structures, all weak soils will need to be densified by an acceptable compaction method (e.g., dynamic deep compaction, rapid impact compaction), or replaced with well compacted engineered fill to safely support conventional foundation members, and to prevent excessive settlements, particularly differential settlements. Such weak soils were encountered beneath the proposed conveyor belts, the screening facility and beneath a large portion of the main process plant complex, particularly at the thickeners and beneath the various storage tanks, located closer to the Sabeto River. The results of the geophysical survey program, together with the historic borehole results, should be used to define the extent of soil improvement or soil removal areas during the feasibility and detailed design stages of the development, and the size (thickness and horizontal extent) of any engineered fill for the structures and foundation members individually.

In addition to the weak soil zones within the overburden, the future foundation design must also take into consideration the potential presence of several apparent fault lines in the area, running beneath the proposed structures along and perpendicular to the Sabeto River. Details (locations and directions) of the apparent local fault lines are visible on the compressional wave velocity profiles, presented in the geophysical report.

In summary, it is evident that the bearing capacity for foundation design is highly variable across the site both horizontally and in depth. For detailed foundation design, it will be necessary during the feasibility stage of the development to review the actual ground conditions beneath each individual section of the proposed structures and determine the most probable ground profiles and relevant soil and rock properties. With these data, the most appropriate ground improvement technique and engineered fill alternative can be selected for each section of the process plant complex.

18.3.2 Tailings Storage Facility

Various TSF site investigations were carried out in 2015 and most recently in 2018, including geotechnical drilling, test-pitting, and geophysical survey programs. Additionally, groundwater level and water quality monitoring well installations, as well as laboratory test work were carried out in July, August, and September 2018 in support of the TSF and associated WMF designs. Limited information is available on the geochemical characteristics of the tailings and the proposed dam construction materials. Therefore, further geochemical characterization testing is required to assess risks and opportunities related to the ML/ARD potentials of these materials and potential future water quality management needs for the TSF.

The site-specific geotechnical and hydrogeological data gathered over the years have been utilized for the TSF designs.

Further details are shown in Section 18.8.

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 Queen’s Road, which is the primary access from Nadi International Airport. From Queen’s Road junction to Nagado Junction (approximately 9 km from the proposed Project site), the majority of the road is paved and in good condition. The section of road from Nagado Junction to the proposed Project site is a public road with a gravel covered surface. Although an all-weather access, this section of the road is narrow and in relatively poor condition due to lack of regular maintenance. Lion One negotiated with Fiji Road Authority (FRA) to share the costs of upgrading the road and bridges to allow heavy freight to be transported to the proposed Project site. Lion One has completed reconstruction of the Nubuyagiyagi Bridge (located approximately 3 km from the proposed Project site), and FRA is currently upgrading sections of this existing road. Lion One has also completed the relocation of the main road past the proposed process plant and the construction of a new bridge at where the new road crosses over the Tuvatu Creek.

18.4.2 Site Internal Roads

The current section of the public road that runs through the mine site has been rerouted closer to the Sabeto River to eliminate public access to the site. A new bridge over Tuvatu Creek was constructed to accommodate the rerouted road.

Haul roads will be constructed to service the main portal, the ventilation portal, and the exploration portal sites. With the proximity of the portals 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. Service roads will also be constructed to access the main diversion ditches above the process plant site and mine portals.

A new road will also be required to link the existing access road to the tailings dam embankment (this includes improvements to off-site roads, including the run-off diversions around the tailings dam as discussed in Section 18.8). This road will be used for tailings dam construction, operations, and maintenance.

18.5 Hydrological Study

In 2017, Lion One retained SMEC, a member of the Surbana Jurong Group, in association with Entec, to undertake hydrologic investigations and drainage infrastructure design for the Tuvatu Gold Mine in western Vitu Levu, Fiji. This subsection is a brief, conclusive summary based on the plant site arrangement upgraded in 2018 and SMEC's September 2017 report.

18.5.1 Hydrology

18.5.1.1 Climate

Fiji has a warm tropical climate with maximum daily temperatures ranging from 28 to 32°C. The minimum daily temperature ranges from 18 to 23°C.

Mean monthly evaporation varies from a low of 130 mm in June to a maximum of 210 mm in December. Rainfall exceeds evaporation in the wet months from December to April, but evaporation exceeds rainfall in the dry months.

18.5.1.2 Precipitation

Rainfall adjacent to the Project site is highly seasonal with January to March contributing approximately 60% of the annual rainfall. There is a significant rainfall gradient near the mine site with a mean annual rainfall of 1,900 mm at Nadi (near the coast approximately 15 km from the mine) and mean annual rainfalls of approximately 3,000 mm in the headwaters of the Sabeto River, upstream of the mine site.

The rainfall intensities provided by FMS for Nadi are summarized in Table 18-2. The intensities are available for various durations and for a range of different return periods.

Table 18-2: Rainfall intensity duration frequency data for Nadi

Duration	Return Period (years)							
	2	5	10	20	50	75	100	150
10 minutes	20.2	24.7	27.6	30.5	34.2	35.8	36.9	38.5
20 minutes	31.8	40.2	45.7	51.0	57.8	60.8	62.9	65.9
30 minutes	41.4	51.1	57.6	63.8	71.8	75.3	77.7	81.2
60 minutes	57.9	71.5	80.6	89.2	100.4	105.4	108.8	113.7
2 hours	71.3	88.6	100.0	111.0	125.1	131.4	135.8	142.0
6 hours	96.3	127.2	147.7	167.3	192.8	203.9	211.8	223.0
12 hours	116.6	170.4	206.0	240.2	284.4	303.8	317.6	336.9
24 hours	148.5	220.0	267.3	312.7	371.5	397.3	415.6	441.3
48 hours	148.6	220.1	267.3	312.7	371.4	397.2	415.4	441.0
72 hours	213.5	308.1	370.7	430.8	508.5	542.7	566.8	600.8

Hourly stream flow gauging data is available for a 13-year period from September 1978 to November 1991, for a station located at Masimasi on the Sabeto River, approximately 10 km downstream of the Project.

A flood frequency analysis was undertaken on the stream flow data to derive estimates for design flows. The results are provided in Table 18-3. Flows at the gauging station were adjusted to provide estimates of flows at the mine site, taking into account the mean annual rainfall and catchment area.

The design flow estimates from flood frequency analysis are not expected to be reliable due to the short duration of record and the long periods of missing data.

Table 18-3: Flood frequency analysis for Sabeto River

Average Recurrence Interval (years)	Masimasi Gauge Station (m^3/s)	Mine Site (m^3/s)
2	160	105
5	335	220
10	500	325
20	710	460
50	1,080	700
75	1,290	835
100	1,460	945

18.5.2 Analysis and Design

18.5.2.1 Diversions

Diversions are required upstream of the process plant area to direct hillside runoff around the process plant building platform. The 6.3 ha catchment upstream of the process plant has a dense grass land cover. It is steep with an average slope of approximately 40%, or 1 vertical to 2.5 horizontal. The upper portion has slopes of 100%, or 1 vertical to 1 horizontal.

Diversions are typically designed to a 100-year ARI storm event but given the short design LOM (approximately six years), there is justification for adopting a lower design criterion. In this case, a 20-year ARI design has been adopted.

Two diversions are proposed upstream of the process plant. The diversions will run approximately parallel to the ground level contours, but at a gradient of 1%.

18.5.2.2 Access Road Culverts

The culverts for two waterways (Tuvatu Creek and the Western Drain) that cross the mine access road adjacent to the process plant have been designed. Tuvatu Creek has a catchment area of 82 ha and Western Drain has a catchment area of 6.4 ha. Both drains have very steep catchments with average slopes of 35 and 40%, respectively.

18.5.2.3 Flow Estimates – Tuvatu Creek

A rainfall runoff model was established to determine design discharges for Tuvatu Creek utilizing the software package XP-RAFTS, which has been widely applied throughout Australia and Asia.

Two options were considered for the Tuvatu Creek crossing, a single span bridge or a series of box culverts. It was assumed that the road crest level was 2.5 m above the creek invert and that a freeboard of 300 mm was required. Should a higher road elevation be adopted, allowing a greater bridge or culvert depth, then the span can be adjusted to provide the flow areas.

18.5.2.4 Flow Estimates – Western Drain

A rainfall runoff model was established to determine design discharges for the Western Drain utilizing the software package XP-RAFTS. The design discharge for the 100-year ARI event is 7.0 m^3/s, with the critical storm duration of 20 minutes.

18.5.2.5 Flood Assessment

In addition to the flow data estimated through regional analysis, a XP-RAFTS software rainfall runoff model was established to determine design discharges for Sabeto River. The XP-RAFTS model was run for a range of storm durations to determine the critical storm duration.

The design flow estimates from rainfall runoff modelling are significantly higher than those from flood frequency analysis but are more reliable as they are derived from a much longer data set. The rainfall runoff model design flows were adopted for the Project.

18.5.2.6 Preliminary Hydraulic Modelling

A preliminary hydraulic model was established using HEC-RAS modelling software to determine the flood levels for channels adjacent to the site. Roughness values implemented in the model were based on site inspection. A value of 0.045 was used for the main channels and a value of 0.15 was used for the overbank areas.

The analysis showed that:

- The proposed finished levels for the process plant area are expected to be above the 100-year ARI flood levels in Sabeto River.
- According to the current site layout design, the Platform for Process Plant along the Tuvatu Creek will be located at 139 m level. The new crusher and screening plant pad at the west side of the Tuvatu Creek is at 141 m, and the location of the previous crusher pad Platform 2 will be at 148 m level. With the design, the process plant is expected not to be inundated by Tuvatu Creek at the 100-year flood.
- The existing exploration facility, which is located approximately 500 m downstream of the process plant, has an existing ground elevation between 119.00 and 120.5 m and may be inundated by Sabeto Creek in the 20-year ARI flood event.

18.6 Water Supply

Reclaim water, run-off water, and mine dewatering will supply the Project site water. In extremely dry conditions, additional water may need to be drawn from the Sabeto River. The possibility of drawing water from the Sabeto River in very dry periods was documented in the 2013 approved EIA.

18.6.1 Raw Water

Currently, mine dewatering is discharged to three settling ponds located near the exploration decline portal. The decanted water from the settling ponds is released to the Sabeto River. For the process plant, the mine water will be rerouted by pumping from the three settling ponds to the raw water storage tank adjacent to the storm water detention pond. This will be the primary source of make-up water for the process plant. Excess water from the raw water tank will be released to the Sabeto River. During dry season, the storm water detention pond will be used as additional raw water storage.

Historical studies were completed on the abstraction of water from Qalibua Stream and Sabeto River. These may be additional raw water sources in case of extreme dry conditions.

18.6.2 Reclaim Water

Supernatant water will be reclaimed from the TSF for reuse as primary process water. During the dry season, run-off water from the diversion ditches above the TSF will be redirected to the TSF and used to augment the reclaim water back to the process plant. The diversion ditches will be returned to normal operation during the remainder of the year.

18.6.3 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 new larger, potable WTP with storage tank will be installed on site near the mine dry. Potable water will be distributed to points of use within the process plant and the main buildings, mine dry, and safety showers.

18.6.4 Sewage

Treated effluent, via a low maintenance, biological-contactor-packaged wastewater treatment plant, will discharge to the plant tailings thickener. As a less expensive alternative, the use of a septic field near the mine dry will be investigated.

18.7 Site Water Management

In general, all divertible fresh water and runoff will be routed around various development areas and facilities to minimize collection and treatment. All contact water will be collected, conditioned, or treated and then released into the environment, as per water management design criteria and upon meeting the release criteria; otherwise the contact water will be retained and reused.

18.7.1 Tailings Storage Facility Water Management

The TSF will have a surface water diversion system upstream of the basin area comprising a surface water diversion pond and high-density polyethylene (HDPE) lined diversion channels to reduce surface water runoff into the tailings storage basin area. The TSF and its associated water management system will be designed as a “collect, monitor, return, condition, and release” system. Over the LOM, supernatant water will be primarily reclaimed back to the mill for reuse in the process under normal climatic conditions. The TSF is designed to retain to up to a 1-in-100-year, 24-hour duration storm event. Under more severe precipitations, excess water will be routed out of the TSF through an emergency spillway. All contact water and potential seepage emanating from the TSF will be collected in the downstream Sediment Control and Monitoring Pond, whereby the water will be clarified of sediment and treated if necessary. Water quality will be systematically monitored and water discharged into the environment upon meeting the release criteria. Otherwise, the collected water will be pumped back to the TSF pond and reclaimed back to the mill for process purposes via a barge-mounted pumping system.

To assess water management requirements given the high rainfall expected at the site and the requirement for full pond coverage of the tailings to mitigate potential acid generation, a site water management model (water balance) was developed for the Project. This assessment was carried out for the first two years of production, as Year 2 is considered representative of the later years of mine operations, based on constant process flows from Year 2 onwards. The water balance incorporates the following aspects:

- TSF rainfall runoff and evaporation
- Predicted tailings supernatant reuse and achieved densities from subaerial and subaqueous deposition
- Tailings porewater storage
- Process plant site water demand
- External storm water runoff from the upstream catchment

Under average climatic conditions with an upstream diversion system in place, the TSF water inventory can be maintained in balance during the LOM. The reclaim return will be sufficient to supply the water demand of the process plant. Therefore, an external water supply is not required. During dry climatic conditions, mill reclaim will be augmented by water collected in the upstream diversion system.

The supernatant pond will cover the deposited tailings over the LOM; therefore, lower tailings density is expected and ponding against the embankment will occur periodically and seasonally.

Under extreme wet conditions that exceed the design criteria, excess water will be routed through the emergency spillway to prevent dam overtopping.

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 unsuitable soils and cut-off trench across the base of the valley, a cofferdam will be constructed that will divert surface water runoff into the diversion system, which will discharge around the embankment.

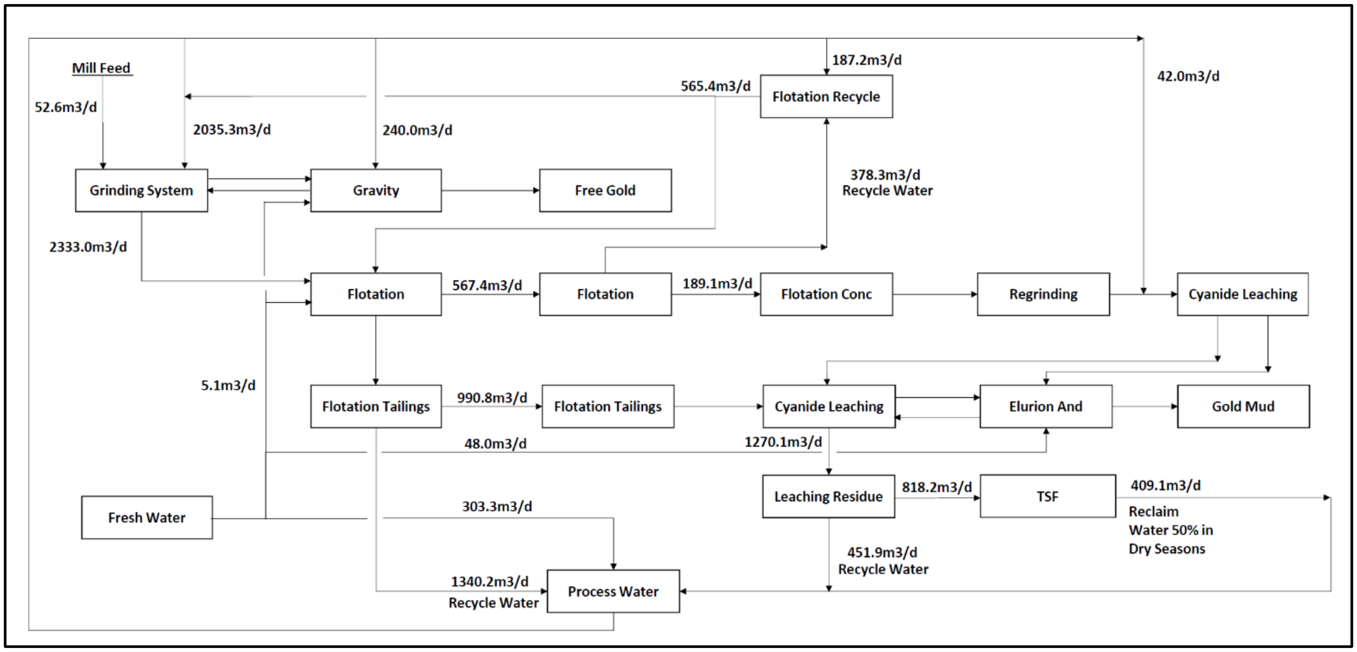
A sediment control structure (SCS), which will become a seepage collection and water quality monitoring pond (SCRMPD) during operation, will be constructed in the valley downstream of the TSF embankment, with a 60 m environmental buffer space to the mine lease boundary at the local public road, to reduce 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 go to the environment downstream if it meets water quality discharge criteria; otherwise it will be pumped back to the TSF. Water collected in the SCS may be used for dust suppression (if suitable).

18.7.2 Process Plant and ROM Water Management

Surface water runoff from the process plant site and ROM pad will be collected in the storm water detention pond to remove sediment. Decanted runoff will be pumped back to the process plant and used for process make-up water requirements and dust suppression (as required) or released upon meeting the release criteria.

Mine water will be routed out of the underground workings and collected and used for make-up water in the mill.

A flow diagram block model has been developed and is shown in Figure 18-9.



Source: Tetra Tech after Jinpeng (2020)

Figure 18-9: Process water balance block model

18.8 Tailings Storage Facility

18.8.1 Design Basis and Criteria

The TSF and associated structure design basis and criteria for the Project are based on the *Guidelines on Tailings Dams* (May 2012) established by the ANCOLD and adopted by the Fiji Government. Since Lion One is a Canadian incorporated and listed mining company, CDA (2014) standards are also used towards developing a stringent facility.

Other applicable environmental, health, and safety regulations are also considered.

18.8.2 Mine Life

The operating life of the mine is expected to be 7 to 10 years at a nominal throughput of 1,000 t/d. Ten years has been used to size the TSF storage capacity. Operation is assumed to be nominally 24 h/d, 365 d/a. The Starter TSF and WMF Dam construction period is assumed to be 12 to 18 months during both the dry and wet seasons. The yearly dam raises are expected to take place mostly in the dry seasons.

18.8.3 Climatic Conditions

The mean average local precipitation and evaporation data are shown in Table 18-4.

Table 18-4: Mean average local precipitation and evaporation data

Precipitation/Evaporation	Value (mm)
Mean Annual Precipitation at Mine Site[1]	2,417
Mean Annual Evaporation at Mine Site[2]	1,796

Notes: [1]Tuvatu Gold Mine Hydrologic Investigations (SMEC 2017)
[2]Data from Nadi Airport assumed to be pan evaporation, factored by 0.85 to represent tailings pond evaporation

18.8.4 Storm and Dry Year Events Data

Storm event precipitation data are shown in Table 18-5.

