Technical Report on the

Preliminary Economic Assessment (PEA)

for the

Auld Creek Gold-Antimony Project, New Zealand

Report for NI 43-101

Prepared for: Rua Gold Inc.

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Date and Signature Page

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Project Location:	Reefton District, South Island, New Zealand
Title of Report:	Technical Report on the Preliminary Economic Assessment for the Auld Creek Gold-Antimony Project, New Zealand
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Abbreviations

°C	degrees Celsius
3D	three-dimensional
1VD	first vertical derivative
2VD	second vertical derivative
AA	access arrangement
AACE	American Association of Cost Engineers
AAS	atomic absorption spectrometry
AEO	authority to enter and operate
Ag	silver
AGC	automatic gain control
AI	artificial intelligence
AIG	Australian Institute of Geoscientists
AISC	all-in sustaining costs
ANCOLD	Australian National Committee on Large Dams
ARI	average recurrence interval
As	arsenic
AS	analytic signal
ASTER	advanced spaceborne thermal emission and reflection radiometer
Au	gold
AuCl	gold chloride
AUD	Australian dollars
AuEq	gold equivalent
AusIMM	Australasian Institute of Mining and Metallurgy
Auzex	Auzex Resources Pty Ltd
Au-TL43	low-level gold analysis
bdsh	bedding shear
BLEG	bulk leach extractable gold
BSc	Bachelor of Science
Ca	calcium
CAD	Canadian dollars
CAF	cut-and-fill
cm	centimetres
CMA	*Crown Minerals Act 1991*
COG	cut-off grade
CRAE	CRA Exploration Limited
CRF	cemented rockfill
CRM	certified reference material
CST	cemented dry-stacked tails
CSV	comma-separated values
Cu	copper
CV	coefficient of variation
DGPS	Differential Global Positioning System
DHSA	drillhole spacing analysis
DOC	Department of Conservation
DQO	data quality objective
DSIR	Department of Scientific and Industrial Research
DST	dry-stacked tailings
ECD	environmental control dam
EGL	effective grinding length
Ei	Young's modulus of intact rock
ELOS	Equivalent Linear Overbreak Slough
EOD	extension of duration
EP	exploration permit
EPA	exploration permit application
EPCM	Engineering, Procurement, and Construction Management
EUR	Euro
ew	extremely weathered
FAA303	30 g fire assay, AAS finish
FAA505	50 g fire assay, AAS finish
FAM303	fire assay, ICP-MS finish
FAS30K	3-g charge, screen fire assay at 75 µm
Fe	iron
FLT	fault
g	grams
G&A	General & Administration
g/t	grams per tonne
GIS	Geographic Information System
GNSS	Global Navigation Satellite Systems
GPS	Global Positioning System
GRES	GR Engineering Services Limited
GRG	gravity recoverable gold
GSI	Geological Strength Index
HBX	host breccia
HDPE	high-density polyethylene
Hg	mercury
HG	high-grade
HNO_3	nitric acid
HQ	core diameter: 63.5 mm
HR	hydraulic radius
HV	high voltage
HV	heavy vehicle
IBC	intermediate bulk container
ICP40Q	4-acid digest, ICP-MS finish
ICP41Q	4-acid digest, ICP-MS finish
ICP-MS	inductively coupled plasma mass spectrometry
ICP-OES	inductively coupled plasma optical emission spectroscopy
IMS40Q	4-acid digest, ICP-MS finish
IP	induced polarisation
IRM	internal reference material
ISRM	field rock strength estimate
IT	integrated tool carrier
IUCM	Improved Unified Constitutive Model
IWF	integrated waste facility
Ja	joint alteration
Jn	joint number
JORC	Joint Ore Reserves Committee
Jr	joint roughness
K	potassium
KCSZ	Krantz Creek Shear Zone
KE	kriging efficiency
kg	kilograms
kPag	kilopascals gauge
KPPG	Knight Piésold Pty Ltd

kt	kilotonnes
ktpa	kilotonnes per annum
km	kilometres
koz	kilo ounces
KNA	kriging neighbourhood analysis
kW	kilowatts
kWh/t	kilowatt-hours per tonne
kV	kilovolts
L	litres
L&M	Lime and Marble Limited
Landpro	Landpro Ltd
LiDAR	light detection and ranging
LG	low-grade
LMUC	localised multivariate uniform conditioning
LOD	limit of detection
LOM	life-of-mine
LOQ	limit of quantification
LV	light vehicle
m	metres
Ma	million years
MAR	mineralised argillite
MBIE	Ministry of Business, Innovation & Employment
MCC	motor control centre
mD	metres diameter
ME-MS23	sodium cyanide leach
MEL	mechanical equipment list
Mg	magnesium
MG	medium-grade
MGK	mineralised greywacke
mH	metres high
MIA	minimum impact activities
MIBC	methyl isobutyl carbinol
min	minutes
MIK	multiple indicator kriging
ml	millilitres
mL	metres long
mm	millimetres
MMU	minimum mining unit
MMCL	Macraes Mining Co Ltd
Mo	molybdenum
MOU	memorandum of understanding
Moz	million ounces
MP	Member of Parliament
MPa	megapascals
MPG	MPG Partnership
MRE	Mineral Resource estimate
MS	magnetic susceptibility
MSO	Mineable Shape Optimiser
MSc	Master of Science
Mt	million tonnes
Mtpa	million tonnes per annum
mV/V	millivolts per volt
mw	moderately weathered
mW	metres wide
MW	megawatts
NA	not available
NaCN	sodium cyanide
NI 43-101	National Instrument 43-101
NIWA	National Institute of Water and Atmospheric Research
No	number
NPI	non-process infrastructure
NPV	net present value
NQ	core diameter: 47.6 mm
NS	not sufficient (data)
nT	nanotesla
NZ	New Zealand
NZD	New Zealand dollars
NZP&M	New Zealand Petroleum and Minerals
NZST	New Zealand Standard Time
NZTM	New Zealand Transverse Mercator
OEM	original equipment manufacturer
OGL	OceanaGold New Zealand Ltd
OHCAF	overhand cut-and-fill
OK	ordinary kriging
ONAN	Oil Natural Air Natural
OPEX	operating expenditure
OSA	on-stream analyser
oz	troy ounce
PAX	potassium amyl xanthate
Pb	lead
$Pb(NO_3)_2$	lead nitrate
PBG	Pitch Black Group Pty Ltd
PBX	pug breccia
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PGMs	platinum group metals
PhD	Doctor of Philosophy
POX	pressure oxidation
PP	prospecting permit
ppb	parts per billion
ppm	parts per million
PPS	pulse per second
PQ	core diameter: 85 mm
pXRF	portable X-ray fluorescence
py	pyrite
QA	quality assurance
QBX	quartz breccia
QC	quality control
QGWK	quartz greywacke
QMAP	quarter million mapping
QP	Qualified Person
QQ	quantile-quantile
QSST	quartz sandstone
QTZ	quartz vein
RBF	radial basis function
RD	relative difference
RGL	Reefton Goldfields Limited
RMA	*Resource Management Act*
RMSCV	root mean square coefficient of variation
RO	reverse osmosis
ROM	run-of-mine
RQD	rock quality designation
RPEEE	reasonable prospects for eventual economic extraction
RRPL	Reefton Resources Pty Ltd

RSC	RSC Consulting Ltd
RTP	reduced to pole
RUA	Rua Gold Inc.
s	seconds
Sb	antimony
SBX	Sb breccia
SD	standard deviation
SDAF	shotcrete drift-and-fill
SEM	scanning electron microscope
SG	specify gravity
SLOS	sub-level open stoping
Sn	tin
SOP	standard operating procedure
SoR	slope of regression
SQL	structured query language
SumN	sum of negative weights
sw	slightly weathered
t	tonnes
t/m/h	tonnes per metre per hour
$t/m^2/h$	tonnes per square metre per hour
TEM	transient electromagnetic method
Th	thorium
tilt	tilt angle filter
TIR	thermal infrared
TKM	tonne-kilometres
TL	tie line
TMI	total magnetic intensity
tpa	tonnes per annum
tpd	tonnes per day
tph	tonnes per hour
TPVM	tonnes per vertical metre
TSF	tailings storage facility
UAV	unmanned aerial vehicle
UCS	uniaxial compressive strength
UHCAF	underhand cut-and-fill
UoM	unit of measure
UPS	uninterruptible power supply
USD	United States dollars
UTC	universal time coordinated
uw	unweathered
VAD	ventilation access drive
VHF	very high frequency
VVVF	variable voltage variable frequency
W	tungsten
wt.%	weight percent
X	latitude
XRT	X-ray transmission
Y	longitude
Z	elevation
Zn	zinc

1 Summary

1.1 Overview

Rua Gold Inc. (RUA) commissioned this Scoping Study/Preliminary Economic Assessment (PEA) of the Auld Creek Gold-Antimony Project (the Project) in the Reefton district of the South Island of New Zealand.

The purpose of the PEA is to evaluate the conceptual economic potential of developing the deposit as a small-scale underground mining operation and to support RUA's application for project approval through the New Zealand Fast-Track Approvals Program. RUA is listed on the Toronto Stock Exchange; therefore, this PEA technical report has been prepared in compliance with National Instrument 43-101: Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43-101F1.

The Project comprises one exploration permit (EP 60648, Golden Point) held by Reefton Resources Pty Limited (RRPL). In November 2024, RUA completed the acquisition of RRPL from Siren Gold Ltd (Siren); consequently, RRPL is now a wholly owned New Zealand subsidiary of RUA. EP 60648 expired on 18 March 2026, and an extension of duration application was submitted on 16 December 2025. The EOD was granted on 28 May 2026, with the new expiry date for the permit of 18 March 2031. This Report documents data and data collection procedures for the Project and current mineral resources. Work on the Project was completed by RRPL prior to RUA's acquisition of RRPL from Siren; all work subsequent to the acquisition has been carried out by RUA. This Report has an effective date of 19 June 2026.

RUA engaged a team of consultants to produce a concept-level scoping study of the Project, including the following contributors to this Technical Report:

- RSC Consulting Ltd (RSC), for geology and Mineral Resource estimation;
- Mining One Pty Ltd (Mining One), for mining scope;
- Pitch Black Group Pty Ltd (PBG), for processing plant and site infrastructure scope; and
- Knight Piésold Pty Ltd (KPPL), for integrated waste storage facility (IWF) scope.

This Scoping Study/PEA includes Class 5 estimates of capital cost and operating cost, as defined by the Association for the Advancement of Cost Estimation (AACE) recommended practice 47R-11. Refer to Table 1-1 for AACE study classifications and estimate accuracy. The accuracy of this PEA is therefore typically expected to be ±30% (noting that the consultants have experience with, and cost databases for, projects of similar size and scope).

Table 1-1: AACE classification guidelines for capital cost estimates (RP 47R-11).

Class of Estimate	Purpose	Accuracy		Level of project completion
		Lower limit	Upper limit	
Class 5 Order of Magnitude Estimate	Scoping Study	-50% to -20%	+30% to +100%	0% to 2%
Class 4 Study Estimate	Prefeasibility Study	-30% to -15%	+20% to 50%	1% to 15%
Class 3 Definitive Estimate	Feasibility Study	-20% to -10%	+10% to 30%	10% to 40%
Class 2 Detailed Estimate	FEED/EPC Bid Preparation	-15% to -5%	+5% to 20%	30% to 70%
Class 1 Check Estimate	Tender	-10% to -3%	+3% to 15%	50% to 100%

On completion of the PEA, RUA plans to immediately commence a Pre-Feasibility Study (PFS) for the Project, with the aim of commencing production in mid-2027.

1.2 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Project is located in the Golden Point exploration permit (EP 60648), which is held in its entirety by RRPL and covers an area of 40.6405 km^2 in the Reefton Goldfield in the West Coast region of the South Island, New Zealand. EP 60648 is located directly south of, and adjacent to, the town of Reefton and is mainly within the Victoria Forest Park Conservation area.

The Project is well connected by state highways and public roads to nearby towns. The closest regional airport is in Hokitika, which connects to Christchurch International Airport, and there is also an airport in Westport with scheduled flights to Wellington. Reefton is connected to New Zealand's rail network, and there are small ports located at Westport and Greymouth, which typically service small-tonnage coastal freight and fishing vessels.

To support its exploration, RUA has an exploration office in the township of Reefton. The exploration office includes a small laboratory for processing soil samples, sample and supply storage, core cutting, and core logging. The West Coast region of the South Island has an active mining industry; therefore, numerous skilled contractors and organisations are available to support exploration and mining activities.

The Project area is in the rain shadow of the Paparoa Range. The climate is wet and temperate, with average annual rainfall in Reefton of 1,920 mm. Spring is the wettest season, and late summer and early autumn are the driest. Average mean temperatures range from 5°C in winter to 17°C in summer. Field work can be conducted year-round, but field activities can be restricted any time of the year by periods of extreme weather (i.e. heavy rain). The surrounding area is typically hilly, with moderate to steep relief in the foothills of the Victoria and Brunner ranges. The topography is locally very steep and varies in elevation from ~100 m to >1,000 m above sea level. Creeks and rivers strongly incise the area, and the steep topography often limits field work access to foot or helicopter.

1.3 Geological Setting and Mineralisation

New Zealand lies on the boundary between the Australian and Pacific plates; the Alpine Fault delineates that boundary in the South Island. The northwest of the South Island comprises the West Coast Basin region, which is mainly composed of broad, approximately north-trending belts of early Palaeozoic rocks that terminate against the Alpine Fault in the southeast (Mortimer, 2004).

The Reefton Goldfield is hosted entirely within Early Ordovician rocks of the Greenland Group in the Buller Terrane of the West Coast Basin (MacKenzie, 2014; Allibone et al., 2020). In the Reefton area, the Greenland Group forms a ~35 km × 15 km north-northeast trending belt that is bound to the north and east by granitic plutons of the Late Devonian–Carboniferous Karamea and Cretaceous Rahu and Separation Point batholiths (Laird and Shelley, 1974; Tulloch, 1988; Muir et al., 1996). In the south and west, the belt is in fault contact with high-grade paragneisses of the Paparoa metamorphic core complex (Ritchie et al., 2015).

The Greenland Group is a turbiditic sequence of alternating greywackes and argillites that were deformed and metamorphosed to lower greenschist facies in the Ordovician–Devonian (450–387 Ma), with the development of illite clay facies (Adams, 2004; Turnbull et al., 2016). The sediments are dominated by greywacke, and beds are typically 0.2–2 m thick and separated by 10–30 cm layers of argillite. The greywackes typically comprise >50% quartz with lesser albite, partially recrystallised rock fragments, and muscovite, whereas the argillites are less quartz-rich and more micaceous (Milham and Craw, 2009). Widespread folding was most likely synchronous with metamorphism, and this deformation predates granitoid emplacement (Mortimer et al., 2013).

Gold (Au) and antimony (Sb) mineralisation in the Project area is orogenic style, and the deposits occur in and around steeply dipping, north to north-northeast trending shear zones that cut across the hinges of earlier folds in weakly altered metasedimentary rocks. The deposits are similar, in many respects, to those at Bendigo and Ballarat in Victoria (Cooper and Tulloch, 1992; Phillips and Hughes, 1996) and Nova Scotia in Canada (Ryan and Smith, 1998; Christie and Brathwaite, 1999).

Most of the Au- and Sb-bearing mineralisation in the Reefton Goldfield, including all of the larger deposits, occurs along an approximately north-trending linear belt that cuts a sequence of deformed metasedimentary Greenland Group rocks (Allibone et al., 2020). This suggests the presence of a deep-seated structure that has permitted mineralising fluids to migrate from their source to the upper crust, where Au and Sb were deposited.

The two dominant styles of Au mineralisation in the Project area are (MacKenzie, 2014):

- coarse native Au associated with minor sulphides in quartz veins; and
- microscopic refractory Au within sulphides in sheared sediments and clay alteration (pug) zones adjacent to quartz veins.

Historical production targeted mainly coarse native Au; however, both mineralisation styles provide important exploration targets (Madambi and Moore, 2013).

1.4 Exploration and Drilling

The nature and extent of historical exploration work undertaken by previous owners are summarised in Section 6.2, and some of these data have been used as a basis for the Mineral Resource estimate (MRE) reported in Section 14. Exploration work conducted by RRPL and RUA over EP 60648, including geophysical surveys, soil sampling, rock-chip sampling, trenching, and drilling, is summarised in Sections 9.1, 9.2 and 9.3. RRPL collected 1,086 conventional soil samples over the Auld Creek area, with a total of 1,336 conventional soils collected from the entire EP 60648 permit area. RRPL collected 46 ultrafine soil samples and 22 whacker samples within EP 60648. RRPL collected a total of 102 rock-chip samples from EP 60648, with 46 from the Auld Creek prospect. Veins were sampled at intervals of 0.3–2.4 m, depending on the width of the outcrop (average ~1 m), and samples were used to identify new reefs and surface extensions.

Following the acquisition of RRPL from Siren, RUA collected a further 1,622 soil samples over EP 60648.

RRPL completed a total of 17 drillholes at the Project for a total of 2,191 m. After the acquisition of RRPL, RUA completed a further 38 drillholes for 8,086.9 m at Auld Creek, bringing the total number of drillholes to 55 for a total of 10,277.9 m.

In September 2024, RRPL announced that three metallurgical samples selected from the Fraternal shoot at Auld Creek (Golden Point) yielded recoveries of >95% Au and Sb (Siren Gold Limited, 2024a).

1.5 Sample Preparation, Analyses and Security

Sampling, sample preparation, and analyses were conducted using industry-standard protocols and independent, ISO/IEC 17025-accredited laboratories, including SGS Westport, SGS Waihi, ALS Brisbane, and LabWest, Perth. Soil, rock-chip, trench, and diamond core samples were collected from the Project by RRPL and RUA. Preparation involved drying, crushing, splitting, and pulverising to appropriate specifications.

Analytical work was primarily undertaken using fire assay with AAS or ICP-MS finish, with screen fire assays applied where coarse Au was anticipated. Quality assurance and quality control (QA/QC) procedures included the systematic insertion of certified reference materials (CRMs), blanks, duplicates, and repeat analyses, with results indicating that the various processes were typically in control, and the data produced are sufficiently accurate and precise for the classification of moderate-confidence mineral resources.

Portable XRF (pXRF) data were used for multi-element screening and geological interpretation. Bulk density determinations were completed using standard water displacement methods, which the QP (Abraham Whaanga) considers acceptable for the classification of moderate-confidence mineral resources, despite minor data management limitations.

Samples were securely stored and transported under controlled conditions, although the QP (Abraham Whaanga) recommends the implementation of formalised chain-of-custody procedures.

1.6 Data Verification

The drillhole data collected by RRPL are comprehensive and were semi-validated at the point of collection through a process of quality assurance and continual quality control. Verification checks completed by the QP (Abraham Whaanga) uncovered several minor errors in the MS Excel workbooks provided by RRPL, which were corrected. The QP (Abraham Whaanga) recommends that all data for the Reefton Project be transferred from the MS Excel workbooks into the relational MS Access database for the Project; this process is underway.

RSC collected a mix of half-core and pulp duplicate samples for verification purposes. The Au samples demonstrate a bias towards the original sample that is likely to be in the order of 4%, which is immaterial with respect to the objectives.

Overall, in the opinion of the QP (Abraham Whaanga), the data on which the mineral resources are based are verified and fit for the purpose of classifying Indicated Mineral Resources. The QP (Abraham Whaanga) recommends RUA conducts further investigations into the check sample bias to ensure that future data are fit for purpose for higher-confidence classifications.

1.7 Mineral Processing and Metallurgical Testing

The metallurgical test work carried out to date indicates the following:

- The ore has moderate hardness.
- Antimony predominantly reports as stibnite within flotation concentrates.
- Gravity recoverable gold (GRG) is present; however, recovery via leaching is low.
- There is potential to use flotation to produce both Au and stibnite concentrates at acceptable grades and recoveries. At a primary grind of 80% passing 106 µm, rougher flotation kinetics are rapid, with high recoveries achieved. Acceptable concentrate grades can be obtained through four stages of cleaning without regrinding.
- Stibnite particles are typically finer than the target grind size, indicating a brittle/friable nature and a tendency to be overground during milling.
- Arsenic (As) is associated with the Au concentrate, representing a potentially significant penalty element. Further work is required to evaluate As levels and their impact on concentrate marketability.
- Initial flotation test work supports the use of potassium amyl xanthate (PAX) as a collector, with lead nitrate as an activator to enhance antimony recovery under natural pH conditions. Further test work is required to confirm and optimise flotation performance in subsequent study phases.

Further metallurgical test work is planned to support the PFS. This work will focus on confirming the initial test results, improving ore characterisation (including variability across the deposit), and optimising the processing flowsheet. Additional test work will also be carried out to investigate the behaviour of penalty and deleterious elements, with the aim of refining process performance and assessing potential impacts on concentrate quality and marketability.

1.8 Mineral Resource Estimate

Geological modelling was conducted in Leapfrog Geo and was based largely on the 2024 RRPL geological interpretation. The estimation domains were subsequently derived from the primary geological and weathering models. Sub-domaining was undertaken in some domains to help constrain high grades. Contact analysis was completed to investigate the boundary conditions of each domain. The variables were estimated in the block model in one or two passes, with variable orientation based on the vein reference surface to guide the ellipsoid direction. Grades were interpolated using ordinary kriging (OK). Block model grades were validated by comparing the input mean grades with the block model mean grade, using swath plots, and visually using cross-sections. Sensitivity testing was undertaken to assess the input parameters.

The QP (Abraham Whaanga) has classified the Mineral Resource for the Auld Creek deposit in the Inferred and Indicated categories (Table 1-2), in accordance with NI 43-101 and the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014). For the Indicated Mineral Resources, geological evidence is derived from adequately detailed and reliable exploration, sampling, and testing, and is sufficient to assume geological and grade or quality continuity between points of observation. For the Inferred Mineral Resources, geological evidence is sufficient to imply, but not verify, geological and grade continuity. The Mineral Resource is based on exploration, sampling, and assaying information gathered through appropriate techniques from trenching and drillholes.

The QP (Abraham Whaanga) is of the opinion that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued drilling. For the Inferred portion of the MRE, confidence in the estimate is not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning in pre-feasibility or feasibility studies.

Cautionary Statement: the PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA will be realised.

The QP (Gary Davison) completed a high-level initial assessment of various factors solely for the purpose of meeting the criteria for reasonable prospects for eventual economic extraction (RPEEE) (refer to Section 14.11 for full details). The cut-off grade United States dollar (USD) value was determined using mining and development costs and modifying factors for an anticipated sub-level, long-hole, open stoping mining method. The resulting MRE is presented as 'RAW MSO' in Table 1-2.

The QP (Gary Davison) refined the mineable shapes by:

- removing shapes manually which were isolated and not near the bulk of the deposit (i.e. not economical to develop); and
- excluding material contained within 20 m of the surface (crown pillar recommended by the Mining One geotechnical engineers).

Table 1-2 also presents the final MRE after exclusion of the crown pillar and the manually removed shapes. In the base case, the QP (Gary Davison) has assumed that the crown pillar can be mostly extracted.

Table 1-2: MRE for the Auld Creek deposit, with an effective date of 19 June 2026.

MSO	Resource category	Tonnes (Mt)	Au (g/t)	Sb (%)	AuEq (g/t)	Contained Au (koz)	Contained Sb (kt)	Contained AuEq (koz)
RAW MSO	Indicated	0.35	3.0	1.0	5.2	34	4	58
	Inferred	1.35	1.8	0.7	3.4	79	10	148
20 m Crown	Indicated	0.03	5.1	1.1	7.4	6	0	8
	Inferred	0.16	1.9	0.8	3.5	9	1	18
Final MSO Set	**Indicated**	**0.28**	**2.5**	**1.0**	**4.6**	**23**	**3**	**42**
	Inferred	**0.98**	**1.9**	**0.7**	**3.5**	**59**	**7**	**109**

Notes:

1. *The Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves (May 2014) were used for the Mineral Resource estimate.*
2. *The Mineral Resource is reported at a cut-off of 1.5 g/t AuEq.*
3. *Metal-equivalent grades were calculated using the following formula: AuEq = Au g/t + 2.15 × Sb%.*
4. *The AuEq factor of 2.15 is calculated using the following prices: USD 3,300/oz Au, and USD 27,000/t Sb. Metallurgical recoveries of 95% Au and 85% Sb, with 95% Au and 90% Sb payable in concentrate were used, where C = payable in concentrate R = metallurgical recovery and P = price.*

$$AuEq\ factor = \frac{(P_{Sb} \div 100) \times C_{Sb} \times R_{Sb}}{(P_{Au} \div 31.10348) \times C_{Au} \times R_{Au}}$$

5. *MSO parameters are outlined in the Report.*
6. *Totals may vary due to rounding.*
7. *Shapes were removed manually if they were <1.8 g/tAuEq and their LEVEL attribute had <5,000 t (not economical to develop).*
8. *Crown pillar was assumed to be ~75% extractable, needs further work at the PFS level to minimise surface impact.*
9. *The Preliminary Economic Assessment (PEA) presented in this Report is based on the Mineral Resources reported within the Project and is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the PEA will be realised.*
10. *The QP (Abraham Whaanga) is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues or any other relevant issue that could materially affect the MRE.*

1.9 Mining Methods

The geotechnical assessment indicates that the Auld Creek deposit is characterised by poor to very poor rock mass conditions, with some improvement at depth. Rock mass quality remains low (Q′ ≈ 1–2; GSI ≈ 22–31), and intact strength is estimated at 25–50 MPa based on field observations. Structural analysis indicates a dominant bedding-parallel fabric dipping moderately to steeply west to northwest, with significant variability (±60°), indicating undulating bedding that is expected to influence local stope stability.

Empirical and numerical analyses indicate that stable unsupported spans are limited, particularly at depth. Recommended hanging wall hydraulic radii are ~4.0 m in the upper domain (600–480 mRL) and 2.0 m in the lower domain (480–250 mRL). Due to the potential for dilution and associated operational risk, stoping methods are considered a potential upside option requiring further investigation at the Pre-Feasibility Study (PFS) stage.

The QP (Gary Davison) evaluated two underground mining methods suitable for the deposit:

- sub-level open stoping (SLOS) with cemented backfill; and
- overhand cut-and-fill (OHCAF) mining using a combination of cemented and uncemented fill.

Based on deposit geometry, selectivity requirements, and geotechnical constraints, the QP (Gary Davison) selected OHCAF as the preferred base case mining method for detailed mine design and scheduling.

The QP (Gary Davison) developed a conceptual underground mine design to access and extract the potentially mineable inventory. The deposit would be accessed via a decline developed from surface portals, positioned to minimise surface disturbance while enabling early access to higher-grade zones (Figure 1-1).

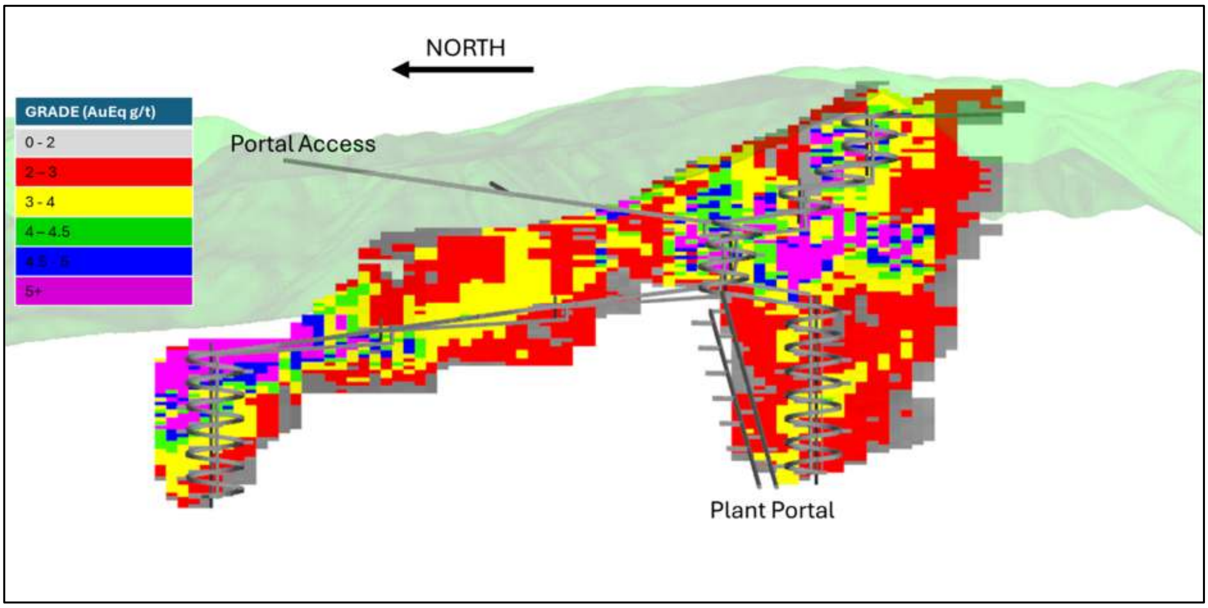


Figure 1-1: Mine design long-section illustrating early high-grade targets.

The QP (Gary Davison) developed a production schedule using benchmark development rates and constrained it to a nominal production rate of ~250 ktpa. The schedule indicates a total mine life of ~6 years, with total production of ~1.4 Mt of mined material (including crown pillar), 85 koz Au, and 9 kt Sb.

The life-of-mine (LOM) plan includes ~16.6 km of capital development and 51.3 km of operating development. The QP (Gary Davison) also assessed the surface disturbance footprint (Table 1-3 and Figure 1-2) expressed in terms of equivalent 'tennis courts' for presentation to the Department of Conservation (DOC).

Table 1-3: Land disturbance area.

Land Disturbance	Area (m^2)	Area in Number of Tennis Courts
Temporary portal	1,600	7
Northern exhaust	100	1
Southern exhaust	100	1
Crown pillar mining	7,800	30
Temporary waste dump	1,600	7

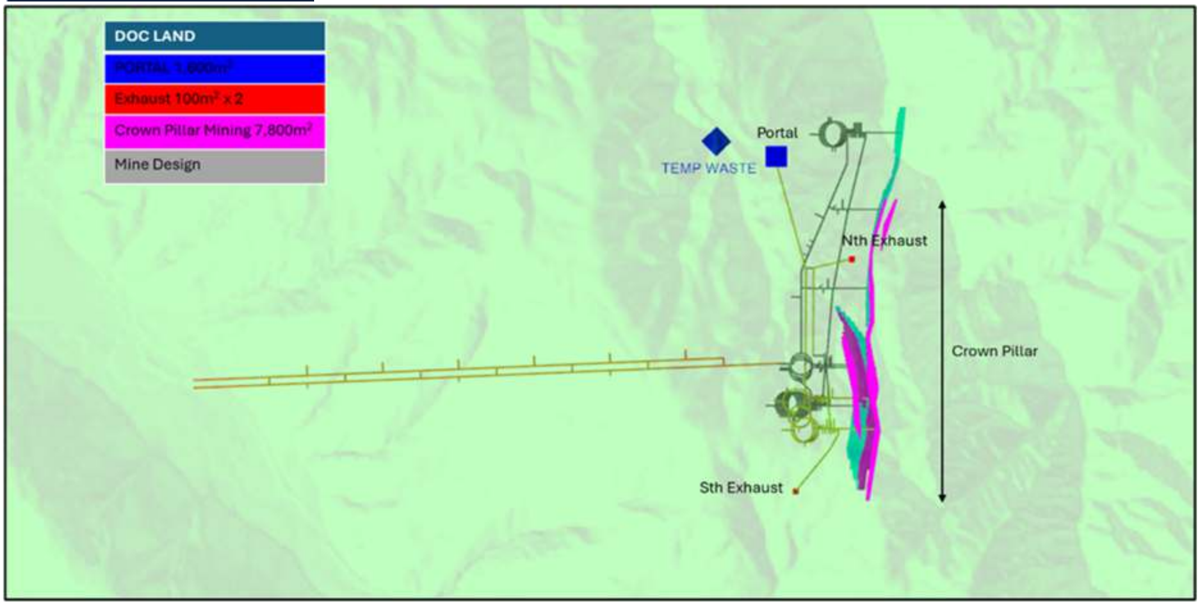


Figure 1-2: Land disturbance on DOC land.

1.10 Recovery Methods

The processing plant design developed by PBG is based on a nominal throughput of 0.25 Mtpa of run-of-mine (ROM) ore, consistent with the initial mine plan. The projected mine life is ~6–7 years based on the current Mineral Resources.

Based on metallurgical test work and the selected throughput, PBG has recommended a processing plant comprising the following components:

- primary crushing using a scalped jaw crusher, followed by closed-circuit secondary cone crushing;
- a mill feed bin and apron feeder system, supplying a single ball mill operating in a closed circuit with hydrocyclones;
- a rougher flotation circuit consisting of six forced-air flotation cells; and
- a four-stage cleaner flotation circuit, upgrading the rougher concentrate to produce an antimony-rich concentrate, with gold reporting to the associated flotation products.

The two saleable products (Sb and Au concentrates) will each be thickened, filtered, and bagged for sale.

In line with recommendations by the QP (Timothy Knowles), tailings would be filtered and stored as a dry-stacked product.

The processing plant is also planned to include supporting infrastructure for the supply and handling of reagents and utilities required for plant operation.

1.11 Project Infrastructure

Site infrastructure for the Project includes the following key components.

- Roads and access: upgrades to existing public roads, including the closure of a section of Soldiers Big River Road that runs through the project site and construction of a bypass via extension of Quigleys Road. A site access road (including a creek ford) is also planned be developed, in addition to internal site roads and tracks.
- Power supply and distribution: high-voltage power is planned to be supplied via connection to the local grid and transmission through overhead powerlines to a high-voltage transformer yard.

Power will then be distributed across the site at 11 kV using a combination of buried cables and overhead powerlines to substations. Preassembled modular switchrooms are planned to house electrical equipment and distribute low-voltage power to end users.

- Water supply and services: raw water is planned to be extracted from Devils Creek to meet process water requirements, potable water supply, mine water demands, and firewater systems. Approximately 10 L/s of raw water and 11 kV power is planned to be supplied to the main and upper mine portals, with provision for the receipt of up to 25 L/s of mine dewatering discharge, based on estimations by Mining One.
- Site facilities and services: infrastructure is planned to include site buildings such as offices, warehouses, a processing plant control room, concentrate storage facilities, a laboratory, workshops, crib rooms, and ablution facilities. Supporting infrastructure is planned to include fuel storage and dispensing, sewage treatment, and solid waste management systems.
- Water management: site drainage systems, environmental control dams (ECDs), water management infrastructure, and active water treatment facilities are planned to be provided to manage excess water prior to discharge to Devils Creek. This scope remains conceptual and is subject to refinement as geochemical characterisation of tailings and waste rock progresses.

The proposed underground mine will require supporting surface infrastructure including:

- mine administration and operational buildings;
- underground workshops and maintenance facilities;
- portal infrastructure and ventilation installations; and
- haul roads connecting the temporary portal to the processing plant.

The underground ventilation system is planned to use adits for both intake and exhaust airflow, eliminating the need for vertical ventilation shafts.

The integrated waste facility (IWF) is planned to store filtered tailings and limited quantities of waste rock within a lined containment system that directs all contact water to an ECD. The current conceptual design provides a total storage capacity of ~1.94 Mt, comprising 0.97 Mt for the first six years of operation, with the potential for an additional 0.97 Mt for expansion if required. The facility is planned to incorporate staged deposition, surface water management controls, and a fully lined basin to ensure geotechnical stability and effective environmental management. The IWF design, layout, and storage capacity will be refined during the PFS as geotechnical, hydrological, and operational inputs are further developed.

PBG has developed a conceptual water balance to evaluate water supply, storage, and treatment requirements for the Project. Water supply is planned to be sourced via a combination of recycled process water and raw water from Devils Creek. Controlled water treatment and discharge systems will be incorporated to maintain operational water balances and support regulatory compliance.

At the PEA stage, significant volumes of material (mined waste or quarried rock) will likely be required for construction of IWF embankments, processing plant foundations, and stockpile pads. Initial assumptions considered the use of mined waste rock; however, this approach results in a shortfall of material for underground backfill, which led to the inclusion of a paste backfill plant in early design iterations.

Uncertainties remain with respect to the suitability of waste rock for construction, including material quality, geochemical characteristics, and timing of availability. These factors will be further assessed during the PFS. For the purposes of this PEA (and particularly the capital and operating cost estimates), the paste backfill plant has been excluded from the scope, with quarried rock assumed as the primary construction material.

1.12 Market Studies and Contracts

The Project is expected to produce Au and Sb in the form of concentrates. Metal price assumptions applied in this PEA were developed using a combination of historical price data and long-term consensus forecasts from mining industry analysts. The Au and Sb prices used in this Report were provided by RUA and are based on price assumptions used in the February 2026 Independent Technical Report (Whaanga, 2026): the 36-month average spot price for Sb (USD 25,000/t) and 24-month average spot price for Au (USD 3,000/oz). RUA then weighted the price to closer align with the long-term market consensus pricing analysts of USD 25,000–32,000/t for Sb and USD 3,600/oz for Au.

Final long-term metal prices adopted for the study are:

- Gold: USD 3,300/oz; and
- Antimony: USD 27,000/t.

RUA has initiated preliminary discussions with a number of potential off-take counterparties for both concentrates. As at the effective date of this Report, no binding terms or agreements have been finalised. For the purposes of this PEA, a payability assumption of 95% has been applied to both Au and Sb concentrates. This assumption is based on high-level market analysis and reflects typical industry expectations, including allowances for minor variability in concentrate specifications and the presence of penalty elements within acceptable limits.

As at the effective date of this Report, no material contracts, off-take agreements, or hedging arrangements are in place. RUA has not committed any portion of future production and retains full exposure to market pricing. RUA considers the market and commercial assumptions applied in this study to be appropriate for a PEA, noting that further work will be required at subsequent study stages to confirm concentrate specifications, payability terms, and commercial agreements.

1.13 Environmental Studies, Permitting, and Social or Community Impact

Environmental baseline information for the Project is limited and based on publicly available data and targeted studies, including ecological (bat survey) and archaeological assessments. Comprehensive baseline programmes have not yet been completed and will be required to support future permitting.

The Project is located predominantly on conservation land within the Victoria Conservation Park, administered by the Department of Conservation (DOC), with smaller areas of freehold and forestry land. To date, no material environmental liabilities or compliance issues have been identified.

RRPL holds exploration permit EP 60648 under the *Crown Minerals Act 1991* (1991a) and has secured, or is in the process of securing, the required land access arrangements. Based on the QP's review, there are no identified material risks preventing ongoing exploration activities.

Key risks relate to the need for further baseline environmental data, permitting requirements associated with conservation land, and potential regulatory processes under the *Resource Management Act 1991* (1991b). These risks are typical for projects at the PEA stage and are considered manageable.

The West Coast region has a strong mining history and comparatively favourable community support for mining. Engagement to date indicates generally positive feedback, and a Social Impact Assessment is underway. Ongoing engagement with mana whenua will be required as part of project development.

Waste management, water management, and mine closure planning are currently conceptual and will be refined through future studies.

The *Fast-track Approvals Act 2024* (2024) (Fast-track) introduces a coordinated approvals regime that may streamline permitting across multiple statutes. While its applicability to the Project is uncertain at this stage, it may provide an opportunity to reduce permitting timelines.

1.14 Capital and Operating Costs

Capital and operating cost estimates were prepared by Mining One, KPPL, and PBG, and consolidated into a Project-wide estimate.

The PEA is supported by a concept-level scoping study and an American Association of Cost Engineers (AACE) Class 5 capital cost estimate, with an expected accuracy range of approximately ±43%. As for typical studies at this level, the estimate is based on limited test work and engineering, with major equipment sized at preliminary level and other quantities derived from analogous projects or factored allowances. Cost inputs are primarily based on historical data and industry benchmarks rather than firm quotations. All costs are presented in USD at a base date of Q2 2026, with no escalation applied. The estimate includes direct and indirect costs, owner's costs (assumed at 3% of direct costs), and a contingency of 30% applied to pre-production costs, consistent with the level of project definition. Closure costs are included, while pre-development costs such as exploration drilling, further test work, feasibility studies, and permitting are excluded.

The total estimated initial capital cost is USD 226.3 million, comprising:

- mining: USD 109.9 million (±50%);
- processing plant: USD 81.7 million (±35%);
- site infrastructure: USD 23.3 million (±35%); and
- IWF: USD 11.4 million (−35% / +50%).

Operating costs were developed from estimates of labour, power, fuel, reagents, consumables, maintenance, and sustaining capital. Inputs are based on equipment sizing, test work (where available), and benchmarking against comparable operations. Indirect corporate costs are excluded.

The total operating cost is estimated at USD 41.8 million per annum, equivalent to USD 167.3 per tonne of ROM ore (dry basis). Mining represents the dominant cost component (78%), followed by processing and site infrastructure (19%) and IWF (3%).

1.15 Economic Analysis

The results of this PEA indicate that the Auld Creek deposit has the potential to support a modest underground mining operation using selective mining methods appropriate for the narrow-vein geometry of the mineralisation.

The OHCAF mining method forms the base case, as it is considered the most technically robust option given the interpreted rock mass conditions and the requirement for selective extraction. Sub-level open stoping remains a potential alternative method; however, further work is required to assess dilution control and operational viability. Based on the base case OHCAF mine plan, the Project generates an estimated net present value (NPV) (5%) of USD ~42 million and an internal rate of return (IRR) of ~17%, with a payback period of ~40 months. Undiscounted cashflow is estimated at USD ~70 million.

The economic model assumes an Au price of USD 3,300/oz and an Sb price of USD 27,000/t. LOM production comprises ~84.5 koz Au and 9.0 kt Sb, with an average grade of 3.36 g/t AuEq and a calculated cut-off grade of 1.79 g/t AuEq.

Total capital cost is estimated at USD ~197 million, including USD 133 million of pre-production capital and USD 64 million of sustaining capital. Average operating costs are estimated at USD ~144/t, with cash costs of USD 1,399/oz and all-in sustaining costs (AISC) of USD 1,835/oz.

The QP (Gary Davison) considers that the Project presents a potential opportunity to establish infrastructure in the Reefton Goldfield that could support future development of additional nearby deposits, including the potential for shared processing capacity.

Alternative development scenarios were evaluated to assess potential value upside, including the use of dry-stacked tailings or paste backfill, and increasing the cut-off grade. These scenarios demonstrate sensitivity of project economics to mining approach, cost structure, and grade selection, with base case conditions providing the strongest overall economic outcome at this stage of evaluation.

Cautionary Statement: the PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA will be realised.

2 Introduction

2.1 Purpose of this Report

Rua Gold Inc. (RUA) commissioned this Preliminary Economic Assessment (PEA) of the Auld Creek Gold-Antimony Project (the Project) in the Reefton district of the South Island of New Zealand. The purpose of the PEA is to evaluate the conceptual economic potential of developing the deposit as a small-scale underground mining operation and to support RUA's application for project approval through the New Zealand Fast-Track Approvals Program. RUA is listed on the Toronto Stock Exchange; therefore, this PEA technical report has been prepared in compliance with National Instrument 43-101: Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43-101F1.

The Project comprises one exploration permit (EP 60648, Golden Point) held by Reefton Resources Pty Limited (RRPL). In November 2024, RUA completed the acquisition of RRPL from Siren Gold Ltd (Siren); consequently, RRPL is now a wholly owned New Zealand subsidiary of RUA. EP 60648 expired on 18 March 2026, and an extension of duration application was submitted on 16 December 2025. The EOD was granted on 28 May 2026, with the new expiry date for the permit of 18 March 2031. This Report documents data and data collection procedures for the Project and current mineral resources. Work on the Project was completed by RRPL prior to RUA's acquisition of RRPL from Siren; all work subsequent to the acquisition has been carried out by RUA. This Report has an effective date of 19 June 2026.

2.2 Sources of Information

The information in this technical report is based on data supplied by RRPL and RUA, in addition to verification data collected by or under the supervision of the QPs. RRPL and RUA provided RSC with .csv files exported from a database of all drilling and sample data available for the Project. Copies of previous reports (geochemical, petrological, geophysical, and metallurgical), core photographs, standard operating procedures (SOPs), and GIS data were also provided.

RRPL/RUA provided RSC with copies of the original logging sheets for drillholes and collar survey files. Original laboratory certificates, data files from ALS and SGS chemical analyses, original portable XRF (pXRF) data, and SOPs were also made available to RSC.

Information relating to property ownership, property titles, and legal and environmental matters was sourced from official documentation.

The mining study used the resource model prepared by RSC, geotechnical information provided by RRPL, and mine planning and scheduling completed by Mining One using Deswik software. The study was based on an overhand cut-and-fill mining method incorporating cemented rockfill, rockfill, and dry-stack tailings backfill. Mining capital and operating cost estimates were developed from contractor quotations, vendor budget pricing, industry benchmarking and Mining One's experience with similar underground mining operations.

A full list of the sources of information, data, and reports reviewed as part of this technical report is provided in Section 31. The QPs listed on the Date and Signature Page take responsibility for the content of this Report and consider the data review to be accurate and complete in all material aspects.

2.3 Qualified Persons

The individuals presented on the Date and Signature Page, by virtue of their education, experience and professional association are considered QPs for this report as defined by NI 43-101. The QPs meet the requirements for independence that are defined in NI 43-101. Table 2-1lists the technical report sections that each QP is responsible for preparing/approving.

Table 2-1: Qualified Persons responsible for preparation/approval of the Report by section.

Title	QP Responsible for Preparation/Approval
1. Summary	All QPs
2. Introduction	All QPs
3. Reliance on Other Experts	All QPs
4. Property Description and Location	All QPs
5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Abraham Whaanga (RSC)
6. History	Abraham Whaanga (RSC)
7. Geological Setting and Mineralisation	Abraham Whaanga (RSC)
8. Deposit Types	Abraham Whaanga (RSC)
9. Exploration	Abraham Whaanga (RSC)
10. Drilling	Abraham Whaanga (RSC)
11. Sample Preparation, Analyses, and Security	Abraham Whaanga (RSC)
12. Data Verification	Abraham Whaanga (RSC)
13. Mineral Processing and Metallurgical Testing	Marius Phillips (PBG)
14. Mineral Resource Estimates	Abraham Whaanga (RSC)
15. Mineral Reserve Estimates	Not Applicable
16. Mining Methods	Gary Davison (Mining One)
17. Recovery Methods	Marius Phillips (PBG)
18. Project Infrastructure	Marius Phillips (PBG) for Sections 18.1 to 18.7 Gary Davison (Mining One) for Sections 18.8.3 and 18.9 Timothy Rowles (KPPL) for Section 18.8 and 18.10
19. Market Studies and Contracts	Abraham Whaanga (RSC)
20. Environmental Studies, Permitting, and Social or Community Impacts	Abraham Whaanga (RSC)
21. Capital and Operating Costs	Gary Davison (Mining One) for Sections 21.1 and 21.2 Marius Phillips (PBG) for Sections 21.1.3 and 21.2.2 Timothy Rowles (KPPL) for Sections 21.1.4 and 21.2.3
22. Economic Analysis	Gary Davison (Mining One)
23. Adjacent Properties	Abraham Whaanga (RSC)
24. Other Relevant Data and information	All QPs
25. Interpretation and Conclusions	All QPs
26. Recommendations	All QPs

2.4 Site Visits

The following site visits have been conducted:

- Abraham Whaanga (RSC) conducted two site visits with the following focus areas:
    - 24–25 January 2025: Auld Creek diamond drilling rig, trench locations, and geology
    - 19–21 January 2026: drill core, logging procedures, and quarter-core check samples.
- Marius Phillips (PBG) and Gino Calderon Vizcarra (KPPL) visited the site on 12 March 2026. During this visit they inspected the proposed site of the processing plant IWF and inspected the general area above the Auld Creek deposit from tracks and from helicopter. They reviewed existing infrastructure (such as public roads, existing buildings) and inspected Devils Creek that runs through the proposed lease.
- Darryl Dyason and Brett Cosgriff (Mining One) visited the site on 17 February 2026 to gain operation context and understand site technical constraints. Gary Davison visited the site on 12 May 2026.The following key information was obtained and used for this study:
    - Mostly narrow-vein deposit geometry, requiring selective underground mining methods.
    - Deposit extends 350 m below surface and requires top access to early high-grade material.
    - Poor to very poor rock mass quality (Q′ ~1–2), requiring supported spans.
    - Requirement to minimise surface disturbance due to permitting restrictions.
    - Portal locations constrained by existing forestry roads and topography.
    - Preference for adit-based ventilation rather than vertical shafts.
    - Crown pillar requirement of ~20 m below surface to maintain ground stability.

3 Reliance on Other Experts

All QPs have relied on certain information provided by RUA, including historical datasets, Project assumptions, and owner-directed development criteria. Reliance on RUA is limited to information of an administrative, logistical, or strategic nature. The QPs have exercised professional judgement in incorporating this information and, where material to technical conclusions, have reviewed it for reasonableness and consistency.

The QPs have not independently verified the legal status of RRPL's mineral permit and has not investigated the legality of any of the underlying agreements that exist concerning the Project. The QP (Abraham Whaanga) reviewed the RRPL permit status information on the Ministry of Business, Innovation and Employment (MBIE) — New Zealand Petroleum and Minerals (NZP&M) website. The QP (Abraham Whaanga) relied on the NZP&M website and the permit certificate issued under the CMA (permit certificate dated 19 March 2021, with a change certificate dated 18 September 2023), which states RRPL's legal status and title of prospecting and exploration. However, the QPs are not qualified to give a legal opinion with respect to property titles contained within this Report and discussed in Section 4.

4 Property Description and Location

4.1 Location

The Auld Creek Gold-Antimony Project is located in the Reefton Goldfield in the West Coast region of the South Island, New Zealand (Figure 4-1). The Project is located directly south of, and adjacent to, the town of Reefton.

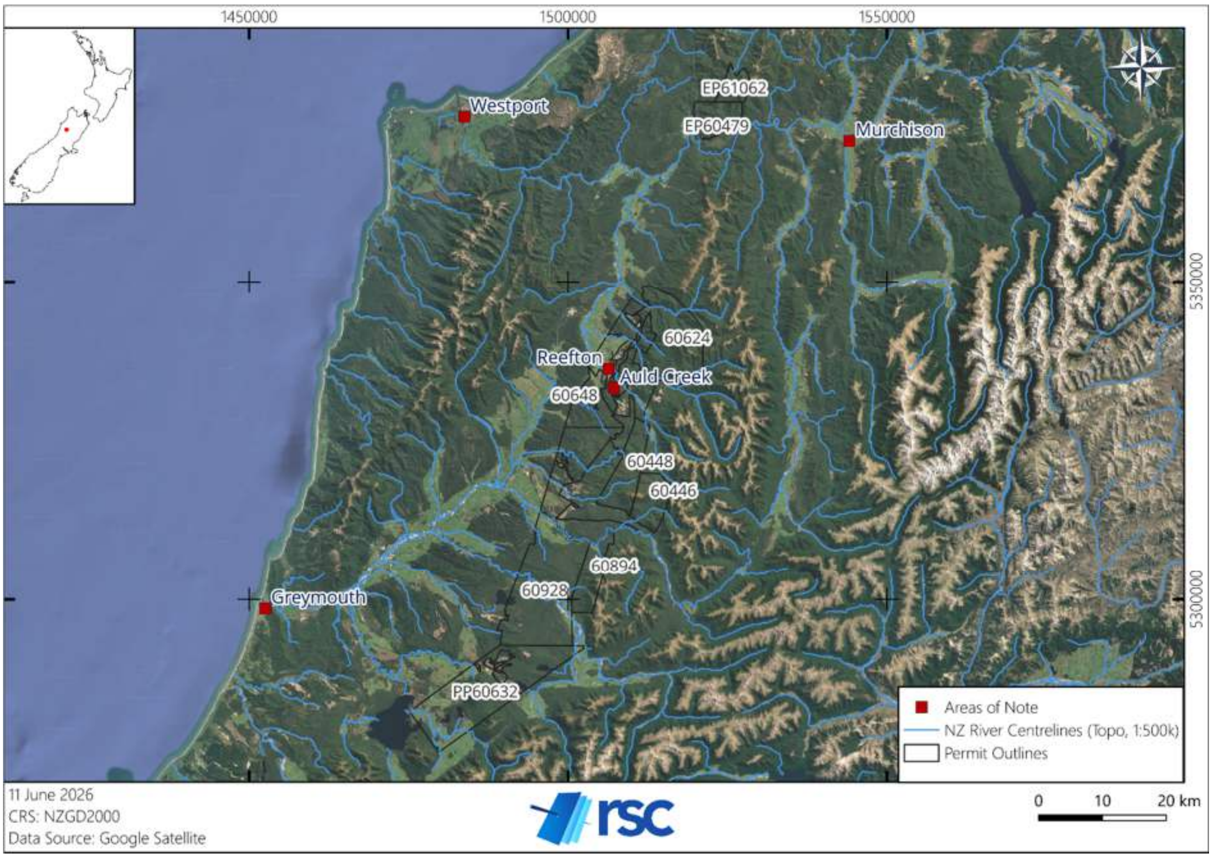


Figure 4-1: Location of the Project.

4.2 Mineral Tenure

4.2.1 Mineral Rights

Within New Zealand, the allocation of rights to prospect, explore, and mine for minerals owned by the Crown is carried out by the granting of prospecting, exploration, and mining permits under the CMA. The Minister of Resources is responsible for the CMA, and the administration of the CMA has been delegated to the Ministry of Business, Innovation and Employment (MBIE), through the brand name NZP&M — the government regulator who manages New Zealand's Crown mineral estate.

Under the CMA, all petroleum, Au, silver (Ag), and uranium (U) in its natural state is deemed to be owned by the Crown, and pounamu (greenstone) is owned by Te Rūnanga o Ngāi Tahu. The granting of a prospecting, exploration, or mining permit provides the permit holder the right to prospect, explore, or mine the minerals specified in the permit. Applications for uranium and thorium minerals will ordinarily be declined.

Permits under the CMA are classified as either Tier 1 or 2, depending on the minerals they relate to, expected work programme expenditure, estimated production or royalty, and where the activities take place. All prospecting permits are classified as Tier 2. Exploration permits for Au are classified as Tier 1 unless the expected total work programme expenditure for the final five years of its life is NZD <1,250,000. Mining permits for Au, Ag, and platinum group metals (PGMs) are classified as Tier

1 if, in any one permit year in the next five years of its life, the annual royalty will be equal to or more than NZD 50,000. Mining permits for any other metallic mineral are classified as Tier 1 if, in any one permit year in the next five years of its life, the annual production will be equal to or more than 500 kt of metallic minerals ore. All underground operations are classified as Tier 1.

4.2.1.1 Exploration Permits

Exploration is any activity undertaken for the purpose of identifying mineral deposits or occurrences and evaluating the feasibility of mining.

An exploration permit gives the permit holder the same rights as a prospecting permit, plus the exclusive right to explore for the Crown-owned minerals referred to in the permit, in the land covered by the permit and in accordance with the permit's conditions. An exploration permit cannot authorise exploration for privately owned minerals (noting, however, that all petroleum, Au, Ag and U, existing in its natural state is deemed to be owned by the Crown under the CMA).

Subject to the permit conditions, the holder of an exploration permit has a prima facie right to be granted a subsequent mining permit, in respect of the land and Crown-owned minerals to which the exploration permit relates, if the exploration is successful.

An exploration permit (for minerals other than petroleum) is typically granted for a period of five years, with the possibility of an extension for a further five years. There are no rights of renewal beyond 10 years, except for appraisal purposes. Appraisal extensions may extend the duration of an exploration permit by up to eight years. When an exploration permit for minerals is renewed, the Minister typically requires relinquishment of half of the permit area.

NZP&M does not specify a maximum size for an exploration permit but does dictate that an exploration permit must not be smaller than 150 hectares.

There is a minimum annual fee for exploration permits that are payable to the Crown. For onshore exploration, the fee is NZD 358.00 per square kilometre, or part of a square kilometre, or NZD 1,610.00, whichever is greater.

EP 60648 is held by RRPL and was issued under the CMA.

4.2.1.2 Revocation of Permits

The Minister may revoke a permit if:

1. the permit holder contravenes a condition of the permit, the CMA, or regulations made under the CMA;
2. the permit is a Tier 1 permit, the permit holder is the permit operator, and the permit holder undergoes a change of control without the Minister's consent; or
3. the permit holder undergoes a change of control without notifying the Minister, or the Minister is not satisfied the permit holder, following the change of control, has the financial capability to meet its obligations under the permit.

The conditions for EP 60648 are in Schedules 1–3 of the permit certificate.

4.2.2 Permit Status

RRPL is 100% holder and operator of EP 60648 issued under the CMA (Table 4-1 and Figure 4-1). The Project covers an area of 40.64 km^2.

Table 4-1: Status of EP 60648.

Owner	Operation Name	Tier	Commodity	Date Granted	Term (years)	Expiry Date	Area (km^2)	Comment
RRPL (100%)	Golden Point	2	metallic minerals	19 Mar 2021	10	18 Mar 2031	40.6405	EOD granted 28 May 2026; status will change to Tier 1 from 19 March 2027

4.2.2.1 Work Programmes

An applicant for a permit under the CMA must propose a minimum work programme for the proposed permit. The Minister will not grant the permit unless the Minister is satisfied the work programme is consistent with the CMA, the purpose of the permit, and good industry practice, and that the applicant is likely to comply with and give proper effect to the work programme. In addition, the work programme for a subsequent permit or permit EOD must be approved by the Minister. A permit holder may apply to the Minister to change the work programme for the permit.

A summary of the minimum work programmes for the Project, including the status and due dates of permit obligations, is given in Table 4-2.

Table 4-2: Minimum work programme for EP 60648.

Item	Type of Activity	Due Date	Status
1a–1g	literature review, geochemical, test pitting, target identification, drilling, data compilation, reporting	19 Mar 2021	completed
2a–2e	geochemical, drilling, data compilation, define Inferred resource, reporting	19 Mar 2026	completed
3a–3f	geochemical, drilling, data compilation, update/define mineral resource, reporting	19 Mar 2029	ongoing
4a–4f	drilling, data compilation, update mineral resource, resource definition drilling, pre-feasibility study, reporting	19 Mar 2031	not yet started

4.3 Surface Rights & Permits

The granting of a permit under the CMA does not confer a right of access to the land covered by the permit, except for certain minimum impact activities.

Subject to some limited exceptions, the permit holder must have an access arrangement with each owner and occupier of the land to carry out more than minimum impact activities on or under the land. The permit holder is also required to give 10 working days' notice to the landowner and occupier prior to conducting activities. An access agreement may be either agreed by the parties or determined by an arbitrator under the CMA. An access agreement is binding on the owner's or occupier's successors in title.

An activity carried out below the surface of the land does not require an access agreement if the activity will not, or is not likely to:

1. cause any damage to the surface of the land or any loss or damage to the owner or occupier of the land;
2. have any prejudicial effect regarding the use and enjoyment of the land by the owner or occupier; or
3. have any prejudicial effect regarding any possible future use of the surface of the land.

Access to Crown land requires permission from the relevant Minister of the Crown with responsibility for the land. To sample Crown land, held or managed under the *Conservation Act 1987* (Conservation Act) or other Acts specified in Schedule 1 of the Conservation Act, the permit holder must gain consent or an access arrangement (AA) from DOC. Permit holders require a concession (this differs from an

AA, which is stricter) from DOC to conduct minimum impact activities (MIA) on conservation land. For all other exploration and mining activities on conservation land, the permit holder will require an AA from DOC. If an AA is sought for conservation land, the Minister of Conservation must determine whether the proposed mining activities are ‘significant’. If the activities are ‘significant mining activities’, the application for land access must be publicly notified with a submission period.

Exploration permits give the permit holder the exclusive right to explore for specified minerals in the permit area using higher-impact exploration methods, such as drilling and earthworks. However, any exploration activity must either be allowed under the *Resource Management Act 1991* (1991b) (RMA) or permitted by a granted resource consent.

The RMA classifies activities into six primary categories: permitted, controlled, restricted discretionary, discretionary, non-complying, and prohibited. These categories determine whether resource consent is required before carrying out an activity, and what will be considered when resource consent application is assessed. National Environmental Standards and Regional and District Plans regulate which category an activity falls into and, therefore, whether resource consent is required.

Much of the land within the Project area was State Forest Land, gazetted in 1981 as the Victoria State Forest Park. This land was subsequently renamed the Victoria Conservation Park and came under the administration of DOC under the Conservation Act. DOC, therefore, has primary responsibility for the conservation of New Zealand’s natural and historical heritage. DOC also has responsibilities under other related legislation, including the *National Parks Act 1980* (1980) and the *Reserves Act 1977* (1977). Parts of the land within the permit area have further conservation protection with the additional gazettal of Wildlife Management Areas, Amenity Areas, and Ecological Areas. Timberlands West Coast administers exotic and some indigenous forest stands. Freehold land forms a minority of the tenement distribution.

An MIA concession gives access to the land to conduct non-mechanical exploration, such as surficial geochemical sampling, geophysical surveying, and mapping. RRPL previously held an MIA (102174-MIA) for EP 60648, which expired on 18 March 2026.

RRPL also previously held an AA for EP 60648 (93190-AA), which expired on 18 March 2026. An AA allows for more intrusive work, including exploration drilling. RUA submitted a variation to extend the AA duration in February 2026 and is currently operating under an interim authority to enter and operate (AEO) while the AA is pending.

The QP (Abraham Whaanga) reviewed RRPL’s agreement with DOC concerning the exploration work programme for EP 60648. No significant risks were identified with regards to RRPL holding sufficient surface rights to allow effective exploration in the permit areas.

4.4 Royalties & Encumbrances

4.4.1 Crown Royalties

One of the purposes of the CMA is to provide a fair financial return to the Crown for its minerals, which is achieved through a system of mandatory Crown royalties.

The Crown Minerals (Royalties for Minerals Other than Petroleum) Regulations 2013 (Royalty Regulations) set out rates and provisions for the payment of Crown royalties on non-petroleum mineral production. The Royalty Regulations provide for the payment of royalties on exploration and mining permits, to the extent minerals are produced from the permits.

Subject to certain thresholds (notably, a net sales revenue threshold of NZD 200,000 per annum), the royalty regime under the Royalty Regulations for Tier 1 permits, for metallic minerals, is:

- for Au and net sales revenue from Au, of not more than NZD 2 million per annum, an ad valorem royalty of 2% of net sales revenue; and otherwise
- the higher of an ad valorem royalty of 2% of net sales revenue or an accounting profits royalty of 10% of accounting profits.

For Tier 2 permits, the royalty regime under the Royalty Regulations for metallic minerals is an ad valorem royalty of 1% of the net sales revenue(s) of the minerals obtained under the permit.

4.5 Environmental Liabilities & Permits

New Zealand's principal environmental legislation is the RMA.

The RMA regulates the impacts of all activities on the natural and physical environment, including land, water, and air. An activity must be permitted under either:

- the relevant district or regional plan (which is administered by the relevant district or regional council);
- a resource consent granted by the relevant district or regional council; or
- the RMA itself, or a regulation made under the RMA.

Activities are typically permitted subject to conditions, such as to mitigate environmental effects in various ways, to monitor and report, or to pay an environmental bond.

The RMA contains a general duty to avoid, remedy, or mitigate any adverse effect on the environment arising from an activity, whether the activity is permitted or not.

If a resource consent is required for an activity, an application must be made to the relevant district or regional council. Regional councils manage resources like water, air, and coastal areas, whereas district councils focus on land use and development. Resource consents may be granted or declined and are subject to appeal procedures. Unless the environmental effects of the activity are minor and written approvals have been obtained from any affected parties, resource consent applications will be notified, and third parties or the general public will be able to submit on whether the activity should be consented and on what conditions.

A variety of injunctive and compensatory enforcement orders are available under the RMA to prevent, remedy, and provide compensation for environmental non-compliance. In serious cases, resource consents can be cancelled. It is an offence to contravene the principal sections of the RMA. Offences attract significant fines of up to NZD 600,000 for a company, with the possibility of an additional penalty in the case of commercial gain.

To the best of RRPL's knowledge, no breaches of the RMA or any other environmental laws have been committed. Neither entity has been the subject of any enforcement proceedings for breaches of environmental obligations.

RRPL holds the necessary permits under the CMA for the current prospecting and exploration activities (see Section 4.4).

RRPL has, or is expected to have, the necessary access agreements in place for its current prospecting and exploration activities (see Section 4.4).

Based on a review of EP 60648, issued to RRPL by NZP&M, concerning exploration in the Reefton area, and other available material, the QP (Abraham Whaanga) has not identified anything to suggest RUA does not hold sufficient permits as of the effective date of this Report to allow effective exploration of the permit area.

4.6 Other Significant Factors & Risks

Mining in New Zealand is a sensitive and political subject and, as in many other countries, there are active anti-mining groups. Exploration and mining projects within New Zealand can be the subject of negative campaigns by emboldened local and online anti-mining groups. Exploration and mining projects within New Zealand can also be the subject of negative social media campaigns by emboldened local and online anti-mining groups. In 2019, Plaman Resources lost its social licence to operate at the Foulden Hills Diatomite Mine, Otago[1]. A negative social media campaign resulted in that project losing funding and thus being unable to proceed. The QP (Abraham Whaanga) notes that while there is some risk of social licence issues, the West Coast, Buller, and Grey regions have stronger support for mining than the rest of New Zealand.

The Reefton area has an extensive mining history, and mining has made a significant contribution to the economy and employment in the area. Accordingly, the social licence for mining in the area is more favourable compared with other areas of New Zealand. A heavy mineral sands project is consented and operating on private land ~42 km northwest of Reefton at Nine Mile Beach, and the Globe-Progress Au mine ~7 km southeast of Reefton operated from 2007–2016; both operations had support from West Coast locals. Recent mining related consents include Endura Mining (previously Federation Mining) for an on-site processing plant at the nearby Snowy River mine in January 2023. Consents were granted by the Buller District Council and the West Coast Regional Council under a limited notification process. The Grey District and West Coast Regional Councils granted consent on 29 April 2024 for the Barrytown mineral sands mine, following a public hearing. The consent was subject to conditions including traffic plan, lighting plan, and avian management plan. This consent was subject to an appeal to the Environment Court, which was resolved when the operating company, TiGa Minerals and Metals, agreed to amend conditions and reduce its hours of operation.

The current New Zealand Government, which is a National-led coalition with New Zealand First and Act parties, strongly supports mining and has discussed plans to double mining earnings over the next decade. This has created a very positive environment for mineral exploration, and the potential to fast-track the permitting processes in the event of a discovery through the *Fast-track Approvals Act 2024* (2024) (Fast-track). The Act establishes a permanent, fast-track approvals regime for projects of national and regional significance. It is administered by the Ministry for the Environment, with other agencies responsible for specific Acts related to project approvals, including DOC, MBIE, the Ministry for Primary Industries, Heritage New Zealand Pouhere Taonga, and the Environmental Protection Authority. The system will be a ‘one-stop shop’ for resource consents, notices of requirement, certificates of compliance under the RMA, and approvals required under several other Acts, including the CMA, the Conservation Act, and the *Wildlife Act 1953* (1953). The Fast-track received royal assent on 23 December 2024.

While there is always some risk of social licence issues, the exploration targets are underground, with minimal anticipated surface impacts. The QP (Abraham Whaanga) is of the opinion that these are more likely than not to have regulatory and public support, as opposed to operations with a larger surface footprint.

The QP (Abraham Whaanga) notes that RUA regularly engages with iwi (indigenous Māori tribes) and hapū (Māori sub-tribes), whose rohe (territory) includes some or all of the Project area, or who may be directly affected by the Project, mainly Ngāi Tahu and Ngāti Waewae.

[1] https://www.newsroom.co.nz/southern-discomfort-at-fossil-mining-plans

On balance, the QP (Abraham Whaanga) considers that the environmental, social, and governance factors are sufficiently well understood and proactively managed to support the MRE's reasonable prospects of economic extraction at the effective date of this Report.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Project is located directly south of, and adjacent to, the town of Reefton. Access is via a mixture of main highways, sealed and unsealed public roads, 4WD tracks, and foot access tracks (Figure 5-1). The permit area is mainly within the Victoria Forest Park Conservation area. Local roads that lead off the highways provide vehicle access to various parts of the Project, and old mining access roads locally provide 4WD access to the major historical mines; DOC maintains recreational walking tracks within the permit area.

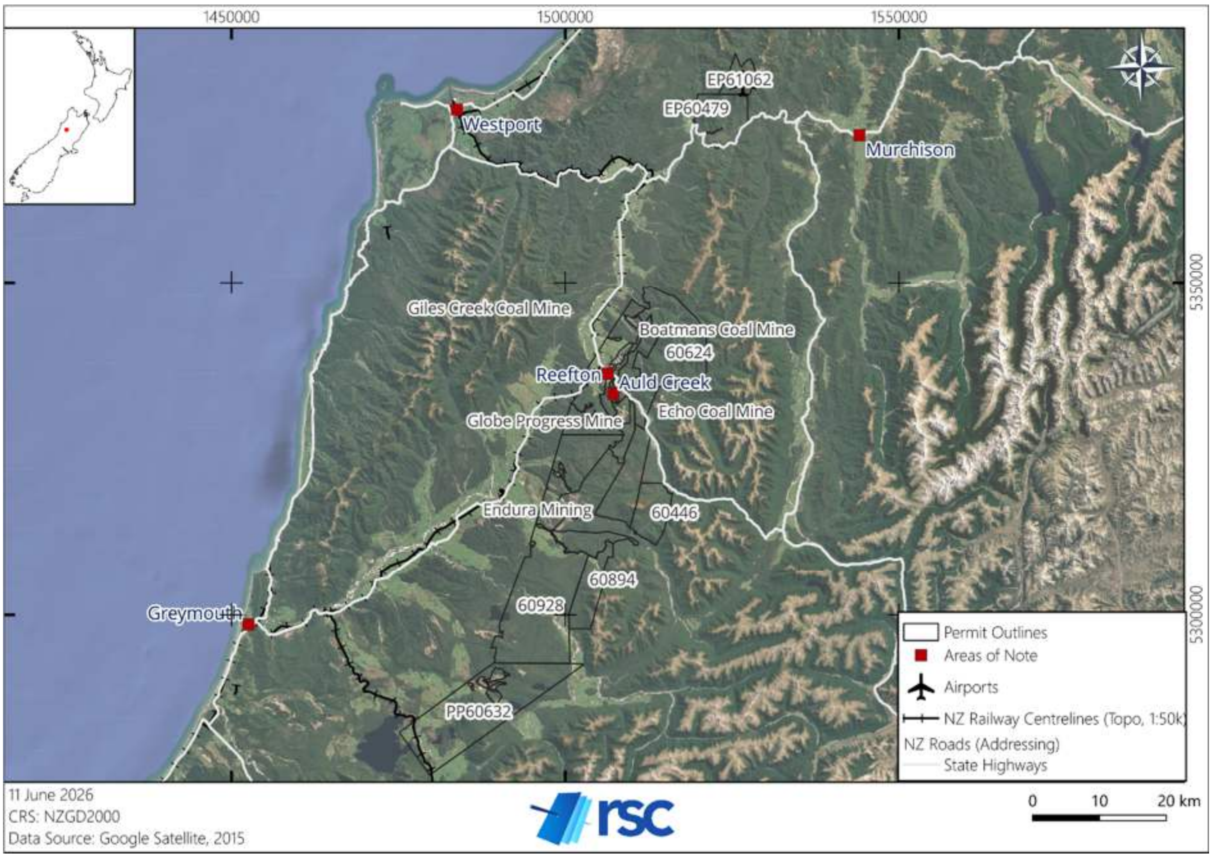


Figure 5-1: Map illustrating accessibility to the Project.

5.2 Climate

The Project area is in the rain shadow of the Paparoa Range. The climate is wet and temperate, with average annual rainfall in Reefton of 1,920 mm. Spring is the wettest season, and late summer and early autumn are the driest. Summer weather is often mild and relatively dry. Frosts and fogs are common in winter, with an average of 2 days of snow and 68 days of ground frost per year. Average mean temperatures range from 5°C in winter to 17°C in summer. Field work can be conducted year-round, but field activities can be restricted any time of the year by periods of extreme weather (i.e. heavy rain).

5.3 Physiography

The Project area is situated mainly in hilly country with moderate to steep relief in the foothills of the Victoria and Brunner ranges (Figure 5-2). The topography is locally very steep and varies in elevation from ~100 m to >1,000 m above sea level. Creeks and rivers strongly incise the area, and the steep topography often limits field work access to foot or helicopter.

Figure 5-2: Typical topography at the Project.

5.4 Vegetation

The dominant vegetation on mountain slopes below 1,000 m in the Reefton area is mixed, regenerating indigenous beech (*Nothofagus* spp.) and podocarp (principally rimu) forest growing on poor and immature soils. Alpine scrublands and grasslands are present at higher altitudes, and flat floodplains are observed along the Grey and Ahaura rivers. There are also areas with exotic pine plantations near the township of Reefton. The vegetation coverage in the Project area can limit access for exploration activities and regenerates rapidly; for example, drill pads cleared ~2.5 years ago at the nearby Alexander River site are already covered by ferns (Figure 5-3).

Figure 5-3: Indigenous beech and drill pad with regenerating vegetation at the nearby Alexander River site.

5.5 Local Resources & Infrastructure

The permit area is located within the Buller district and is well connected by state highways and public roads to nearby towns. The nearest hospital is in Greymouth. The closest regional airport is in Hokitika, which connects to Christchurch International Airport, and there is also an airport in Westport with scheduled flights to Wellington. Reefton is connected to New Zealand's rail network. There are small ports located at Westport and Greymouth, which typically service small-tonnage coastal freight and fishing vessels.

To support its exploration, RUA has an exploration office in the township of Reefton. The exploration office includes a small laboratory for processing soil samples, sample and supply storage, core cutting, and core logging.

Cell phone coverage for much of the Project area is poor. VHF radios are used for communication between the RUA base and drill sites; GPS units, with satellite communication functions (Garmin InReach), are used for communication between the RUA base and surface sampling teams.

Water at the drill sites is sourced from the nearest creek. Depending on the distance to the nearest water source, a series of pumps are used to pump water to the drill sites. Power at the camp and drill sites is provided by diesel-fuelled generators.

The West Coast region of the South Island has an active mining industry; therefore, there are numerous skilled contractors and organisations in the area that can support exploration and mining activities.

6 History

The history of the Project area has been described in detail in prior technical disclosure, including Aldrich and Whaanga (2024) and Whaanga (2026). Sections 6.1 to 6.4 provide a brief summary of the exploration/drilling, operating, and production history of the Project.

6.1 Tenure & Operating History

The Project is located in the Reefton Goldfield, which contains numerous alluvial and hard-rock Au mines. Much of the Reefton Goldfield was previously held by Lime and Marble Limited (L&M) between 1970 and 1971 (Riley and Ball, 1971), and then by CRA Exploration Limited (CRAE) between 1981 and 1990. Macraes Mining Co Ltd (MMCL) (later OceanaGold New Zealand Ltd; OGL) then held most of the goldfield from the late 1990s until the Capleston and Globe-Progress areas were relinquished in 2018 and 2020, respectively.

6.2 Exploration History

The Project area has been variably explored by several operators prior to RRPL's tenure (Table 6-1) (Aldrich and Whaanga, 2024; Whaanga, 2026).

Table 6-1: Summary of historical exploration in the Project area.