Table 18-5: Storm event and dry year precipitation data

Storm Event Precipitation	Value (mm)
1-in-10-year Rainfall (24 h) (EDF for Sedimentation Pond)[1]	257.3
1-in-100-year Rainfall (24 h) (EDF for TSF Pond and Diversion Ditches)[1]	393.8
Probable Maximum Precipitation (24 h) (IDF for TSF Pond)[2]	1,152.0
1-in-50-dry-year Annual Precipitation[3]	1,092.0

Notes: EDF – Environmental Design Flood; IDF – Intensity-Duration-Frequency
[1]Based on short duration (5 min to 24 h) IDF data for Nadi Airport (Data Period: 1951 – 2006), using Gumbel double exponential distribution for annual extremes
[2]Based on short duration (5 min to 24 hr) IDF data for Nadi Airport (Data Period: 1951 – 2006), using Hershfield method.
[3]Minimum value reported at Nadi Airport assumed to represent 1-in-50-year, prorated by 1.266 (Tuvatu Gold Mine Hydrologic Investigations, SMEC 2017)

18.8.5 Wind

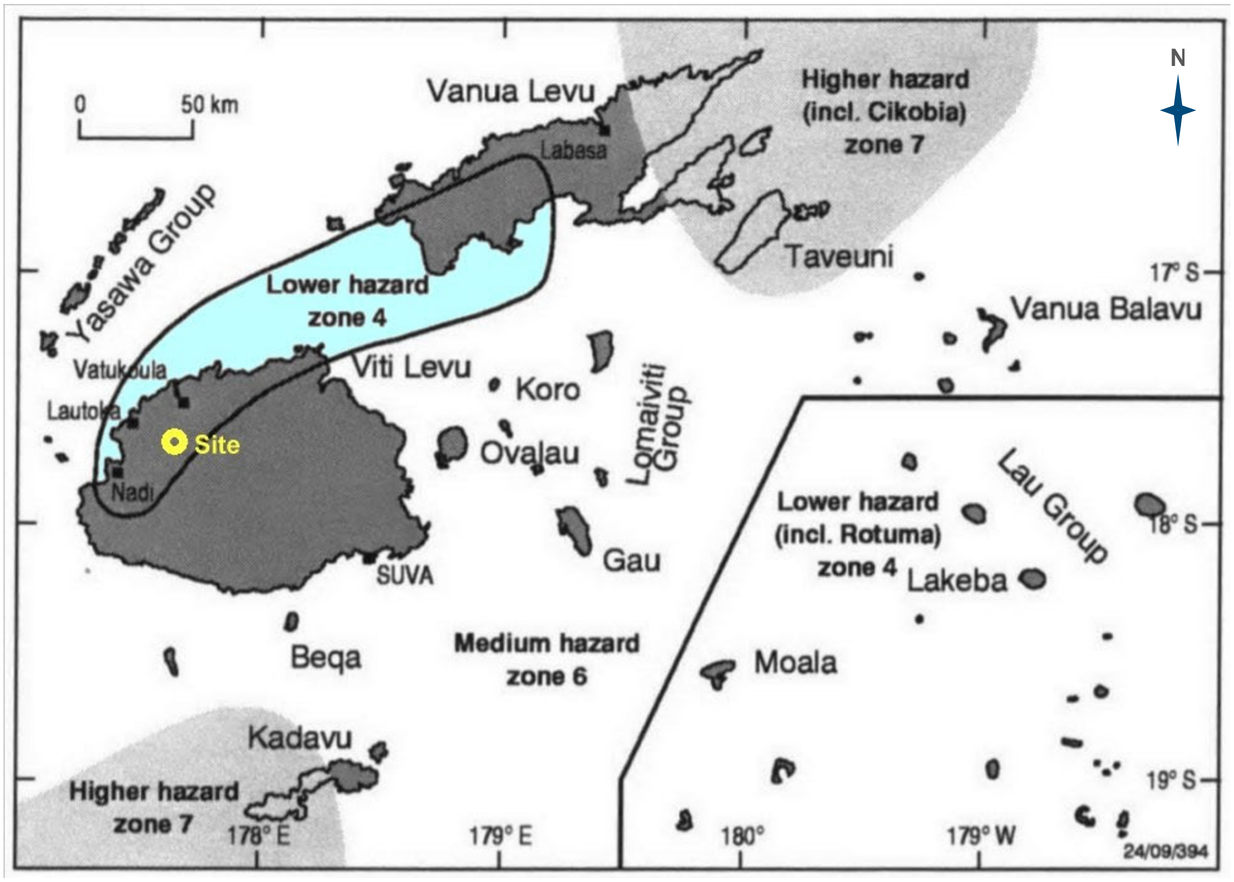
Winds at Nadi Airport do not show spatial consistency and are primarily from east to southeast and west to northwest during the daytime hours and from east to southeast during the night-time hours. Maximum wind gusts at Nadi Airport by month range from approximately 38 knots (70 km/h) in June to 110 knots (204 km/h) in January.

18.8.6 Site Seismicity

According to the Probabilistic Earthquake Hazard Assessment for Fiji (1998), the hazard estimates for most of Viti Levu are comparable to moderate values in New Zealand. The primary reason for such comparison is the fact that the Structural Provisions for Earthquake Loads are in the National Building Code for Fiji.

An earthquake risk map of Fiji, as presented in the National Building Code of Fiji, is shown on Figure 18-10. According to the map, the proposed TSF site is located within lower hazard zone 4.

Based on internationally published data given in various seismic hazard maps for Pacific Islands countries including Fiji, the horizontal peak ground acceleration (PGA) with 10% probability of exceedance in 50 years (475-year return period), may be considered as 0.24 *g*. This is the same value that had been recommended by a local engineering company (Entec) for the mine site in their 2017 report. Other PGA values are shown in Table 18-6.



Source: National Building Code for Fiji (1990)

Figure 18-10: Earthquake risk map for Fiji

Table 18-6: Other peak horizontal ground acceleration (PGA)

Return Periods	PGA (*g*)
1-in-475-year PGA	0.24
1-in-2,475-year PGA	0.50
1-in-4,750-year PGA	0.72
1-in-10,000-year PGA	0.90

18.8.7 Tailings Properties and Disposal

The following are considered in the TSF design:

- The detoxification process is to reduce the total WAD cyanide to approximately 1 ppm in the tailings slurry.
- Tailings are to be thickened to 55% solids (w/w), as per Lion One. However, due to ramp up and feed rate variation, an initial solids content of 50% solids is expected for the first year of operation.
- Based on available test results, the tailings were identified to be PAG with carbonate neutralizing potential ratios (NPR) of less than 1 (NPR of approximately 0.9). However, the lag time to acid onset for the tailings and their potential for neutral / acidic ML is not known. Overall, existing geochemical characterization information for the tailings is limited; as such, a geochemical testing program has been proposed. For the purpose of this Technical Report, the following geochemical assumptions were made:
  - The tailings become net acid generating in the near-term (i.e., within weeks to months of exposure).
  - The tailings will leach metals prior to and following the onset of acidic conditions.
  - Steps to mitigate these risks will need to be taken for TSF operation and closure.
- Settling and rheology testing were completed at Pocock Industrial (Pocock) laboratories in July 2018 with undrained settled dry densities reported as 1.03, 1.04, and 1.11 g/cm^3 for 50, 55, and 60% solid content (by mass), respectively. Recent settling testing at SGS Canada Inc. in Lakefield, Ontario (SGS Lakefield) in August 2018 showed similar undrained settled dry densities at 1.06, 1.09, and 1.13 g/cm^3 for 50, 55, and 60%, respectively. For design purposes, the initial in-situ dry density has been selected to be at 1.00 g/cm^3 or 1.0 t/m^3.
- Air drying testing was performed in August and September 2018 at SGS Lakefield to simulate the tailings beach drying process. The final dry densities were reported at 1.73, 1.78, and 1.88 g/cm^3 for 50, 55, and 60% solid content, respectively. The drying tests have demonstrated that a competent tailings beach can be established for the proposed centreline dam raising. For design purposes, the ultimate whole in-situ dry density has been selected to be at 1.40 g/cm^3 or 1.40 t/m^3.
- SG of the tailings is determined at 2.77 for one sample (ID. CD1) by Pocock.
- Slurry and reclaim water will be transported via single-walled HDPE pipelines, as per Lion One's instruction.
- Disposal types will be subaerial and subaqueous.
- As previously described, tailings were considered to be acid generating for the purposes of this study given the limited geochemical information. However, the lag time to acid onset is not known. Therefore, to be conservative, the deposition plan will incorporate considerations for ARD management for the subaerially deposited tailings, potentially including frequent deposition of fresh tailings over exposed tailings beaches or other steps to maintain saturation of the beached tailings. Alternatively, water quality management (i.e., treatment) may be required. Determining the lag time to acid onset would support identification of tailings management needs for operations, and this approach will be refined once additional information is available. At closure, tailings will be stored under a water cover and any beaches will be covered with a low permeability cover to minimize infiltration.
- Spigotting / end discharge from pipelines on dam crest at 100 m spacing will be used.
- Tailings beach slopes will be 0.5% (subaerial) and 2% (subaqueous).

18.8.8 Mill Operation Data

The mill operation data are summarized in Table 18-7.

Table 18-7: Mill operation data

Criterion	Units	Year 1	LOM	Total
Mill Feed Production				
Resource	kt	-	-	2,555
Underground Mine	kt	365	2,555	2,555
Design Production Rate	t/d	1,000	1,000	-
Tailings Production and Properties				
Tailings Mill-feed Ratio	-	-	-	1.0
Discharge Slurry Solid Content by Mass (Mass Solids/Mass Solids + Mass Water) per Lion One	% (solids by mass)	50	55	-
Specific Gravity of Tailings Solids	-	-	2.77	-
Project Design Life	years	1 and 2	8	10
Deposited Tailings				
Dry Density (after Desiccation and Consolidation)	t/m^3	1.0	1.4	Tested/Assumed
Reclaim Water Decant Strategy	-	Barge mounted pumps in TSF pond		

18.8.9 Dam Design Criteria

The TSF and associated dam structures have been designed based on a well-established design criteria, which are documented in a separate technical report (Wood, 2019).

18.8.10 Dam Construction Materials

18.8.10.1 General Approach for Starter Dam Construction

Given the distinct dry and wet seasons, effort will be made to construct the TSF dam year-round, using clay-rich soil (Zone 5 – Random Fill) and rock fill (Zones 2A and 2B) in the dry season, and rock fill (Zones 2A and 2B) only in the wet season. Based on the limited available rock fill from the initial mine development and its uncertain geochemical nature, an approximate ratio of 60% soil and 40% rock (from both mine and TSF basin) will be established in consultation with Lion One. In addition, a construction duration of 12 to 18 months has been anticipated by Lion One.

It is to be noted that sources of dam raising fill materials are yet to be defined for the detailed design stage and will need to be reviewed when a more definitive mine plan is available.

18.8.10.2 Clay Core and Basin Liner – Zone 1

The TSF basin is naturally overlaid by a thick, completely weathered rock overburden, geologically called saprolite, which is rich in clay content. Laboratory geotechnical testing has demonstrated that it has a very low permeability of less than $1x10^{-5}$ cm/s, suitable to form a natural low permeability dam core and liner over the TSF basin. The saprolite layer at some areas along the creek bed has been washed out somewhat, requiring placement and compaction of the low permeability layer to create a continuous seepage barrier over the entire basin.

18.8.10.3 Waste Rock / Rock from TSF Basin – Zones 2A (Waste Rock) and 2B (Non-potentially Acid Generating / Non-metal Leaching Rock)

Waste rock from the mine development and weathered/sound/intact rock from TSF basin are proposed to be used as one of the main components of the TSF dam structure, pending appropriate results from geochemical characterization of these materials.

18.8.10.4 Sand, Gravel, and Crushed Rock – Zones 3 and 4

Natural sand, gravel, and crushed rock products will be used as filter (Zone 3) and transition (Zone 4) zones. These materials require geochemical characterization and must be non-potentially acid generating (NPAG) and non-metal leaching (NML).

18.8.10.5 Random Fill from TSF Basin – Zone 5

A mixture of saprolite and partially weathered, NPAG/NML rock from the TSF basin is proposed to be used as another main source of dam building materials (Zone 5 – Random Fill), which is still rich in clay content and of low permeability, suitable to form a natural low permeability dam core and liner over the TSF basin.

The partially weathered and sound rock in the TSF basin has yet to be tested for PAG/ML potentials.

18.8.11 Site Preparation and Earthworks

18.8.11.1 Clearing and Grubbing, Stripping

If required, close cut clearing of trees and vegetation will be carried out in the TSF impoundment area and TSF dam footprint followed by grubbing and striping. Clearing, grubbing, and stripping will be executed in stages to support the development of TSF dam raises, thus minimizing the impact on the environment. All loose, wet, low-strength, fine-grained soils, which are unsuitable in the TSF dam foundation, will be removed.

Stripped topsoil and overburden soil will be stockpiled in a designated area within the mine lease boundary and used for future site reclamation.

18.8.11.2 Dam Foundation Preparation

The following foundation preparation measures are anticipated for TSF, sediment control, seepage retention, monitoring, and diversion pond dam footprints:

- Remove all organics.
- Remove all unsuitable overburden soils, such as soft/loose, wet/saturated, fine-grained soils.
- Excavate cut-off key trench to competent low permeability soil stratum and backfill with compacted Zone 1 clay-rich, low-permeability fill, where needed, to eliminate piping potential and minimize uncontrolled seepage.

18.8.11.3 TSF Dam Construction

The TSF dam will be constructed primarily out of geochemically suitable random fill and mine rock / TSF rock materials borrowed from within the TSF valley. For the purposes of this Technical Report, the design criteria for mine rock use in TSF construction are based on conservative assumptions since available geochemical information is very limited. The design criteria will be refined as opportunities and risks are further defined with additional geochemical testing.

TSF Starter Dam and Subsequent Raises

A starter dam will be required to store the first year of tailings production. Annual raises will be subsequently required to support ongoing tailings disposal after that time.

TSF dam will be raised by approximately 5 to 6 m following the first year of operation; for Year 2 tailings storage, centerline construction method will be used. The yearly raising height is decreased to 1 to 3 m for the final years of tailings disposal operation.

18.8.12 Sediment Control, Seepage Retention, and Monitoring Pond Dam

The SCRMPD is located downstream of the TSF dam and has dual purposes of collecting, retaining, and monitoring contact water from the TSF dam's downstream slope plus potential seepage from the TSF.

This pond is designed to contain 1-in-10-year, 24 h storm event. Beyond this event, excess runoff will be discharged via an emergency spillway to the environment under the assumption that water quality will be acceptable due to significant dilution.

The sediment/retention pond will discharge to the environment by pumping if water quality is acceptable; otherwise the pond will discharge to the TSF. The pond will be operated in a nearly empty condition in preparation for storing runoff from a storm event.

Water quality estimates for the TSF contact water have not been developed due to limited available information. Additional geochemical testing to support the development of water quality estimates has been proposed.

18.8.13 Tailings Reclaim Pond and Pump Barge

The tailings reclaim pond will be formed toward the back (south side) of TSF basin by discharging tailings from the dam crest. The tailings beach slope is assumed to be approximately 0.5% above water and about 2% below the pond level.

The pond will be capable of containing 1-in-100-year, 24 h storm event, beyond which, excess runoff will be routed out of TSF via an emergency spillway to the environment under the assumption that water quality will be acceptable to discharge due to significant dilution.

A floating pump barge is planned to send tailings water back to the mill with nominal pumping capacity of 1,000 m^3/d.

Water quality estimates for the TSF contact water have not been developed due to limited available information. Additional geochemical testing to support the development of water quality estimates has been proposed.

18.8.14 Dam Performance and Water Quality Monitoring

Dam stability; movement; deformation; and seepage, surface, and ground water quality will be monitored with adequate instrumentation, routine inspections, and monitoring wells, using standard Western practice.

18.9 Power Supply

18.9.1 Fiji Grid and Generating Capacity

An 11 kV transmission line crosses the Project site from a nearby EFL hydroelectric generation facility. Due to the national shortfall in power supply from the grid, despite supplementary thermal generating capacity, the Project will generate its own power by a hybrid power generation system, consisting of seven, 1,200 kW containerized gensets (five in operation and two in standby), two, 800 kW gensets used for mining preproduction, and a solar power system that will be capable of providing full power supply to the Project during day time.

18.9.2 Project Power Requirements

The estimated mine power requirements and estimated process plant and infrastructure power requirements are summarized in Table 18-8.

Table 18-8: Predicted Project power requirements

Area	Total Connected Power (MW)	Average Power Consumption (MW)
Mine (Surface, Portal, and Underground)	1.6	1.1
Process Plant	6.0	4.4
Infrastructure (Allowance)	0.5	0.3
Total	**8.1**	**5.8**

18.9.3 Power Generation

A hybrid solar power generation system / containerized diesel power station is planned to be constructed to provide electrical power for the overall mine site. This system will operate primarily on solar power during daylight hours and automatically switch to diesel-generated power at night. The hybrid system will include all required automated control systems, switchgear, and substation. The solar panels will be located near Korobebe Village. The diesel gensets, control system, switchgear, and substation will be located at the proposed mine site. An overhead transmission line will be used to carry the solar power to the mine site.

The mine site primary power will be 6.6 kV, three phases, 50 Hz. The solar power generation system will be designed for 7.03 MWh with an annual projected production of 10.31 GWh. The diesel generation power station will be comprised of seven diesel generator units each rated at 1,200 kW and two, 800 kW diesel generator units. The two, 800 kW units will be used for pre-production mining ahead of the installation of the solar farm and other 1,200 kW units. For production start, all nine units will be tied together along with the solar power system in normal operation. The diesel power station is designed to operate as a N+2 redundancy configuration to service the overall operating load of approximately 5.8 MW. This hybrid configuration will ensure reliability of supply and provide sufficient reserve to start the ball mill motors. Power of 6.6 kV will be provided to the main load centers, such as the process plant, and for distribution to the mine access portals and ventilation shafts. Underground distribution will be 1 kV with further voltage reductions as dictated by mining equipment and dewatering pumps. At the process plant, the voltage will be stepped down to 415 V for the low-voltage drives and as appropriate to supply ancillary equipment requirements. Remote facilities such as the TSF and the administration office at the exploration camp will be equipped with standalone generators.

The hybrid system will be constructed under an “across the fence” power supply contract. The contract structure requires no initial capital expenditure by Lion One as the capital is amortized in the power cost. The power cost was estimated to be USD$0.30/kWh based on delivered diesel cost of USD$0.75/L. The delivered diesel price will include a USD$0.05/L concession from Fiji Revenue and Customs. The total power cost will vary with delivered diesel price. The power supply contract is structured such that the total power cost will drop to USD$0.28/kWh after payback of initial capital.

18.10 Communications

The existing mobile tower and communications systems at the mine site are inadequate for mine operations. Vodafone, a local wireless communication provider, has upgraded the communication system, including installing a new microwave tower adjacent to the existing microwave tower above the mine site to improve mobile phone and internet service. The site mobile phone and internet systems have been upgraded and expanded for current exploration and operation use. The construction cost for the new tower and internet system upgrade are borne through Lion One’s monthly internet payments over a five-year contract. No additional capital costs for communications are required.

The plant control room and substations will be linked by a hard-wired control network, with remote stop/start capability. The reclaim pumps and seepage collection system control will be performed by telemetry with back up by radio network, which will also be used for surface communications and emergency response.

Underground voice communications will consist of primary head-end equipment and a radiating cable (leaky feeder) network system with capability for very-high frequency digital underground two channel operation. One underground channel will be linked to a surface channel. The radiating cable will serve as the antennae for radio communication and will also provide power for amplifiers and repeaters required to maintain signal integrity over the entire length of the radiating cable. An electronic tagging system for personnel and equipment location may be considered.

18.11 Logistics and Other Infrastructure

18.11.1 Administration, Security, and Emergency Medical Facilities

The administration building will be located immediately west of the existing exploration camp. The administration building will house mine management, information technology, clerical, procurement, human resources, community relations, environmental, and safety personnel. Additional functions, including general administration, accounting, government and public relations, and geographic information systems, will be housed in Lion One’s Nadi office.

There will be a gatehouse at the western junction of the existing Navilawa Road and the new rerouted section of the Navilawa Road. The entire site will be fenced, including the exploration portal. The gatehouse, with manned boom gate, will include security offices and a first-aid room. A paramedic will be stationed on site at all times. The site ambulance and fire response vehicles will be located adjacent in dedicated parking bays.

18.11.2 Maintenance Shop and Warehouse

The maintenance shop and warehouse will be located immediately west of the administration office. The maintenance shop is intended to service light vehicles and provide general mechanical and electrical repair services. The warehouse will include indoor storage and an outdoor, uncovered, fenced storage area.

18.11.3 Mine Infrastructure

The mining operations will require surface facilities such as mine dry, mine truck shop, and explosives magazines. The mine dry and mine truck shop will be located adjacent to each other and directly south of the crushing plant. Access to the area will be through the main gate.

The mine dry will be sized for 70 personnel per shift. The dry will be equipped with shower and toilet facilities, hanging baskets, and lockers. The mine truck shop will initially have two service bays with additional bays to be added in the future as required. It will include lubricant/oil storage and a disposal storage area for waste oil. A vehicle wash area with wash water containment and treatment will also be located adjacent to the truck shop.

Mining explosives will be housed in three separate 6 m long sea containers converted to storage magazines. These magazines will be located approximately 150 m east of the existing exploration portal, well away from the main process plant and public road. Access to the magazines will be via the main gate. One magazine will be used to store ammonium nitrate and fuel oil (ANFO), one to store detonators, and one to store primer/power gel. The ANFO and primer/power gel magazines will be stored on one platform level separated by an earth bund. The detonator magazine will be located on a separate platform elevated above the other magazines. Both explosives storage platforms will be securely fenced with video monitoring. Lion One will own the storage magazines but the explosives vendor will be responsible to supply, transport, and unload the explosives into the magazines. This magazine site has been approved by the regulators.

18.11.4 Accommodation

Proximity to Nadi, Latouka, and local villages provides sufficient accommodation for contractors and mine operation personnel. Local landowners will be contracted to provide transportation of workers to site. The contractor workforce will be sourced from local communities where possible. Only key personnel or specialist personnel may be mobilized from elsewhere and housed in nearby communities. Mine operation personnel will be sourced locally where possible. Personnel with specific expertise not available locally will be sourced from other parts of Fiji or offshore and will also be housed in nearby communities.

18.11.5 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.

18.11.6 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 on the southeast of the island. The port of Lautoka is capable of handling bulk, containerized, and break-bulk cargo, and has a 100t 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 Project at Lautoka from any point of origin. A significant number of sea containers have already arrived at the Port of Lautoka for the Project and cleared customs with minor or no delays.

18.11.7 Ground Transport

Transport to site from the Port of Lautoka is approximately 40 km by road. The last 9 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 ft. semi-trailer. Any loads above 32 t must be transported via low loader, of which there are several available in Fiji.

No issues are anticipated with weight restrictions on bridges on the route to site. There are, however, some height restrictions due to low power lines. Loads up to 4.2 m high may travel without restriction; above this height, a permit from the FRA will be required.

18.11.8 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.

18.11.9 Logistics Service Providers

There are several established logistics/transport/customs clearance agents in Fiji that could manage all Project freight from export port to delivery at the Project site. Carpenters Fiji Ltd. (Carpenters), UB Freight Ltd. (UB Freight), and Williams and Gosling Ltd. are all capable of providing these services. Carpenters and UB Freight can also look after all customs clearance and assist with permit applications for special cargo.

18.11.10 Import Duty

Import duties are levied under the 2012 Fiji Harmonized Customs Tariff Schedule. For companies involved in a new mine, the applicable code is 252, which states that machineries and specialized equipment (except for hand tools of a kind for general purpose use), motor vehicles, and mobile equipment (except for parts and accessories) used for mining are duty exempt but still attract a 15% Value Added Tax (VAT), which will be refunded upon application.

This exemption is not automatic. An application must be submitted, and the exemption formally approved by issued letter. As reported by Lion One, the Project received a duty waiver for underground mobile mining equipment and it is expected that all the mineral processing equipment should be under the same exemption.

18.12 Rehabilitation and Closure

18.12.1 General

The Project closure plan will commence 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 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.

18.12.2 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, mine 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.

19.0 MARKET STUDIES AND CONTRACTS

No marketing study has been completed for the Project. The large numbers of available gold purchasers allow for gold production to be sold on a regular and predictable basis.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

An EIA was completed for the Project and approved on September 27, 2013. Lion One agreed to prepare a CEMMP, which is based on the Terms of Reference provided by the Fiji DE, as part of the approvals process. In April 2018, Lion One submitted a supplemental EIA to straighten Tuvatu Creek at the north side of the existing Navilawa Road, which is necessary for the rerouting of Navilawa Road and subsequent restriction of public access to the Tuvatu plant site. Approval of the Tuvatu Creek diversion EIA was granted on May 29, 2018.

Data have been collected that describe the physical, biological, and socio-cultural environments of the Project area as part of the EIA. The following sections present brief summaries of this information and are not intended to be comprehensive.

Both a Construction Environmental Management Plan and Operational Environmental Management Plan were submitted to the regulatory authorities and approved on July 30, 2014. A Rehabilitation and Closure Plan was also required to be submitted. This was submitted in 2014, though no formal approval was required. All three plans will need to be updated as the development process continues.

Quarterly surveys of the water quality and macroinvertebrate communities have been undertaken since September 2014 to determine the baseline condition of the watercourses located adjacent to the site.

A range of parameters were measured in the field at the time of sampling including pH, conductivity, temperature, and dissolved oxygen concentration. Dissolved oxygen saturations were later calculated based on field temperature.

Laboratory analysis of water samples collected was undertaken for total suspended solids, turbidity, ammonia, total oxidized nitrogen, total kjeldahl nitrogen, total nitrogen, dissolved reactive phosphorus, total phosphorus, sulphide, a range of total metals (arsenic, cadmium, chromium, copper, iron, lead, manganese, mercury, nickel and zinc), and bacteria (*E. coli*, faecal coliforms, and total coliforms). Samples were collected under the guidance of Lion One's Environmental team and independent consultants from New Zealand.

20.1 Physical Environment

20.1.1 Topography and Soils

The topography in the vicinity of the proposed Project is characterized by steep hills and cliffs with deeply incised, small streams. Soils are primarily alluvium near creeks, while hill slopes support Vunatoto gritty loam and Namuna stony sand clay soils.

Soil erosion and sedimentation from earthworks are the primary environmental effects anticipated. These will be mitigated through the implementation of the Soil Erosion and Sediment Control Management Plan. Implementation of the Prevention of Land Contamination Management Plan will also assist in reducing potential effects to local soils from spills of materials such as fuels and lubricants.

20.1.2 Land Use and Landscape

The Project area is characterized by hill slopes with pockets of forest and grassland. Occasional livestock farming occurs as well as subsistence farming along the banks of the Sabeto River. The Sabeto River is also subject to boulder and gravel removal in some areas. The proposed Project will introduce a new industrial activity to the area.

20.1.3 Air Quality

The air quality in the vicinity of the Project area is considered typical of isolated rural settings. Vehicular traffic in the area is minimal and industrial developments are largely absent. The primary air-quality related effects anticipated from the Project relate to the generation of dust, smoke, fumes, and odours, both in the vicinity of the plant site as well as along mine site access roads. Immediate mitigation measures will be to spray dust-generating areas with water during extreme dry conditions and to cover stockpiles with tarps. The implementation of the Air Quality (Fugitive Emissions) Management Plan will also assist in mitigating potential effects.

20.1.4 Noise

Pre-disturbance noise levels for the project area are considered typical of rural environmental settings. Noise levels are expected to increase with development of the site as a result of construction activities, the use of heavy equipment, occasional blasting, and an increase in vehicular traffic. Mitigation measures for noise include maintaining equipment in good working order and operating the equipment in accordance with manufacturers' specifications. In addition, potentially noisy construction activities will be scheduled to occur during daylight hours only in order to offset noise effects. Further details are provided in the Noise Management Plan.

20.2 Biological Environment

20.2.1 Surface Water

Surface water quality and quantity were characterized as part of the EIA. Areas assessed in the vicinity of the proposed Project, and particularly downstream of the mine, show signs of disturbance from various human activities, such as gravel extraction and erosion due to farming (sugar cane in particular). Physical parameters were found to be within acceptable standards for tropical surface waters (per Australian and New Zealand Environment and Conservation Council ANZECC (2000) Guideline for Protection of Aquatic Ecosystems and Fiji's *Environmental Management Act* (EMA) 2005), with the exception of conductivity, pH, temperature, and dissolved oxygen. Metal levels are also evaluated regularly and have been found to be low and well within recommended guideline limits.

Water from local creeks and drainages will be one of the sources of water for mine operations (e.g., processing plant). Fresh water from creeks will also be treated and used for drinking water at the mine. A Water Quality Management Plan has been established for the Project that will guide activities involving water use.

20.2.2 Freshwater Ecology

Freshwater vertebrates and invertebrates were characterized from various watercourses in the vicinity of the proposed Project. The fish documented during the studies are important food sources for local communities living near the Sabeto River. They are characteristic of degraded systems and spend part of their life cycle at sea. None of the fish are endemic to the Fiji archipelago.

Periphyton and macroinvertebrates were sampled from the Sabeto River and were characteristic of tropical streams/rivers with varying degrees of disturbance (e.g., sampling sites that were less disturbed supported healthier communities of macroinvertebrates, compared to areas further downstream, particularly downstream of gravel extraction sites, that were more highly disturbed).

20.2.3 Groundwater

Groundwater has been monitored in relation to the groundwater discharge emanating from the existing Tuvatu adit. It is anticipated that groundwater will continue to be intercepted by the Tuvatu underground adit and any effects would be localized. Groundwater from proposed mine dewatering will be a primary source of water for mineralized material processing operations.

Inhabitants living close to creeks have typically installed their own water wells for drinking and general use. Lion One will maintain the quality of water downstream of the proposed mine during operations and after mining has ceased so that good quality water will remain accessible to those who are reliant on it.

20.2.4 Flora

The area surrounding the proposed Project is characterized by dry lowland rainforest, grassland, and shrub vegetation. Riparian vegetation and gully forests are also present. These vegetation types and forest systems are typical of areas located on the leeward side of larger Fijian islands such as Viti Levu.

Much of the area shows signs of historical and more recent disturbance from activities such as burning, agricultural development, and livestock grazing. Forests are considered secondary with native and alien species present in both the understory and overstory. Grasslands are prevalent and cover many of the slopes and ridges at lower elevations.

None of the plants identified during the EIA process were considered rare, threatened, or endangered; they are all common and widespread throughout the area and extend to other provinces and islands in Fiji.

Potential effects to local flora will be managed in accordance with the Flora and Fauna Management Plan established for the Project.

20.2.5 Fauna

Wildlife surveys conducted as part of the EIA process included documenting birds, herptiles (reptiles and amphibians), and mammals. The birds identified were characterized as wide-ranging generalist species that are common to secondary and disturbed habitats. No birds of conservation significance were recorded. Previous sightings of a listed sub-species of Peregrine Falcon were, however, recorded in 2011. Peregrine Falcons are no longer considered threatened by the International Union for Conservation of Nature; however, it is still regarded as threatened in Fiji.

The reptile and amphibian species recorded as part of the baseline environmental studies are also common to the types of habitats present. None are of conservation concern.

Of the six terrestrial mammal species documented, only one is native to the area (Pacific Flying Fox). It is not threatened and is common throughout the Fiji archipelago.

Potential effects to local fauna will be managed in accordance with the Flora and Fauna Management Plan established for the Project.

20.3 Socio-cultural Environment

The proposed Project will be the first gold mine within the Nadi area. This Project has the potential to provide employment and skills development to locals within the Sabeto catchment region, thus contributing to the social, economic, and institutional development of communities. Indirect economic benefits will help maintain and improve the quality of life and wellbeing in these communities.

Potential negative effects of the Project may include a breakdown of the social system within local communities, such as an increase in school drop-outs or disrespect or disinterest in communal values and beliefs. This trend has been visible in many capital projects elsewhere in Fiji, whereby the local quality of life was worse at the end of a project.

Regular community engagement and the responsible management of the Project will assist in avoiding this scenario with Tuvatu. The community of Sabeto and the surrounding community of Nadi are very supportive of this development due to the benefits it would bring.

The area was assessed for archaeological and historical sites. The area is of great cultural significance and contains evidence and historical accounts of the people whose descendants currently reside in the nearby villages of Korobebe, Navilawa, and Nagado. All areas identified for project development and operations are located well away from any historical site. The Archaeological and Cultural Heritage Management Plan identifies strategies to manage the identification and handling of archaeological or cultural materials that may be discovered as part of project activities.

20.4 Waste Management

A Waste Management Plan has been developed for the Project that outlines the handling of solid and liquid waste, sewage, and waste oil. Waste generated by the Project will be disposed of at approved facilities. Recycling practices will also be implemented where possible to reduce the amount of waste overall reporting to landfill sites.

20.5 Acid Mine Drainage

All contact water and potential seepage (including potential acid mine drainage from the leaching of sulphides) emanating from the TSF will be collected in a downstream Sediment Control and Monitoring Pond. Water entering the pond from the TSF will be clarified of sediment and conditioned prior to being released into the receiving environment. Water quality will be systematically monitored and water discharged to the receiving environment only upon meeting specified water quality discharge criteria. Water not meeting release criteria will be pumped back to the TSF pond and reclaimed back to the mill for use in processing. Establishing a dense plant root system within the Sediment Control and Monitoring Pond will also assist with the trapping and settlement of suspended solids and flocculated particles.

It is understood that the overall percentage of sulphides within the Tuvatu gold deposit is low (up to 5% in the mineralized lodes, but considerably less than 1% in the host rock). Nevertheless, regular monitoring will be undertaken, particularly if the sulphide content changes in other areas of the mineralized body that have yet to be explored.

20.6 Rehabilitation Plan

Lion One has identified general planning and development objectives that are consistent with national and international environmental guidelines and best management practices for rehabilitation. The primary objectives of the Rehabilitation Plan for the Project include:

- Adhering to statutory requirements.
- Providing a site configuration that is stable over the long-term and attains a beneficial post-mining land use.
- Rehabilitating mine-related disturbances that are compatible with the surrounding environment.
- Eliminating public safety hazards.
- Implementing progressive rehabilitation, concurrent with mining, where possible.
- Allocating sufficient funds to implement these objectives.

In order to achieve the rehabilitation objectives, Lion One plans to adopt basic principles that are compatible with the long-term, post-mining land use of the area, such as salvaging and stockpiling soil as a future growth medium, identifying alternative substrates for use as a growth medium (e.g., overburden), incorporating the use of local and native plant species in the revegetation plan, reshaping areas disturbed by mining operations so they achieve site stability, and minimizing erosion. In addition, Lion One plans to monitor and manage rehabilitated areas until a self-sustaining cover of vegetation has been achieved and other reclamation objectives have been met.

The Rehabilitation Plan will be updated periodically to reflect changes to the mine plan as well as changes to anticipated end land uses, as more information becomes available.

21.0 CAPITAL AND OPERATING COSTS

A PEA is preliminary in nature and includes Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the Technical Report will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The capital cost and operating estimates shown in this section for this Technical Report are based on the 2020 costing. No update for cost estimates has been conducted since the 2020 PEA.

21.1 Capital Cost Estimate

Tetra Tech prepared a capital cost estimate for the Project Technical Report with inputs from Entech, Wood, and Lion One.

Tetra Tech established the capital cost estimate using a hierarchical work breakdown structure. The accuracy range of the estimate is +35%/-30%. The base currency of the estimate is Canadian dollars. Table 21-1 shows the foreign exchange rates for Canadian dollar (CDN$) to US dollar (USD$), Chinese Renminbi (RMB¥), and Fiji dollar (FJD$), which were applied as required.

Table 21-1: Foreign exchange rates

Base Currency (CDN$)	Currency
1.00	USD$0.75
1.00	RMB¥5.15
1.00	FJD$1.60

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is USD$66.8 million (CDN$89.1 million), including an average contingency of approximately 16% of the total direct costs. Table 21-2 shows a summary breakdown of the initial capital cost.

Table 21-2: Capital cost summary

Description		Capital Cost Estimate (CDN$ million)*	Capital Cost Estimate (USD$ million)*
Direct Costs			
10	Overall Site	4.3	3.2
25	Underground Mining	27.8	20.8
40	Process	18.2	13.7
50	TSF**	5.4	4.1
70	On-site Infrastructure	2.4	1.8
Direct Cost Subtotal		**58.1**	**43.6**

table continues…

Description		Capital Cost Estimate (CDN$ million)*	Capital Cost Estimate (USD$ million)*
Indirect Costs			
X	Project Indirect Costs	15.4	11.5
Y	Owner's Costs	6.4	4.8
Z	Contingency	9.2	6.9
Indirect Cost Subtotal		**31.0**	**23.2**
Total Capital Cost Estimate		**89.1**	**66.8**

Notes: *Numbers may not total due to rounding.
**Estimate based on Wood Material Take-off Revision F, dated January 23, 2019.

21.1.1 Exclusions

The following items are excluded from the capital cost estimate:

- Working or deferred capital
- Financing costs
- Refundable taxes and duties
- Land acquisition
- Currency fluctuations
- Lost time due to severe weather conditions
- Lost time due to force majeure
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- Warehouse inventories, other than those supplied in initial fills
- Any project sunk costs (studies, exploration programs, plant site preparation, access road upgrading, etc.)
- Mine reclamation costs (included in financial model)
- Mine closure costs (included in financial model)
- Escalation costs
- Community relations

21.1.2 Mining Capital Cost Estimate

The mine plan execution is based on owner management and owner technical services staff, with the initial three years of underground mining to be executed using an underground mining contractor. After three years, the operation plans to transition to owner-operator mining. Lion One engaged an underground mining contractor to provide pricing in a fixed and variable pricing format.

The mining capital cost estimate is based on the UG mining contractor pricing and other Lion One supplied inputs; the costs have not been directly estimated by Entech. Entech has reviewed the mining capital cost inputs and considers them appropriate. All mining costs are presented in US dollars unless otherwise specified. The estimated level of accuracy of the projected operating and capital costs is ±25%.

The mining capital costs include mining equipment, surface and underground infrastructure, establishment costs, as well as capital lateral and vertical development. A total of USD$23.97 million (CDN$31.96 million) was estimated for initial underground mining capital cost, including initial capital mining contingency and pre-production operating costs. Table 21-3 shows a summary of initial, sustaining, and LOM mining capital costs.

The fleet costs described in Table 21-3 are applicable to equipment that is planned to be directly purchased by Lion One, as opposed to equipment that is provided by the mining contractor and has been priced into the mining rates. Mobilization and site establishment costs include mobilization and demobilization of mining equipment and personnel, in addition to the costs associated with establishing any mining-related facilities.

Table 21-3: Mining capital cost summary

Description	Total Mining Capital (USD$ million)*	Initial Capital (USD$ million)*	Sustaining Capital (USD$ million)*
Lateral Development	22.11	7.16	14.95
Vertical Development	3.97	2.31	1.66
Mining Equipment and Infrastructure Development	10.03	7.98	2.05
Additional Mine Site Surface Facilities	0.26	0.26	-
Initial Capital Mining Contingency	3.11	3.11	-
Total Mining Capital Cost Estimate	**39.48**	**20.82**	**18.66**
Mining Pre-Production	3.15	3.15	-
Total Mining Capital Cost Estimate, with Pre-Production	**42.63**	**23.97**	**18.66**

Note: *Numbers may not total due to rounding.

Lateral development average cost is estimated to be USD$2,060/m over the LOM, and vertical development average cost is estimated to be USD$2,729/m over the LOM.

21.1.3 Processing and Overall Site Infrastructure Capital Cost Estimate

Major mechanical costs are based on detailed quotations from vendors. All equipment and material costs are included as free carrier or free board marine manufacturer plants and are exclusive of spare parts, taxes, duties, freight, and packaging. These costs, if appropriate, are covered in the indirect cost section of the estimate.

Earthwork costs estimated are based on actual cost spent to date and quotations received from contractors. Infrastructure, material, and equipment erection costs were based on quotations from contractors.

Project indirect costs, including construction indirects, spare parts, and freight and logistics are based on quotations. Costs for initial fills for grinding media, reagents, lubricants, and fuel were provided by vendors. Engineering, procurement, and construction management; commissioning; and start-up costs are estimated based on quotations. Owner's costs are based on the estimates from Lion One according to the construction plan. The estimated contingencies are allowances for undefined items of work incurred within the defined scope of work covered by the estimate.

The overall direct initial capital cost for processing and overall site infrastructure (excluding TSF) was estimated to be USD$18.68 million (CDN$24.91 million), including an initial direct capital cost of USD$13.67 million (CDN$18.23 million) for processing-related facilities.

21.1.4 Tailings Storage Facility Capital Cost Estimate

The initial TSF capital costs include installation labour cost based on Fiji contractors' rates. The material take-off for earthworks and mechanical equipment were provided by Wood.

Pipeline costs for both supply and installation were based on the in-house data.

Table 21-4 shows the TSF capital cost summary.

Table 21-4: Tailings storage facility capital cost summary

Description	Total (CDN$ million)	Total (USD$ million)
TSF (Starter Dam)	3.61	2.71
Sediment Control and Monitoring	0.24	0.18
Diversion Channel, Surface Water Control, and Supply Pond/Dam	1.11	0.83
Seepage Control	0.43	0.32
Total TSF Capital Cost Estimate*	**5.39**	**4.04**

Note: *Estimate based on Wood Material Take-off Revision F, dated January 23, 2019; numbers may not total due to rounding.

21.2 Operating Cost Estimate

The on-site average operating costs, at a mill feed rate of 1,000 t/d were estimated to be USD$97.35/t (CDN$129.81/t) of material processed. The operating costs are defined as the direct operating costs including mining, processing, site servicing, and G&A costs, including related freight costs. Table 21-5 and Figure 21-1 show the cost breakdown for various areas.