Operator	Geochemical Sampling (No. of Samples)			Trenching & Channel Sampling	Drilling	Geophysics
	Soil	Stream	Rock-Chip			
CRAE	550	-	82	11 trenches (80 m)	-	Ground magnetics
L&M	-	-	-	3 trenches (13.22 m)	-	-
MMCL	323	55	-	10 trenched (109 m)	324.6 m (3 holes) – diamond drilling	-
OGL	273	-	-	-	1,630.9 m (13 holes) – diamond drilling	-

In 2011, NZP&M, on behalf of the New Zealand Government (the Crown), commissioned an airborne magnetic survey of the West Coast region, including the Project area (Vidanovich, 2013). The West Coast Airborne Geophysical Survey was conducted between February 2011 and March 2013 (Figure 6-1). The aeromagnetic grids produced during the survey included several different industry-standard variants, including total magnetic intensity (TMI), reduced to pole (RTP), first vertical derivative (1VD), second vertical derivative (2VD), analytical signal (AS), and automatic gain control (AGC).

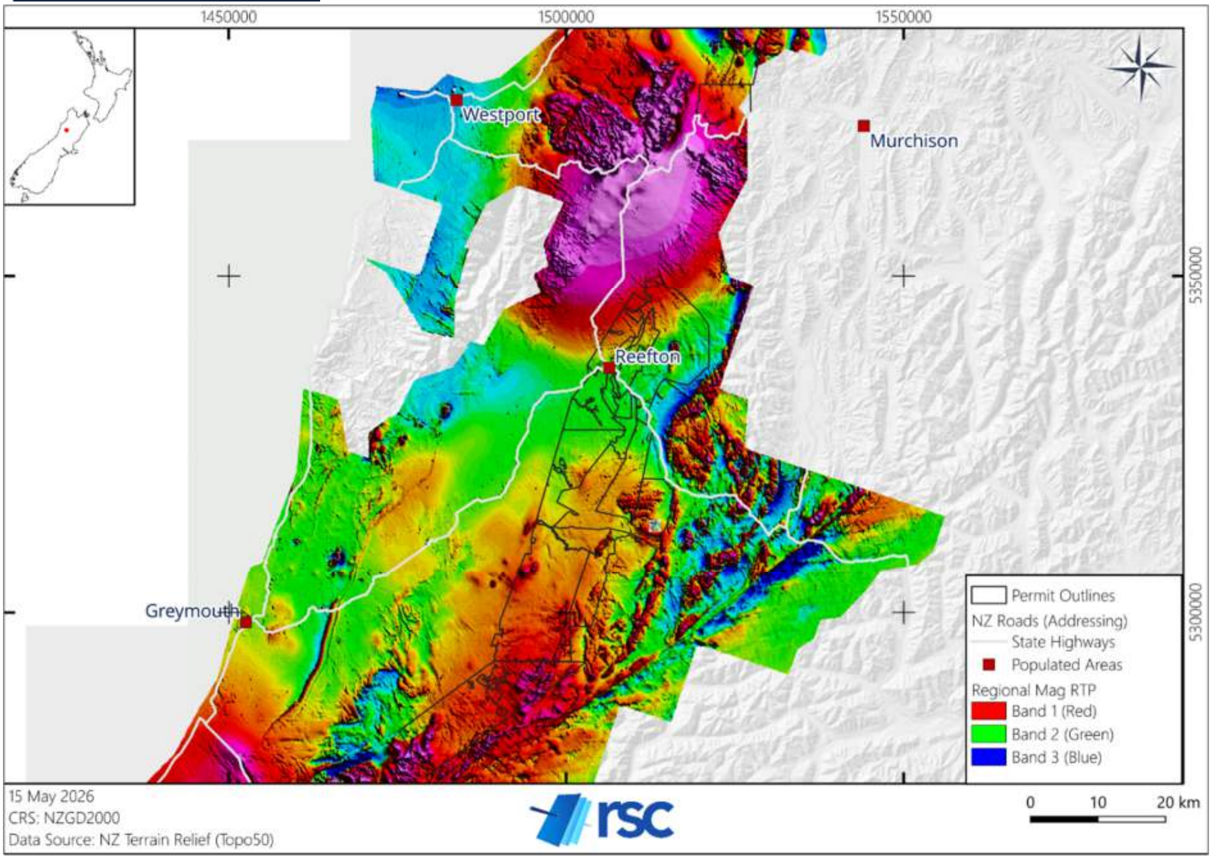


Figure 6-1: Magnetics image (RTP) over the Reefton and Lyell goldfields.

6.3 Production History

The mining history of the wider Reefton Goldfield includes both alluvial and hard-rock mining (Figure 6-2). Previous Au production has been significant, totalling ~11 Moz Au from alluvial, open-cut, and historical underground mines (Siren Gold Limited, 2023). Stibnite, which is an antimony (Sb) sulphide mineral (Sb_2S_3), is also present in quartz lodes of the Reefton Goldfield, where quartz veins have been reported to contain up to 10–30% stibnite (Finlayson, 1909). However, stibnite has not been exploited commercially from the Reefton Goldfield to date.

The first discovery of Au within the Reefton area was made by John Redman in 1866, during the peak of the first West Coast gold rush (Barry, 1993). Mining in the Reefton Goldfield began in the 1870s, with initial production from several areas around Reefton, including Murray Creek, Andersons Creek, and the Golden Fleece shoot (Barry, 1993). Mining operations continued throughout the Goldfield in the 1870s and 1880s, until discoveries of new deposits dropped off in the late 1890s and early 1900s. Other than the discovery of the Alexander reefs to the south of Reefton in 1920, there were no further significant discoveries, and Au production steadily declined as the Globe-Progress, Wealth of Nations, and Keep-it-Dark mines closed in the 1920s and 1930s (Barry, 1993).

At the outbreak of World War II, the Big River and Blackwater mines were the only producers. Wartime labour shortages were responsible for the closure of the Big River Mine in September 1942. When the Blackwater shaft collapsed on 9 July 1951, the ventilation and drainage systems of the Blackwater Mine were disabled and 81 years of continuous quartz-hosted Au mining activity in the Reefton Goldfield came to an end (Barry, 1993).

Hard-rock Au mining did not recommence in Reefton until 2007 when OGL reopened the Globe-Progress mine. Construction on the Project started in 2005, consisting of a surface mine and processing plant to grind and concentrate the mined ore. The mine yielded 606 koz Au over the life of the open-pit operation, which ceased production in 2015. In total, 12.89 Mt of ore were processed, with an average grade of 1.8 g/t Au. Globe-Progress transitioned to closure and rehabilitation in 2016 and is now known as the Reefton Restoration Project (Edwards, 2020).

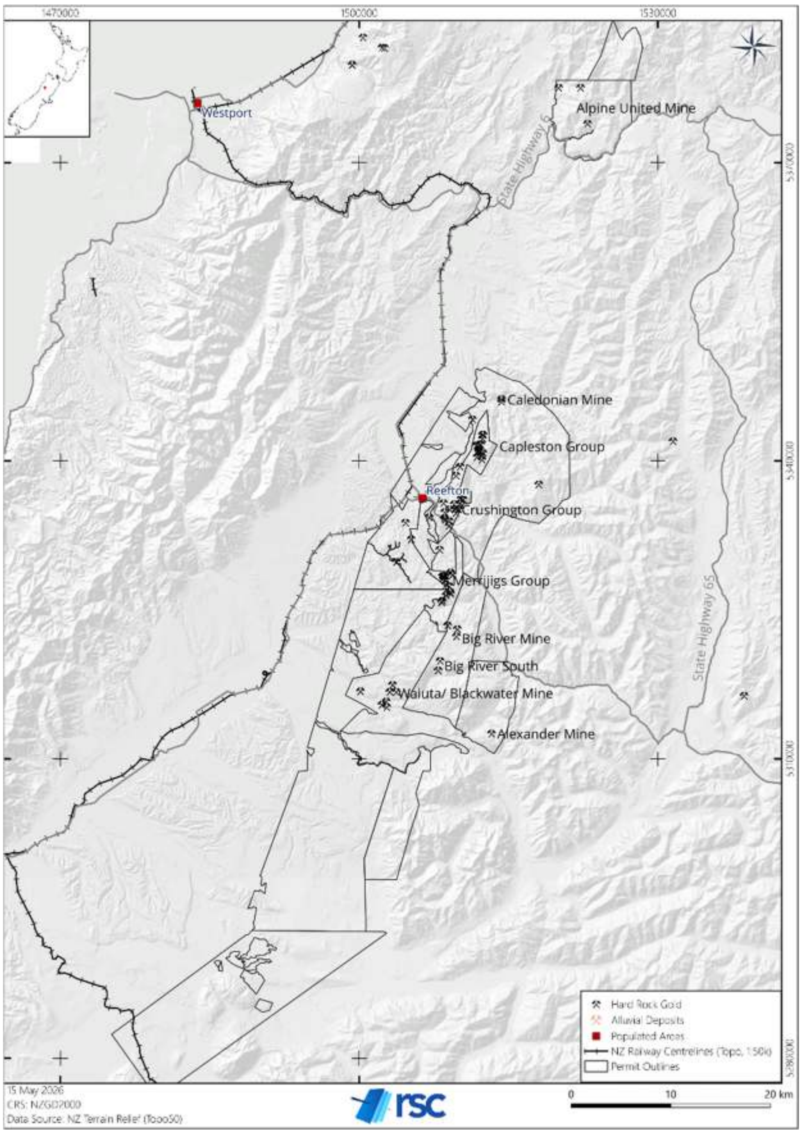


Figure 6-2: Locations of historical hard-rock and alluvial mining centres in the vicinity of the Project.

6.4 Historical Mineral Resource Estimates

In September 2024, RRPL reported Au and Sb resources for the Project (McCulloch, 2023b). A further historical estimate was reported by RUA in October 2024 (Aldrich and Whaanga, 2024). These historical estimates are summarised in Sections 6.4.1 and 6.4.2.

6.4.1 RRPL

In September 2024, RRPL completed a historical estimate for the Auld Creek prospect in the Golden Point permit area with an effective date of 17 September 2024 (Siren Gold Limited, 2024b). The historical estimate was assessed using historical MMCL and OGL drilling data alongside RRPL drillhole and trench data, including a geological database, QA/QC procedures and results, laboratory results, topographic surfaces, and geological interpretations (Table 6-2).

Table 6-2: Auld Creek prospect historical estimate as of 17 September 2024 (Siren Gold Limited, 2024b).

Zone	Classification	Tonnes (kt)	Au (g/t)	Contained Au Ounces (koz)	Sb (%)	Sb (kt)	AuEq (g/t)	Contained AuEq (koz)
Fraternal	Inferred	614.1	3.9	77	1.4	9	7.1	104
Bonanza East	Inferred	234.4	3.6	27	2.5	6	9.3	70
Total	**Inferred**	**848.5**	**3.8**	**105**	**1.7**	**15**	**7.7**	**210**

Notes:

1. *All figures are rounded to reflect appropriate levels of confidence.*
2. *Differences in totals may occur owing to rounding.*
3. *Reported in accordance with the JORC Code (2012).*
4. *The QP (Abraham Whaanga) has not done sufficient work to classify the historical estimate as current mineral resources, and RUA is not treating the historical estimates as current mineral resources.*
5. *Refer to additional disclosure below for this historical estimate.*

Required disclosure under Section 2.4 of NI 43-101 (Disclosure of Historical Estimates)

- The September 2024 Auld Creek historical estimate was reported in accordance with the JORC Code (JORC Code, 2012) and included in a Competent Person's JORC report with an effective date of 17 September 2024 (Siren Gold Limited, 2024b).
- The September 2024 Auld Creek historical estimate is considered reliable and relevant by the QP (Abraham Whaanga), as it was an updated resource estimate. However, it has been superseded by the current MRE disclosed in Section 14 of this Report.
- The September 2024 Auld Creek historical estimate was reported at a cut-off grade of 1.5 g/t AuEq, which was considered appropriate for an underground mining operation. An Au equivalent formula of AuEq = Au g/t + 2.25 × Sb% was based on the average metal prices for the preceding 12 months of USD 2,160 per ounce of Au and USD 15,625 per tonne of Sb. Metallurgical recoveries of 90% were used for both Au and Sb, based on metallurgical test work results announced in September 2024.
- The September 2024 Auld Creek historical estimate uses similar categories to those set out in Section 1.2 of NI 43-101 but was classified using the JORC Code (2012), which uses resource classifications similar to the resource classifications under the CIM Definition Standards (May 2014).
- The QP (Abraham Whaanga) has not done sufficient work to classify the September 2024 Auld Creek historical estimate as current mineral resources, and RUA is not treating this historical estimate as current mineral resources, as it has been superseded by the current MRE disclosed in Section 14 of this Report. The purpose of stating this historical estimate in the Report is to fully disclose past historical estimates for the Auld Creek prospect.

6.4.2 RUA

In 2024, RUA completed a historical estimate for the Auld Creek prospect with an effective date of 30 October 2024 (Aldrich and Whaanga, 2024).

The October 2024 historical estimate was reported in accordance with the CIM Definition Standards (May 2014), incorporating geological and assay data from 26 drillholes for a total of 3,170 m and 12 trenches for a total of 103.2 m. RUA based the October 2024 historical estimate on average metal prices of USD 2,020 per ounce of Au and USD 15,000 per tonne of Sb. An Au recovery of 97% was used based on metallurgical test work, and an Sb recovery of 85% was used based on metallurgical test work carried out by RRPL. The October 2024 historical estimate involved geological interpretation and wireframing in Leapfrog Geo based on AuEq, hard-boundary compositing in Leapfrog using the Edge module, variography and OK in Leapfrog Edge, and block model estimation in Leapfrog. The Au and Sb were estimated individually, and the AuEq was calculated based on the results. Composites for each element were based on 2-m composites, and outlier grades were assessed by reviewing composite histograms of Au grades. The October 2024 historical estimate has a cut-off of 2.5 g/t AuEq and is detailed in Table 6-3.

Table 6-3: Auld Creek prospect historical estimate as of 30 October 2024 (Aldrich and Whaanga, 2024).

Domain	Classification	Tonnes (Mt)	Au (g/t)	Contained Au Ounces (koz)	Sb (%)	Contained Sb (kt)	AuEq (g/t)	Contained AuEq (koz)
Bonanza	Inferred	0.3	2.2	19	1.0	3	4.2	35
Fraternal 1	Inferred	0.4	3.6	49	1.2	5	5.8	79
Total	**Inferred**	**0.7**	**3.1**	**67**	**1.1**	**8**	**5.2**	**110**

Notes:

1. *All figures are rounded to reflect appropriate levels of confidence.9*
2. *Differences in totals may occur owing to rounding.*
3. *The Mineral Resource is reported at a cut-off of 2.5 g/t AuEq.*
4. *Metal-equivalent grades were calculated using the following prices: USD 2,025 per ounce Au, and USD 15,000 per tonne of Sb and calculated using the formula AuEq = Au g/t + 1.9 × Sb%.*
5. *Reported in accordance with the CIM Definition Standards (May 2014).*
6. *The QP (Abraham Whaanga) has not done sufficient work to classify the historical estimate as current mineral resources, and RUA is not treating the historical estimates as current mineral resources.*
7. *Refer to additional disclosure below for this historical estimate.*

Required disclosure under Section 2.4 of NI 43-101 (Disclosure of Historical Estimates)

- The October 2024 Auld Creek historical estimate was reported in accordance with the CIM Definition Standards (May 2014) and included in a technical report compliant with NI 43-101, with an effective date of 30 October 2024, and filed on RUA's SEDAR+ profile (Aldrich and Whaanga, 2024).
- The October 2024 Auld Creek historical estimate is considered reliable and relevant by the QP (Abraham Whaanga), as it was an updated resource estimate. However, it has been superseded by the current MRE disclosed in Section 14 of this Report.
- In addition to the key assumptions, parameters, and methods described above, the October 2024 Auld Creek historical estimate was reported at a cut-off grade of 1.5 g/t AuEq.
- The QP (Abraham Whaanga) has not done sufficient work to classify the October 2024 Auld Creek historical estimate as current mineral resources, and RUA is not treating this historical estimate as current mineral resources, as it has been superseded by the current MRE disclosed in Section 14 of this Report. The purpose of stating this historical estimate in the Report is to fully disclose past historical estimates for the Auld Creek prospect.
- The QP (Abraham Whaanga) is not aware of any other recent historical estimates for the Auld Creek prospect.

7 Geological Setting and Mineralisation

The geological setting and mineralisation of the Project area have been described by detail in Aldrich (2024) and Aldrich and Whaanga (2024). A brief summary is provided in Sections 7.1 to 7.3.

7.1 Regional Geology

New Zealand lies on the boundary between the Australian and Pacific plates; the Alpine Fault delineates the boundary in the South Island (Figure 7-1). The northwest of the South Island comprises the West Coast Basin region, which is mainly composed of broad, approximately north-trending belts of early Palaeozoic rocks that terminate against the Alpine Fault in the southeast (Mortimer, 2004).

7.2 Local & Property Geology

The Reefton Goldfield is hosted entirely within Early Ordovician rocks of the Greenland Group in the Buller Terrane of the West Coast Basin (Figure 7-2) (MacKenzie, 2014; Allibone et al., 2020). In the Reefton area, the Greenland Group forms a ~35 km × 15 km north-northeast trending belt that is bounded to the north and east by granitic plutons of the Late Devonian–Carboniferous Karamea and Cretaceous Rahu and Separation Point batholiths (Laird and Shelley, 1974; Tulloch, 1988; Muir et al., 1996). In the south and west, the belt is in fault contact with high-grade paragneisses of the Paparoa metamorphic core complex (Ritchie et al., 2015).

The Greenland Group is a turbiditic sequence of alternating greywackes and argillites that were deformed and metamorphosed to lower greenschist facies in the Ordovician–Devonian (450–387 Ma), with the development of illite clay facies (Adams, 2004; Turnbull et al., 2016). The sediments are dominated by greywacke, and beds are typically 0.2–2 m thick and separated by 10–30 cm layers of argillite. The greywackes typically comprise >50% quartz with lesser albite, partially recrystallised rock fragments, and muscovite, whereas the argillites are less quartz-rich and more micaceous (Milham and Craw, 2009). Widespread folding was most likely synchronous with metamorphism, and this deformation predates granitoid emplacement (Mortimer et al., 2013).

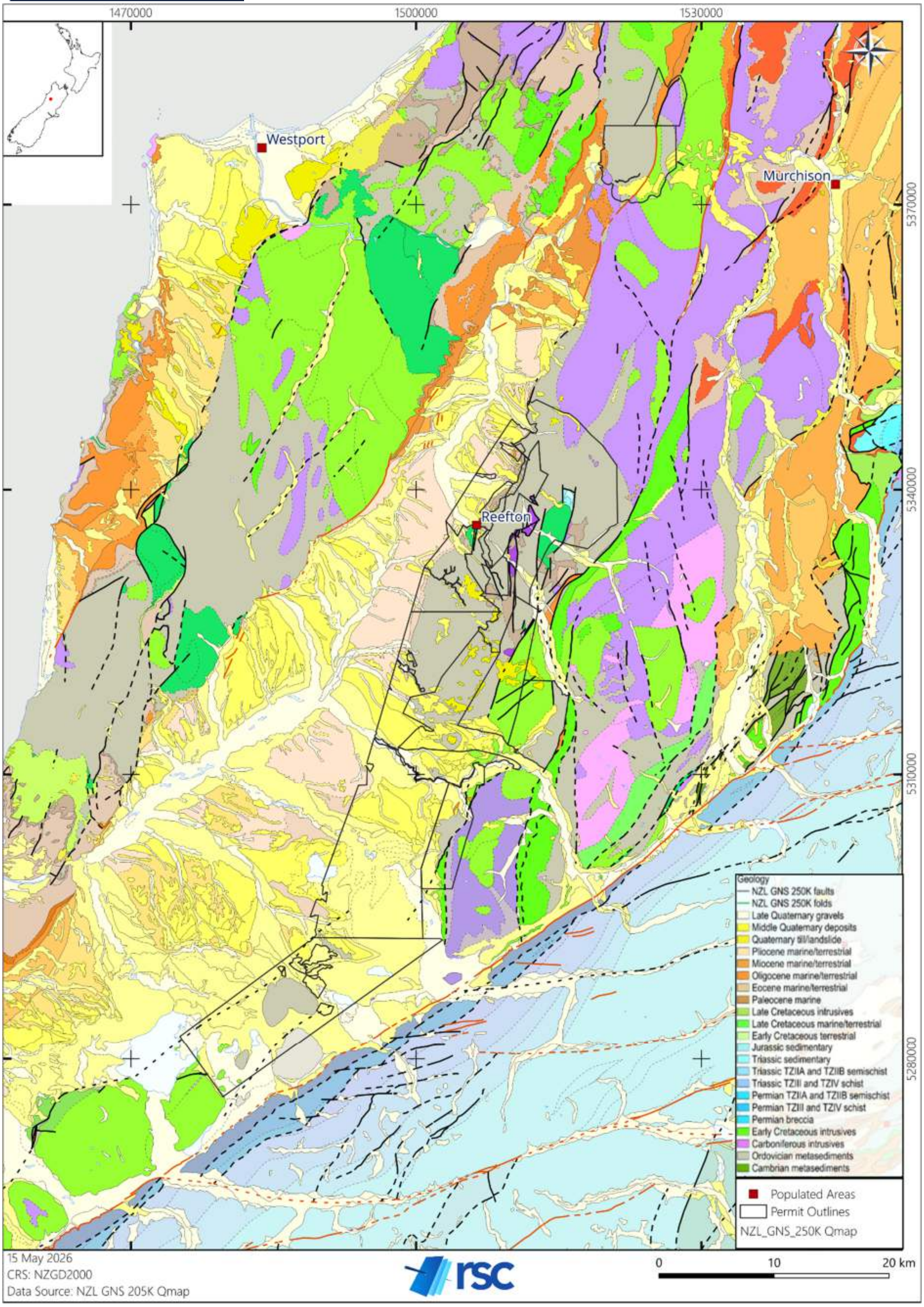


Figure 7-1: Regional geological map; QMAP 1:1,000,000 detail.

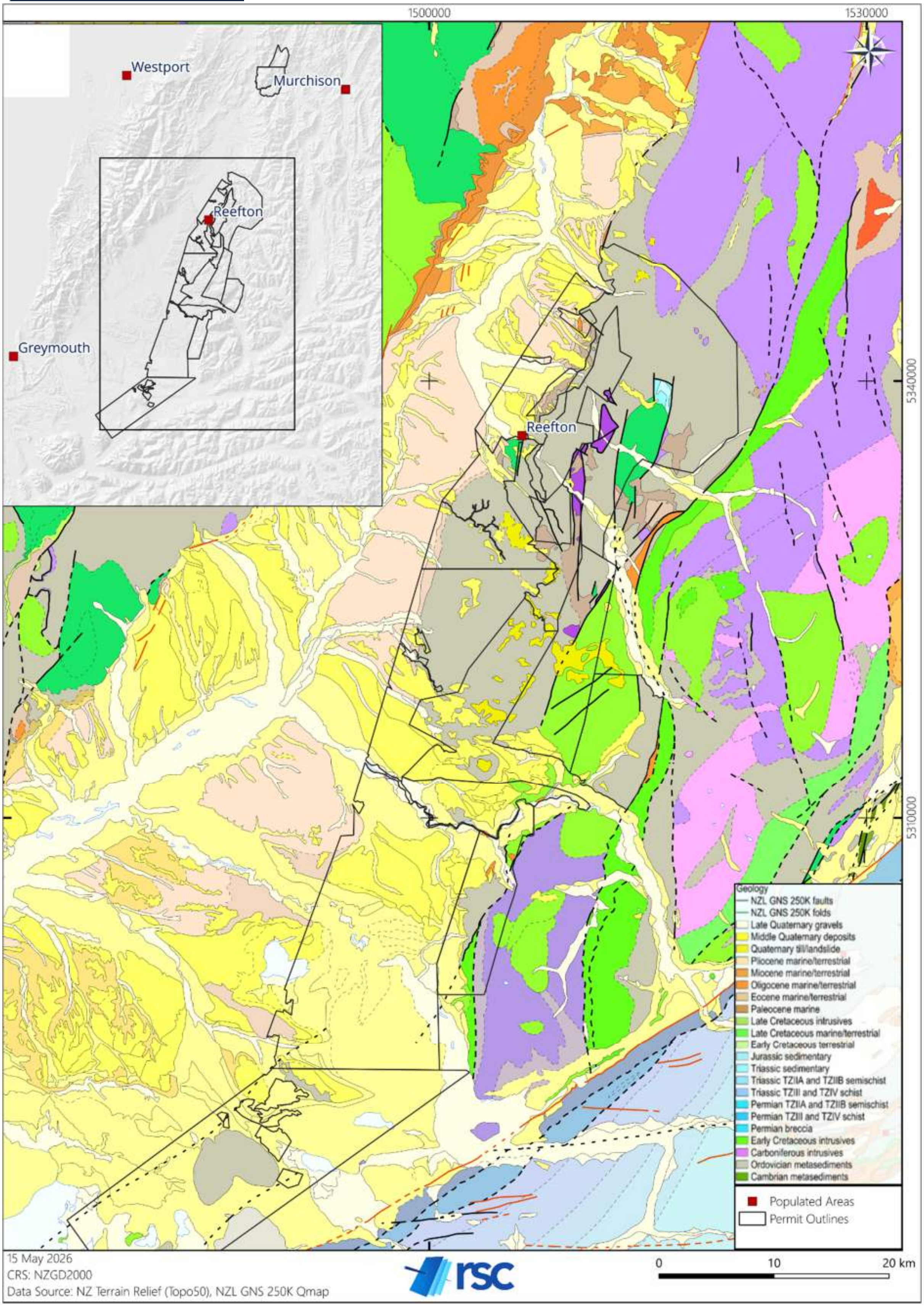


Figure 7-2: Map of geological units in the Reefton area; QMAP 1:250,000.

7.3 Mineralisation

The Au and Sb mineralisation in the Project area is orogenic style, and the deposits occur in and around steeply dipping, north to north-northeast trending shear zones that cut across the hinges of earlier folds in weakly altered metasedimentary rocks. The deposits are similar, in many respects, to those at Bendigo and Ballarat in Victoria (Cooper and Tulloch, 1992; Phillips and Hughes, 1996) and Nova Scotia in Canada (Ryan and Smith, 1998; Christie and Brathwaite, 1999).

Most of the Au- and Sb-bearing mineralisation in the Reefton Goldfield, including all of the larger deposits, occurs along an approximately north-trending linear belt that cuts a sequence of deformed metasedimentary Greenland Group rocks (Allibone et al., 2020). This suggests the presence of a deep-seated structure that has permitted mineralising fluids to migrate from their source to the upper crust, where Au and Sb were deposited.

The two dominant styles of Au mineralisation in the Project area are (MacKenzie, 2014):

1. coarse native Au associated with minor sulphides in quartz veins; and
2. microscopic refractory Au within sulphides in sheared sediments and clay alteration (pug) zones adjacent to quartz veins.

Historical production targeted mainly coarse native Au; however, both mineralisation styles provide important exploration targets (Madambi and Moore, 2013).

8 Deposit Types

There are two main hard-rock mineralisation types in the Reefton area: orogenic Au deposits in the Greenland Group, and younger intrusion-related Au ± Cu deposits related to the Devonian–Carboniferous Karamea suite of plutonic rocks on the Goldfield's eastern margin. The mineralisation types and their similarities to deposits elsewhere have been described in full by Aldrich (2024) and Aldrich and Whaanga (2024); a brief summary is provided in the following paragraphs.

The orogenic mineral system includes a diverse group of mineral deposits that form during orogenesis (e.g. Tavares Nassif et al., 2022). Associated orogenic Au and Sb lodes form in mid- to shallow-crustal metamorphic rocks in compressional settings, where Au-Sb-bearing fluids (often considered to be derived from dehydrated metamorphic rocks) migrate from depth via structural conduits and precipitate Au and Sb, typically within quartz veins, following cooling and decompression (e.g. Fyfe and Henley, 1973; Gaboury, 2019). Lode Au is the predominant economic deposit type found within metamorphic belts; however, these settings may also host Au-dominant intrusion-related deposits, as well as deposits with non-typical metal associations (e.g. Groves et al., 2003).

The oldest intrusive rocks in the Reefton area are the Devonian–Carboniferous Karamea Suite of plutons, which intrude the Greenland Group along the eastern margin of the goldfield. Various porphyry-Mo occurrences, rich in molybdenite-chalcopyrite, are associated with discrete high-level tonalite, granodiorite, and granite bodies that have intruded along pre-existing regional-scale faults that run subparallel to and along the margins of the Karamea batholith (e.g. west Nelson) (Pirajno, 1985). The Cretaceous Rahu Suite also intrudes along the eastern margin of the Reefton Goldfield and has the potential for porphyry-style mineralisation. Some discrete granitic plutons that intrude Greenland Group rocks, near the western boundary of the igneous belt, have low levels of Mo and are associated with greisen hydrothermal alteration (e.g. the McConnochie granite, Kirwans Hill, Bateman Creek and Farmer Creek) (Pirajno and Bentley, 1985).

9 Exploration

RRPL conducted LiDAR surveying across the permit area. Conventional soil sampling was completed at Auld Creek by RRPL to test various mineralised structures. RRPL used rock-chip sampling to identify exposed mineralisation across the Project area. RRPL also conducted trench sampling within the Project area, including resampling and/or extension of historical trenches in addition to new trenches. RRPL's main exploration activities in the Project area, until October 2024, are detailed in Aldrich and Whaanga (2024) and summarised in Sections 9.1, 9.2, 9.3, and 9.4.

9.1 Trenching

RRPL sampled 30 trenches within EP 60648 for a total of 241 m (Table 9-1), 14 (total of 108.9 m) of which were at the Auld Creek prospect and were used in the current resource estimate reported in Section 14. Trench locations were established using a Garmin GPS or surveyed by a professional surveyor using a Trimble real-time kinematic (RTK) GNSS with R10 rover and base units. Positions were checked against 1-m LiDAR contour maps. Trench orientations were measured using tape and a compass. Due to difficulty in obtaining accurate surveyed GPS z-values for trenches (due to steep slopes and bush cover), RRPL adjusted trench z-values by draping them onto the LiDAR surface. A full review of the trench data quality is reported in Section 11.5.

Table 9-1: Summary of RRPL trenches in EP 60648.

Date	No. Trenches	Total Length (m)
2022–2023	30	241

RRPL carried out chip sampling of trenches using a hammer and chisel, with an average sample size of ~2 kg. Veins were sampled in intervals of 0.3–2.4 m, depending on the width of the outcrop, and averaged ~1 m. Before sampling, outcrops were cleared of debris and alluvial sediments using shovels and hammers to uncover the full extent of the veins. RRPL collected field repeat samples from visible mineralisation at a rate of one per trench. Trenches were treated as drillholes, with collar, survey, lithology, and assay data compiled into an MS Excel workbook. Trench locations are reported in Table 9-2.

Table 9-2: Trench locations sampled by RRPL within EP 60648.

Trench ID	Easting (NZTM)	Northing (NZTM)	Elevation (m)	Length (m)	Year	Used in MRE	Comment
BZTR001	1507181.4	5333135.3	538.3	17.5	2022	Y	Extended (both directions)
BZTR002	1507146.7	5333151.8	504.1	5.2	2022	N	Extended (eastwards)
BZTR003	1507165.2	5333226.2	520.1	6.6	2022	N	Re-excavated
BZTR004	1507137.0	5333225.0	545.2	1.9	2022	N	New
BZTR005	1507133.0	5333245.1	556.2	4	2022	N	New
BZTR006	1507161.5	5333183.9	513.1	4	2022	N	New
BZTR007	1507132.6	5333135.7	539.1	6	2022	N	Re-excavated
BZTR008	1507190.9	5333106.9	540.6	10	2023	Y	New
BZTR009	1507199.2	5333067.7	598.9	4	2023	N	New
BZTR010	1507135.7	5333133.6	531	3.7	2023	N	Re-excavated
BZTR011	1507140.3	5333104.8	540	5	2023	N	New
FTTR001	1507243.7	5333075.2	550	13.5	2022	Y	Extended (both directions)
FTTR002	1507233.6	5333075.9	543	1.5	2022	Y	Re-excavated

Trench ID	Easting (NZTM)	Northing (NZTM)	Elevation (m)	Length (m)	Year	Used in MRE	Comment
FTTR003	1507234.9	5333166.8	519.3	7	2022	Y	Re-excavated
FTTR004	1507258.0	5333363.0	467	7.8	2022	N	Extended (westward)
FTTR005	1507239.1	5333033.5	573.1	12.8	2022	N	Extended
FTTR006	1507232.2	5333306.0	479	5.6	2022	Y	Extended
FTTR007	1507177.0	5333243.8	577	7.7	2022	N	Re-excavated
FTTR008	1507188.2	5333259.8	582.5	9.2	2022	N	New
FTTR009	1507238.0	5333483.0	438.2	10	2022	Y	New
FTTR010	1507260.6	5332902.3	606.7	5.7	2022	N	New
FTTR011	1507259.1	5332953.7	608.2	4	2022	N	Extended (eastern)
FTTR012	1507267.7	5333411.4	468	8.8	2023	N	New
FTTR013	1507229.3	5333208.8	517.7	4.8	2022	N	New
FTTR014	1507228.0	5333509.0	442.3	2.7	2023	N	New
FTTR015	1507250.3	5332956.5	621.3	11	2023	N	Re-excavated
FTTR016	1507258.6	5332985.1	597.1	10.5	2023	N	New
FTTR017	1507240.3	5333131.3	542.2	8	2023	N	New
FTTR018	1507244.3	5333019.4	563	12.5	2023	N	New

RRPL completed 30 trenches within EP 60648 (29 at Auld Creek), including 16 new trenches, re-excavation of seven historical trenches, and extension of a further seven historical trenches. One trench was excavated at the extrapolated position of the Morning Star reef track; however, no mineralisation was observed, and no samples were assayed. At Auld Creek, the trenching was successful in defining the strike, extent, thickness, and grade of mineralised shoots (Au + As + Sb) and was consistent with anomalies identified during soil sampling. The significant intercepts are summarised in Table 9-3. As of the effective date of this Report, RUA had not carried out any further trenching in the Project area.

Table 9-3: Significant trenching intercepts for Auld Creek, full mineralised zone composites (1.5 g/t Au cut-off).

Trench ID	From (m)	To (m)	Interval (m)	True Width Interval (m)	Au (g/t)	Sb (%)	Mineralised Zone
BZTR001	0.0	17.5	17.5	8.5	1.66	0.67	Bonanza
BZTR004	0.0	1.0	1.0	0.9	1.89	0.04	Bonanza
BZTR008	1.0	10.0	9.0	6.6	3.80	0.31	Bonanza
FTTR001	3.5	11.9	8.4	7.2	17.21	5.46	Bonanza
FTTR002	0.0	1.5	1.5	0.5	16.34	8.48	Fraternal
FTTR003	3.0	5.0	2.0	1.7	14.15	12.95	Fraternal
FTTR005	4.0	12.8	8.8	6.6	2.88	0.25	Fraternal
FTTR018	2.2	6.6	4.4	2.2	2.82	0.62	Fraternal

9.2 Soil Sampling

RRPL collected a total of 1,086 conventional soil samples at Auld Creek to test mineralised structures (Aldrich and Whaanga, 2024). RRPL also conducted ultrafine soil sampling (46 samples in total) in areas of glacial till at Auld Creek (Aldrich and Whaanga, 2024).

As of the effective date of this Report, RUA had collected 1,622 soil samples within EP 60648.

9.3 Rock-Chip Sampling

RRPL collected a total of 102 rock-chip samples across EP 60648.

As of the effective date of this Report, RUA had not collected any further samples in the Project area.

9.4 Geophysics

A LiDAR survey was facilitated by RRPL in 2020 and covered parts of the Project area (McCulloch, 2023a; Reefton Resources Pty Ltd, 2023a, b, c).

9.5 Mapping

Prior to 2019, most geological mapping covering the Project area was conducted on a regional scale, targeting historical mines and possible lode extensions (e.g. Allibone, 2012). Following the acquisition of RRPL, RUA carried out further mapping over the Project area.

9.6 Three-Dimensional Solid Geological Modelling

RRPL generated a 3D geological model of the Auld Creek vein system. The model is based on downhole vein intercepts of Fraternal and Bonanza — the two main structures at Auld Creek. The model intervals are based on geological observations and were refined with assay data (Figure 9-1).

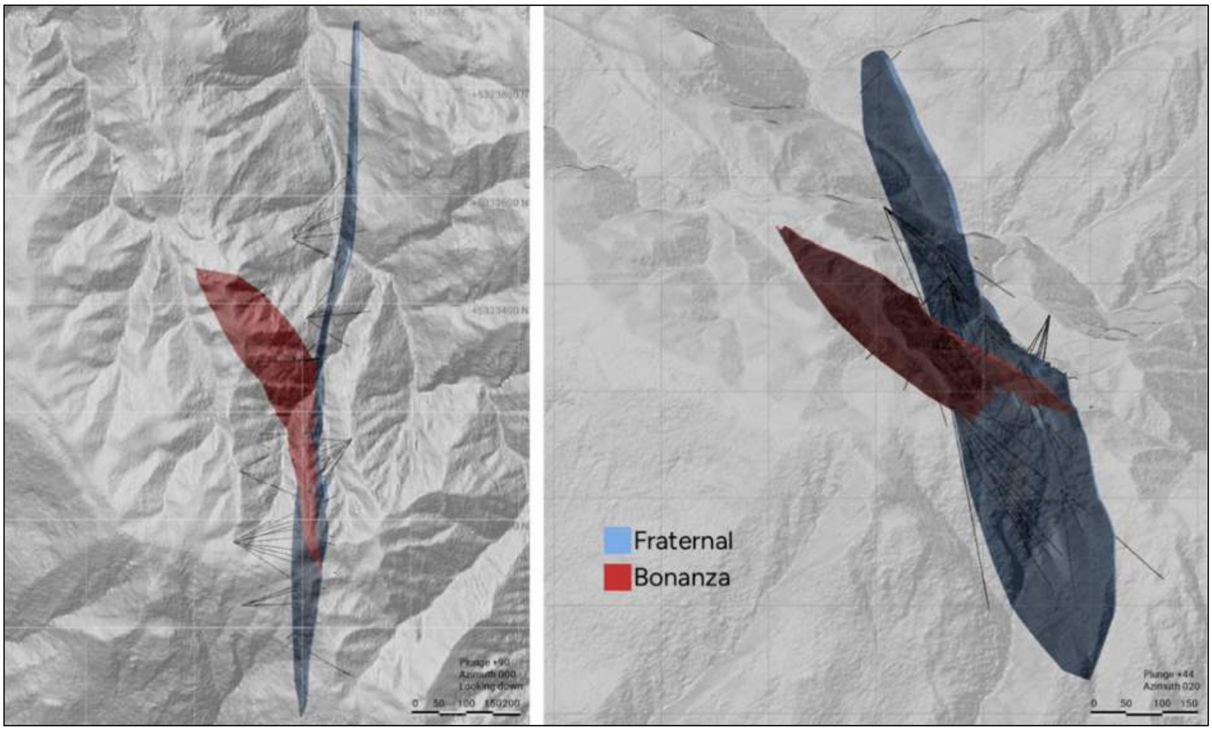


Figure 9-1: Auld Creek 3D model.

9.7 Exploration Target Interpretation

The Reefton Goldfield is a classic orogenic system containing numerous structurally controlled Au deposits, hosted in tightly folded, greenschist-facies Greenland Group sediments. The Au deposits are predominantly clustered along the intersection of north-trending shear and fault zones and areas of intense folding. Quartz veins are typically discordant to bedding and strike parallel to axial surfaces of regional-scale north-plunging folds. Two end-member styles of mineralisation are described.

1. An early generation of relatively undeformed Au-and-arsenopyrite-bearing quartz veins (e.g. Phoenix-Inglewood at Murray Creek).

2. A later, shallower, more brittle event that is characterised by Au and Sb-bearing quartz breccias (e.g. Golden Treasure).

10 Drilling

RRPL completed 21 diamond drillholes in EP 60648 for a total of 2,659 m (Figure 10-1); of which 18 drillholes for 2,304.1 m are within the Project area. Between July 2024 and February 2026, RUA completed a further 13 diamond drillholes in the Project area, for a total of 2,737.3 m. Drilling data from these holes and historical drilling undertaken by previous owners (Section 6.2) were used as a basis for the MRE reported in Section 14.

Significant intercepts from full mineralised zone composites are reported in Table 10-2. The QP (Abraham Whaanga) notes that the true width of mineralisation will be smaller than the downhole width of mineralisation due to the high intersection angles — DOC consent restrictions necessitate numerous drillholes being drilled from one pad. While the QP (Abraham Whaanga) recommends optimising the drill pattern for the reef orientation where possible, it is unlikely that this risk can be mitigated.

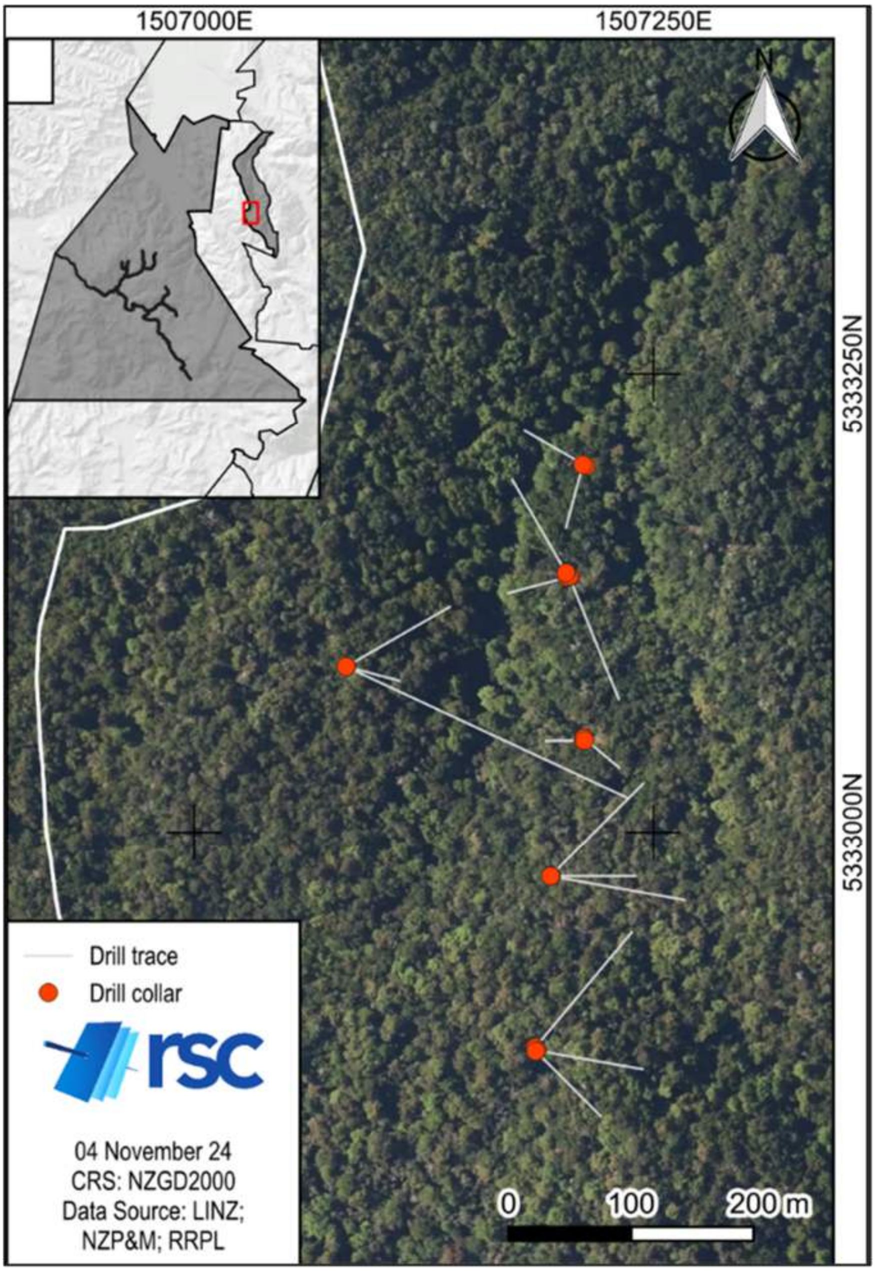


Figure 10-1: RRPL drillhole collar locations at Golden Point.

RRPL conducted drilling in EP 60648 between 2021 and 2024. In total, 21 diamond drillholes were completed, including 18 holes (2,304.1 m) at the Project, targeting the Fraternal and Bonanza East shoots (Figure 10-2). Drilling was conducted by Ecodrilling Ltd, using a CS1000 rig.

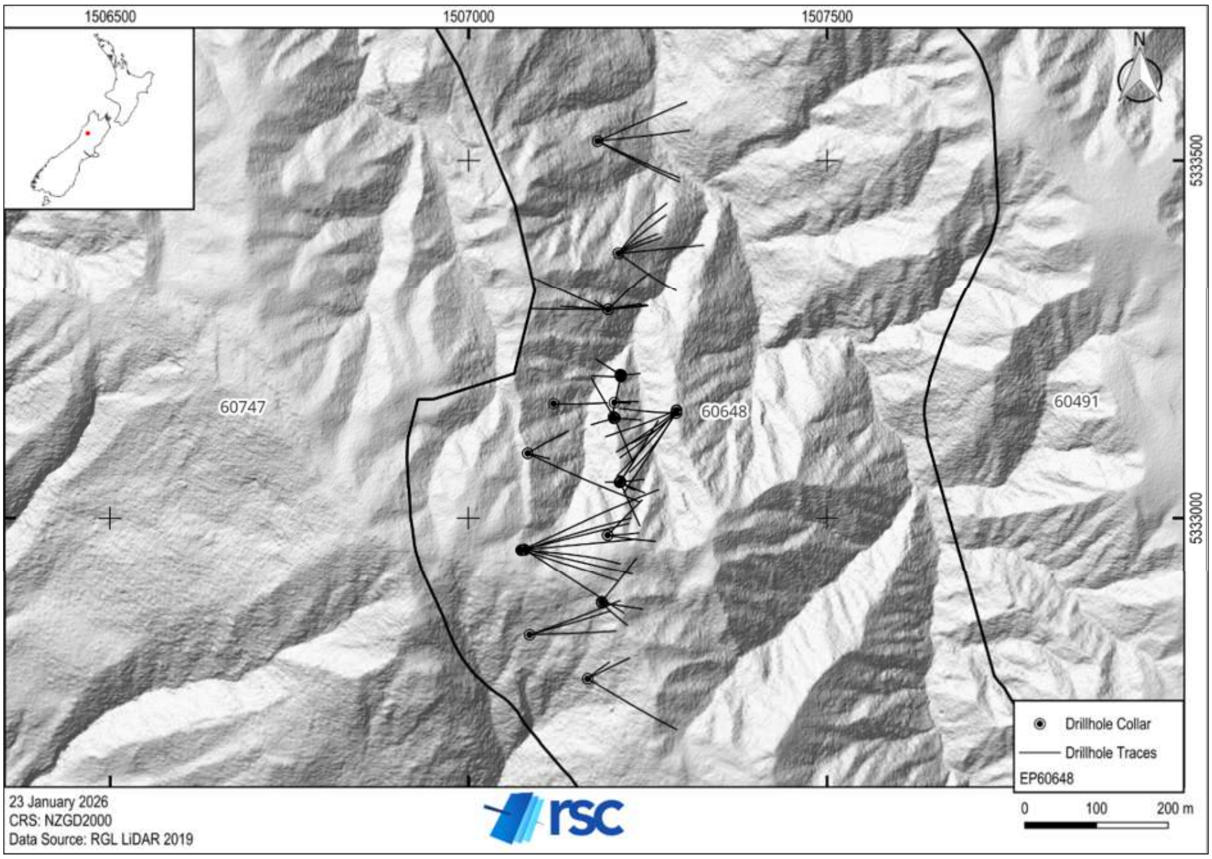


Figure 10-2: Auld Creek drillhole collars and traces.

Drilling at the Project was conducted from six drill pads, with 2–4 holes drilled from each pad. Drillhole dips varied from -50° to -90°, and diamond core diameter decreased from PQ to HQ at depth. A summary of the drill collars is presented in Table 10-1, and the significant intercepts are presented in Table 10-2.

RRPL collected oriented core using REFLEX orientation tools, and downhole surveys were completed using a REFLEX EZ-TRAC or a Precision Gyro, with measurements taken every 5–15 m.

Due to the orientations of parallel north-striking shoots, many drillholes intercepted both Bonanza and Fraternal 1. Drillhole ACDDH015 returned 12.4 m at 5.19 g/t Au and 13.7% Sb from 69.6 m, and 27.4 m at 3.67 g/t Au from 105 m.

Table 10-1: RRPL drillholes.

Drillhole	Easting (NZTM)	Northing (NZTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
ACDDH004	1507194.2	5332976.5	602.4	142.6	45	-60
ACDDH005	1507194.4	5332976.0	602.6	147.4	100	-60
ACDDH006	1507194.1	5332976.3	602.4	177.4	90	-75
ACDDH007	1507185.6	5332882.9	604.3	154.3	40	-58
ACDDH008	1507186.7	5332881.3	604.1	110	100	-58
ACDDH009	1507185.9	5332881.0	604.1	181.5	135	-74
ACDDH010	1507211.8	5333050.0	565.6	40.8	270	-60

Drillhole	Easting (NZTM)	Northing (NZTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
ACDDH011	1507212.1	5333051.6	565.6	161	130	-81
ACDDH012	1507212.5	5333050.2	565.5	39.2	270	-65
ACDDH013	1507203.1	5333139.1	533.6	52	255	-50
ACDDH014	1507204.9	5333139.6	534.5	70.4	255	-90
ACDDH015	1507204.1	5333139.7	534.2	136	158	-58
ACDDH016	1507202.5	5333141.5	533.1	101.9	330	-55
ACDDH018	1507083.0	5333090.3	584.0	262.1	60	-55
ACDDH019	1507082.8	5333090.5	584.0	143.8	115	-50
ACDDH020	1507213.3	5333199.8	584.0	124.3	105	-78
ACDDH021	1507211.6	5333200.4	511.0	146.3	300	-72

Table 10-2: Significant RRPL drilling intercepts, full mineralised zone composites (1.5 g/t Au cut-off).

Hole ID	From (m)	To (m)	Downhole Interval (m)	True Width Interval (m)	Au (g/t)	Sb (%)	Mineralised Zone
ACDDH004	51.7	57.9	6.2	1.2	1.82	0.01	Bonanza
ACDDH004	116.2	136.8	20.7	7.8	4.91	2.18	Fraternal
ACDDH005	65.8	80.4	14.6	7.8	2.12	0.10	Fraternal
ACDDH007	123.2	148.5	25.2	8.0	3.34	0.08	Fraternal
ACDDH011	75.3	83.4	8.1	2.2	2.19	3.00	Bonanza
ACDDH015	69.6	82.0	12.4	3.6	5.01	13.88	Bonanza
ACDDH015	105.0	132.4	27.4	4.4	3.56	0.18	Fraternal
ACDDH016	65.0	90.0	25.0	7.8	6.45	0.25	Bonanza
ACDDH024	105.0	115.0	10.0	5.6	2.24	1.24	Fraternal
ACDDH025	120.0	124.0	4.0	3.4	3.24	4.38	Bonanza
ACDDH027	148.0	168.0	20.0	7.1	3.03	0.07	Fraternal
ACDDH028	209.5	212.0	2.5	2.1	6.17	3.29	Fraternal
ACDDH031	310.5	312.5	2.0	1.2	5.57	13.67	Fraternal

Following the acquisition of RRPL, RUA commenced drilling at the Project in 2024. RUA has completed a further 38 drillholes, for a total of 8,086.9 m, targeting the Fraternal and Bonanza shoots (Table 10-3). The drilling was carried out by Ecodrilling Ltd, using a CS1000 rig and an LF70 rig.

The drilling was conducted from eight pads, with 3–9 holes drilled from each pad. Drillhole dips varied from -37° to -83°, and diamond core widths decreased from PQ to HQ at depth. A summary of the drill collars is presented in Table 10-3.

Table 10-3: RUA drillholes.

Drillhole	Easting (NZTM)	Northing (NZTM)	RL (m)	Length (m)	Azimuth (°)	Dip (°)
ACDDH022	1507212	5333199	511	108.5	193	-54
ACDDH023	1507212	5333199	511	51.5	85	-60
ACDDH024	1507290	5333146	539.2	156.3	220	-37
ACDDH025	1507290	5333146	539.2	180.9	248	-54
ACDDH026	1507290	5333146	539.2	200	231	-59
ACDDH027	1507290	5333146	539.2	193.4	212	-45
ACDDH028	1507079	5332952	500	243.5	104	-50

Drillhole	Easting (NZTM)	Northing (NZTM)	RL (m)	Length (m)	Azimuth (°)	Dip (°)
ACDDH029	1507079	5332952	500	256	120	-50
ACDDH030	1507079	5332952	500	268.5	85	-53
ACDDH031	1507079	5332952	500	336	74	-65
ACDDH032	1507079	5332952	500	351.6	108	-70
ACDDH033	1507079	5332952	500	291.1	92	-64
ACDDH034	1507212	5333199	511	100	267	-72
ACDDH035	1507209	5333371	511.61	194.3	85	-53
ACDDH036	1507209	5333371	511.61	200.8	125	-60
ACDDH037	1507209	5333371	511.61	189.3	49	-62
ACDDH038	1507209	5333371	511.61	154.5	72	-67
ACDDH039	1507209	5333371	511.61	188	60	-75
ACDDH040	1507209	5333371	511.61	238.7	39	-67
ACDDH041	1507195	5333291	512.26	154	270	-45
ACDDH042	1507166	5332776	656	206.1	120	-45
ACDDH043	1507195	5333291	512.26	207.3	273	-72
ACDDH044	1507195	5333291	512.26	200	294	-54
ACDDH045	1507166	5332776	656	174.2	64	-68
ACDDH046	1507195	5333291	512.26	215.2	295	-83
ACDDH047	1507166	5332776	656	252	51	-81
ACDDH048	1507085	5332837	574	251.7	88	-62
ACDDH049	1507195	5333291	512	122.5	85	-63
ACDDH050	1507195	5333291	512	172	84	-73
ACDDH051	1507085	5332837	574	308.8	71	-66
ACDDH052	1507195	5333291	512	209.3	45	-70
ACDDH053	1507085	5332837	574	325.4	75	-71
ACDDH054	1507195	5333291	512	240	81	-79
ACDDH055	1507180	5333527	450.8	175.9	115	-45
ACDDH056	1507074.23	5332955.32	606.24	276.5	65	-43
ACDDH057	1507180	5333527	450.8	240	118	-59
ACDDH058	1507180	5333527	450.8	209.8	84	-52
ACDDH060	1507180	5333527	450.8	243.3	66	-59

The QP (Abraham Whaanga) is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results underpinning the MRE in section 14.

11 Sample Preparation, Analyses and Security

For the sake of clarity and transparency, relevant aspects of the RRPL, RUA, and historical sampling and analyses that were used to underpin the current resource estimate (presented in Section 14) are summarised in this section (Section 11).

11.1 Sample Preparation

Sample preparation was undertaken at SGS laboratories in Westport and Waihi, and at LabWest Minerals Analysis Pty Ltd (LabWest) in Perth, Western Australia. SGS and LabWest maintain ISO/IEC 17025 accreditation and are independent from RRPL and RUA.

11.1.1 Soil Samples

RRPL carried out conventional and ultrafine soil sampling across EP 60648 (Figure 11-1). For the conventional soil sampling, soil sample points were loaded onto a handheld GPS for guidance, and actual locations were marked and recorded using GPS in the field. Soil augers or spades were used to acquire a ~300-g sample, which was put in a wet-strength paper sample bag with wire ties. Samples were typically collected from the B or C horizons, although sample depths varied. Samples were logged on MS Excel spreadsheets in the field, including sample ID, depth, colour, horizon, slope, sample description, sampler, basement, and comments. Samples were sent to SGS Westport for sample preparation, after which the pulp samples were sent to SGS Waihi for Au analysis. At SGS Westport, the samples were dried for 24 hours and crushed to pass 2 mm. The crushed sample material was split using a rotary splitter before a sub-sample was pulverised to 75 µm.

For ultrafine sampling, soil augers were used to acquire 200–300 g clay samples from the first clay layer intercepted and put in wet-strength sample bags. Samples were sent to LabWest for preparation, where 40 g of soil was settled in water with a dispersant, and a minimum of 0.2 g of the 2 µm fraction was collected for assay.

For RUA's soil sampling, a bulk sample of ~0.5–1 kg was collected in the field and taken back to RUA's office for preparation. Samples were dried in a customised incubator, set at 38°C, for a minimum of 2 days. Once the samples were fully dried, they were sieved to <180 µm in size. A sub-sample of 50–100 g was scooped from the <180 µm size fraction for analysis. The remaining material was retained and stored in the RRPL core shed in Reefton.

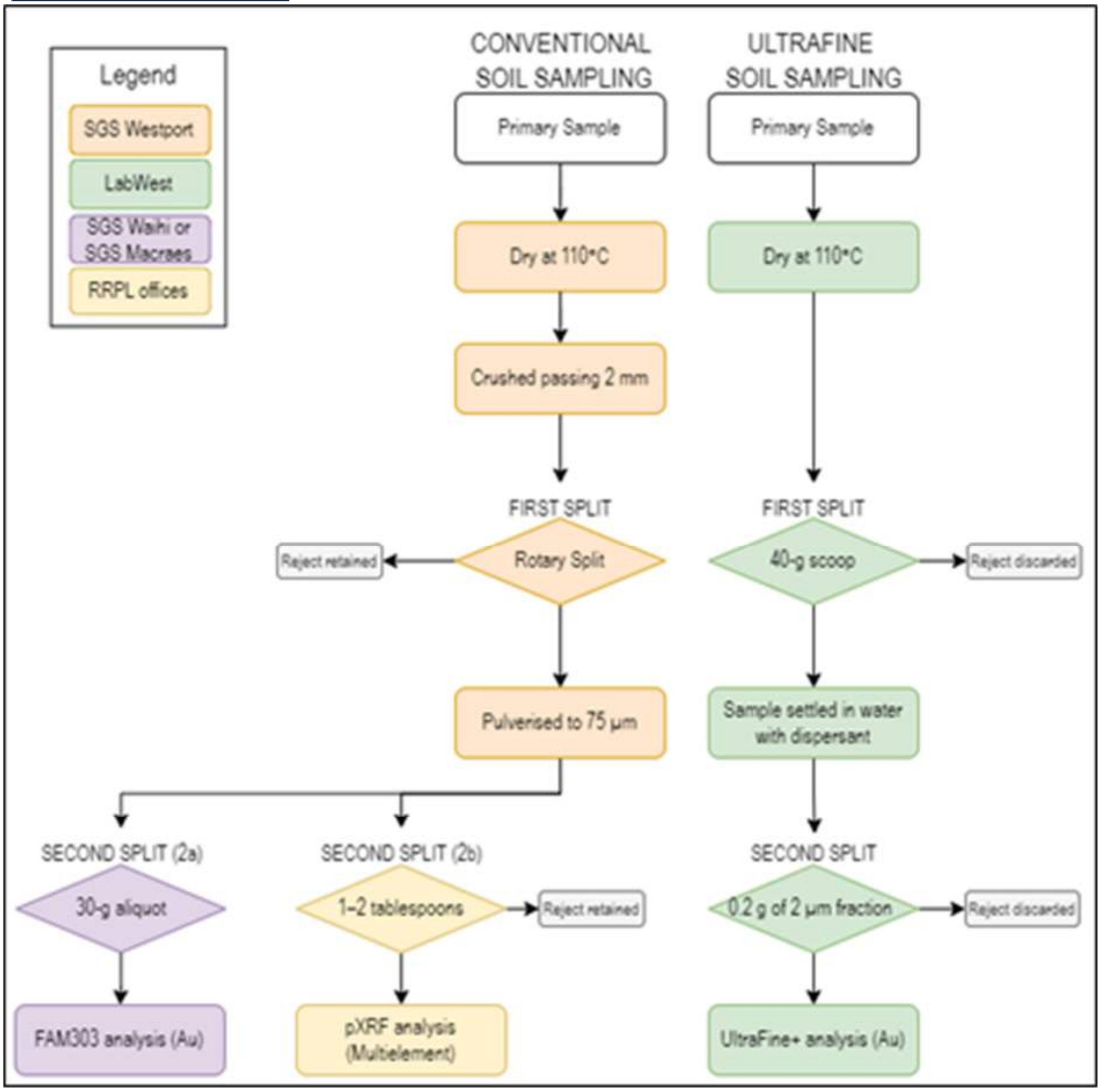


Figure 11-1: Flowchart outlining RRPL's soil sampling process.

11.1.2 Rock-Chip Samples

RRPL collected ~0.5–1 kg samples, which were described in the field before being analysed by pXRF for a first-pass indication of elements of interest (e.g. Au, As, Sb, Cu, Pb, Zn). Relevant samples were sent to SGS Westport for preparation with Au certified reference materials (CRMs), then to SGS Waihi for analysis. At SGS Westport, the samples were crushed to 2 mm and then pulverised to 75 µm. Samples larger than 1 kg were split using a rotary splitter prior to being pulverised.

Rock-chip samples collected by RUA were sent to SGS Westport for sample preparation. Samples were crushed and pulverised to 85% passing 75 µm. The pulverised rock-chip samples were split into two samples: an ~50-g sample sent for laboratory analysis, and the reject returned to RUA for pXRF analysis and storage.

11.1.3 Core Samples

RRPL employed selective sampling according to industry standards, typically 4–5 m on either side of an area of interest or known zone of mineralisation. Diamond core was used to obtain samples for geological logging and sampling. Core samples were photographed and cut in half lengthways using a core saw in intervals of 1 m, unless determined otherwise by lithology (e.g. quartz vein contacts).

Maximum and minimum sample lengths of 1.2 m and 0.15 m, respectively, were collected. All half-cut core samples were placed in poly-weave sacks and delivered to SGS Westport for preparation, before being shipped to SGS Waihi for Au assay.

RRPL core samples were dried and then crushed to a nominal 90% passing 2 mm. If required, samples were split using a rotary splitter to 250 g, with the coarse residue retained and the remaining split pulverised to 90% passing 75 µm in a vertical spindle pulveriser. A 1-kg barren sand flush was pulverised, and compressed air was used to clean all crushers and grinders after every sample with visible Au.

RUA typically sampled drillholes in full, following 1-m sample intervals unless geological contacts (i.e. dolerite intrusions) dictated otherwise. PQ and HQ core were sampled as half-core.

RUA drill core samples were sent to SGS Westport for sample preparation. Core was crushed to 75% passing 2 mm, and a 1-kg split of material was pulverised (to 85% passing 75 µm). No split duplicates were collected during the crushing steps. Two scoops were taken from the pulveriser bowl: one for laboratory analysis (~150 g) and the other for pXRF analysis (~100 g); the pulp reject was stored at the RRPL site in Reefton.

11.1.4 Trench Samples

All trench sampling was conducted by RRPL.

Trenches were located with a Garmin 66i GPS, using the waypoint averaging function for 30 minutes, and positions were checked against 1-m LiDAR contour maps. Trenches were treated as drillholes, with collar, survey, lithology, and assay data compiled into a validated database. Chip sampling of trenches was completed using a hammer and chisel, with an average sample size of ~2 kg, and field duplicate samples were taken from visible mineralisation at a rate of one per trench. Sample intervals were typically 1 m, but intervals of 0.2–1.2 m were collected to allow for geological contacts. Trench samples were stored in calico bags and sent to SGS Westport for preparation, which comprised drying, crushing, splitting (if required), and pulverising to obtain an analytical sample of 250 g, with >95% passing 75 µm.

11.2 Analysis

Analysis was completed by SGS Waihi, LabWest, and ALS Geochemistry (ALS), Brisbane. SGS, LabWest, and ALS maintain ISO/IEC 17025 accreditation and are independent from RRPL and RUA.

11.2.1 Laboratory Analysis: Soil Samples

Analysis of conventional soil samples collected by RRPL was undertaken at SGS Waihi by 30-g fire assay with an inductively coupled plasma mass spectrometry (ICP-MS) finish (SGS method FAM303; Table 11-1), with a detection range of 1–2,000 ppb.

Ultrafine analysis was undertaken by LabWest (LabWest Ultrafine+ method; Table 11-1) and involved analysis of the reactive 2-µm clay fraction by microwave digestion and ICP-MS for Au and 48 other elements.

For the RUA soil samples, a 50–100 g fine-sieved (<180 µm) soil sample was sent to ALS for Au-TL43 low-level Au analysis. The analysis consisted of 25-g sample digestion by aqua regia, followed by trace Au analysis by ICP-MS. The detection limit for Au by this method is 1 ppb.

11.2.2 Laboratory Analysis: Rock-Chip Samples

RRPL rock-chip samples were sent to SGS Waihi for Au analysis by 30-g fire assay with AAS finish (SGS method FAA303; Table 11-1), or screen fire assay (SGS method FAS30K) if visual Au was present. Samples were fused with a Pb oxide flux at 1,000°C, and a Pb button containing Au and Ag was recovered. The button was then cupelled in a magnesia cupel and the doré prill recovered, which was transferred to a Pyrex test tube and digested in HNO3 to dissolve the Ag. Hydrochloric acid was added to generate aqua regia, which dissolved the Au. The resultant solution was diluted with demineralised water and mixed thoroughly. After the AgCl had precipitated and the solution was free of sediment, it was read for Au on an AAS instrument against Au standard calibration solutions prepared from 99.9999% pure Au metal. Data were reported with an accuracy of ±15% to reflect the sample preparation component and particulate Au in the assay process.

RUA's pulverised rock-chip samples were analysed by 50 g fire assay with AAS finish at SGS Waihi (SGS Code FAA505). The detection limit for Au by this method is 0.01 ppm.

11.2.3 Laboratory Analysis: Core Samples

RRPL's diamond drill core samples were sent to SGS Waihi for Au analysis. Samples were analysed for Au by 30 g fire assay with AAS finish (SGS method FAA303; Table 11-1), and screen fire assays (SGS method FAS30K) were used if there was visible Au in the core. The detection limit for FAA303 and FAS30K was 0.01 g/t Au.

RUA's pulverised drill core samples were analysed by 50 g fire assay with AAS finish at SGS Waihi (SGS Code FAA505). The detection limit for Au by this method is 0.01 ppm. As part of SGS' internal quality control, SGS conduct repeat analyses, also at a rate of ~5%.

11.2.4 Laboratory Analysis: Trench Samples

Trench samples were sent to SGS Waihi for Au analysis. Samples were analysed for Au by 30 g fire assay with AAS finish (SGS method FAA303; Table 11-1), and screen fire assays were used if visible Au was noted by the RRPL geologist.

Trench samples were analysed using the same CRMs as those used in core sampling (Table 11-3). Blanks were inserted at the start of the sample chain for each trench, and field duplicate samples were taken at a ratio of one per trench over visually mineralised intervals. Samples were submitted with a CRM.

11.2.5 Portable X-Ray Fluorescence

All pXRF analyses were conducted by RRPL in house. Multi-element analysis was undertaken by pXRF on the core, soil, trench, and rock-chip sample pulps returned to RRPL after Au analysis. An RRPL SOP (*RRL_pXRF_SOP*) was used for the operation and analysis of samples, and detailed the steps involved in collecting multi-element data using the following method.

A plastic teaspoon was used to scoop material from the sample pulp into a sample cup with 4-μm polypropylene film. Sample cups were filled to approximately three-quarters full (~1–2 tablespoons) and tapped to form a smooth surface against the film. Sample cups were cleaned between samples by wiping with alcohol wipes or alcohol (e.g. ethanol) on soft tissues.

Pulp samples were analysed in batches, as per laboratory submissions, using an Olympus M-series Vanta pXRF instrument with a 4-W, 50-kV rhodium anode in Geochem3-AuTe mode, with 20 s for each of the three beams. In total, 42 elements were analysed, and the analytical run was initiated with three CRMs and one blank. A CRM was then analysed every 20 pulp samples, and a blank was analysed every 50 samples. Analyses were repeated at a rate of 1 in every 20 samples. The instrument was operated using a field test stand and a laptop with Vanta PC software. The approach followed industry best practice, as outlined in Fisher et al. (2014) and Gazley and Fisher (2014).

The pXRF data, including QC analyses (e.g. blanks, CRMs, replicates, and repeats), were exported from the instrument and compiled into a master MS Excel workbook.

Core was analysed by pXRF during logging for a multi-element suite, including As and Sb at 1-m intervals, with more detailed analysis around zones of interest. The SOP (*RRPL_pXRF_SOP)* stated that the core should be analysed for 10–15 s, using the trigger to manually start and stop the analysis. The QP (Abraham Whaanga) notes that this is much shorter than the full test time of 60 s when using three-beam mode and stopping the analysis after 15 s would mean the second and third beams were not engaged. The QP (Abraham Whaanga) does not consider this best practice, and the data will be indicative only. Results for each interval were recorded both on the core and in the geology log, and these data were used to select the intervals for laboratory analysis. The QP (Abraham Whaanga) notes that Sb is difficult to analyse using a pXRF instrument, and a 50-kV beam mode is required.

The pXRF analysis of pulps and drill core was completed in-house by trained RRPL and RUA staff.

Table 11-1: Summary of laboratory method codes for assay and geochemical analyses.

Analysis Type	Operator	Sample Type	Laboratory	Method	Description
Low-Level Au	RUA	Soil	ALS Brisbane	Au-TL43	Low-level aqua regia digest
Fire Assay	RRPL	Soil	SGS Waihi	FAM303	30 g charge, ICP-MS
High-Level Au	RUA	Soil	ALS Brisbane	Au-AROR43	Aqua regia digest
Ultrafine	RRPL	Soil	LabWest	Ultrafine+	Microwave digestion, ICP-MS, OES
pXRF	RRPL	Soil, rock-chip, core, trench	RRPL/RUA	-	pXRF of <2 mm pulp samples
Fire Assay	RUA	Rock-chip, core	SGS Waihi	FAA505	50 g charge FA, AAS finish
Fire Assay	RRPL	Core, trench, rock-chip	SGS Waihi	FAA303	30 g charge FA, AAS finish
Screen Fire Assay	RRPL, RUA	Core	SGS Waihi	FAS30K	30 g charge, 75 µm, lead collection

11.3 Density & Moisture Content

Density assessments were conducted by RRPL based on 106 drill core samples (median core length 0.12 m) from EP 60648. The QP (Abraham Whaanga) notes that the RRPL SOP (*RRL SG SOP*) confuses specific gravity (SG) and in situ bulk dry density, but the procedure outlined is otherwise consistent with the common-practice water displacement method described by Lipton and Horton (2014).

RUA conducted an additional 204 bulk density measurements over the Project area (Table 11-2).

Table 11-2: Sample information for RUA bulk density measurements.

Shoot	Number of Samples	Median Core Length (m)
Bonanza East	5	0.19
Fraternal	115	0.17
Fraternal North	84	0.17

Bulk density was calculated automatically using the *Density Master* worksheet when water temperature, dry weight, and wet weight data were input. The bulk density calculation was completed in two steps.

1. The weight of the water volume displaced was divided by the water density to account for the temperature difference.
2. The mass (dry weight) was divided by the volume of water displaced (wet weight, corrected for temperature).

Measurements were collected from competent diamond drill core. Geological domains were determined according to different lithologies and degrees of weathering, and samples were collected from the same domain in different drillholes at varying depths, and from both unmineralised host rock and mineralised zones. The Density Master spreadsheet indicates that sample lengths were typically 10–25 cm and could be cut to fit within this range; however, the QP (Abraham Whaanga) notes that the Density Master spreadsheet contains some outliers that may be transcription errors. These outliers have a negligible impact on the overall bulk density data, which are otherwise typically consistent; however, the QP (Abraham Whaanga) recommends RUA checks the data for any errors and moves towards the use of an appropriate database instead of MS Excel spreadsheets to help limit such errors in the future. The QP (Abraham Whaanga) also notes that there is likely to be some potential for selection bias towards more competent pieces of core, therefore marginally overcalling the true in situ bulk dry density values.

11.4 Security

All samples collected for laboratory analysis were securely packaged on site and transported to SGS Westport by RRPL (prior to October 2024) and RUA (after October 2024) staff for sample preparation. All samples were stored in a locked core shed until dispatch. Sample sheets for submission to SGS were in both paper and digital form. While the QP (Abraham Whaanga) did not observe the dispatch of samples during the site visits, the samples were observed to be held in a secure core shed.

Drilling programmes were typically helicopter supported. Core was flown out as required to a staging area or directly to a core handling area. The QP (Abraham Whaanga) recommends RUA develops an SOP covering sample transport and chain-of-custody details to capture this process once drilling details and logistics have been confirmed.

11.5 Data Quality

11.5.1 Data Quality Objective

Every data collection process implicitly comes with expectations for the accuracy and precision of the data being collected. Data quality can only be discussed in the context of the objective for which the data are being collected. In the minerals industry, the term ‘fit for purpose’ is typically used to convey the principle that data should suit the objective. In the context of data quality objectives (DQOs), fit for purpose could be translated as ‘meeting the DQO’. For the Auld Creek Gold-Antimony Project, data should be fit for the purpose of classifying at least Indicated Mineral Resources. The data used for the MRE were obtained mainly from diamond drillhole samples. A discussion on QA/QC for trench data is also presented for the sample collection and preparation steps used for trench sampling across the Reefton Project rather than specifically for the Auld Creek Gold-Antimony Project. This approach is considered appropriate by the QP (Abraham Whaanga), as trench sampling was primarily undertaken to support early-stage exploration and target generation, rather than detailed resource evaluation.

11.5.2 Quality Assurance

Quality assurance (QA) is about error prevention and establishing processes that are repeatable and self-checking. The simpler the process and the fewer steps required the better, as this reduces the potential for errors to be introduced into the sampling process. This goal can be achieved using technically sound, simple, and prescriptive SOPs and management systems.

In discussing the suitability of QA systems for the data collection that might underpin a future MRE, and the potential impact of these processes on the resource classification, the QP (Abraham Whaanga) determined whether:

1. processes are clearly documented in an SOP and represent good practice;
2. the SOP includes clearly defined data quality objectives;
3. the SOP includes clear details on quality control (QC) measures; and
4. the site visit confirmed adherence to the SOPs.

For each part of the sampling, preparation, and analytical process, a comment on the expected associated risk with respect to resource classification is provided.

11.5.2.1 Diamond Drilling Samples

11.5.2.1.1 Collar Location

An SOP covering the collection of collar location data was not available for review; however, the QP (Abraham Whaanga) was able to review the survey data. The RRPL, RUA, and OGL drillhole collars were surveyed by a professional surveyor using a Trimble real-time kinematic (RTK) GNSS with R10 rover and base units. The measured accuracy of these surveys was between position 0.05 m, and height 0.10 m when RTK lock was achieved, and position 0.5 m, and height 1–2 m when RTK lock was not possible. OGL drillholes were surveyed using a mixture of GPS and the process described above. The QP (Abraham Whaanga) notes that the GPS was prone to large errors (~5 m) and recommends resurveying (using differential (D) GPS) all collars surveyed by GPS.

The collar location collection process was not audited by the QP (Abraham Whaanga). However, the QP (Abraham Whaanga) considers that collar location pick-up processes pose limited risk for the intended resource classifications.

11.5.2.1.2 Downhole Orientation Survey

Downhole surveys on RUA and RRPL drillholes were conducted using a REFLEX EZ-TRAC downhole instrument or a Precision Gyro. OGL drillholes were surveyed using a single-shot digital camera, or REFLEX EZ-TRAC, every 30 m. The drilling contractors for RRPL surveyed the drillholes at 15-m intervals, consistent with the manufacturer's instructions for operating the survey tool. The drilling contractors for RUA surveyed the drillholes at intervals that vary from 3–30 m depending on lithology and whether the drilling intersected a mineralised zone, consistent with the manufacturer’s instructions. Downhole surveys were conducted by the drilling contractors; however, this process should ideally have been monitored by the rig geologist. The QP (Abraham Whaanga) reviewed RRPL’s instrument output files, which indicated that quality checks were in place, including pass/fail checks.