Table 21-5: Operating cost summary

Description	Operating Cost (CDN$/t milled)	Operating Cost (USD$/t milled)
Mining**	62.99	47.24
Process	55.33	41.49
Reclaim Water Handling	0.40	0.30
G&A	8.88	6.66
Site Services	2.21	1.66
Total Operating Cost Estimate*	**129.81**	**97.35**

Notes: *Numbers may not total due to rounding.
**LOM average, excluding pre-production related costs.

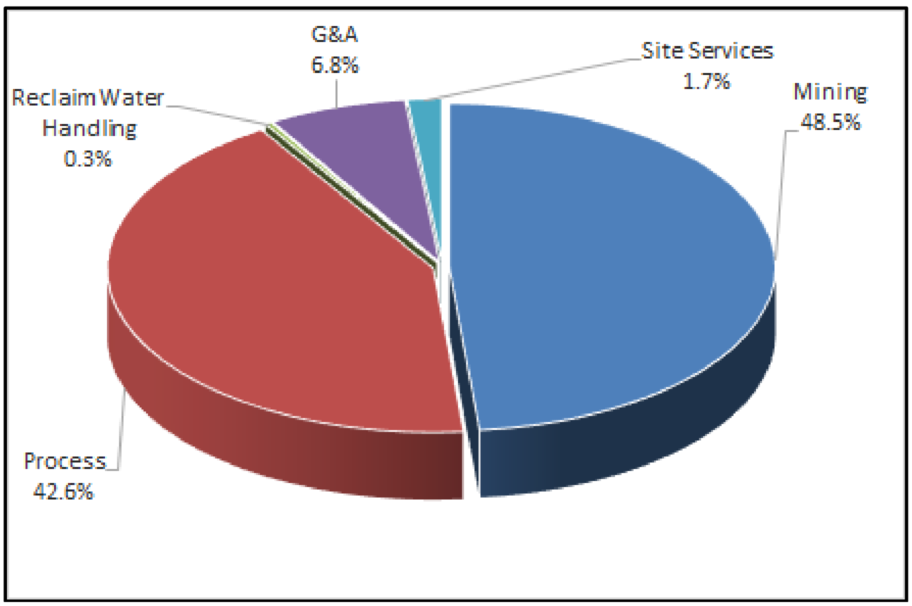


Figure 21-1: Operating cost distribution by area

The cost estimates in this section are based on the consumable prices and labour salaries/wages from Q2 2020, or information from Tetra Tech and other consulting firms' in-house database.

The expected accuracy range of the operating cost estimate is +35%/-30%. All the costs have been estimated in US dollars, unless specified. Table 21-1 shows the foreign exchange rates used for the estimates.

It is assumed that operation personnel will reside in towns or villages nearby. There will be no accommodation or catering services provided at site. Personnel will be bussed to site by the Owner.

The operating costs exclude shipping and refining charges for the doré produced; these costs are included in the financial analysis.

21.2.1 Mining Operating Cost Estimate

The mining operating costs include, labour, consumables, equipment costs, and mining contractor rates. The costs are allocated to either lateral development, vertical development or stoping. Average mining cost for the LOM is USD$49.32/t milled, as summarized in Table 21-6 and Figure 21-2.

Table 21-6: Mining operating cost summary

Description	Total (USD$ millions)	Pre-production (USD$ millions)	Operations (USD$ millions)	Unit Costs (USD$/t)**	Unit Costs (CDN$/t)**
Lateral Development	35.21	2.12	33.09	23.91	31.88
Vertical Development	2.37	0.03	2.34	1.69	2.25
Stoping	30.96	1.00	29.96	21.65	28.86
Total Mining Operating Costs*	**68.54**	**3.15**	**65.39**	**47.24**	**62.99**

Notes: *Numbers may not total due to rounding.
**LOM average, excluding pre-production related costs.

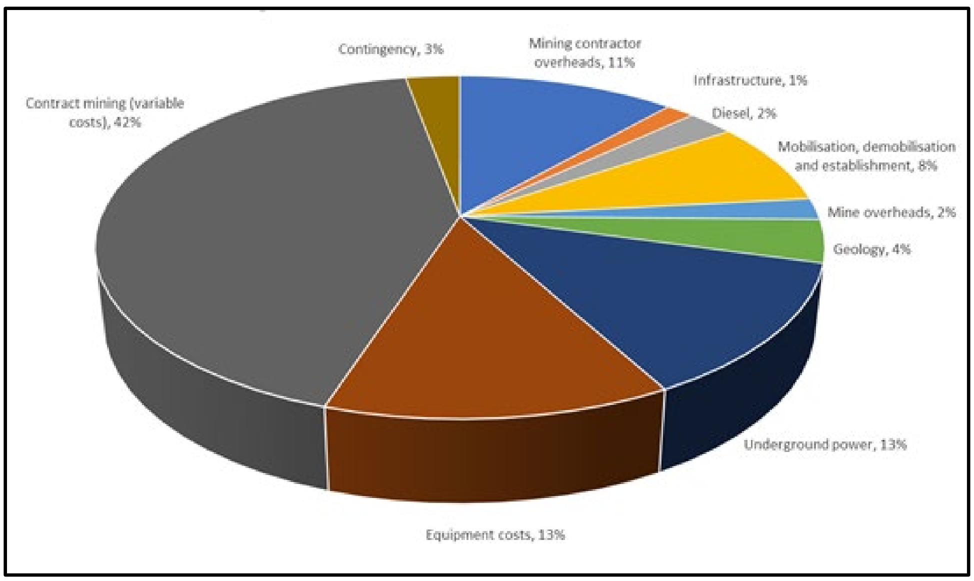


Figure 21-2: Mining operating cost distribution by area (including LOM capital and operating costs)

21.2.2 Processing Operating Cost Estimate

The unit process operating cost was estimated at USD$41.49/t milled (CDN$53.33/t milled) at a nominal processing rate of 1,000 t/d, or 330,000 t/a, including the power cost for the processing plant. The estimate is based on 8-hour shifts, 24 h/d, 365 d/a, excluding the crushing circuit and loaded carbon handling circuit. The crushing circuit will operate on the day shift only for 10 h/d. The loaded carbon ADR circuit will operate over two, 8-hour shifts per day.

Table 21-7 and Figure 21-3 show the breakdown for the estimated process operating cost.

Table 21-7: Process operating cost summary

Description	Unit Cost (CDN$/t milled)*	Unit Cost (USD$/t milled)*
Manpower (68 persons)	5.93	4.45
Metal/Liner Consumables	3.49	2.62
Reagent Consumables	10.17	7.62
Maintenance Supplies	3.35	2.51
Operating Supplies	2.13	1.59
Power Supply	27.63	20.72
Others	2.63	1.98
Total Process Operating Cost Estimate	**55.33**	**41.49**

Note: *Rounded to the nearest hundredth.

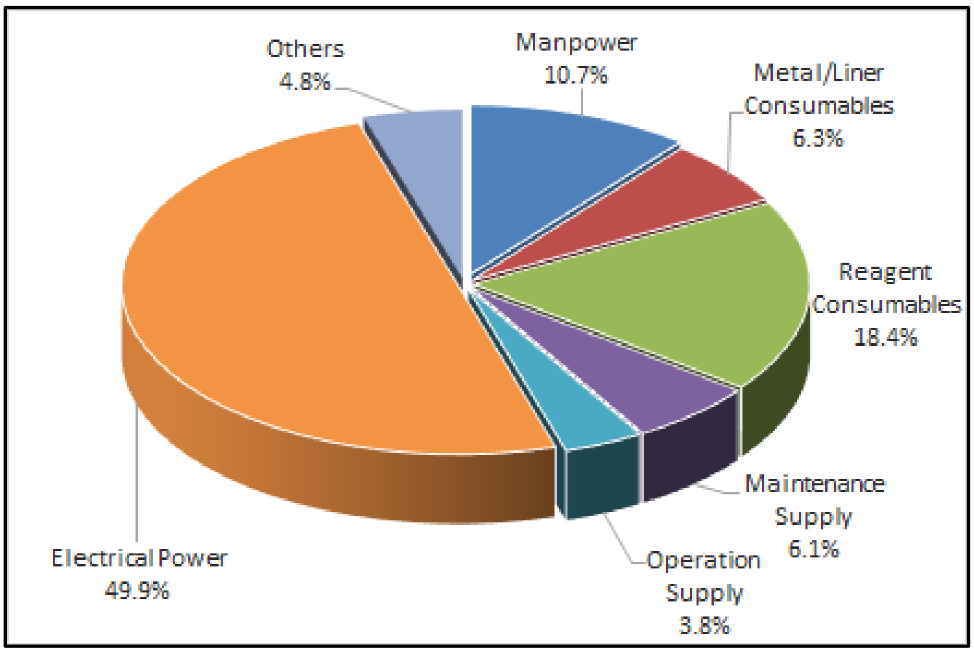


Figure 21-3: Process operating cost distribution by area

The process operating cost estimate includes:

- Personnel requirements including supervision, technical support, operation and maintenance, and salary/wage levels, including burdens, based on Q2 2020 labour rates estimated by Lion One according to the collected local labour rates.
- Crusher and ball mill liner and grinding media consumption, steel ball consumption estimated from the Bond ball mill work index and abrasion index equations and Tetra Tech's experience, steel ball prices based on the quotation from a local supply, and ball mill liner and crusher liner consumptions and prices estimated based on the potential process equipment supplier in China.

- Maintenance supplies, based on approximately 12% of major equipment capital costs or estimated based on the information from Tetra Tech's database/experience.
- Reagent consumptions based on test results and Tetra Tech's database/experience; reagent prices based on the quotation from a local supply or from Tetra Tech's database.
- Other operation consumables, including laboratory and service vehicles consumables.
- Power consumption for the processing plant based on the preliminary plant equipment load estimates and a power unit cost of USD$0.30/kWh (CDN$0.40/kWh), which is estimated based on the on-site power generation from a hybrid solar/diesel genset power system.

All operating cost estimates exclude taxes unless otherwise specified.

21.2.2.1 Personnel

The estimated average personnel cost, at a nominal processing rate of 1,000 t/d, is USD$4.45/t milled (CDN$5.93/t milled). The projected process personnel requirement is 68 persons, including:

- 12 staff for management and technical support, including personnel at laboratories for quality control, process optimization, and assaying.
- 39 operators servicing for overall operations from crushing to doré production and leach residue detoxification.
- 17 personnel for equipment maintenance, including maintenance management team.

The salaries and wages, including burdens, are based on Q2 2020 labour rates estimated by Lion One according to the local labour rates collected.

Labour required for the tailings and reclaimed water management are excluded in this estimate but are included in the tailings and reclaimed water management cost estimate.

21.2.2.2 Consumables and Maintenance / Operation Supplies

The operating costs for major consumables and maintenance/operation supplies were estimated at USD$14.34/t milled (CDN$19.14/t milled), excluding the costs associated with off-site shipment and refining costs for the doré produced. The costs for major consumables, which include metal and reagent consumables, were estimated to be USD$10.24/t milled (CDN$13.66/t milled). The consumable unit prices are based on the quotation from a local supplier and ball mill liner and crusher liner prices are based on a potential Chinese process equipment supplier.

The cost for maintenance/operation supplies was estimated at USD$4.10/t milled (CDN$5.48/t milled). Maintenance supplies were estimated based on approximately 12% of major equipment capital costs and/or based on information from Tetra Tech's database/experience.

21.2.2.3 Power

The total process power cost was estimated at USD$20.72/t milled (CDN$27.63/t milled). Electricity is planned to be generated on site from a hybrid solar/diesel genset power system, which will be constructed under an "across the fence" power supply contract. The power unit cost used in the estimate is approximately USD$0.30/kWh (CDN$0.40/kWh). The contract structure requires no initial capital expenditure by Lion One as the capital is amortized in the power cost. The basic power cost will be USD$0.30/kWh based on delivered diesel cost of

USD$0.75/L. The delivered diesel price will include a USD$0.05/L concession from Fiji Revenue and Customs. The total power cost will vary with delivered diesel price. The power supply contract is structured such that the total power cost will drop to USD$0.28/kWh after payback of initial capital.

The power consumption was estimated from the preliminary power loads estimated from the process equipment load list. The average annual power consumption was estimated to be approximately 23 GWh.

21.2.3 General and Administrative and Site Services Operating Cost Estimate

G&A and site services costs include expenditures that do not relate directly to mining or process operating costs. These costs were estimated at USD$6.66/t milled (CDN$8.88/t milled) for G&A and USD$1.66/t milled (CDN$2.21/t milled) for site services, based on a nominal mill feed processing rate of 1,000 t/d. The G&A and site service costs include personnel and general expenses. Personnel includes general manager and staffing in accounting, purchasing, environmental, security, site maintenances, human resources, and other G&A departments. The estimated total employees are 31 for G&A and 8 for site services. Only personnel working at the Project site are included. Personnel at Lion One's corporate headquarters are not included. Salaries and wages are based on the Q2 2020 labour rates estimated by Lion One according to the local labour rates collected. The salaries and wages include the payment burden.

General and site services expenses include general administration, contractor services, insurance, security, medical services, legal services, human resources, travel, communication services/supports, external assay/testing, overall site maintenance, electricity and fuel supplies, engineering consulting, and sustainability, including an environmental auditing and community liaison.

Table 21-8 shows a summary of the G&A and site services cost estimates. The costs for management and service personnel were estimated at USD$2.11/t milled (CDN$2.81/t milled) for G&A and USD$0.37/t milled (CDN$0.50/t milled) for site services. The other costs estimated for G&A and site services are USD$84.55/t milled (CDN$6.07/t milled) and USD$1.28/t milled (CDN$1.71/t milled), respectively.

Table 21-8: G&A and site services operating cost summary

Description	Manpower	Annual Cost (CDN$/a)*	Annual Cost (USD$/a)*	Unit Cost (CDN$/t milled)	Unit Cost (USD$/t milled)
G&A					
Labour	31	927,600	695,700	2.81	2.11
Other Costs	-	2,002,700	1,502,000	6.07	4.55
Subtotal	**31**	**2,930,300**	**2,197,700**	**8.88**	**6.66**
Site Services					
Labour	8	164,700	123,600	0.50	0.37
Other Costs	-	564,700	423,500	1.71	1.28
Subtotal	**8**	**729,400**	**547,100**	**2.21**	**1.66**

Note: *Rounded to the nearest hundredth.

21.2.4 Tailings Storage Facility Operating Cost Estimate

Overall reclaimed water management cost was estimated at approximately USD$0.31/t milled (CDN$0.41/t milled). The costs associated with the tailings dam construction and the closure were excluded and estimated separately as initial capital costs and sustaining capital costs.

The initial capital and sustaining capital costs are based on the material take-off estimated by Wood. The TSF capital and sustaining costs will be further refined during the next design stage of the Project.

22.0 ECONOMIC ANALYSIS

This Technical Report is a summary compilation of additional work completed on the Project since the 2015 PEA (Freudigmann et al. 2015). A PEA is preliminary in nature and includes Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the Technical Report will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. As such, the financial modeling of the Project does not demonstrate economic viability in accordance with NI 43-101 requirements. There is no certainty that the economic results presented in this study will be realized.

The financial analysis data shown in this section for this Technical Report are based on the 2020 costing. No update for the financial analysis has been conducted since the 2020 PEA.

22.1 Introduction

Tetra Tech prepared an economic evaluation of the Project based on a pre-tax model and an after-tax financial model prepared by Lion One. For the five-year LOM and 1.348 Mt of mine plan tonnage and the gold price of USD$1,400/oz (base case), the following financial parameters were calculated:

- Pre-tax IRR of 60.3%
- After-tax IRR of 50.9%
- Pre-tax NPV of USD$155.8 million (CDN$207.7 million) at a 5% discount rate
- After-tax NPV of USD$121.7 million (CDN$162.2 million) at a 5% discount rate
- 1.5-year payback (pre-tax) on USD$66.8 million (CDN$89.1 million) of initial capital
- After-tax all-in sustaining cost (AISC) of USD$713/oz
- After-tax all-in cost of USD$915/oz

22.2 Assumptions and Qualifications Basis of Estimate

This section includes forward-looking information regarding cash flow forecasts as a result of the study's projected mine production rates, projected gold recoveries, and associated process construction and mine development schedules. Factors that may cause actual results to differ materially from those presented in this economic analysis include:

- The ability to obtain skilled labour, major construction equipment, or long-lead items in a timely fashion.
- The ability to obtain the many permits required on an appropriate timely basis in order to construct and/or operate the mine and process facilities.
- To achieve the assumed mine production rates at the assumed grades.

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.

The basis of the economic evaluation of the Project was obtained from a variety of information sources including:

- In discussions with Lion One, Tetra Tech, with mining capital cost estimates from Entech and TSF material take-off inputs from Wood, prepared the capital cost estimates and expenditure schedules.
- Entech prepared the mine operating cost estimates and mine schedule.
- Owner's capital costs, sustaining capital costs, and closure costs were estimated based upon assumptions or on detail provided by Tetra Tech, Entech, Wood, and Lion One.
- Tetra Tech, in discussions with Lion One, estimated the process operating, G&A, and site services costs.
- Metal pricing, royalties, and refining charges were based upon guidelines provided by Lion One and Tetra Tech.
- The projected gold recovery was derived from metallurgical test work results by various test programs. The economic evaluation reflects one metal price scenario and considered only cash flows from the start of the construction period, based on an 18-month design and construction period, assuming the expenditures for construction initiate in Month 7 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%.
- Working capital has been excluded from the capital cost estimate.
- Capital depreciation has been considered based on a declining balance 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.
- 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 G&A costs during operations.
- 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.

22.2.1 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 USD$1,400/oz representing the approximate average spot gold price for the last two years. A refinery gold payable rate of 99.5% has been applied with a refining charge of USD$0.75 per payable ounce.

22.2.2 Foreign Exchange Rate

Table 22-1 lists the foreign currency exchange rates used in the economic analysis.

Table 22-1: Project production summary

	CDN$	RMB¥	FJD$	USD$
USD$1.00	1.33	6.87	2.13	-
CDN$1.00	-	5.15	1.60	0.75

22.2.3 Royalties

The royalty model inputs provided by Lion One are all based on gold revenue and total 6.5%. They include:

- Government royalty: 5.0% of gold revenue.
- Laimes Global Inc. royalty: 1.5 % of gold revenue.

Total royalty and payments amount to approximately USD$30.1 million (CDN$40.2 million) over the five-year LOM.

22.2.4 Smelting Terms

The refining terms applied in the financial analysis are shown below:

- Percentage payment: 99.5% of delivered gold
- Refining charge: USD$0.75/oz of delivered gold.

22.2.5 Transportation and Insurance

The transportation, assay, and insurance costs applied in the financial model are detailed below:

- Transportation cost: USD$0.75/oz of delivered gold from mine site to refinery
- Assay and insurance costs: 0.1% of the payable value

22.3 Fiji Tax Regime

The following general tax regime was recognized as applicable during the writing of this Technical Report. The pre-production duration for this project is expected to be approximately 18 months.

Resident and non-resident companies are subject to income tax at 20% in Fiji. Income tax losses can only be carried forward for a period of 4 years, and there is no provision for carry back of tax losses. The carry forward of income tax losses has two tests: continuity of ownership and continuity of business.

Tax depreciation in Fiji may be calculated on the cost of a business asset on a straight-line or diminishing-value basis. The prescribed rates of depreciation are based on the estimated life of the asset. Plant and machinery used in manufacturing and mining are eligible to be depreciated at 30% diminishing value (i.e., declining balance method) or straight line at 20% per annum. Motor vehicles, buses and minibuses with a seating capacity of less than 30 passengers, goods vehicles with a load capacity of less than 7 tonnes, computers and data handling equipment, and construction equipment and earthmoving equipment are eligible to be depreciated at 40% diminishing value or straight line at 25% per annum. Any depreciable asset not included in another category, including building category, are eligible to be depreciated at 20% diminishing value or straight line at 12.5% per annum.

The cost of the acquisition of a mining lease or tenement and the cost of development of mines may also be written off in equal installments in any five of the first eight years, commencing with the year in which the expenditure was incurred.

22.4 Economic Results

The reader is cautioned that this study includes the use of Inferred Mineral Resources, which 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 Technical Report is preliminary in nature and uses 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.

Table 22-2 presents a summary of the production information on which the cash flow model is based. A total of 1.384 Mt mill feed with an average head grade of 8.6 g/t Au will be processed at an average recovery of 87.3% to recover 331,369 oz of gold.