RRPL did not have an SOP covering the downhole survey procedures. RUA did have an SOP in place, which was of a good standard and consistent with industry best practice; however, it did not include any information on data quality objectives. The downhole orientation survey process was not audited by the QP (Abraham Whaanga); however, based on discussions between RRPL/RUA geologists and the QP (Abraham Whaanga), the QP (Abraham Whaanga) considers that there is low risk with respect to the DQO, and this has been considered when classifying the mineral resources.

11.5.2.1.3 Bulk Density

An SOP detailing the measurement of bulk density was available for the QP (Abraham Whaanga) to review. The process was not audited by the QP (Abraham Whaanga). The SOP describes a process that is consistent with the water displacement method described by Lipton and Horton (2014), which is industry standard, although the SOP does not provide tolerance thresholds for density measurements or details on quality control processes, e.g. the collection of repeat or duplicate measurements. The QP (Abraham Whaanga) is of the opinion that the process poses a low risk with respect to the DQO but recommends revising the SOP to provide tolerance thresholds and to detail QC procedures.

11.5.2.1.4 Primary Sample

An SOP detailing the drilling of diamond core by RRPL was available for the QP (Abraham Whaanga) to review. The SOP briefly covered aspects of logistics, preparation, safety in relation to the drilling campaign, downhole surveying, and core recovery. However, the SOP did not note the minimum recovery threshold required or provide guidance for dealing with low recoveries.

Core was sampled along 1-m intervals, except in zones of distinct mineralisation (e.g. quartz veins or sulphide enrichment), where the sample interval was adjusted for lithological breaks; pXRF was used to help determine mineralised zones. The SOP defined minimum and maximum sample intervals of 0.15 m and 1.2 m, respectively; however, the drillhole database indicated intervals as long as 1.8 m had been sampled.

The SOP followed by RUA also indicated that core should be sampled at 1-m intervals, except in zones of distinct mineralisation. The SOP defined minimum and maximum sample intervals of 0.3 m and 1.0 m, respectively; however, entries in the drillhole database indicated that intervals as long as 1.6 m had been sampled.

The QP (Abraham Whaanga) visited a drill rig at Auld Creek during the January 2025 site visit, and discussed site set-up, drilling conditions, and rig safety with the drill contractor. During the January 2026 site visit, the QP (Abraham Whaanga) checked core boxes while undertaking verification sampling and compared recoveries in the database with core photographs.

Based on the SOP and observations made during the January 2026 site visit, the QP (Abraham Whaanga) considers the core recovery process poses a low risk with respect to the DQO.

11.5.2.1.5 First Split

All PQ and HQ core was half-core sampled by RRPL and RUA. No NQ core was sampled for the drillholes of interest from RRPL or RUA. The QP (Abraham Whaanga) recommends updating the SOP to include different procedures for core with different diameters.

The marking, selecting, and cutting procedures were not audited by the QP (Abraham Whaanga) during the site visits. However, the QP (Abraham Whaanga) reviewed the remaining core, sample marks, and sampling documentation during the site visit. The RRPL SOP states core should be cut perpendicular to features of interest (e.g. shearing, faulting, significant veins, and stockworks), and in the absence of these features, core should be cut perpendicular to the rock fabric. The remaining half-core was retained in the core tray for future reference and check analyses. However, the QP (Abraham Whaanga) recommends marking and cutting core along the orientation line (or a few degrees off it to preserve the line) and consistently sampling the same half of the core, in line with industry best practice, to prevent introducing sampling bias. In core drilling campaigns where core orientation is not carried out, or where it is difficult to align core in broken zones, cut lines may be biased to preserve visible Au in the core, potentially leading to biased sampling.

Based on the SOP and observations made by the QP (Abraham Whaanga) during the site visits, the QP (Abraham Whaanga) considers the first-split process poses a minor risk with respect to the DQO. The QP (Abraham Whaanga) recommends making changes to the core cutting procedures at the Project to minimise the risk of introducing selection bias and recommends updating the SOP to document the process for determining the location of the cut line through mineralised or broken intervals.

11.5.2.1.6 Second Split

Crushing of the sample and the second split occurred at SGS Westport. The crushing parameters were set to ~90% (RRPL samples) or 75% (RUA samples) passing 2 mm, consistent with the standard passing for this step for the respective analytical methods used by RRPL and RUA. An ~1-kg split was collected by SGS Westport. An SOP for the second-split process was not available for the QP (Abraham Whaanga) to review; however, SGS Westport is an ISO/IEC 17025-accredited laboratory, and although there is some residual risk with this part of the process not having been audited, the QP (Abraham Whaanga) is conversant with SGS laboratories and their SOPs around the world, and considers the risk associated with the second-split procedures to be low.

11.5.2.1.7 Third Split

Following crushing and splitting, SGS Westport pulverised the samples to 85% passing 75 µm before taking a 30-g (RRPL) or 50-g (RUA) split for analysis. An SOP for this third-split process was not available for the QP (Abraham Whaanga) to review, and the third split was not audited by the QP (Abraham Whaanga); however, the QP (Abraham Whaanga) is familiar with SGS's SOPs and considers the risk associated with the third-split procedure to be low.

11.5.2.1.8 Analytical Process

Pulverised diamond drill core samples were analysed for Au at SGS Waihi (RRPL and RUA) by fire assay (methods FAA303 and FAA505 for RRPL and RUA samples, respectively) and screen fire assay (FAS30K) for core with visible Au. No SOP for the analytical process was available for the QP (Abraham Whaanga) to review, and the process was not audited. However, SGS Waihi is an ISO 17025 accredited laboratory, and although there is some residual risk with the process not being audited, the QP (Abraham Whaanga) is familiar with SGS laboratories and its procedures and considers the risk associated with Au analysis to be low with respect to the DQO.

A selection of samples from the Project was sent to ALS Brisbane for Sb analysis. No SOPs outlining the multi-element or Sb analytical processes were available for the QP (Abraham Whaanga) to review from ALS Brisbane; however, the lab is ISO 17025 accredited, and the QP (Abraham Whaanga) is conversant with the procedures followed by ALS laboratories. Therefore, the QP (Abraham Whaanga) considers the risk associated with the analytical processes to be low.

Multi-element analysis of returned core pulps was completed by RRPL geologists, using an Olympus M-series Vanta pXRF and following RRPL's SOP for pXRF analysis. The SOP provides prescriptive steps on how to analyse a sample using the pXRF and includes screenshots of the device software. The QP (Abraham Whaanga) recommends also adding photographs to demonstrate the pXRF set-up with test stand, laptop, and sample cups.

The RRPL pXRF SOP outlines a robust process of collecting QC data (e.g. analysing blanks, CRMs, and repeat samples as well as collecting a replicate measurement) but does not outline what to do with the QC data once collected (e.g. use the CRM data to calibrate the pXRF data). The pXRF analytical process was not audited by the QP (Abraham Whaanga). The QP (Abraham Whaanga) considers that the risk associated with the pXRF analytical process is low; however, the risk associated with data handling and processing is moderate to high, as there is no written procedure detailing how to correct or calibrate the pXRF data.

As of the effective date of this Report, RUA had not carried out any additional pXRF analysis on the samples of interest.

11.5.2.2 Trench Samples

11.5.2.2.1 Trench Location

An RRPL SOP outlining the process to determine the location of the trench samples was not available for review. Following discussions with RRPL geologists, trench locations were picked out by compass and tape measure, after which a handheld GPS or registered surveyor was used to determine the coordinates. The trench location collection process was not audited by the QP (Abraham Whaanga); however, the QP (Abraham Whaanga) considers that there is low risk for surveys conducted by a professional surveyor and some risk for surveys collected using GPS. The QP (Abraham Whaanga) recommends the trenches located by GPS be resurveyed by a professional surveyor.

11.5.2.2.2 Primary Sample

The trenches were typically dug by hand (except for a few that used an excavator), and samples were collected using a geological hammer. An SOP outlining this process was not available for review. The

QP (Abraham Whaanga) did not audit the collection of the primary sample. Based on a discussion of the sampling procedures with RRPL geologists, the QP (Abraham Whaanga) considers the collection of the trench primary samples to be low risk with respect to the DQO.

11.5.2.2.3 First Split

An SOP outlining the first split was not available for review. The first split was conducted at SGS Westport, where the sample was crushed to 2 mm then split using a rotary splitter. SGS Westport is an ISO/IEC 17025-accredited laboratory. Although there is some residual risk with this part of the process not having been audited by the QP (Abraham Whaanga), the QP (Abraham Whaanga) is conversant with SGS laboratories and their SOPs around the world and considers the risk associated with the first-split procedures to be low.

11.5.2.2.4 Second Split

The second split was conducted at SGS Westport, where the sample was pulverised, and a scoop was used to collect the aliquot (30 g) for analysis. An SOP for the second split was not available for review. The QP (Abraham Whaanga) did not audit the SGS preparation facilities or the second-split process. The QP (Abraham Whaanga) is familiar with SGS laboratories and their procedures; therefore, the QP (Abraham Whaanga) considers the collection of the second split to be of a low risk with respect to the DQO.

11.5.2.2.5 Analytical Process

Pulverised trench samples were analysed at SGS Waihi for Au by fire assay (FAA303). An SOP outlining the analytical procedures was not available for review; however, the QP (Abraham Whaanga) is conversant with SGS laboratories and their analytical methods. As both laboratories are ISO/IEC 17025 accredited, the QP (Abraham Whaanga) considers the risk associated with the analytical process to be low.

11.5.3 Quality Control

The purpose of QC is to detect and correct errors while a measuring or sample collection system is in operation. The outcome of a good QC programme is that it can be demonstrated that errors were fixed during operation and that the system delivering the data was always in control. Together with good QA (covered in Section 11.5.2), QC ensures that the DQO is met.

Good QC is achieved by inserting and constantly evaluating checks and balances. These checks and balances can be incorporated at every stage of the sample process (location, primary sample collection, preparation, and analysis) and, if in place, should be monitored during data collection, allowing the operator to identify and fix errors as they occur.

11.5.3.1 Diamond Drilling Samples

11.5.3.1.1 Collar Location

QC of the collar location data, as derived from a combination of drillhole collar positions and downhole surveys, should occur on site as surveys are being conducted, by performing check measurements and applying performance thresholds. RRPL and RUA collected multiple hand-held GPS measurements at each collar and validated the collar coordinates against high-resolution LiDAR imagery. The QP (Abraham Whaanga) recommends RUA records the repeat GPS measurements to allow quantitative assessment of the quality of the location data. However, based on the SOP and LiDAR verification, the QP (Abraham Whaanga) is of the opinion that the risk associated with the QC on collar location is low with respect to the DQO.

11.5.3.1.2 Downhole Survey

No quantitative control data (e.g. magnetic field strength, magnetic dip, gravity) were recorded over the course of the drilling programme to monitor the quality of the downhole survey data; however, discussions between the QP (Abraham Whaanga) and RUA indicated that the quality of the downhole surveys was monitored via communication between the drilling contractors and geologists. The Precision Gyro survey device seeks out true north with no risk of magnetic interference and has internal QC procedures that flag surveys as failed if certain parameters exceed predetermined limits. The downhole surveys and the associated QC aspects were managed by the drillers. Because the downhole equipment software used with the Precision Gyro and REFLEX EZ-TRAC is easy to operate, and because it auto-validates the survey data, the QP (Abraham Whaanga) considers the downhole survey process to have been in control throughout the programme.

11.5.3.1.3 Bulk Density

The QP (Abraham Whaanga) reviewed the Density Master spreadsheet, which indicates that water temperature was measured for each sample. Before each batch, a piece of reference HQ core of a known weight was measured to check the scales were performing well. These data were not recorded by RRPL; therefore, the QP (Abraham Whaanga) is unable to determine whether the density data collection was in control prior to December 2024.

RUA collected a further 204 bulk density measurements from core from EP 60648 between December 2024 and February 2026. Calibration checks of the scales were performed at the start of each measurement series and recorded in the Density Master spreadsheet. The QP (Abraham Whaanga) identified a small number of outliers in the data with respect to sample lengths, which were inconsistent with the weights entered, but the bulk density values were typically consistent, and the QP (Abraham Whaanga) is of the opinion that these may be transcription errors. The QP (Abraham Whaanga) recommends checking the data for any erroneous values and improving the digital record-keeping of this process by using an appropriate database rather than an MS Excel workbook. RUA geologists reviewed the density measurements as they were collected; however, because no tolerance thresholds were stipulated, this process could not be verified. The QP (Abraham Whaanga) notes that by providing tolerance thresholds to geologists/scale operators, QC on scale performance can be exercised as the measurement process is in operation, which is the goal of good QC.

No duplicate or repeat density measurements were collected; therefore, RSC is unable to confirm that the process was always in control. The QP (Abraham Whaanga) considers this acceptable for the classification of low to moderate-confidence mineral resources, but the QP (Abraham Whaanga) recommends collecting repeat/duplicate data going forward to support the classification of higher-confidence mineral resources.

11.5.3.1.4 Primary Sample

The primary sample is collected at the drill bit. The quality and consistency of the primary sample for diamond drilling is monitored, by proxy, by assessing core recovery.

The drillers used drill blocks to record drill recovery, and these were checked by RRPL and RUA geologists during core mark-up. When poor core recovery was identified by the geologist, the geologist would alert the drillers. The QP (Abraham Whaanga) recommends documenting this process more clearly in an SOP, including guidelines specifying what is acceptable core recovery.

Recovery at the Project was not always in control, with step-drops and out-of-threshold recoveries demonstrated throughout the different parts of the drilling campaign (Figure 11-2). The decrease in sample recovery (moving average) corresponds to poor recoveries linked to certain drillholes (ACDDH010, ACDDH013, and ACDDH016). In response to the poor drill core recovery, especially within the mineralised zone, RRPL re-drilled ACDDH010 on a slightly different angle using different techniques (e.g. different mud mix). The re-drilled hole was ACDDH012.

Cyclic dips in the recovery correspond to the start of new holes, as the ground was typically more weathered at the surface. The QP (Abraham Whaanga) considers that there is a low risk associated with the diamond core sampling consistency with respect to the DQO.

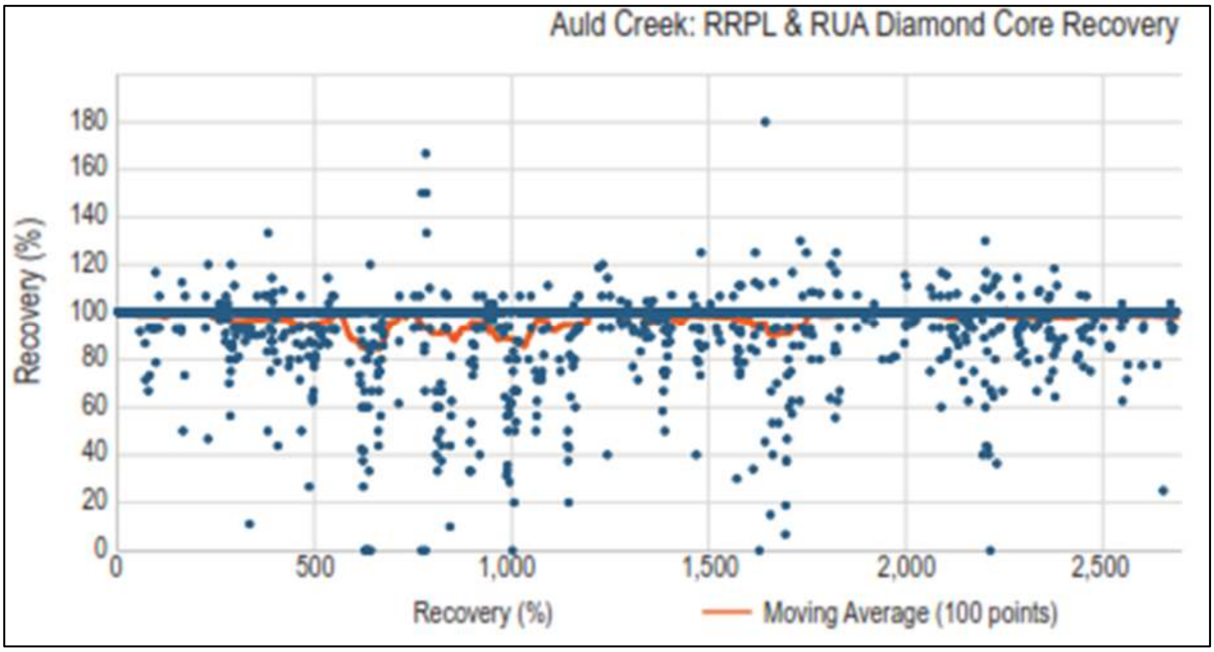


Figure 11-2: Sample recovery for Auld Creek.

11.5.3.1.5 First Split

The quality of the first-splitting process is typically monitored by the collection of duplicate or repeat samples. The consistency of the splitting process can be broadly assessed by tracking the relative difference (RD) of the duplicate or repeat pairs over time. For diamond drilling, the first split occurs when the core is cut.

RRPL collected quarter-core duplicate samples during drilling at the Project. Quarter-core duplicate samples were submitted to SGS for the same preparation and analytical methods as the primary core samples.

The RD ranges from approximately -10% to +10%. No major trends or marked step jumps are observed in the duplicate pairs (Figure 11-3), indicating that the splitting process was largely in control, e.g. no evidence of preferential sampling of sides of the core, or bias related to analytical method was observed.

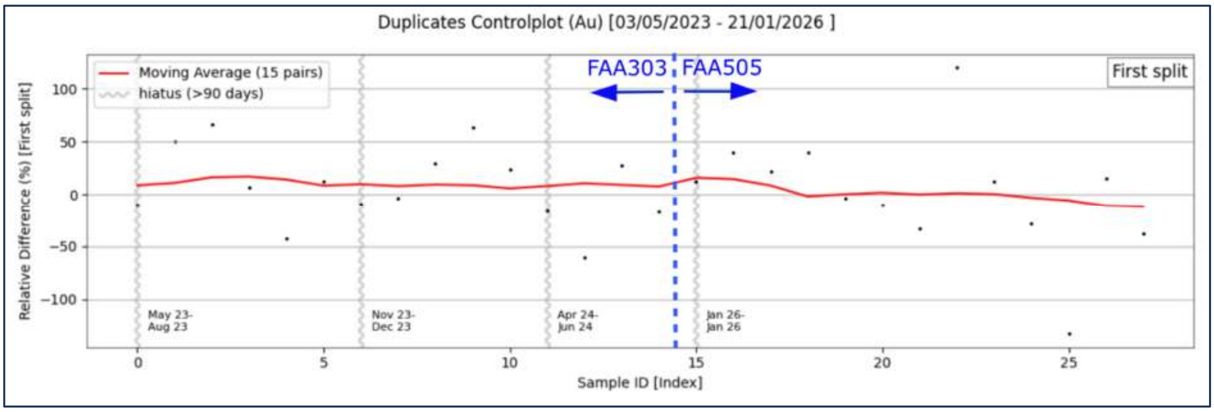


Figure 11-3: Relative difference in Au grades between quarter-core duplicate samples. Gold was analysed by FAA303 and FAA505 at SGS Waihi.

The QP (Abraham Whaanga) recommends RUA increases the collection of first-split duplicate samples for any diamond drillholes that are to be included in future resource estimates.

11.5.3.1.6 Second Split

No second-split duplicates of core samples were collected by RRPL during the crushing stage; therefore, the QP (Abraham Whaanga) cannot determine if the second-split process was in control. In the QP (Abraham Whaanga)'s opinion, this is acceptable for the purpose of delineating exploration targets and moderate-confidence MREs.

Second-split duplicates were collected at SGS Westport for the RUA samples. Only 11 second-split sample pairs were available, of which only six were above the limit of detection (LOD), which is insufficient for meaningful statistical analysis. The QP (Abraham Whaanga) recommends collecting second-split duplicates from the same samples that have core-split duplicates for any future resource delineation drilling programmes.

11.5.3.1.7 Third Split

Further reduction of the drill core sample (pulverisation) was carried out at the laboratory, after which another split was collected. SGS Westport collected a duplicate sample at a frequency of one per batch. The RD between sample pairs reporting above the Au LOD (>0.1 g/t Au), as a broad indication of splitting control, is depicted in Figure 11-4. The RD plot exhibits a step jump following the implementation of the FAA505 analytical method, characterised by an increase in both the magnitude and variance of the RD values. This shift is interpreted as special-cause variation associated with the change in analytical method and/or laboratory conditions, rather than inherent sampling variability. While the subsequent reduction in the moving average suggests a return towards more stable conditions, the number of samples analysed using FAA505 is limited. As such, limited data are available to confirm that the analytical process was consistently in control under the new analytical method. The QP (Abraham Whaanga) considers that the observed behaviour is consistent with a transition between analytical regimes; however, continued QC monitoring is required to confirm. The QP (Abraham Whaanga) recommends that RUA closely monitors duplicate performance and broader QC datasets as additional FAA505 data become available to confirm that the analytical process is operating within acceptable control limits.

Pulp repeat samples collected from the pulp bag were also analysed by pXRF. The RD plots for Sb exhibit a positive bias, with a median RD of +7% in analyses conducted by pXRF SN841694 (Figure 11-5). Analyses conducted by pXRF SN843701 do not have any trends or step jumps and yield a median RD of +1% Sb. A review of other elements (including Al, As, Ca, Fe, and Si) analysed by pXRF SN841694, conducted at the same time, indicates no trends. Antimony can be more difficult for a pXRF to accurately measure, due to interference with Ca for the Kα peak, and the location of the higher energy peaks in the area of high background in the spectra.

Based on the third-split repeat pairs, the QP (Abraham Whaanga) considers the third-split process to have been in control for pXRF analyses but needs improving for fire assay Au analyses.

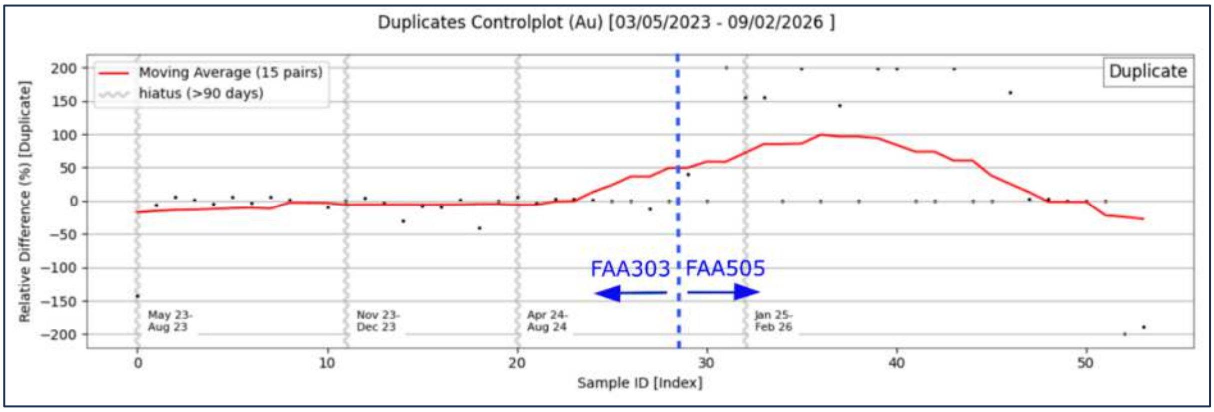


Figure 11-4: Relative difference in Au grades between the original and third-split repeat pairs (core samples only). Gold was analysed by FAA303 and FAA505 at SGS Waihi.

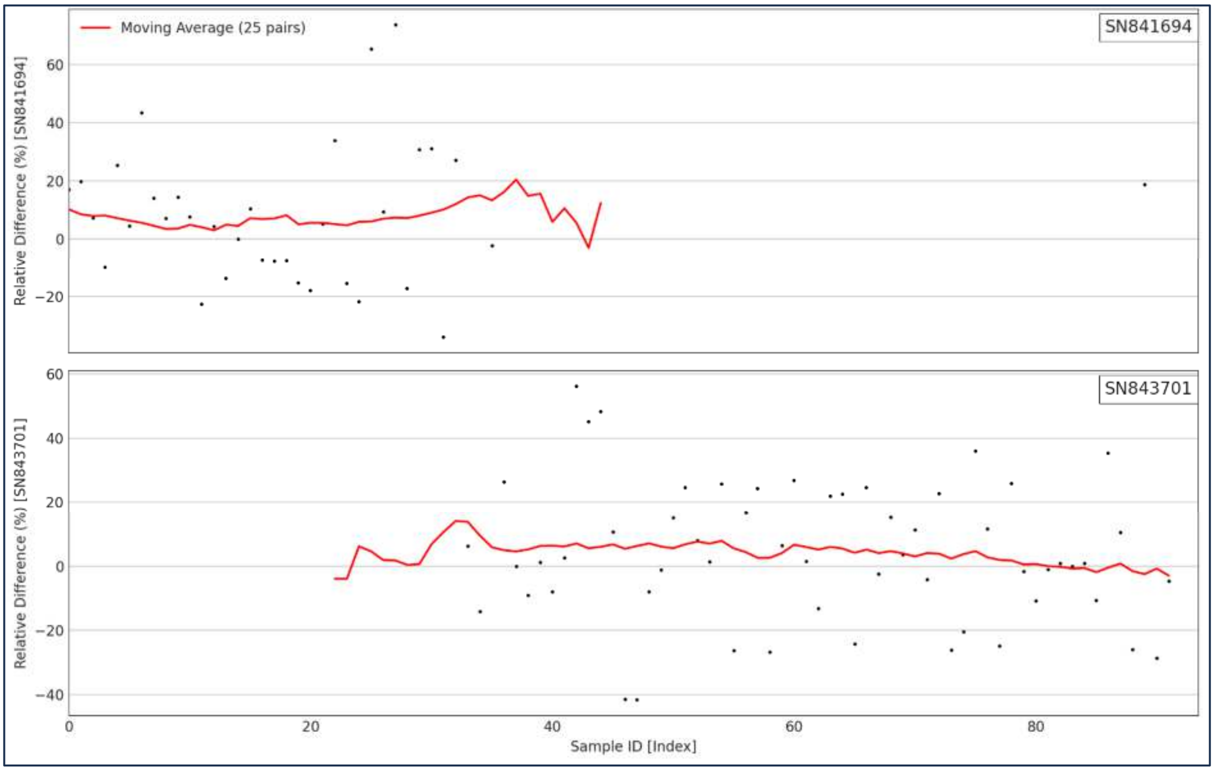


Figure 11-5: Relative difference in Sb grades between the original and third-split duplicate pairs against time (core samples only). Antimony was analysed by pXRF at the Reefton office.

11.5.3.1.8 Analytical Process: SGS

Quality control of the analytical process involves the repeated and continuous evaluation of CRMs. As part of its requirements under ISO accreditation, the laboratory inserts such reference materials into the sample stream, evaluates these, and makes corrections to the system when errors occur. The QP (Abraham Whaanga) notes that the analytical results of the internal reference material (IRM) used by the laboratory were typically already corrected (e.g. QC had already taken place, the system stopped when transgressions were identified, and the values were replaced by new and correct values).

It is common in the minerals industry for companies to submit their own (disguised) CRMs. However, in the QP (Abraham Whaanga)'s experience, this only achieves its intended purpose when the data are immediately and properly reviewed and correct inferences are drawn from the data. The timeframe between analysis and evaluation of the results means that correcting a system in real time is not possible; therefore, QC cannot be effectively carried out.

Quality control results obtained from blanks and CRMs demonstrate that analytical processes were in control and performing within acceptable limits. These controls are applied routinely by laboratories and provide an assessment of general laboratory performance independent of specific projects. By comparison, duplicate or repeat samples generated from the various splitting stages are project-specific and are used to evaluate whether the sampling and sample preparation processes are in control for the materials under investigation. The discussions around QC in this section with respect to CRMs and blanks are informed by data collected during analysis of samples from across the Reefton Project, which the QP (Abraham Whaanga) considers appropriate for determining whether the processes producing the data underpinning the MRE presented in section 14 were in control throughout the analyses.

For the RRPL samples, at least one blank was submitted per drilling instruction/drillhole. Two blanks were typically included: one as the first sample and one placed between predicted higher-grade samples. If the submission included fewer than 15 samples, only one blank was typically submitted. Blank samples consisted of coarse basalt sourced from Blackhead Quarry, Dunedin. Quartz washes were inserted after any samples with visual Au. At the start of the programme, two CRMs were included per submission; later, after RRPL reviewed its QC procedures, RRPL inserted a CRM after every 20 samples. A laboratory repeat sample was requested every 25 samples. SGS undertook one laboratory repeat per submission.

RRPL inserted 12 certified Rocklabs CRMs (Table 11-3) to monitor the quality of the analytical process. If a CRM was reported outside the limits accepted by RRPL (three times the standard deviation), the job was repeated by SGS. The QP (Abraham Whaanga) conducted a *post hoc* review of the CRM data to determine the consistency of the analytical process that delivered the data.

Table 11-3: Certified reference materials inserted by RRPL during analysis.

CRM	Source	Certified Value Au (ppm)	Standard Deviation	Number of Analyses
SG66	Rocklabs Ltd	1.086	0.032	44
SL51	Rocklabs Ltd	5.909	0.136	51
SJ53	Rocklabs Ltd	2.637	0.048	61
SN50	Rocklabs Ltd	8.685	0.180	51
SK52	Rocklabs Ltd	4.107	0.088	20
SH41	Rocklabs Ltd	1.344	0.041	45
SL61	Rocklabs Ltd	5.931	0.177	46
Si54	Rocklabs Ltd	1.780	0.034	45
SE68	Rocklabs Ltd	0.599	0.013	27
SF57	Rocklabs Ltd	0.848	0.030	14
OxH97	Rocklabs Ltd	1.278	0.009	21
OxJ95	Rocklabs	2.337	0.018	12

The control on the analytical process was assessed using RSC's in-house QC tool. Westgard rules 1x3s, 2x2s, 4x1s, 7x, and 6t (Table 11-4) (Westgard et al., 1981; Sterk, 2015) were used for the detection of special-cause variation.

Table 11-4: Explanation of the Westgard rules.

Rule	Explanation
1x3s	One result outside of three standard deviations from the mean.
2x2s	Two consecutive results outside two standard deviations from the mean.
4x1s	Four consecutive results outside one standard deviation from the mean.
7x	Seven consecutive results on one side of the mean.
6t	Six consecutive results trending in the same direction (e.g. six results where every result is higher than the previous).

Westgard rule violations, indicating the presence of special-cause variation, were reported in all CRMs, except for SH41 (Figure 11-6). The most common rule violation was 1x3s, where one or more analyses were reported outside three times the standard deviation.

RSC plotted all CRM analyses on a heat map to identify periods in which multiple CRMs demonstrated special-cause variation (Figure 11-7). Heat maps overlay results for all CRMs alongside the average fail rate per CRM on a given date. This approach addresses the problem faced when following the standard Westgard rules and carrying out the data review process after it has already been completed, where some CRMs exhibit special-cause variation while others for the same batch/period do not. The heat map approach is a more pragmatic and holistic approach; it enables identification of the periods in which multiple transgressions occurred across various CRMs and provides a more practical way to evaluate whether there were any significant issues with consistency at the laboratory. The heat map approach also illustrates the importance of having CRMs that span a representative portion of the grade range to assess whether any issues at the laboratory are consistent across the grade range. A review of the heat map indicates that six analyses across three CRMs failed over a 2-day period in January 2022; however, overall, there is no indication of significant special-cause variation.

RRPL and RUA inserted a total of 403 coarse sample blanks across the various work orders for the Reefton Project (including Alexander River, Big River, and Auld Creek) drilling programmes (Figure 11-8). Most samples returned an Au grade below the LOQ (0.1 g/t Au); however, four returned Au grades at or above the LOQ. These four samples were assayed within a short timeframe (less than one month), in different batches or work orders, with other blanks inserted at the same time performing well. RRPL and RUA reviewed QC data as they were returned, and three of the four batches that included a blank reporting >0.1 g/t Au were re-assayed at SGS Waihi by fire assay. The fourth blank in question was analysed by screen fire assay, and the batch was not re-assayed in full.

No major trends or step changes are identified in the RD plot for analytical repeat pairs analysed by FAA303 and FAA505 (Figure 11-9), based on a moving average of 15 pairs. However, the QP (Abraham Whaanga) notes that, after filtering the data to an LOQ of 0.03 ppm Au, only 24 repeat pairs remain for the entire reporting period, and only four of these are from the period after June 2024 (i.e. after the change in analytical method). As such, insufficient data are available for meaningful statistical analysis of the FAA505 method in isolation, and the QP (Abraham Whaanga) cannot conclusively determine that the analytical process was always in control. As SGS Waihi is ISO 17025 certified and is thus required to monitor QC and investigate out-of-control results/apply corrective actions where necessary, the QP (Abraham Whaanga) considers the lack of certainty to present a low risk with respect to the classification of moderate-confidence mineral resources. However, the QP (Abraham Whaanga) recommends RUA monitors the performance of analytical repeat pairs using monthly QA/QC reports to ensure that any issues are identified and resolved.

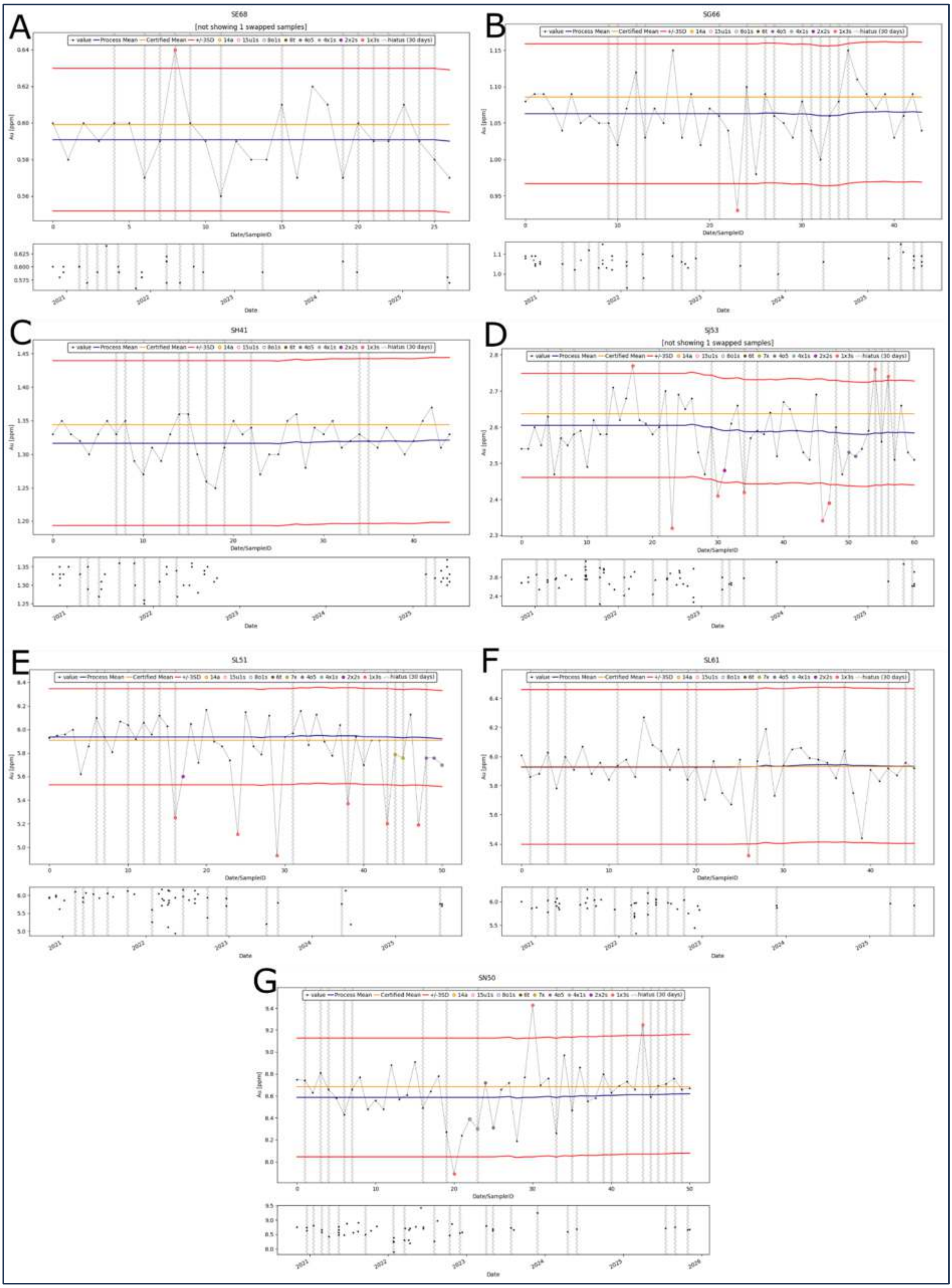


Figure 11-6: Control plots for A) SE68, B) SG66, C) SH41, D) SJ53, E) SL51, F) SL61, and G) SN50, analysed for Au by FAA303 and FAA505.

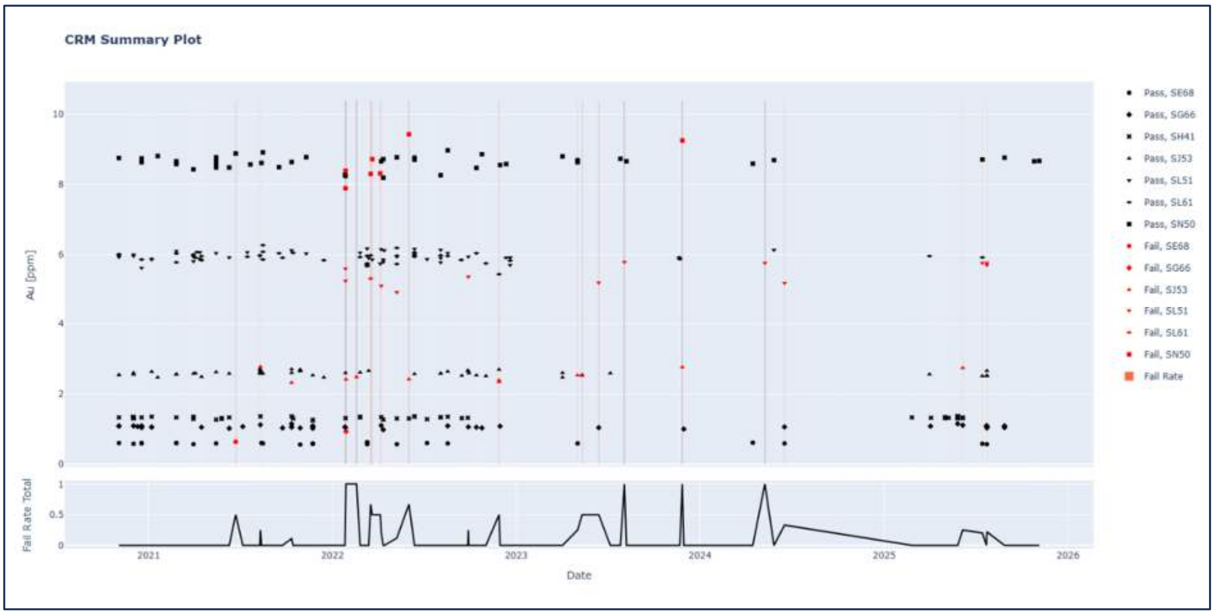


Figure 11-7: CRM heatmap illustrating pass and fail instances of seven standards (n > 25, Z-score < 10) used by RRPL during analysis.

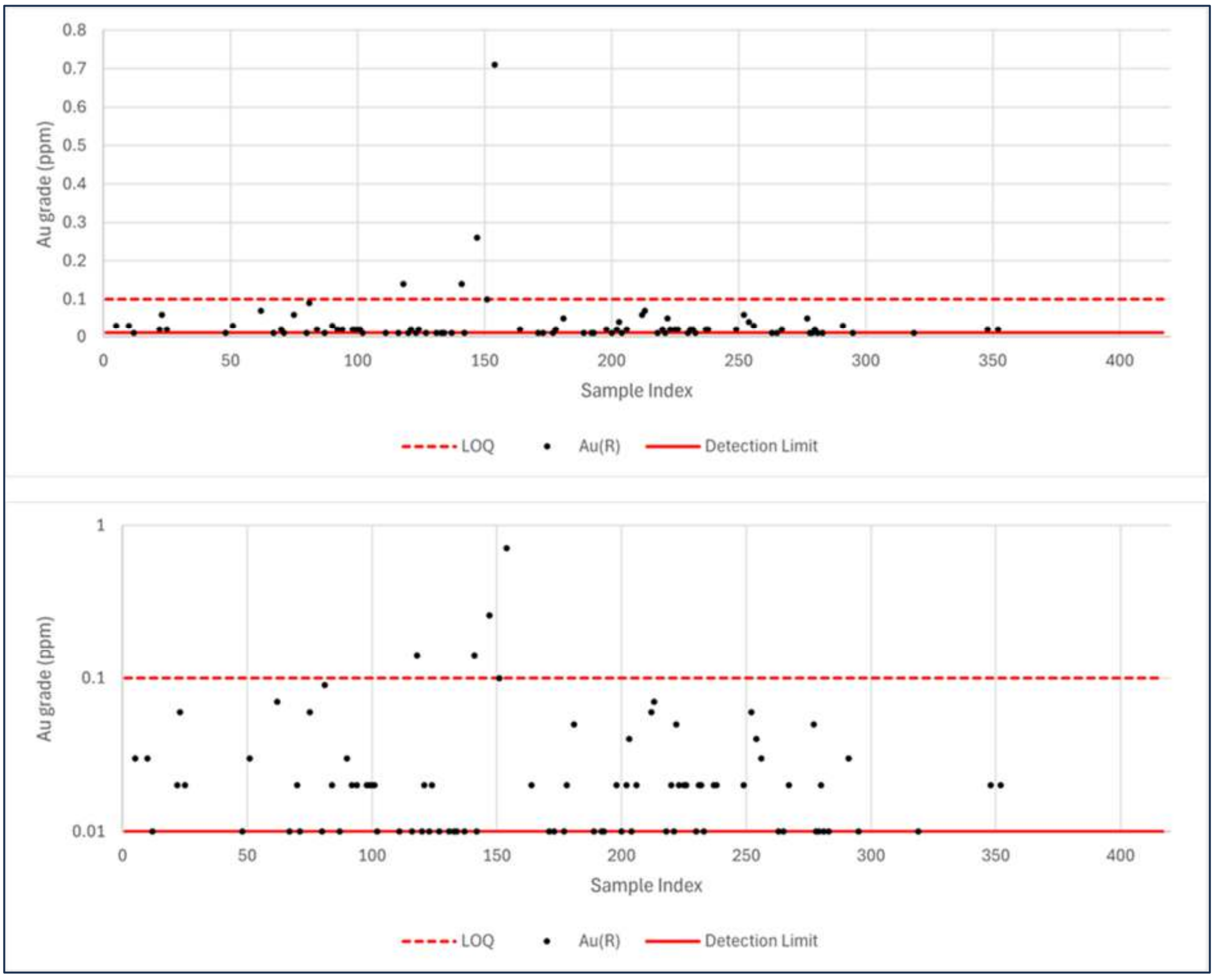


Figure 11-8: Linear (above) and logarithmic (below) plots of blanks by Au grade and sample index. Samples returning below detection limit values are not shown.

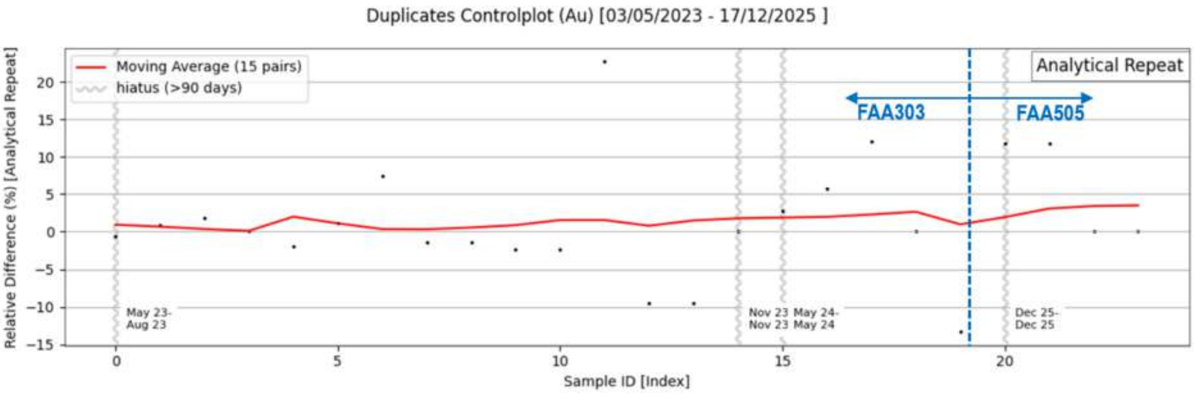


Figure 11-9: Relative difference in Au grades between analytical lab repeat analyses by sample index (core samples only). Gold was analysed by FAA303 and FAA505 at SGS Waihi.

In the opinion of the QP (Abraham Whaanga), the practice of reviewing the CRM and blank data and re-assaying a batch if it does not meet certain thresholds is excellent; however, the QP (Abraham Whaanga) recommends also re-assaying original blanks that fail during a programme of re-analysis, rather than just the primary sample material. Overall, the QP (Abraham Whaanga) considers the analytical process to have been largely in control, and any demonstrated special-cause variation was considered by the QP (Abraham Whaanga) in classifying the resource.

11.5.3.1.9 Analytical Process: pXRF

Soil, core, and trench sample pulps were analysed by RRPL using pXRF. Procedures for QC varied among the projects, with QC samples inserted into the sample stream at a frequency of 1:25 to 1:40. Where no significant Sb (<5,000 ppm) was expected, CRMs OREAS 903, OREAS 245, and OREAS 277 were used. Where Sb concentrations were expected to be elevated, CRMs OREAS 292 (4.5% Sb) and OREAS 245 (0.34% Sb) were used. One blank and three CRMs were analysed prior to analysis.

CRMs were inserted into the sample stream to allow post-processing correction of the data and to monitor the consistency of the pXRF during analysis. Blanks were inserted to ensure that any contamination of the instrument was identified before analysis began. Duplicates were used to test the precision of the instrument. Repeat samples were used to test the variability of the sample material. However, the pXRF data were not calibrated against the OREAS standards by RRPL. The QP (Abraham Whaanga) recommends correcting all data using calibration plots after each upload into the database; the calibration plots should be based on the expected values for each element in the CRMs plotted against the analysed values of the CRMs. The gradient of the linear fit between the expected and analysed values defines the correction factor used to calibrate the collected geochemical data (Fisher et al., 2014; Gazley and Fisher, 2014).

RRPL used two different pXRF instruments (of the same model) to perform the multi-element analysis. The QP (Abraham Whaanga) reviewed the QC data specific to each instrument to ensure that the analytical process of both instruments was in control.

The analytical process was assessed using RSC's in-house QC tool. Westgard rules 1x3s, 2x2s, 4x1s, 7x, and 6t (Table 11-4) (Westgard et al., 1981; Sterk, 2015) were used for the detection of special-cause variation. Control plots of the different OREAS CRMs analysed for Sb are presented in Figure 11-10.

The control plots indicate that, for As, 7x is the most frequent Westgard rule violation and is predominantly recorded in analyses completed by pXRF SN841694. No trends are evident in the As data; however, a step jump was observed in OREAS277 in December 2022.

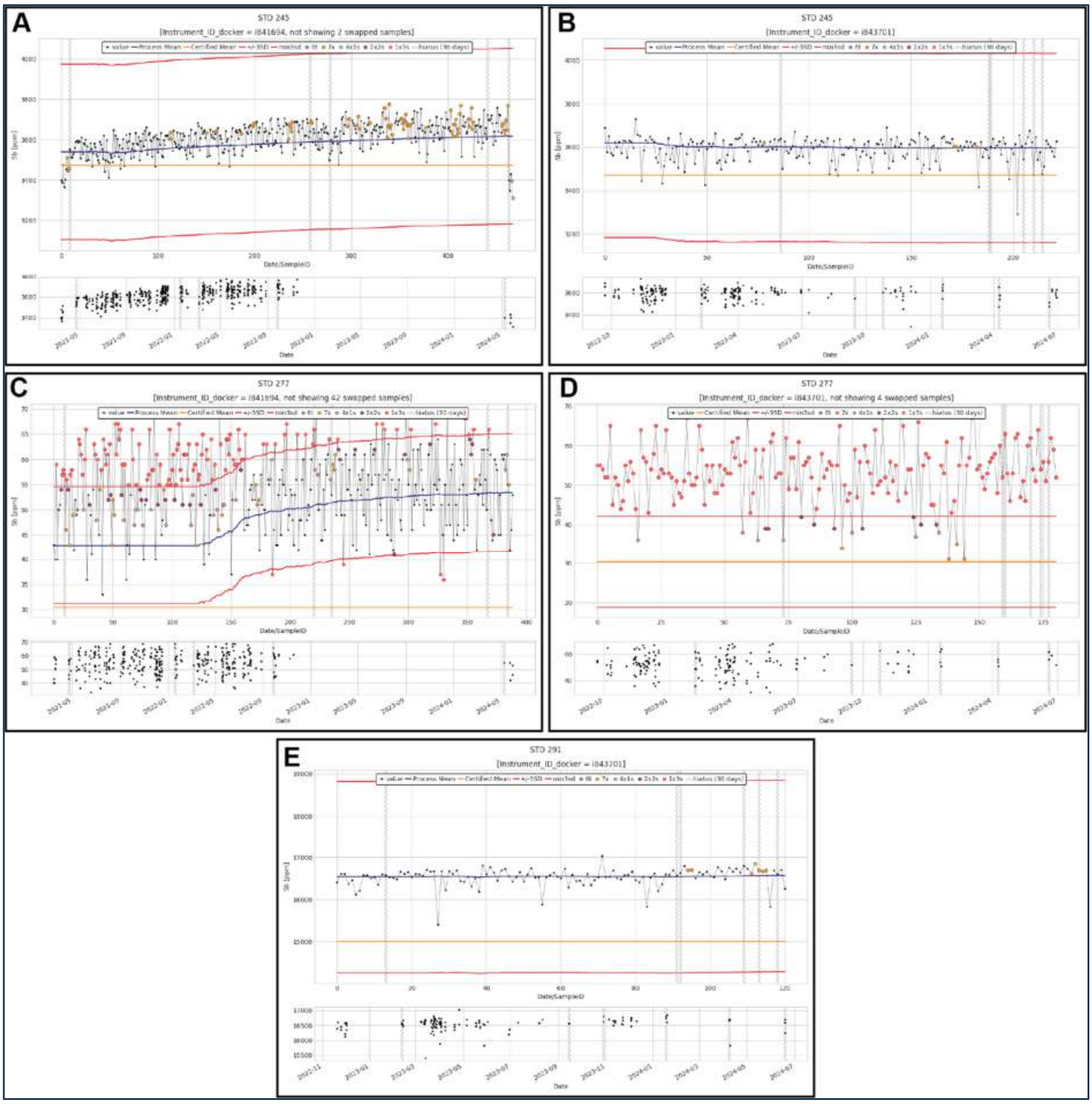


Figure 11-10: Control plots analysed for Sb. A) OREAS245 analysed by pXRF SN841694. B) OREAS245 analysed by pXRF SN843701. C) OREAS277 analysed by pXRF SN841694. D) OREAS277 analysed by pXRF SN843701. E) OREAS291 analysed by pXRF SN84370.

Several 7x, 1x3s, 2x2s rule violations, and trends were observed across the different CRMs analysed for Sb. The data collected from OREAS 245 by pXRF SN841694 has a positive trend over time. This suggests the pXRF analysis was not always in control. No other CRMs have a trend in either the Ab or Sb data.

The consistency of the pXRF analytical process can also be assessed by proxy, by reviewing the RD between the original and replicate measurements. A replicate measurement is obtained by taking a second pXRF measurement without moving the sample. The RD varied from approximately -30% to +30 %, similar to the third split (pulp) repeat sample. Several outliers (>+40% or >-40%) were also present. No major trends were visible.

Following a review of the CRM and replicate data, the QP (Abraham Whaanga) considers the laboratory analytical process to have been predominantly in control; however, the CRM data highlight that issues may arise when analysing Sb using pXRF. The location of the low-energy peak corresponds to Ca and

K, whereas the high-energy peaks are measured at part of the spectra where the background is high; thus, the limit of quantification for Sb is likely to be relatively high.

The RD plot in Figure 11-9 indicates the analytical process was predominantly in control, with no trends or step changes observed in the data.

11.5.3.2 Trench Samples

11.5.3.2.1 Trench Location

Trench data was used to inform early exploration work, whereas recent exploration has been informed primarily by drilling completed at the Project. Trench location data were validated against the collar coordinates and high-resolution LiDAR imagery. The QP (Abraham Whaanga) recommends RUA collects and records repeat GPS measurements to allow quantitative assessment of the quality of the location data. Based on the LiDAR validation, the QP (Abraham Whaanga) is of the opinion that the risk associated with the trench location data is low with respect to the DQO.

11.5.3.2.2 Primary Sample

RRPL collected field repeat samples at a rate of approximately one repeat sample per trench. The consistency of the primary sampling process can be broadly assessed by tracking the RD of the repeat pairs over time.

The RD between the repeat pairs ranges from -41% to +67% (Figure 11-11). No trends or step jumps are observed in the repeat pairs, indicating the sampling process was in control.

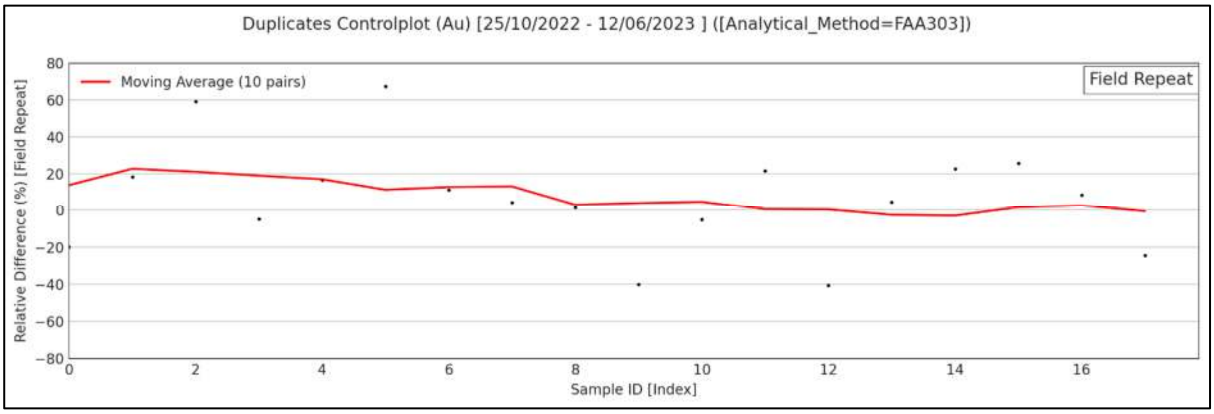


Figure 11-11: Relative difference in Au grades between trench field repeat samples. Gold was analysed by FAA303 at SGS Waihi. Data filtered above the LOQ (0.1 g/t Au).

11.5.3.2.3 First Split

No duplicate or repeat samples were collected following the coarse crush. Therefore, the QP (Abraham Whaanga) could not establish whether the first split stage was always in control. The QP (Abraham Whaanga) recommends that duplicate samples are collected during the first split to monitor the performance of the splitting stage.

11.5.3.2.4 Second Split

The second split stage was monitored by the collection of repeat samples of the trench samples pulverised by SGS Westport. The RD between sample pairs reporting above the Au LOQ (0.1 g/t Au) is depicted in Figure 11-12. The RD plot does not exhibit any trends or step jumps. The RD in Au grade between the second split repeat pairs is -94% to +67%. Based on the RD of pulp repeat samples, the QP (Abraham Whaanga) considers the second split process to have been in control.

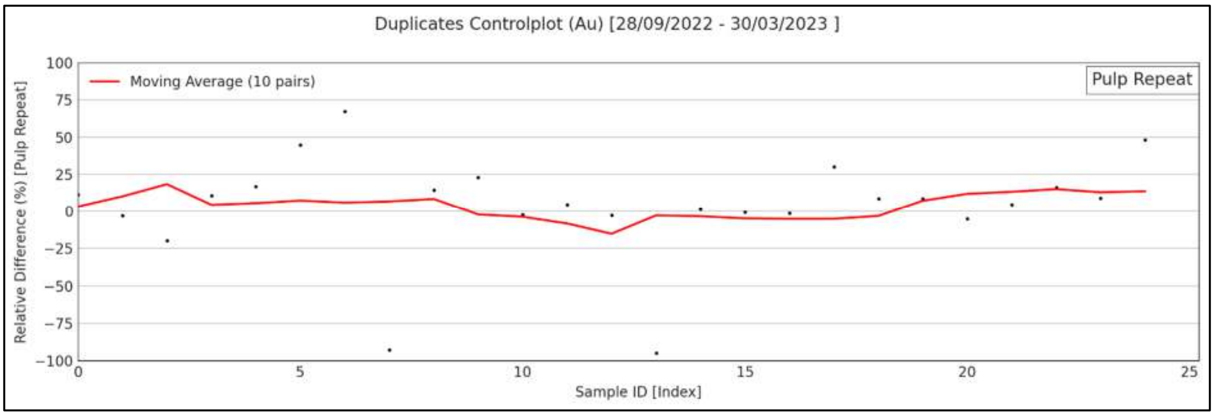


Figure 11-12: Relative difference in Au grades between trench pulp repeat samples. Gold was analysed by FAA303 at SGS Waihi. Data filtered above the LOQ (0.1 g/t Au).

11.5.3.2.5 Analytical Process: SGS

Sample blanks and CRMs were inserted into the sample stream along with the trench samples. All sample blanks returned Au below the LOQ (0.1 g/t).

The CRM results are reported in Section 11.5.3.1.8 and indicate the analytical process was in control.

11.5.4 Quality Acceptance Testing

Quality acceptance testing is where a final judgement of the data is made by assessing the accuracy and precision of the data for those periods where the process was demonstrated to be in control, and separately for those periods where the process was demonstrated to be not in control. Accuracy and precision are evaluated, and a final pass/fail assessment is made based on the DQO.

11.5.4.1 Diamond Drilling Samples

11.5.4.1.1 Collar Location

No quantitative quality data were available for the collar location collection process; therefore, accepting the quality (accuracy and precision) of the collar location data based on statistically defined thresholds is not possible. Based on reviews of the processes, systems, and tools available to determine collar locations described above, the collar location data are considered by the QP (Abraham Whaanga) to be fit for the purpose of defining exploration targets and classifying moderate-confidence mineral resources.

11.5.4.1.2 Downhole Survey

No quantitative downhole survey data were collected; therefore, the quality of the analytical process, as determined by accuracy and precision, cannot be determined. Based on the adequacy of the operating procedures (Section 11.5.2.1.2), and a quantitative threshold review of dogleg severity, the downhole survey data are considered by the QP (Abraham Whaanga) to be fit for the purpose of defining exploration targets and moderate-confidence mineral resources.

11.5.4.1.3 Bulk Density

Samples were weighed using either WS201-10K Wedderburn scales (as instructed by the density measurement SOP) or kitchen scales (not included in the SOP). The Wedderburn scales had a readability of 0.5 g, whereas the kitchen scales had a readability of 1 g. The QP (Abraham Whaanga) considers the readability of both scales to be acceptable with respect to the DQO, but the Wedderburn scales should be given preference.

Water temperature data were recorded to ensure the correct density of water was used in the calculation of the sample density. RRPL did not record the mass of reference core digitally; therefore, no quantitative quality data were collected, and the quality of density data collection cannot be determined in terms of accuracy and precision. RUA collected and recorded these data from December 2024 to February 2026; however, insufficient data were collected for meaningful statistical analysis. The QP (Abraham Whaanga) has reviewed the Density Master spreadsheet and is of the opinion that the results are mostly consistent, other than a small number of outliers that are likely to be transcription errors, and considers this acceptable for the classification of low to moderate-confidence mineral resources.

Based on the adequacy of the operating procedures (Section 11.5.2.1.3), the QP (Abraham Whaanga) is of the opinion that the risk associated with the density data is low with respect to the DQO. However, the QP (Abraham Whaanga) recommends that, going forward, the SOP should be updated to include tolerance thresholds and the collection of repeat samples, and all QC data should be entered into a suitable database, to ensure that data are of sufficient quality to support the definition of high-confidence mineral resources.

11.5.4.1.4 Primary Sample

A practical means of checking and verifying the quality of a sample is to validate it against, or compare it with, a sample with a known grade. In simple terms, the difference between the analysed value and the ‘known’ value is then defined as the bias, which is a measure of sample quality. Precision can be benchmarked by comparing the variance in the measurements of samples with the variance in check samples. This is the principle behind the utility of laboratory CRMs.

For the primary samples, i.e. the sample collected at the drill bit, such options are limited. The next practical way to determine the quality of the primary sample is to compare it with a sample of similar or better quality taken at the same location. This process is often called twinned drilling, but it can be used whenever a sample from drill/sample A is close enough to a sample from drill/sample B.

As of the effective date of this Report, no twin drilling has been conducted at the Project. In the QP (Abraham Whaanga)’s opinion, this is acceptable for early-stage exploration programmes and moderate-confidence mineral resources; however, twin drilling, particularly of significant intersections, is recommended as the Project progresses to higher-confidence resource definition.

The quality of the primary sample can be assessed by proxy by assessing sample recovery rates. Sample recovery was actively monitored by RRPL and RUA during drilling (Section 11.5.2.1.4). Drill core recovery averaged 97.2% for intervals returning >1 g/t Au and was consistent across the different hole diameters (Table 11-5). Lower than average recovery (as low as 74%) was recorded for three holes at Auld Creek.

The QP (Abraham Whaanga) notes that an average recovery of 95% is a standard recovery target for diamond drilling under most conditions Annels and Dominy (2002). The data are considered by the QP (Abraham Whaanga) to be fit for the purpose of classifying moderate-confidence mineral resources.

As a check for primary sample quality, sample recovery can be used as a proxy to investigate the impact of grade distribution. No trend is observed between sample recovery and Au or Sb grades (Figure 11-13).

Table 11-5: Summary of drill recovery for intervals returning >1 g/t Au.

Operator/Core Size	Auld Creek (%)
OGC/MMCL	-
OGL	98.8
RRPL	99.0
RRPL PQ	99.7
RRPL HQ	92.7

Operator/Core Size	Auld Creek (%)
RUA	98.2
RUA PQ	95.9
RUA HQ	99.5
All Holes	98.7

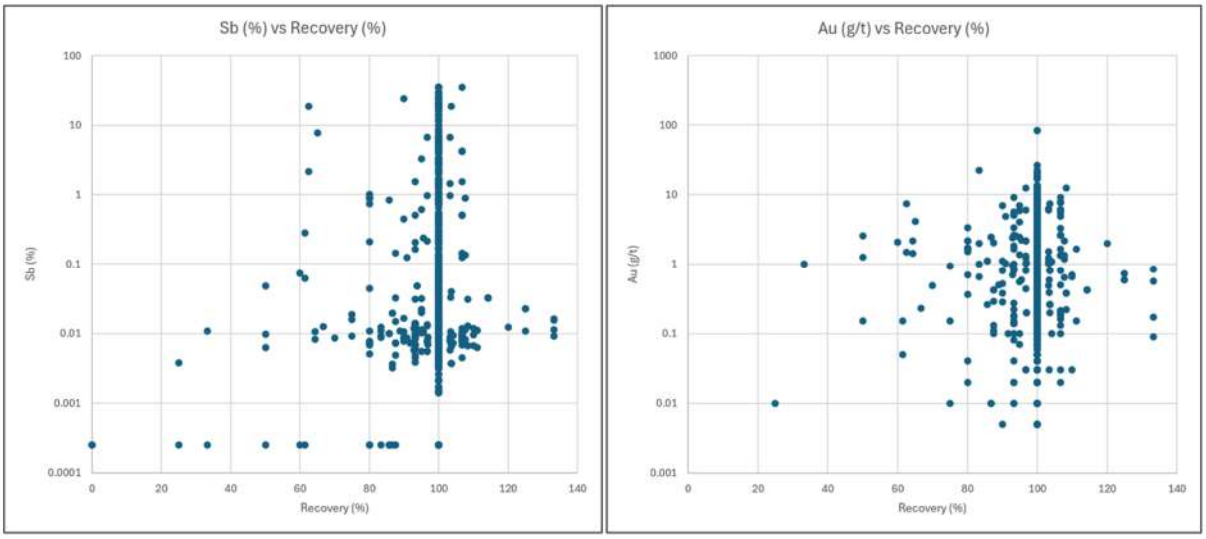


Figure 11-13: Sample recovery vs grade at Auld Creek for Au (left) and Sb (right).

11.5.4.1.5 First Split

RRPL collected only 15 quarter-core duplicate samples from the Project at a rate of ~one duplicate per drillhole or one in every 50 primary samples; therefore, it is not possible for the QP (Abraham Whaanga) to determine the accuracy and precision of the first split based on statistically defined thresholds. The QP (Abraham Whaanga) conducted a *post hoc* analysis of quarter-core duplicate data from the Reefton Project as a proxy to assess whether analytical data obtained from the first split are likely to be sufficiently accurate and precise. RRPL collected 110 quarter-core duplicate samples from across the Reefton Project at a rate of around one per drillhole or one in every 50 primary samples. Sixty (60) quarter-core duplicate samples returned an Au grade above the LOQ (0.03 ppm). Figure 11-14 presents scatter and QQ plots for Au. A Wilcoxon signed-rank test indicates no statistically significant bias was introduced during the first split (Table 11-6 and Figure 11-14). No quarter-core duplicate samples were collected by RUA.

Based on the adequacy of the operating procedures (Section 11.5.2.1.5), the QP (Abraham Whaanga) considers the sub-sampling methodology appropriate for the style of mineralisation, and the quality of the data acceptable with respect to the DQO. The QP (Abraham Whaanga) recommends RUA conducts first-split duplicate sampling to identify any quality issues at this stage of splitting.

Table 11-6: Precision summary for quarter-core sample pairs from the Reefton Project.

Analyte	Split	N pairs	LOQ	Wilcoxon p-Value	Wilcoxon Verdict	RMSCV (%)
Au	Quarter-Core	60	0.03 ppm	0.455	Accept H_0	30

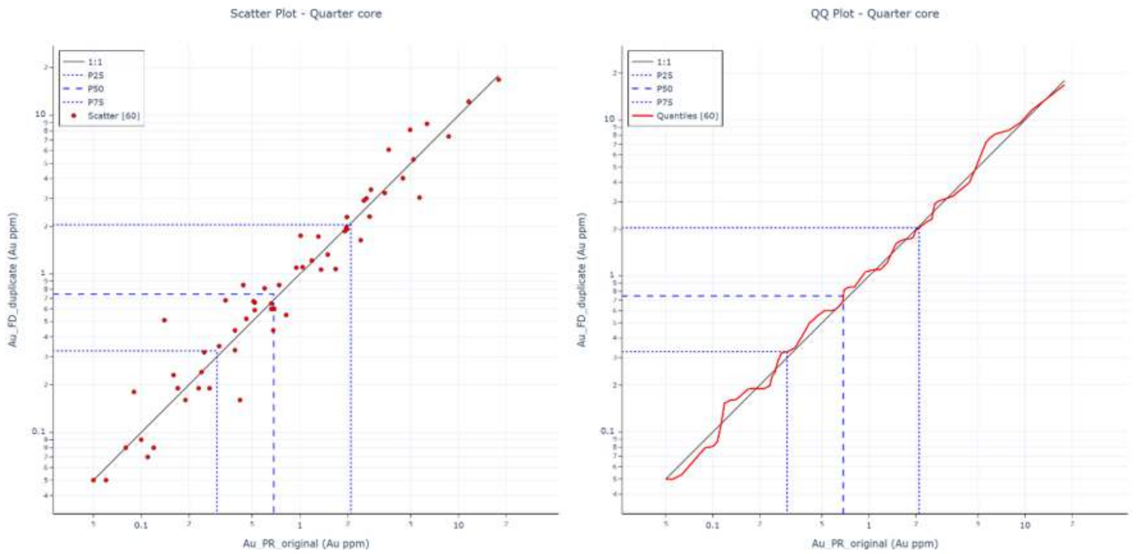


Figure 11-14: Scatter (left) and QQ (right) plots of quarter-core sample pairs from across the Reefton Project. Data filtered to an LOQ of 0.03 ppm.

11.5.4.1.6 Second Split

No second-split duplicate samples were collected by RRPL, and only 11 sample pairs were available from the RUA drilling; therefore, it is not possible for the QP (Abraham Whaanga) to determine the accuracy and precision of the second split based on statistically defined thresholds. Based on the adequacy of the operating procedures (Section 11.5.2.1.6), the QP (Abraham Whaanga) considers the sub-sampling methodology appropriate for the style of mineralisation, and the quality of the data acceptable with respect to the DQO. The QP (Abraham Whaanga) recommends RUA continue to conduct second-split duplicate sampling to identify any quality issues at this stage of splitting.

11.5.4.1.7 Third Split

The quality of the third split could be assessed following the determination that the third-splitting process was largely in control (Section 11.5.2.1.7).

A Wilcoxon signed-rank test was performed for Au (fire assay) and As and Sb pXRF repeat pairs and has no statistically significant bias (Table 11-7). The QP (Abraham Whaanga) also visually reviewed the scatter and QQ plots and did not observe a bias (Figure 11-15). The RMSCV value for the third split is 32% for Au, which is in line with industry expectations for this mineralisation style and this comminution stage, thereby indicating good precision.

Table 11-7: Precision summary for third split (pulp) sample pairs.

Analyte	Analytical Method	pXRF Serial No.	Split	N Pairs	LOQ	Wilcoxon p-Value	Wilcoxon Verdict	RMSCV (%)
Au	Fire assay [1]	NA	Third (pulp)	28	0.05 ppm	0.706	Accept H_0	32
Sb	pXRF	841694	Third (pulp)	35	10 ppm	0.171	Accept H_0	22
Sb	pXRF	843701	Third (pulp)	57	10 ppm	0.243	Accept H_0	19

Notes:

1. *Pre-2025 = 30 g charge weight; 2025 = 50 g charge weight.*

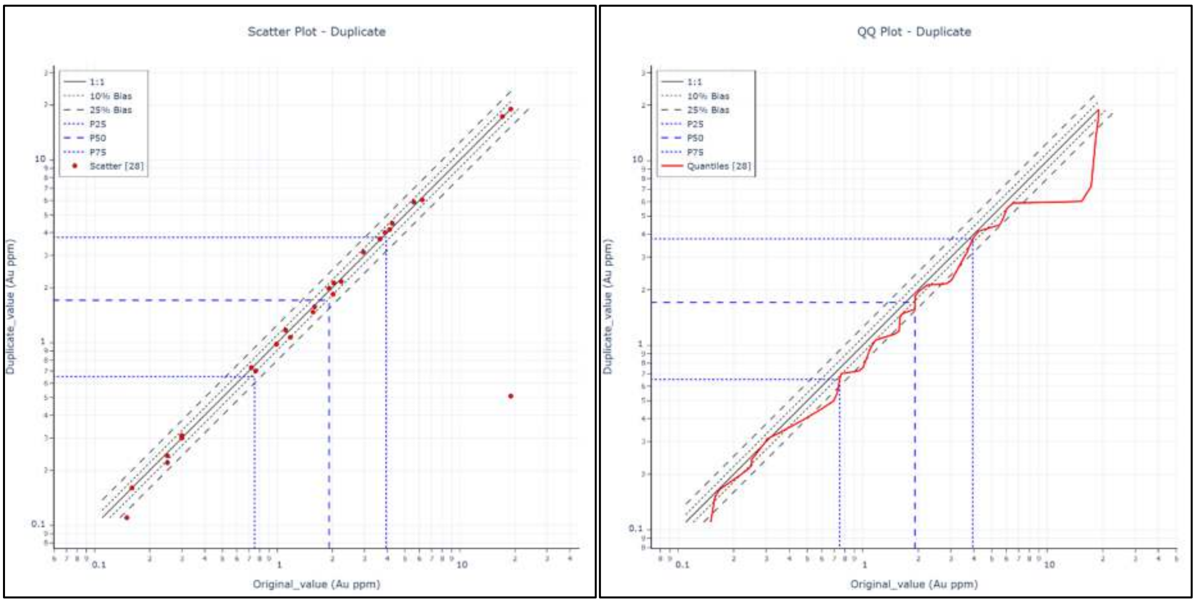


Figure 11-15: Scatter and QQ plots of third split (pulp) repeat pairs from diamond drilling samples collected by SGS Westport. Data filtered to a LOQ of 0.05 ppm.

11.5.4.1.8 Analytical Process: SGS

Blanks and CRMs are used as part of a laboratory's routine internal QA/QC programme and provide a general measure of analytical accuracy and precision independent of a particular project. These controls are applied consistently across all sample batches processed by the laboratory. By comparison, duplicate or repeat samples derived from the various splitting stages are project-specific and are used to evaluate the precision and accuracy of the sampling and sample preparation processes applied to the project samples. The discussions regarding quality acceptance testing in this section with respect to CRMs and blanks are informed by data collected during analysis of samples from across the Reefton Project, which the QP (Abraham Whaanga) considers appropriate for determining the accuracy and precision of the data underpinning the MRE presented in section 14.

RRPL inserted 12 different CRMs (Table 11-3). The QP (Abraham Whaanga) performed a *post hoc* review of data from seven of the CRMs that met RSC's minimum insertion threshold of 25 analyses. The CRMs suggest the analytical results were mostly precise and accurate (Table 11-7), with all but one CRM (SJ53) within the acceptable range (marginal to excellent) for the precision and accuracy z-score tests. A negative bias was recorded for all CRMs (-0.3% to -2.3%). The CRMs inserted were synthetic and not matrix matched; the QP (Abraham Whaanga) notes that Sb-rich mineralisation can cause issues with fusion during fire assay. While the bias is consistent across the different CRMs and follows the same trend (biased low), the QP (Abraham Whaanga) considers the analytical data acceptable with respect to the DQO for the definition of moderate-confidence mineral resources. The QP (Abraham Whaanga) strongly recommends changing the source of reference material to be matrix-matched to the style of mineralisation at the Project.

The RRPL blank data returned elevated Au results in three batches. The samples in these batches were re-assayed by SGS; however, the original blanks were not included as part of this re-assay programme. The QP (Abraham Whaanga) recommends re-assaying original blanks that fail, rather than just the primary sample material during a programme of re-analysis, and not just the primary sample material. The QP (Abraham Whaanga) also recommends including blank samples after suspected high-grade intervals. All 10 blanks for the RUA samples analysed by method FAA505 returned results below the detection limit (0.01 ppm).

The quality of the analytical lab repeat process could be assessed following the determination that the lab repeat process was in control (Section 11.5.3.1.8). A Wilcoxon signed-rank test was performed for

23 Au (fire assay) repeat pairs and demonstrates a statistically significant bias, with a p-value of 0.146 and Wilcoxon verdict of ‘Accept H_0’ (Table 11-8). In addition, the root mean square coefficient of variation (RMSCV) value for the lab repeats is 7% for Au, which is in line with expectations for this mineralisation style and this comminution stage. The QP (Abraham Whaanga) visually reviewed the scatter and QQ plots, which do not indicate a bias (Figure 11-16). Although statistical tests may demonstrate a statistically significant bias, they do not quantify the magnitude or *impact* of the bias, merely that a bias exists. It is thus important that statistical tests are assessed with a good dose of common sense to pass a judgement on whether an issue indeed exists. The lack of an obvious bias in the QQ plot suggests that any bias, *albeit statistically significant*, is very small and of low impact.

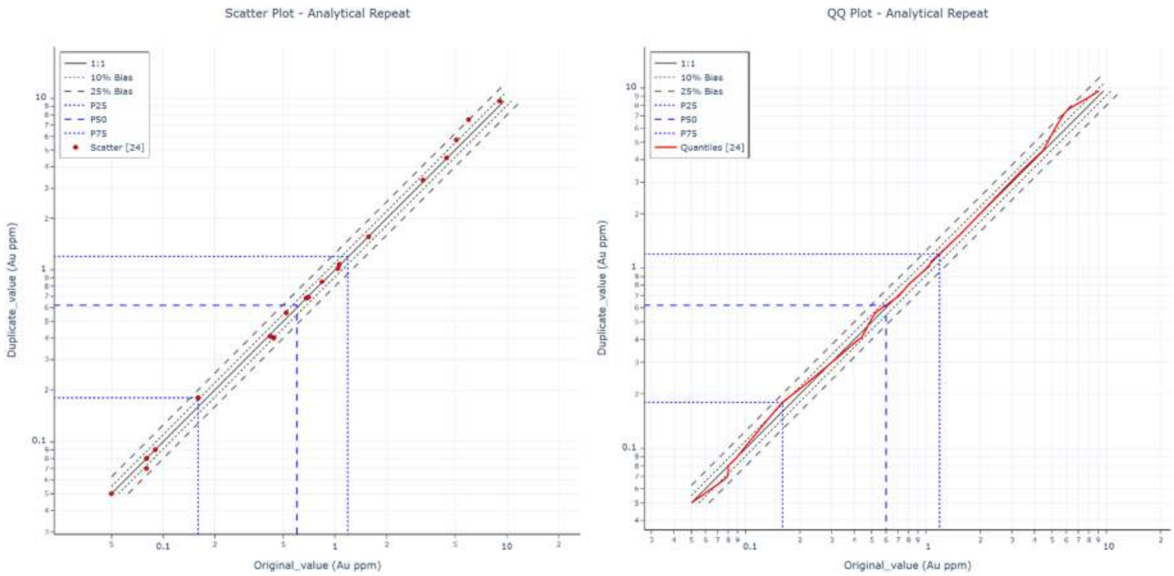


Figure 11-16: Scatter and QQ plots of analytical lab repeat analyses from diamond drilling samples collected by SGS Westport. Data filtered to an LOQ of 0.03 ppm.

Table 11-8: Precision summary for analytical lab repeat analyses (diamond drilling samples).

Analyte	Split	N pairs	LOQ	Wilcoxon p-Value	Wilcoxon Verdict	RMSCV (%)
Au	Analytical Repeat	23	0.03 ppm	0.146	Accept H_0	7

11.5.4.1.9 Analytical Process: pXRF

Quantitative acceptance criteria for the performance of CRMs, based on statistical thresholds, are set in RSC’s QC WebApp and match the expectations of the DQO. Precision acceptance is assessed by comparing the total variance of the analysis of CRMs, as determined by the laboratory, with the certified variance for each CRM. This is carried out using a Fisher test, which determines whether the variance in the laboratory assay data of the CRMs is statistically different from the certified variance at 95% confidence. Accuracy was assessed by comparing the process mean grade of the analysis of CRMs, as determined by the laboratory, with the certified mean value of the CRM, using t-tests or absolute average z-score tests (Table 11-9). The t-tests determine whether the difference between the two grades is statistically significant at a 95% confidence limit.

A review of the CRM data (four CRMs analysed by two pXRF instruments) indicates all CRMs except OREAS 277 (Sb) meet the precision thresholds determined by the DQO (Table 11-10). However, only one CRM meets the accuracy thresholds enforced with respect to the DQO (OREAS 903 for As, measured by pXRF SN841694). Most CRMs have a high bias (-3% to 9% for As and 3% to 79% for Sb). This is not unexpected, as the pXRF data were not calibrated.

In some instances, RRPL had both laboratory and pXRF Sb data for samples from Auld Creek. A review of these data by the QP (Abraham Whaanga) indicates that the pXRF under-reports Sb at low to moderate grades (<5,000 ppm; Figure 11-17).

Antimony is a difficult element to quantify using a pXRF, due to interference with Ca on the K_α beam. Antimony can be measured using different beams with a higher beam energy; however, this decreases the performance of the pXRF when analysing samples with a low Sb grade. This is observed in the CRM data, where OREAS 291, which has the highest certified value for Sb, performs the best.

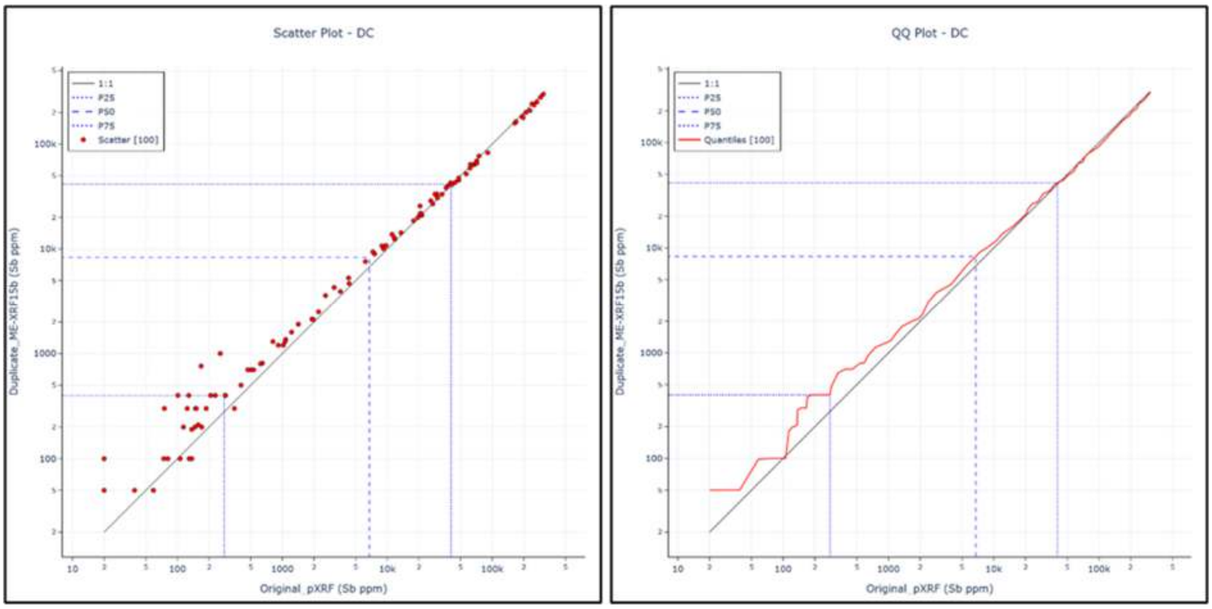


Figure 11-17: Scatter and QQ plots comparing pXRF and laboratory Sb analysis on pulp samples from Auld Creek diamond drill core samples.

Another way to determine the accuracy and precision of the pXRF data is by proxy of repeat measurements. Insufficient pXRF data from the Project were available for meaning statistuical analysis; therefore, as a proxy to assess whether the analytical data are likely to be sufficiently accurate and precise, the QP (Abraham Whaanga) assessed repeat measurements collected by RRPL from DD and trench samples as a whole across the entire Reefton Project at a rate of approximately 1:20. Data were available for a total of 100 sample pairs. After filtering the data using a lower LOQ of 20 ppm and an upper LOQ of 1,000 ppm, 98 pairs remained for analysis. A Wilcoxon signed-rank test was performed for the Sb pXRF repeat pairs, with the results indicating no significant bias (Table 11-11). The QP (Abraham Whaanga) also visually reviewed the scatter and QQ plots and did not observe a bias (Figure 11-18). The RMSCV value for the Sb pXRF repeat pairs is 19%, which is consistent with industry expectations for this mineralisation style and analytical stage, thereby indicating good precision Table 11-11).

While there are issues surrounding the Sb pXRF data, as indicated by the CRM review, the QP (Abraham Whaanga) considers the pXRF analytical data to be fit for the purpose of classifying moderate-confidence mineral resources. The QP (Abraham Whaanga) strongly recommends re-assaying Sb-bearing samples using laboratory methods (e.g. ME-XRF15b or ME-XRF15c at ALS Brisbane), and implementing a programme of calibrating the pXRF data to compensate for longer-term trends in pXRF analytical results related to the instrument.

Table 11-9: Performance summary of CRMs submitted by RRPL for the Project.

Analyte	Duration	CRM ID	CRM Certified Value	CRM SD	N	Process Mean	Avg. Z-Score	Avg. Abs Z-Score	Process Std	Bias (%)	F-Test (p)	F-Test Result (a=95.0)	Precision	Precision Z Result	Student-t (p)	Student-t Result (a=95.0)	Accuracy	Accuracy Z Result
Au	1829	SE68	0.599	0.013	27	0.592	-0.550	1.125	0.018	-1.193	0.072	Accept H0	Pass	Marginal	0.052	Accept H0	Pass	Acceptable
Au	1829	SG66	1.086	0.032	44	1.062	-0.756	1.085	0.04	-2.227	0.125	Accept H0	Pass	Marginal	0.006	Reject H0	Fail	Acceptable
Au	1829	SH41	1.344	0.041	45	1.321	-0.558	0.684	0.028	-1.703	0.026	Reject H0	Pass	Good	0.009	Reject H0	Fail	Acceptable
Au	1829	SJ53	2.637	0.048	61	2.575	-1.287	1.83	0.093	-2.342	0	Reject H0	Fail	Not Acceptable	0	Reject H0	Fail	Not Acceptable
Au	1829	SL51	5.909	0.136	51	5.834	-0.552	1.436	0.285	-1.271	0	Reject H0	Fail	Marginal	0.062	Accept H0	Pass	Acceptable
Au	1829	SL61	5.931	0.177	46	5.91	-0.117	0.635	0.165	-0.35	0.365	Accept H0	Pass	Good	0.312	Accept H0	Pass	Excellent
Au	1829	SN50	8.685	0.18	51	8.634	-0.286	0.968	0.25	-0.593	0.04	Reject H0	Fail	Good	0.155	Accept H0	Pass	Good

Table 11-10: Precision summary table of CRMs analysed by RRPL using pXRF.

Analyte	Duration	pXRF Serial No.	CRM ID	Certified Value	CRM SD	N	Process Mean	Process Variance	Process Std	Bias (%)	F-Test (p)	F-Test Result (a=95.0)	Precision	Student-t (p)	Student-t Result (a=95.0)	Accuracy
Sb	647	843701	OREAS 245	3471	145	222	3590.748	3114.189	55.805	3.45	0	Reject H_0	Pass	0	Reject H_0	Fail
Sb	647	843701	OREAS 277	30.4	3.9	181	52.552	61.015	7.811	72.87	0	Reject H_0	Fail	0	Reject H_0	Fail
Sb	647	843701	OREAS 291	15000	760	121	16540.157	44338.333	210.567	10.268	0	Reject H_0	Pass	0	Reject H_0	Fail
Sb	1173	841694	OREAS 245	3471	145	468	3613.019	5492.396	74.111	4.092	0	Reject H_0	Pass	0	Reject H_0	Fail
Sb	1173	841694	OREAS 277	30.4	3.9	389	54.635	55.325	7.438	79.72	0	Reject H_0	Fail	0	Reject H_0	Fail

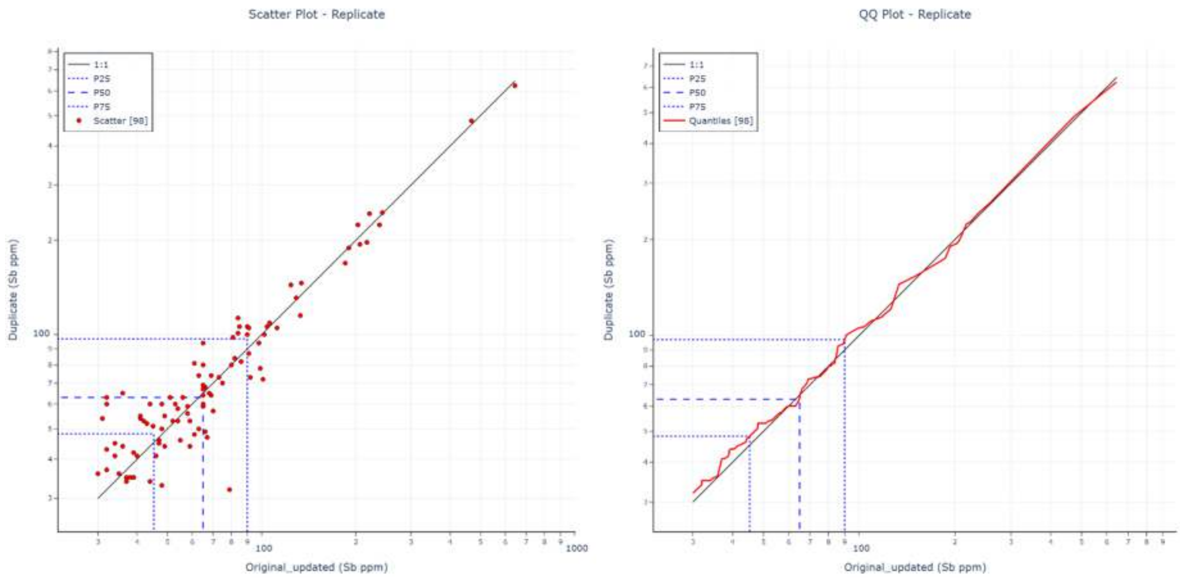


Figure 11-18: Scatter and QQ plots for diamond core and trench repeat analyses analysed by RRPL for Sb by pXRF (LOQ = 20–1,000 ppm).

Table 11-11: Precision summary for repeat analyses (diamond core and trench samples only).

Analyte	Split	N pairs	Lower LOQ (ppm)	Upper LOQ (ppm)	Wilcoxon p-Value	Wilcoxon Verdict	RMSCV (%)
Sb	Third (pulp)	98	20	1,000	0.186	Accept H_0	19

11.5.4.2 Trench Samples

11.5.4.2.1 Trench Location

No quantitative quality data were available for the trench location collection process; therefore, accepting the quality (accuracy and precision) of the trench location data based on statistically defined thresholds is not possible. Based on reviews of the processes, systems, and tools available to determine trench locations described in Section 11.1.4–11.5.2.2.1, the trench location data are considered by the QP (Abraham Whaanga) to be fit for the purpose of delineating moderate-confidence mineral resources.

11.5.4.2.2 Primary Sample

RRPL collected 38 field repeat trench samples at a rate of approximately one field repeat per trench. Additionally, Kent collected nine field repeat trench samples. In total, 28 repeat samples report an Au grade above the LOQ (0.1 g/t Au). Figure 11-19 presents scatter and QQ plots for Au analysed by SGS (FAA303). A Wilcoxon signed-rank test indicates no statistically significant bias was introduced during the repeat sampling (Table 11-12). The sample populations for the Kent field repeat samples and RRPL samples analysed by SGS (GO_FAP30V10) are low; therefore, statistically meaningful conclusions regarding the data's accuracy and precision cannot be made.

Based on the RRPL field repeat sample pairs (analysed by fire assay), in the opinion of the QP (Abraham Whaanga), the data produced by the field repeat sampling are of an acceptable quality with respect to the DQO.

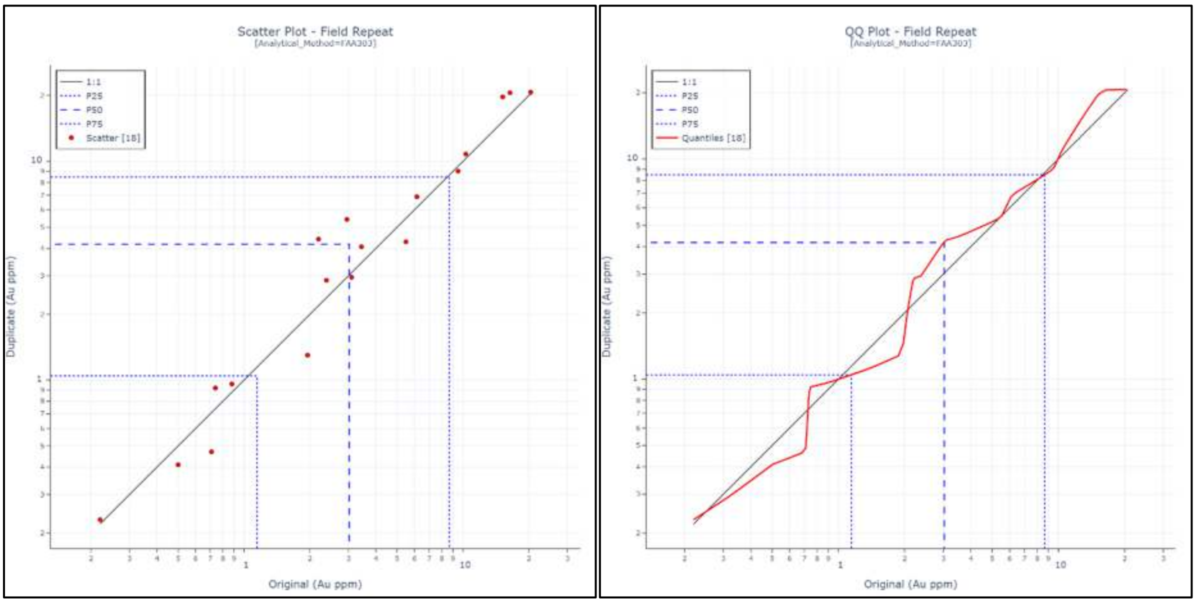


Figure 11-19: Scatter and QQ plots of field repeat pairs from trench samples collected by RRPL and analysed by FAA303 for Au by SGS.

Table 11-12: Precision summary for trench field repeat sample pairs.

Analyte	Analytical Method	Company	Split	N pairs	LOQ	Wilcoxon p-Value	Wilcoxon Verdict	RMSCV (%)
Au	FAA303	RRPL	Field Repeat	18	0.1 g/t	1.08	Accept H_0	26
Au	GO_FAP30V10	RRPL	Field Repeat	4	0.1 g/t	0.875	Accept H_0	75
Au	FAA515	Kent	Field Repeat	6	0.1 g/t	0.312	Accept H_0	28

11.5.4.2.3 First Split

No first-split duplicate samples were collected; therefore, it is not possible to determine the accuracy and precision of the second split based on statistically defined thresholds. Based on the adequacy of the operating procedures (Section 11.5.2.2), the QP (Abraham Whaanga) considers the sub-sampling methodology appropriate for the style of mineralisation, and the quality of the data is acceptable with respect to the DQO. The QP (Abraham Whaanga) recommends the laboratory routinely conduct duplicate sampling to understand any quality issues at this stage of splitting.

11.5.4.2.4 Second Split

The quality of the second split could be determined following the determination that the second-splitting process was in control (Section 11.5.3.2.4).

A Wilcoxon signed-rank test was performed for Au (fire assay) duplicate pairs and demonstrates no statistically significant biases (Table 11-13). The QP (Abraham Whaanga) also visually reviewed the scatter and QQ plots and did not observe a bias (Figure 11-20). The RMSCV value for the third split is 31% for Au.

Based on the second split (pulp) repeat samples, the QP (Abraham Whaanga) considers the second-split data are fit for purpose with respect to the DQO.

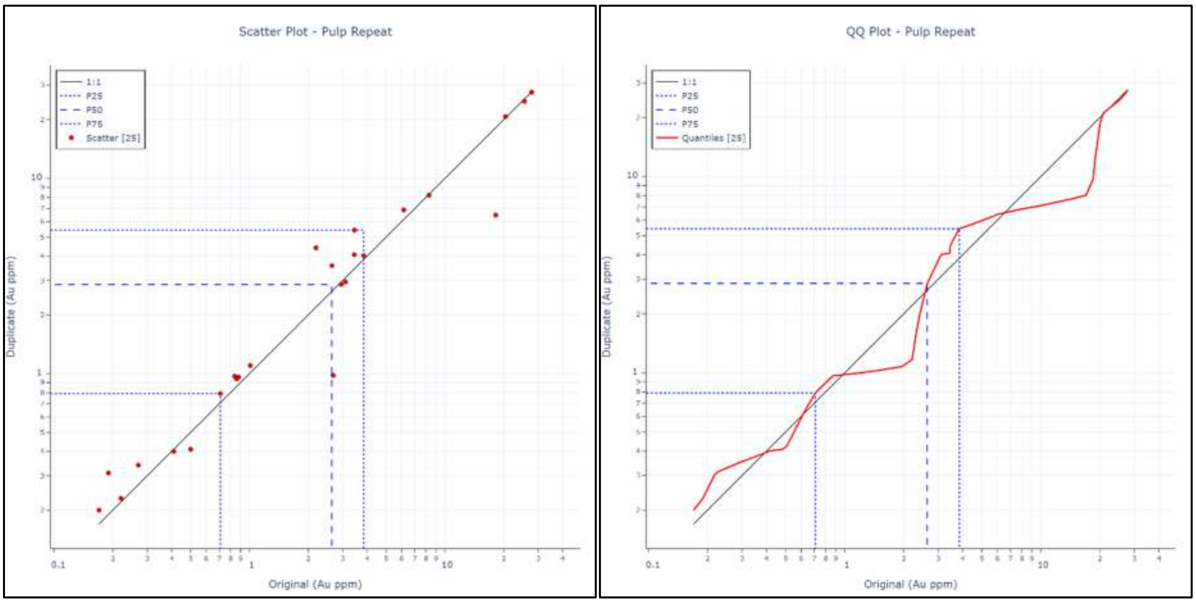


Figure 11-20: Scatter and QQ plots of second-split duplicate pairs from trench samples collected by RRPL and analysed by FAA303 for Au by SGS.

Table 11-13: Precision summary for trench field repeat sample pairs.

Analyte	Analytical Method	Company	Split	N pairs	LOQ	Wilcoxon p-Value	Wilcoxon Verdict	RMSCV (%)
Au	FAA303	RRPL	Second	25	0.1 g/t	0.339	Accept H_0	31

11.5.4.2.5 Analytical Process

A review of the CRM data inserted to determine the accuracy and precision on the analytical process is reported in Section 11.5.3.1.8.

11.6 Summary

Following a review of the available data quality and SOPs, the QP (Abraham Whaanga) considers that the logging/sampling, sample preparation, security, and analytical procedures are standard and adequate for assessing the quality of the procedures for the purpose of this PEA. The QP (Abraham Whaanga) considers he location, density, sample preparation, sampling, and analytical data to be fit for the purpose of defining moderate-confidence mineral resources. A summary of the data quality is presented in Table 11-14, where the process has been divided into the various sampling and preparation stages.

Table 11-14: Summary of data quality review for the Project. NA = not available.

Sample Type	Data Type	QA	QC	Accuracy	Precision	Fit for Purpose	Comment
Drill Sample	Collar Location	NA	Pass	Unknown	Unknown	Yes	No SOP. Collar locations professionally surveyed and compared to LiDAR. Data fit for purpose.
	Downhole Survey	NA	NA	Unknown	Unknown	Yes	No SOPs or quantitative control data were available. Process was standard; data fit for purpose.
	Density	Pass	NA	Unknown	Unknown	Yes	SOP available for review. No quantitative data recorded. Process was standard; data fit for purpose.

Sample Type	Data Type	QA	QC	Accuracy	Precision	Fit for Purpose	Comment
	Primary Sample	Pass	Pass	Pass	Pass	Yes	SOP available to review but did not include a minimum recovery threshold or guidance on dealing with low recoveries. Good drill recovery; data fit for purpose.
	First Split	Pass	Pass	Pass	Pass	Yes	SOP available for review. Insufficient data for analysis of the Project in isolation, but no bias observed between RRPL quarter-core pairs for the Reefton Project as a whole. Data fit for purpose. No quarter-core samples collected by RUA. The QP (Abraham Whaanga) recommends the collection of first-split duplicates.
	Second Split	NA	NA	Unknown	Unknown	Yes	No SOP available. Insufficient data for meaningful statistical analysis. Crushing process carried out using a standardised method at the preparation laboratory; data fit for purpose.
	Third Split	NA	Pass	Pass	Pass	Yes	No SOPs available for review. No bias observed between Au pulp repeat samples. Data fit for purpose. Insufficient Sb data for meaningful statistical analysis. Process was standard; data fit for purpose.
	Analytical Process: Au SGS	NA	Pass	Pass	Pass	Yes	No SOP for review. Minor low bias observed. The QP (Abraham Whaanga) recommends using matrix-matched CRMs. Data fit for purpose.
	Analytical Process: pXRF	Pass with issues	Pass with issues	Fail	Pass	Yes	CRMs report high compared to their certified values. Positive trend exhibited in OREAS 245 for Sb but not replicated for other elements. Due to the interference with Ca, it can be difficult to analyse Sb by pXRF. Data fit for purpose (moderate-confidence mineral resources). SOP fails to provide procedures regarding the use of CRM data. The QP (Abraham Whaanga) recommends conducting laboratory analysis on Sb-rich samples.
Trench Sample	Trench Location	NA	Pass	Unknown	Unknown	Yes	No SOP. Some trench locations were professionally surveyed. All data were compared to LiDAR. Data fit for purpose, but the QP (Abraham Whaanga) recommends professionally surveying remaining trench locations.
	Primary Sample	NA	Pass	Pass	Pass	Yes	No SOP. No bias observed between primary and repeat sample pairs. Data fit for purpose.

Sample Type	Data Type	QA	QC	Accuracy	Precision	Fit for Purpose	Comment
	First Split	NA	NA	Unknown	Unknown	Yes	No SOPs or quantitative control data were available. The QP (Abraham Whaanga) strongly recommends the collection of coarse crush duplicates. Process was standard; data fit for purpose.
	Second Split	NA	Pass	Pass	Pass	Yes	No SOP for review. No bias observed between pulp repeat samples. Data fit for purpose.
	Analytical Process: Au SGS	NA	Pass	Pass	Pass	Yes	No SOP for review. Minor low bias observed. The QP (Abraham Whaanga) recommends using matrix-matched CRMs. Data fit for purpose.

12 Data Verification

The QP (Abraham Whaanga) confirms that RUA has not conducted any work since the January 2026 site visit that is considered to be material to the MRE reported in Section 14, as of the effective date of this Report. To verify this, the QP (Abraham Whaanga) regularly discussed the Project's progress with RUA during this period, and monitored any public announcements made by RUA.

The QPs (Marius Phillips and Timothy Rowles) confirm that RUA has not conducted any work since the PBG and KPPL site visit on 12 March 2026 that is considered to be material to the metallurgical testing discussed in Section 13, recovery methods proposed in Section 17, and infrastructure proposed in Section 18.

The QP (Gary Davison) confirms that RUA has not conducted any work since the February 2026 site visit that is considered to be material to the mining methods proposed in Section 16, capital and operating costs discussed in Section 21, and economic analysis covered in Section 22.

12.1 Drillhole Database

The Project now has a relational Microsoft Access database containing all data underpinning the MRE in Section 14. Historical data from across the wider Reefton Project will also be validated and imported into this database for ongoing use by RUA.

The QP (Abraham Whaanga), or RSC staff under the QP (Abraham Whaanga)'s supervision, has independently verified the RRPL and RUA drilling, soil sample, and rock-chip databases, and supporting records, logs, and photographs.

Prior to May 2026, all exploration data was stored in MS Excel workbooks. The workbooks contain data from both trenches and drilling undertaken by RUA, RRPL, MMCL, and OGL. Each workbook was split into the following tabs.

- Collar: Hole_ID, location, depth planned orientation, operator, and drilling company;
- Survey: downhole depth and orientation, survey tool, and quality comments;
- Geology: depth by lithology, validated lithology codes, weathering, orientate core measurements, structure field, and alteration;
- Recovery: run interval, total core recovered, and recovery; and
- Assay: lab_ID, date, Sample_ID, sample type, depth, QC note, analytical data including Au by fire assay, screen fire assay, and multi-element pXRF.

The QP (Abraham Whaanga) independently verified the RRPL drilling workbooks and supporting records, logs, and photographs. Each RRPL drillhole was logged in an MS Excel workbook via a laptop. The drill log had the required data fields to provide the information necessary to support an MRE.

The QP (Abraham Whaanga) verified a representative number of collars, sampling, lithology logs, and assay data (Au, As, and Sb) against the digital database. An appropriate number of datapoints were checked, including selected zones of significant mineralisation. A small number of transcription errors, missing data, and incorrect values in the recovery data and geotechnical logs were identified during the verification process and quickly fixed by RRPL. No transcription errors were identified in the newer data provided by RUA.

12.2 Collar Locations

During the August 2024 site visit, RSC visited drill pads at Auld Creek. Due to consent restrictions, numerous drillholes had been drilled from one pad at the Auld Creek site. Due to the size of the drill pad and GPS error (likely to be higher due to steep slopes and bush cover), only one GPS check point was collected on each pad. RSC checked 16 collars at Auld Creek. The largest error was noted where there was an ~20-m difference between the GPS check and the coordinates in the database. RUA has since resurveyed the collars around the drill pad using DGPS.

RSC also recorded trench locations Auld Creek during the August 2024 site visit and observed exposed veins. Due to the difficulty in obtaining accurate surveyed GPS Z-values for trenches (due to steep slopes and bush cover), the values had been adjusted by RRPL by draping them onto the LiDAR surface.

The QP (Abraham Whaanga) visited a drill rig at Auld Creek and trench collar locations at Fraternal during the January 2025 site visit.

The QP (Abraham Whaanga) reviewed the collar verification work completed by RSC in previous site visits and determined that the process for the location of drillhole and trench collars poses a low risk to the classification of resources in the intended categories.

12.3 Sampling Verification

The QP (Abraham Whaanga) conducted a site visit to the Project for sampling verification from 19–21 January 2026 and checked a representative number of database entries against the core retained on site (Figure 12-1 and Figure 12-2). During previous site visits, RSC also verified a representative number of database entries against the core retained on site. Full details are provided in Aldrich (2024), Aldrich and Whaanga (2024), and (Whaanga, 2026).

Figure 12-1: Example of core tray samples selected by the QP (Abraham Whaanga) for verification sampling (ACDDH050: Box 61 and Box 62).

Figure 12-2: Core logging verification conducted by the QP (Abraham Whaanga).

Intervals for check logging and check sampling were identified during the site visit. The QP (Abraham Whaanga) checked drill core recoveries and run depth and noted no issues. Zones of core loss or potential cave had been identified by RUA during drilling and were clearly marked on the blocks. Drilling reference lines were clear, as were the orientation lines, and the level of confidence when tracing the orientation line between drilling intervals was marked on the core. Bedding and cleavage were noted and marked on the core prior to logging. Metre/sample intervals were clearly marked on the sides of the core tray inners; these were checked by the QP (Abraham Whaanga) and confirmed to be accurate. The core blocks were observed to be consistent with the recoveries noted by the drill plods and database. The QP (Abraham Whaanga) also reviewed drill logs of the selected intervals and found no issues.

The QP (Abraham Whaanga) discussed sampling and cutting procedures with site personnel, who confirmed that half-core samples were sent for assay. Prior to December 2025, RUA sampled with a minimal sampling interval of one (1) metre. The QP (Abraham Whaanga) notes that the new minimum sampling interval is 0.3 m, to capture the variation in Sb and Au grade. The logging geologist marked a cut line for zones identified as being potentially mineralised; outside of these zones, the cut was determined by the field technician operating the diamond saw. Orientation lines were preserved for future examinations. However, several of the samples checked during the site visit exhibited significant deviation between the cut line and the orientation line, and the variation was inconsistent across the samples. The QP (Abraham Whaanga) notes that this issue was identified on a previous site visit (Aldrich and Whaanga, 2024). Marking and cutting core along the orientation line (or a few degrees off it to preserve the line), in line with industry best practice, to ensure no sampling bias is introduced, is recommended.

Overall, in the QP (Abraham Whaanga)'s opinion, the recent drilling data on which the updated mineral resources are based are verified and fit for purpose.

12.4 Core & Pulp Check Sample Analysis

12.4.1 August 2024

RSC collected a representative number of check samples from Auld Creek to verify mineralisation and grade tenure (Figure 12-3). The check samples consisted of quarter-core samples and pulp repeat samples.

The verification samples for check analysis were selected based on geology. A selection of drillholes was pre-selected prior to the site visit to ensure they could be located and placed on the racks before RSC arrived on site.

Figure 12-3: Quarter-core verification sampling conducted by RSC in August 2024. A) Quarter-core sample and corresponding pulp collected from AXDDH055. B) Quarter-core sample and corresponding pulp collected from BRDDH027. C) Quarter-core samples and corresponding pulps collected from ACDDH005. D) Quarter-core samples collected from 97RDD020. E) Close-up of the check sample labelling.

RSC reviewed the lithological logging and sampled intervals logged as quartz veins, mineralised greywacke, mineralised argillite, and breccia, as these are associated with the mineralisation. A total of 30 quarter-core samples were selected.

While half-core is preferred, RRPL preferred not to lose entire intervals of the core. This is understandable given the development stage of the Project, and the limited amount of mineralised core available. Quarter-core sampling (regardless of the volume variance effect) allows check sampling to be conducted while also retaining core for archive purposes.

Where possible, corresponding pulp samples were also selected. However, where pulps could not be located or were too small, additional mineralised pulps were selected based on geological logging to ensure a representative number of pulp check samples were analysed.

A total of 25 CRMs were included in the submission, which also included samples from elsewhere in the Reefton Project. Samples were prepped at SGS Westport and then sent to ALS Brisbane for Au analysis (fire assay Au-AA26 or screen fire assay Au-SCR24) and Sb analysis (ME-XRF15c).

Based on a Wilcoxon signed-rank test, the quarter-core samples do not exhibit a statistically significant bias at Auld Creek for Au or for Sb (Table 12-1). A review of the data (scatter and QQ plots) for the Project (i.e. check sample data) indicates there is a statistically significant bias towards the original sample, in the order of 3% (Figure 12-4).

Table 12-1: Precision summary table for half-core check samples.

Split Type	Prospect	Analyte	No. pairs	LOQ	Wilcoxon p-Value	Wilcoxon (p95)
Quarter-Core	Auld Creek	Au	30	0.1 ppm	0.109	Accept H_0
Quarter-Core	Auld Creek	Sb	18	0.05%	0.799	Accept H_0

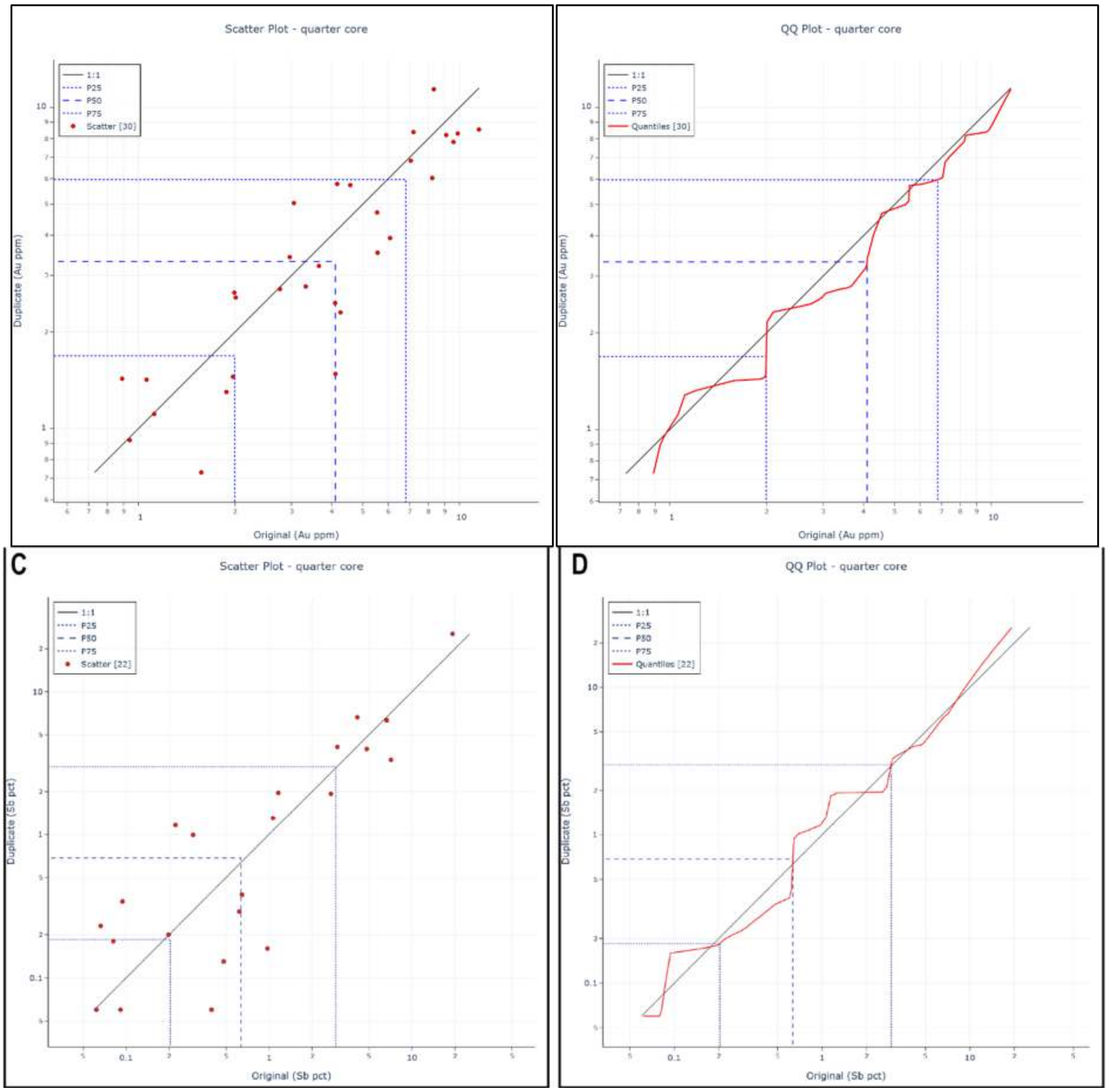


Figure 12-4: Scatter and QQ plots comparing quarter-core check sample pairs for A–B) Au and C–D) Sb.

The pulp samples analysed for Au also exhibit a statistically significant but small bias, as indicated by the Wilcoxon signed-rank test, towards the original sample, in the order of 4% (Table 12-2). No statistically significant bias at a 95% confidence level is observed in the Sb pulp samples collected from Auld Creek, as indicated by a Wilcoxon signed-rank test (Table 12-2, Figure 12-5).

Table 12-2: Precision summary table for pulp check samples.

Split Type	Prospect	Analyte	No. pairs	LOQ	Wilcoxon p-Value	Wilcoxon (p95)
Pulp	Auld Creek	Au	30	0.1 ppm	0.000	Reject H_0
Pulp	Auld Creek	Sb	19	0.05%	0.891	Accept H_0

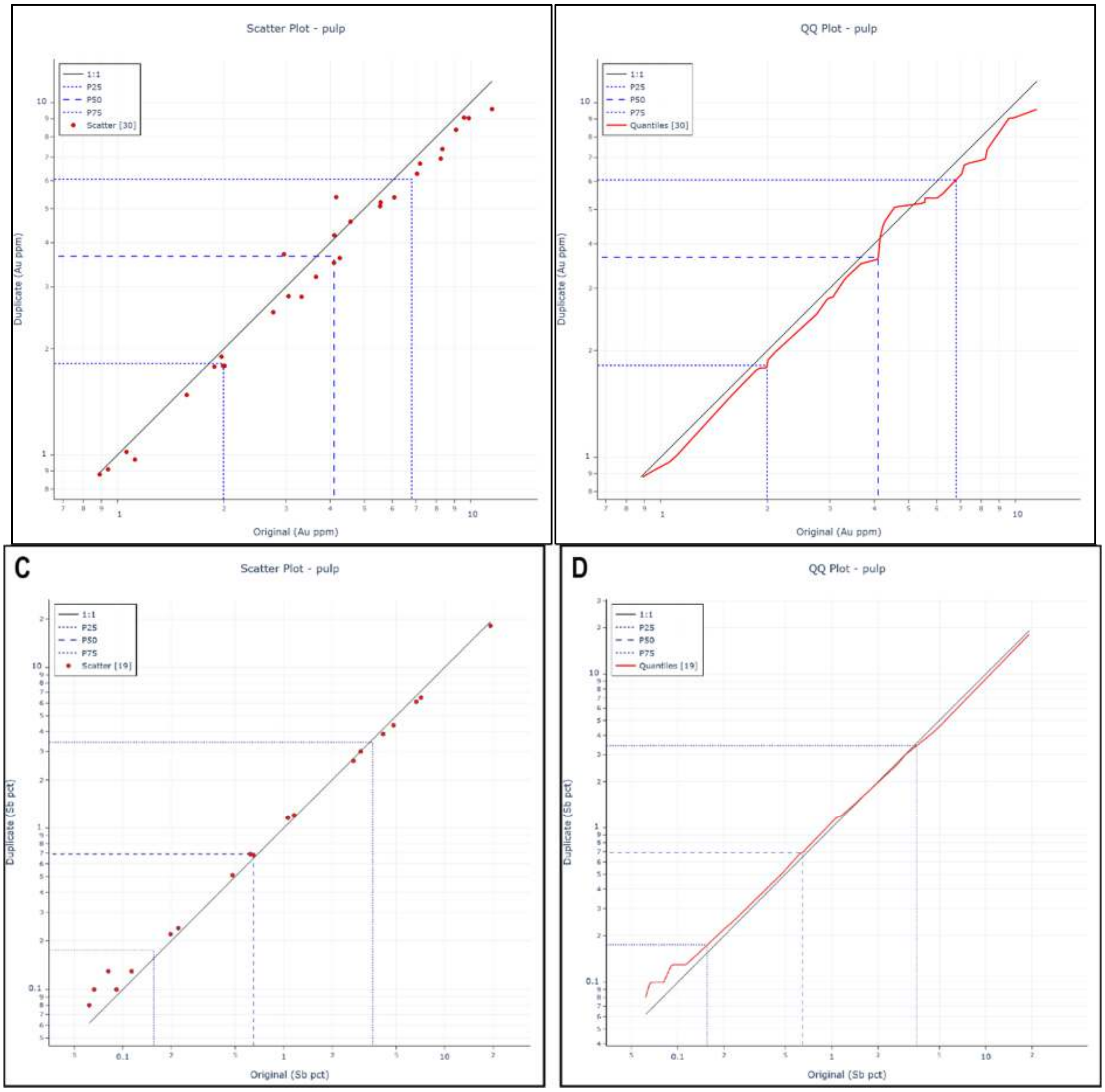


Figure 12-5: Scatter and QQ plots comparing the pulp check sample pairs for A–B) Au, and C–D) Sb.

Following a review of the CRMs inserted alongside the check samples and as part of RRPL's QC procedures, it was determined that the analytical process was in control and the bias was not analytical in nature. It is possible the bias was introduced during the sub-sampling steps; however, this is inconclusive, and RSC recommends RUA undertakes further investigations to identify the source of the bias.

RSC notes that pulp samples that were selected by RSC for verification had been in storage for more than 2 years. Over time, dense Au particles settle, thereby introducing heterogeneity in the pulp sample (Dominy et al., 2000), which can be exacerbated by vibrations and motion during sample transport. Once at the laboratory, laboratory technicians may have incorrectly taken a scoop from the top of the bag for analysis; thus, with the Au settling at the bottom of the bag, the verification sample may be biased towards anomalously low values. Oxidation of sulphides over time may also create biases. RSC recommends further pulp resubmissions to investigate any likely analytical and preparation bias.

12.4.2 January 2026

During the January 2026 site visit, the QP (Abraham Whaanga) collected check samples from the Project to verify mineralisation and grade tenure. The check samples consisted of quarter-core samples collected by RUA.

The verification samples for check analysis were selected by the QP (Abraham Whaanga) based on intervals of known mineralisation. The drillholes were selected on site by the QP (Abraham Whaanga), in consultation with RUA geologists, to have a significant impact on the Auld Creek MRE (Figure 12-6).

The QP (Abraham Whaanga) reviewed the lithological logging, and sampled intervals logged as quartz veins, mineralised greywacke, mineralised argillite, and breccia, as these are associated with the mineralisation. A total of 13 samples were selected across the Project (Table 12-3).

Figure 12-6: Quarter-core verification sampling conducted by RSC in January 2026: half-core verification samples prior to cutting (left); quarter-core check samples relabelled (right).

Table 12-3: Original and check assays from the January 2026 site visit.

Prospect	HoleID	From (m)	To (m)	Interval (m)	Original Au Assay (ppm)	Sample Type	Check Au Assay (ppm)	Method
Auld Creek	ACDDH050	139	140	1	5.32	Core	6.01	FAA505
Auld Creek	ACDDH050	144	145	1	0.02	Core	0.03	FAA505
Auld Creek	ACDDH050	147	148	1	0.04	Core	0.05	FAA505
Auld Creek	ACDDH050	155	156	1	0.02	Core	0.03	FAA505
Auld Creek	ACDDH056	241.4	241.8	0.4	7.89	Core	7.63	FAA505
Auld Creek	ACDDH056	242.5	242.9	0.4	6.79	Core	6.1	FAA505
Auld Creek	ACDDH056	242.9	243.3	0.4	10.4	Core	7.47	FAA505
Auld Creek	ACDDH056	247	248	1	0.01	Core	0.04	FAA505
Auld Creek	ACDDH058	168.8	169.8	1	0.39	Core	0.44	FAA505
Auld Creek	ACDDH058	171.7	172.8	1.1	0.36	Core	0.27	FAA505

Prospect	HoleID	From (m)	To (m)	Interval (m)	Original Au Assay (ppm)	Sample Type	Check Au Assay (ppm)	Method
Auld Creek	ACDDH058	173.5	174	0.5	20.5	Core	4.14	FAA505
Auld Creek	ACDDH058	174.2	174.9	0.7	12.6	Core	14.7	FAA505
Auld Creek	ACDDH058	175.3	175.8	0.5	0.38	Core	0.26	FAA505

One outlier check sample (ACDDH058 173.5–174 m) returned a grade of 4.14 g/t Au compared with the original sample of 20.5 g/t Au. The QP (Abraham Whaanga) has discussed the results with RUA and compared them with core photographs and is satisfied that the discrepancy is not indicative of a significant issue with sampling. The QP (Abraham Whaanga) is of the opinion that the orientation of the mineralisation relative to the quarter-core taken for the check sample is the primary reason for the discrepancy between the two values (Figure 12-7). As stated in Section 12.4.1, half-core sampling is preferred over quarter-core sampling, and half-core sampling would reduce the volume variance effect; however, the QP (Abraham Whaanga) notes that RUA has elected to undertake quarter-coring to retain some sample for archive.

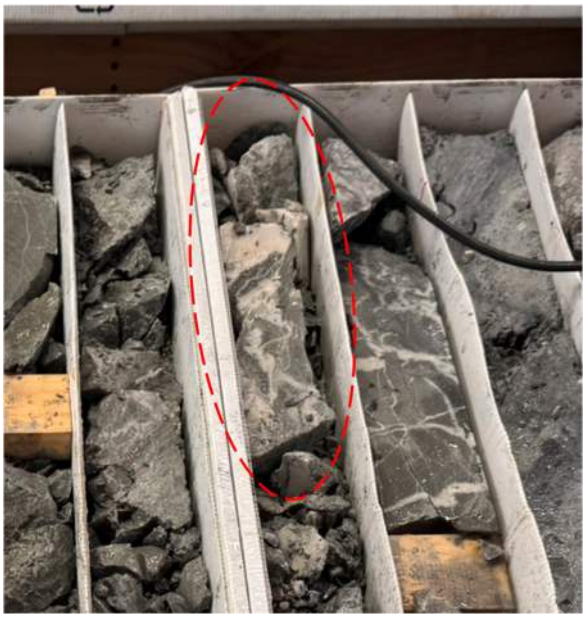


Figure 12-7: Variable mineralisation orientation for check sample ACDDH058 173.5–174 m prior to quarter-core sampling.

12.5 Summary

The drillhole data collected by RRPL were comprehensive and semi-validated at the point of collection through a process of quality assurance and continual quality control. Verification checks completed by the QP (Abraham Whaanga) uncovered several minor errors in the MS Excel workbooks provided by RRPL, which were corrected. The QP (Abraham Whaanga) recommends that all data for the Reefton Project be transferred from the MS Excel workbooks into the relational MS Access database for the Project.

RSC collected a mix of half-core and pulp samples. The Au samples demonstrate a bias towards the original sample that is likely to be in the order of 4%, which is immaterial with respect to the DQO.

Overall, in the opinion of the QP (Abraham Whaanga), the data on which the mineral resources are based are verified and fit for the purpose of classifying Indicated Mineral Resources. The QP (Abraham

Whaanga) recommends RUA conducts further investigations into the check sample bias to ensure that future data are fit for purpose for higher-confidence classifications.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

The metallurgical response of the Auld Creek Au-Sb ore has been assessed via the completion of metallurgical test work on three representative composite samples. Twenty-five crushed core samples were combined to form the three composite samples which represent three grade ranges as summarised in Table 13-1.

Table 13-1: Core sample selection for metallurgical testing.

Sample	Drillhole	From (m)	To (m)	Au (g/t)	Sb (%)	As (%)	Mass (kg)	Comment
AC001	ACDDH005	59.39	63.27	3.0	0.08	0.31	27.3	Gold, no stibnite
		75.07	76.37					
AC002	ACDDH004	126.35	131.81	5.9	5.80	0.48	26.6	High gold, high stibnite
AC003	ACDDH004	120.10	125.14	4.1	1.35	0.38	27.0	High gold, lower stibnite

The test work was aimed at defining the physical characteristics of the Auld Creek ore while simultaneously generating an understanding of the mineralogical composition and resultant metallurgical response. Both gravity and flotation recovery methodologies were assessed. Valuable mineral deportment was investigated by size-by-size analysis and multi-stage diagnostic Au leach testing.

13.2 Mineralogy

A sub-sample of composite AC002 was subjected to a rougher flotation test and the resulting concentrate was subjected to mineralogical analysis via QEMSCAN® to assess stibnite deportment, liberation and association.

Bulk mineralogy on the flotation concentrate sample indicated that the flotation concentrate was comprised predominantly of stibnite (50.8%), arsenopyrite (20.8%), and pyrite (16.2%). Non-sulphide gangue minerals comprised quartz (7%) and muscovite (3%).

As depicted in Figure 13-1, many of the other sulphide and gangue particles appear to have small grains of stibnite attached to them. These stibnite grains appear to have smeared on the mineral surface during the milling process and could adversely impact the differential flotation approach by allowing other sulphides and non-sulphide gangue to misreport to the stibnite concentrate.

Elemental deportment confirms that Sb is present as stibnite and As deports as arsenopyrite. Sulphur is contained within stibnite (53%), arsenopyrite (15%), and pyrite (32%).

The cumulative passing size of the key minerals confirms that sample has a combined P_{80} of 64 µm; however, stibnite had a P_{80} of 33 µm, suggesting a degree of preferential breakage has occurred, which in turn has contributed to overgrinding of the stibnite. Arsenopyrite and pyrite returned a P_{80} of 73 µm and 72 µm, respectively (Figure 13-2).

With respect to Au mineralogy, only a single Au grain was identified, which was fully enclosed within arsenopyrite and was ultrafine-grained at <2 µm. No Ag was detected.

Laser ablation was also conducted to determine whether the stibnite was Au-bearing (aurostibnite) and to confirm the hypothesis that Au is contained within the sulphides and consequently Au-stibnite separation should be possible. A polished section of the sample was submitted to University of Tasmania for laser ablation testing. This assessment confirmed that the Au content of the stibnite is low (0.114 g/t ± 0.024 ppm) and stibnite therefore accounts for <0.1% of the sample Au content.

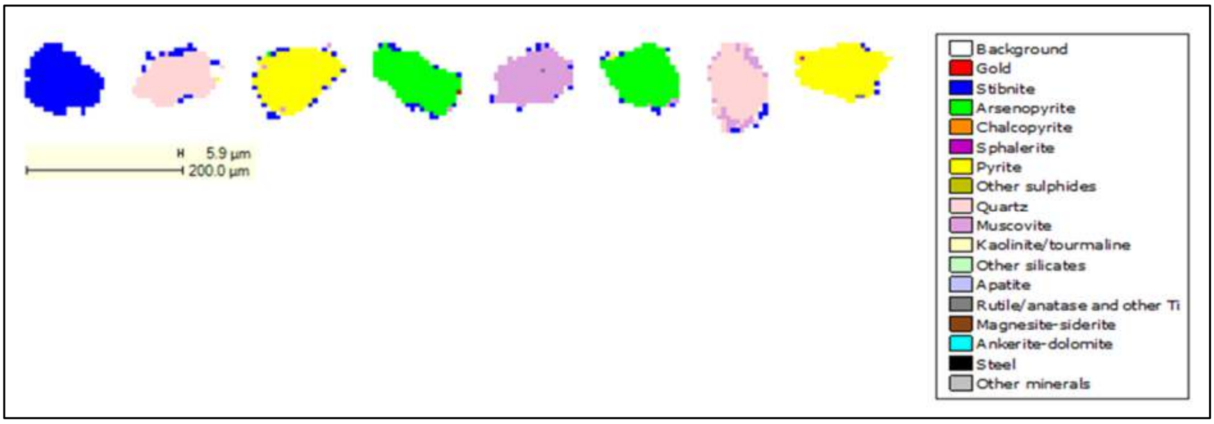


Figure 13-1: Stibnite association with sulphides/gangue.

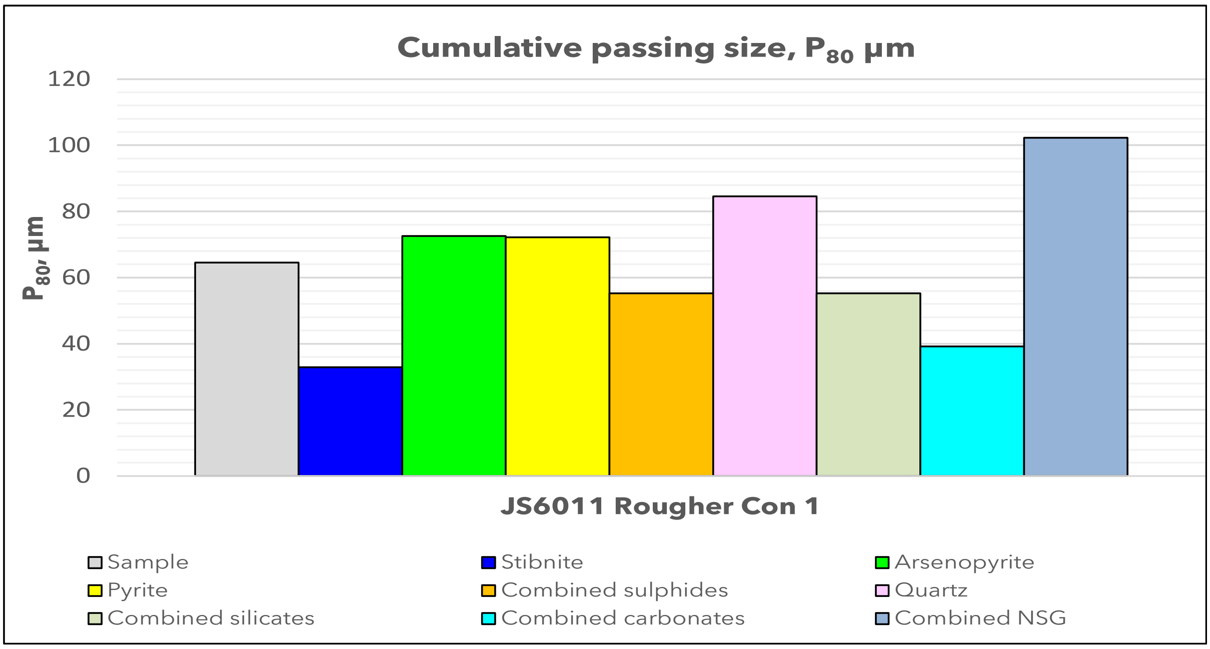


Figure 13-2: Key mineral particle size.

13.3 Comminution Test Work

13.3.1 Bond Ball Mill Work Index

Each of the three composite samples was subjected to a standard Bond Ball Mill Work Index (BBWi) determination. The BBWi test work was completed using a 150 µm closing screen. The BBWi classifies the Auld Creek ore as moderately hard, with an average BBWi 15.3 kWh/t. Results from the BBWi are summarised in Table 13-2.

Table 13-2: Bond ball mill work index results.

Sample	F_{80} (µm)	P_{80} (µm)	Gbp (g/rev)	Closing Screen (µm)	Bond BWi (kWh/t)
AC001	1,989	111	1.427	150	16.0
AC002	2,032	114	1.525		15.3
AC003	2,000	117	1.648		14.6

13.4 Head Characterisation

13.4.1 Head Assays

A multi-element inductively coupled plasma mass spectrometry (ICP-MS) analysis was conducted on each of the individual sample composites to generate an understanding of the chemical composition of the samples. The Au and Ag content of the individual samples was determined by fire assay. The head analysis results for the individual samples are presented in Table 13-3. A true SG determination was also completed for each composite sample.

Table 13-3: Composite sample head assays.

Element	Unit	AC001	AC002	AC003
Gold (Au)	g/t	3.40 / 3.26	6.15 / 6.05	4.49 / 4.45
Silver (Ag)	g/t	<0.3	<0.3	<0.3
Arsenic (As)	wt. ppm	3,531	4,902	5,368
Copper (Cu)	wt. ppm	16	24	20
Iron (Fe)	wt. %	3.30	2.66	3.00
Sulphur (S) – Total	wt. %	0.64	3.04	1.34
Sulphur (S) – Sulphide	wt. %	0.56	0.94 / 0.76	0.74 / 0.76
Antimony (Sb)	wt. %	0.07	5.29	1.34
Silica (SiO_2)	wt. %	67.9	68.0	68.1
Tellurium (Te)	wt. ppm	<0.2	<0.2	<0.2
Zinc (Zn)	wt. ppm	56	54	44
Specific Gravity		2.86	2.91	2.86

Pertinent observations from the head analysis determination include:

- Gold head assay confirmed the drill core results with a close correlation between the core sample and composite sample Au assays. Very little variance in Au head assay is evident, suggesting no free Au or 'spotty' Au influence on head assay. Visible Au was not seen in drill core logging.
- Silver assays are low and thus Ag is unlikely to present any economic value.
- Sulphur assays are variable and range from 0.64–3.04%. These variable S grades are likely to result in a variable flotation response. The high S grades are likely to adversely affect flotation performance with respect to achievable concentrate grade. Cleaner test work will need to consider processes to minimise S deportment to final concentrate. The differential between sulphide S and total S suggests some oxidation has occurred or other S-bearing species are present. Future work will be needed to define this difference.

- Arsenic head assays are high, indicating the presence of arsenopyrite. Arsenic deportment to concentrate will require monitoring, given As is a penalty element which incurs significant smelter penalty charges. The presence of arsenopyrite may also be indicative of the presence of refractory Au and future work will need to define the Au-arsenopyrite association.
- Base metal assays (Pb, Zn) are low, and these elements are not anticipated to report to the concentrate phase in concentrations that will incur penalty charges. Likewise, should a flowsheet base on Au leach be considered, these low concentrations are unlikely to adversely impact cyanide consumption.
- The Sb (stibnite) grade of composite samples AC002 and AC003 is very high. These samples will likely produce saleable Sb grade concentrates. Within an Au leach circuit, stibnite can solubilise at high pH and form a passivating oxide layer on the Au surface, retarding leach performance. This surface passivation effect will need to be tested in future programs and can be mitigated by operating at a lower leach pH (but with potential consequences for cyanide gas evolution).

13.4.2 Size-by-Size Head Assays

Each of the metallurgical composite samples were ground to a P_{80} of 106 µm using a laboratory rod mill and then subjected to a particle size distribution determination via sieve sizing, with screen sizes ranging from 150 µm to 25 µm. Each of the screen size fractions were then analysed for Au, Sb, As and S to determine the size-by-assay distribution of the samples.

The size-by-assay distribution for each sample is presented in Figure 13-3 (AC001), Figure 13-4 (AC002), and Figure 13-5 (AC003). The size-by-size distribution consistently indicates a correlation between the Au and As, consistent with the hypothesis that Au is predominantly associated with arsenopyrite. Gold also appears to be uniformly distributed across the tested size fractions below -75 µm.

The Sb minerals appear to be concentrated within the fine fractions (-25 µm), suggesting potential overgrinding has occurred. This is not an unexpected outcome as stibnite is a friable/brittle mineral which is prone to overgrinding and fine slimes production due to preferential breakage, all of which must be mitigated in industrial-scale grinding applications.

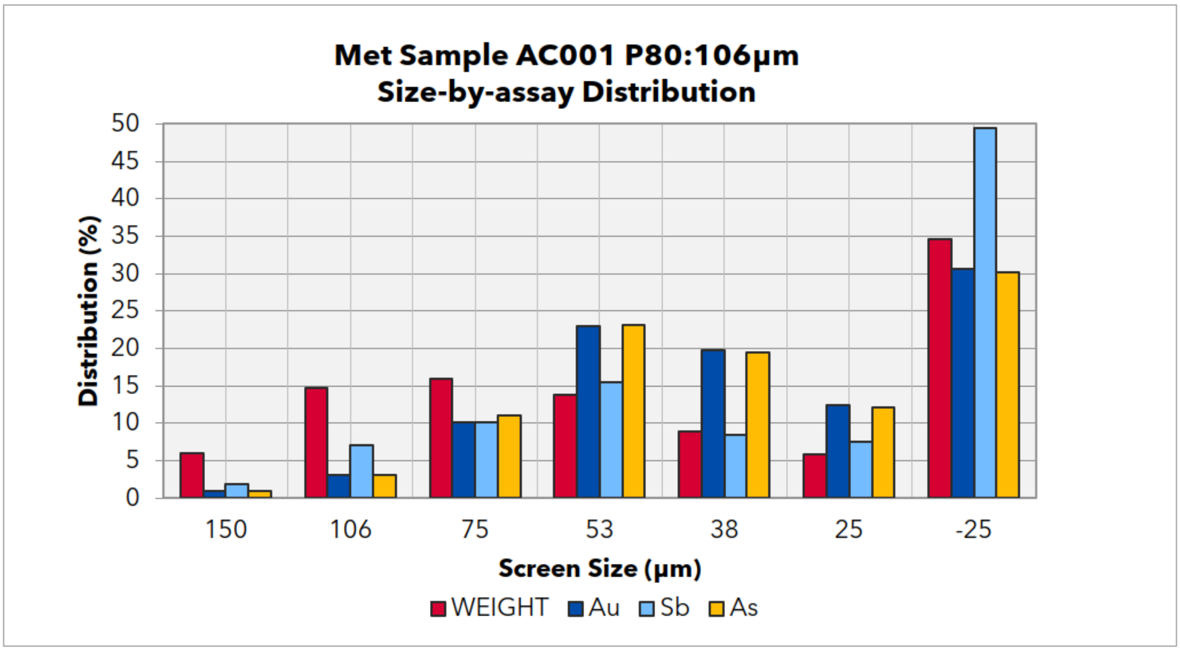


Figure 13-3: AC001 size-by-assay distribution.

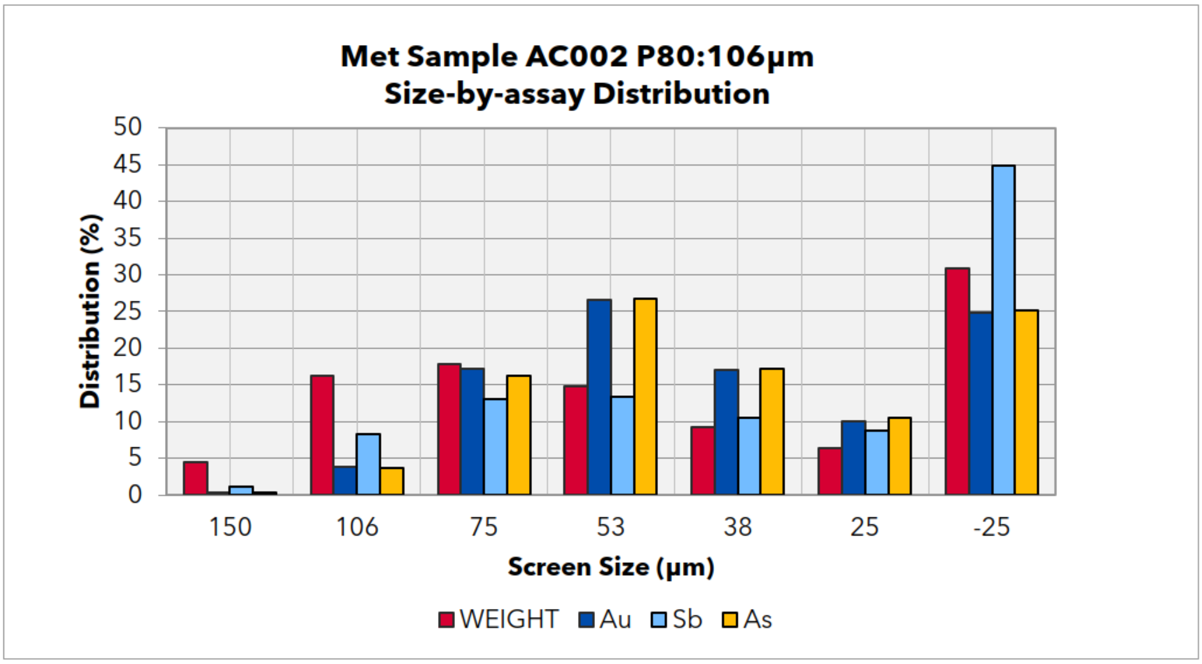


Figure 13-4: AC002 size-by-assay distribution.

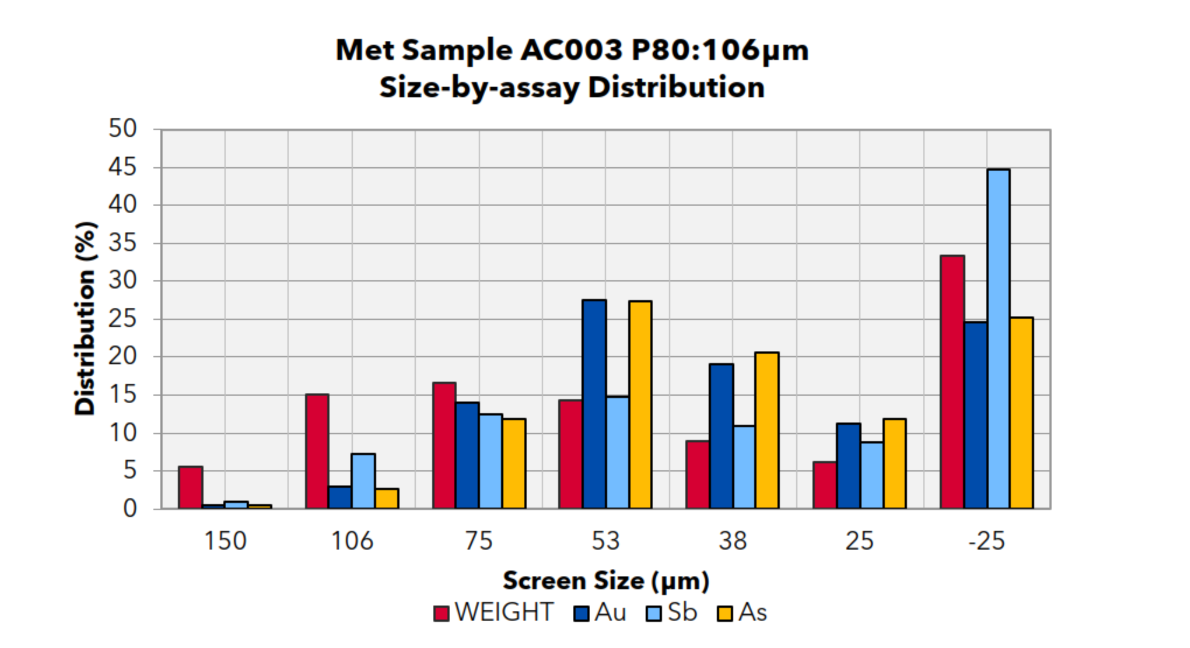


Figure 13-5: AC003 size-by-assay distribution.

13.5 Gravity Recovery Test Work

The samples were subjected to gravity recovery using a 3” laboratory Knelson concentrator to assess whether the Auld Creek samples contain gravity recoverable gold (GRG). Gold-bearing gravity concentrate from the Knelson concentrator was then subjected to mercury (Hg) amalgamation to clean the gravity concentrate. Amalgamation is adopted to obtain a mass yield to concentrate which more closely approximates the mass yield in practice, and thus the gravity Au recovery achievable by industrial-scale gravity operations. Results from the gravity recovery test work are provided in Table 13-4.

Table 13-4: Gravity recovery results.

Product	Sample AC001			Sample AC002			Sample AC003		
	Mass (%)	Au (g/t)	Au Rec. (%)	Mass (%)	Au (g/t)	Au Rec. (%)	Mass (%)	Au (g/t)	Au Rec. (%)
Gravity/Amalgam	-	0.05	1.36	-	0.01	0.17	-	0.01	0.22
Amalgam Tailings	6.35	29.0	55.5	7.26	45.5	54.3	6.52	40.8	58.0
Knelson Tailings	93.7	1.53	43.2	92.7	2.98	45.5	93.5	2.05	41.8
Calculated Head	**100**	**3.32**	**100**	**100**	**6.08**	**100**	**100**	**4.59**	**100**
Assay Head	**-**	**3.33**	**-**	**-**	**6.10**	**-**	**-**	**4.47**	**-**

Although all three of the samples tested returned a high GRG content, a very low Au recovery was achieved when these samples were subjected to Hg amalgamation. This suggest the recovered Au is either very fine grained or unliberated. The QP (Marius Phillips) recommends that bespoke GRG test work be conducted in future test work programs.

13.6 Flotation Recovery Test Work

13.6.1 Baseline Rougher Flotation

Baseline rougher flotation tests were conducted to assess the flotation response of the composite samples. The samples (with P_{80} of 106 µm) were diluted to achieve a flotation pulp density of 35 wt.% solids. The slurry was conditioned with copper sulphate (10 g/t) and a potassium amyl xanthate (PAX) collector (50 g/t) and flotation was conducted at natural pH of 7.6. Four rougher concentrates were recovered over a total flotation time of 9 minutes. Staged addition of PAX (25 g/t) and W24 frother (20 g/t) was used.

With respect to Au recovery, all three samples responded very well to rougher flotation and yielded an overall Au recovery to concentrate of 95.8–98.3% (Table 13-5), with a corresponding concentrate Au content of 51.2–62.1 g/t. Mass pull to concentrate was variable, ranging from 5.9–11.4%, with the high-S sample (AC002) yielding the highest mass pull, implying that a correlation exists between mass pull and sample S grade.

Table 13-5: Baseline rougher flotation test results.

Sample	Test No.	Composite of Rougher Concentrate 1–4								
		Mass Yield (%)	Au		As		Sb		S_{Tot}	
			Grade (g/t)	Rec. %	Grade (wt. %)	Rec. %	Grade (wt. %)	Rec. %	Grade (wt. %)	Rec. %
AC001	JS6010	5.90	51.2	95.8	6.16	95.1	0.69	89.7	9.7	92.4
AC002	JS6011	11.40	60.4	98.3	5.23	98.5	32.4	64.6	19.9	73.0
AC003	JS6012	7.42	62.1	97.8	6.40	96.3	13.0	71.3	15.2	85.3

Rougher flotation kinetics was fast with 65–74% of the Au recovered after 1 minute of flotation and flotation essentially complete after 7 minutes (Figure 13-6). Likewise, As recovery consistently exceeded 95%, with a distinct correlation existing between Au recovery and As recovery.

Sample AC001 showed excellent selectivity for sulphides against non-sulphide gangue and achieved very high S recoveries into a low mass concentrate. Conversely, samples AC002 and AC003 returned lower S recoveries into a higher mass yield concentrate.

Samples AC002 and AC003 yielded an inferior Sb recovery response to sample AC001, so additional test work was implemented to optimise the flotation response of these two samples.

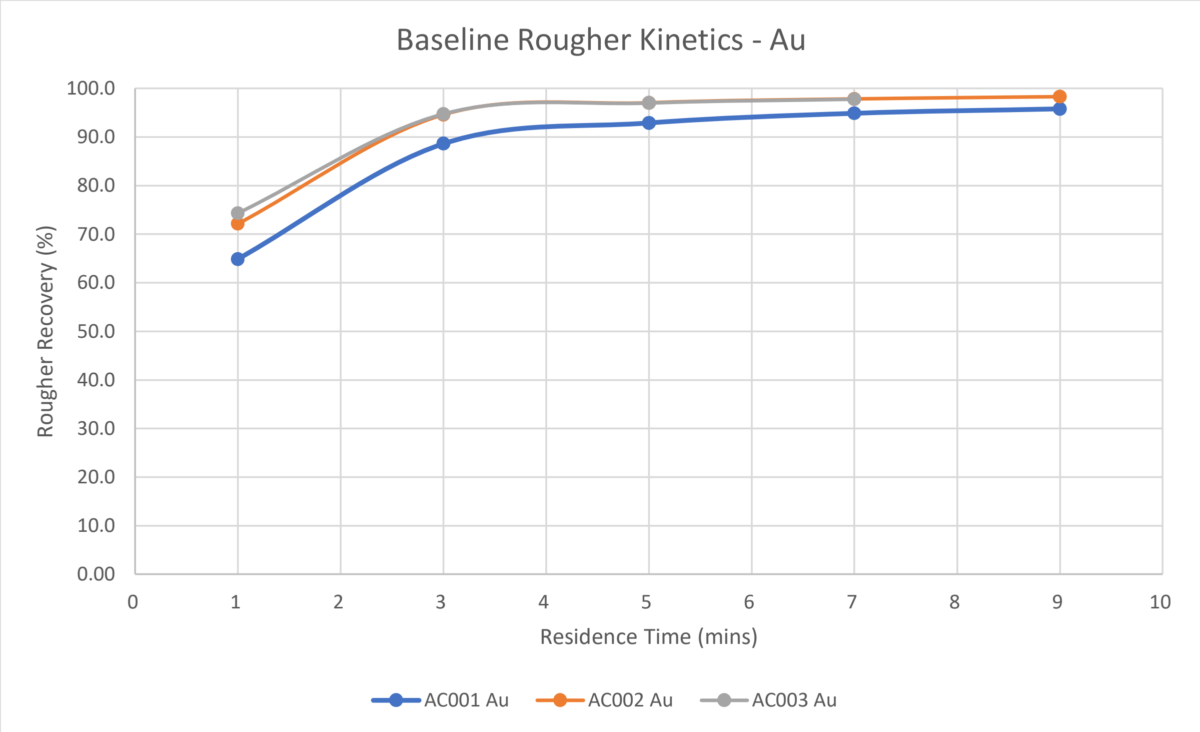


Figure 13-6: AC001 baseline rougher flotation kinetics.

13.6.2 Optimisation Rougher Flotation

Rougher flotation optimisation test work was completed following the poor baseline rougher flotation results achieved by AC002 and AC003. This test work focused on the impact of grind size (liberation or over grinding), pH effects and the impact of alternate activators.

Results from the rougher optimisation test work are summarised in Table 13-6.

The rougher optimisation tests are compared with the baseline flotation results below.

For sample AC002:

- Halving the PAX addition and not adding any copper sulphate to the flotation circuit resulted in a marginal reduction in Au and As recovery (2% and 3%, respectively), and a more pronounced recovery reduction for Sb and S (25% and 16%, respectively).
- Increasing the slurry pH to 9 resulted in a similar response, reducing Au and As recovery by 2% and 4%, respectively, while Sb and S recovery was reduced by 30% and 24%, respectively.

Table 13-6: Rougher optimisation flotation test results.