The LOM capital cost for the Project is estimated at USD$94.3 million (CDN$125.7 million), with a pre-production and peak capital expenditure of USD$66.8 million (CDN$89.1 million).

Table 22-2: Project production summary

Project Production Summary	Unit	Basis of Estimate
Total Mill Feed Processed	t	1,384,067
Average Head Grade	g/t Au	8.57
Contained Gold in Mill Feed	oz Au	381,532
Recovered Gold, including Refining Loss	oz Au	331,369

table continues…

TETRA TECH

Project Production Summary	Unit	Basis of Estimate
Average Gold Recovery	%	87.3
Production Mine Life	years	5
Nominal Production Rate	t/a	330,000
Average Annual Production	oz Au	77,969

Table 22-3 illustrates the Project cash flow summary.

Table 22-3: Project cash flow summary

Project Cash Flow Summary	Project (USD$ million)	Project (USD$/t mill feed)	Project (USD$/oz Au)
Mine Operating Cost	65.39	47.24	197.33
Processing Cost	58.64	42.37	176.97
G&A and Site Service Cost	11.66	8.43	35.20
Smelting and Refining Cost	0.96	0.70	2.91
Sub-total Cash Operating Cost	**136.66**	**98.74**	**412.40**
Royalties	30.15	21.79	91.00
Total Cash Operating Cost	**166.81**	**120.52**	**503.40**
Initial Revenue	**463.92**	**335.18**	**1400.00**
Operating Cash Flow	**297.11**	**214.66**	**896.60**
Initial Capital Cost	**66.82**	**48.28**	**201.65**
Sustaining Capital Cost	**27.44**	**19.83**	**82.82**
Total Capital Cost	**94.27**	**68.11**	**284.47**
AISC (Pre-tax)	**194.26**	**140.35**	**586.22**
All-in Cost (Pre-tax)	**261.08**	**188.63**	**787.88**
Estimated Tax	**42.02**	**30.36**	**126.82**
AISC (After-tax)	**236.28**	**170.71**	**713.04**
All-in Cost (After-tax)	**303.10**	**218.99**	**914.69**

Note: Numbers may not total due to rounding.

At a gold price of USD$1,400.00/oz, the Project economics is estimated as:

- Pre-tax:
  - Discount rate: 5%
  - IRR: 60.3%
  - Pay-back period: 1.5 years, after production commences
  - NPV: USD$155.8 million (CDN$207.7 million)
- After-tax:
  - Discount rate: 5%
  - IRR: 50.9%
  - Pay-back period: 1.7 years, after production commences
  - NPV: USD$121.7 million (CDN$162.2 million).

22.4.1 Annual Cash Flow Chart

Table 22-4 and Figure 22-1 show the annual pre-tax and after-tax cash flows for the pre-production and production years. Cash flows become positive in Year 2 of operations.

Table 22-4: Annual pre-tax and after-tax cash flows

Year	-1.5	-1	1	2	3	4	5
Net Cash Flow, USD$ millions pre-tax	-13.3	-53.5	36.7	62.5	91.0	65.4	14.7
Net Cash Flow, USD$ millions after-tax	-13.3	-53.5	31.9	52.0	75.2	54.7	14.5
Cumulative Cash Flow, USD$ millions pre-tax	-13.3	-66.8	-30.1	32.4	123.4	188.7	203.4
Cumulative Cash Flow, USD$ millions after-tax	-13.3	-66.8	-34.9	17.0	92.2	146.9	161.4

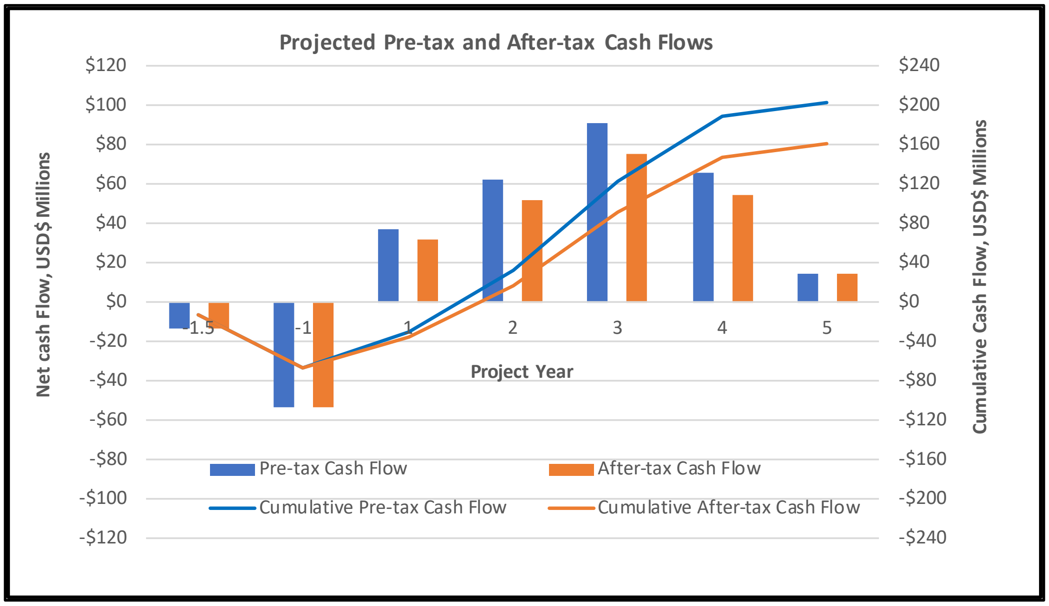


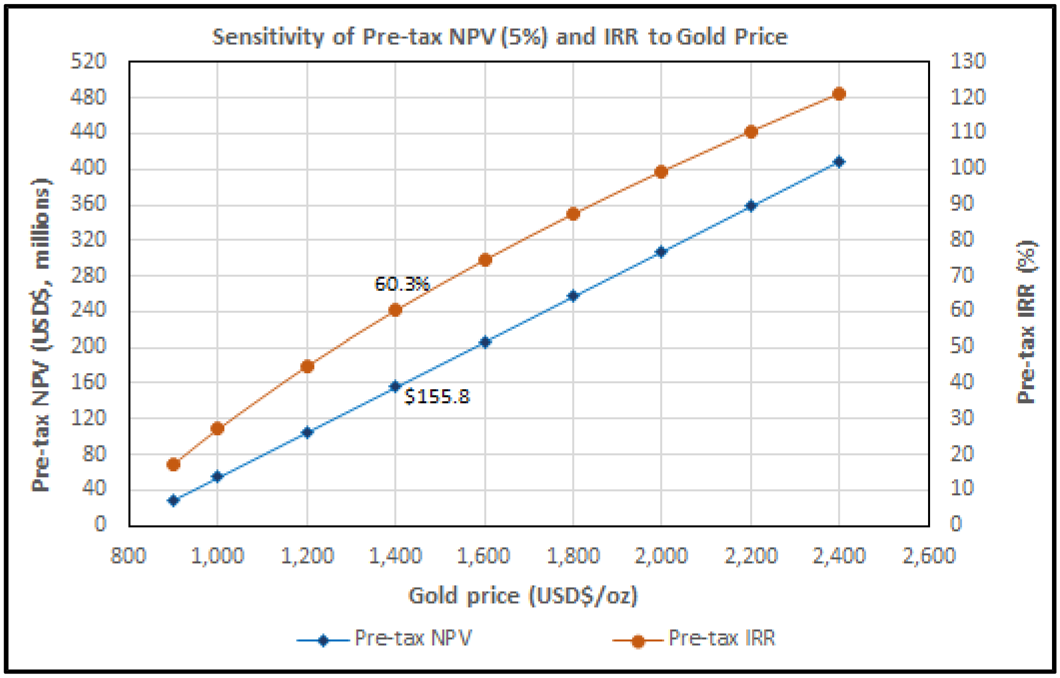
Figure 22-1: Annual pre-tax and after-tax cash flows

22.5 Sensitivity Analysis

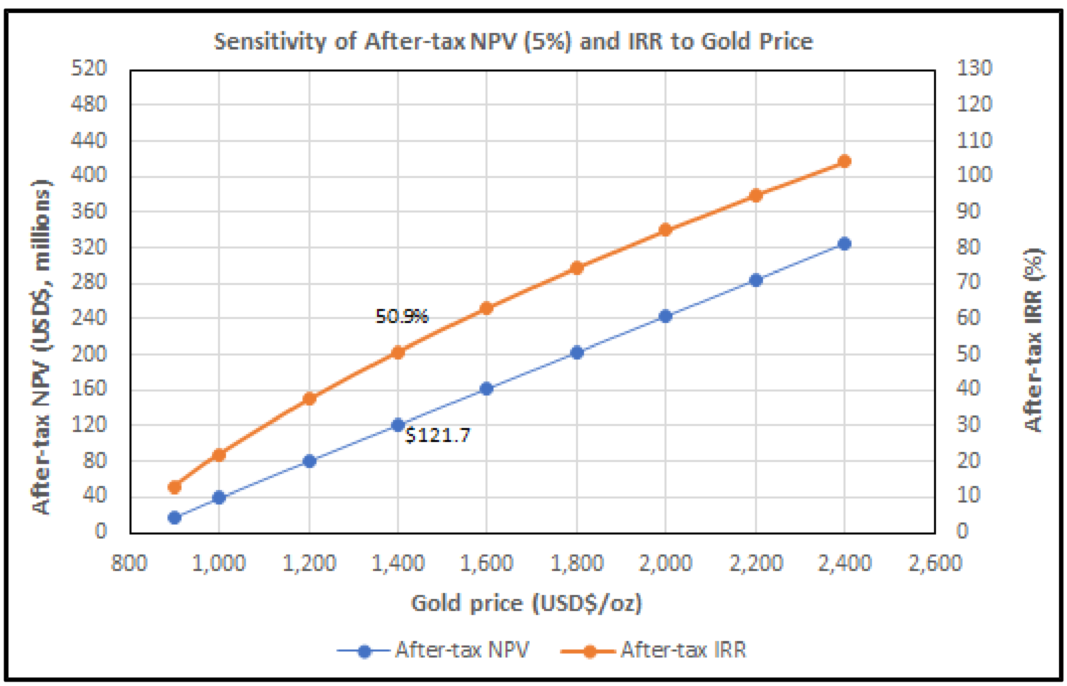
The sensitivity response of the calculated pre-tax and after-tax IRR and NPV (5%) to variations in gold price is shown in Table 22-5 and Figure 22-2, while the sensitivity response of the calculated after-tax IRR and NPV (5%) for capital cost, operating costs, and currency exchange rate are illustrated in Figure 22-3 and Figure 22-4, respectively.

Table 22-5: Sensitivity of pre-tax and after-tax NPV and IRR to variations in gold price

	Unit	Parameter							
Gold Price	USD$/oz	1,000	1,200	1,400	1,600	1,800	2,000	2,200	2,400
Pre-tax NPV at 5% Discount	USD$, M	54.4	105.1	155.8	206.5	257.2	307.9	358.7	409.4
Pre-tax IRR	%	27.1	44.7	60.3	74.4	87.3	99.3	110.5	121.1
After-tax NPV at 5% Discount	USD$, M	40.0	80.9	121.7	162.2	202.8	243.4	284.0	324.5
After-tax IRR	%	22.1	37.4	50.9	63.2	74.5	85.0	94.9	104.3



(A) Pre-tax



(b) After-tax

Figure 22-2: Sensitivity of NPV and IRR to variations in gold price

A USD$200/oz increase in gold price to USD$1,600/oz would increase the Project pre-tax IRR to 74% and after-tax IRR to 63%, increase the discounted pre-tax NPV to USD$206.5 million (CDN$275.3 million) and after-tax NPV to USD$162.2 million (CDN$216.3 million), and decrease the payback to 1.2 years (pre-tax). A USD$200/oz decrease in gold price to USD$1,200/oz would reduce the Project pre-tax IRR to 45% and after-tax IRR to 37% and discounted pre-tax NPV to USD$105.1 million (CDN$140.1 million) and after-tax NPV to USD$80.9 million (CDN$107.9 million), and increase the payback to 1.9 years (pre-tax). The analysis indicates that the Project is most sensitive to gold price.

The IRR and payback period were found to be most sensitive to gold price followed by Fiji currency exchange rate, capital costs, and on-site operating costs, while the NPV is most sensitive to gold price followed by Fiji currency exchange rate, on-site operating costs, and capital costs. Table 22-6 and Figure 22-5 show payback year variation with gold price.

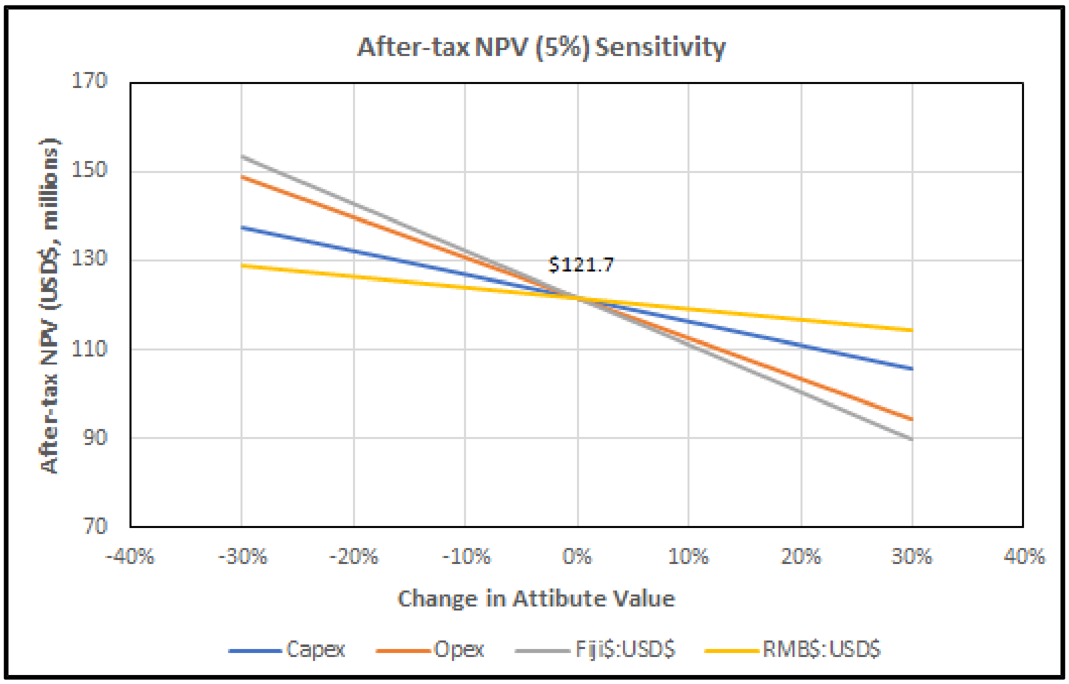


Figure 22-3: Sensitivity of after-tax NPV (5% discount) to variations in project inputs

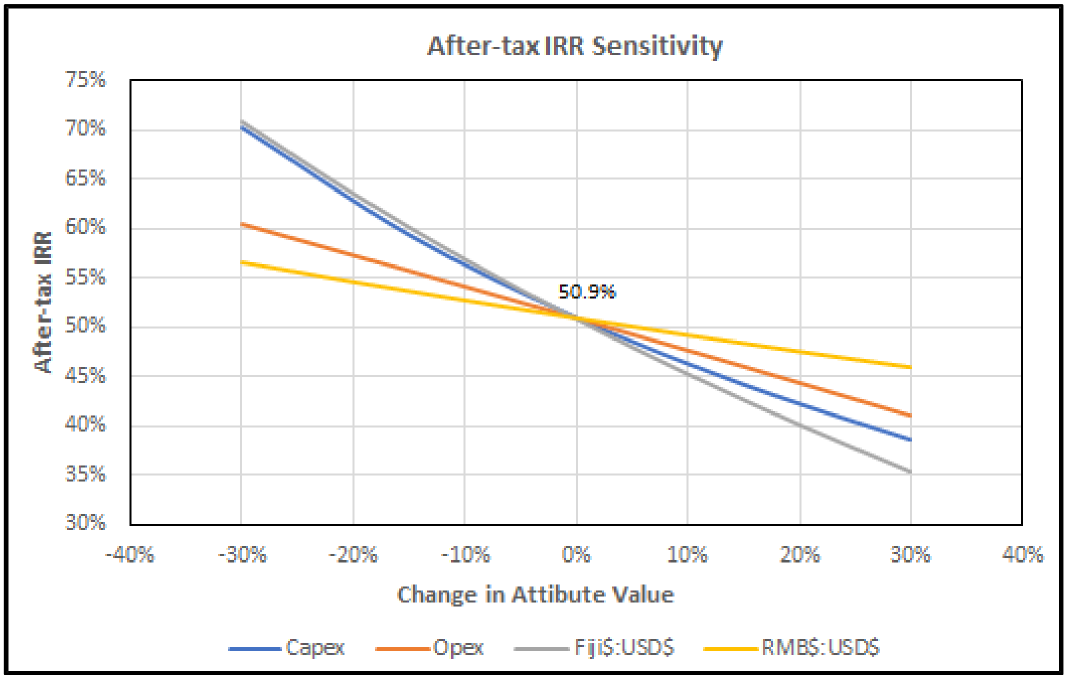


Figure 22-4: Sensitivity of after-tax IRR to variations in project inputs

Table 22-6: Payback year vs. gold price

Gold Price, USD$/oz	1,000	1,200	1,400	1,600	1,800	2,000	2,200	2,400
After-tax Payback Year	2.61	2.09	1.67	1.38	1.19	1.04	0.91	0.80
Pre-tax Payback Year	2.45	1.92	1.48	1.22	1.05	0.88	0.75	0.66

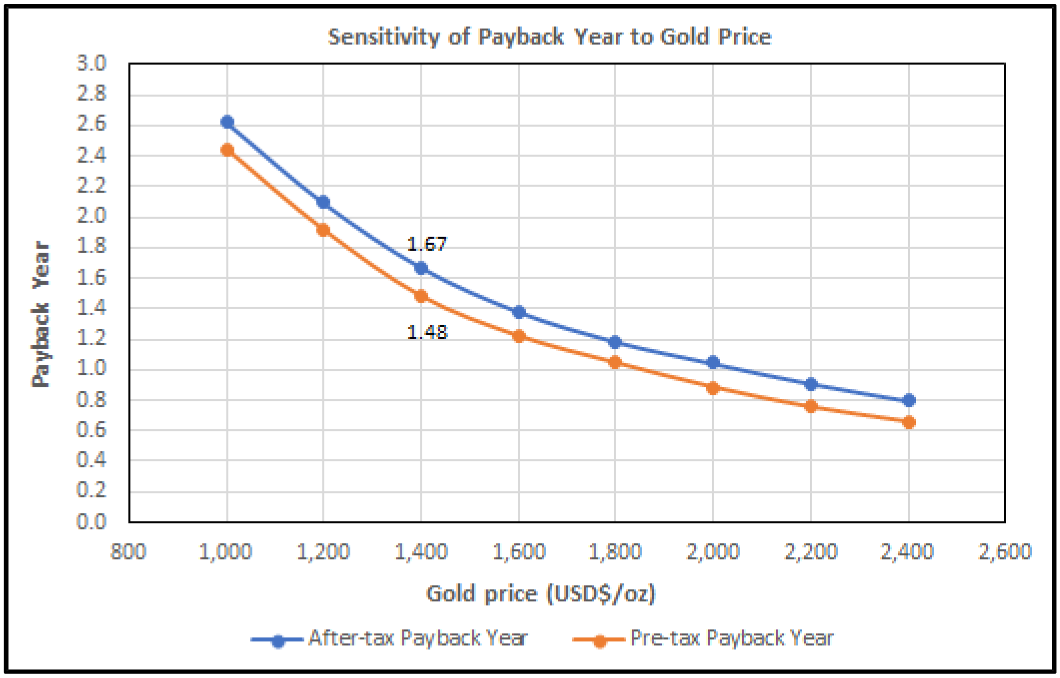


Figure 22-5: Payback year vs. gold price

23.0 ADJACENT PROPERTIES

There are three tenement or tenement applications adjacent to the Lion One Properties in the Sabeto Valley, namely the following summarized in Table 23-1.

Table 23-1: Tenement and tenement application list of adjacent properties

License No.	Name	License Holder	First Granted	Expiry Date	Notes
SPL 1360	Toge	Goldbasin Mining (Fiji) Pte Limited	June 18, 1993	October 17, 2024	-
CX861	Vuda	Ding Jin Mining Pte Ltd	Application	-	Ex SPL 1368
CX857	Nawaka	Akura Limited	Application	-	Petroleum

Source: Mineral Resources Department, Fiji (2020)

No work has been undertaken on these tenements (or tenement applications) since the previous holders relinquished the properties.

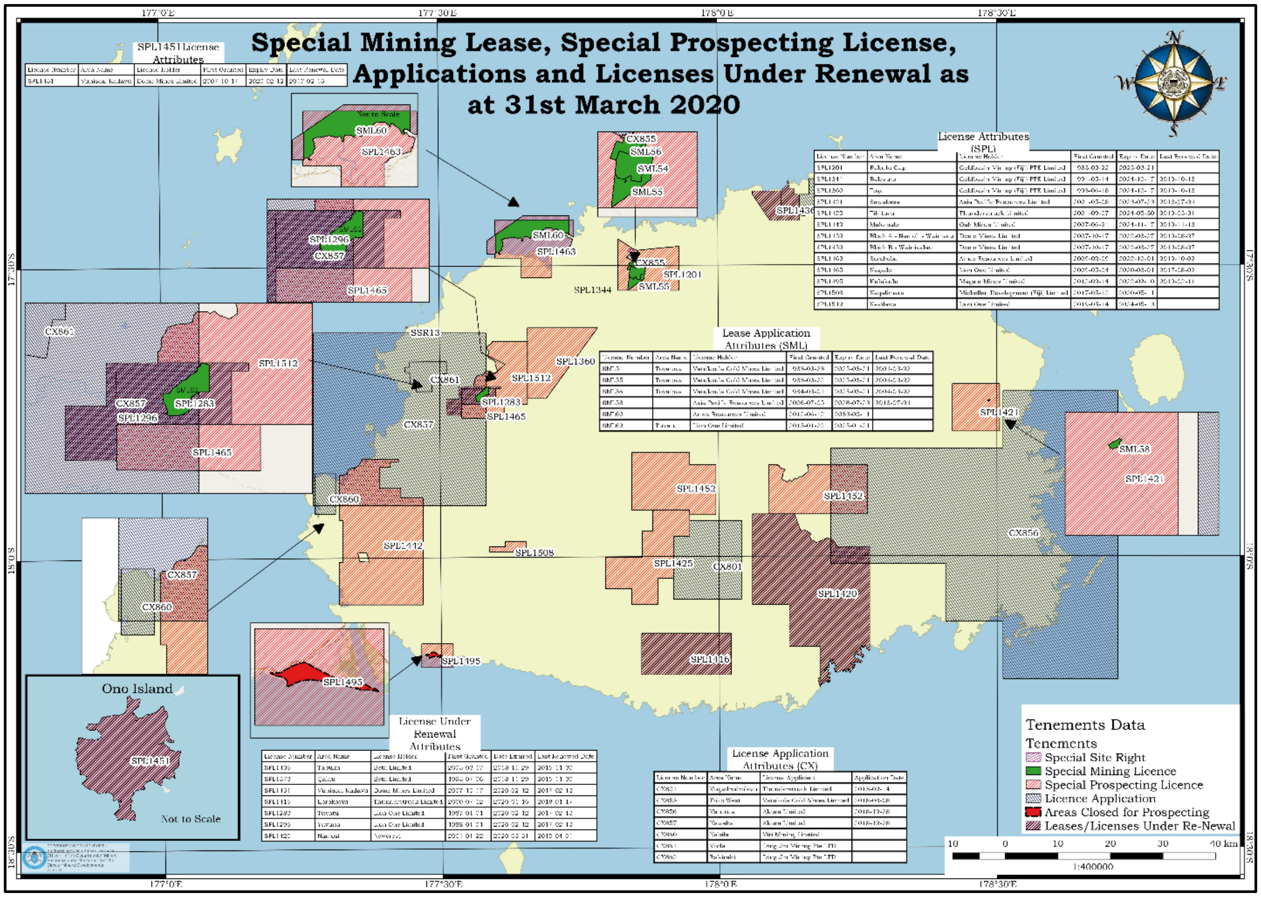
23.1 SPL 1360

SPL 1360 lies to the east and northeast of Lion One tenement SPL 1512. No work has been reported on this tenement in recent times.

23.2 Tenement Application CX861 (Formally SPL 1368)

Geopacific (a subsidiary of Geopacific Resources Ltd.) had been exploring the Vuda Prospect area targeting high-grade, vein-style epithermal/mesothermal gold. The Vuda Project consisted 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 approximately 85 km² (Figure 23-1). Geopacific describes 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 2005). Tuvatu is located 3 km east of these previous tenements. Exploration on the Sabeto/Vuda Project had focussed on the search for gold-copper porphyry mineralization, and Geopacific believed that Sabeto and Vuda have potential to host porphyry related gold-copper mineralization. In 2012, three deep (235 to 400 m) DDHs were completed on the Sabeto Porphyry Project. Geology within the drill holes confirmed observations made from surface mapping and sampling program 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 provided a vector towards potential porphyry-related gold-copper mineralization within an area around and to the south of one of those drill holes, SBDD001. This drill hole displayed the most proximal alteration and mineralization assemblage. While mineralization in SBD001 was seen to be associated with a syenite porphyry, it was thought that the actual mineralizing 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 DDHs to test the nature of the deeper mineralization. However, no further work was undertaken on this tenement area since 2013.



Source: Lion One (2020)

Figure 23-1: Tenement holdings in Fiji (March 2020)

24.0 OTHER RELEVANT DATA AND INFORMATION

A preliminary construction plan, including the construction elements required to successfully execute construction management for the Project has been developed. Figure 24-1 shows the preliminary construction schedule. The Project is expected to be completed in approximately one and a half years, including site preparation, mining development and pre-production, process plant construction and commissioning, and TSF construction.

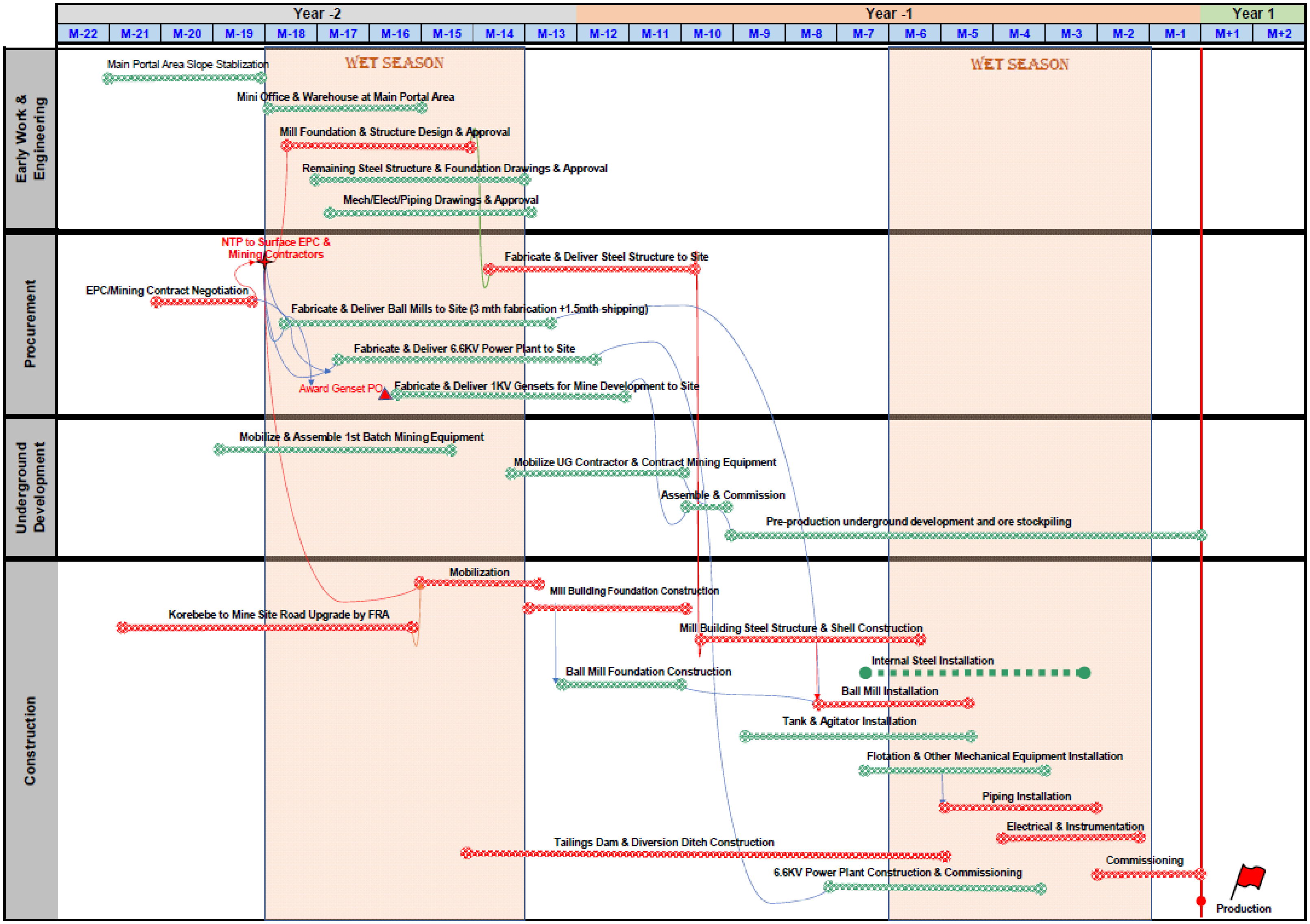


Figure 24-1: Preliminary Project construction schedule

TETRA TECH

25.0 INTERPRETATION AND CONCLUSIONS

The interpretation and conclusions shown in this section for this Technical Report are based on the 2020 PEA study. No update for the interpretation and conclusions has been conducted since the 2020 PEA.

25.1 Geology and Mineral Resources

Following the completion of the 2016/2017 diamond drilling program and field exploration, MA undertook a similar study to update the resources with the results of that drilling program and other work completed by Lion One to that date. In particular, drilling added significant additional information in the HT Corridor zone of mineralization (H and Tuvatu Lodes) and the Western Veins (which are interpreted to be the western extension of the Murau Lodes). Stricter parameters and tighter controls than those used for the 2014 estimate (which was put together for the 2015 PEA study) were used in this Technical Report. As a consequence of these tight controls, the Mineral Resource estimate related to some lodes was reduced in tonnes and/or grade.

The Property contains a sequence of volcaniclastic units intruded by a monzonite intrusion complex with the narrow epithermal style vein mineralization occurring as structurally controlled sets and networks of narrow veins and cracks. Gold mineralization is dominantly hosted in the monzonite units but also occurs in the adjacent volcanics. The local geology is quite well understood from recent drilling and underground mapping.

Lion One engaged MA on November 24, 2017 to prepare an updated Mineral Resource model suitable for mine design and scheduling of the resources for the Project. At Lion One's request, MA created a new model using additional data around the Tuvatu and H Lodes. The new drilling (New Surface Drilling Holes TUDDH-441 to TUDDH-471) has been used to update the Tuvatu global resource in this area of the Project. At a cut-off grade of 3 g/t Au, a total of 23,700 oz of gold was estimated as Inferred Resource for the H Lodes.

The Mineral Resource has been estimated for each vein individually based on the current drill hole database, historic block models, and geological wireframes. With a depletion of 3,500 t at 9.06 g/t Au from the existing development and a cut-off grade of 3 g/t Au, the total Indicated Mineral Resource is 1,007,000 t at 8.48 g/t Au for 274,600 oz of gold and an Inferred Mineral Resource of 1,325,000 t at 9.0 g/t Au for 384,000 oz of gold. The effective date for the Mineral Resource estimate is January 8, 2018.

25.2 Mining

Entech prepared a PEA-level mining assessment of the Project, which provides a basis for underground mine development. The mine operations will consist of narrow vein mining, which has resulted in constraints to mining productivity and the estimated costs being higher than for wide vein operations as expected. Mine planning is an ongoing activity at an operating mine, and as such, the mine plan, costs, and productivities will be updated as mining progresses.

The Project has a mineralization body amenable to longhole mining methods and has a positive economic outlook based on the mining work compiled by various consultants. Further work related to determining the economic viability of expanding the mineralized material drives and minimum mining width combined with additional geotechnical data is likely to show that the Project is viable.

25.3 Metallurgy and Processing

The processing facility has been designed as a 1,000 t/d on-site facility to produce gold doré based on metallurgical test work conducted to date. Significant metallurgical test work has been conducted on various samples. The tested gold recovery methods included whole-ore leaching, gravity concentration, flotation and gravity-flotation-cyanidation combined treatments. Preliminary pre-treatment prior to cyanidation, including intensive cyanidation, were also tested. The metallurgical test work indicates that there is a significant amount of the gold occurring in nugget gold form. The Tuvatu mineralization is amenable to gravity concentration and flotation followed by cyanidation processes.

The proposed process plant will consist of a three-stage crushing, two-stage grinding including a gravity concentration in the primary grinding stage to recover coarse-free gold grains, followed by flotation processes, and a respective cyanide leaching on both the flotation concentrates after regrinding and floatation tailings. Gold dissolved in the pregnant solutions will be extracted via a series of operations, including a common CIP adsorption, acid elution, electrowinning, and refining steps to produce the final gold doré. The leaching residues will be treated by SO_2/air to lower the WAD cyanide level to less than 1 ppm and then transferred to the TSF.

25.4 Project Infrastructure

The Property is accessible via existing roads and bridges, but a civil engineering upgrading will be required to allow future mining activities. Internal roads, including haul roads, services roads, and connection road from the mine site to the TSF will be constructed. The proposed on-site infrastructure for the Project will include:

- Access and site roads
- A process plant, consisting of gravity and flotation concentration followed by conventional CIP cyanidation and related gold recovery from loaded carbon
- Vehicle maintenance shop and warehouse
- A hybrid solar power and diesel power generation system (contracted)
- Electrical substations and power distribution system
- Potable and fire water storage and distribution system
- Site sewage treatment facilities
- Fuel storage and fueling station
- A TSF
- A geochemical and metallurgical laboratory located at the Nadi office site
- Administration offices

25.4.1 Site Geotechnical Investigations

The geotechnical investigations and some site preparation work (rough grading at the proposed process plant and crusher areas) were reviewed by Wood for this Technical Report. As detailed in Section 18.0, various geotechnical testing programs were conducted at the proposed process plant site, adjacent structures, and TSF site. The test results and the hydrogeological data were used for the conceptual TSF design. For the process plant site and crusher/screen structures, the locations have been selected by Lion One and have been under active development when preparing the geotechnical review work by Wood. The site preparation involved cut-and-fill earthworks with some perimeter slopes. However, the bearing capacity for foundation design of the processing plant is highly variable across the site. Wood recommends that the results of the geophysical survey program, together with the historic borehole results, should be used to define the extent of soil improvement, including the size (thickness and horizontal extent) of any engineered fill for the structures and foundation members individually.

25.4.2 Utilities

The utility facilities will provide water and electrical power consumed in mining, processing, and tailings management operations, as well as mobile phone and internet services. Site water will be sourced from reclaim water, runoff water, and mine dewatering. Additional make-up water may be required from the Sabeto River. The Project will generate its own power via a hybrid solar power generation system / containerized diesel power station. An overhead transmission line will be used to carry the solar power to the mine site. A new mobile phone and internet tower will be constructed and provide required services.

25.4.3 Buildings/Structures

The on-site buildings/structures will be constructed for administration, emergency medical services, general maintenance, warehouse, as well as for mining operation services, including mine dry, truck shop, and explosive magazines. No on-site accommodations will be required as proximity to Nadi, Latouka, and local villages can provide enough capabilities.

25.4.4 Tailings Storage Facility

The TSF has been designed to contain the generated tailings at a total capacity of 2,555 kt in dry solids mass, based on the dual regulations of the local government and the CDA. Higher capacity is achievable with substantially higher earthwork quantities. Various testing programs were conducted and indicated the tailings materials are PAG and/or ML. A tailings deposition plan has been included in this design for ARD management. The TSF dam will be constructed using centerline construction method with varied raising height each year. The associated facilities with the TSF will include a floating pump barge to send tailings water to the process plant, as well as a sedimentary control pond for water quality monitoring and potential seepage collection.

25.4.5 Site Water Management

A site water management plan has been developed for the Project to meet the requirement for a full coverage of the tailings to mitigate acid generation. Surface water diversions will be constructed for the TSF, process plant site, and ROM pad areas. The supernatant water from TSF will be monitored and treated to meet the release criteria before discharge. A sediment control structure will be constructed to reduce the impacts of construction activities.

25.5 Environmental

The studies conducted to date for the Project indicate that the aspects of greatest environmental concern are water quality and freshwater flora and fauna. Potential effects to these resources will be mitigated by engineering design, the CEMMP, and other management plans. Development of the project also has to adhere to the terms and conditions presented in the EIA approval documents issued by the Fiji DE that have been received to date (the first dated September 27, 2013 for the primary EIA and the second dated May 29, 2018 for the Tuvatu Creek realignment).

Lion One will continue to refine their management plans and Rehabilitation Plan in order to reduce potential effects on the biological and socio-cultural environment.

25.6 Capital and Operating Costs

The capital cost and operating estimates shown in this section for this Technical Report are based on the 2020 costing. No update for cost estimates has been conducted since the 2020 PEA.

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project, including mining pre-production, is USD$66.8 million (CDN$89.1 million). This total includes all direct costs, indirect costs, Owner's costs, and contingencies. A summary breakdown of the initial capital cost is provided in Table 25-1.

Table 25-1: Capital cost summary

Description		Capital Cost Estimate (CDN$ million)	Capital Cost Estimate (USD$ million)
Direct Costs			
10	Overall Site	4.3	3.2
25	Underground Mining	27.8	20.8
40	Process	18.2	13.7
50	TSF	5.4	4.1
70	On-site Infrastructure	2.4	1.8
Direct Cost Subtotal		**58.1**	**43.6**
Indirect Costs			
X	Project Indirect Costs	15.4	11.5
Y	Owner's Costs	6.4	4.8
Z	Contingency	9.2	6.9
Indirect Cost Subtotal		**31.0**	**23.2**
Total Capital Cost Estimate		**89.1**	**66.8**

Note: Numbers may not total due to rounding.

On average, the on-site operating costs for the Project were estimated to be USD$97.35/t of material processed. The operating costs included mining, processing, surface services, and G&A costs, excluding gold doré shipping and refining charges. The operating costs by area are summarized in Table 25-2.

Table 25-2: Operating cost summary

Description	Operating Cost (CDN$/t milled)	Operating Cost (USD$/t milled)
Mining**	62.99	47.24
Process	55.33	41.49
Reclaim Water Handling	0.40	0.30
G&A	8.88	6.66
Site Services	2.21	1.66
Total Operating Cost Estimate*	**129.81**	**97.35**

Notes: *Numbers may not total due to rounding.
**LOM average, excluding pre-production related costs.

25.7 Economic Analysis

The financial analysis data shown in this section for this Technical Report are based on the 2020 costing. No update for the financial analysis has been conducted since the 2020 PEA.

A PEA is preliminary in nature and includes Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the Technical Report will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

A preliminary economic evaluation was prepared for the Project based on a pre-tax financial model for the five-year LOM and 1.348 Mt of mill feed tonnage. The following pre-tax financial parameters were calculated using the base case gold price of USD$1,400/oz and exchange rates shown in Table 21-1:

- 60.3% IRR
- USD$155.8 million (CDN$207.7 million) NPV at a 5% discount rate
- 1.5-year payback on USD$66.8 million (CDN$89.1 million) of initial capital

Lion One prepared a tax model for the after-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes. The following after-tax financial results were calculated:

- 50.9% IRR
- USD$121.7 million (CDN$162.2 million) at a 5% discount rate
- 1.7-year payback on USD$66.8 million (CDN$89.1 million) of initial capital

Sensitivity analyses were conducted to analyze the sensitivity of the Project financial parameters (NPV, IRR, and payback periods) to changes in gold price, operating costs, capital costs, and exchange rate at a 5% discount rate. The analysis indicates that the Project is most sensitive to gold price. The Project's pre-tax and after-tax NPV and IRR were found to be most sensitive to gold price and Fiji currency exchange rate followed by capital costs (IRR) and on-site operating costs (IRR). The payback period was found to be most sensitive to gold price followed by Fiji currency exchange rate, capital costs, and on-site operating costs.