Sample	Test No. and Conditions	Composite of Rougher Concentrate 1–4								
		Mass Yield (%)	Au		As		Sb		S_{Tot}	
			Grade (g/t)	Rec. %	Grade (wt. %)	Rec. %	Grade (wt. %)	Rec. %	Grade (wt. %)	Rec. %
AC002	BF2720 (No $CuSO_4$; Half PAX)	7.24	85.3	96.0	7.59	95.2	28.9	39.4	21.8	54.3
	BF2721 (pH 9 with Lime)	7.24	78.4	95.9	7.05	94.8	27.6	34.4	20.9	49.1
AC003	BF2722 (P80 150µm)	5.49	82.2	96.4	8.57	94.3	11.2	47.5	17.8	74.8
	BF2723 (Lead nitrate Activator)	9.84	44.8	97.6	4.60	96.2	13.2	**97.3**	12.7	96.5

For sample AC003:

- Coarsening the grind size from a P_{80} of 106 µm to a P_{80} of 150 µm resulted in a marginal reduction in Au and As recovery of 1% and 2%, respectively, and a more pronounced recovery production for Sb and S of 24% and 11%, respectively.
- Replacing the copper sulphate activator with a lead nitrate activator resulted in a negligible difference in Au and As recovery; however, recovery for Sb and S improved substantially, by 26% and 11%, respectively.

The QP (Marius Phillips) recommends that future test work programs explore whether the use of a coarser grind and lead nitrate activator would offer any further improvement in rougher flotation performance.

13.7 Cleaner Flotation Test Work

Sighter cleaner flotation test work was conducted using composite sample AC001 (containing negligible stibnite) to assess the impact of regrind on cleaner performance and Au recovery. Two tests were conducted: the first testing cleaner performance on the 'as received' rougher concentrate sample, and the second testing the impact of a 45 µm target regrind size.

Results from the cleaner test work are summarised in Table 13-7.

Table 13-7: AC001 cleaner flotation test results.

Sample	Product	Concentrates								
		Mass Yield (%)	Au		As		Sb		S_{Tot}	
			g/t	Rec. %	%	Rec. %	%	Rec. %	%	Rec. %
No Regrind	Cleaner Conc. 1–4	2.61	120.6	91.9	13.0	90.8	1.30	62.4	19.9	87.8
	Ro Conc. 1–5	5.91	55.7	95.9	6.0	95.0	0.60	65.4	9.2	92.1
Regrind*	Cleaner Conc. 1–4	2.03	136.6	87.7	15.2	86.1	1.73	47.0	25.1	85.4
	Ro Conc. 1–5	6.32	47.7	95.6	5.4	94.8	0.59	49.8	8.7	92.1

*P_{80} 45 µm target, actual achieved P_{80} 21 µm

Cleaner flotation on the 'as received' concentrate sample successfully separated non-sulphide gangue from the rougher concentrate, with negligible losses of the target mineral to cleaner tailings. After cleaner flotation, 91.9% of the Au was recovered into a cleaner concentrate containing 121 g/t Au. Mass yield to cleaner concentrate was low (2.6%). Given cleaner test work was conducted on AC001, which was the 'no stibnite' sample, the cleaner concentrate contained negligible Sb. However, As content was very high, and future test work programs will need to consider measures to reduce the As content of the concentrate, given that As is a penalty element when concentrate sale to a third party is being considered.

Regrinding of the rougher concentrate resulted in a P_{80} of 21 µm, which was far lower than the 45 µm target. This overgrinding suggests that concentrate species are brittle/friable. The subsequent cleaner flotation test yielded only a reduction in Au recovery; after regrinding, 87.7% of the Au was recovered into a cleaner concentrate containing 137 g/t Au. Mass yield to cleaner concentrate was low (2.0%).

Regrinding of concentrates is not required for this sample.

13.8 Multi-Stage Diagnostic Leach Test Work

A three-stage diagnostic leach was completed to assess the nature and association of Au in feed, Au deportment with various minerals, and to gain a better understand Au losses to tailings. The diagnostic leach was completed on sub-samples of the composite samples.

The three-stage diagnostic leach involves subjecting the sample to increasingly aggressive leach conditions, as follows:

- Stage 1 – intensive cyanidation (3 g/L sodium cyanide; NaCN) for 24 hours to determine the free Au/cyanide soluble Au content;
- Stage 2 – residue from Stage 1 subject to an Aqua Regia digest to determine pyrite/sulphide mineral locked Au; and
- Stage 3 – residue from Stage 2 subject to a fire assay to determine silicate encapsulated Au.

Results from the three-stage diagnostic leach are summarised in Table 13-8.

Table 13-8: Gold deportment results – diagnostic leaching.

Diagnostic Leach	Sample AC001		Sample AC002		Sample AC003	
	Au (g/t)	Au Dist'n (%)	Au (g/t)	Au Dist'n (%)	Au (g/t)	Au Dist'n (%)
Free / Cyanide-Recoverable Gold	0.11	4.0	0.04	0.8	0.02	0.5
Gold in Pyritic Sulphide	2.56	93.1	5.34	96.6	3.88	96.8
Gold Encapsulated in Silicates	0.08	2.9	0.14	2.6	0.11	2.7
Calculated Head	**2.75**	**100**	**5.52**	**100**	**4.01**	**100**
Assayed Head	**3.33**	**-**	**6.10**	**-**	**4.47**	**-**

The diagnostic leach test work concluded that negligible free Au/cyanide soluble Au is present in the samples, consistent with the gravity recovery test work results described in Section 13.5. For all three samples, >93% of the Au is associated with sulphides, most likely arsenopyrite. This is consistent with rougher flotation test work results, which indicated a strong correlation between As recovery and Au recovery. Less than 3% of the available Au is locked in silicates.

The QP (Marius Phillips) recommends future test work consider leaching of the flotation circuit tails as a means to improve Au recovery. Low pH test work along with the use of oxidants (lead nitrate) and pre-aeration should be tested.

13.9 Cleaner Optimisation Test Work

Baseline rougher and cleaner flotation test work on the three Auld Creek composite samples (AC001, AC002, and AC003) concluded that an Au-As association exists, and that stibnite can be activated by lead nitrate to yield a very favourable flotation response.

Sample AC003 is considered to be 'typical' of the Auld Creek ore, and was therefore used to conduct further cleaner optimisation test work, with the aim of producing a concentrate containing >50% stibnite (Sb_2S_3) and <1% As.

This cleaner optimisation test work retained the primary grind size P_{80} of 106 µm. The sample was diluted to a flotation pulp density of 35 wt.% solids and then conditioned with lead nitrate (250 g/t). Staged addition of PAX collector (50 g/t + 25 g/t) and W24 frother (40 g/t) was adopted. Flotation was conducted at natural pH 7.6 and four rougher concentrates were recovered over a total flotation time of 7 minutes. No regrinding of the rougher concentrate was adopted.

Following rougher flotation, rougher concentrates were subjected to five stages of cleaning, for a cumulative cleaner flotation time of 15 minutes. Staged addition of H27 frother was adopted (20–40 g/t), with frother added to each respective cleaning stage. Cleaner flotation was conducted at natural pH 7.8.

Results from the cleaner optimisation test work are summarised in Table 13-9.

Table 13-9: AC003 cleaner optimisation flotation test results.

Product	Mass Yield (%)	Au		As		Sb		S_{Tot}	
		g/t	Rec. %	%	Rec. %	%	Rec. %	%	Rec. %
Cleaner Concentrate 5	1.69	4.74	1.84	0.39	1.37	64.5	81.2	28.2	36.3
Cleaner Concentrate 4	1.84	12.1	5.12	0.89	3.41	61.7	84.6	28.4	39.9
Cleaner Concentrate 3	2.10	22.2	10.7	2.15	9.36	56.2	87.8	28.8	46.0
Cleaner Concentrate 2	2.73	49.8	31.2	5.09	28.8	45.5	92.3	29.0	60.2
Cleaner Concentrate 1	3.58	73.3	60.2	7.79	57.8	35.6	94.8	27.7	75.5
Rougher Concentrate 1–5	5.94	69.8	95.3	7.64	94.2	22.0	97.2	21.0	95.0
Calculated Head	**100**	**4.36**	**100**	**0.48**	**100**	**1.34**	**100**	**1.31**	**100**
Assayed Head	**-**	**4.47**	**-**	**0.54**	**-**	**1.34**	**-**	**1.34**	**-**

The use of lead nitrate as an activator in lieu of copper sulphate yielded excellent results, with sample AC003 yielding an overall Au recovery to rougher concentrate of 95.3%. Likewise, high As and stibnite recoveries of 94.2% and 97.2%, respectively, were achieved. Sulphur recovery to concentrate was also high, at 95.0%. Mass pull to rougher concentrate was low, at 5.9%.

Following five stages of cleaner flotation, a stibnite recovery of 81.2% was achieved into a concentrate containing 64.5% stibnite, well above the target range of 50% stibnite. Gold recovery to cleaner concentrate was low, with 1.8% of the Au recovered into a concentrate grading 4.7 g/t Au. This selectivity through the cleaner stages provides an opportunity to produce a separate stibnite concentrate and a separate Au concentrate.

Figure 13-7 demonstrates the metal deportment through the cleaner flotation test. Given the target stibnite concentrate grade (>50% Sb; <1% As), Cleaner 4 concentrate was selected as the 'final' stibnite concentrate – under this operating regime, 84.6% of the stibnite is recovered into a concentrate containing 61.7% stibnite with corresponding As and Au grades of 0.9% and 12.1 g/t, respectively. Arsenic and Au recovery to Sb concentrate is low, at 3.4% and 5.1%, respectively. Within this 'differential' cleaner circuit, Cleaner 4 tailings represents the final Au concentrate, with 90.1% of the Au recovered into a concentrate assaying 90.1 g/t Au. The As content of the Au concentrate is 10.7% and the QP (Marius Phillips) recommends considering measures to reduce As grade in future works. Mass pull to Sb concentrate is low (1.8%), while mass pull to Au concentrate is 4.1%.

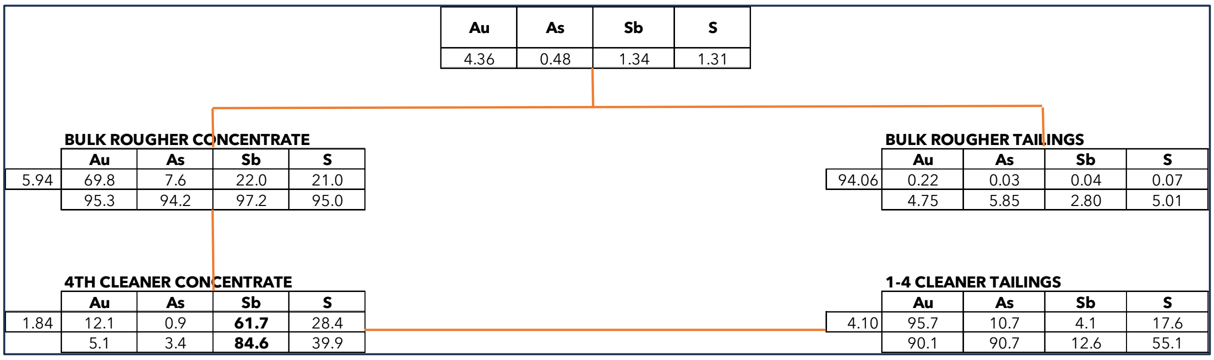


Figure 13-7: Cleaner flotation metal deportment.

To improve stibnite recovery, Cleaner 3 concentrate could also be considered as the 'final' stibnite concentrate. Under this operating regime, 87.8% of the stibnite is recovered into a concentrate containing 56.2% stibnite; however, the As grade increases and this concentrate contained 2.2% As and 22.2 g/t Au. Within this three-stage 'differential' cleaner circuit, Cleaner 3 tailings represents the final Au concentrate, with 84.6% of the Au recovered into a concentrate assaying 95.8 g/t Au. The As content of the Au concentrate remained consistent at 10.6%.

Following this initial cleaner optimisation test, two additional cleaner optimisation tests were completed. These tests adopted a differential rougher approach followed by two stages of cleaning on the Sb concentrate. The first test (summarised in Figure 13-8), run at natural pH, adopted a differential rougher float with reduced collector addition, and produced an Au-bearing arsenopyrite concentrate. Rougher tailings was then conditioned with lead nitrate to promote stibnite flotation. This approach produced an Au concentrate containing 91.5 g/t Au, 10.0% As, and 12.3% stibnite, with a corresponding Au recovery of 92.0% and a stibnite recovery of 39.7%. The two-stage cleaner circuit recovered 52.6% of the stibnite into a concentrate containing 43.1% Sb and 0.53% As. Given the poor stibnite recovery, this circuit configuration was not explored further.

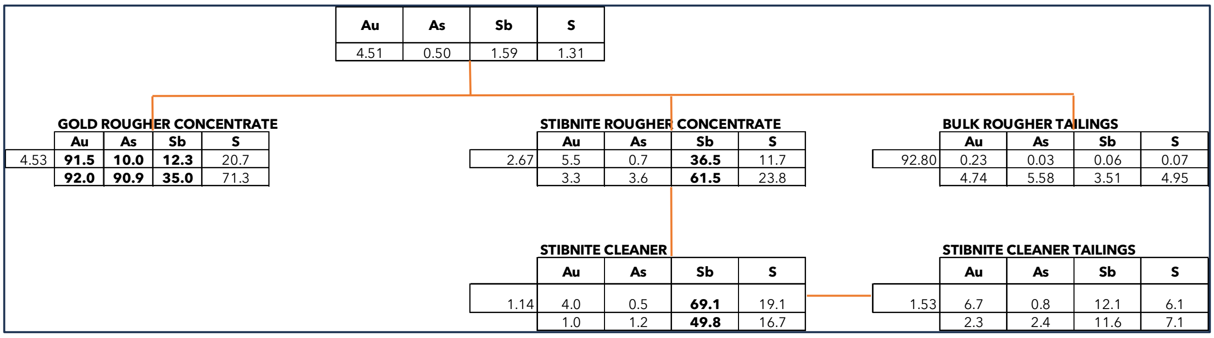


Figure 13-8: Cleaner flotation metal deportment – differential rougher at natural pH.

As a variation to the above, the same test was repeated at elevated pH 10 to assess whether increased pH would improve the differential flotation response. This approach (Figure 13-9) produced an Au concentrate containing 95.3 g/t Au, 10.2% As, and 7.4% stibnite, with a corresponding Au recovery of 84.6% and a stibnite recovery of 21.0%. The two-stage cleaner circuit recovered 67.2% of the stibnite into a concentrate containing 51.3% Sb and 2.3% As. Although increasing the Au rougher circuit pH reduced the stibnite deportment to rougher concentrate, Au and As recovery also reduced, resulting in additional As reporting to the stibnite concentrate. Given the poor stibnite recovery, this circuit configuration was not explored further.

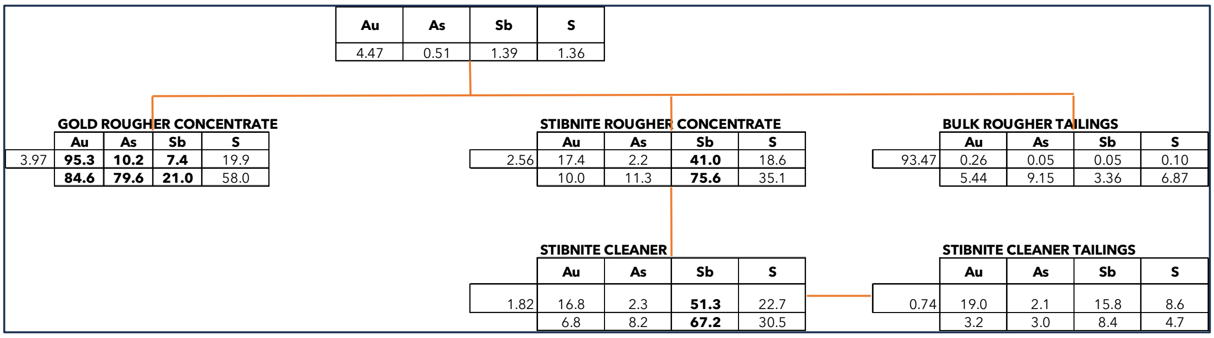


Figure 13-9: Cleaner flotation metal deportment – differential rougher at elevated pH.

14 Mineral Resource Estimates

14.1 Informing Data

The data were provided by RUA in the form of MS Excel workbooks containing drillhole data (Table 14-1) and uploaded by RSC into an MS Access database. The drillhole database *Auld Compiled* contains collar, geology, recovery, survey, and assay information. Assay method information was compiled and provided for each element, and an Au_Best_ppm field was created by RUA using a priority system, with the highest priority taking precedence. Density information was provided in a separate *RUA_Density_Master* MS Excel spreadsheet containing raw density data. The spreadsheet included a calculation for bulk density and was sorted by mineralised domain. The trench database *August 2024 Auld Creek Trench Database* contained collar, survey, lithology, and assay information. Trench z-values were adjusted by RRPL by draping them onto the LiDAR surface due to difficulties in obtaining accurate surveyed GPS z-values for trenches. Both z-values (original and draped to the LiDAR surface) were stored in the Leapfrog trench database.

Table 14-1: Data used for the Auld Creek MRE.

Type	Drillholes	Metres
Diamond drilling	64	11,296.3
Trench	14	108.9

14.2 Interpretation & Model Definition

14.2.1 Geological Domains

The interpretation of geological domains is crucial for providing a first-order constraint on grade populations and ensuring the geological controls on mineralisation guide the modelling of estimation domains.

Gold mineralisation is hosted in quartz reefs within tightly folded sandstone and siltstone units of the Greenland Group. Disseminated mineralisation comprises silicified acicular arsenopyrite within adjacent siltstone and sandstone and forms halos surrounding mineralised quartz reefs. A summary of the Project geology and controls on mineralisation is provided in Section 7 (see Aldrich and Whaanga, 2024, for further details).

Geological modelling was conducted by the QP (Abraham Whaanga) in Leapfrog Geo, using interval selection and the vein system tools to create a geological model. An oxide model compiled by the QP (Abraham Whaanga) was based on interval selection of weathering (Figure 14-1). The weathering profile was taken as an offset surface from the topography, as drillhole coverage is not sufficient to create a surface on its own. Two categories were created: oxidised, including the weathering selections slightly weathered (sw), moderately weathered (md), and extremely weathered (ew); and fresh, i.e. unweathered (uw). Selections for individual drillholes were adjusted to fit the surrounding data.

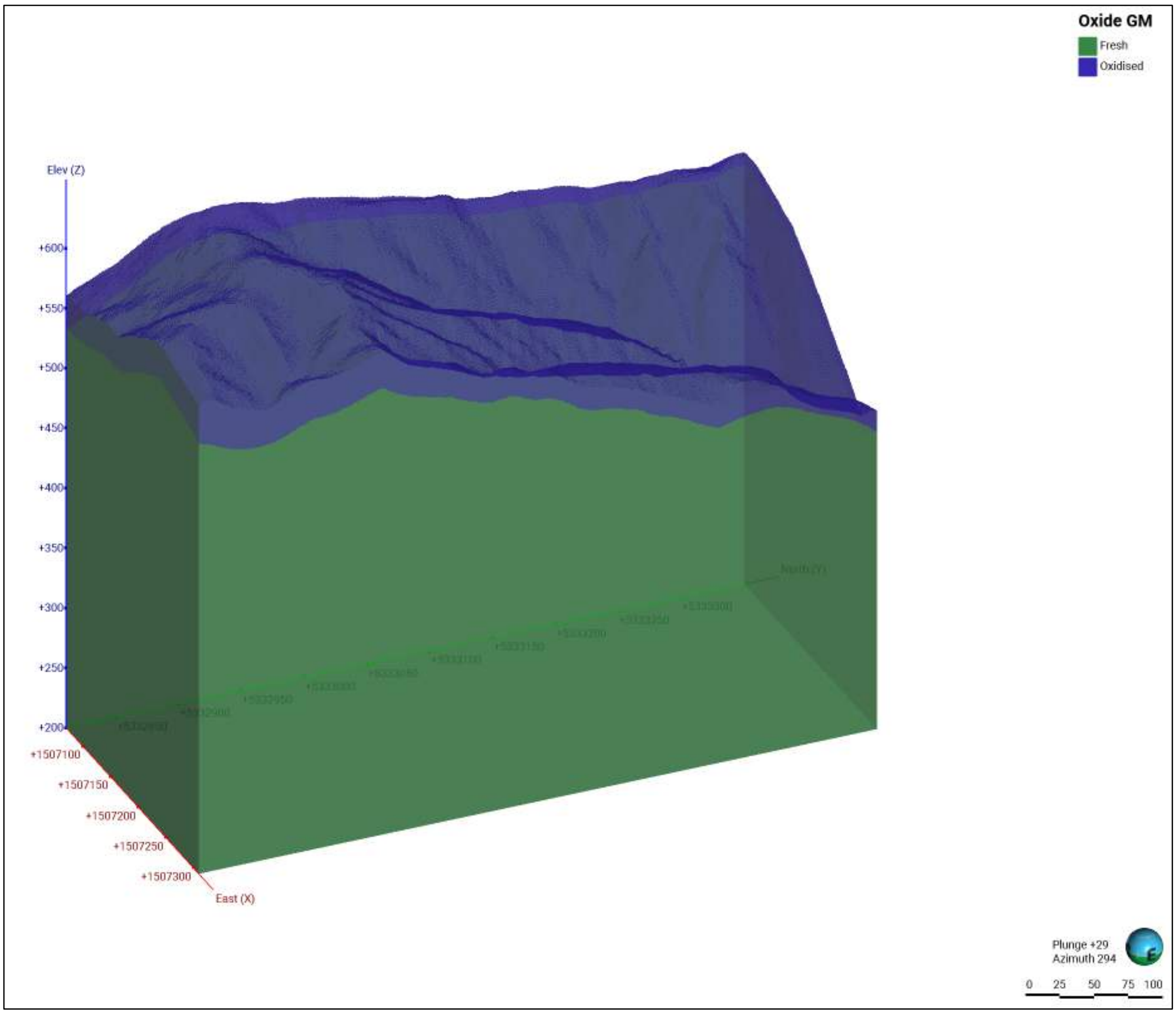


Figure 14-1: Auld Creek oxide model. Blue = oxidised; green = fresh.

The geological model was largely based on the 2024 RRPL geological model interpretation. Infill and extensional drilling into this model conducted by RUA in 2025 has confirmed grade continuity around the intersection of Bonanza East and Fraternal and geological continuity down dip and along strike within Fraternal. The geological model interpretation was validated by the QP using surface trench data and is considered a reasonable interpretation of the reef trends and orientation present at Auld Creek. In the geological model created by the QP, two structural domains were guided by lithological codes and a review of core photos covering the full width of the Bonanza and Fraternal shear zones. The QP created a merged drillhole and trench table with full lithology and assay information, with summary statistics reviewed, grouping lithology (Sum Log) and assay data. The modelled geological domains for Auld Creek lie within these two structural domains and represent the mineralised portions of the shear zones, based on cross-sections supplied by RUA (Figure 14-2 and Figure 14-3). The logged lithologies with the highest Au grades, ranging from a mean of 11 g/t to 2.5 g/t, were used in a filter with Au grades of >0.5 g/t. Intervals matching these criteria were selected in combination with the observation of the drilling cross-sections, indicating the presence of a quartz reef or high-grade vein (Table 14-2). Multiple mineralised intercepts were identified within the north of the Fraternal shear zone. Two minor lodes oriented subparallel to Fraternal were also modelled (Figure 14-4).

Table 14-2: Auld Creek summary statistics sorted by highest mean Au grade.

Lith code	Length (m)	Mean Au (g/t)	Mean Sb (%)	Description
SZ	0.8	11.14	2.85	Shear zone
SBX	12.8	8.98	34.74	Sb breccia
QTZ	3.0	7.78	1.28	Quartz breccia
QBX	11.7	6.98	3.22	Quartz vein
MS	0.8	6.04	34.74	Massive sulphide
FLT	41.9	4.60	0.41	Fault
MGK	196.6	2.99	0.78	Mineralised greywacke
MAR	21.4	2.93	0.08	Mineralised argillite
PBX	16.8	2.51	1.85	Pug breccia

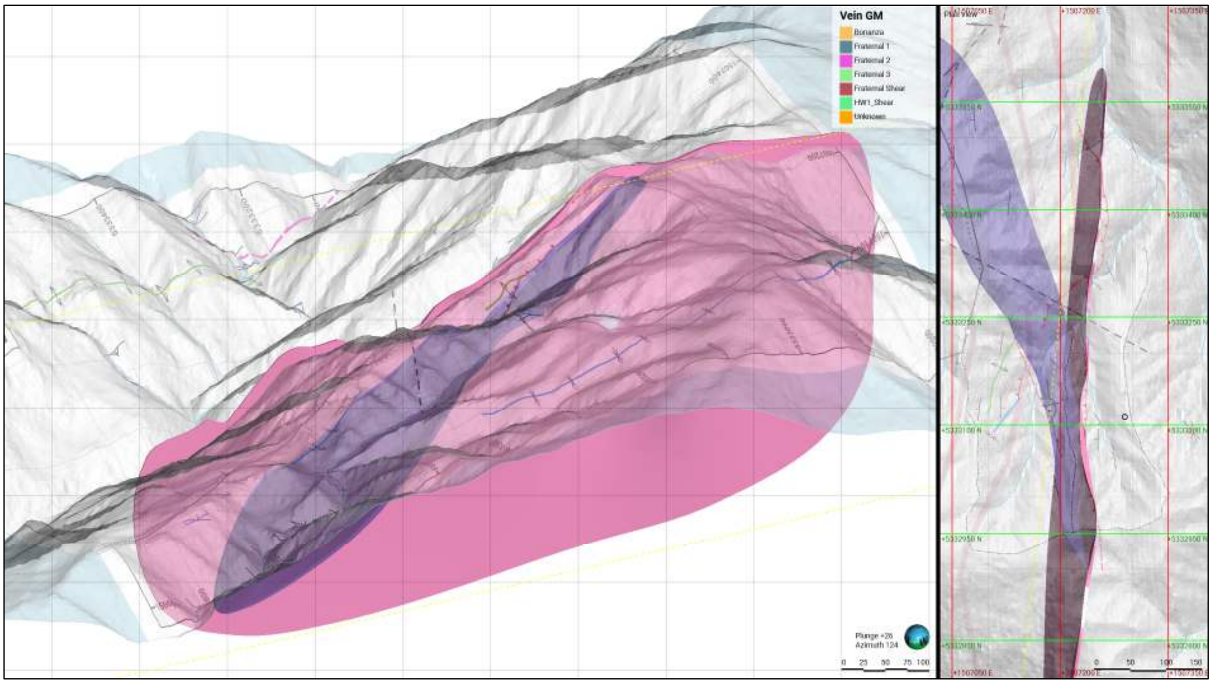


Figure 14-2: Modelled structural domains for Fraternal and Bonanza.

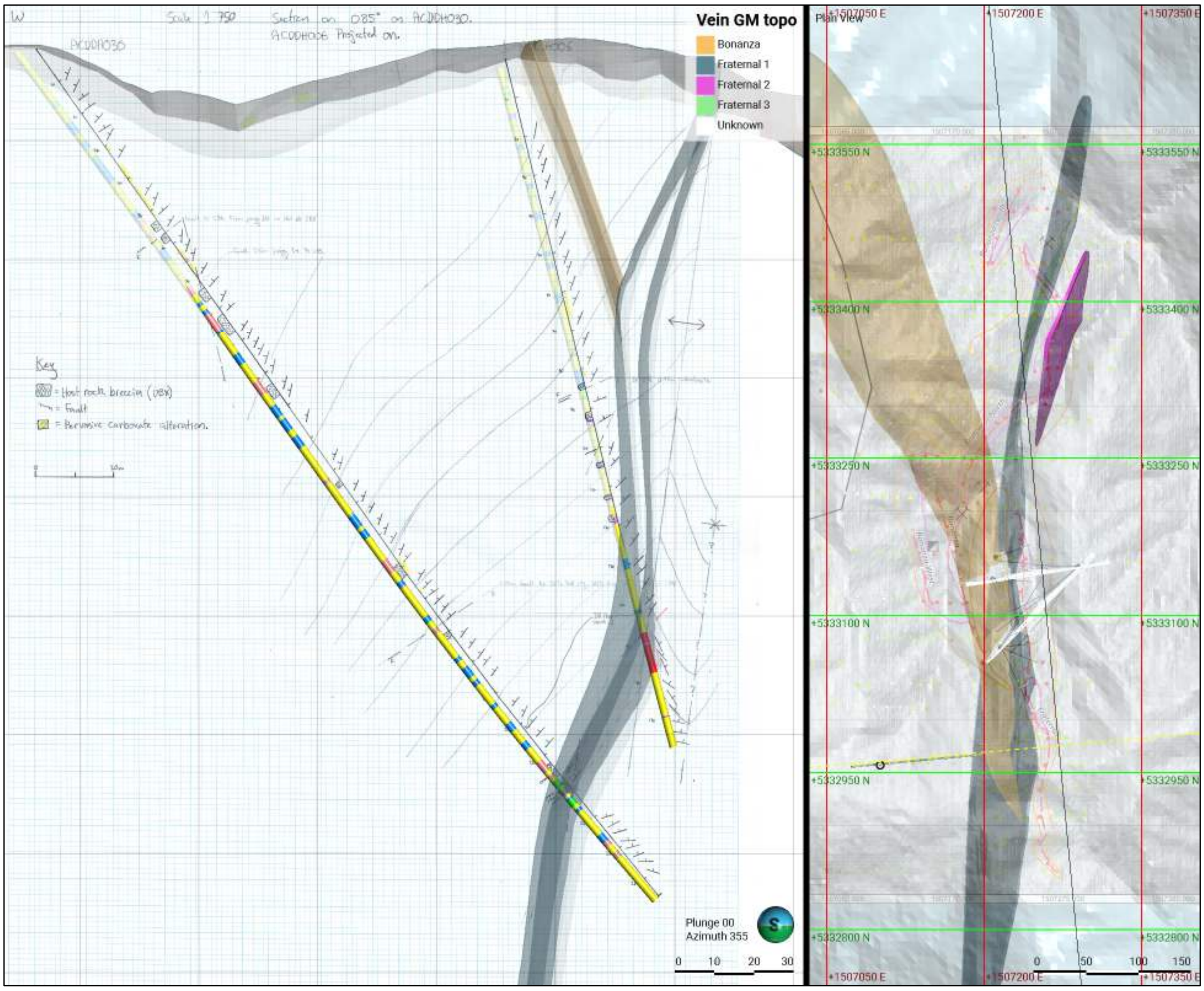


Figure 14-3: Paper cross-sections with geological interpretations for Fraternal and Bonanza.

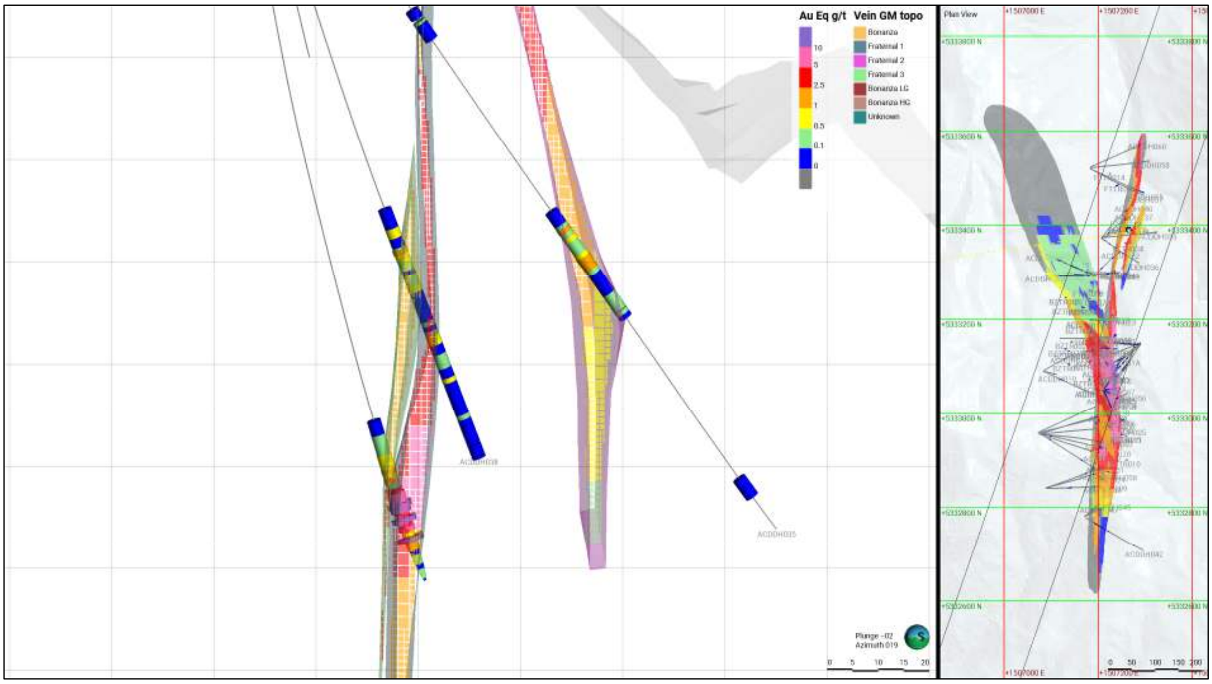


Figure 14-4: Cross-section of Fraternal 3 (left), Fraternal 1 (centre), Fraternal 2 (right).

14.2.2 Estimation Domains

The estimation domains were derived from a combination of the geological and weathering models. The two Au estimation domains displayed monomodal distributions with low coefficient of variation (CV) values and reasonable adherence to intrinsic stationarity assumptions.

The QP (Abraham Whaanga) completed contact analysis to investigate the boundary conditions of each domain (Figure 14-5 and Figure 14-6). The mean grade was reviewed inside the domains, around the boundaries, and outside the domains. There are clear transitions between the mineralisation hosted in quartz reefs and disseminated mineralisation occurring in halos surrounding the quartz reefs. This is consistent with the geological interpretation and logging of mineralisation from drill core. Hard boundaries were used for both domains during estimation to protect the distinct boundaries between estimation domains. Figure 14-5 illustrates a transition at the boundary from 0–4 g/t Au within the domain. As the boundary is a distinct geological domain, a hard boundary was used, leaving the variability within the mineralised domain.

Analysis of Sb mineralisation within the individual geological domains of Fraternal and Bonanza (Figure 14-7) indicated a highly skewed distribution that can be separated into two sub-domains either side of a 0.1 % threshold: a monomodal distribution for Sb < 0.1% and a long tail of high Sb values for Sb > 0.1%, resulting in a global distribution with a CV of 2.6. A review of the distribution of high and low grades indicates that they cluster. It is likely that high- and low-grade shoots are present; however, at the current drill spacing, there are insufficient geological data to separate the domains. Therefore, grades were used to derive two Sb estimation domains from the data by defining an indicator at a top-cut level of 0.1% and estimating the proportions of mineralisation above and below the top-cut separately, then combining the Sb estimates of each individual portion to obtain the final Sb values.

Drilling carried out in 2025 into the northern portion of Bonanza demonstrated continuity of the shear zone; however, a clear break in mineralisation was observed, with low Au and Sb grades present within the weakly mineralised shear zone (Figure 14-8). High- and low-grade Au estimation sub-domains were derived from the logging and sample data. Soft boundaries were used to create a smooth transition between the two domains, and the results were validated visually and by review of the global block model mean compared with the declustered composite mean (section 14.6.1).

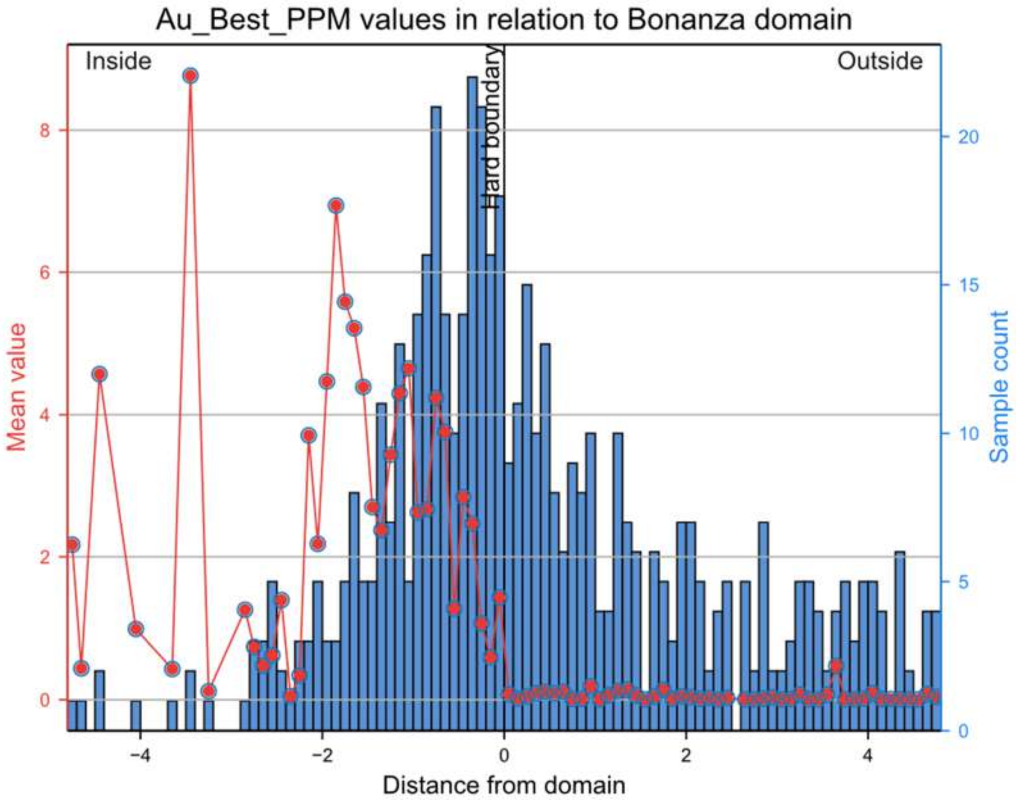


Figure 14-5: Contact analysis plot for the Bonanza Au domain.

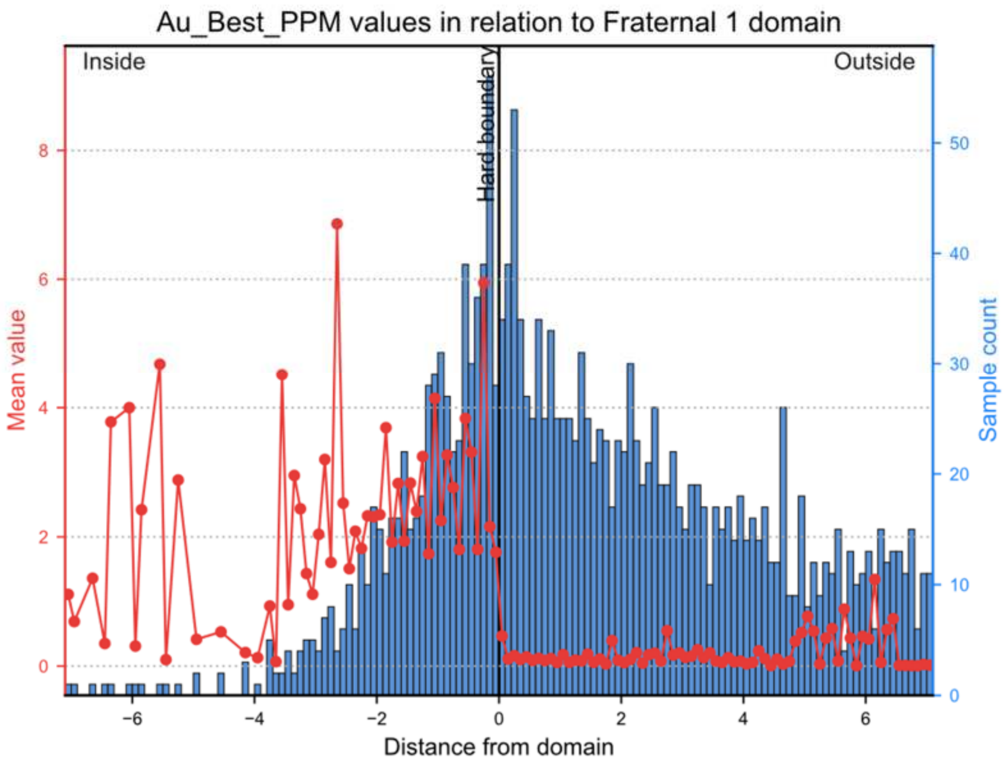


Figure 14-6: Contact analysis plot for the Fraternal Au domain.

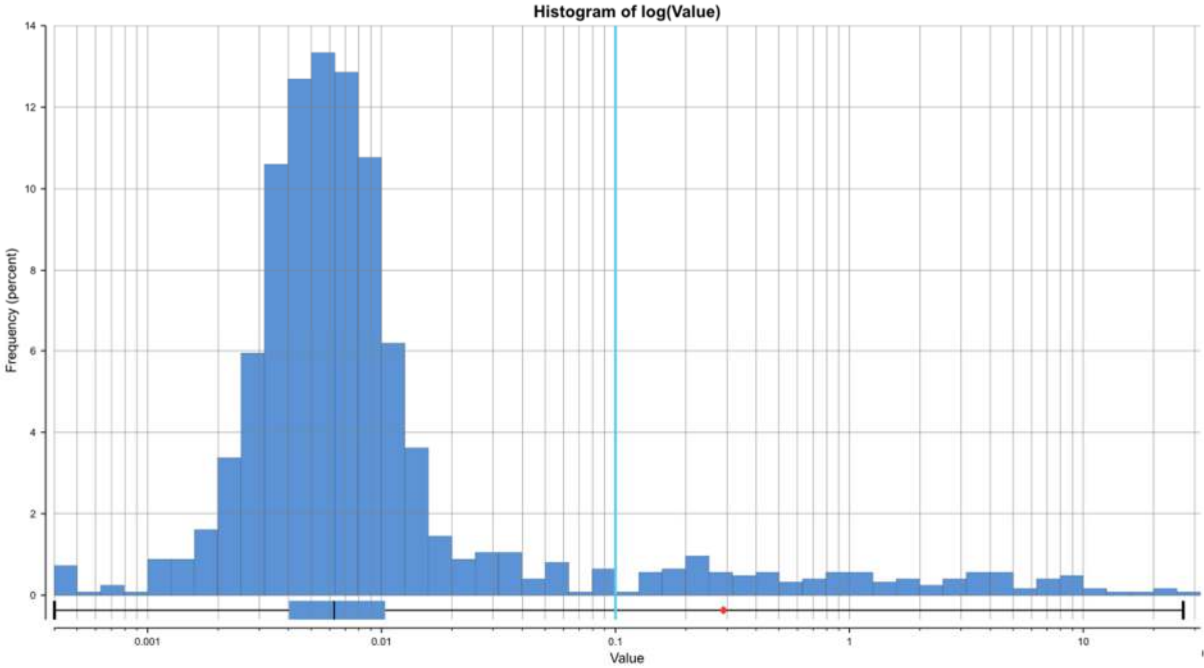


Figure 14-7: Histogram Sb sample distribution illustrating a bimodal population and 0.1% cut-off.

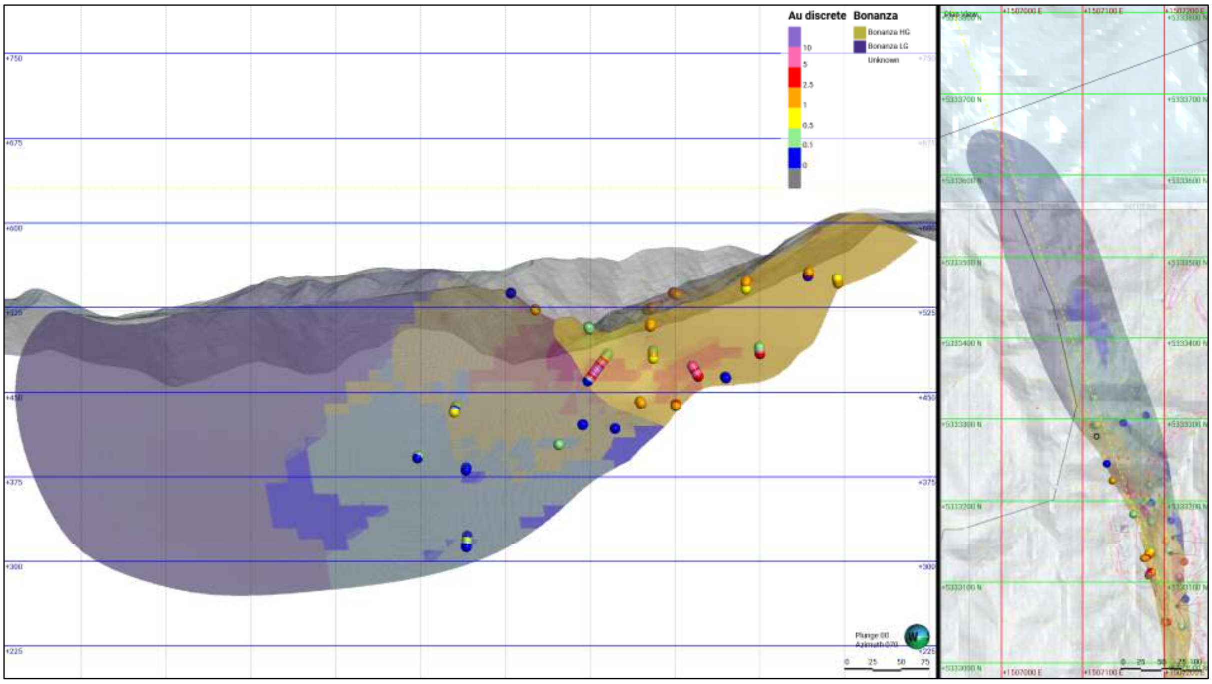


Figure 14-8: Bonanza sub-domaining (low-grade left, high-grade right).

14.2.3 Extrapolation

Extrapolation of the mineralised intersections varies from ~10 m to ~70 m; the extrapolation distances typically relate to the local drillhole spacing. The most extreme distances occur at the north end of the Bonanza reef, where a small number of widely spaced drillholes define a ~70-m reef. The QP considers this to be a reasonable interpretation based on the available drilling data and surface trench data. Several drillholes completed in 2025 extended the Fraternal lode both down dip and to the north. ACDDH031 intersected 1.7 m (true width) of mineralisation at 4.2 g/t Au and 10.4% Sb ~80 m down dip from the previous interpretation, demonstrating grade and geological continuity. ACDDH032, ~80 m south of ACDDH031, intersected 1.8 m (true width) of mineralisation at 1.2 g/t Au and 0.02% Sb, demonstrating geological continuity. Fifteen drillholes extended the Fraternal lode 400 m to the north, demonstrating geological and grade continuity (Figure 14-9).

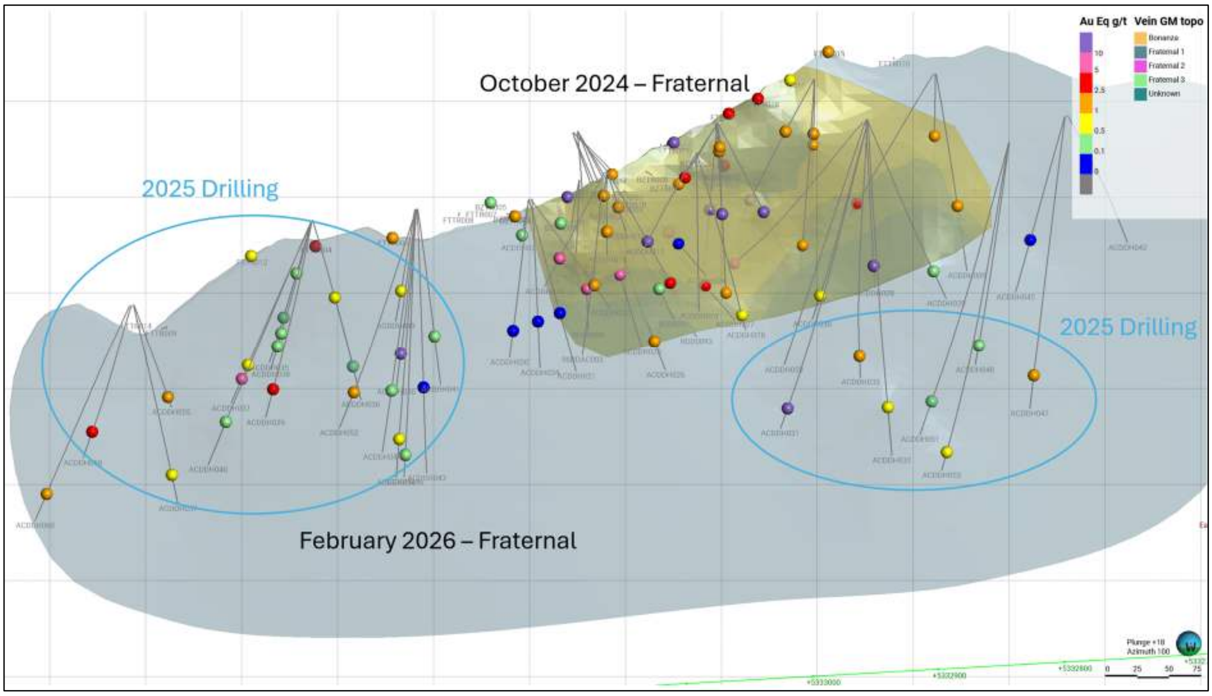


Figure 14-9: Comparison of Fraternal between October 2024 and February 2026.

14.2.4 Alternative Interpretations

At a large scale, the controls on mineralisation are well understood and supported by the data and geology. However, the geological domains may vary considerably between known locations, and additional sampling may provide locally alternative interpretations.

The QP (Abraham Whaanga) considers that, at this stage in the Project and at this level of data resolution, alternative interpretations of the geology and mineralisation are possible; however, they are not likely to generate models or estimates that are significantly different.

14.2.5 Summary Statistics & Data Preparation

The distribution of interval lengths indicates that 73% of the intervals sampled had a length of 1 m. Assay data were composited to 2-m intervals, as this provides a good compromise between respecting the complexity of the estimation domains, due to low drill intersection angles (Figure 14-10 and Figure 14-11), and assuring that no mineralised intervals are split in the compositing process. Sensitivity to the compositing scheme and treatment of the residual intervals were tested as part of the sensitivity analysis.

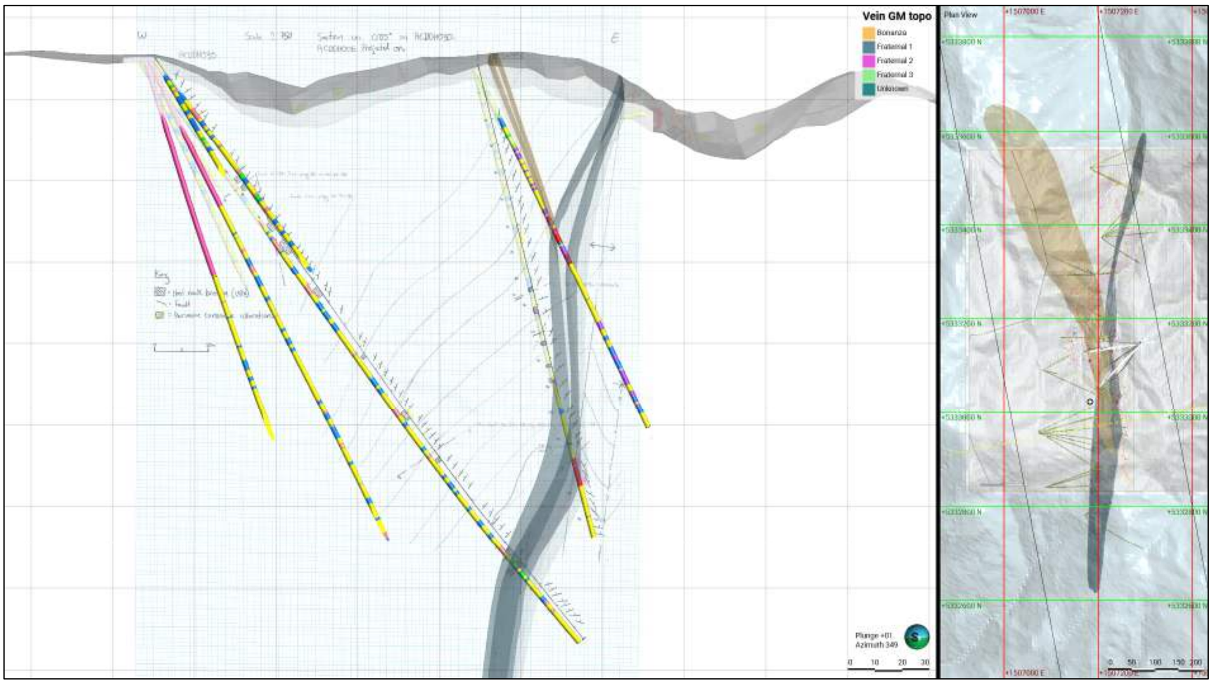


Figure 14-10 Section view illustrating low drillhole intersection angles in the Fraternal domain.

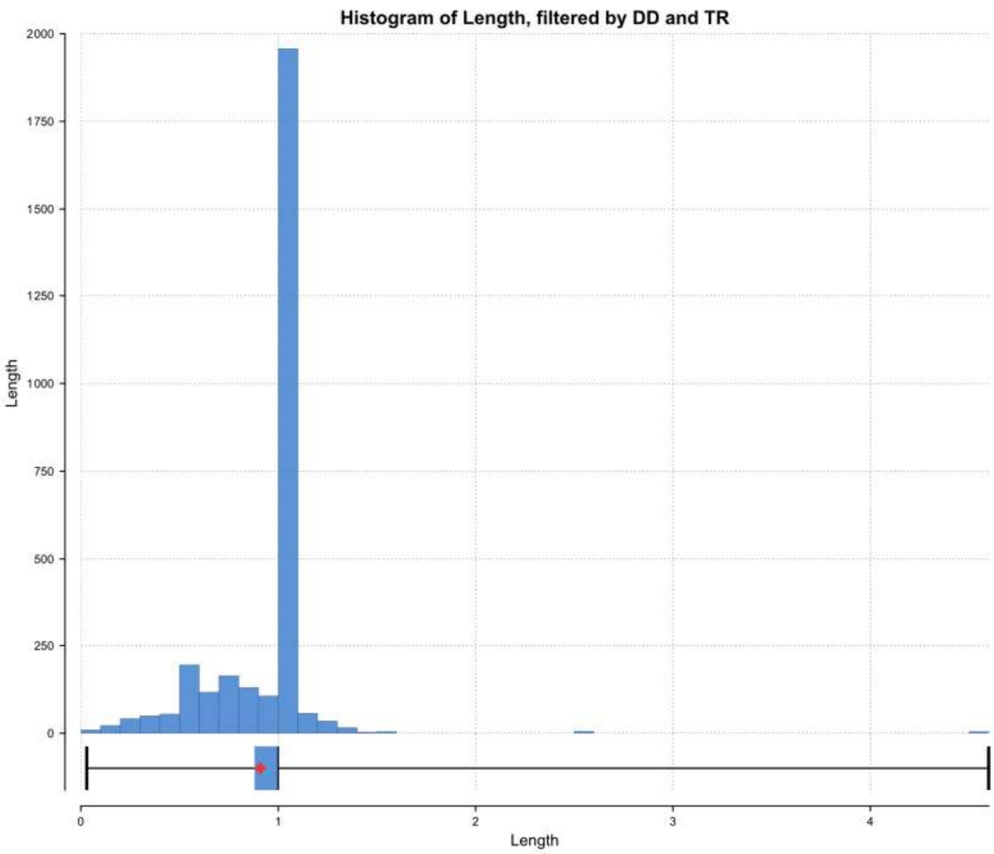


Figure 14-11: Histogram of interval lengths for the Auld Creek deposit.

All grade variables are characterised by skewed distributions and low to moderate CVs. The CVs range from 1.1 to 1.6 before top-cutting (Table 14-3 and Table 14-4). Example log histograms are presented in Figure 14-12 to Figure 14-15.

Table 14-3: Auld Creek domain statistics for Au (2-m composites).

Assay	Domain	Count	Length	Mean	SD	CV	Variance	Min	Q1	Q2	Q3	Max
Au (g/t)	Bonanza	87	166.9	2.27	2.92	1.28	8.51	0.01	0.28	1.16	3.37	16.18
	Fraternal	264	519.3	2.43	3.89	1.60	15.14	0.01	0.42	1.28	3.47	47.02

Table 14-4: Auld Creek domain statistics for Sb (2-m composites).

Assay	Domain	Count	Length	Mean	SD	CV	Variance	Min	Q1	Q2	Q3	Max
Sb >0.1%	Bonanza	25	49.7	4.87	7.89	1.62	62.18	0.10	0.28	0.94	4.74	28.06
	Fraternal	90	179.4	2.45	3.65	1.49	13.32	0.10	0.31	0.86	3.29	20.03
Sb <0.1%	Bonanza	63	119.6	0.01	0.02	1.10	0.00	0.00	0.01	0.01	0.02	0.08
	Fraternal	174	340.0	0.02	0.02	1.08	0.00	0.00	0.01	0.01	0.02	0.09

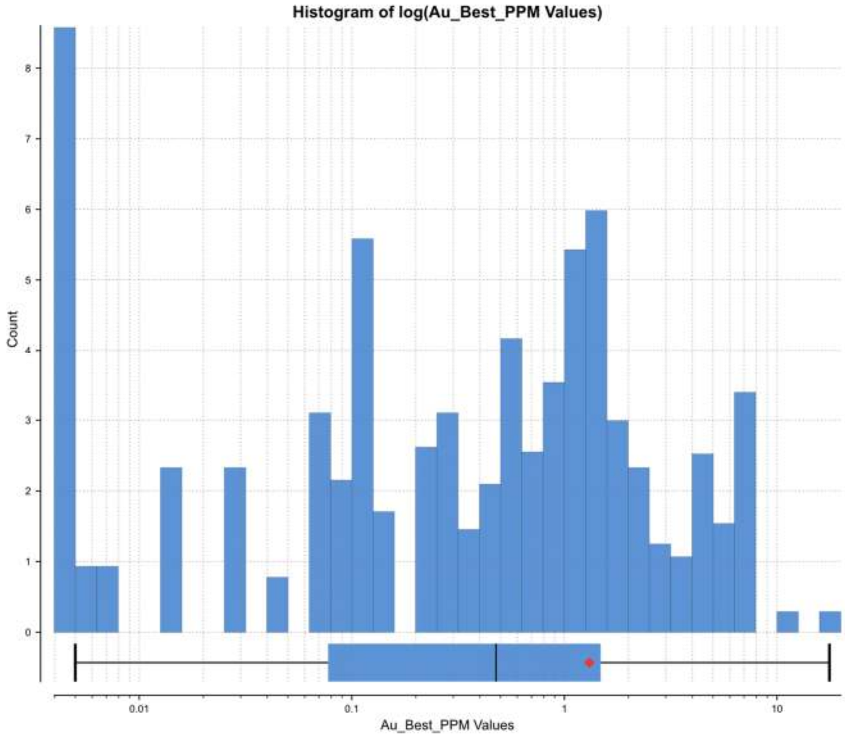


Figure 14-12: Bonanza log histogram of the Au grade variable (declustered 2-m composites).

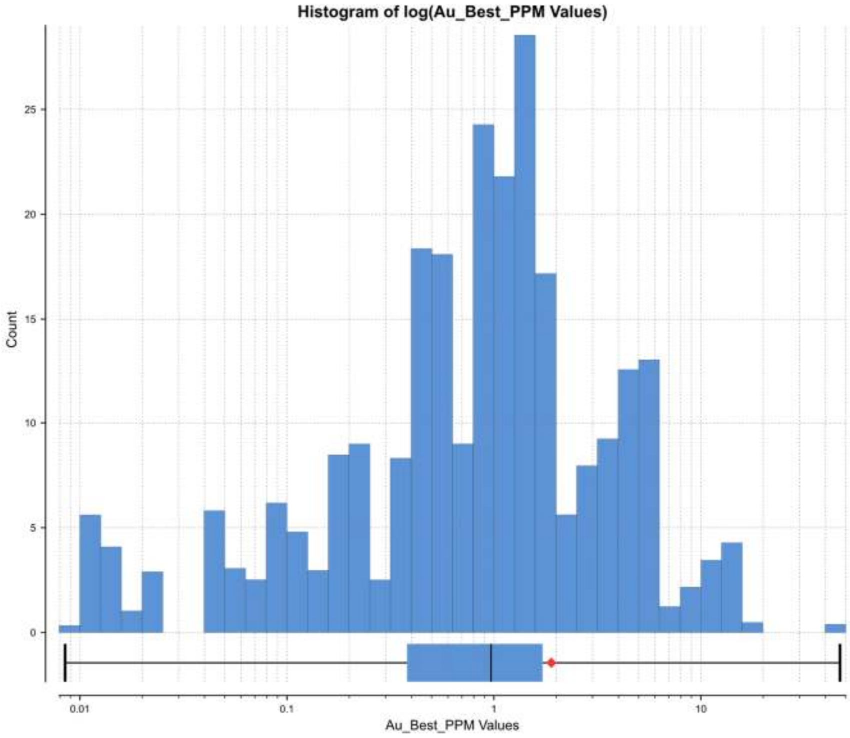


Figure 14-13: Fraternal log histogram of the Au grade variable (declustered 2-m composites).

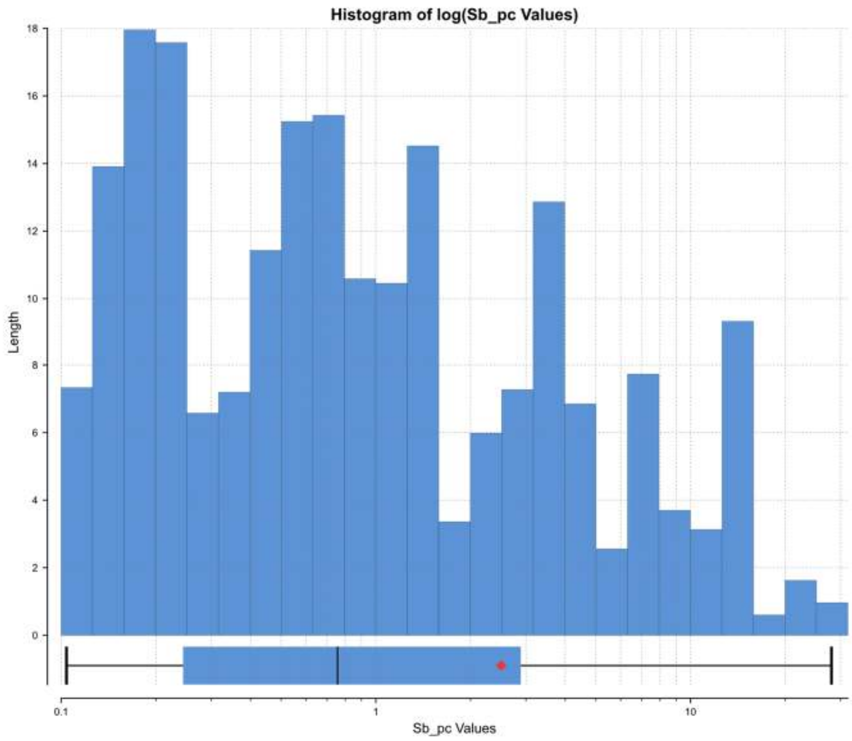


Figure 14-14: Log histogram (>0.1%) of the Sb grade variable (2-m composites, Bonanza and Fraternal combined).

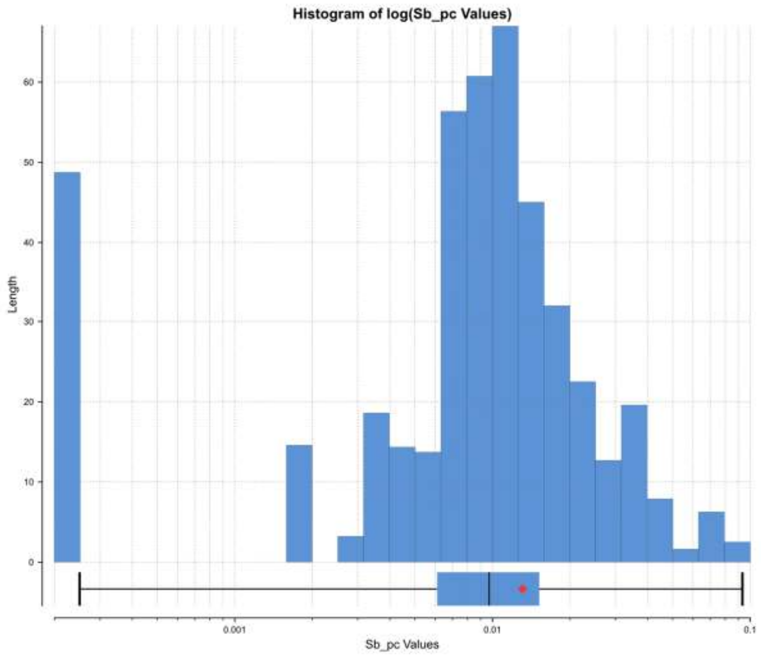


Figure 14-15: Log histogram (<0.1%) of the Sb grade variable (2-m composites, Bonanza and Fraternal combined).

14.2.6 Bulk Density

Density data for the mineralised domain have a normal distribution. A total of 142 data points were available in the estimation domains, with a mean density of 2.75 g/cm^3. Mineralisation at Auld Creek with high Sb content has density values >3.0 g/cm^3. Bulk densities are grouped in the estimation domain model (Figure 14-16).

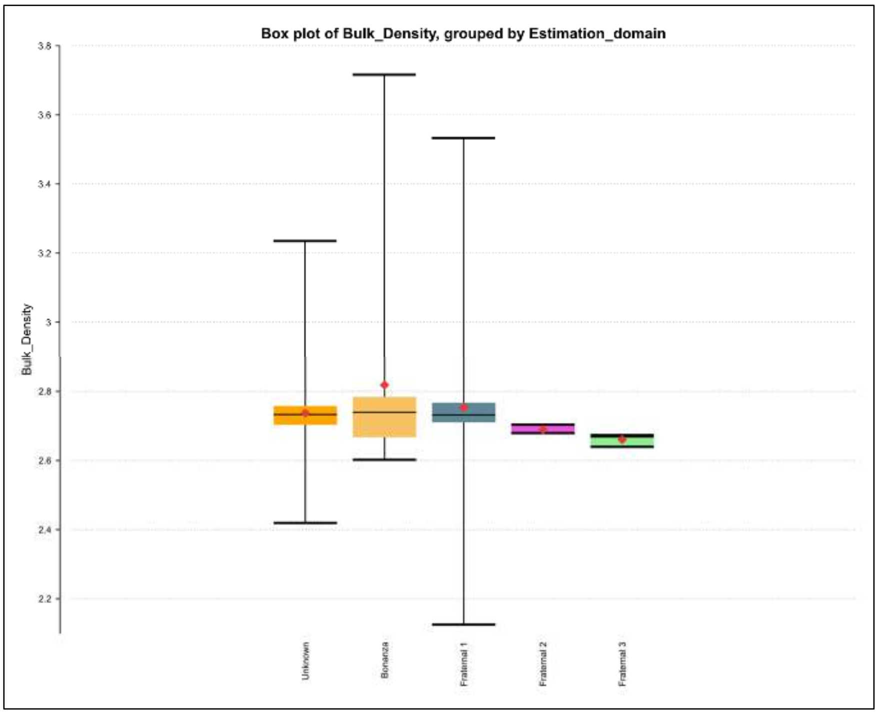


Figure 14-16: Density grouped by estimation domain.

14.3 Spatial Analysis & Variography

14.3.1 Gold

Experimental variography was completed on the normal-scores transform of the composited Au grades within each estimation domain. Variogram models were fitted using one spherical structure. Variogram models were fitted with a relatively low γ_0 ratio (30%), with practical ranges (at which 90% of the variance is reached) of 75–85 m in the major direction, 50 m in the semi-major direction and 6–10 m in the minor direction. Variogram model parameters are presented in Table 14-5. An example semi-variogram and associated model is presented in Figure 14-17. The back-transformed continuity models were then used to assign weights in the estimation. The variogram models fitted support the level of confidence required for the classification objective.

Table 14-5: Auld Creek modelled variogram parameters for Au estimation domains.

Variogram parameters		Domain	
		Bonanza	Fraternal
Normalised Nugget		0.3	0.3
S1	**Normalised Sill**	0.7	0.7
	Major	120	113
	Semi-Major	80	82
	Minor	8	10
	Dip	76.7	86.3
	Dip Azimuth	71.8	273.4
	Pitch	27.0	157.4

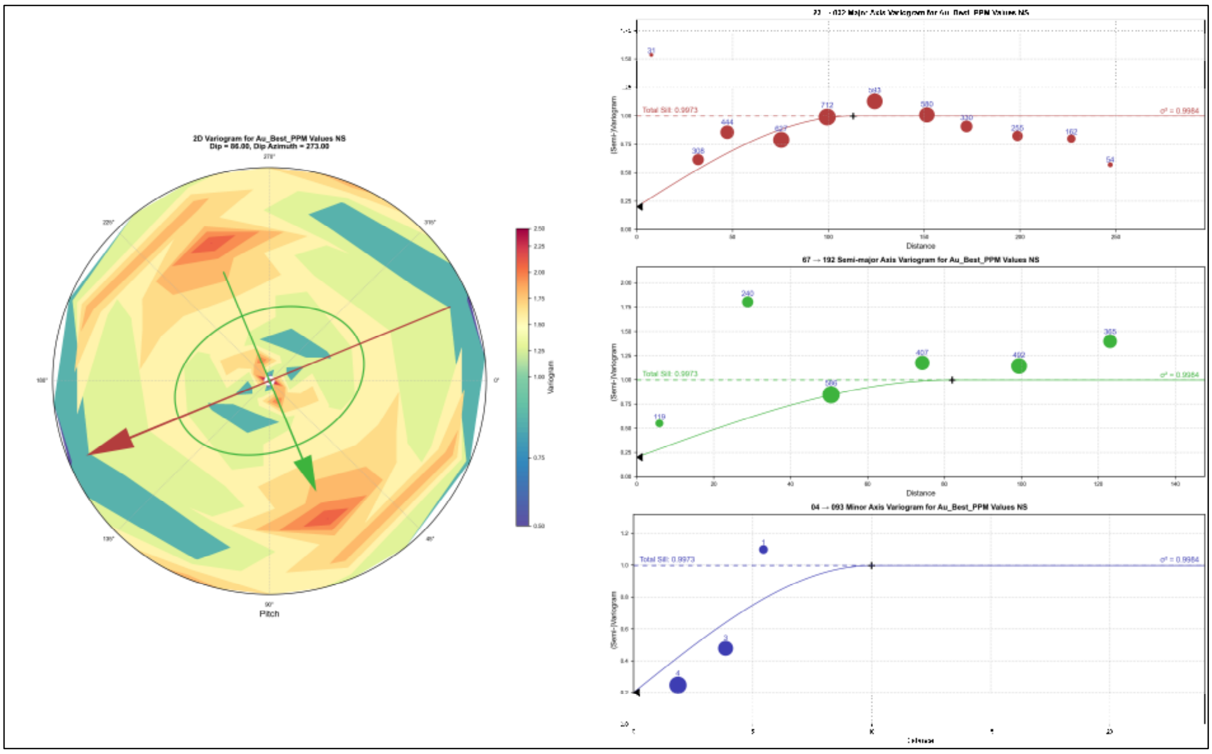


Figure 14-17: Fraternal Au semi-variogram.

14.3.2 Antimony

Experimental variography was completed on the normal-scores transform of the composited Sb grades within each estimation domain. Variogram models were fitted using one spherical structure. Variogram models were fitted with a low to moderate γ_0 ratio (ranging from 20% to 60%), with practical ranges (at which 90% of the variance is reached) of 50–70 m in the major direction, 30–50 m in the semi-major direction and 6–8 m in the minor direction. Variogram model parameters are presented in Table 14-6 and

Table 14-7. The back-transformed continuity models were then used to assign weights in the estimation. The variogram model fits the experimental data well and supports the level of confidence required for the classification objective.

Table 14-6: Auld Creek modelled variogram parameters for >0.1% Sb estimation domains.

Variogram parameters		Domain	
		Bonanza >0.1%	Fraternal >0.1%
Normalised Nugget		0.6	0.6
S1	**Normalised Sill**	0.4	0.4
	Major	80	80
	Semi-Major	60	60
	Minor	5	5
	Dip	70.0	81.2
	Dip Azimuth	70.0	275.4
	Pitch	30.8	160

Table 14-7: Auld Creek modelled variogram parameters for <0.1% Sb estimation domains.

Variogram parameters		Domain	
		Bonanza <0.1%	Fraternal <0.1%
Normalised Nugget		0.2	0.4
S1	**Normalised Sill**	0.8	0.6
	Major	80	75
	Semi-Major	40	40
	Minor	8	8
	Dip	70.0	81.2
	Dip Azimuth	70.0	275.4
	Pitch	19.2	146.8

14.3.3 Bulk Density

Density data were assigned in the block model to match the mean density for fresh and oxidised material within the oxide model.

14.4 Block Model

The block model parameters are detailed in Table 14-8. The block model was left un-rotated, as the mineralisation strikes north-northeast to north-northwest. Block dimensions were chosen to represent half the drill spacing along strike y (20 m) and across strike x (10 m) and sufficiently represent the changes in orebody dip along z (10 m). The geometry of estimation domains was approximated using sub-blocks of 0.625 m × 1.25 m × 1.25 m (x, y, and z) to satisfactorily represent changes in strike and dip that are typical of the narrow, high-grade shoot geometry.

Discretisation of 3 × 5 × 3 points along the x, y, and z directions was selected.

Table 14-8: Auld Creek block model description.

Number of Parent Blocks	50 × 65 × 53 = 172,250
Sub-Blocks per Parent	16 × 16 × 8 = 2,048
Sub-Block Model	Octree
Base Point: x, y, z (m)	1507070, 5332810, 630
Parent Block Size: x, y, z (m)	10, 20, 10
Minimum Sub-Block size: x, y, z (m)	0.625, 1.25, 1.25
Boundary Size (m)	500, 1,300, 530
Leapfrog Rotation	None
Azimuth	0°
Dip	0°
Pitch	0°

14.5 Estimation

The variables were estimated in the block model in one or two passes, with variable orientation based on the vein reference surface to guide the ellipsoid direction for Au and Sb. Search distances and minimum samples, maximum samples, and samples per drillhole search neighbourhood are detailed in Table 14-9 and Table 14-10.

Table 14-9: Auld Creek search neighbourhood parameters.

Estimation Name	Ellipsoid Range Maximum	Ellipsoid Range Intermediate	Ellipsoid Range Minimum
Kr Au Bonanza LG Pass1	100	80	80
Kr Au Bonanza LG Pass2	100	80	80
Kr Au Bonanza HG Pass1	140	70	70
Kr Au Bonanza HG Pass2	140	70	70
Kr, Au Fraternal Pass1	80	60	60
Kr, Au Fraternal Pass2	100	70	40
Kr, I1 Sb Bonanza	90	70	60
Kr, I1 Sb Fraternal	150	90	90
Kr, Sb cut Bonanza	90	70	60
Kr, Sb cut Fraternal	150	90	90

Table 14-10: Auld Creek number of samples per pass.