25.8 Risks and Opportunities

There are numerous risks and opportunities that influence any mining venture, and as such, there are also risks and opportunities for the Project. External factors such as fluctuations in metal prices and exchange rates are not within control of the Project, while other risks are usually associated with insufficient technical information, such as resource estimate, underground mining associated issues (underground geotechnical and hydrological conditions), unforeseen weather conditions, and gold recovery projections. Other factors that may impact the project economics include permit acquisitions and local skilled labour sources.

Potential opportunities for further improvement of the Project economic viability may include an increase in mineral resources with further exploration, further improvement in gold extraction, gold recovery flowsheet optimization, and mine plan optimization. Potential silver value, which may receive some credits as by-product, has not been included in the Project economic evaluation.

26.0 RECOMMENDATIONS

The recommendations shown in this section for this Technical Report are based on the 2020 study. No update for the recommendations has been conducted after the 2020 PEA, excluding the budget spent shown in Table 26-1 since the 2020 PEA.

The Project is considered economically viable based on this Technical Report. Approvals have been received for initial and updated environmental assessments. It is recommended to advance the Project to the next stage. Meanwhile, the following activities are suggested to further improve the Project economics and/or lower the Project risks. Total cost for future recommended work is approximately USD$7.7 million; Table 26-1 shows the cost breakdown by discipline for the budgets estimated by the 2020 study and spent amounts by April 2022 as provided by Lion One.

Table 26-1: Estimated costs for future work

Area	Budget Amount (USD$)	Spend Amout ($USD)	% of Budget
Geology and Mineral Resources	5,210,000	3,450,000	66%
Mining	352,950	290,000	82%
Mineral Processing and Metallurgical Testing	300,000	105,000	35%
General Site Infrastructure	112,500	140,000	124%
Plant Site Geotechnical Design	75,000	40,000	53%
TSF Design, including Foundation Design	1,275,000	350,000	27%
Geochemical Testing for Waste Rocks and Tailings	225,000	30,000	13%
Hydrology and Plant Site Water Balance	38,300	15,000	39%
TSF Site Water Balance	37,500	7,000	19%
Environmental	75,000	40,000	53%
Total	**7,701,250**	**4,467,000**	**58%**

26.1 Geology and Mineral Resources

The following recommendations have been made after this review of the technical data:

- Continue aggressive confirmation drilling of the deeper enriched portions of the Tuvatu Vein system and discovery of deeper rooted feeder structures; drilling may include wedge holes as veins extend over 600 m below the surface.
- Critically assess Inferred Resources within the mine design (either infill drill to bring resources up to Indicated or assess access from proposed underground development), and consider suitable drill platforms that can be constructed in advance of production.
- As Lion One develops a mine at Tuvatu, exploration drilling would continue to expand the current identified Mineral Resources present.

The estimated cost for the recommended work is approximately USD$1,763,800.

26.2 Exploration

In 2019, Lion One consolidated ownership of the extensive mineral system situated inside and peripheral to the Navilawa Caldera. Surface exploration is challenged by steep terrain, and the distribution of gold in the near surface environment is not necessarily indicative of the mineralized structures or gold endowment in the subsurface. Nevertheless, Lion One has successfully delineated a far larger surface footprint than previously understood, through analysis of the project area using stream sediment bulk leach extractable gold analysis in the drainage system, an effective method of studying the distribution of fine particle gold over large areas in this type of environment. Since acquiring the balance of the surrounding Caldera in 2019 through the grant of SPL 1512, Lion One has identified multiple high quality exploration targets by cross referencing its database of extensive high grade surface samples with deep coincident resistivity gradients highlighted from CSAMT survey. This technique has been utilized successfully in other jurisdictions to explore and expand major alkaline gold systems, and demonstrates that the Navilawa system has the potential to host additional gold deposits of equal if not greater size and scale than that of Tuvatu alone.

Little drilling has taken place outside the Tuvatu resource area itself. Lion One's cBLEG program objectively confirmed that there are several other catchments beyond Tuvatu that host significant mineral systems. The surface sampling and reconnaissance has highlighted multiple prospects with the potential for other significant deposits. Lion One's current exploration strategy involves:

1. Deep drilling for extensions and high-grade feeder zones to Tuvatu identified from the CSAMT and recent work. The estimated budget is approximately USD$1,763,800.

2. Infill CSAMT sampling as follow-up to the successful 2019 sampling program. The estimated budget is approximately USD$200,000.

3. District-scale benching, mapping, and sampling across multiple prospects based on results of cBLEG sampling, previous geological mapping, CSAMT, and projected extensions to existing mineralization at Tuvatu. The estimated budget is approximately USD$296,500.

4. Prioritisation of shallow to deep drill targets based on CSAMT geophysics (combined with geochemistry and geology). The estimated budget is approximately USD$501,600.

Of the district targets, the following are considered high-priority:

1. Matanavatu – With confirmed high-grade structure in weathered rock and potentially deep feeder structures identified on CSAMT.

2. Banana Creek – Pending review of scout drilling, re-mapping, and sampling program, with perhaps the best targets at deeper than 200 m beneath surface.

3. Jomaki-Ura-Kuba – Based on CSAMT, the previous drilling did not drill deep enough. Holes deeper than 400 m are recommended.

4. Upper Qalibua – Extensive anomalism in highly leached rocks, with strong tellurium-copper-silver pathfinders for alkaline-type systems.

The estimated cost for work on the district targets is approximately USD$998,300.

Lion One's drilling objectives are twofold: confirm proof of concept that the Navilawa Caldera could host a major alkaline gold system in excess of 10 million ounces, whilst developing an economically viable high grade mining operation through aggressive exploration and expansion of the current resource.

26.3 Mining

The following recommendations are made for mining at Tuvatu:

- A review of minimum mining widths should be conducted. Lion One should evaluate the risk of having to mine at wider stope widths on the Project economics.
- Further geotechnical studies should be carried out to improve confidence in rock mass rating as well as other geotechnical criteria. This information should be considered with the chosen mining method, as the original AMC report was concerned with handheld shrinkage / breast stoping and not longhole stoping / mechanized cut and fill.
- The proposed mining operations will consist of narrow vein mining, which has resulted in constraints to mining productivity and the estimated costs being higher than for wide vein operations as expected. Mine planning is an ongoing activity at an operating mine, and as such, the mine plan, costs, and productivities will be updated as mining progresses.

The costs for conducting the underground geotechnical investigations and the mine plan studies are estimated to be USD$352,500, including USD$258,000 for geotechnical related work.

26.4 Metallurgy and Processing

The following recommendations are made for metallurgy and processing at Tuvatu:

- Further metallurgical test work, including mineralogical evaluations, are recommended to optimize the flowsheet. The test work should further confirm metallurgical performance for various variability samples, including representative samples for initial two year mill feeds and various lithological domains. Optimization of treatment methods for gold recovery of the flotation concentrate should be investigated to improve gold recovery and project economics. Some of the design-related data, such as thickening rates, should be further confirmed. The estimated cost of these programs is USD$187,500, inclusive of sample shipping and short-term storage costs. Costs for sample collection (i.e., drilling) are not included, as this recommendation assumes samples would be derived from core stored on site or ongoing exploration drilling programs.
- The process equipment sizing and plant arrangement may need to be further reviewed and optimized to reduce process plant construction related costs. This plant optimization processes will be part of ongoing activities. The estimated cost is approximately USD$112,500.

26.5 Project Infrastructure

26.5.1 Overall Plant Infrastructure Update

The overall site infrastructure layout should be updated to optimize the infrastructure design and better define the related capital costs, including detailed construction schedule and construction laydown areas. The cost related to relevant the design work is estimated to be approximately USD$112,000.

26.5.2 Plant-site Structures

The following recommendations are made for the plant-site structures at Tuvatu:

- Based on information obtained during several geotechnical investigations, extensive removal of unsuitable soils will be required followed by the placement of well-compacted, engineered fill to support foundation structures and floor slabs with adequate safety.
- Specific geotechnical design and foundation preparation recommendations are required for each major structure at the next phase of studies. The cost to carry out this assessment is estimated at approximately USD$75,000.

26.5.3 Tailings Storage Facility and Site Foundations

The following recommendations are made for the TSF and site foundations at Tuvatu:

- Limited baseline information is available. By establishing comprehensive baseline conditions, it will determine if the discharge water quality meets design criteria using Australian and Canadian guidelines. Currently no available information is available to support the development of water quality estimates and identify potential water quality management (i.e., treatment) needs. The cost to establish proper baseline conditions for subsequent studies is estimated at approximately USD$150,000.
- It is recommended that flow monitoring stations and sampling locations be placed at the upstream and downstream of the Savuskia Creek entering Sabeto River. Flow rates and samples should be recorded before and after the construction of the TSF to establish baseline conditions and to ensure results after constructions are within acceptable guideline limits. The cost to establish flow monitoring and sampling station for subsequent studies is estimated at approximately USD$75,000.
- Geochemical testing for waste rock is currently limited to one sample, which had an uncertain acid generating potential. The ML potential of waste rock is not known. Geochemical testing for other rock from the TSF basin has not been conducted. Additional testing of representative waste rock and other rock materials is required to assess risks and opportunities related to ML/ARD and the potential use of these materials for TSF construction. The use of PAG/ML mine rock (Zone 2A) on the downstream side of the dam body is not recommended. A prefeasibility-level geochemical characterization study will be required to understand the risks and opportunities related to tailings and construction material (e.g., mine rock, other rock, aggregates) ML/ARD potential and associated considerations for the TSF design, operation, and closure. These studies have been proposed in a separate memorandum. The preliminary cost to perform such sampling and test work is estimated at approximately USD$225,000.
- The current design has some assumptions and can be optimized with the support of additional technical and economic analyses and trade-off studies, such as comparison of soft soil removal versus stabilization toe berms and on-site filter preparation versus off-site sand hauling. The cost to carry out design optimization with trade-off studies for subsequent assessment is estimated at approximately USD$75,000.
- Additional dam foundation soil classification with field investigation and strength verification testing and tailings consolidation behaviour are needed to optimize TSF dam and storage sizing. The cost to carry out such classification and optimization is estimated at approximately USD$375,000.
- The present Technical Report contains the PEA update study design of the TSF; a prefeasibility, feasibility, and a detail design of TSF dams are required, including detailed dam material specifications and construction drawings. The cost to carry out these studies and design is estimated at approximately USD$600,000.

26.5.4 Hydrology and Site Water Balance

26.5.4.1 Hydrology and Process Plant Site Water Balance

Further plant site hydrological studies should be conducted to further evaluate the site hydrology. The recommended work includes:

- Updating climate data and further assessing the site hydrology
- Conducting detailed diversion ditch design, including preparation of detailed design drawings and updating material take-off estimation and cost estimation
- Conducting detailed culvert design, including preparation of detailed design drawings and updating material take-off estimation and cost estimation
- Updating site water balance
- Preparing design report

The estimated cost for completing the recommended work is USD$38,300.

26.5.4.2 TSF Water Balance

The current water balance represents the first two years of operation only. It is recommended that the water balance be updated to assess start-up water management and to represent continuous operations over LOM. This should include an assessment of sequential wet and dry years to ensure that TSF water inventory can be maintained in the design range under a range of expected climatic conditions. It should also include sensitivity analysis for key parameters. The cost to carry out this study is estimated at approximately USD$37,500.

26.6 Environment

Further refining environmental management plans and rehabilitation plan should be conducted in order to reduce potential effects on the biological and socio-cultural environment. The cost associated with further environmental work is estimated to be approximately USD$75,000. Further hydrological and geochemical characterization investigations should be conducted. The associated costs are estimated in the sections above.

27.0 REFERENCES

27.1 General

Wang et al. (2020). Preliminary Economic Assessment (PEA) Update (Technical Report), Tuvatu Gold Project, Lion One Metals. Effective Date September 25, 2020.

Emperor Mines Limited. (2003). Tuvatu Gold Project – Summary report. Internal Report.

Emperor Mines Limited. (2005). 2005 annual report. Published report to the Australian Stock Exchange.

Emperor Mines Limited. (September 2005). Tuvatu Gold Project – Summary report. Internal Report. Fiji Bureau of Statistics, 2007 Census.

Freudigmann et al. (2015). Lion One Metals Tuvatu Gold Project preliminary economic assessment. Effective Date June 1, 2015.

27.2 Property Description, History, and Geology

A-Izzeddin, D. (1997 & 1999). *The Tuvatu Gold Project*. Internal Reports. Tuvatu Gold Mining Company Pty Ltd.

A-Izzeddin, D. (2000). Powerpoint presentations. Tuvatu Gold Mining Company Pty Ltd. Alcaston Mining NL. October 30, 2006, Prospectus.

Hatcher, R. (1997). *The Tuvatu epithermal gold prospect, Upper Sabeto Valley, Viti Levu, Fiji*. Unpublished Summary Report on Honours Thesis. Queensland University of Technology, Queensland Australia.

Hathway, B. (1993). The Nadi Basin: Neogene strike-slip faulting and sedimentation in a fragmented arc, Western Viti Levu, Fiji. *Journal of the Geological Society, London*, 150, 563-581. 11 figs printed in Northern Ireland.

Lion One Limited, Fiji. SPL1283, Compensation Agreement between Lion One Limited and the Mataqali Nacokula, Rawawaqa, Naboucawa, Nabouwalu, Natauvesi, Vadalo, Nawaqali and Natubakitoga. July 2013.

Mineral Resources Department, Fiji. (2020). Renewal of special prospecting license for Lion One Limited – SPL 1283 Bebe. Letter, September 4, 2020.

Mineral Resources Department, Fiji. (2020). Renewal of special prospecting license for Lion One Limited – SPL 1296 Yavuna. Letter, September 4, 2020.

Scherbarth, N. L. & Spry, P. G. (2006). Mineralogical, petrological, stable isotope and fluid inclusion characteristics of the Tuvatu gold-silver-telluride deposit, Fiji: Comparisons with the Emperor Deposit. *Economic Geology*, *101*, 135-158.

Tuvatu Gold Mining Co Ltd. (August 2000). *In-house feasibility study*. Internal Report.

Vigar & Associates Brisbane. (May 2000). Geology overview & resource estimate, Tuvatu Project, Fiji. Prepared for Tuvatu Gold Mining Co Ltd.

Vigar, A.J. (March 2009). *A technical review of exploration and resource estimates of the Tuvatu Gold Deposit Viti Levu, Republic of Fiji*. Internal report prepared by Mining Associates Pty Ltd for American Eagle Resources Inc and X-Tal Minerals Corp, dated March 30, 2009.

27.3 Deposit Types and Exploration

Baker, E. M. (1989). *The bearing of fluid inclusion and isotope results on the economic potential of the Tuvatu Prospect, Fiji. Emperor Gold Mines. Fiji*.

Canadian Institute for Mining, Metallurgy and Petroleum. (2014). *CIM Definition Standards for Mineral Resources & Mineral Reserves*.

Hedenquist, J. W. (2000). Exploration for epithermal gold deposits. *Society of Economic Geology Reviews*, *3*, 245-277.

Hennigh, Q. (2019). Communication to executives at geological during site visit March 2019. Lion One Metals Ltd.

Holden, D. (2019). *Evidence assessment of an alkali gold system at Tuvatu*. Internal Company Report, April 1, 2019.

Holden, D. (2018). Geospy Report #12, November 2018 *Tuvatu Gold Project: Magmatic intrusion-related gold and implications for extensional and regional exploration*. Lion One Ltd. Lion One.

Inclusion characteristics of the Tuvatu gold-silver telluride deposit, Fiji: Comparisons with the Emperor deposit. *Economic Geology*, *101*, 135–158.

Jensen, E. P. & Barton, M. D. (2000). Gold deposits related to alkaline magmatism. *Society of Economic Geology Reviews*, *13*, 279-314.

Kelley, K. D. & Luddington, S. (2002). Cripple Creek and other alkaline-related gold deposits in the Southern Rocky Mountains, USA: Influence of regional tectonics. *Mineralium Deposita*, *37*, 38-60.

Lion One Limited, Fiji. SPL's 1283 and 1296, Tuvatu. Annual report (1st draft), Exploration Report for the Period January 2008 to December 2008, Annual Report to Fiji Mineral Resources Department.

Mining Associates Pty Ltd. (March 2009). *A technical review of exploration and resource estimates of the Tuvatu gold deposit, Viti Levu, Republic of Fiji*.

Mining Associates Pty Ltd. (2014). *Independent technical report and resource estimate on the Tuvatu gold deposit*. Available Mining Associates Pty Ltd, Level 4, 67 St Paul's Terrace, Spring Hill QLD.

Mining Associates Pty Ltd. (January 2018). Tuvatu Gold mineralisation model. Document No. MA1708-2-3, January 8, 2018.

Ministry of Lands and Mineral Resources. (1997). Investment guide. Fiji Mineral Resources Department.

Ministry of Lands and Mineral Resources. (1998). Airborne geophysical signatures of localities of epithermal gold deposits in Fiji. Fiji Mineral Resources Department.

P&E Mining Consultants Inc. (October 2010). *Technical report and resource estimate on the Tuvatu Gold Property, Viti Levu, Fiji*.

Planchuelo-Ravelo A.M. (September 1984). *Agroclimatic survey of southern Pacific Island groups: Fiji, Gilbert Islands (Kiribati), Solomon Islands, Tonga, Vanuatu – final report*. USAID.

Richards, J. P., Bray, C. J., Channer, D. M. D., & Spooner, E. T. C. (1997). Fluid Chemistry and Processes at the Porgera Gold Deposit. *Mineralium Deposita*, (32), 119-132.

Rodda, P. (May 1967). Outline of the geology of Viti Levu. Fiji Geological Survey, Suva, May 1967.

Scherbarth, N. L. & Spry, P. G. (2006). Mineralogical, petrological, stable isotope and fluid inclusion characteristics of the Tuvatu gold-silver-telluride deposit, Fiji: Comparisons with the Emperor Deposit. *Economic Geology*, *101*, 135-158.

Schwartz, M.O. (1986). Interpretation of the Conolly contour diagram of fault-related veins. *Mineralium Deposita*, *21*(1), 1-11.

Sillitoe, Richard H & Thompson, John (1999). Intrusion-related vein gold deposits: Types, tectono-magmatic settings and difficulties of distinction from orogenic gold deposits. *Resource Geology*, *48*(2), 237-250.

Sillitoe, Richard H. (2002). Some metallogenic features of gold and copper deposits related to alkaline rocks and consequences for exploration. *Mineralium Deposita*, *37*, 4-13.

Spry, P. (2007). A genetic relationship among epithermal gold tellurides, calderas, and the Viti Levu lineament, Fiji, circum-pacific tectonics, geologic evolution, and ore deposits, *A symposium in honor of William R. Dickinson*, Tucson, Arizona, Arizona Geological Society.

Spry, P.G. & Scherbarth, N.L. (2002). The vanadium-tellurium-gold association at the epithermal Tuvatu gold deposit, Fiji: Implications for ore deposition [abs.]: Geological Society of Australia, Abstracts 67, p. 342. Mineralogy and Petrology (2006) 87: 171–186.

Tawake, A.K. (January 2005). *Technical report on geological and geo-technical investigations of the semo quarry and selected aggregate sources in South and Western Viti Levu, Fiji*. South Pacific Applied Geoscience Commission.

Vigar, A.J., (May 2005). *Epithermal gold*. Internal Report. Mining Associates Pty Ltd.

Vigar, A.J. (March 2009). *A technical review of exploration and resource estimates of the Tuvatu Gold Deposit Viti Levu, Republic of Fiji*. Internal report prepared by Mining Associates Pty Ltd for American Eagle Resources Inc and X-Tal Minerals Corp, dated March 30, 2009.

Weis, T. (2019a). Review of 3-D IP/resistivity inversion models for the Tuvatu Project, Fiji. Internal Company Report for Lion One Ltd.

Weis, T. (2019b). Tuvatu Fiji Project, Review of 3-D IP/resistivity inversion models for the Tuvatu Project. Fiji, August 15, 2019. Thomas V. Weis & Associates, Denver, CO, USA.

Weis, T. (2020a). *Tuvatu Fiji Project, CSAMT geophysical report*. Internal Company Report for Lion One Ltd.

Weis, T. (2020b). *Tuvatu Fiji Project, CSAMT geophysical report*. Report prepared for Lion One Ltd, February 22, 2020. Thomas V. Weis & Associates, Denver, CO, USA.