Estimation Name	Minimum Number of Samples	Maximum Number of Samples	Maximum Number of Samples per Drillhole
Kr Au Bonanza LG Pass1	5	20	3
Kr Au Bonanza LG Pass2	2	25	
Kr Au Bonanza HG Pass1	5	20	3
Kr Au Bonanza HG Pass2	2	20	
Kr, Au Fraternal Pass1	5	20	3
Kr, Au Fraternal Pass2	2	25	
Kr, I1 Sb Bonanza	5	25	2
Kr, I1 Sb Fraternal	4	25	2
Kr, Sb cut Bonanza	5	25	
Kr, Sb cut Fraternal	4	25	

14.5.1 Gold

Grades were interpolated using ordinary kriging (OK). Search neighbourhoods were optimised for global accuracy to yield sufficient samples for estimation and create an acceptable level of smoothing while minimising conditional bias. Search neighbourhoods were 80–140 m in the major direction (y), 60–80 m in the semi-major direction (x), and 40–80 in the minor direction (z). The variable orientation tool was used to account for strike and dip changes in the wireframes. A minimum of five and a maximum of 25 samples were used to inform the estimate in most domains. A maximum number of three samples per drillhole was used to ensure a minimum of two drillholes were included per estimate. Second-estimate passes had a minimum of two samples, and the maximum number of samples per drillhole limit was removed to fill blocks within the domain.

One outlier grade of 47 g/t Au was identified within the Fraternal domain. An outlier restriction of 50% of the search at a grade of 20 g/t Au was used for pass 1 and 25% of the search at a grade of 20 g/t Au was used for pass 2.

14.5.2 Antimony

Grades were interpolated using OK. Search neighbourhoods were optimised for global accuracy to yield sufficient samples for estimation and create an acceptable level of smoothing while minimising conditional bias. Search neighbourhoods were 90–150 m in the major direction (y), 70–90 m in the semi-major direction (x) and 60–90 in the minor direction (z). The variable orientation tool was used to account for strike and dip changes in the wireframes. A minimum of 4 and a maximum of 25 samples were used to inform the estimate in most domains. A maximum number of two samples per drillhole was used to ensure a minimum of three drillholes were included per estimate for the <0.1% domains, and the drillhole restriction removed for the >0.1% Sb domains.

No further top-cuts were applied for Sb domains.

14.5.3 Bulk Density

Density data were not composited due to sparse availability, so raw sample data were used in the estimate. An average density of 2.65 g/cm^3 was applied to oxidised material, and 2.75 g/cm^3 was applied to fresh material. A comparison of density by estimation domain illustrates that density at Bonanza is 3% higher than the average density (Table 14-11).

Table 14-11: Density data statistics by estimation domain.

Estimation Domain	Count	Length (m)	Mean (g/cm^3)	SD	CV	Median (g/cm^3)	Upper quartile (g/cm^3)	Maximum (g/cm^3)
Fraternal 1	97	13.03	2.74	0.15	0.06	2.73	2.79	3.53
Fraternal 2	5	0.54	2.69	0.01	0.01	2.68	2.70	2.70
Fraternal 3	3	0.78	2.66	0.02	0.01	2.67	2.67	2.67
Bonanza	37	5.26	2.82	0.25	0.09	2.74	2.82	3.72
Unknown	158	22.8	2.74	0.06	0.02	2.73	2.76	3.24
Total	300	42	2.75	0.13	0.05	2.73	2.76	3.72

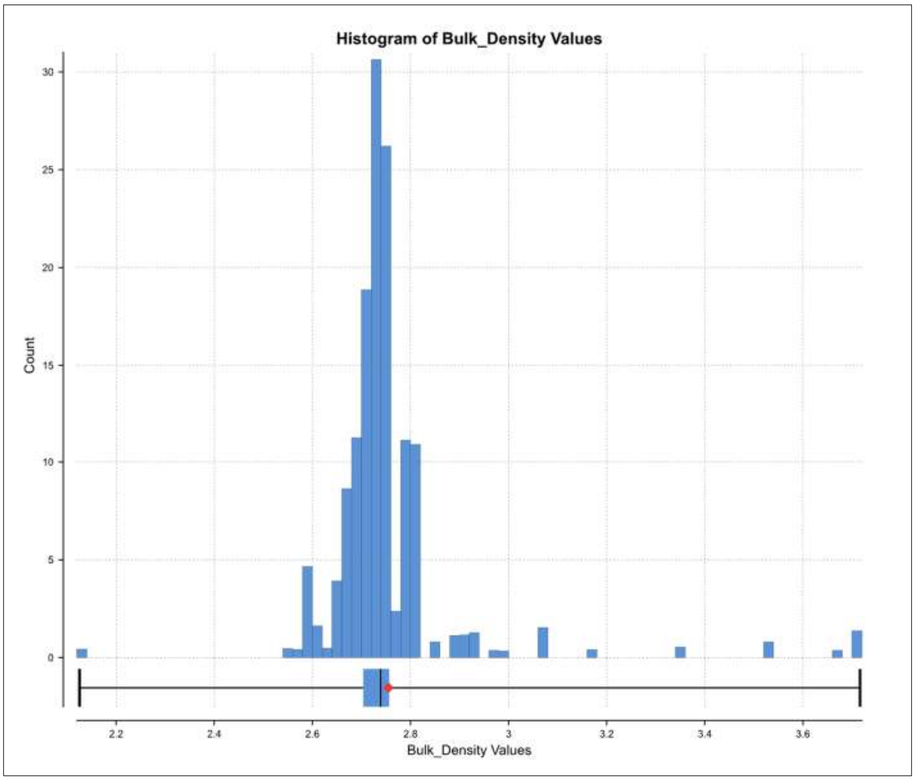


Figure 14-18: Histogram of density (Bonanza and Fraternal).

14.6 Validation

Block model grades were validated by comparing the input mean grades with the block model mean grade using swath plots and visually using cross-sections. The QP (Abraham Whaanga) considers the block model to be robustly estimated.

14.6.1 Global Mean Validation

The mean Au and Sb grade block model and composite comparisons are presented in Table 14-12 and Table 14-13.

Areas on the periphery of the Bonanza and Fraternal domains with wide-spaced drilling are at increased risk of overestimation of Sb grades without sample support. Sensitivity testing conducted without restricting the high-grade Sb population demonstrated that these areas of lower sample support are significantly overestimated, with the high-grade Sb spread throughout the domain.

Table 14-12: Auld Creek mean Au grade block model and composite comparisons.

Domain	Block Mean Au Grade (g/t)	Composite Mean Au Grade (g/t)	Declustered Composite Mean Au Grade (g/t)	Relative Difference between Block Grade & Declustered Composite Au Grade (%)
Bonanza LG	0.27	0.28	0.25	9%
Bonanza HG	1.89	2.81	2.09	-9%
Fraternal	1.82	2.43	1.88	-4%

Table 14-13: Auld Creek mean Sb grade block model and composite comparisons.

Domain	Block Mean Sb Grade (%)	Composite Mean Sb Grade (%)	Declustered Composite Mean Sb Grade (%)	Relative Difference Between Block Grades and Declustered Composite Sb Grade (%)
Bonanza	0.54	1.61	0.53	1%
Fraternal	0.69	0.83	0.65	6%

14.6.2 Swath Plot Validation

Block model Au and Sb grades were validated by comparing the declustered input mean composite grades with the block model mean grade from OK, nearest neighbour, and inverse distance estimates using swath plots supported by visual cross-section validation. These swath plots were generated for Au and Sb in the x and z directions and across strike in all estimation domains. Example swath plots are presented in Figure 14-19 to Figure 14-20. The Au plots indicate the estimation results are unbiased and appropriately smoothed, and that outliers did not lead to bias in areas of low sample support. The Sb indicator domain demonstrates non-stationarity, where the reconstituted Sb grade combining the proportions of mineralisation above and below the top-cut results in lower grades in areas of low sample support (Figure 14-19).

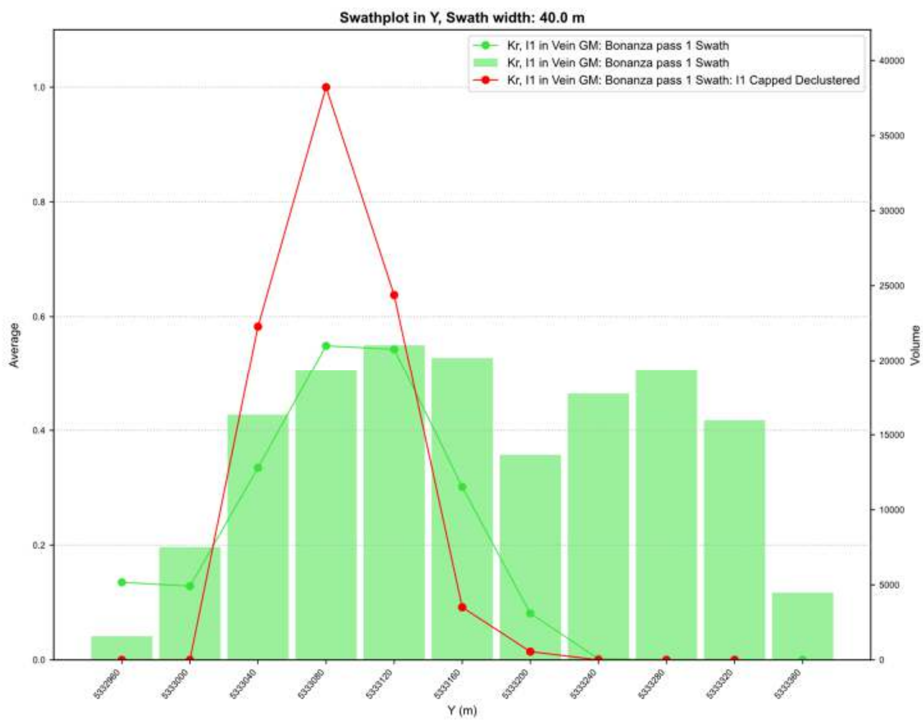


Figure 14-19 : Swath plot for Bonanza Sb indicator domain (y direction).

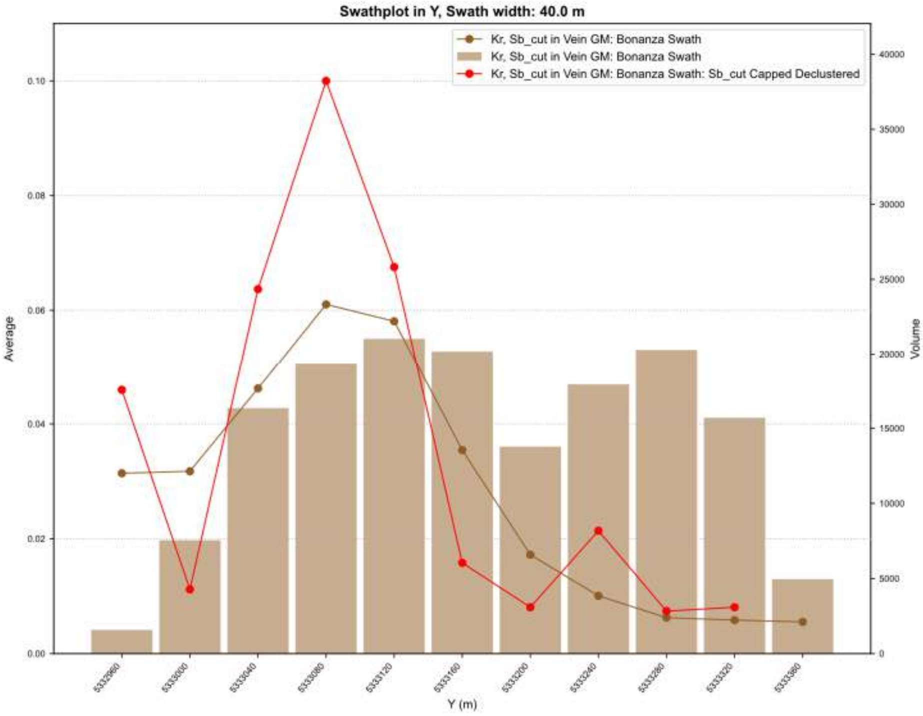


Figure 14-20 : Swath plot for Bonanza <1% Sb (y direction).

14.6.3 Visual Validation

Visual validation along cross-sections demonstrated good correlation between the input grade and OK estimated block grades (Figure 14-21 and Figure 14-22). As expected from the smoothing effect of OK estimation, fluctuations between zones of internal dilution and zones of higher-grade mineralisation are attenuated in the smoothed block grade profiles. Some drillholes presented in the visual validations are off-plane due to deviation and may not spatially align with block grades.

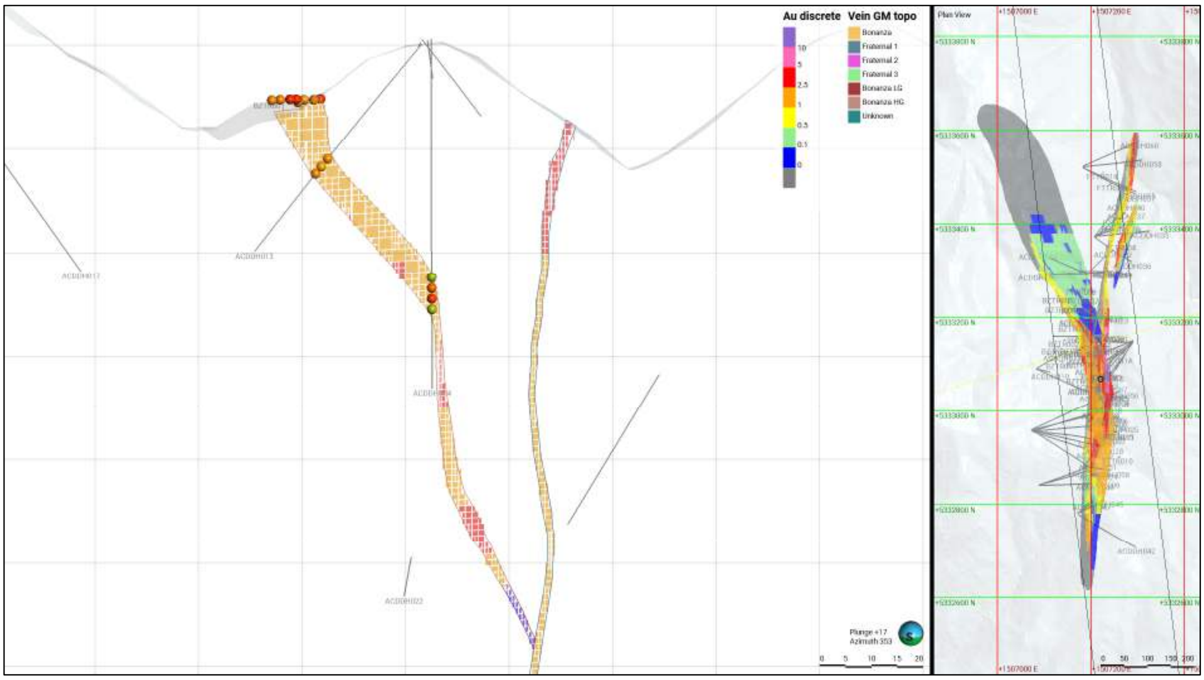


Figure 14-21: Bonanza (left) and Fraternal (right) section and plan views illustrating the estimated block model Au and 2-m Au composite (looking northwest).

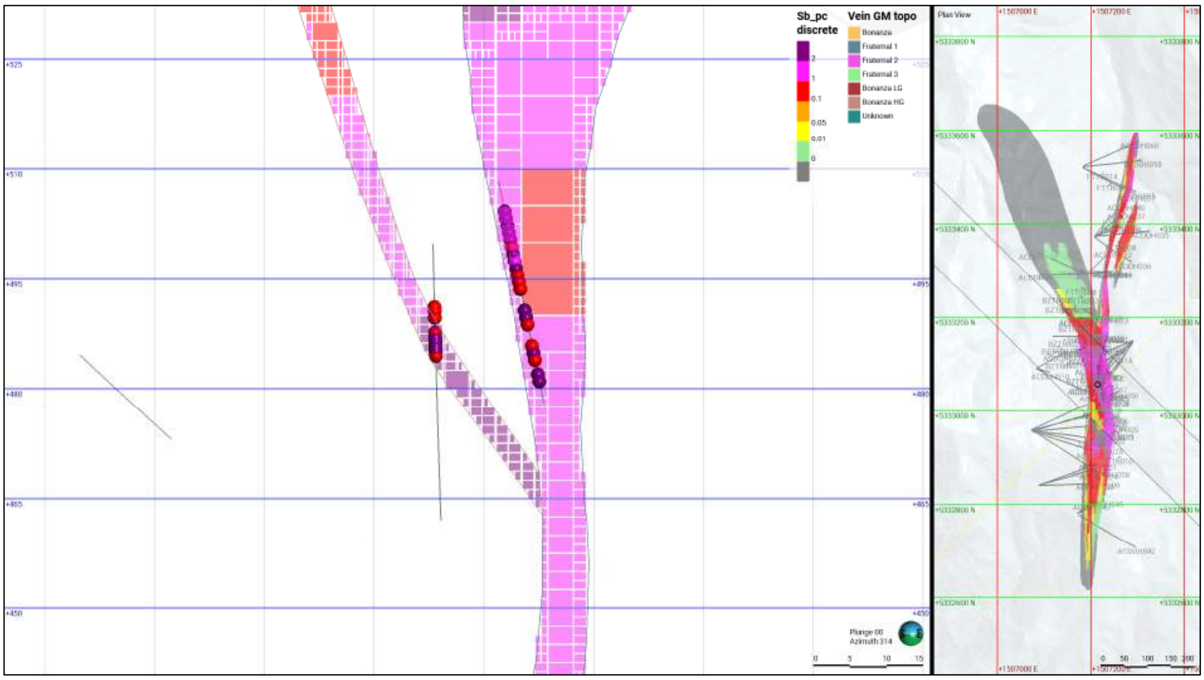


Figure 14-22: Bonanza (left) and Fraternal (right) section and plan views illustrating the estimated block model Sb and 2-m Sb composite (looking northwest).

14.7 Sensitivity Testing

The following five methods were used to assess the sensitivity of the OK estimate to the input parameters:

1. Adjusting the maximum number of samples in the estimation parameters.
2. Adjusting the search parameters to a percentage of the variogram range.
3. Adjusting the maximum number of samples allowed per drillhole in the estimation parameters.
4. Estimating with 1-m composites using the same estimation parameters with updated variograms and top-cuts.
5. Creating a non-linear estimate using an indicator interpolant at the top-cut level (0.1% Sb).

14.7.1 Estimating with Different Numbers of Samples

Kriging neighbourhood analysis (KNA) was conducted to determine the maximum number of samples per estimate (in the end 25 was selected), maximising the slope of regression (SoR) and kriging efficiency (KE) while reducing the sum of negative weights in the estimate (SumN) (Figure 14-23).

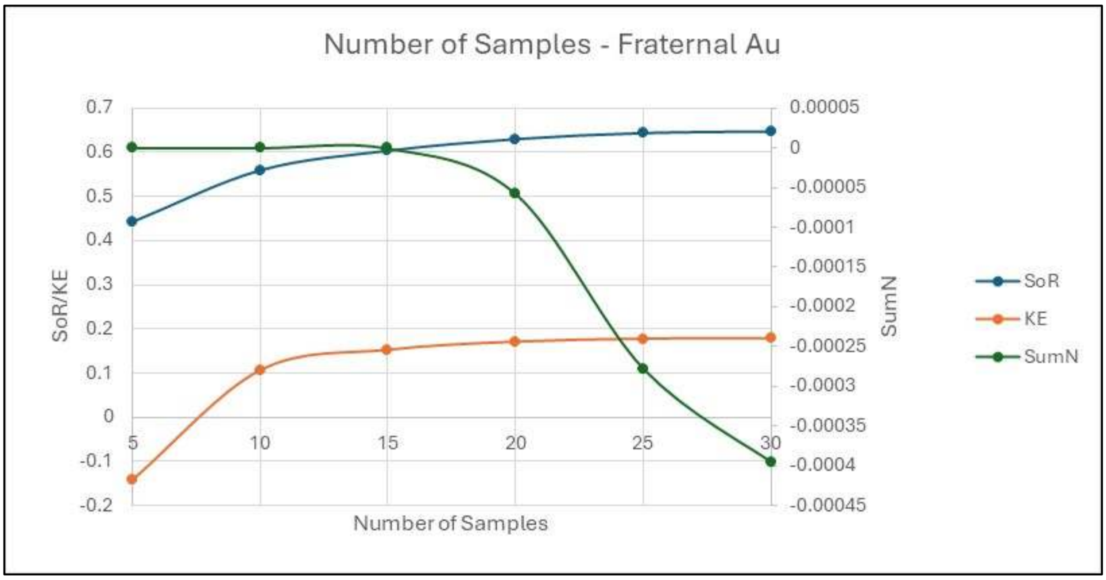


Figure 14-23: Fraternal Au: maximum number of samples per estimate.

14.7.2 Estimating with Different Ellipsoid Search Ranges

KNA was conducted to help validate the optimum search range, maximising the SoR and KE while reducing the SumN (Table 14-14 to Table 14-15 and Figure 14-24).

Table 14-14: Fraternal Au search ranges.

Percentage	Major Search Range (m)	Semi-Major Search Range (m)
100%	115	80
80%	90	65
70%	80	60
60%	70	50
40%	45	35

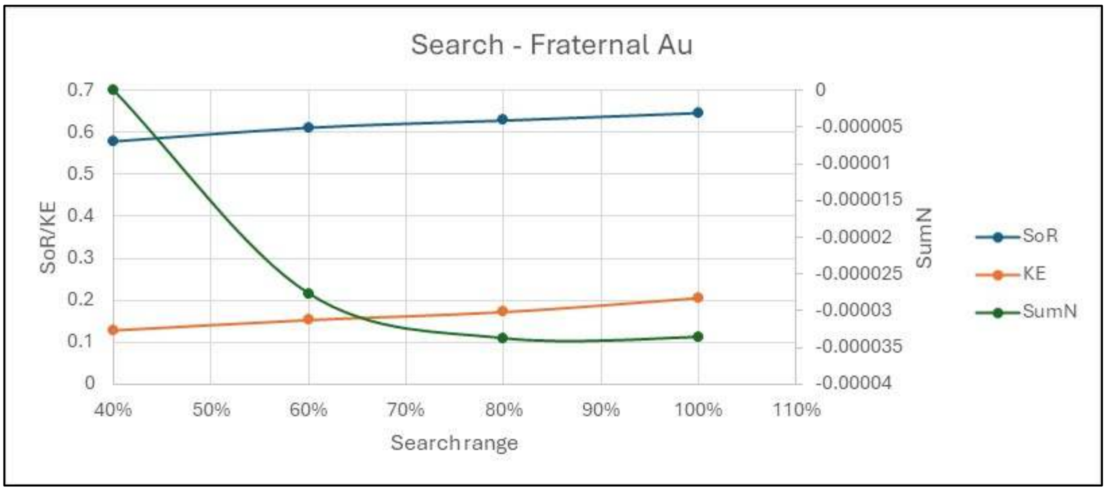


Figure 14-24: Fraternal Au search ranges.

Table 14-15: Bonanza Au search ranges.

Percentage	Major Search Range (m)	Semi-Major Search Range (m)
100%	140	70
80%	110	55
60%	85	40
40%	55	30

Table 14-16: Fraternal Sb search ranges.

Percentage	Major Search Range (m)	Semi-Major Search Range (m)
100%	145	60
80%	115	50
60%	90	40
40%	60	25

Table 14-17: Bonanza Sb search ranges.

Percentage	Major Search Range (m)	Semi-Major Search Range (m)
100%	140	75
80%	115	60
60%	85	45
40%	55	30

14.7.3 Estimating with Different Maximum Numbers of Samples Allowed per Drillhole

The OK estimation sensitivity was tested after first setting the search ranges and maximum number of samples per drillhole. A minimum number of samples of five and a maximum number of samples per drillhole of three required two drillholes per block estimate. Both domains with unestimated blocks required a second pass with no restriction on the maximum number of samples per drillhole and the minimum number of samples per block estimate reduced to two.

14.7.4 Estimating Different Composite Lengths

The OK estimation of Au was tested using a 1-m sample compositing length, and the final grades were compared for all domains. The comparison resulted in changes to the mean grade of up 7% in the selected estimation domains, demonstrating moderate sensitivity to compositing selection (Table 14-18 and Table 14-19).

Table 14-18: Auld Creek Au sensitivity analysis comparing different compositing lengths.

Domain	1-m Composite Mean Declustered Grade (Au g/t)	2m Composite Mean Declustered Grade (Au g/t)	Relative Difference (%)
Bonanza	1.40	1.46	4%
Fraternal	1.88	1.88	0%

Table 14-19: Auld Creek Sb sensitivity analysis comparing different compositing lengths.

Domain	1-m Composite Mean Declustered Grade (Sb%)	2-m Composite Mean Declustered Grade (Sb%)	Relative Difference (%)
Bonanza	0.71	0.67	-7%
Fraternal	0.72	0.74	2%

14.7.5 Estimating with an Indicator Interpolant

An indicator interpolant at a 0.1% Sb cut-off was used to sub-domain the bimodal Sb grade population. The domains above and below the top-cut were estimated separately and then combined to produce the final Sb value (Figure 14-25 and Figure14-26).

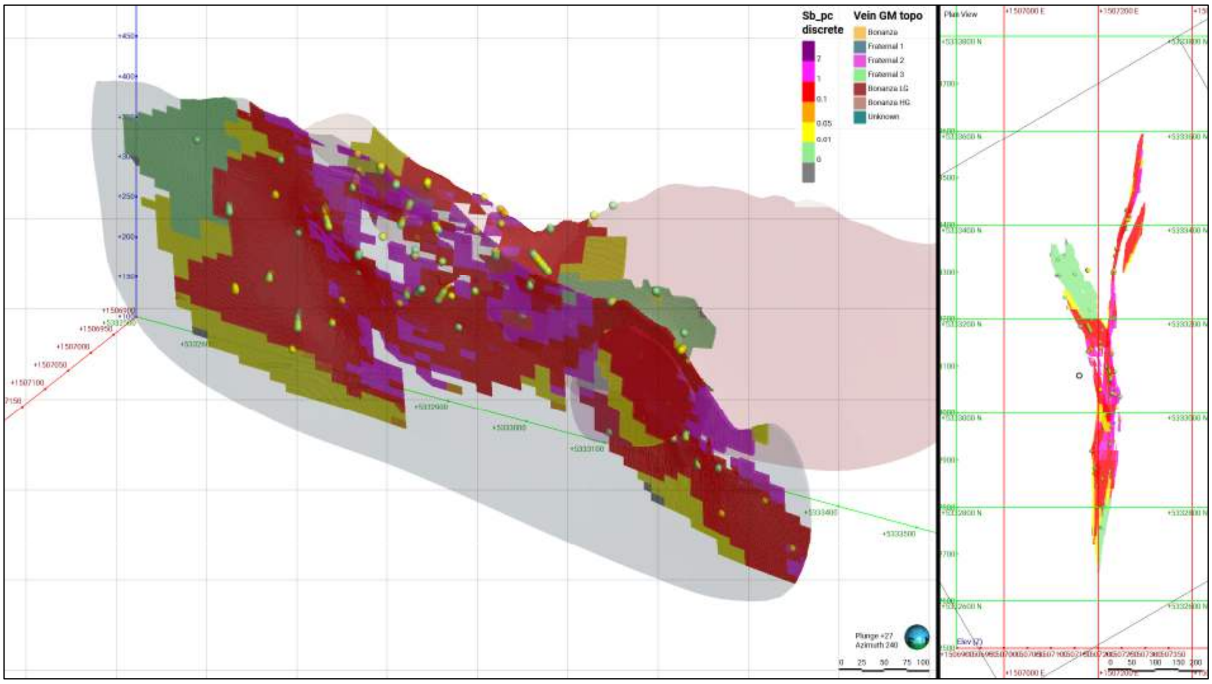


Figure 14-25: Block model illustrating the <0.1% Sb domain with 2-m composites.

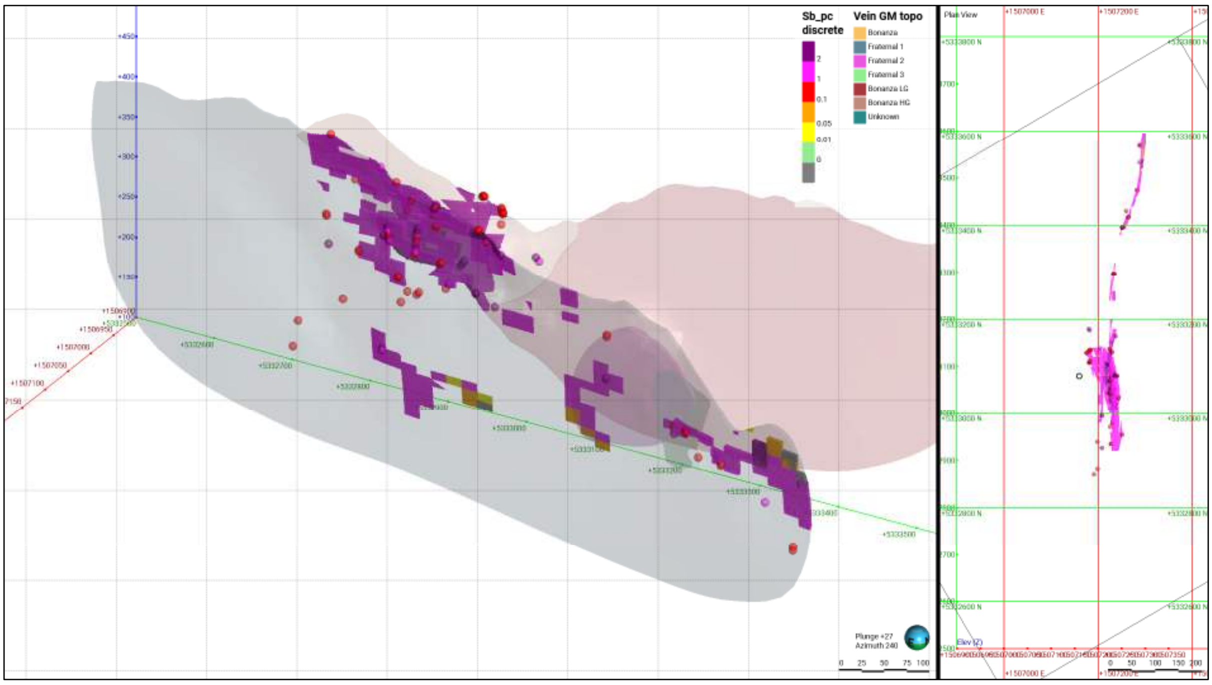


Figure14-26: Block model illustrating the >0.1% Sb domain with 2-m composites.

14.8 Depletion

There are no known historical mine workings in or around the Fraternal and Bonanza reefs; therefore, no depletion was applied.

14.9 Drillhole Spacing Analysis

Drillhole spacing analysis (DHSA) was conducted at Auld Creek to determine minimum drill spacing requirements to properly control, with a high degree of confidence, the estimation of grade and volumes to meet specific resource classification objectives. The drillhole spacings required to attain specific levels of global estimation precision were based on the Global Estimation Variance methodology (Postolski et al., 2014). This method combines elementary variances to assess the global level of precision achievable, when estimating in situ grades over volumes corresponding to set mining periods (RSC considered 6 months, 1 year, and 2.5 years at Auld Creek).

14.9.1 Global Estimation Variance Methodology

The Global Estimation Variance methodology, summarised by Postolski et al. (2014), can lead to very robust first-pass answers that may better qualify the link between drilling density and estimation quality, according to the geological grade continuity of the deposit. The methodology assumes that the sampling grid is regular (with mesh v) and covers exactly volume V (corresponding to a fixed mining period) as illustrated in Figure 14-27.

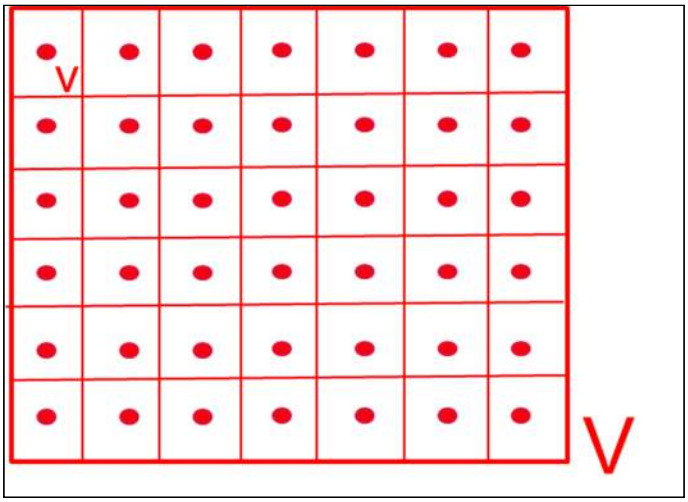


Figure 14-27: Systematic sampling of extraction volume V, by a regular sampling grid v.

Given that s_e^2 is the elementary variance of estimation for the grade of interest over each elementary volume v from its central sample (easily obtained once a variogram has been modelled in the relevant estimation domain for the variable of interest), and s_E^2 the variance of estimation of the grade over volume V as the mean of the samples contained in V, it can be demonstrated (Postolski et al., 2014) that: $s_E^2 = \frac{s_e^2}{\mathrm{N}}$.

The knowledge of s_E^2 allows the characterisation of the coefficient of variation of the sampled mean $\frac{s_E}{\mathrm{m}}$ over V, which can then be set to assure targeted confidence intervals are met.

If, for example, the global mean is required to remain within ±15% over a 1-year period, at a 90% confidence level, and if a Gaussian distribution for the global mean is assumed, then the 90% confidence interval for the estimation of the global mean can be approximated by 3.3 standard deviations, and so $\frac{s_E}{\mathrm{m}}$ must be bound by $\frac{30\%}{3.3}$, which is approximately 9.1%. The spacing, ensuring that $\frac{s_E}{\mathrm{m}}$ remains below that level of 9.1% over a 1-year period, will be a suitable candidate for that variable-domain combination to reach the target confidence intervals compatible with the definition of Indicated Mineral Resources.

14.9.2 Results

The methodology was applied assuming yearly productions of 275–300 kt per year. The variogram models used are the ones established for the MRE of the key variables (Au, Sb, Domain indicator). A minimum drill spacing of 50 m seems required to properly control the estimation of Au to within ±30% over a 2.5-year period, this period corresponds to the Inferred category and the life of the Project. A minimum drill spacing of 20 m seems required to properly control the estimation of Au to within ±15% over a 1-year period, this period corresponds to the Indicated category and 1 year of production.

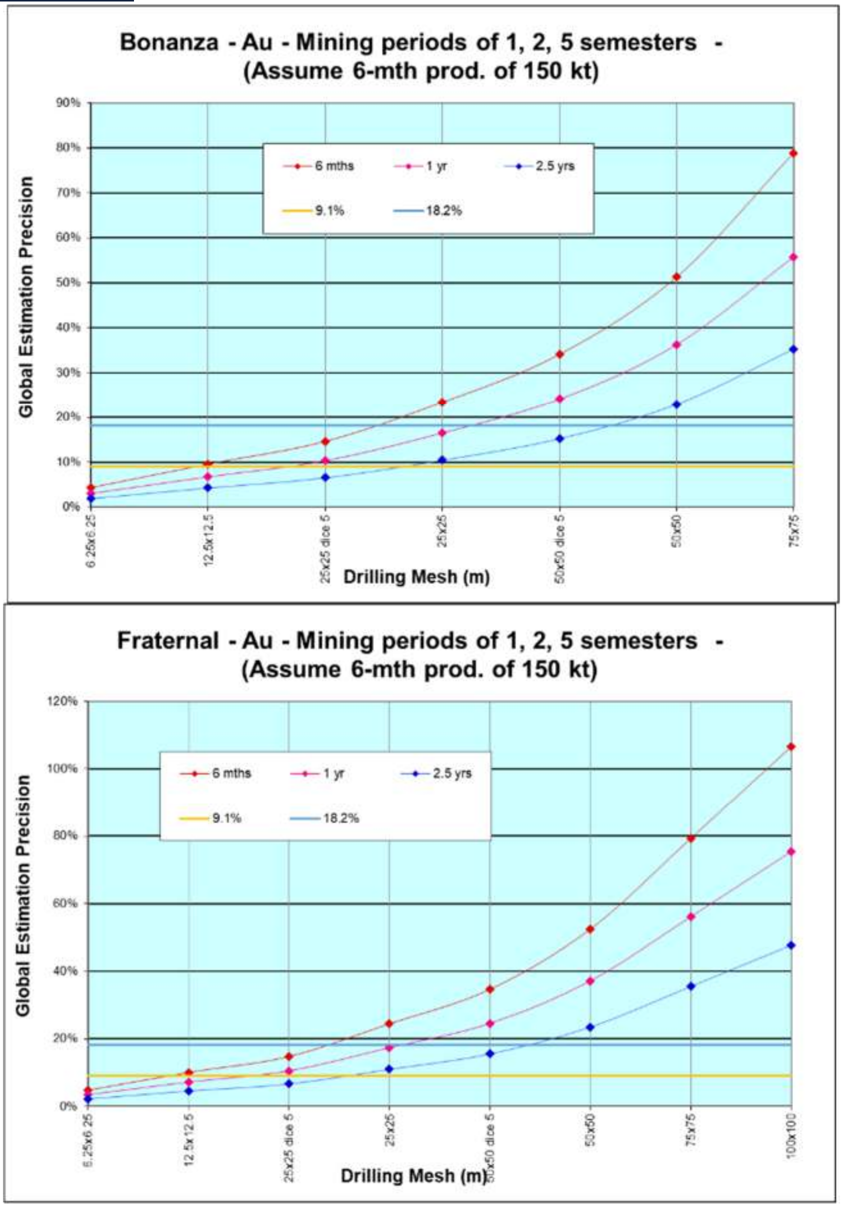


Figure 14-28: Global estimation variance methodology applied to Au for the Bonanza domain (left) and Fraternal domain (right).

Following the results of above analysis of drill spacing, and given the consistency of the applied drill grid, the QP (Abraham Whaanga) has based the resource classification at Auld Creek on an unconstrained estimate to determine the Cartesian average distance to the nearest three drillholes using one sample per drillhole, known as the three-hole rule (Table 14-20). To apply a consistent drilling infill grid pattern from Inferred to Indicated, the QP (Abraham Whaanga) has chosen 25 m as the minimum spacing requirement for Indicated. That spacing implies a lower level of confidence of meeting the classification than the tighter 20-m spacing requirement defined by DHSA. Consequently, the QP applied a conservative post processing step to retain the better informed areas in the Indicated portion (Figure 14-29).

Table 14-20: Characterisation of local estimation precision.

Classification	Three-Hole Distance
Indicated	12 m to less than 25 m
Inferred	25 m to less than 50 m

The QP (Abraham Whaanga) post processed the resulting classification by assessing local geological and grade continuity to refine the resulting classification resolution and eliminate artefacts potentially created by the three-hole rule (Figure 14-29).

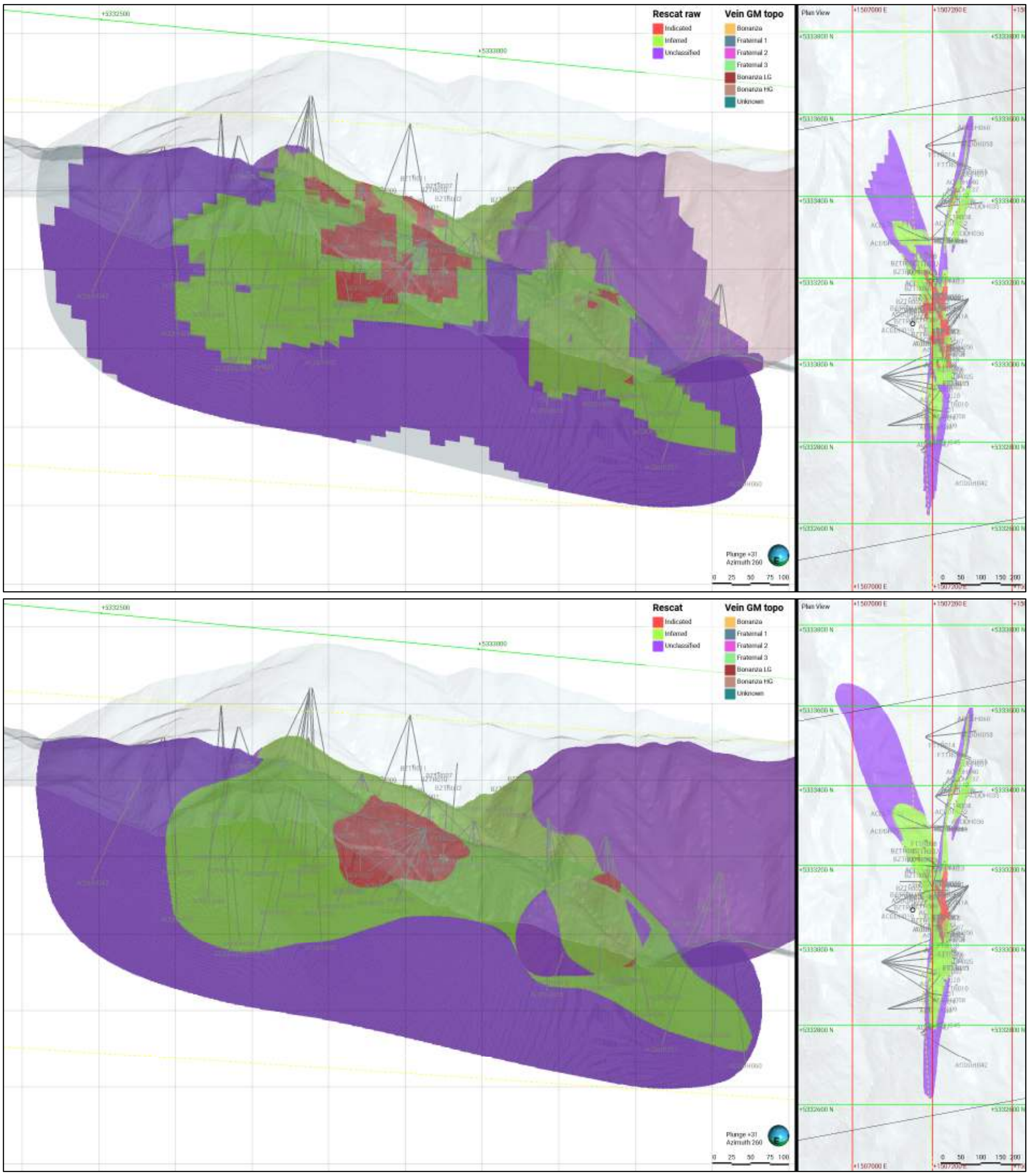


Figure 14-29: Resource classification defined by the three-hole rule (top) and post processed (bottom).

The QP (Abraham Whaanga) reviewed block model statistics by classification using the slope of regression (SoR), kriging efficiency (KE), number of samples per estimate (NS), and number of drillholes per estimate (NDh) (Figure 14-30), with higher numbers indicating a higher quality estimate. In the QP's opinion, the increase in estimation quality supports the final classification.

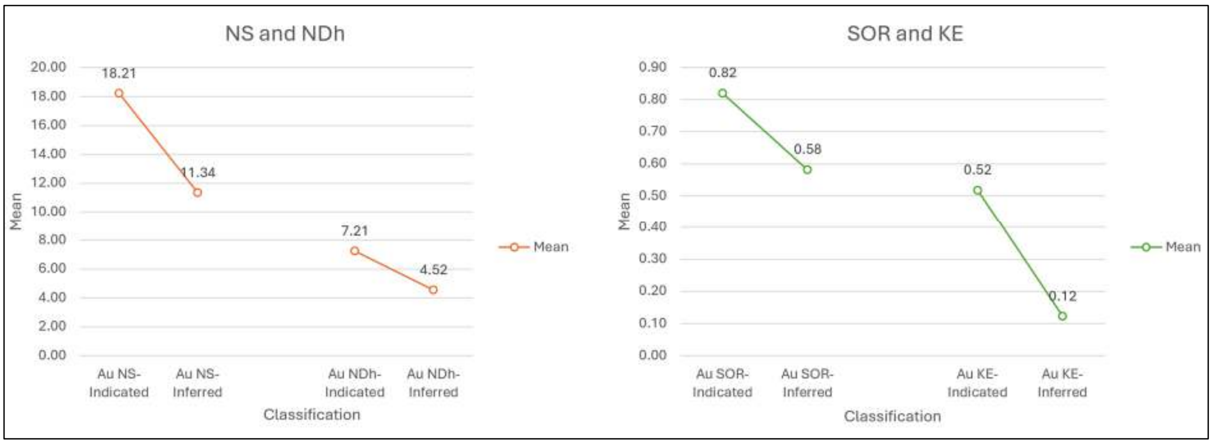


Figure 14-30: Block model statistics by classification.

14.10 Mineral Resource Classification

The QP (Abraham Whaanga) has classified the Mineral Resource in the Inferred and Indicated categories in accordance with NI 43-101 and the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014) (Table 14-21). For the Indicated Mineral Resources, geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. For the Inferred Mineral Resources, geological evidence is sufficient to imply but not verify geological and grade continuity. The Mineral Resource is based on exploration, sampling, and assaying information gathered through appropriate techniques from trenches and drillholes.

It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued drilling. For the Inferred Mineral Resource portion of the MRE, confidence in the estimate is not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning in pre-feasibility or feasibility studies.

Cautionary Statement: the PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA will be realised.

Future work should seek to decrease the drillhole spacing, improve sample and analytical quality control, and improve the resolution of the Au and Sb estimation domains.

The QP (Gary Davison) refined the mineable shapes by:

- removing shapes manually which were isolated and not near the bulk of the deposit (i.e. not economical to develop); and
- excluding material contained within 20 m of the surface (crown pillar recommended by the Mining One geotechnical engineers).

Table 14-21 presents the final MRE after exclusion of the crown pillar and the manually removed shapes. In the base case, the QP (Gary Davison) has assumed that the crown pillar can be mostly extracted.

Table 14-21: MRE for the Auld Creek deposit (effective date 19 June 2026) with updated cut-off grade and commodity pricing.

MSO	Resource category	Tonnes (Mt)	Au (g/t)	Sb (%)	AuEq (g/t)	Contained Au (koz)	Contained Sb (kt)	Contained AuEq (koz)
RAW MSO	Indicated	0.35	3.0	1.0	5.2	34	4	58
	Inferred	1.35	1.8	0.7	3.4	79	10	148
20 m Crown	Indicated	0.03	5.1	1.1	7.4	6	0	8
	Inferred	0.16	1.9	0.8	3.5	9	1	18
Final MSO Set	**Indicated**	**0.28**	**2.5**	**1.0**	**4.6**	**23**	**3**	**42**
	Inferred	**0.98**	**1.9**	**0.7**	**3.5**	**59**	**7**	**109**

Notes:

1. *The Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves (May 2014) were used for the mineral resource estimate.*
2. *The Mineral Resource is reported at a cut-off of 1.5 g/t AuEq.*
3. *Metal-equivalent grades were calculated using the following formula AuEq = Au g/t + 2.15 × Sb%.*
4. *The AuEq factor of 2.15 is calculated using the following prices: USD 3,300/oz Au, and USD 27,000/t Sb. Metallurgical recoveries of 95% Au and 85% Sb, with 95% Au and 90% Sb payable in concentrate, were used, where C = payable in concentrate R = metallurgical recovery and P = price.*

$$AuEq\ factor = \frac{(P_{Sb} \div 100) \times C_{Sb} \times R_{Sb}}{(P_{Au} \div 31.10348) \times C_{Au} \times R_{Au}}$$

5. *MSO parameters are outlined in the Report.*
6. *Totals may vary due to rounding.*
7. *Shapes were removed manually if they were <1.8 g/tAuEq and their LEVEL attribute had <5,000 t (not economical to develop).*
8. *Crown pillar was assumed to be ~75% extractable, needs further work at the PFS level to minimise surface impact.*
9. *The Preliminary Economic Assessment (PEA) presented in this Report is based on the Mineral Resources reported within the Project and is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the PEA will be realised.*
10. *The QP (Abraham Whaanga) is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues or any other relevant issue that could materially affect the MRE.*

14.11 Cut-Off Grade and Reasonable Prospects for Eventual Economic Extraction

The QP (Gary Davison) noted the key block model fields that will be essential in guiding the mine design and schedule (Table 14-22).

Table 14-22: Block model fields.

BM Field	Meaning	UoM
AuEq	Gold equivalent (estimated by RSC)	g/t
Density	In situ nulk density	t/m^3
Au	Gold grade	g/t
Sb	Stibnite grade	%
Rescat	Mineral Resource classification	Indicated, Inferred, or Unclassified

Grade-tonnage curves were generated for the three grade fields in the block model to understand potential resource. These are captured in Figure 14-31 to Figure 14-33.

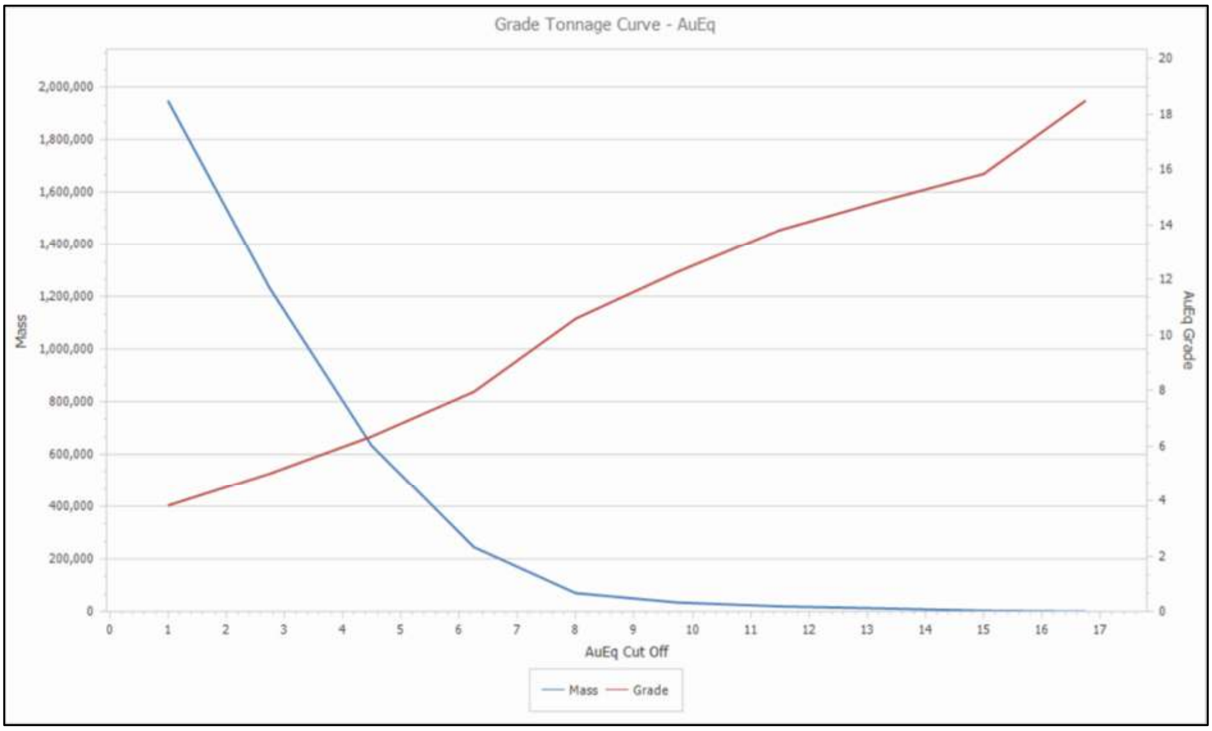


Figure 14-31: Grade-tonnage curve for AuEq.

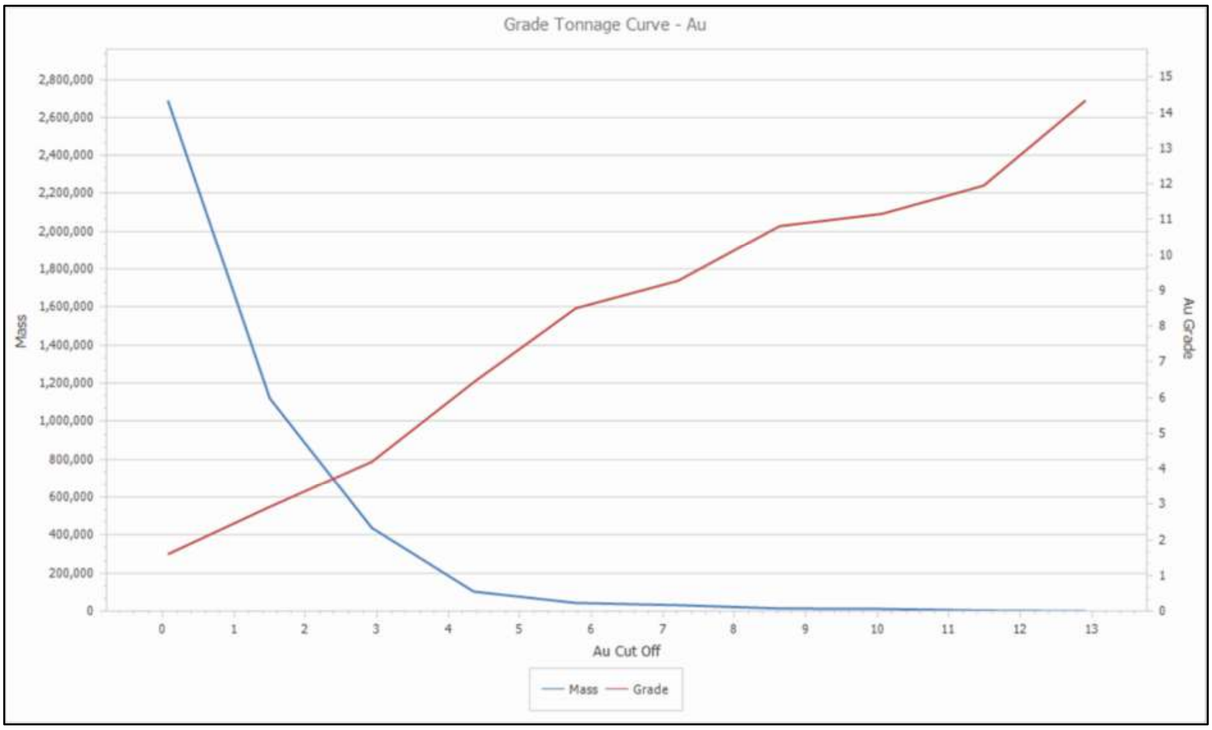


Figure 14-32: Grade-tonnage curve for Au.

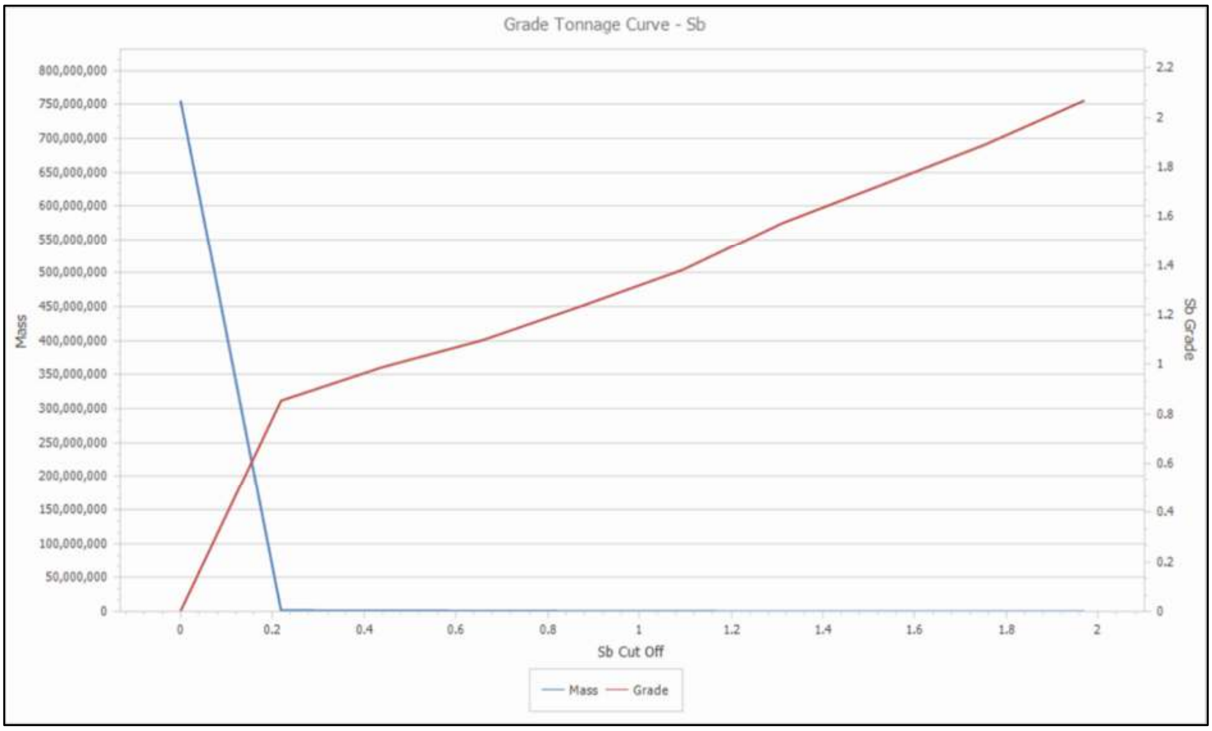


Figure 14-33: Grade-tonnage curve for Sb.

If a cut-off-grade of 1.0 g/t AuEq were chosen (Figure 14-31), the resultant Mineral Resource could be as much as 1.9 Mt of material at 4.0 g/t AuEq. This provides a benchmark for the success of the MSO run. Note that the grade-tonnage curves in Figure 14-31 to Figure 14-33 contain a portion (~10%) of Unclassified material. This was excluded from the mine schedule and economic model and represents a potential exploration target.

14.11.1 High-Level Mining Method Selection

The QP (Gary Davison) assessed mining method suitability based on deposit grade, width, and strike. A range of technical constraints were also considered, such as the requirement for minimal surface disturbance and the desired portal locations. Figure 14-34, which depicts a grade shell generated above 1.5 g/t AuEq, illustrates that the deposit is ~350 m below surface.

Potential mining methods are summarised in Table 14-23. From this analysis, The QP (Gary Davison) concludes that sub-level open stoping with cemented fill and OHCAF are likely the best options for the mine. Some potential exists for handheld mining, particularly in the crown pillar areas of the deposit.

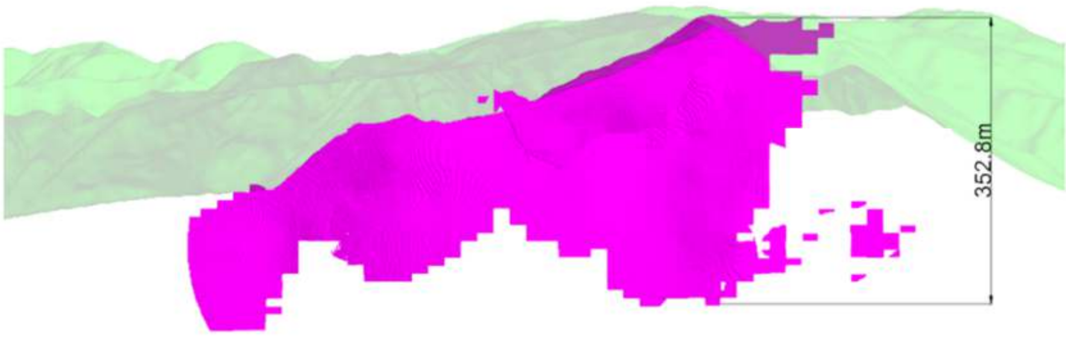


Figure 14-34: 1.5 g/t AuEq grade shell.

Table 14-23: High-level analysis of mining methods amenable to the Auld Creek deposit.

Potential Method	Applicable?	Reason
Sub-Level Open Stoping (SLOS)	YES	Existing operations similar (Costerfield, VIC). Suitable orebody. Pending Geotech analysis.
Overhand Cut-And-Fill (OHCAF)	YES	Selective mining. Disposal of waste. Limited excavation size.
Underhand Cut-And-Fill (UHCAF)	NO	Mining below cemented fill may require high cement content in fill Good selectivity. High unit-cost.
Caving Options (SLC, BC)	NO	Narrow, tabular orebody. Regional disturbance in DOC land.
Avoca Stoping	NO	Lower cost than C&F methods / methods involving cement. High capital cost for dual access (2 declines).
Modified Avoca (Single Access)	MAYBE	Lower cost. Tight firing. Potential for substantial dilution.
Shrinkage Stoping	NO	Top-bottom access for stopes not favourable owing to single access drives. Not industry best-practice for safety.
Handheld Development & Rise Mining (Airleg)	MAYBE	Highly selective and flexible mining. Permitting and SLO could be problematic (Risk assessment). Poor ground conditions / safety.

14.11.2 First-Pass Cut-Off-Grade

The QP (Gary Davison) used metal prices and cost ranges (low and high) to calculate a preliminary range of cut-off grades (COGs). The commodity prices and costs used are summarised in Table 14-24.

Table 14-24: Summary of COG inputs.

Cost Assumption	High[1]	Low[2]	UoM
Processing Cost	49	32	USD/t
Mining Cost	297	130	USD/t
G&A Cost	14	7	USD/t
Sustaining Capital	10	7	USD/t
Gold Price	5,000	3,300	USD/oz

Notes:

1. *High end costs are based on a comparable narrow-vein Au mine in Victoria, Aus, which RUA has been using as a basis of estimate prior to engaging Mining One.*
2. *Low end costs are based on a) processing cost first-pass estimate from PBG of 46 AUD/t (Converted to 32 USD/t), and b) previous Mining One studies on a similar mine.*

Resultant COGs are presented in Table 14-25. The QP (Gary Davison), alongside RUA management, mutually agreed to use a COG of 1.5 g/t AuEq for this PEA.

The selected COG represents a reasonable mid-point within the range of cost-derived values, with preference given to lower cost assumptions considered more representative of the anticipated operating conditions based on comparable operations within a similar geographic setting. The adoption of a relatively lower COG also facilitates the inclusion of a broader suite of MSO shapes, supporting a more comprehensive assessment of the mineralised envelope and exploration potential.

Importantly, this approach provides flexibility for subsequent optimisation, whereby lower-grade shapes may be selectively excluded to evaluate the sensitivity of the mine design and project economics to varying COG assumptions.

Table 14-25: First-pass COG calculation.

COG (AuEq g/t)	Costs (Low)	Costs (High)
Low Au Price USD 3,300/oz	1.90	3.62
High Au Price USD 5,000/oz	1.25	2.39

14.11.3 Mineable Shape Optimiser

Many MSO runs were undertaken to understand the sensitivity of the block model to geometrical constraints or COG variations. The final MSO inputs are summarised in Table 14-26 for UHCAF and Table 14-27 for SLOS.

Table 14-26: MSO parameters – OHCAF.

Key MSO Input	Amount	UoM
Length	10	m
Height (level interval)	3.5	m
Maximum Mining Width	3.0	m
Minimum Mining Width	2.9	m
HW Dilution (ELOS)	0	m
FW Dilution (ELOS)	0	m
COG	1.5	g/t AuEq

Table 14-27: MSO parameters – SLOS.

Key MSO Input	Amount	UoM
Length	10	m
Height (level interval)	10	m
Maximum Mining Width	10	m
Minimum Mining Width	0.9	m
HW Dilution (ELOS)	0.3	m
FW Dilution (ELOS)	0.3	m
COG	1.5	g/t AuEq

The resultant MSOs yielded the tonnes and grade profiles captured in Table 14-28. The drop in equivalent grade, when compared to the block model grade-tonnage curves, is explained by the application of dilution (SLOS) and 3 m minimum mining width (OHCAF).

Table 14-28: MSO outputs.

MSO Scenario	Tonnes (kt)	Grade (AuEq)	Equivalent Ounces (oz)
UHCAF	1,735	3.70	206,352
SLOS	1,938	3.56	222,012

14.11.4 Sensitivity to Cut-Off-Grade

The QP (Gary Davison) varied COG from 1.5–3.65 g/t to generate Table 14-29, which illustrates the potentially mineable inventory sensitivity to COG variation.

Table 14-29: MSO sensitivity to COG (based on 10 mW × 10 mH stope shapes).

MSO Scenario (COG)	MSO (Mt)	Grade (AuEq)	Equivalent Ounces (koz)
1.5	1,9	3.56	222
2.0	1,7	3.93	210
2.5	1,4	4.33	192
3.0	1,1	4.76	172
3.65	0.9	5.31	146

14.11.5 Markets

See section 19 for details on market analysis.

14.11.6 Comparison with Historical Estimate

A summary of changes comparing the current MRE for Auld Creek with the Auld Creek historical estimate, dated 30 October 2024 and discussed in Section 6.4.2 of this Report, is displayed in Table 14-30. In addition, a portion of the historical estimate in the Inferred Mineral Resource category has been upgraded to the Indicated Mineral Resource category in the current MRE.

A total of 44 drillholes (9,190 m) have been drilled since the October 2024 historical estimate and one trench for 11 m was added to the interpretation. This drilling has increased the vertical extent of the Fraternal shoot by 120 m, from 160 m to 280 m, and the strike extent from ~350 m to almost 1,000 m (Figure 14-35).

Table 14-30: Summary of changes from 30 October 2024 historical estimate to the current MRE.

Classification	Change in Tonnes (Mt)	Change in Tonnes (%)	Change in Contained Au Ounces (koz)	Change in Contained Au Ounces (%)	Change in Contained Sb (kt)	Change in Contained Sb (%)	Change in Contained AuEq (koz)	Change in Contained AuEq (%)
Indicated	0.3	100	30.5	100	3.8	100	57.1	100
Inferred	0.6	79	12.4	18	3.2	40	46.6	42

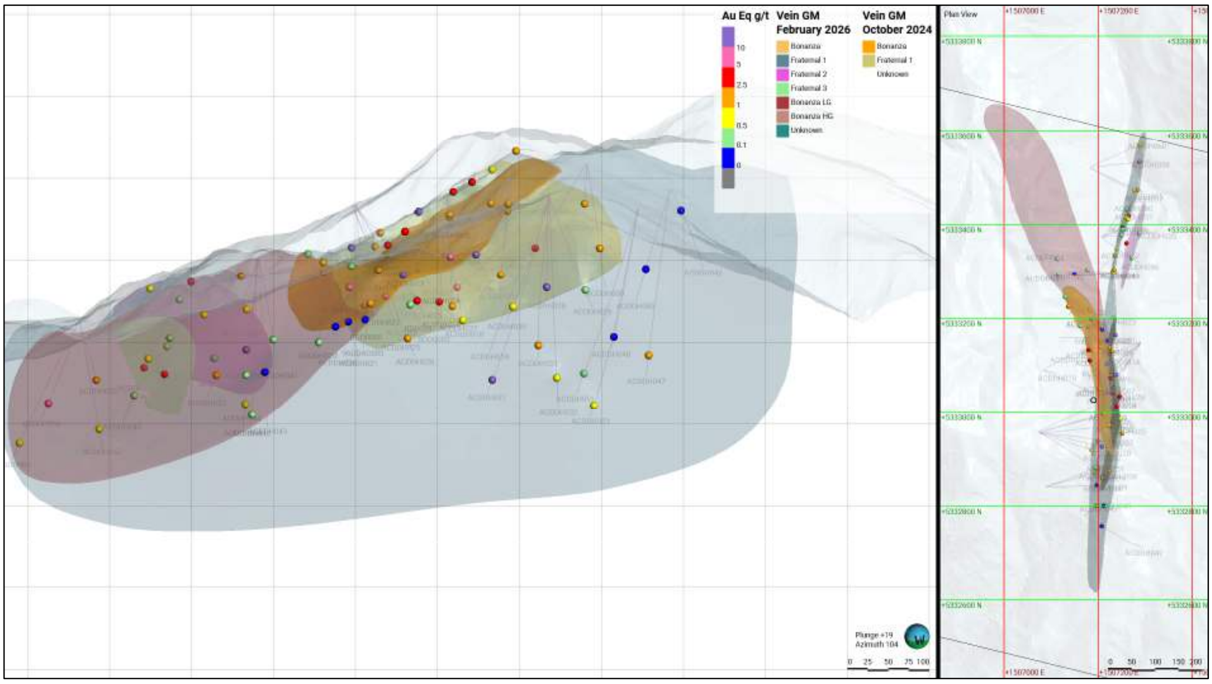


Figure 14-35: Long-section looking east, illustrating the increase to the vertical and strike extent of Fraternal (full vein width composites coloured by g/t AuEq).

The DHSA has tightened the resource classification from a drillhole spacing of ~100 m × 50 m for Inferred to 50 m × 50 m and defined a drillhole spacing of 25 m × 25 m as a minimum requirement for Indicated.

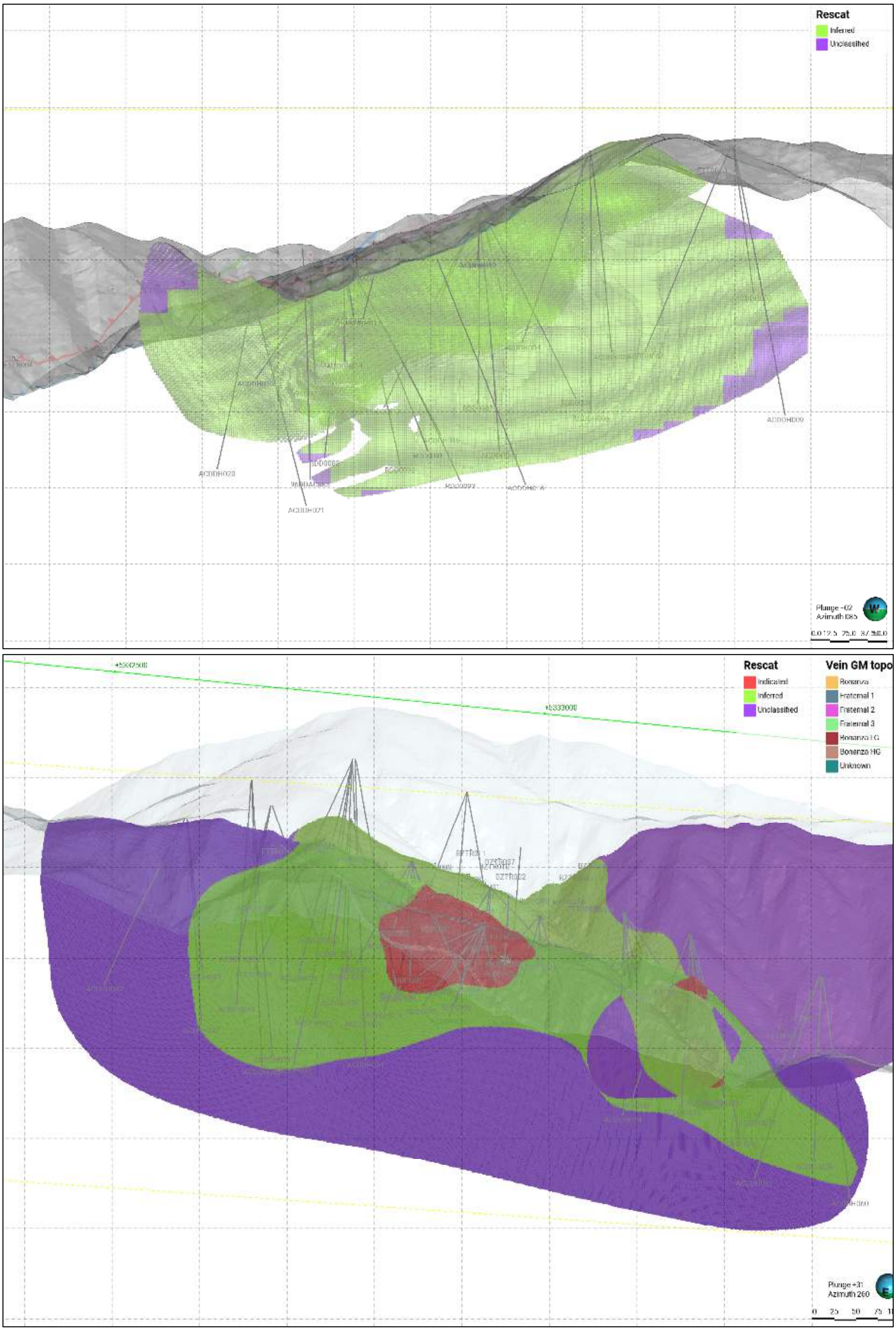


Figure 14-36: Mineral Resource classification – October 2024 historical estimate (top), current February 2026 estimate (bottom).

14.12 Risks

In line with best practice, the QP (Abraham Whaanga) undertook a risk assessment for the Project. RSC's risk assessment considers the availability of data and gives a performance scorecard and risk rating (Table 14-31). RSC's risk score matrix is given in Figure 14-37. The risks involved in the modelling and estimations for all prospects are summarised in Table 14-32. The most pertinent risks have also been noted throughout this Report.

Table 14-31: Guide to the rating system used in this Report.

Availability of Data	
Absent	Entirely absent
Poor	Incomplete MS Excel/export files, no metadata
	Briefly described in report
Average	Basic MS Excel/export files, limited metadata
	Briefly described in report
Good	Advanced MS Excel/export files
	Well described in report and supporting appendices available
Excellent	Industry best practice SQL or MS Access database
	Well described and supported by extensive SOPs

Performance Score Card	
0	Complete failure or erroneous
0–3	Largely incorrect
3–5	Largely correct
5–8	Correctly undertaken and industry standard
8–10	Exceeds industry standard and is best practice

Impact Score	
1–Low	Low impact with respect to the purpose (Inferred, Indicated, or Measured resources, or to Probable or Proven reserves) or deposit (bulk commodity or orogenic gold, etc.)
2	Low–medium impact
3	Medium impact
4	Medium–high impact
5–High	High impact with respect to the purpose (Inferred, Indicated, or Measured resources, or Probable or Proven reserves) or deposit (bulk commodity or orogenic gold, etc.)

Risk Factor	
None	No risk to resource or project
Low	Low risk to Mineral Resource, Ore reserves classification. Can be interpreted to read: a low possibility of a negative outcome or loss[2].
Moderate	Moderate risk to Mineral Resource, Ore Reserves classification. Can be interpreted to read: a moderate possibility of a negative outcome of loss[1]. There may be some residual subjectivity.

[2] *Negative outcome of loss:*

a) *eventual mine-to-mill reconciliation outside the company's defined tolerances; or*
b) *a fatal flaw that would objectively prevent classification in the target category.*

Risk Factor	
High	High risk to Mineral Resource, Ore Reserves classification. Can be interpreted to read: a high possibility of a negative outcome or loss[1].
Extreme	Extreme risk to Mineral Resource, Ore Reserves classification. Can be interpreted to read: an extreme likelihood of a negative outcome or loss[1].

		Impact				
		1	2	3	4	5
Performance Score	1	10	20	30	40	50
	2	9	18	27	36	45
	3	8	16	24	32	40
	4	7	14	21	28	35
	5	6	12	18	24	30
	6	5	10	15	20	25
	7	4	8	12	16	20
	8	3	6	9	12	15
	9	2	4	6	8	10
	10	1	2	3	4	5

Project Risk: Performance vs Impact Score			
1-9	Low	20-24	High
10-12	Low-Medium	25-28	High-Extreme
13-15	Medium	29+	Extreme
16-19	Medium-High		

Figure 14-37: RSC's risk score matrix.

Table 14-32: Overview of risk factors impacting the MREs.

Item	Data/Info Availability	Score (1–10)	Impact Factor (1–5)	Risk Factor	Comments
Database Format	Good	8	3	Moderate	The Project has a relational database. Data were collated by RRPL geologists from digital drillhole logging files into MS Excel. Data collected by RRPL were comprehensive and semi-validated at the point of collection. Checks completed by the QP uncovered several errors in the MS Excel workbooks provided by RRPL, which were corrected.
Drilling and Primary Sampling Techniques	Good	8	1	Low	Sampling was nominally 1 m; composite widths were adjusted for veins and contacts. Drilling was a mixture of PQ and HQ, using triple-tube techniques. An SOP detailing the drilling of diamond core was available for review. The SOP briefly covered aspects of logistics, preparation, safety in relation to the drilling campaign, downhole surveying, and core recovery. However, it did not note the minimum recovery required or provide guidance for dealing with low recoveries.
Drilling and Primary Sampling Recovery	Good	8	1	Low	Diamond drill sample recovery averaged ~95% across the prospects and core sizes. No correlation between recovery and grade was determined.

Item	Data/Info Availability	Score (1–10)	Impact Factor (1–5)	Risk Factor	Comments
Logging	Good	6	2	Low	The QP visually inspected the core and noted the lithologies. Only minor variations were noted. A logging SOP was available for review. Logging detail was comprehensive and at a level of detail that provided good-quality geological domains. The QP recommends updating the logging procedure and integrating it into a data management system.
Sub-sampling Techniques and Sample Preparation	Average	5	3	Moderate	An SOP regarding the first split of diamond core was available to review and stated that core was sampled along 1-m intervals, except in zones of distinct mineralisation. The SOP stated core should be cut perpendicular to features of interest, and where these features were absent, core should be cut perpendicular to the rock fabric. During the site visit, the QP reviewed sections of cut core, which indicated the SOP was followed. However, the QP notes it is best practice to mark and cut core along the orientation line (or a few degrees off it to preserve the line), and it is important to always sample the same half of the core to ensure no sampling bias is introduced. Based on the SOP and observations made by the QP, the QP considers that the first-splitting process poses a moderate risk with respect to the DQO. The QP recommends that changes are made to the core- cutting procedures at the Project to minimise the risk of introducing selection bias. No second-split duplicates of core samples were collected. The QP recommends collecting second-split repeat samples for any future resource delineation drilling programmes. The QP recommends updating the sub-sampling SOP and including details on the process for integrating the data into a data management system.
Quality of Assay Data and Laboratory Tests	Average	5	2	Low to moderate	RRPL inserted Rocklabs CRMs. While the CRM are mostly accurate and precise, the QP notes that the CRMs were not matrix matched with the rock type and mineralisation. Future CRMs should be sourced from similar metasediments with elevated Sb and As, or the consideration could be given to making a CRM from mineralised samples. The QP recommends that the analytical SOP be updated and detail the process for integrating the data into a data management system.
Verification of Sampling and Assaying	Good	7	2	Low	RRPL did not conduct umpire assays. During the site visit, the RSC collected a representative number of check samples from the Project to verify mineralisation and grade tenure (mix of half-core and pulp samples). The Au samples demonstrate an exact bias towards the original sample that is likely to be in the order of 4%, which falls within the realms of the DQO.
Location of Data Points	Good	6	2	Low	There was no SOP for collar set-out, pick-up, or downhole surveying. Collars have been picked up using DGPS and downhole survey tools. Collar heights have been adjusted onto a LiDAR surface. The QP notes there were some data management issues with some of the location data provided; these were noted, adjusted, and changed.

Item	Data/Info Availability	Score (1–10)	Impact Factor (1–5)	Risk Factor	Comments
					The QP recommends that a collar location SOP be developed, detailing the process for integrating the data into a data management system.
Data Spacing and Distribution	Good	6	3	Moderate	Due to DOC consent restrictions, numerous drillholes were often drilled from one pad, resulting in inconsistent data spacing. However, the QP notes that the drillhole spacing is appropriate to assume and infer the geological and grade continuity for the classification of at least Inferred Mineral Resources.
Bulk Density	Good	5	2	Low	An SOP for bulk density was available for review. The SOP confused specific gravity (SG) with bulk density; however, the procedure undertaken was consistent with the water displacement method described by Lipton and Horton (2014). The QP also recommends collecting duplicate measurements. The method used by RRPL was also prone to selection bias, and the QP recommends collection of larger cores with defects and trying alternative methods.
Orientation of Data/Drilling	Good	4	2	Moderate	Due to DOC consent restrictions, numerous drillholes were often drilled from one pad. This resulted in many drillholes intersecting the mineralisation at high angles. While the QP recommends optimising the drill pattern for the reef orientation where possible, it is unlikely that this risk can be mitigated.
Sample Security	Average	5	1	Low	An SOP for the security or chain-of-custody was not available. Samples collected for laboratory analysis were securely packaged on site and transported to SGS Westport for sample preparation. All samples were stored in a locked core shed until dispatch.
Database Integrity	Average	5	3	Moderate	The Project has a relational database. Data were provided via a series of MS Excel files. This made validating and querying data difficult and time consuming. Where errors in the MS Excel workbooks were noted by the QP, they were communicated to RRPL and corrected.
Geological Interpretation	Good	7	3	Moderate	Geological logging plus surface observations from outcrops and trenching were used to develop the geological model. The measured vein and reef orientations near the surface and orientation of historical workings at depth provide an understanding of mineralisation control. Subsequent drilling has provided detailed core logging to support the continuing understanding of the mineralisation at depth. In general, the mineralisation can be visually logged as veins, faults, and breccias surrounded by broad zones of disseminated arsenopyrite and stibnite. Logging was supported by pXRF to help confirm mineralised and unmineralised zones.
Estimation and Modelling: Domaining	Good	6	4	Moderate	The estimation domains are considered robust, being based on geological observations that align with the understanding of controls on mineralisation. In general, the CVs were below 2; however, the CVs were high in several domains at Auld Creek. This was managed by indicator kriging at Auld Creek for the Sb estimate.

Item	Data/Info Availability	Score (1–10)	Impact Factor (1–5)	Risk Factor	Comments
Estimation and Modelling: Compositing	Good	7	2	Low	A composite length of 2 m was selected owing to the high angle at which the drilling intersects the mineralisation in many domains. This offers an acceptable compromise between capturing the desired precision of the geological and estimation domain modelling and matching the likely selectivity of the underground operation.
Estimation and Modelling: Grade Capping	Excellent	7	2	Low	No top-cuts were applied.
Estimation and Modelling: Variography	Excellent	7	2	Low	Experimental and modelled variograms display satisfactory structure and an acceptable level of confidence for the estimation of Inferred and Indicated Mineral Resources.
Estimation and Modelling: Interpolation and Extrapolation	Excellent	7	3	Low to Moderate	Grades were estimated using ordinary kriging and validated against nearest neighbour and inverse distance estimation methods. Sensitivity testing indicated the estimation was not sensitive to the number of samples, variogram models, or compositing length. Extrapolation is typically 50% of the drillhole spacing laterally, and the QP considers the extrapolation distance reasonable.
Estimation and Modelling: Checks and Validation	Excellent	7	3	Low to Moderate	The QP considers the block models to be robustly estimated, based on a comparison of input mean grades with the block model mean grade using swath plots and visually on cross-sections.
Estimation and Modelling: Cut-Off	Good	8	4	Low to Moderate	A cut-off grade of 1.7 g/t Au was selected for the reporting of the Mineral Resource based on a brief assessment of potential modifying factors.
Estimation and Modelling: Density	Average	6	2	Low	The bulk density measurement procedure was consistent with the water displacement method described by Lipton and Horton (2014). Estimations were completed using an RBF interpolant rather than OK due to the widely spaced clustered data.
Estimation and Modelling: Classification	Good	8	4	Low to Moderate	A cut-off grade was selected for the reporting of the Mineral Resources based on a brief assessment of potential modifying factors. The assessment of RPEEE was carried out using a re-blocking approach. RPEEE categories were assigned after re-blocking the model to regular minimum mining units.

The MRE for the Auld Creek deposit has been prepared by RSC and supplied to Mining One as the basis for potential inventory generation.

The QP (Gary Davison) has relied on the resource block model and classification fields provided by RSC and RUA for the purposes of cut-off grade analysis, MSO generation, mine design and economic assessment.

RUA supplied Mining One with a resource model (*Auld Creek Block Model February 2026 ITR.dm*) for the Auld Creek deposit. The QP (Gary Davison) used this block model to generate economical shapes using Deswik Mineable Shape Optimiser (Deswik.SO) software. These shapes were then manually culled to arrive at an inventory.

15 Mineral Reserve Estimates

This section is not applicable, as no Mineral Reserves have been estimated.

16 Mining Methods

The mining methods reported in this section are at a PEA level of study, based on the Mineral Resources reported within the Project. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves. There is no certainty that the PEA will be realised. No Pre-Feasibility Study or Feasibility Study in accordance with NI 43-101 and the CIM definition standards has been completed here or previously.

16.1 Geotechnical

16.1.1 Available Data

Geotechnical parameters for the Auld Creek deposit were derived from drilling data provided by RUA. Two principal datasets were used in the analysis:

- lithology/structural logging dataset, containing information on rock strength estimate and defect surface conditions; and
- core recovery dataset, containing rock quality designation (RQD) values.

No laboratory rock testing results were available for The QP (Gary Davison) to review.

The analysis used the following information from the drillhole datasets:

- RQD;
- joint condition and infill parameters;
- joint orientation and bedding measurements; and
- field estimates of intact rock strength.

Due to the absence of a geological model, spatial filtering was applied to target drillhole intervals located within 10 m of the deposit, capturing rock mass conditions expected to influence stope walls and near-mineralised zone stability.

16.1.2 Geotechnical Domains

In the absence of a detailed geological model, geotechnical domains were defined based on depth and rock mass quality indicators derived from the drillhole dataset.

Two domains were identified based on differences in RQD values (Table 16-1).

Table 16-1: Geotechnical domains.

Domain	Elevation Range	Description
Upper Domain	600–480 RL	Shallower portion of the deposit
Lower Domain	480–250 RL	Deeper section of the deposit

The division between domains was selected as the RQD data indicated a noticeable improvement in rock mass quality at depth.

16.1.3 Rock Mass Characterisation

16.1.3.1 Rock Quality Designation

RQD values were calculated from the recovery dataset for intervals located within 10 m of the deposit (Figure 16-1 and Figure 16-2).

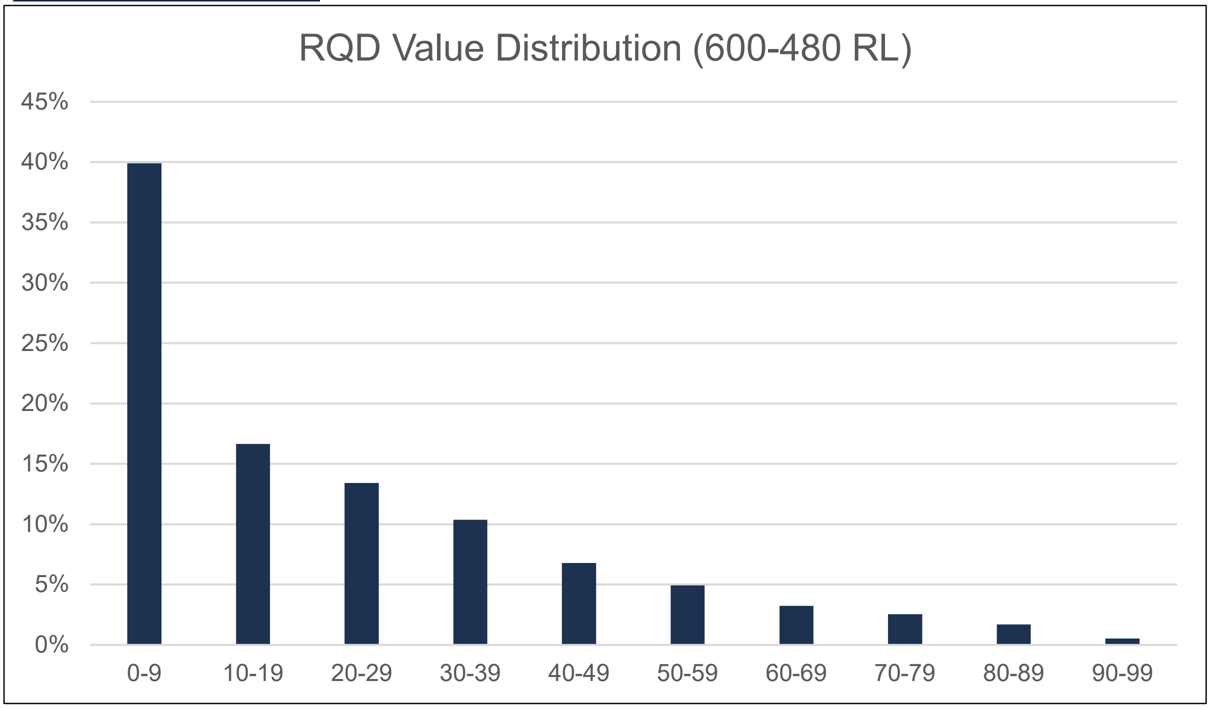


Figure 16-1: RQD values (600–480 RL).

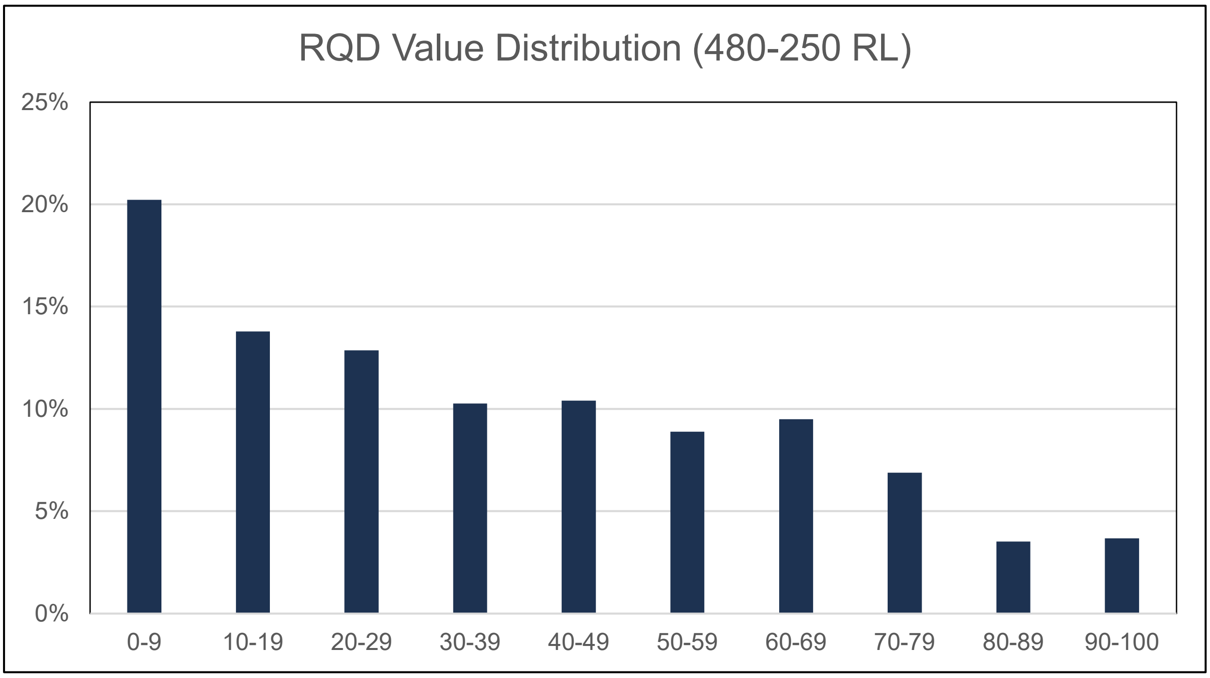


Figure 16-2: RQD values (480–250 RL).

Weighted median RQD values were determined to represent typical rock mass quality within each domain (Table 16-2).

Table 16-2: RQD by domain.

Domain	Median RQD (%)	Weighted Median RQD (%)
600–480 RL	16	18
480–250 RL	33	35

16.1.3.2 Joint Condition Parameters

Joint condition parameters were derived from the structural logging dataset. The parameters considered include:

- Jn – joint set number;
- Jr – joint roughness; and
- Ja – joint alteration.

The joint condition parameters were found to be relatively insensitive to depth within the analysed dataset. Consequently, similar values were adopted for both the 600–480 RL and 480–250 RL geotechnical domains.

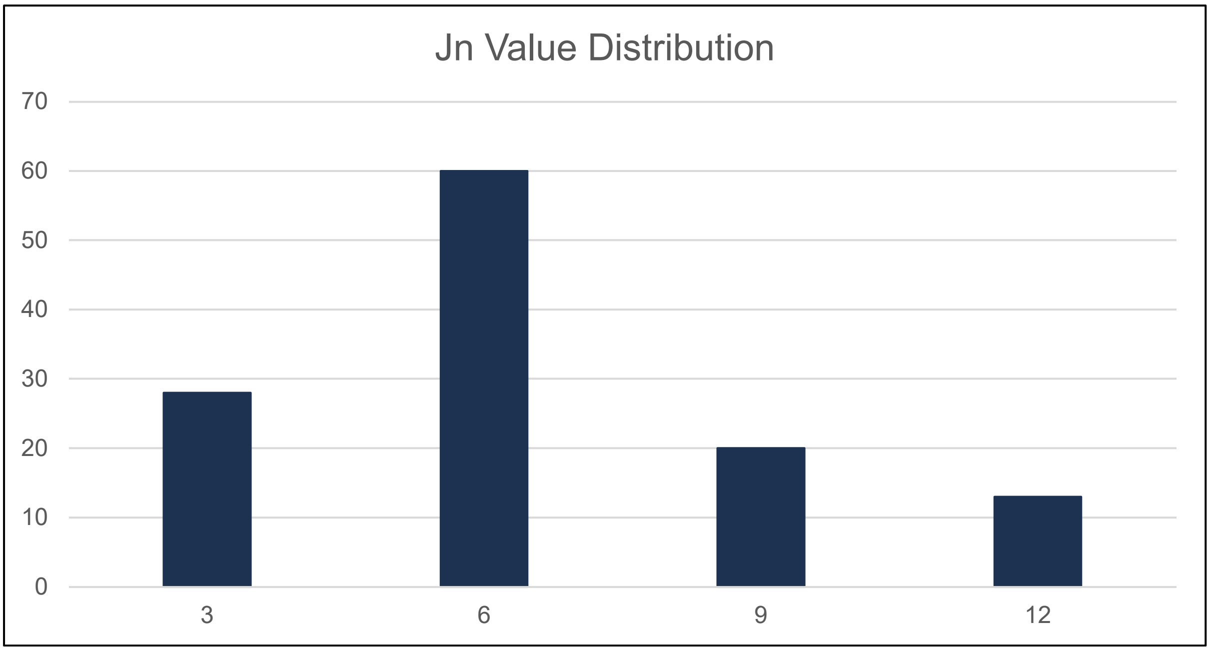


Figure 16-3: Jn values.

Weighted median values were calculated using the logged interval lengths (Table 16-3).

Table 16-3: Q parameters used in Mathews Stability analysis.

Domain	Jn	Jr	Ja
600–480 RL	6	2	6
480-250 RL	6	2	6

These values indicate a moderately jointed rock mass with smooth to slightly rough joint surfaces and moderate alteration.

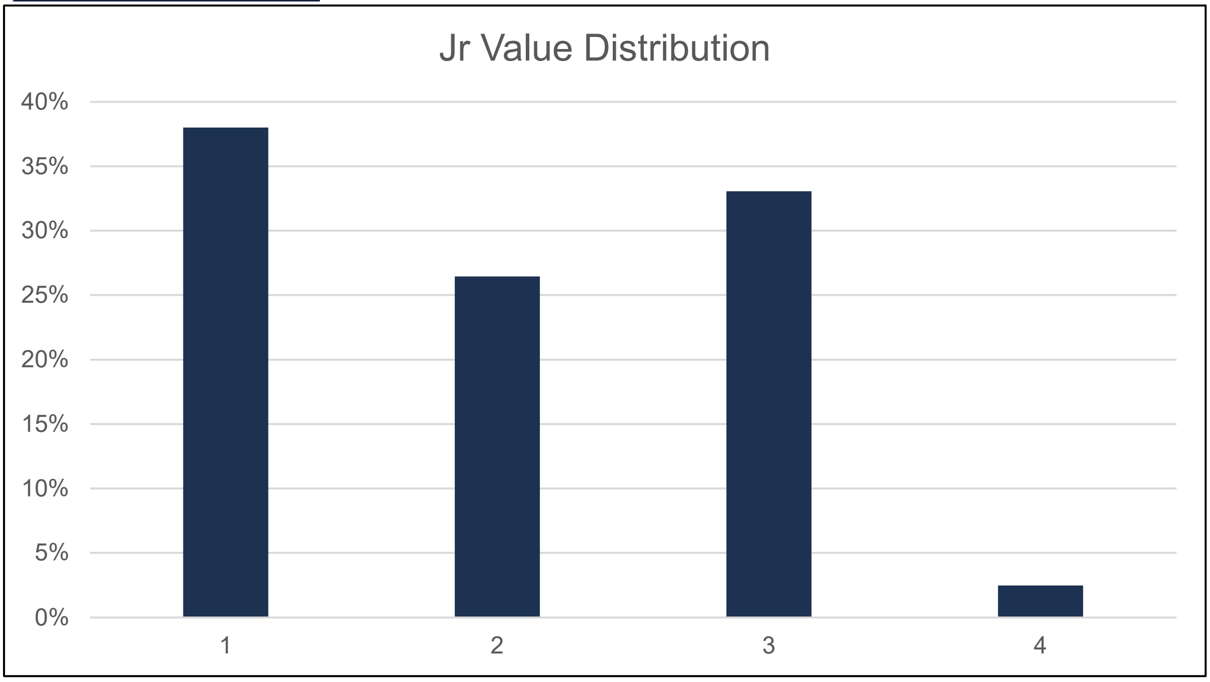


Figure 16-4: Jr values.

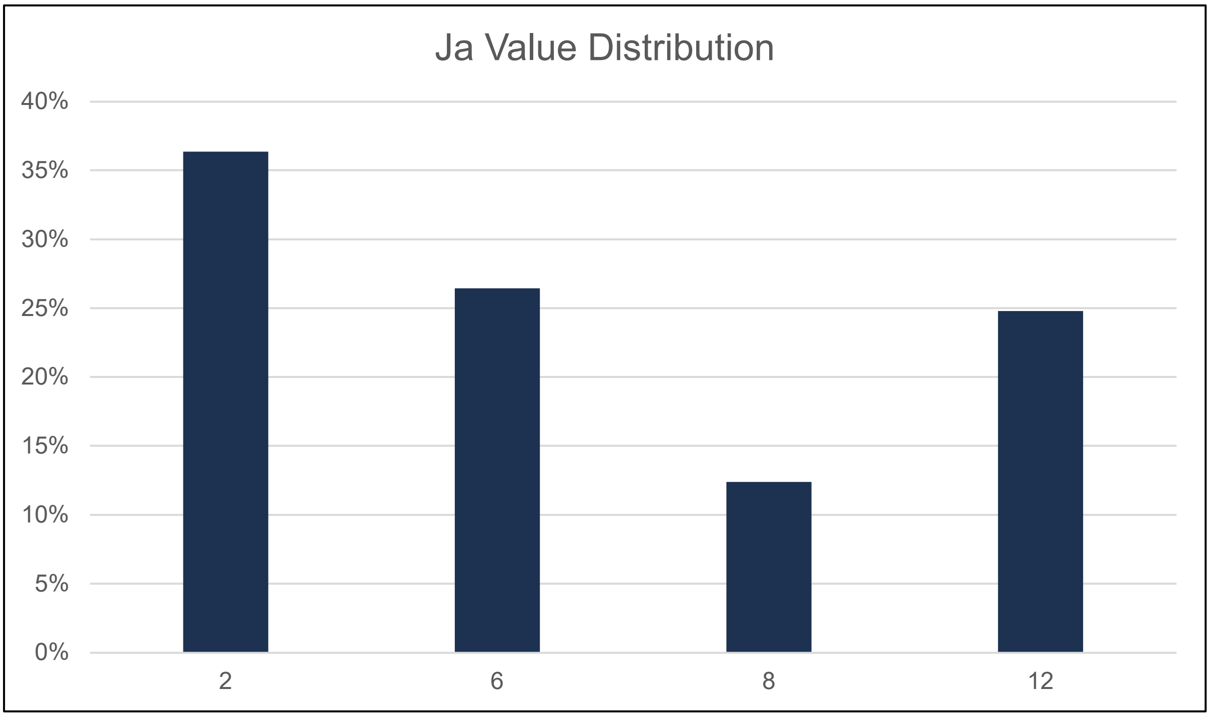


Figure 16-5: Ja values.

16.1.3.3 Intact Rock Strength

Intact rock strength was estimated from field strength classifications recorded during geotechnical logging (Table 16-4, Figure 16-6, and Figure 16-7).

Table 16-4: Field rock strength estimate (ISRM).

Class	Description	Estimated UCS (MPa)	Field indication
R0	Extremely weak	0.25–1	Indented by thumbnail
R1	Very weak	1–5	Crumbles under firm hammer blow, can be peeled by knife
R2	Weak	5–25	Can be broken with geological hammer
R3	Medium strong	25–50	Requires more than one hammer blow to break
R4	Strong	50–100	Difficult to break with hammer
R5	Very strong	100–250	Only chipped by hammer
R6	Extremely strong	>250	Hammer rebounds, very difficult to chip

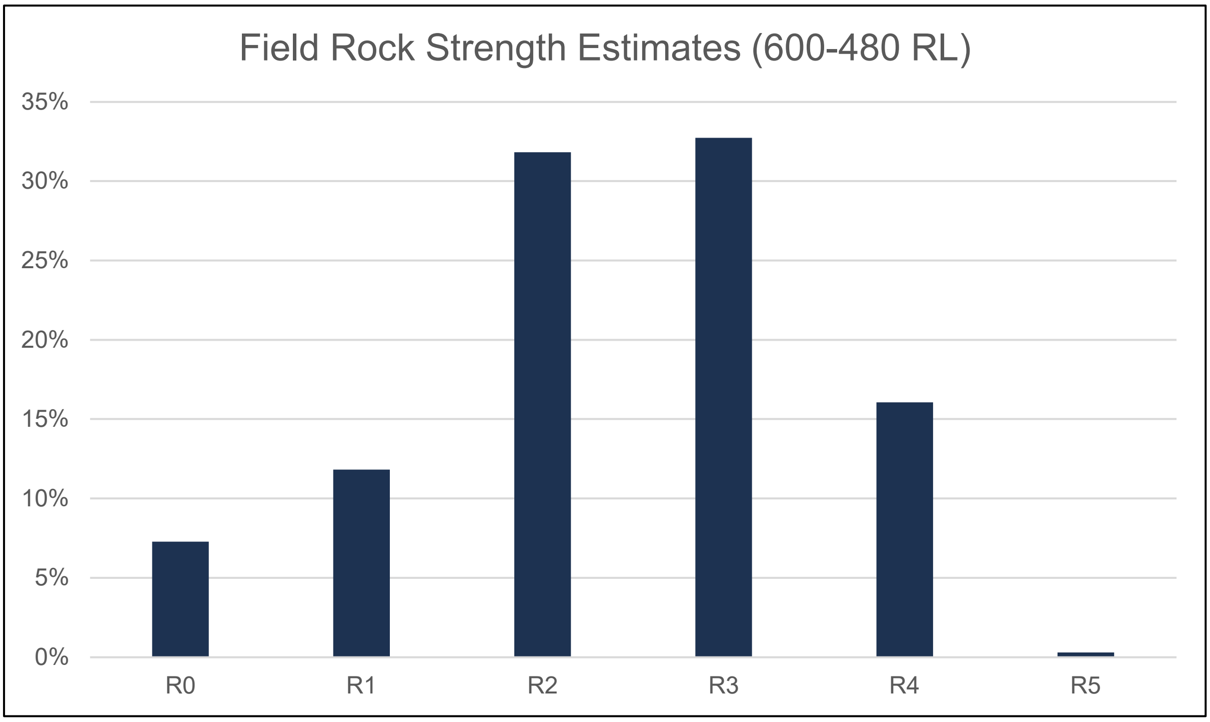


Figure 16-6: Field rock strength estimates (600–480 RL).

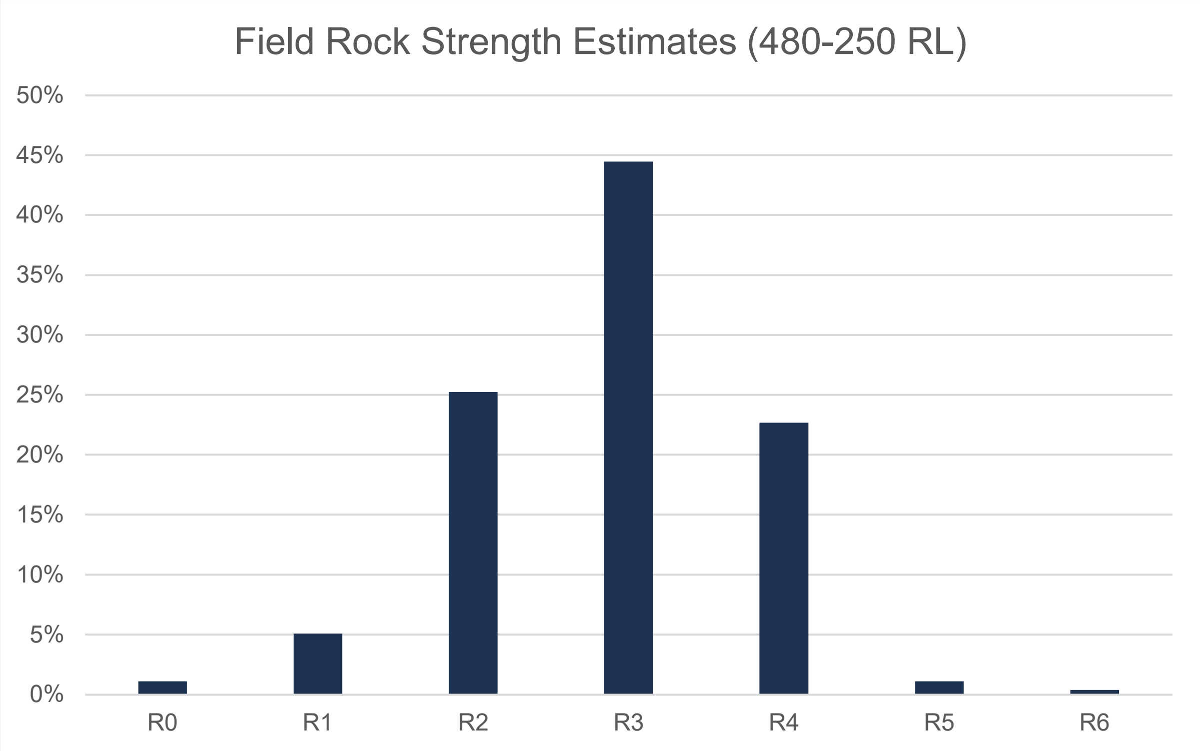


Figure 16-7: Field rock strength estimates (480–250 RL).

The dominant strength class observed in the dataset corresponds to R2–R3 categories, which are typically associated with intact compressive strengths in the range of 25–50 MPa (Table 16-5).

Table 16-5: Field UCS estimates by domain.

Domain	Field Estimated UCS (MPa)
600–480 RL	25-50
480–250 RL	25-50

These values represent approximate estimates only, as no laboratory uniaxial compressive strength (UCS) testing results were available at the time of the study.

16.1.4 Structural Analysis

Structural discontinuities recorded in the drillhole logging were analysed using stereonet projections. Two datasets were evaluated:

- bedding planes; and
- all logged structural defects.

The analysis used lower hemisphere equal-angle stereonet projections of poles to planes, with contour density plots used to identify dominant structural orientations.

The stereonet analysis identified a dominant discontinuity set within the rock mass. These structural features are expected to influence potential failure mechanisms in underground excavations.

The presence of multiple joint sets indicates potential for:

- structurally controlled planar failures;
- wedge failures; and

- localised overbreak along persistent discontinuities.

The critical defect sets were considered when assessing stope stability using the Mathews Stability method.

16.1.4.1 Bedding Orientation

The stereonet projection of bedding poles indicates a well-defined dominant bedding orientation within the dataset (Figure 16-8). The mean bedding orientation derived from the stereonet analysis is:

- dip: 56°
- dip direction: 278°.

This indicates bedding typically dips moderately to steeply towards the west-northwest.

The bedding poles form a clear concentration cluster on the stereonet, suggesting that bedding orientation is relatively consistent across the analysed intervals.

Bedding-parallel structures were also recorded in the logging database, including bedding shear (bdsh) structures, which may locally weaken bedding planes and contribute to anisotropic rock mass behaviour.

Given the moderately steep dip of the bedding planes, these structures may influence hanging wall stability, depending on the stope wall orientation relative to bedding.

Bedding orientations exhibit variability exceeding ±60° in both dip and dip direction, indicating moderate to high undulation of the bedding planes. This variability is likely to significantly influence local stope stability conditions. It is therefore recommended that, in subsequent stages of the study, bedding planes be modelled as wireframes for use in stability analyses.

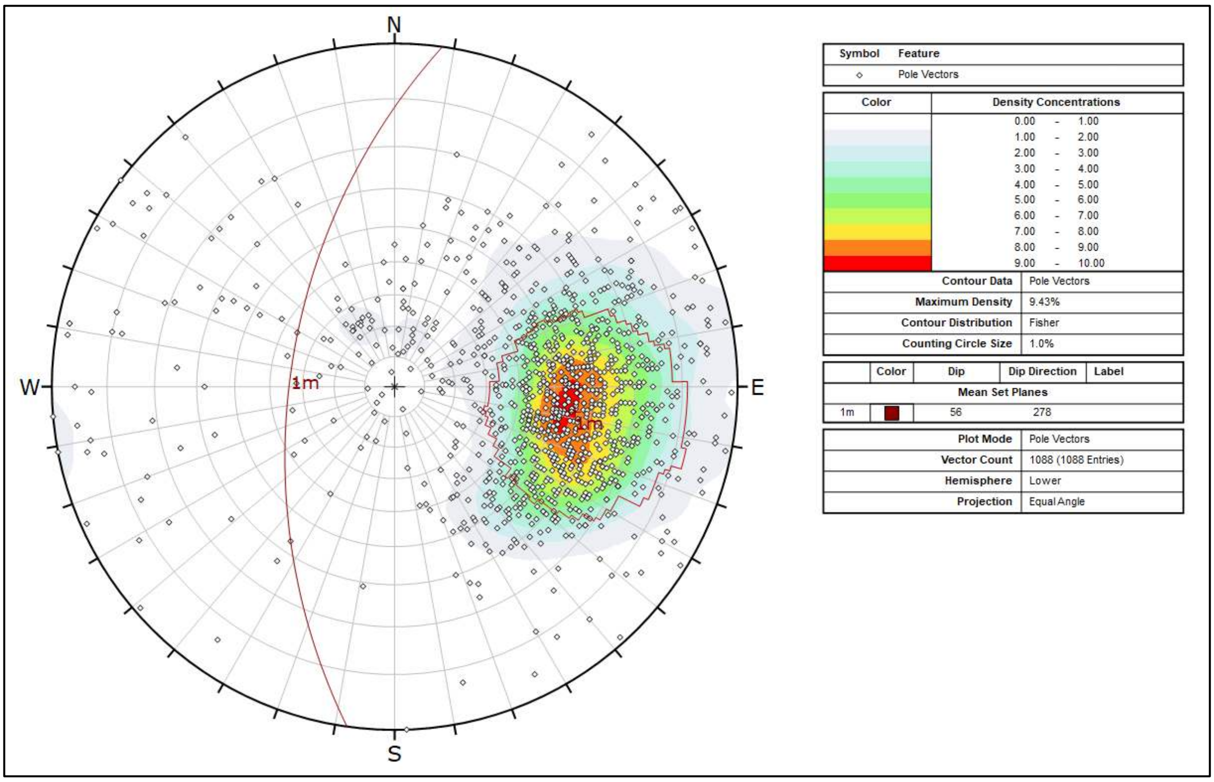


Figure 16-8: Stereonet projection of bedding poles.

16.1.4.2 Major Defect Set Orientation

All structural defects recorded in the drillhole logging were combined and analysed to assess the broader structural fabric of the rock mass.

The stereonet projection of defect poles (Figure 16-9) indicates a dominant structural set with the following mean orientation:

- dip: 57°
- dip direction: 268°.

This orientation is broadly similar to the bedding orientation, suggesting that a significant proportion of the logged defects are either bedding-controlled structures or sub-parallel discontinuities.

The pole density contours indicate a strong clustering of poles in the eastern quadrant of the stereonet, reflecting a consistent set of moderately to steeply dipping structures striking approximately north-south.

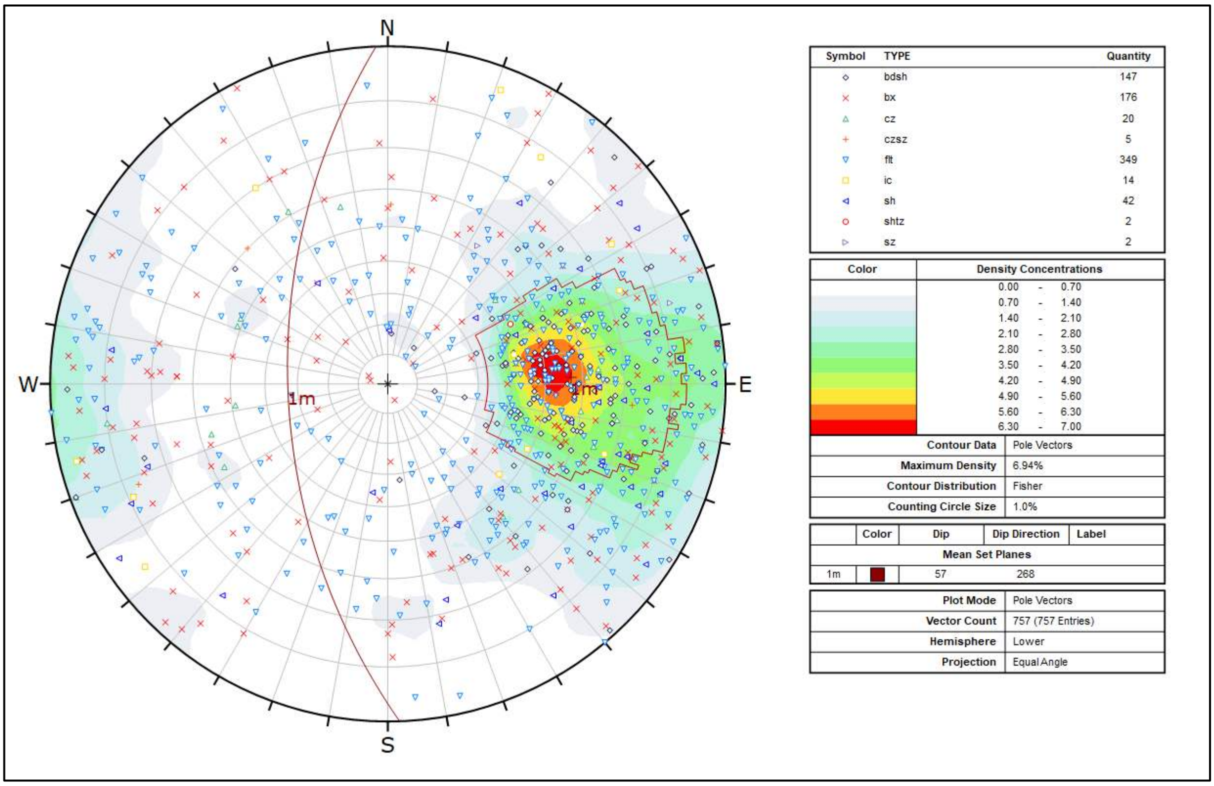


Figure 16-9: Stereonet projection of defect poles.

16.1.5 Rock Mass Classification

Rock mass quality was estimated using the Q-system classification (Barton et al., 1974).

The weighted median Q′ values were calculated using the representative parameters derived from the dataset.

The geotechnical logging data were weighted based on the core intervals. The weighted mean of all lithologies is presented in Table 16-6. Note that the Geological Strength Index (GSI) was derived from the equation below (Hoek et al., 2013).

$$GSI = \frac{52 J_r/J_a}{(1 + J_r/J_a)} + RQD/2$$

Table 16-6: Rock mass parameters.

Domain	RQD	Jn	Jr	Ja	Q'	GSI
600–480 RL	18	6	2	6	1	22
480–250 RL	35	6	2	6	2	31

These values indicate:

- Upper Domain: very poor to poor rock mass
- Lower Domain: poor rock mass.

16.1.6 In Situ Stress

The in situ stress conditions at the Auld Creek deposit were estimated using a regional stress desktop study. The study evaluated the regional tectonic setting of the Reefton Goldfield and analysed stress orientations derived from earthquake focal mechanism inversions across the South Island of New Zealand.

The analysis indicates that the regional stress regime in the Reefton area is most consistent with a strike-slip stress environment associated with tectonic deformation along the Alpine Fault, which forms the boundary between the Australian and Pacific plates.

The relative motion between these tectonic plates results in significant shear deformation along the Alpine Fault, which strongly influences the regional stress field across the South Island.

16.1.6.1 Principal Stress Orientation

Based on the inversion of earthquake focal mechanisms in the vicinity of the Alpine Fault, the principal stresses for the Reefton region are presented in Table 16-7.

Table 16-7: In situ stress orientation.

Principal Stress	Orientation	Description
σ_1	05° / 115°	Sub-horizontal maximum compressive stress
σ_2	85° / 295°	Approximately vertical
σ_3	00° / 025°	Sub-horizontal minimum compressive stress

This configuration indicates that the maximum principal stress is horizontal, while the vertical stress approximates the intermediate principal stress.

16.1.6.2 Stress Magnitude Relationships

Based on regional stress ratios derived from the earthquake focal mechanism analysis, the principal stress magnitudes are estimated to follow the relationships indicated in Table 16-8.

Table 16-8 : Stress magnitude relationships.

Stress Component	Relationship
σ_1	$1.5 \times \sigma_v$
$\sigma_2 \approx \sigma_v$	Vertical stress
σ_3	$0.67 \times \sigma_v$

where: $\sigma_v = \rho g D$

and:

- ρ = rock density
- g = gravitational acceleration
- D = depth below surface.

These relationships are consistent with a strike-slip tectonic regime and are considered appropriate for preliminary underground mine design assessments at the Project.

16.1.6.3 Stress Ratio

The stress ratio parameter derived from the focal mechanism inversion is:

$$R = \frac{\sigma_1 - \sigma_2}{\sigma_1 - \sigma_3} = 0.60 \pm 0.24$$

This value indicates that the differences between the principal stresses are relatively balanced, which is typical of strike-slip tectonic environments.

16.1.6.4 Influence of Local Structure

The geology of the Reefton Goldfield comprises folded Ordovician sedimentary rocks with significant faulting and shearing. Gold mineralisation occurs primarily within quartz veins associated with structural zones trending approximately north-south.

These structural features have the potential to locally influence the regional stress regime by concentrating stresses or modifying the orientation of local stress fields.

In particular, the cross-cutting northwest-striking faults and bedding-parallel shear zones may locally disturb the regional stress orientation.

16.1.6.5 Limitations

No direct in situ stress measurements are currently available for the Auld Creek deposit. The stress field described above therefore represents a regional estimate based on tectonic analysis and earthquake focal mechanism inversion.

To improve confidence in the stress model for future project stages, the QP (Gary Davison) recommends that direct stress measurements be collected when underground development commences. However, considering the relatively shallow depth of the deposit and the adequately constrained regional stress regime, direct stress measurement is not considered a priority at this stage.

16.1.7 Stope Stability Assessment

16.1.7.1 Methodology

Stope stability was evaluated using the Mathews Stability Graph method (Mathews et al., 1981), including the extended version (Mawdesley et al., 2001), using a larger database of open stope performance cases.

The extended stability graph provides an empirical relationship between rock mass quality, excavation geometry, and observed stope performance in underground open stoping operations. The method is widely used at early project stages to evaluate the stability of proposed stope geometries.

The stability assessment is based on two primary parameters:

1. **Stability Number (N')**

   A measure of the rock mass ability to remain stable, calculated using Q′ (Q-prime), which is a modified form of the Q-system rock mass classification.

   Q′ represents the structural component of the Q-system, derived from the following parameters:

   - RQD – rock quality designation
   - Jn – joint set number
   - Jr – joint roughness
   - Ja – joint alteration

   and is calculated as:

$$Q' = \frac{RQD}{Jn} \times \frac{Jr}{Ja}$$

   Representative Q′ (Q-prime) values were derived for each geotechnical domain from the drillhole dataset and used as inputs to the Mathews Stability analysis.

2. **Hydraulic Radius**

   A measure of excavation size calculated as:

$$HR = \frac{Area}{Perimeter}$$

   where:

   - area = surface area of the stope wall
   - perimeter = perimeter of the stope wall.

The calculated values of N′ and HR are plotted on the Mathews Stability Graph to determine the expected excavation performance.

Rock mass parameters used in the stability analysis were derived from the drillhole database and included:

- RQD;
- joint condition parameters (Jn, Jr, Ja);
- estimated intact rock strength; and
- structural orientation data.

The analysis was conducted for two geotechnical domains defined in Section 16.1.2.

The Mathews Stability analysis was completed using the inputs in Table 16-9.

Table 16-9: Mathews Stability analysis inputs.

Parameter	Domain	Value
Maximum depth	600–480 RL	120 m
Maximum depth	480–250 RL	350 m
Vertical stress, (\sigma_v)	600–480 RL	3.2 MPa
Maximum horizontal stress, (\sigma_H)	600–480 RL	4.9 MPa
Vertical stress, (\sigma_v)	480–250 RL	9.5 MPa
Maximum horizontal stress, (\sigma_H)	480–250 RL	14.2 MPa

Parameter	Domain	Value
Uniaxial compressive strength (UCS)	All Domains	37 MPa
Critical joint set angle	All Domains	56/268°
Wall surface dip angle	All Domains	85°

The method provides an empirical indication of likely stope performance and is considered appropriate for scoping-level evaluation of stope geometries.

16.1.7.2 Mathews Stability Assessment Results

The Mathews Stability assessment indicates that rock mass conditions within the Auld Creek deposit fall within the poor to very poor rock mass quality range, with representative Q′ values of ~1.0 for the upper domain (600–480 RL) and 2.0 for the lower domain (480–250 RL).

The calculated stability number (N′) and hydraulic radius (HR) values for the proposed stope geometries were plotted on the extended Mathews Stability chart (Mawdesley et al., 2001). In conjunction with the MSO shapes, the following stope geometries were evaluated to assess excavation stability:

- Stope height (H) = 10 m, strike length (L) = 10 m, width (W) = 5 m.

The hydraulic radius and stability number were calculated for the primary excavation surfaces, including the crown, hanging wall, footwall and sidewalls.

Upper Domain (600–480 RL)

The results indicate that the crown surfaces plot well below the major failure boundary, suggesting that the crown is unlikely to remain stable under the analysed conditions (Table 16-10 and Figure 16-10). This indicates that crown stability will be a critical design consideration and will likely require additional ground support or operational controls to manage exposure. The hanging wall, footwall and sidewall surfaces plot within the stable zone, indicating that minimal dilution is anticipated.

Table 16-10: Mathews Stability summary – Upper Domain (600–480 RL).

Factor	Hanging wall	Footwall	Crown	N Endwall	S Endwall
Q'	1.0	1.0	1.0	1.0	1.0
A	1.0	1.0	0.45	1.0	1.0
B	0.8	1.0	0.7	1.0	1.0
C	7.0	7.0	1.0	8.0	8.0
N'	5.6	7.0	0.1	8.0	8.0

Lower Domain (480–250 RL)

For the lower domain (480–250 RL), the higher rock mass quality (Q′ ≈ 2.0) results in an improved stability number and therefore more favourable stability conditions (Table 16-11and Figure 16-11). Under these conditions, the hanging wall, footwall and sidewall surfaces plot well into the stable region of the chart. As in the upper domain, the crown remains the most critical surface, plotting within the unstable region and therefore representing the primary stability constraint.

Table 16-11: Mathews Stability summary – Lower Domain (480–250 RL).

Factor	Hanging wall	Footwall	Crown	N Endwall	S Endwall
Q'	2	2	2	2	2
A	1.0	1.0	0.1	1.0	1.0
B	0.8	1.0	0.7	1.0	1.0
C	7.0	7.0	1.0	8.0	8.0
N'	11.2	14.0	0.1	16.0	16.0

Overall, the Mathews Stability assessment indicates that stope geometries of 10 m height and 5 m width, with strike lengths up to 10 m, are appropriate for both the upper and lower domains, if crown stability is effectively controlled.

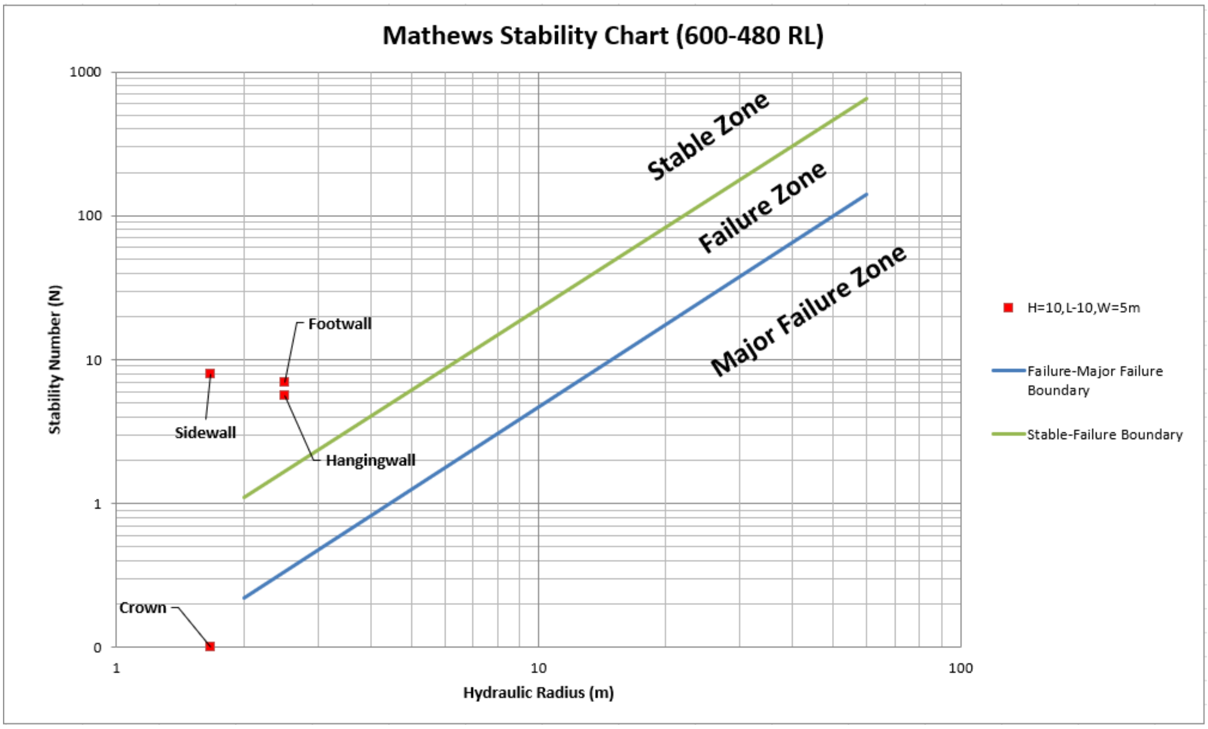


Figure 16-10: Mathews Stability chart (600–480 RL).

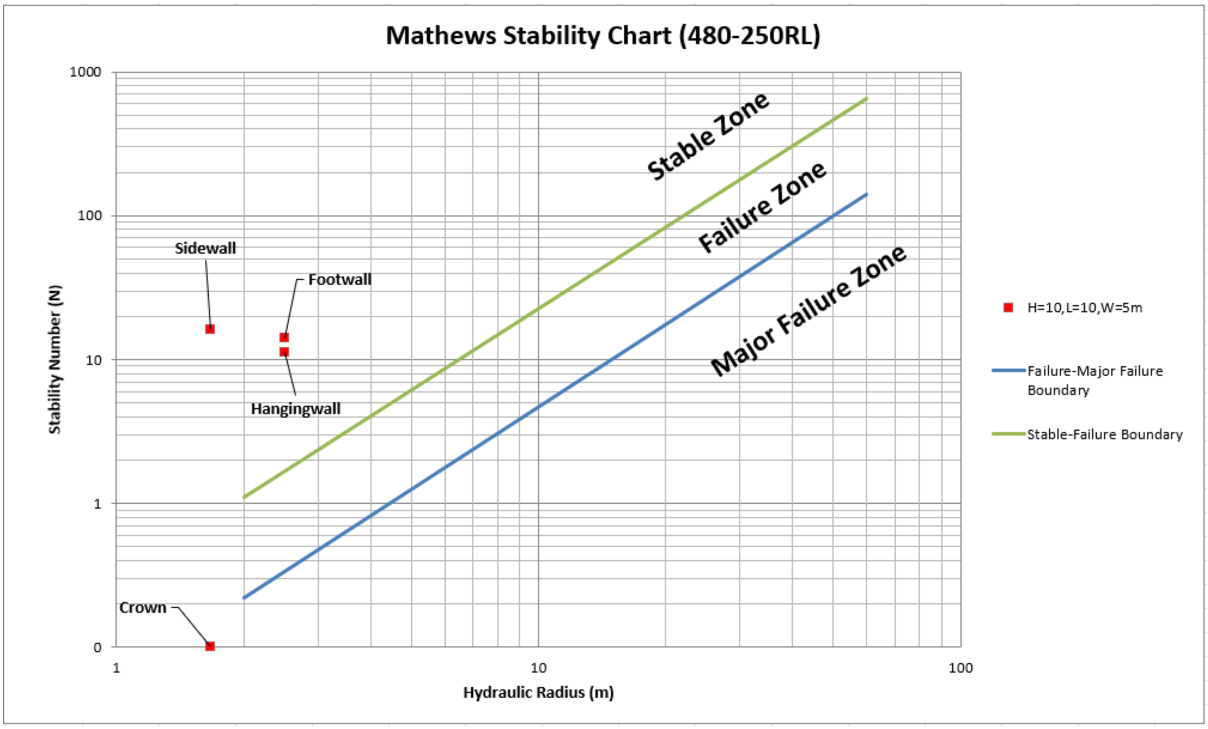


Figure 16-11: Mathews Stability chart (480–250 RL).

16.1.7.3 Limitations

The following limitations are noted:

- Strength anisotropy of the intact rock due to the presence of a foliation (the fabric) is not considered in the method. Intact strength anisotropy has a different impact on stope stability when compared to the effect of prominent open joints (effect of prominent joint sets is included in the Mathews Stability method).
- While empirical stability charts can be effective for predicting whether a stope will be geotechnically stable or unstable, they are unreliable for predicting the amount of dilution.
- The effect of discrete large-scale structures such as faults were not considered in the Mathews Stability method.

16.1.8 Numerical Stope Size Assessment

In addition to the empirical stable stope span analysis, the QP (Gary Davison) has undertaken a numerical assessment, accounting for the limitation of the empirical method. The numerical approach used involves the simulation of multiple potential stope geometries, with variability and uncertainty accounted for in the simulated geotechnical conditions. For the purposes of this stoping study, the assessment was split by elevation (RL) to account for a difference in material properties associated with depth. Table 16-12 shows the applied Improved Unified Constitutive Model (IUCM) material properties for the assessment.

Table 16-12: Numerical stope stability assessment material parameters.

Property	Value	Comment
Density	2.8 t/m^3	
Ei	40,000 MPa	Young's modulus of intact rock
GSI	31/22	600–480 RL/480–250 RL
mi_{max}	18	
UCS_{MAX}	37.5 MPa	Variability: SD = 4 (Field Estimate: 25–50 MPa)
Anisotropy Orientation	56°/278°	Uncertainty: +/-20°
mi_{MIN}	9	
Anisotropy Factor	2	

The generic stope geometry controls are summarised in Table 16-13, noting that the level spacing and strike length of the stope is automatically varied in the assessment to build a database covering potential sizes.

Table 16-13: Numerical stope stability assessment geometry parameters.

Variable	Unit	Value	Description
Level interval	m	15–50	Planned mining level spacing (can be varied with the min/max level spacing input if the parameter is not yet defined)
Stope (hanging wall) dip	°	80	Dip of hanging wall for analysis
Stope (hanging wall) dip direction	°	98	Dip direction of hanging wall for analysis
Stope depth below ground surface	m	120/350	Depth of the stope below ground surface (used for in situ stress calculations)
Stope/deposit width	m	3	Stope width/deposit width
Volumetric strain failure criteria	%	1	Refer to Figure 16-12
Acceptable overbreak	%	10	The target overbreak for solver to target (0% is not recommended, values up to 10% are suitable)

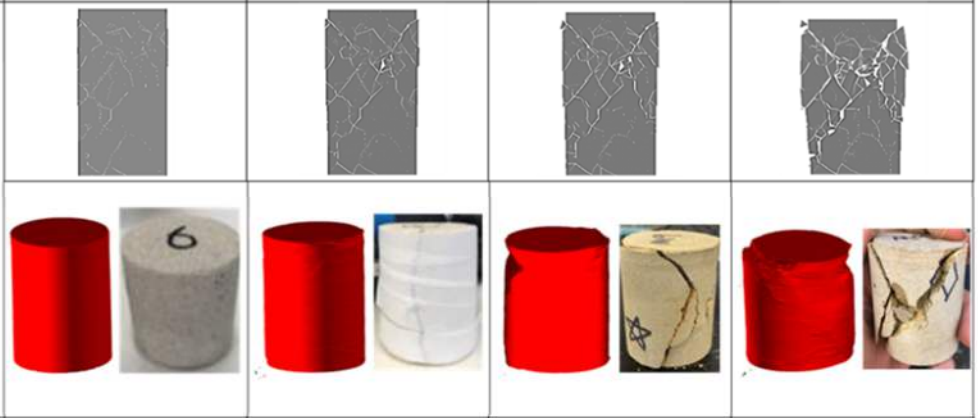
	Class I	Class II	Class III	Class IV
Volumetric Strain	0.5%	1.0%	2.0%	5.0%
Visual Appearance				
Conditions in Massive Rock Mass	– Onset of cracking – Less than 20% of blocks mobilised – Probability of failure is less than 20% – Less than 5% Cohesion Loss – Rock bridges exist along 80% of cracks	– Intermediate cracking – Around 50% of blocks mobilised – Probability of failure is around 50% – Around 20% Cohesion Loss – Rock bridges exist along 50% of cracks	– Extensive cracking – Around 80% of blocks mobilised – Probability of failure is around 80% – Around 50% Cohesion Loss – Rock bridges exist along 20% of cracks	– Very extensive cracking – More than 80% of blocks mobilised – Probability of failure is more than 80% – More than 50% Cohesion Loss – Minimum Rock bridges existing
Conditions in Highly Jointed Rock Mass and in high confinement	– cracking and joint opening – Less than 50% of blocks mobilised – Probability of failure is less than 20% – Less than 5% Cohesion Loss	– Around 80% of blocks mobilised – Probability of failure is around 50% – Around 20% Cohesion Loss	– Most blocks are mobilised – Probability of failure is around 80% – Around 50% Cohesion Loss	– Most blocks are mobilised – Probability of failure is more than 80% – More than 5% Cohesion Loss
Conditions in Highly Jointed Rock Mass and in low confinement	– Deep cracking and joint opening – Less than 50% of blocks mobilised – Probability of failure is less than 50% – Less than 5% Cohesion Loss	– Around 80% of blocks mobilised – Probability of failure is around 80% – Around 20% Cohesion Loss	– Most blocks are mobilised – Probability of failure is more than 80% – Around 50% Cohesion Loss	– About 80% of blocks mobilised – Probability of failure is more than 80% – More than 5% Cohesion Loss

Figure 16-12: Volumetric strain – definition and correlation with observed damage.

16.1.8.1 RL 480 m Stope Stability

A total of 2,328 simulated combinations of stope shapes and material properties were modelled. The methodology automatically generates a higher density of data points around the target dilution value, which in this case was 10%.

The results indicate that a hanging wall hydraulic radius of 4.0 m is appropriate for the design of stopes above RL 480 m. The modelling indicates that at HR ≤4.5, 90% of the modelled stopes have a hanging wall dilution of 10% or less (Figure 16-13). Notably, the modelled footwall dilution contributes to the total modelled overbreak, with sensitivity to the orientation of anisotropy (Figure 16-14). On average the modelled relationship between footwall overbreak and hanging wall overbreak is:

$$FW\ Dilution = 0.82\ \times HW\ Dilution$$

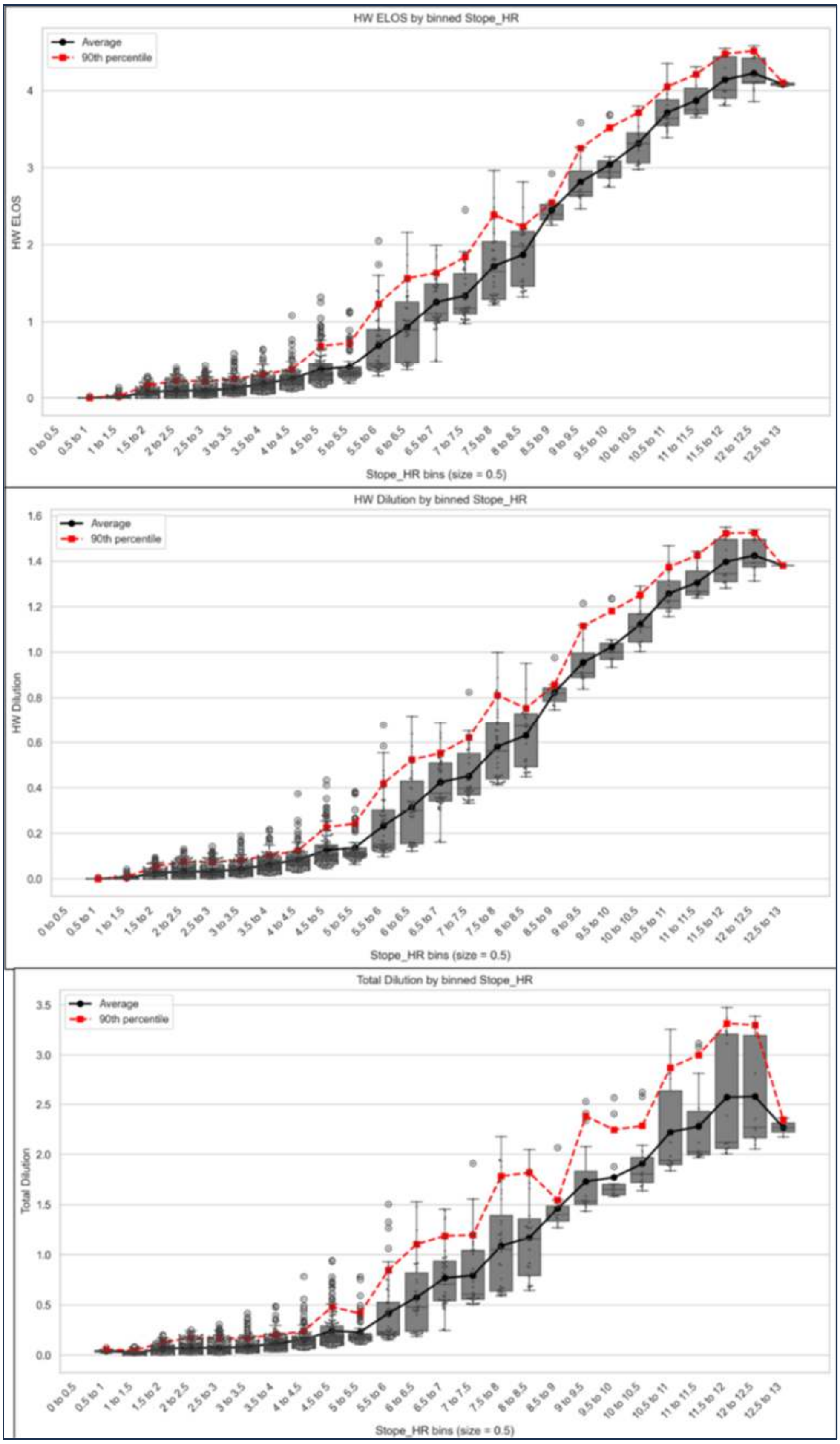


Figure 16-13: Simulated overbreak results for RL 480 m.

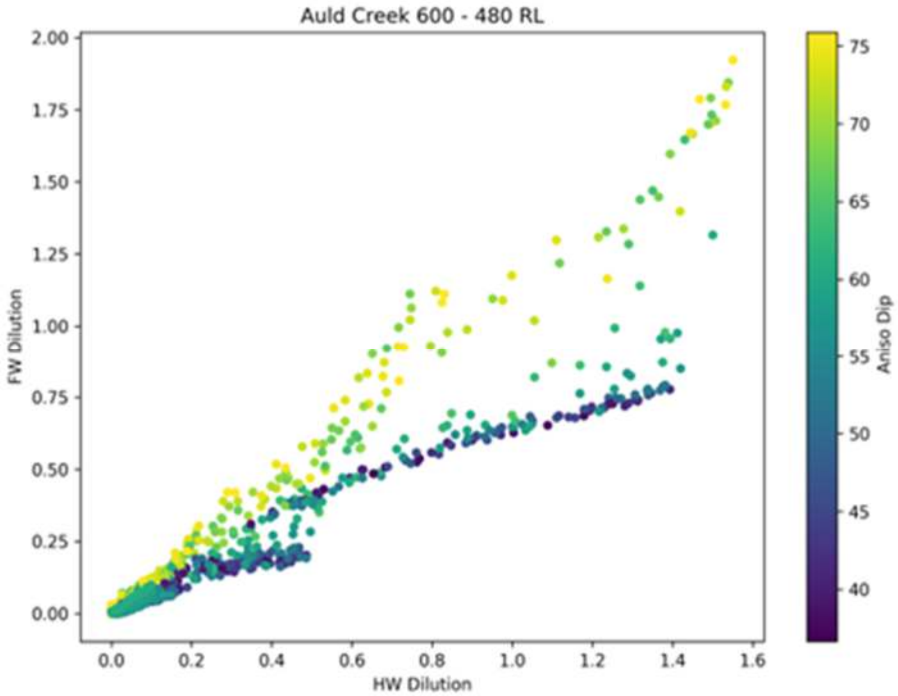


Figure 16-14: Footwall vs hanging wall modelled dilution; colour ramp on anisotropic dip.

16.1.8.2 RL 250 m Stope Stability

The reduced GSI and higher stress in this zone results in a significant reduction in stable stope span. Simulations using a hanging wall HR of 2 m indicate that 90% of stopes have <10% hanging wall dilution (Figure 16-15).

The use of ground support to increase stable span is likely to provide a benefit, given the limited unsupported stable span in the deeper zone.

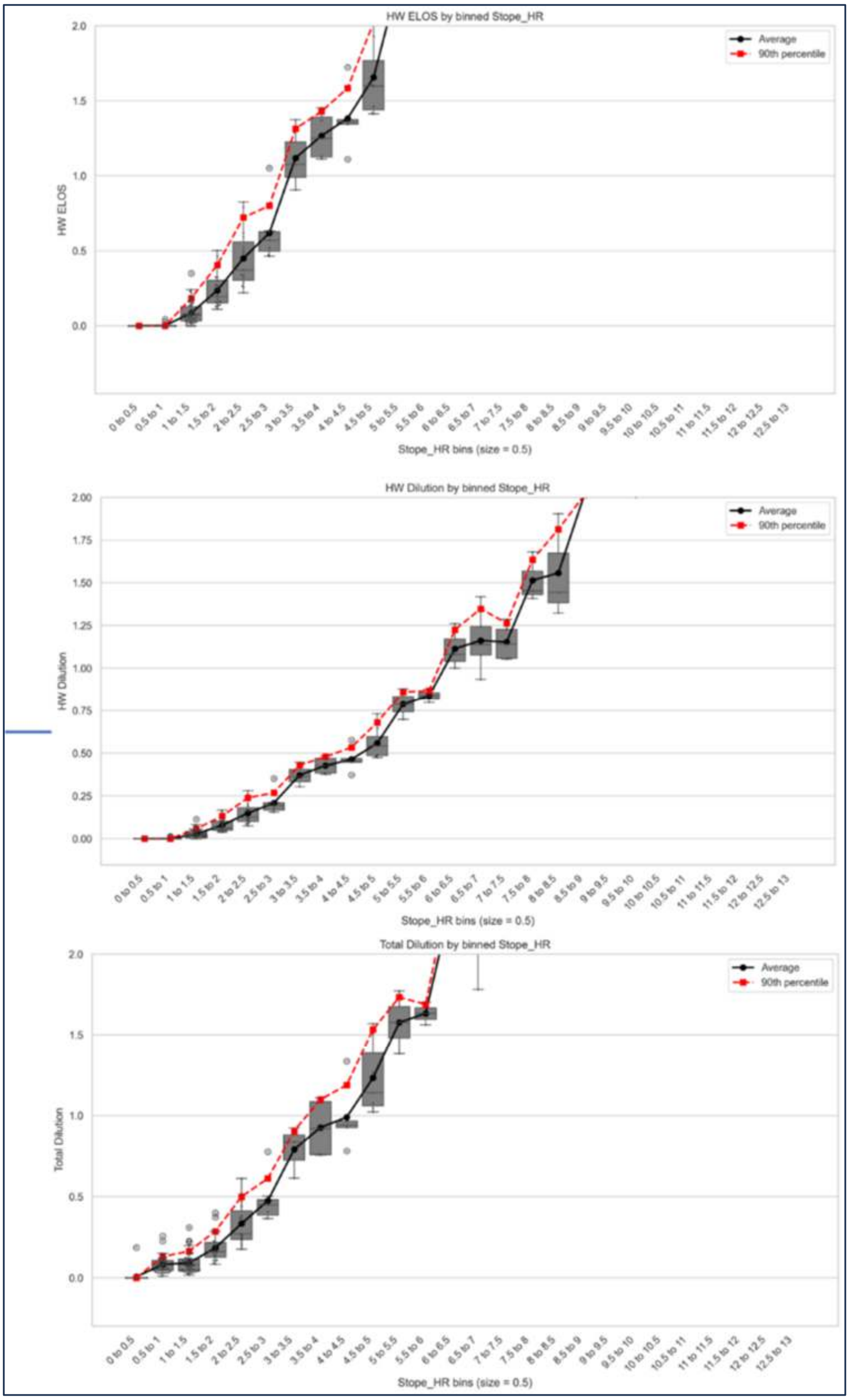


Figure 16-15: Simulated overbreak results for RL 250 m.

16.1.8.3 Numerical Stope Design Recommendations

The selection of stope spans is a balance of managing acceptable overbreak against operating cost. For the purposes of the scoping study, the recommended design spans are summarised in Table 16-14. However, the QP (Gary Davison) recommends that a cost-benefit analysis be conducted to establish the economic impact of increased stope span and accepting greater dilution.

Table 16-14: Numerical stope size assessment results.

Zone	Design Hanging Wall HR	Simulated Dilution Hanging Wall (90th percentile)	Simulated Hanging Wall ELOS (90th percentile)	Simulated Dilution All Walls (90th percentile)
600–480 RL	4 m	10%	0.27 m	20%
480–250 RL	2 m	10%	0.4 m	30%

Applying the results of the numerical stope size assessment to the Auld Creek deposit means designing appropriate HR to result in predictable stope dilution. The QP (Gary Davison) generated MSOs using 0.3 m Equivalent Linear Overbreak Slough (ELOS) on both the hanging wall and footwall (for 0.6 m total linear overbreak). This was based on the long-term dilution assumptions implemented by a comparable narrow-vein gold mine in Victoria, Australia.

Table 16-15 and Table 16-16 depict realistic stope dimensions that would satisfy HR recommendations for 600–480 RL and 480–250 RL, respectively.

Table 16-15: Upper portion of deposit (600–480 RL) – realistic stope dimensions.

Level Spacing	Stope Strike to Maintain HR = 4 m
15	17
20	13
25	11

Table 16-16: Lower portion of deposit (480–250 RL) – realistic stope dimensions.

Level Spacing	Stope Strike to Maintain HR = 2 m
10	7
12	6
15	5
20	5

16.1.9 Conclusions

The Auld Creek deposit is characterised by poor to very poor rock mass conditions, with some improvement at depth (RQD increasing from ~18% to ~35%). Joint conditions are consistent across domains, and intact rock strength is estimated at 25–50 MPa based on field logging.

Structural analysis indicates a dominant bedding-parallel fabric dipping moderately to steeply west-northwest, with significant variability (±60°), suggesting undulating bedding that will influence local stability. Potential failure mechanisms include planar and wedge failures.

The regional stress regime is interpreted as strike-slip and evaluations are considered adequate for this stage of study.

Mathews Stability analysis indicates that wall stability is generally acceptable for the proposed stopes, while crown stability is the primary constraint. Numerical modelling confirms limited stable spans, particularly at depth, with recommended hanging wall hydraulic radii of ~4.0 m (Upper Domain) and ~2.0 m (Lower Domain).

Overall, stoping is feasible, but stope dimensions will be constrained by rock mass quality, structural variability, and stress conditions, with dilution and crown stability being key risks.

16.1.10 Recommendations

The QP (Gary Davison) makes the following recommendations:

1. **Laboratory testing:** Conduct UCS and tensile strength testing to improve confidence in rock strength parameters for the PFS.
2. **Geotechnical model:** Develop a high-level geological and geotechnical model, including bedding wireframes, major faults, and bedding intensity.
3. **Structural data:** Improve structural confidence through enhanced logging and/or oriented core data.
4. **Numerical modelling:** Once the mining method and schedule are defined, undertake detailed numerical modelling to refine stope stability, dilution, infrastructure placement, and cemented rockfill (CRF) sill performance.
5. **Stress measurements:** Consider in situ stress measurements at later stages when underground access is available.
6. **Design optimisation:** Complete trade-off studies between stope span, dilution, and support requirements.

16.2 Mine Design

This section discusses the PEA (concept-level) mine design. The QP (Gary Davison) considered two mining methods:

- sub-level open stoping with cemented backfill (SLOS); and
- overhand cut-and-fill (OHCAF) with hybrid cemented fill, loose fill, and dry-stacked tailings backfill.

At the PEA stage, only the OHCAF was designed and scheduled using mine planning software. Indicative production rates were calculated using benchmark development speeds for the SLOS option, but it is recommended that this be further explored with a detailed mine plan at the PFS level.

16.2.1 Overhand Cut-and-Fill Design

Cut-and-fill (CAF) mining is achieved by developing laterally along the deposit strike, filling the development with waste or tailings (either loose fill or a cemented fill) and then mining on top of (OHCAF) or underneath (underhand cut-and-fill; UHCAF) the fill. This method of mining is highly selective and geotechnically favourable (as all spans are supported) but has a higher unit price than other bulk mining methods.

The QP (Gary Davison) recommends OHCAF as the most appropriate mining method for the following reasons:

- It is unlikely that the cost of cemented fill using UHCAF would be offset by the grade of the deposit; and
- OHCAF typically allows for higher productivity than UHCAF owing to the slower nature of cemented filling.

The QP (Gary Davison) recommends that, at the PFS level, UHCAF using pastefill could be investigated in a trade-off study. This could give the advantage of reducing tailings storage requirements.

16.2.1.1 Cut-and-Fill Mining Sequence

The proposed OHCAF mine design uses a top-down mining sequence, split into panels containing six lifts. Level accesses are spaced at 21.0 m vertically (6 m × 3.5 m lifts per access). Within the 21.0 m panels, lifts are extracted bottom-up. This means that the bottom-most lift in each level must be filled with cemented fill (in this case CRF or cemented tailings) as it will be undercut by mining of the panel below. Figure 16-16 illustrates the mining sequence and nomenclature for lifts in an isometric view.