27.4 Mineral Processing and Metallurgical Testing

ALS Metallurgy. (February 2016). *Metallurgical testwork conducted upon a bulk sample from the Tuvatu Project for Lion One Limited*. Report No. A16743.

Amdel Limited Mineral Services. (April 2000). Comminution tests – Tuvatu Project.

AMMTEC Ltd. (January 1998). *Comminution tests – Tuvatu ore*. Report No. N8448.

AMMTEC Ltd. (December 1997). *Metallurgical testwork conducted upon samples of ore from the Tuvatu gold deposit for the Emperor Gold Mining Company Ltd*. Report A5922.

AMMTEC Ltd. (June 1997). *Comminution testwork conducted upon samples of ore from Fiji Tuvatu gold deposit for Emperor Gold Mining Company Limited, Fiji*. Report No. A5724.

APSAR Applied Petrologic Services & Research. (February 2018). Petrologic studies of drill core from the Tuvatu precious metal deposit/prospect, Viti Levu, Fiji. APSAR 77002.

Beneficiation Branch of Yantai Oriental Metallurgy Design and Research Institution. (April 2015). *Beneficiation of Fiji Gold Ores Test Report*. Report No. 1513.

Bureau Veritas Commodities Canada Ltd. (June 2019). *Phase II metallurgical testing report – The Tuvatu Gold Project, Fiji*. Project No. 1801807 & 1803310.

Bureau Veritas Commodities Canada Ltd. (September 2018). *Preliminary metallurgical testing of a sample from the Tuvatu Gold Project*. Fiji, Project No. 1801004.

Gekko Systems Pty Ltd. (June 2012). Python – GRG – Leach amenability test work update, 2 stage CGR and flotation results.

Gekko Systems Pty Ltd. (July 2012). Lion One *Testwork report, GRG, CGR, VSI and leach amenability*. Report No. T0886.

Gekko Systems Pty Ltd. (July 2012). *Flotation optimisation & final product analysis*. Report No. T0941.

Hamish, G. (August 2012). Memo – Tuvatu Project metallurgical studies, observations, conclusions, recommendations.

Metcon Laboratories Pty Ltd. (May 2000). Metallurgical development testwork on Tuvatu drill core samples – variability response tests.

Met-Solve Laboratories Inc. (June 2020). *MS1944 Lion One Metals report*. Report No. MS1944.

Orway Mineral Consultants. (March 1998). *Tuvatu Project – Redesign of the grinding circuit following evaluation of Navisi Adit Ore*. Report No. 38075.

Orway Mineral Consultants. (December 1997). Batch Treatment Campaign.

Yantai Xinhai Mining Research & Design Co., Ltd. (January 2018). *Brief Report of Preliminary Test Work on Tuvatu Gold Mill in Fiji*.

Yantai Xinhai Mining Research & Design Co., Ltd. (February 2018). *Interim Test Work Report on Tuvatu Gold Mill in Fiji* (in Chinese).

Yantai Xinhai Mining Research & Design Co., Ltd. (March 2018). *Mineral Processing Test Report on Tuvatu Gold in Fiji*.

27.5 Mining Methods

AMC Consultants Pty Ltd. (November 2017). *Tuvatu Gold Project geotechnical assistance*, Lion One Metals Limited, 9 Havelock Street, West Perth, AMC Project 717028, November 01, 2017.

AMC Consultants Pty Ltd. (2000). *Tuvatu Gold Project geotechnical assessment*. Available, 9 Havelock Street, West Perth.

Entec Limited Engineering & Science Consultants. (July 2017). Geotechnical engineering investigation for Tuvatu mine tailings processing plant, Sabeto, Nadi for Lion One Metals Limited. Reference No.: 0320217, July 04, 2017.

Entech Mining Ltd. (July 2018). *Tuvatu – Life of mine plan update*. Report Number: ENT_0420_LIO, July 2018.

Knight Piésold Pty Ltd. (2014c). Tuvatu Project technical memorandum – preliminary groundwater assessment KP, PE14-01018, dated October 14, 2014.

27.6 Project Infrastructure

27.6.1 General

Canadian Dam Association. (2014). Canadian Dam Association (CDA) standards.

National Building Code of Fiji (1990).

Watkins, L. & Fiddes, D. (1984). Highway and urban hydrology in the tropics. Pentech Press, London.

World Meteorological Organization (WMO). (2009). *Manual on estimation of probable maximum precipitation (PMP)*. WMO-No. 1045, Geneva, Switzerland, 2009.

Yantai Oriental Metallurgy Design and Research Institution. (2018). Various plant site layouts.

27.6.2 Geotechnical and Tailings Storage Facility

Australian National Committee on Large Dams (ANCOLD). (2012). Guidelines on tailings dams, planning, design, construction, operation and closure.

Australian Standards. (2001). Concrete structures, AS 3600.

GBG MAPS Pty Ltd. (September 2018). Geophysical investigation for determination of in-situ seismic wave velocities at a proposed tailings storage facility, Tuvatu Gold Mine, Viti Levu Fiji. Report Ref: 70438_TSF, September 18, 2018.

Knight Piésold Pty Ltd. (2014a). Tuvatu Project technical memorandum – preliminary seismic hazard assessment, PE14-00926, dated September 19, 2014.

Knight Piésold Pty Ltd. (2014b). Tuvatu Project technical memorandum – preliminary assessment of ore geochemistry, PE14-00981, dated October 6, 2014.

Qingdao Geotechnical Investigation and Surveying Research Institute. (October 2018). The geotechnical investigation for Tuvatu Mine processing plant. Reference No: K2018-213.

SMEC, Surbana Jurong Group. (September 2017). Hydrologic investigations. Reference No: 5035014, September 29, 2017.

Wood Environment & Infrastructure Solutions, Wood Canada Limited. (June 2018). Technical memo – geotechnical review – process plant area. Ref: TC180305, June 25, 2018.

Wood Environment & Infrastructure Solutions, Wood Canada Limited. (March 2019). *Tailings storage facility prefeasibility study design report*. TC 180305, March 2019.

27.6.3 Hydrology

SMEC Australia Pty Ltd. (September 2017) Tuvatu gold mine hydrologic investigations. Reference No: 5035014.

27.7 Environmental

Argo Environmental Ltd. (2014). *Tuvatu Project report – Study of operational environmental management & monitoring plan, 2014*.

27.8 Adjacent Properties

Geopacific Resources. (2005). Prospectus; Australian Stock Exchange. Geopacific Resources. 2014, Company website.

Golden Rim Resources Ltd. Annual reports 2008; 2009, Australian Stock Exchange.

28.0 CERTIFICATES OF QUALIFIED PERSONS

Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM

I, Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM do hereby certify:

- I am a Director and Principal Consultant with GeoSpy Pty Ltd. with a business address at 7d Amherst Street, Fremantle, Western Australia 6160.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic AssessmentUpdate for the Tuvatu Gold Project, The Republic of Fiji" with the effective date of April 29, 2022 (the "Technical Report").
- I am a graduate of the University of Western Australia (Bachelor of Science with honours, Geology, 1994) the University of Notre Dame Australia (Doctor of Philosophy, 2019). I am a Fellow in good standing of the Australasian Institute of Mining and Metallurgy (F.AusIMM, #226201).
- My relevant experience with respect to mineral exploration geology includes more than 27 years of involvementas both a consultant and employee on various companies. I have worked as a mine-geologist in Western Australia, geological modeller and as an exploration geologist, including holding positions in companies such as Vice President Geoscience, and Chief Executive Office, for exploration to discovery stage projects in Canada, the United States, Australia, and the Pacific.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I visited the Property that is the subject of this Technical Report on February 16 to 23, 2020; December 1 to 7,2019; October 27 to November 3, 2019; September 18 to 30, 2019; June 29 to July 7 2019; April 7 to 14, 2019;March 7 to 15, 2019; and a total of 12 other times, for about 7 to 10 days each time, between 2017 to 2018. The purpose of the visit is to conduct exploration, mapping, data reviews, reporting, and develop sampling protocols
- I am not independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- My previous experience with the Property that is the subject of this Technical Report includes providing advisory services to Lion One Metals Limited for approximately 5 years. I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 2.2.2, 8.0, 9.0, 12.3.2, 12.4.2, 26.2, and 27.3 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of April 2022.

(Signed) *"Darren Holden"*

Darren Holden, B.Sc. (Hons), Ph.D., F.AusIMM
Director and Principal
GeoSpy Pty Ltd

Davood Hasanloo, M.Sc., M.A.Sc., P.Eng.

I, Davood Hasanloo, M.Sc., M.A.Sc., P.Eng., do hereby certify:

- I am a Water Resources Engineer with Tetra Tech Canada Inc. with a business address at Suite 1000 – 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled “Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji” with the effective date of April 29, 2022 (the “Technical Report”).
- I am a graduate of the University of British Columbia (M.A.Sc., 2013). I am a member in good standing of Engineers and Geoscientists British Columbia (#42950). My relevant experience with respect to water resources engineering includes more than 10 years of involvement in hydrology and hydraulic analysis and design, mine surface water management, and numerical modeling.
- I am a “Qualified Person” for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Property that is the subject of this Technical Report.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 1.8.4, 12.4.7, 18.5, 26.5.4.1, and 27.6.3 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed, sealed, and dated this 29th day of April 2022.

(Signed and Sealed) *“Davood Hasanloo”*

Davood Hasanloo, M.Sc., M.A.Sc., P.Eng.
Water Resources Engineer
Tetra Tech Canada Inc.

Hassan Ghaffari, P.Eng., M.A.Sc.

I, Hassan Ghaffari, P.Eng., M.A.Sc., do hereby certify:

- I am a Director of Metallurgy with Tetra Tech Canada Inc. with a business address at Suite 1000 – 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled “Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji” with the effective date of April 29, 2022 (the “Technical Report”).
- I am a graduate of the University of Tehran (M.A.Sc. Mining Engineering, 1990) and the University of British Columbia (M.A.Sc. Mineral Processing Engineering, 2004). I am a member in good standing of Engineers and Geoscientists British Columbia (#30408). My relevant experience with respect to mining and processing engineering includes more than 27 years of involvement in mining and plant operation, project studies, management, and engineering.
- I am a “Qualified Person” for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Property that is the subject of this Technical Report.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 1.8 (except 1.8.1, 1.8.3, and 1.8.4), 1.10 (except 1.10.2), 18.1, 18.2, 18.4, 18.6, 18.7 (except 18.7.1), 18.9 to 18.12, 21.1 (except 21.1.2), 25.4 (except 25.4.1, 25.4.4, and 25.4.5), 25.6, 26.5.1, and 27.6.1 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed, sealed, and dated this 29th day of April 2022.

(Signed and Sealed) *“Hassan Ghaffari”*

Hassan Ghaffari, P.Eng., M.A.Sc.
Director of Metallurgy
Tetra Tech Canada Inc.

Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)

I, Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP), do hereby certify:

- I am a Principal Geologist with Mining Associates with a business address at L6 445 Upper Edward Street Spring Hill Queensland 4004.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji" with the effective date of April 29, 2022 (the "Technical Report").
- I am a graduate of James Cook University (B.Sc. [Hons], 1993) and Edith Cowan University (Graduate Certificate in Geostatistics, 2013). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (#110090). My relevant experience includes more than 25 years in the minerals industry. My work experience includes resource geology, production geology in open pit and underground mines, and exploration roles. I have worked more recently as a consulting geologist and have consulted primarily in relation to gold resource estimates including epithermal gold (high and low sulphur systems), alkaline gold, copper-gold and gold-molybdenum (porphyries), skarn, VMS, and unconformity-related uranium resource projects in Australia, Indonesia, Papua New Guinea, Philippines, Fiji, Myanmar, Turkey, and Columbia.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I visited the Property that is the subject of the Technical Report during the periods of February 25 to 28, 2014; July 31 to August 5, 2017; and again from September 28 to October 3, 2017 to review the geological setting, locate some drill collars, inspect drill core and sample storage, view underground lode development, and discuss geological models with site-based geologists.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- My previous experience with the Property that is the subject of this Technical Report includes the Lion One 2020 PEA.
- I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 2.2.1, 3.1, 4.0, 5.0, 6.0, 7.0, 10.0, 11.0, 12.0 (except 12.3.2 to 12.3.5 and 12.4.2 to 12.4.8), 14.0, 23.0, 25.1, 26.1, 27.2, and 27.8 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of April 2022.

(Signed) *"Ian Taylor"*

Ian Taylor, B.Sc. (Hons), G.Cert. Geostats, F.AusIMM (CP)
Principal Geologist
Mining Associates

Jianhui (John) Huang, Ph.D., P.Eng.

I, Jianhui (John) Huang, Ph.D., P.Eng., do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Canada Inc. with a business address at Suite 1000 – 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji" with the effective date of April 29, 2022 (the "Technical Report").
- I am a graduate of North-East University (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals (M.Eng., 1988), and Birmingham University (Ph.D., 2000). I am a member in good standing of Engineers and Geoscientists British Columbia (#30898). My relevant experience with respect to metallurgy and mineral processing engineering includes more than 35 years of involvement in mineral processing technology development, processing plant design, metallurgy and processing consulting, especially for various precious metal and base metal ores.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Property that is the subject of this Technical Report. I visited the metallurgical testing laboratories including BV on August 8, 2018; Jinpeng Group on December 2, 2017; and Xinhai on December 4, 2017.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- My previous experience with the Property that is the subject of this Technical Report includes reviewing processing plant designs by Jinpeng Group and Xinhai, and reviewing/supervising test programs conducted by BV. I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 1.0, 1.5, 1.7, 1.9, 1.10.2, 1.12, 2.0 (except 2.2.1 to 2.2.4), 3.0 (except 3.1 and 3.3), 12.3.5, 12.4.6, 13.0, 17.0, 19.0, 20.0, 21.2 (except 21.2.1), 24.0, 25.3, 25.5, 25.8, 26.0 (except 26.1 to 26.3 and 26.5), 27.1, 27.4, and 27.7 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed, sealed, and dated this 29th day of April 2022.

(Signed and Sealed) *"Jianhui (John) Huang"*

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Canada Inc.

Laszlo Bodi, M.Sc., P.Eng.

I, Laszlo Bodi, M.Sc., P.Eng., do hereby certify:

- I am a Principal Geotechnical Engineer with Wood Environment & Infrastructure Solutions, a Division of Wood Canada Limited, with a business address at 2020 Winston Park Drive, Suite #600, Oakville, Ontario, Canada, L6H 6X7.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji", with the effective date of April 29, 2022 (the "Technical Report").
- I am a graduate of the Budapest University of Technology and Economics, Budapest, Hungary (B.Sc. Civil Engineering, 1977) (M.Sc. Civil, Highway, Railway and Geotechnical Engineering, 1978). I am a member in good standing of the Professional Engineers Ontario (#90235631), the Ontario Society of Professional Engineers (#11386128), the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (L4236), the Engineers and Geoscientists British Columbia (#45219), and the Canadian Geotechnical Society (#060398). My relevant experience includes more than 42 years of involvement in consulting and lecturing in geotechnical and foundation engineering fields, including waste disposal and infrastructure studies for mining applications at scoping, prefeasibility, feasibility and detailed design levels in Canada, Spain, Macedonia, Russia, Kazakhstan, Armenia, Turkey, USA, Gabon, Republic of the Congo, Burkina Faso, Fiji, and Ivory Coast. I have extensive experience in foundation design and construction support for buildings, bridges, hydroelectric and nuclear power generating stations, pumping stations for TransCanada pipelines, transmission tower and pole foundations, transformer stations, large elevated storage tanks, and wind turbine foundations, etc. I completed design and construction support for seepage control systems (perimeter drainage and cut-off walls) around various waste disposal sites in Canada and around the world.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I visited the Property that is the subject of this Technical Report from April 4 to 7, 2018 to review the geotechnical conditions across the site.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 1.8.1, 1.8.3, 2.2.3, 2.2.4, 12.3.3, 12.3.4, 12.4.3, 12.4.4, 18.3, 18.8, 25.4.1, 25.4.4, 26.5.2, 26.5.3, and 27.6.2 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed, sealed, and dated this 13th day of May 2022.

(Signed and Sealed) *"Laszlo Bodi"*

Laszlo Bodi, M.Sc., P.Eng.
Principal Geotechnical Engineer
Wood Environment & Infrastructure Solutions
a Division of Wood Canada Limited

Maureen Phifer, P.Eng., B.Sc.

I, Maureen Phifer, P.Eng., B.Sc., do hereby certify:

- I am the Mining Division Manager with Tetra Tech Canada Inc. with a business address at Suite 1000 – 10th Floor, 885 Dunsmuir St., Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji" with the effective date of April 29, 2022 (the "Technical Report").
- I graduated in 2013 from the Montana Technical University with a B.Sc. in Mining Engineering. I am a member in good standing of Engineers and Geoscientists British Columbia (#176335). My relevant experience includes 10 years of experience working in precious metals, onsite operational experience, and consulting.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Property that is the subject of this Technical Report.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 1.11, 3.3, 22.0, and 25.7 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed, sealed, and dated this 29th day of April 2022.

(Signed and Sealed) *"Maureen Phifer"*

Maureen Phifer, P.Eng., B.Sc.
Manager, Mining Division
Tetra Tech Canada Inc.

Norman Schwartz, M.Sc.Eng., P.Eng.

I, Norman Schwartz, M.Sc.Eng., P.Eng., do hereby certify:

- I am a Senior Water Resources Engineer with Wood Environment & Infrastructure Solutions, a Division of Wood Canada Limited, with a business address at 2020 Winston Park Drive, Suite #600, Oakville, Ontario, Canada, L6H 6X7.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, in the Republic of Fiji", with the effective date of April 29, 2022 (the "Technical Report").
- I am a graduate of Queen's University, Kingston, Ontario, Canada (B.Sc. Civil Engineering, 1987 and M.Sc. Civil Engineering, 1989). I am a member in good standing of the Professional Engineers Ontario (#90258161). My relevant experience includes 31 years of involvement in water resources engineering consulting, specifically in mining hydrology, as well as dam safety reviews, dam breach studies, and assimilative capacity studies. My experience includes projects in Canada, USA, Mexico, Guyana, Peru, Bolivia, Venezuela, Romania, and Australia. I have extensive experience in water balances and water management designs for tailings facilities, waste rock dumps and overall mine sites, flood routing, design of spillways, decants and drainage systems, and sizing of pumping systems for mine water management in Canada and in other countries.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Property that is the subject of this Technical Report.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 12.4.5, 18.7.1, 18.8.4, 25.4.5, and 26.5.4.2 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed, sealed, and dated this 13th day of May 2022.

(Signed and Sealed) *"Norman Schwartz"*

Norman Schwartz, M.Sc.Eng, P.Eng.
Senior Water Resources Engineer
Wood Environment & Infrastructure Solutions
a Division of Wood Canada Limited

Shane McLeay, B.Eng. Mining (Hons), F.AusIMM

I, Shane McLeay, B.Eng. Mining (Hons), F.AusIMM, do hereby certify:

- I am a Principal Consultant with Entech Pty Ltd. with a business address at 8 Cook St., West Perth, WA 6005, Australia.
- This certificate applies to the technical report entitled “Technical Report and Preliminary Economic Assessment Update for the Tuvatu Gold Project, The Republic of Fiji” with the effective date of April 29, 2022 (the “Technical Report”).
- I am a graduate of Curtin University (B.Eng. Mining, 1995). I am a fellow in good standing of the Australasian Institute of Mining and Metallurgy (#222752). My relevant experience with respect to mining engineering includes more than 15 years of involvement in mine project execution, development, and production, and 10 years consulting primarily in mining studies and execution.
- I am a “Qualified Person” for the purposes of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for the sections of the Technical Report that I am responsible for preparing.
- I have not visited the Property that is the subject of this Technical Report.
- I am independent of Lion One Metals Limited as defined by Section 1.5 of NI 43-101.
- My previous experience with the Property that is the subject of this Technical Report includes provision of technical services from Entech Pty Ltd. to Lion One Metals Limited. I was a co-author Qualified Person of the previous technical report on the Tuvatu Gold Project with an effective date of September 25, 2020.
- I am responsible for Sections 1.6, 12.4.8, 15.0, 16.0, 21.1.2, 21.2.1, 25.2, 26.3, and 27.5 of this Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for preparing have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for preparing contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 29th day of April 2022.

(Signed) *“Shane McLeay”*

Shane McLeay, B.Eng. Mining (Hons), F.AusIMM
Principal Consultant
Entech Pty Ltd.