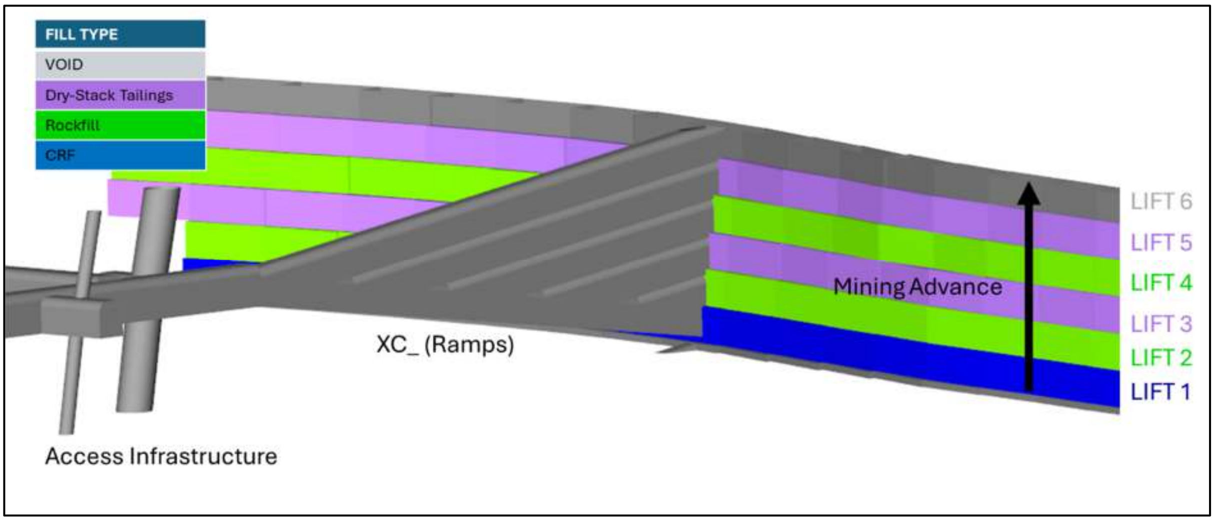


Figure 16-16: Naming convention for OHCAF lifts and sequence. (Oblique View).

Subject to test work, the QP (Gary Davison) proposes that dry-stacked tailings and cemented tailings can be used as the fill medium. This allows some mined waste rock to be used for surface construction demand (tailings storage facility (TSF) and pad building for the processing plant).

16.2.1.2 Level Access Design

The QP (Gary Davison) named design entities and assigned development sizes in accordance with the standard outlined in Table 16-17. Drive widths were selected so that ground support could be properly installed. There is potential to reduce the minimum development drive width to 2.0 m wide (mW) to minimise dilution and this scenario should be investigated at the PFS level. The QP (Gary Davison) notes that MSOs were run for the smaller mining width and resulted in an overall potentially mineable inventory increase from 3.70 g/t AuEq to 3.88 g/t AuEq.

Table 16-17: Activity Type (ACT_TYPE) convention used in schedule.

ACT_TYPE	Meaning	Profile
DEC	Decline / Incline	3.5 mW × 4.0 mH
ACC	Level Access	3.5 mW × 4.0 mH
VAD	Ventilation Access Drive	3.5 mW × 3.5 mH
ERA	Escapeway Access	3.0 mW × 3.5 mH
SP	Stockpile	3.5 mW × 4.5 mH
SMP	Sump	3.0 mW × 3.5 mH
XC	Crosscut	3.0 mW × 3.5 mH
PMP	Pump Chamber	3.0 mW × 3.5 mH
SUB	Substation	3.0 mW × 3.5 mH
RC	Refuge Chamber	3.0 mW × 3.5 mH
OD	Ore Drive	3.0 mW × 3.5 mH
VR	Vent Rise	3.5 mD
LDW	Ladderway	1.4 mD

Level accesses (Figure 16-17) were designed with 2:1 pillars between horizontal capital development where possible. Each access typically contained the following essential infrastructure:

- a ventilation return connection;
- a 20-m stockpile (4.5 mH to allow truck tipping);
- an escapeway or refuge chamber cuddy;

- cross-cuts (ramps) to access the deposit; and
- a sump for water collection and pump location.

It was assumed that electrical infrastructure (e.g. a distribution board or jumbo boxes) could be installed in the escapeway access drive. Stockpiles were designed along the decline every 150 m. Substations and pump chambers were also designed as cuddies off the decline.

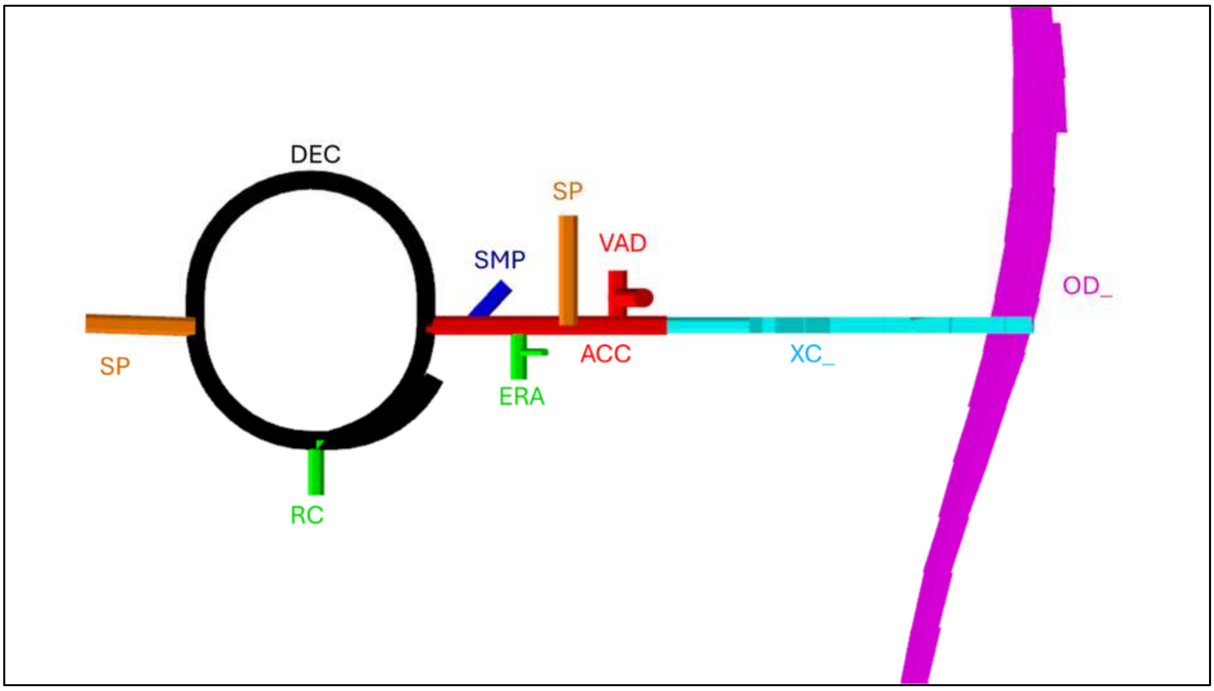


Figure 16-17: Level access layout.

16.2.1.3 Decline Design and Portal Locations

The decline was positioned to best access some early high-grade material in the upper sections of the mine. The decline stands off from the deposit by ~90 m. This allows horizontal offset for:

- 1:7 ramps to reach the six lifts in each access (42 m required); and
- 50 m off access development, allowing for pillars.

Figure 16-18 illustrates the decline location and the ore drive shapes coloured by grade. The high-grade northern pocket of potentially mineable material is also targeted as early as possible; however, the QP (Gary Davison) notes that this high-grade material is unclassified by Mineral Resource category, so it was not prioritised over the sufficiently drilled southern area.

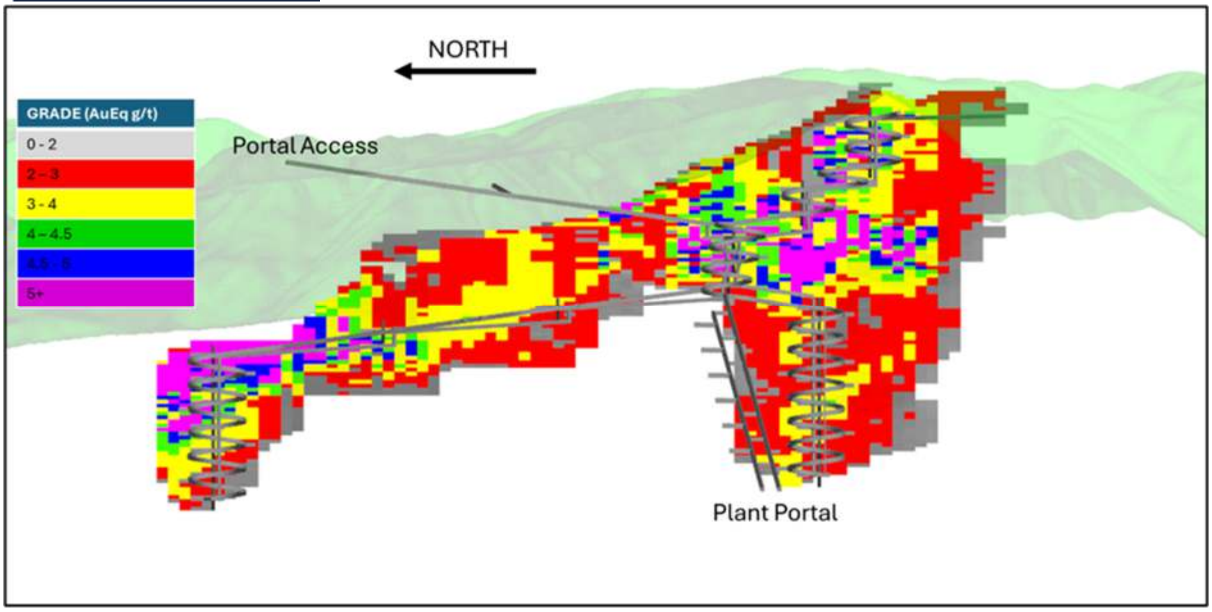


Figure 16-18: Decline design to target early high-grade material.

The first portal location (Figure 16-19) was determined considering:

- the earliest possible access to high-grade material, and
- feasible location on surface relative to the existing Forestry Road and public bike path.

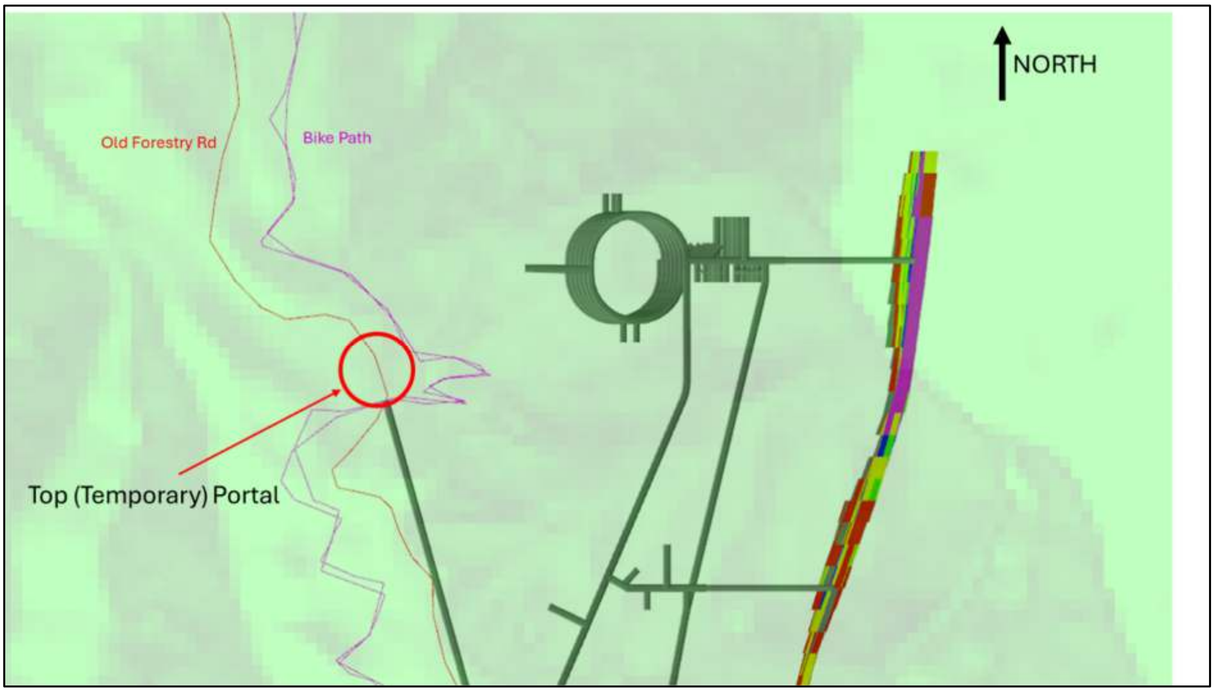


Figure 16-19: Temporary portal location.

The QP (Gary Davison) considers that, to reduce surface disturbance and tonne-kilometres (TKMs), there would be need for an additional portal closer in proximity to the processing plant. This additional portal is illustrated in Figure 16-20. Note that a 1 km-long twin decline must be mined (for ventilation of the active heading) to connect this portal location to the decline design. The QP (Gary Davison) recommends that, at the PFS level, this drive should be optimised. It is possible that some of the twin-drive could be removed, saving on development costs.

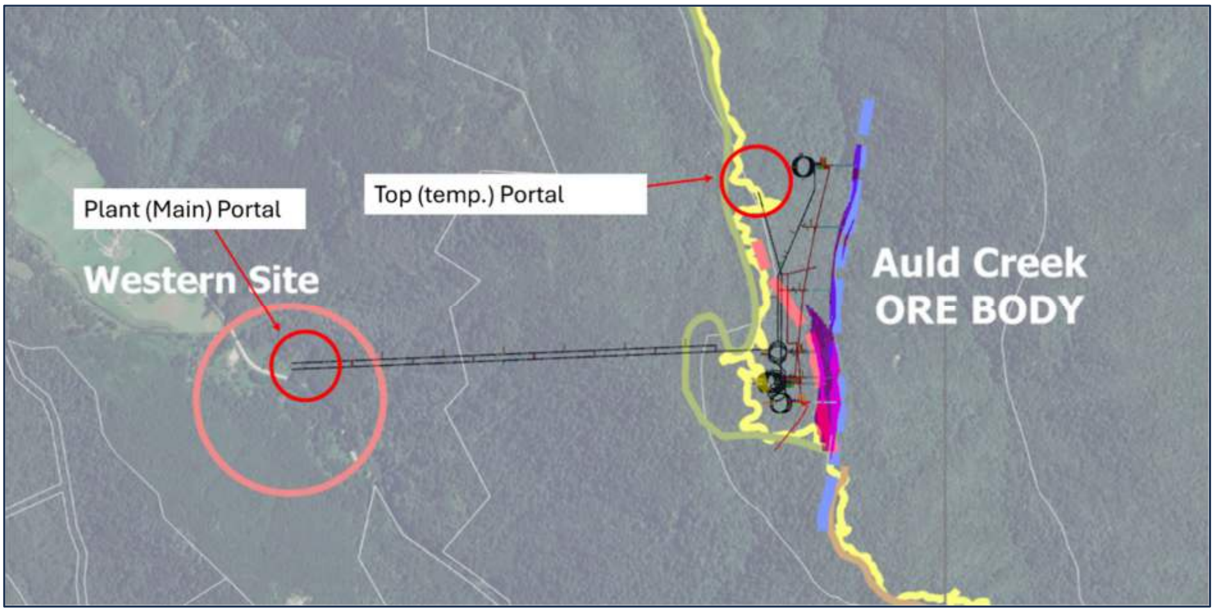


Figure 16-20: Process plant portal.

16.2.1.4 Wider Areas of the Deposit

Some areas of the deposit (particularly those which are well drilled) are much wider than the deposit's average width. In these areas, a single development drive along strike will not be sufficient to extract the entire deposit. The QP (Gary Davison) recommends an innovative mining method known as Shotcrete Drift-and-Fill (SDAF) for these areas (Figure 16-21).

Figure 16-21: Mining method: Shotcrete Drift-and-Fill (SDAF).

SDAF is a low-cost variant of Drift-and-Fill (DAF) mining, using shotcrete to separate drives and eliminate the need for cemented fill or the leaving of pillars. Note that this method would require the use of sprayer rigs and agitator trucks.

16.2.2 Sub-Level Open Stoping Design

MSOs were compiled for stopes in line with the geotechnical analysis. The MSOs were constrained to an HR of 2.5, corresponding to equivalent overbreak (ELOS) of 0.5 m. The resultant MSOs are summarised in Table 16-18.

Table 16-18: SLOS MSO scenarios.

MSO Scenario	COG (g/t AuEq)	Tonnes (Mt)	Grade (g/t AuEq)
10 mW × 10 mH	1.5	1.94	3.56
9 mW × 12 mH	1.5	1.95	3.53

Using industry benchmark rates for vertical development advance per annum, the QP (Gary Davison) provided approximate annual metal production, potentially mineable inventory production, and mine life for the SLOS option in Table 16-19. Crown pillar tonnes were removed from this calculation. These indicative rates make no allowance for unplanned dilution or mining recovery. The advantages and disadvantages of the SLOS option are listed in Table 16-20.

Table 16-19: SLOS potential production.

Mining Method	MSO Tonnes (@1.5 g/t) – 20 m Crown	Grade (g/t)	TPVM[1]	Estimated Production (ktpa) Low - High	Mine Life (Yrs)
SLOS	1.7	3.53	5,099	250–350	5–6

[1]TPVM = Tonnes per vertical metre of deposit depth.

Table 16-20 : Advantages and disadvantages of the SLOS mining method.

Advantages of SLOS (Compared to OHCAF)	Disadvantages of SLOS
Potentially higher production rates.	Dilution > 1.0m would result in a worse schedule in terms of grade.
Simpler ventilation and fewer required active working areas.	Narrow deposit and poor ground conditions means stope extraction is challenging.
Fewer jumbos required, smaller fleet, lower operational unit cost.	Cemented fill must be used throughout, or pillars left in situ which would diminish project value.

A trade-off between the OHCAF and SLOS schedule will only be fully understood once a 3D schedule is created for the SLOS option. The QP (Gary Davison) recommends that this be completed at the PFS level, owing to the timeframe restrictions on this PEA study. At this stage, the QP (Gary Davison) considers that the OHCAF and SLOS schedules are likely to have similar project value.

16.2.3 Ventilation

The proposed design contains three exhaust points:

- two exhausts at the top of mine (north and south); and
- one exhaust near the processing plant which is used for mining of the plant haulage drive.

Figure 16-22 depicts a long-section of the ventilation system, including the locations of intakes and exhausts; these accesses are all adits. No ventilation shafts are required for this mine design. Additionally, the QP (Gary Davison) has designed these ventilation adits to break through from the underground to the surface, so there will be no additional requirement for haul road access to mine these exhausts.

The North Exhaust intersects the surface at existing drill pad #4 and the South Exhaust intersects the surface at drill pad #20. Using sites with existing access agreements that are already cleared of vegetation will make the permitting process simpler.

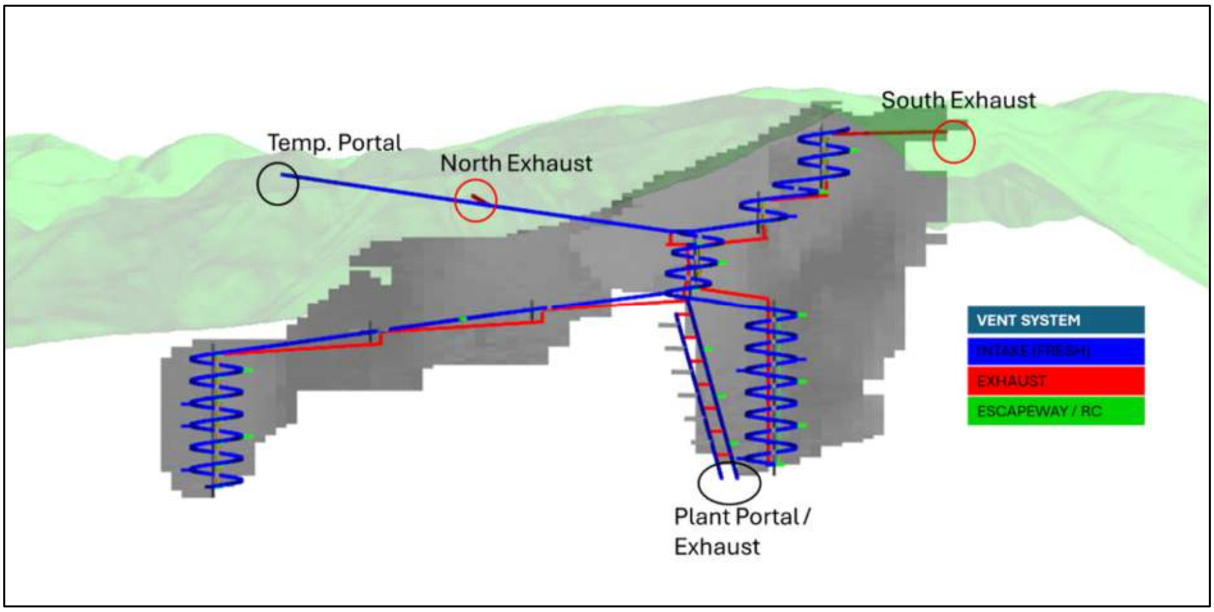


Figure 16-22: Ventilation system.

16.2.3.1 Ventilation Design

With respect to the New Zealand ventilation guidelines (*Health and Safety at Work (Mining Operations and Quarrying Operations) Regulations, 2016*), the QP (Gary Davison) notes that the following are essential to a safe design:

- Minimum airflow (secondary heading) must be 0.3 m^3/s for each m^2 of development profile (e.g. for a 3.0 mW × 3.0 mH heading, the minimum flow would be 9.0 m^2 × 0.3 m^3/s = 0.27 m^3/s).
- Volume of air is not less than 0.05 m^3/s per rated diesel engine power or not less than 3.5 m^3/s (whichever is greater).

Based on the requirement for 0.05 m^3/s per kW of rated diesel engine power, and the scheduled fleet requirements, primary flow demand was calculated (Table 16-21). A utilisation factor was applied to each piece of machinery. This factor represents the portion of a shift that the machine would be using its diesel motor and is based on Mining One's previous ventilation studies.

Table 16-21: Primary flow requirements based on diesel engine power.

Equipment	Number	Utilisation Factor	Engine Size (kW)	Volume Required (m^3/s)
Light Vehicles	16	0.50	151	60
Sandvik TH320 UG Haul Truck (20 t)	3	1.00	235	35
Sandvik LH307 UG Loader (7 t)	3	0.95	160	23
Sandvik LH203 UG Loader (3 t)	10	0.95	71	34
Volvo Integrated Tool Carrier (IT)	3	0.60	170	15
Agitator	2	0.80	185	15
MUKI FF NV Jumbo	9	0.35	55	9
Narrow Vein Charge Basket (JCB)	3	0.50	50	4
Sandvik DD321 Twin-Boom Jumbo	2	0.30	124	4
Fibrecrete Sprayer	2	0.30	120	4
Normet Charmec SF404	2	0.35	96	3
Total	55			205 m^3/s

The QP (Gary Davison) then assumed the portion of flow that would be attributable to each portal (Table 16-22).

Table 16-22: Primary exhaust weighting assumptions.

Primary Exhaust	Portion of Flow (Estimated)	Q (m^3/s)
Plant (Main) Portal	0.3	61.5
North Portal	0.4	82
South Portal	0.3	61.5

Using the volumes derived in Table 16-21, pressure loss through the ventilation system was calculated from first principles, assuming one fan at each portal location. The resultant pressure calculations are captured in Figure 16-23.

Airway description	Width [m]	Height [m]	z_entry [m]	z_exit [m]	NVP Case	Δρ [kg/m3]	P [m]	A [m^2]	L [m]	k [Ns^2/m^4]	R [Ns^2/m^8]	Q [m^3/s]	V (m/s)	Frictional Presure Loss
North Decline (In)	3.5	4	270	370	Low NVP	0.0324	15	14	2183	0.00917	0.1094	82	5.9	736
North Vent Return	3.5	4	370	270	High NVP	0.0972	15	14	962	0.00917	0.0482	82	5.9	324
Plant Decline Return	3.5	4	0	270	High NVP	0.0972	15	14	1000	0.00917	0.0501	61.5	4.4	190
Plant Decline (In)	3.5	4	270	370	Med NVP	0.0648	15	14	1000	0.00917	0.0501	61.5	4.4	190
Southern Exhaust	3.5	4	370	270	High NVP	0.0972	15	14	621	0.00917	0.0311	61.5	4.4	118
Southern Decline (In)	3.5	4	270	370	Med NVP	0.0648	15	14	2328	0.00917	0.1167	61.5	4.4	441
Total														1998

Figure 16-23: Primary system pressure loss calculations.

The approximate duty required for the Auld Creek OHCAF design is 205 m^3 at 1,998 Pa.

At this early PEA stage, the primary fan requirements are estimated in Table 16-23. At the PFS level, ventilation modelling (in specialist software such as Ventsim™) is required to have full confidence in these calculations.

Table 16-23: Primary fan requirement.

Primary Exhaust	Pressure (Pa)	Q (m^3/s)	Fan Details
Plant (Main) Portal	380	61.5	1 × 90 kW
North Portal	1,060	82	2 × 90 kW
South Portal	559	61.5	1 × 90 kW

16.2.4 Egress and Entrapment

The New Zealand mining regulations (2016) state that before production operations begin in an underground metalliferous mine, the mine operator must ensure that there are at least two trafficable exits on foot from the mine workings.

The North Exhaust and South Exhaust (once mined to surface) could offer second and third means of egress, respectively. The plant haulage ramp also offers an additional means of egress from the mine workings. The QP (Gary Davison) has also included provisions in the schedule for a connected system of raise-bored ladderways which connect the level accesses. Where it was not possible to connect the level access, development has been allocated for refuge chambers.

Additionally, New Zealand mining regulations state that mining of a single-entry heading for more than 200 m is considered a high-risk activity and is notifiable to the mining authority. Risk assessment would be the best course of action for mining blind headings and would involve the use of refuge chambers in most ore drives.

16.2.5 Materials Handling

There are two distinct phases for the mine in terms of handling material:

1. Haulage of waste from the working face to the temporary surface stockpiles (in the mountainous region) and haulage of ore from the top portal to a temporary run-of-mine (ROM) pad (ideally near the planned main portal).
2. Haulage from the working areas to the plant decline and eventually to the dedicated ROM set up near the processing plant (only available once the plant decline has connected with the main decline).

Figure 16-24 illustrates the breakdown of material haul destination across the mine design. At this stage, the QP (Gary Davison) recommends that all material be hauled to surface using a fleet of underground trucks (Sandvik TR315 or similar). High-CAPEX, bulk materials handling methods (such as a hoist or conveyor) were discounted at this stage owing to the relatively small deposit. However, the QP (Gary Davison) recommends revisiting other materials handling options at the PFS stage of study.

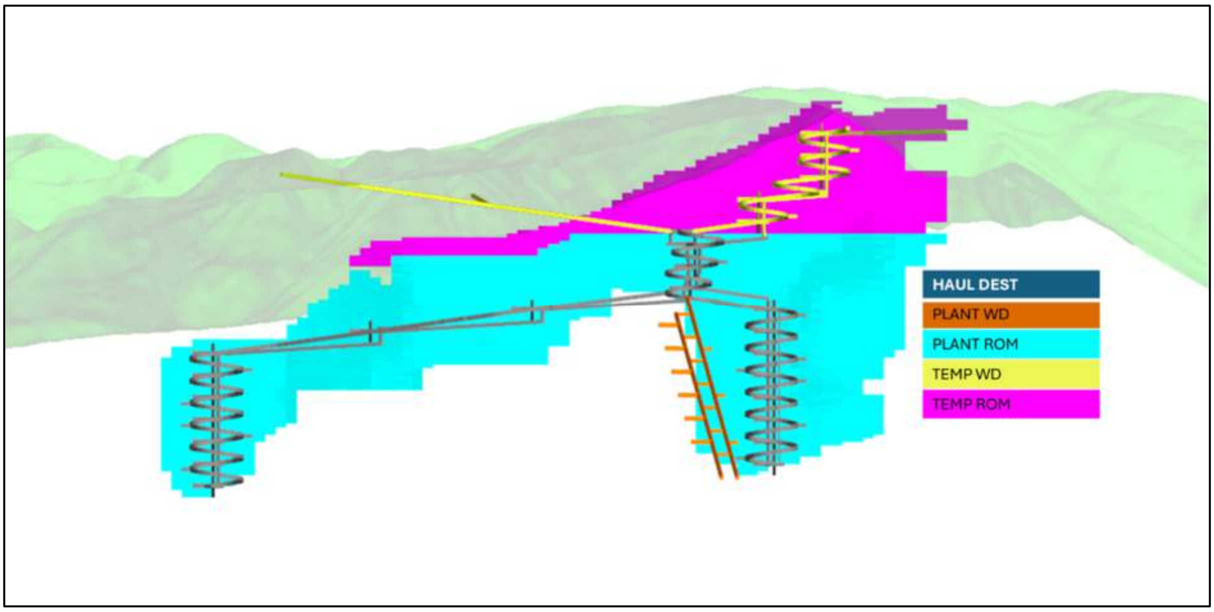


Figure 16-24: Mine design depicting materials haul destination.

Haulage distances were input as global constants into the mine schedule and used to calculate tonne-kilometres (TKMs) (Table 16-24) based on the preliminary dump locations in Figure 16-25. Design for these dumps is further discussed in Section 18.

Table 16-24: Surface haulage distances.

Global Constant	Distance (m)
Plant Portal to Plant ROM	350
Plant Portal to Plant Waste Dump	400
Temporary Portal to Temporary ROM	4,000
Temporary Portal to Temporary Waste Dump	900

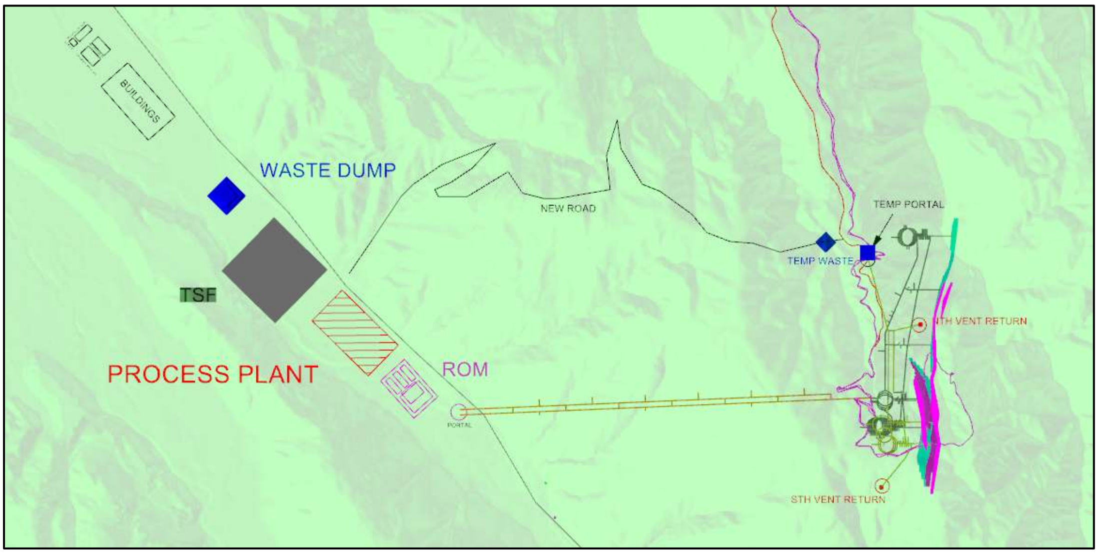


Figure 16-25: Concept surface infrastructure size and locations.

Material must be hauled to the ROM pad from the temporary portal via a new haul road until the plant decline intersects the main decline. Allowance has been made for this road in the financial model, but the design and location have not been finalised. Based on the processing schedule (compiled by Mining One in the economic model), ~50 kt of storage is required at the temporary ROM pad while the processing plant is built and commissioned. An additional ~35,000 m^3 of waste storage is required for development of the temporary (top) portal.

16.2.6 Potential to Recover Crown Pillar

The QP (Gary Davison) recommends that a crown pillar of 20 m be left in situ (Figure 16-26). Based on the OHCAF MSO scenarios, this represents 235 kt of material at 4.0 g/t AuEq. This material could be, at least partially, extracted at the start of the mine life, providing some early value to the operation.

This material could be mined from accesses originating from existing drill pads (#10, 12, 14, 16, and 19 are well positioned to access the deposit). All provided schedules in this PEA assumed that the crown pillar can be mostly extracted (applying a recovery factor of 75%).

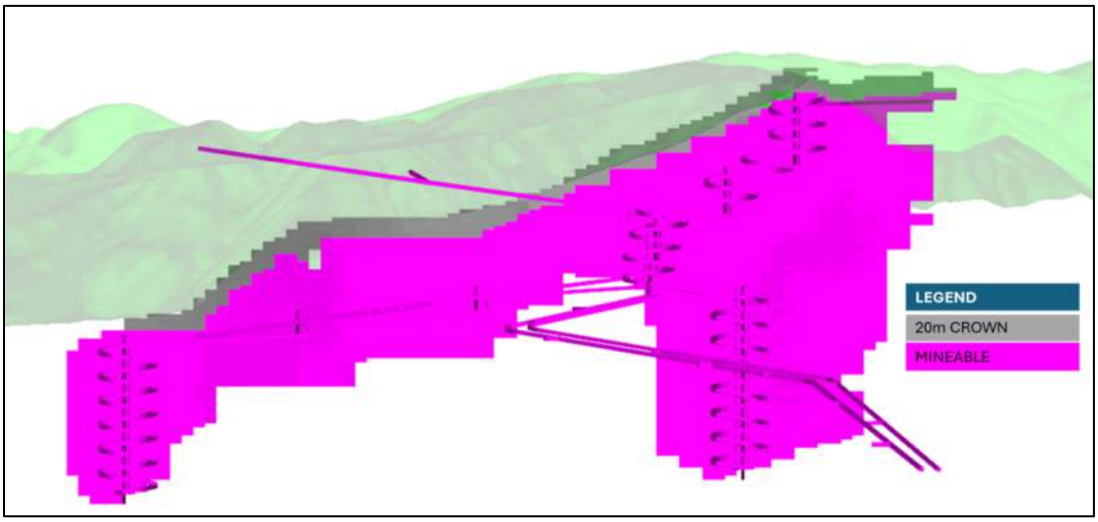


Figure 16-26: Crown pillar of 20 m.

16.2.7 PFS Mine Design Opportunities

The QP (Gary Davison) recommends that the following be explored at the PFS level; these factors represent potential upside to the production rates and grades of the operation:

- Minimum mining width could be reduced to 1.8–2.0 m. This presents operational challenges with installing ground support but use in other operating mines provides evidence to support the concept.
- A trade-off study between cement usage and number of lifts per access to optimise CAF productivity (accesses with multiple internal ramps provide more concurrent working areas which could improve productivity but increases the demand for cemented fill in sill lifts).
- A full mine design and schedule for the stoping operation, which the QP (Gary Davison) expects to be more productive in terms of annual production, but carries more operational risk when compared to the OHCAF option (dilution and ground conditions).

16.3 Mine Schedule

16.3.1 Schedule Set-Up

This section summarises the OHCAF schedule assumptions and outputs for the base case. Table 16-25 lists the key schedule inputs.

Table 16-25: Schedule assumptions.

Task	Resource	Task Rate	Resource Rate	UoM
Capital Development	Jumbo	100	150	m/mo
Ore Development	MUKI FF	90	90	m/mo
Ladderway	Raisebore	5	5	m/day
Vent Rise[1]	Raisebore	1	1	m/day
RF Backfilling	LH203	120	120	t/day
CRF Backfilling	LH203	100	100	t/day
Tailings Backfilling	LH203	100	100	t/day

Notes:

1. *Raisebore rate for ventilation rises was calculated to include pilot hole drilling, reaming and shrink supporting of the raise.*

The schedule was also levelled using the quantity constraints in Table 16-26.

Table 16-26: Quantity constraints.

Key Schedule Driver	Constraint	UoM
Total Production Tonnes	250	ktpa

The global constants captured in Table 16-27 were also used in generation of the schedule.

Table 16-27: Global constants.

Key Schedule Driver	Amount	UoM
Total Production Tonnes	250	ktpa
C&F Dilution[1]	0	%
Mining Recovery	95	%
Waste Development Overbreak	10	%
Cement Content	5	%
Swell Factor	30	%
Process Recovery (Au)	95	%
Process Recovery (Sb)	85	%
Surface Haul: Temp. Portal – Temp. ROM	4,000	m
Surface Haul: Temp. Portal - WD	900	m
Surface Haul: Plant Portal - Plant ROM	350	m

Key Schedule Driver	Amount	UoM
Surface Haul: Plant Portal - Plant WD	400	m
Troy Ounce Conversion	31.10348	g/oz
Ventilation Design Contingency[2]	25	%

Notes:

1. *In the OHCAF schedule, dilution is assumed to be accounted for by minimum mining width of 3 m.*
2. *Ventilation design contingency applies an extra 25% of development length to all ventilation access drives (VADs) to allow for some dog-legs in a more realistic design.*

16.3.2 Schedule Outputs

Schedule key outcomes for the base case are summarised in Figure 16-27 through Figure 16-32. Figure 16-30 refers to recovered metal, and this is derived from the processing schedule in the economic model, assuming 95% processing recovery. It was assumed that mining can start in January 2028 and material will only be processed starting March 2028 (allowing for a small ~50 kt production material stockpile for processing plant commissioning).

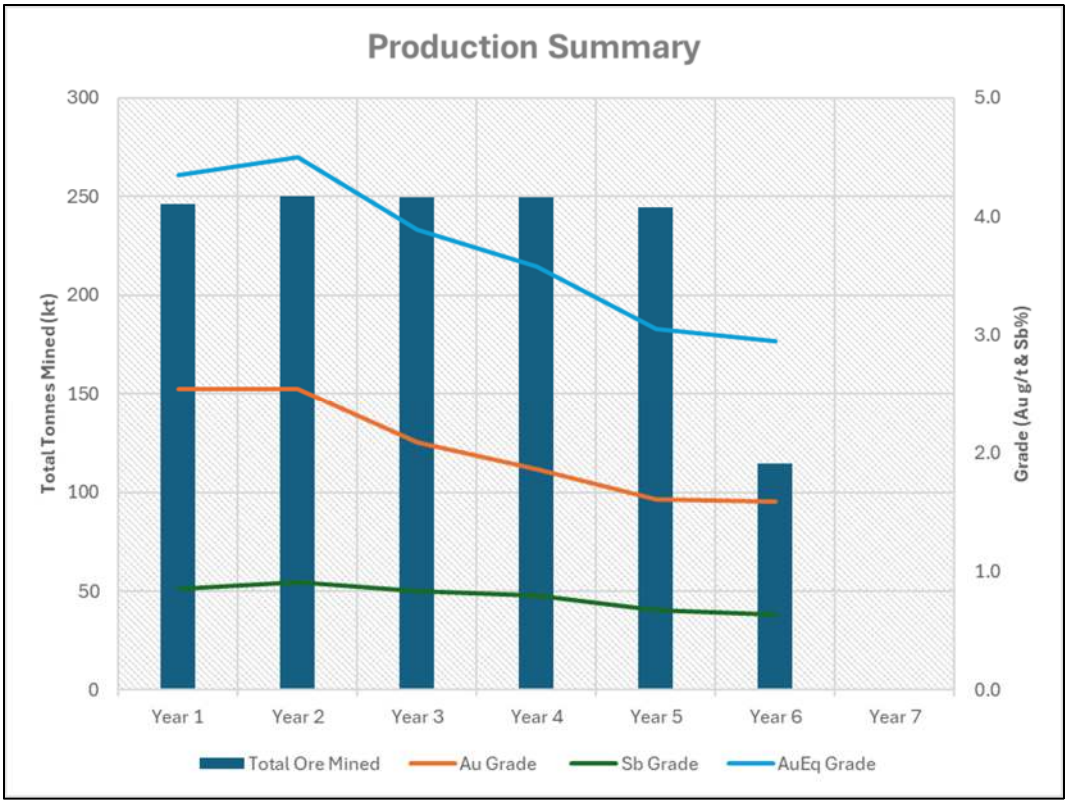


Figure 16-27: Production summary.

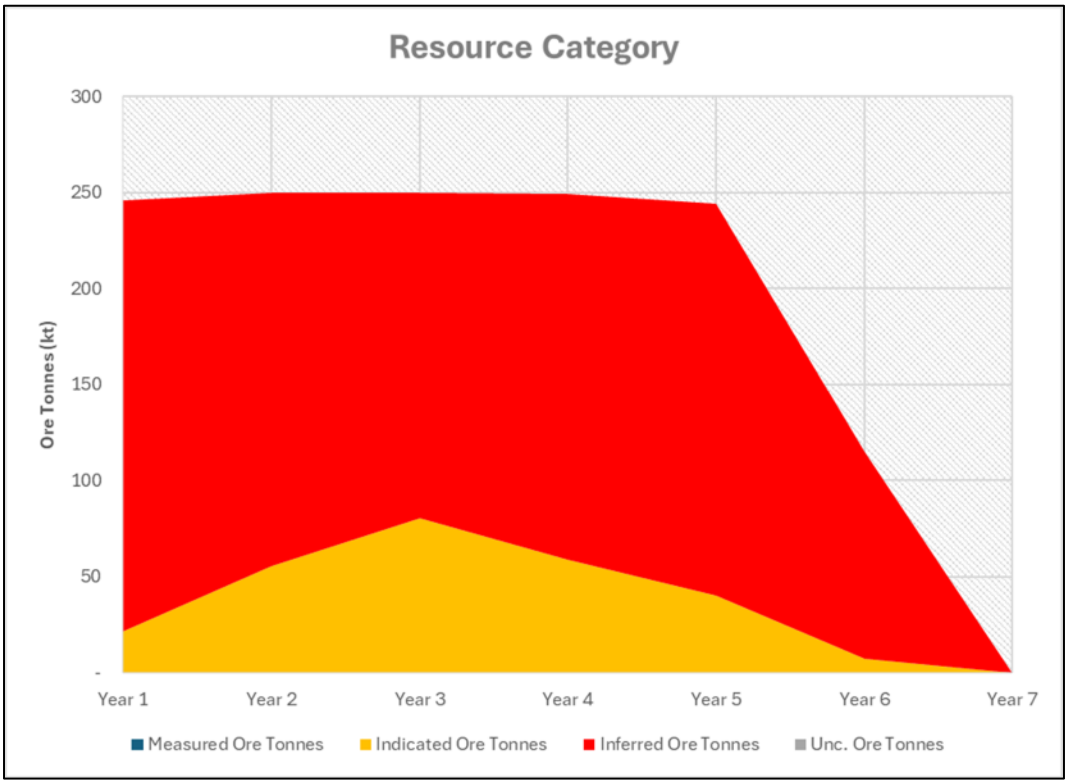


Figure 16-28: Mineral Resource classification.

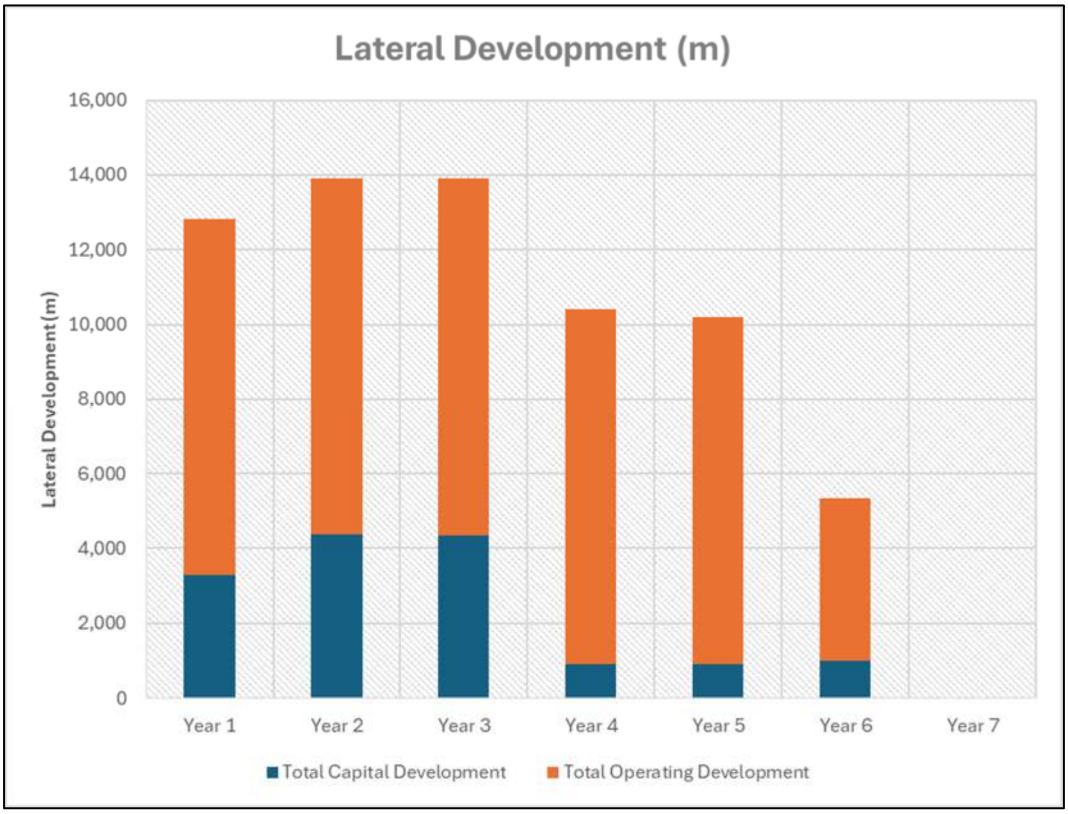


Figure 16-29: Lateral development summary.

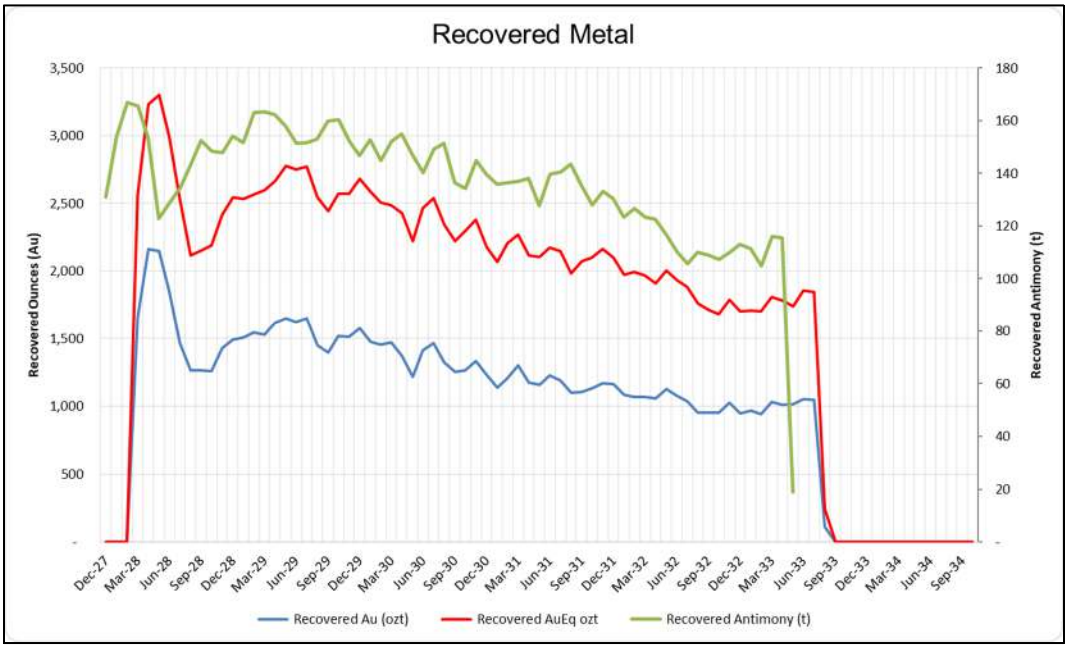


Figure 16-30: Recovered metal.

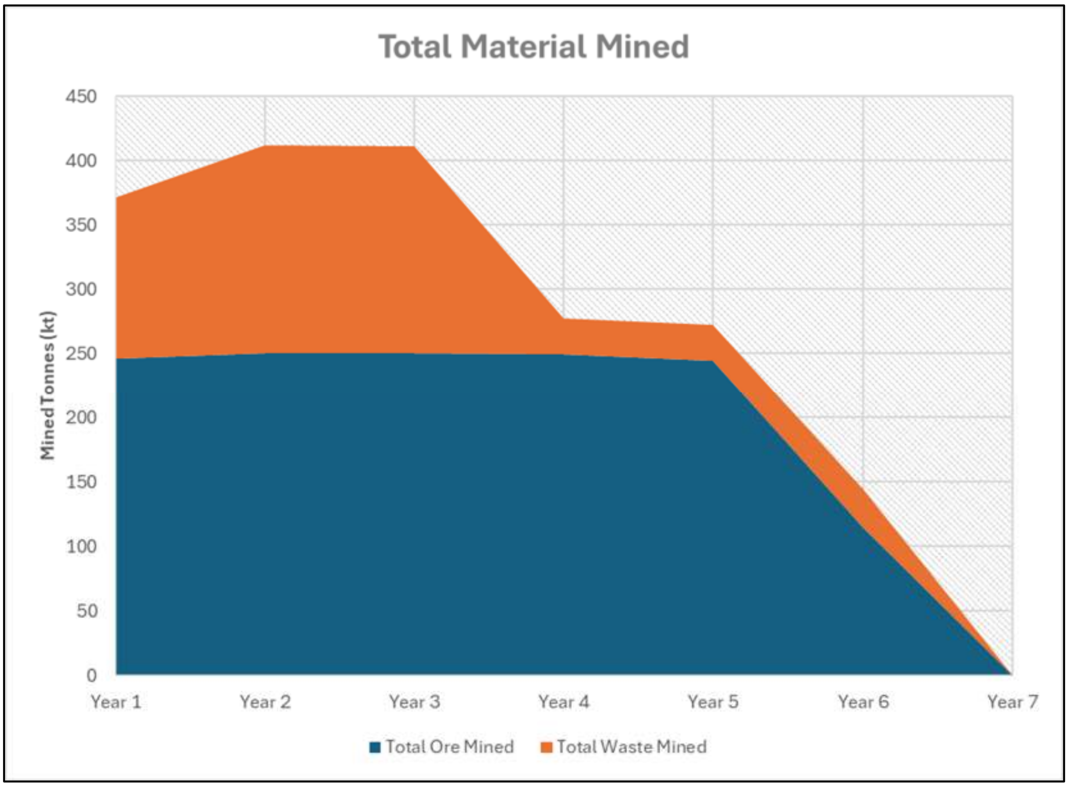


Figure 16-31: Total material mined.

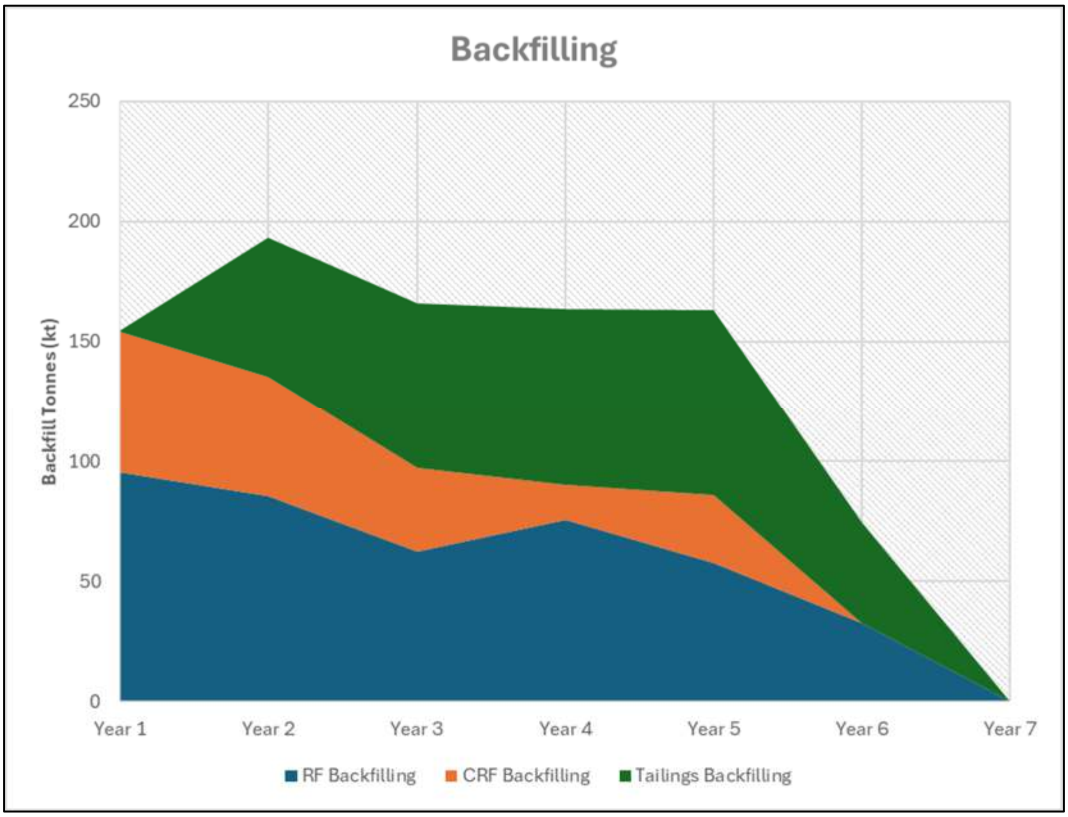


Figure 16-32: Backfilling quantities.

Table 16-28: Production summary.

KPI	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	TOTAL	UoM
Total Capital Development	3,304	4,370	4,354	906	915	985	3,304	m
Total Operating Development	9,505	9,537	9,544	9,499	9,288	4,347	9,505	m
Total Lateral Development	12,808	13,908	13,898	10,405	10,203	5,332	12,808	m
Total Vertical Development	176	91	224	159	0	0	176	m
Total Production	246	250	250	249	244	115	246	kt
Au Grade	2.5	2.5	2.1	1.9	1.6	1.6	2.5	g/t
Sb Grade	0.8	0.9	0.8	0.8	0.7	0.6	0.8	%
AuEq Grade	4.4	4.5	3.9	3.6	3.0	2.9	4.4	g/t
Au Mined Metal	20	20	17	15	13	6	20	koz
Sb Mined Metal	2,076	2,282	2,088	1,986	1,642	725	2,076	t
AuEq Mined Metal	34	36	31	29	24	11	34	koz
Total Waste Mined	125	162	161	28	28	30	125	kt
TMM	372	415	415	280	272	145	372	kt
RF Backfilling	95	86	62	76	57	32	95	kt
CRF Backfilling	59	50	35	15	28	0	59	kt
Tailings Backfilling	0	58	69	74	77	42	0	kt
Prod. TKM	1,034,878	963,672	693,178	378,989	299,255	94,568	1,034,878	TKM
Waste TKM	110,132	40,126	73,458	11,761	10,838	11,670	110,132	TKM
Total TKM	1,145,010	1,094,253	767,496	390,750	310,093	106,238	1,145,010	TKM
Measured Prod. Tonnes	21	55	80	59	40	7	21	kt
Indicated Prod. Tonnes	225	195	170	191	204	107	225	kt
Inferred Prod. Tonnes	3,304	4,370	4,354	906	915	985	3,304	kt

16.3.2.1 Waste Deficit

The mined waste balanced against backfill demand is presented in Figure 16-33. Mined waste cannot fulfil the CRF and RF schedule demand for the OHCAF scenario as it falls short by ~60 kt. Options to offset this waste deficit include increasing the size of capital development profiles or using tailings as the primary fill medium underground, or quarrying waste material from the surrounding areas. These options are offered as potential schedule scenarios in Section 22.

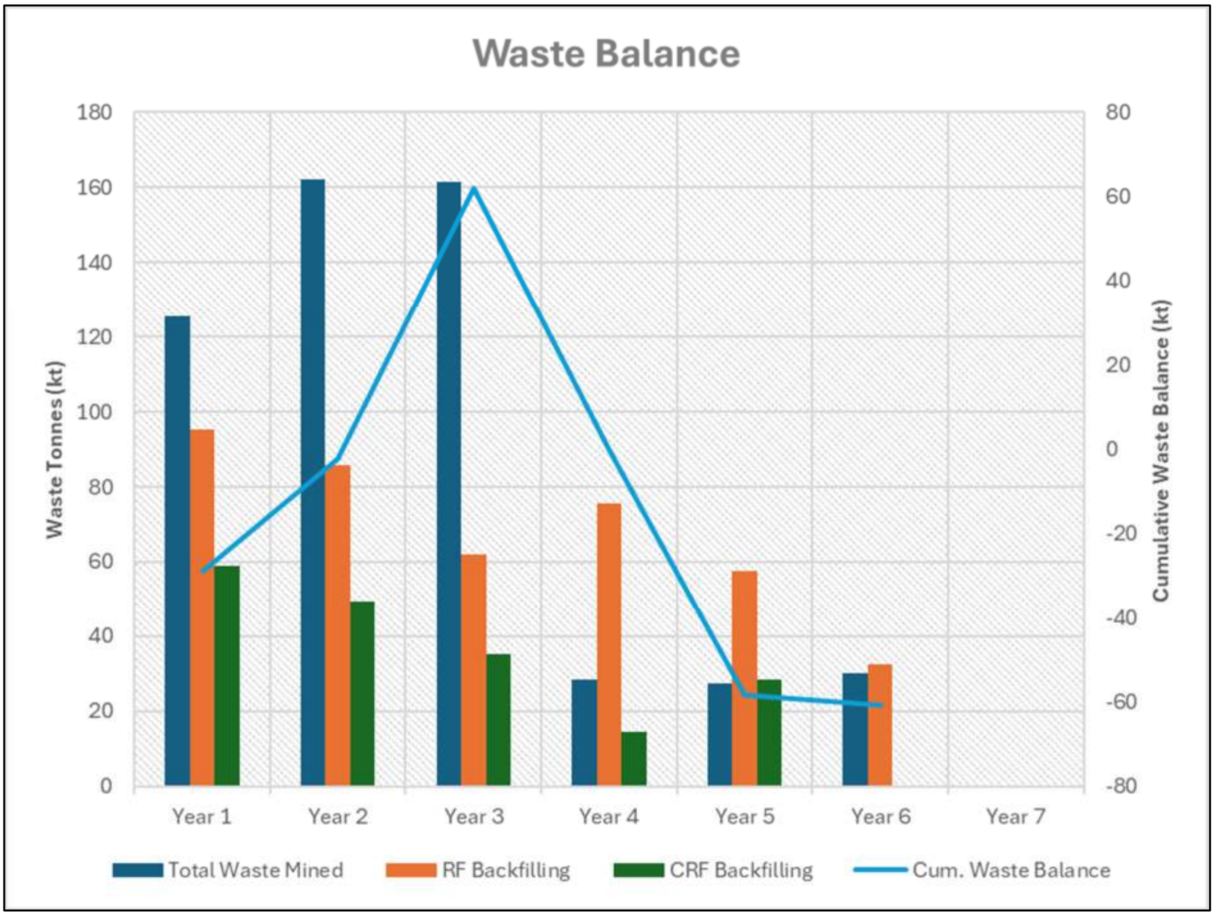


Figure 16-33: Waste balance.

16.3.2.2 Required Mining Fleet

Narrow-vein jumbo, twin-boom jumbo, raisebore, and narrow-vein loader requirements were exported directly from the 3D schedule using a work summary. Other machines (e.g. trucks, charge wagons, and fibrecrete sprayers) summarised in Table 16-29 were calculated using cycle times.

Table 16-29 : Required mining fleet summary.

FY	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
MUKI FF Jumbo	9	9	9	9	9	9	2
Raisebore	1	1	1	1	-	-	-
Sandvik LH203	8	10	8	8	8	8	2
Twin-Boom Jumbo	2	2	2	1	-	-	-
UG Haul Trucks	3	3	3	3	2	2	
Larger Loaders	3	3	3	3	2	2	1
Agitator	2	2	2	1	1	1	1
Sprayer	2	2	2	1	1	1	1

FY	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Cablebolter	0	0	0	0	0	0	1
ITC	3	3	3	3	3	3	0
Light Vehicles	16	16	16	16	16	16	3
Charge Machine	2	2	2	1	1	1	16
Small IT (JCB)	3	3	3	3	3	3	1

17 Recovery Methods

17.1 Battery Limits

The battery limits for processing plant design are as follows.

- ROM ore delivered to ROM dump hopper or crusher pocket by haul truck or wheel front end loader. Sufficient room is allowed on the ROM pad to facilitate ore blending.
- Antimony concentrate that has been filtered, bagged, sampled, registered and is stored within the concentrate storage shed.
- Gold concentrate which has been filtered, bagged, sampled, registered and is stored within the concentrate storage shed.
- Trash from crushing, grinding, and flotation areas into bins or holding bays.
- Filtered processing plant tailings discharged onto a dedicated filtered tailings stockpile at the processing plant, before transport to a dry-stack TSF.
- Paste discharge at discharge flange of positive displacement paste pump (no surface or underground reticulation included).
- Reagents, consumables and fuel delivered to stores or receiving area.
- Raw water delivered into respective holding tanks within the plant site.
- Excess process water to water treatment.

17.2 Principal Sources of Information

The principal sources of information used for the PEA processing plant design are listed below.

- *"ALS Metallurgical Test Work Report: Report No. A25728, Metallurgical Test Work Conducted Upon Auld Creek Project Ore Samples, August 2024".*
- *"ALS Metallurgical Test Work Report: Report No. 101824-MIN6956 (A25728), Quantitative Automated Mineralogical Analysis Conducted on One Rougher Concentrate from the Auld Creek Project, October 2024".*
- *"University of Tasmania Report: P1824. Gold Contents of Stibnite in One Sample for ALS (101824-MIN6956_A25728_LAICPMS_Report), October 2024".*
- *"Auld Creek Metallurgy Update, Leo Consulting, August 2024".*
- *"Auld Creek Metallurgy Update Rev 1, Leo Consulting, August 2024".*
- *"Auld Creek Cleaner Flotation on AC003, Leo Consulting, September 2024".*
- Pitch Black Group Pty Ltd benchmarking data, industry data and recommendation.
- Project meeting outcomes and guidance issued by RUA.
- Industry guidelines as provided by international organisations and committees, such as the World Health Organization (WHO) and the International Cyanide Management Code (ICMC).

17.3 Metallurgy Derived Design Criteria

Based on the test work completed to date, sufficient information exists to compile the design basis for the Project to a scoping study level of definition (Table 17-1). Further test work is required to refine the physical parameters, mineralogical understanding, metallurgical response, and metallurgical variability.

In essence, the recovery circuit will comprise a bulk rougher flotation circuit followed by four stages of differential cleaning, to produce a separate Sb concentrate and a separate Au concentrate.

Table 17-1: Metallurgy derived process design criteria.

Parameter	Unit	Value	Source
Ball Mill Work Index – Closing Screen	µm	150	ALS Metallurgical Report A25728
Ball Mill Work Index	kWh/t	16.0	ALS Metallurgical Report A25728
Solids SG		2.86	ALS Metallurgical Report A25728
Flotation Grind Size	µm	106	ALS Metallurgical Report A25728
Rougher Flotation Residence Time - Lab	min	7	
Cleaner Flotation Residence Time - Lab	min	15	

17.4 Process Description

17.4.1 Overview

The Auld Creek processing plant facility includes all ore processing activities from crushing to storage of concentrate in bulk bags (Figure 17-1). It further includes a tailings filtration circuit, with filtered tailings transported to dry-stacked tailings storage areas or used within the paste plant for underground paste backfill production.

The design incorporates a two-stage crushing circuit, crushed ore storage and reclaim, ball milling with cyclone classification, bulk rougher flotation to produce a bulk Sb-Au concentrate, four stages of differential cleaner flotation, Sb concentrate thickening, filtration and bagging, Au concentrate thickening, filtration and bagging, high-rate thickening and filtration of tailings and the ancillary air and water services. A concept-level 3D view of the processing plant is illustrated in Figure 17-2.

The Auld Creek circuit design is based on a preliminary mine plan and sustains an annual Sb concentrate production of 2,798 tpa together with an annual Au production of 14,613 oz through the processing of 0.25 Mtpa of ROM ore feed.

The design basis for the principal process areas is summarised as follows:

- The operating basis for the crushing and ore receipt area will be 365 days, 18 hours per day with continuous two shifts per day. The operating basis for all other areas will be 365 days available per year, with two continuous shifts per day at 12 hours per shift.
- The crushing area will receive and crush ROM material, with a nominal top size of -500 mm, at a nominal rate of 0.25 Mtpa, and produce a 14 mm product for subsequent size reduction within the grinding circuit.
- The mill feed bin will provide 24 hours' total capacity at the nominal milling throughput rate of 0.25 Mtpa. A side discharge ore stockpile will provide additional storage capacity, with material recovered from the stockpile via front end loader.
- The grinding area will produce a flotation feed slurry with a nominal product size of 80% passing 106 µm.
- The bulk rougher flotation circuit will provide sufficient volumetric capacity to sustain a flotation residence time of 15 minutes, at an ore throughput of 0.25 Mtpa, while achieving the desired mass yield to concentrate of 6–8%.
- The four-stage cleaner flotation circuit will provide sufficient volumetric capacity to sustain a flotation residence time of 38 minutes, at an ore throughput of 0.25 Mtpa, while achieving the desired mass yield to final Sb concentrate of 1.4–1.8%. Tailings from the four-stage cleaner circuit will comprise final Au concentrate.
- Antimony concentrate will be thickened prior to filtration. Filtered Sb concentrate will be bagged (1-tonne bags) for shipment off-site.

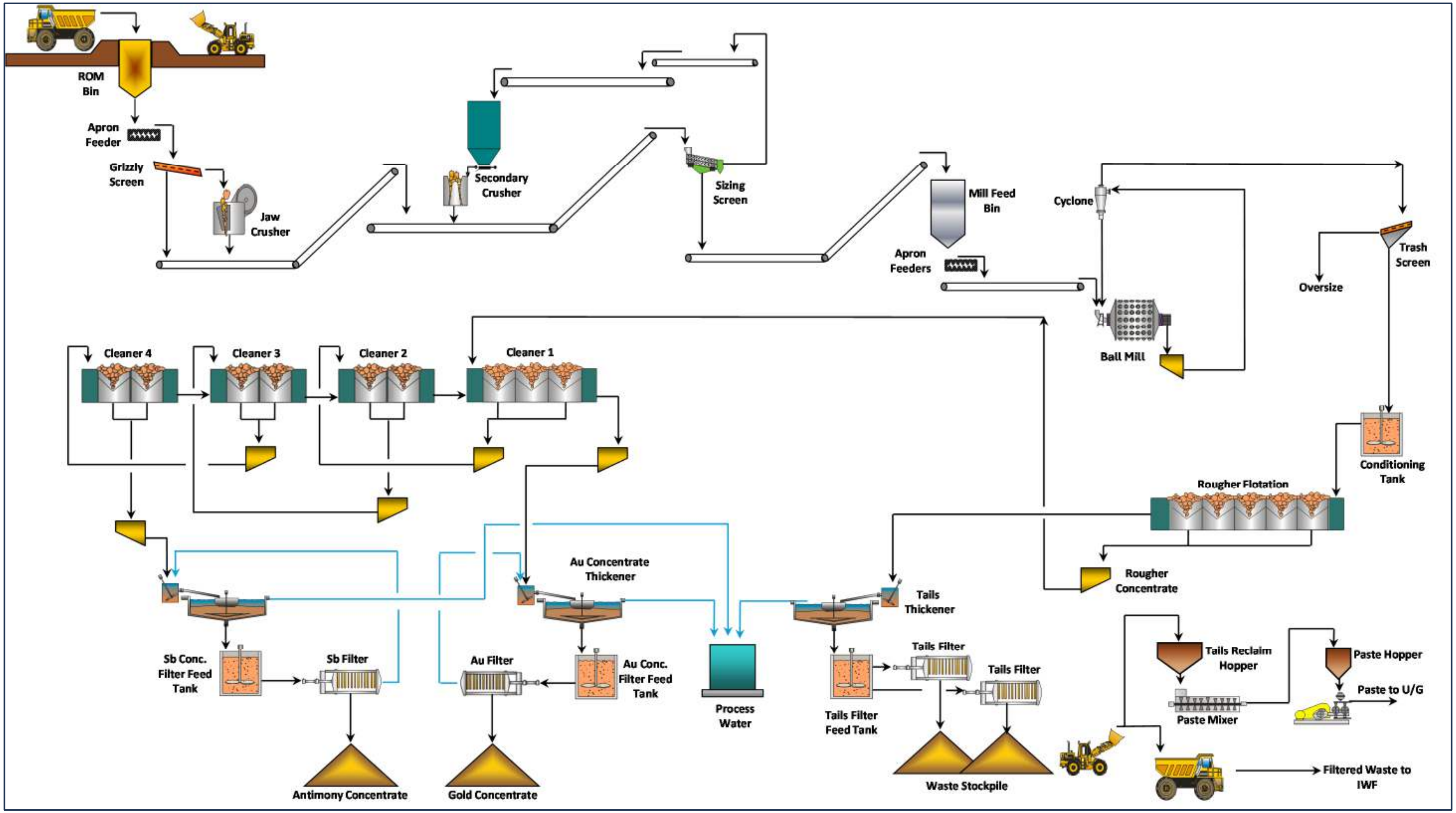


Figure 17-1: Auld Creek processing plant facility schematic flowsheet.

- Gold concentrate will be thickener prior to filtration. Filtered Au concentrate will be bagged (1-tonne bags) for shipment off-site.
- Rougher flotation tailings will be thickened prior to filtration. Filtered tailings will be transported to the integrated waste facility (IWF) for disposal or used for paste backfill production.
- A dedicated spillage handling thickener is incorporated into the design. The small volumetric capacity of the flotation circuit necessitates the controlled return of all spillage to the main processing circuit, to prevent over-dilution of the circuit and unstable flotation circuit operation.

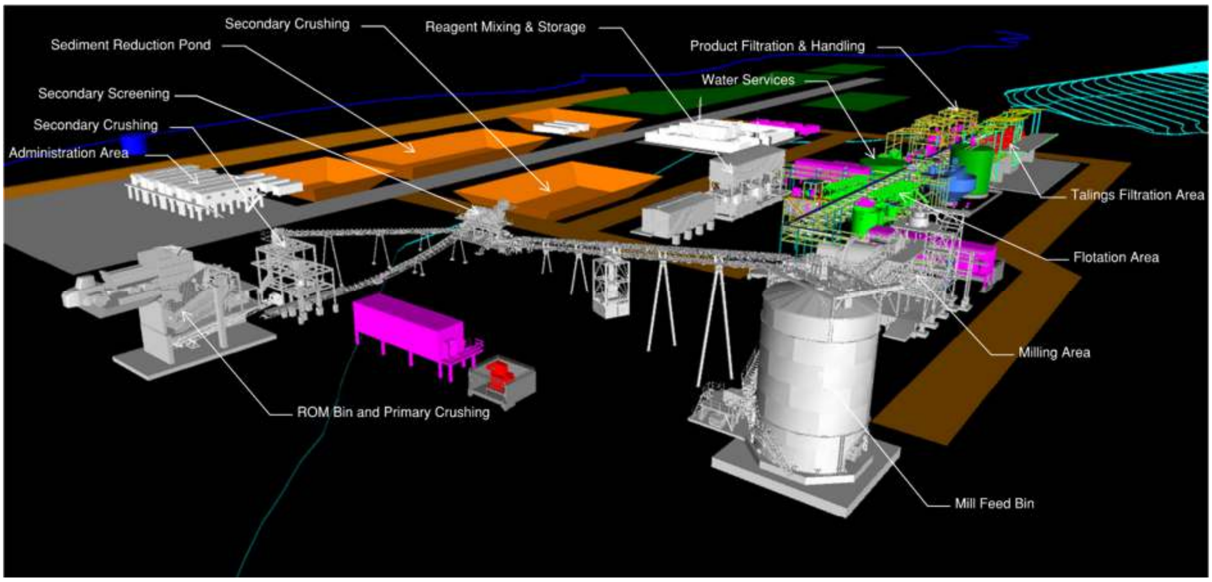


Figure 17-2: 3D view of processing plant (view towards north-northwest).

17.4.2 Crushing

The crushing circuit will operate 18 hours per day, 365 days per year, with an availability of 70%. This is equivalent to operating 4,599 hours per annum at a nominal crushing rate of 54 tonnes per hour (tph).

ROM ore from the underground mine, with a maximum lump size of 500 mm and a P_{80} size of 250 mm, will be delivered to the ROM bin by reardumping articulated haul trucks. The dump pocket will be designed to allow single truck dumping and will have the capacity for two truckloads (40 t). The ROM bin will be fitted with a 700 mm static grizzly to prevent the ingress of oversize ore. A mobile rock breaker will be used to manage ROM oversize.

The ROM pad will have 10,000 t of ROM ore storage capacity and ore not direct tipped to the ROM bin will be stored on these stockpiles. Reclaim from these stockpiles to the ROM bin will be via front end loader.

ROM ore will be withdrawn from the ROM bin by an apron feeder, onto a vibrating grizzly screen, with 75 mm bar spacing. Oversize material from the grizzly screen will discharge directly to the open circuit primary jaw crusher. The C80 jaw crusher can accept a maximum lump size of 600 mm and has a maximum rated capacity of 110 tph at the selected jaw crusher closed side setting of 80 mm. Jaw crusher product (P_{80} 80 mm) will discharge onto the primary crusher discharge conveyor, where it will re-combine with the grizzly feeder undersize, and be conveyed to the double deck primary screen, via the screen feed conveyor.

The screen feed conveyor will be fitted with a weightometer which will be used for monitoring the crushing area throughput and for mass accounting. A continuous belt magnet will be used to remove tramp metal.

Oversize ore from the sizing screen top deck (35 mm) and lower deck (20 mm) will be conveyed to the secondary crusher feed bin, via the secondary crusher feed conveyor. The HP200 cone crusher has a maximum capacity of 120 tph at the nominated closed side setting of 13 mm. Ore will be withdrawn from the secondary crusher feed bin (10 t) by a variable speed vibrating feeder, discharging directly to the closed-circuit secondary cone crusher. Cone crusher product will discharge to the screen feed conveyor and return to the sizing screen.

A metal detector and static magnet will be located on the secondary crusher feed conveyor for tramp metal removal. A bypass chute will allow tramp metal to bypass the crusher feed bin, upon a metal detect event.

Undersize from the sizing screen (P_{80} 15 mm) will discharge onto the primary screen discharge conveyor and be conveyed to the mill feed bin, via the mill feed bin feed conveyor.

Any dust generated at the grizzly screen and primary jaw crusher discharge points will be collected by a dust collector. Dust collected from this system will be discharged onto the sacrificial conveyor. Any dust generated at the secondary cone crusher discharge will be collected by a dust collector. Dust collected from this system will be discharged onto the sizing screen feed conveyor.

Any spillage generated within the crushing area will be mechanically recovered using a bobcat and transported to the mill feed conveyor spillage reclaim hopper. Supernatant liquor will be pumped to the spillage thickener.

17.4.3 Ore Storage and Mill Feed

Crushed ore will be conveyed to the 750-tonne live capacity mill feed bin which will provide 24 hours of surge capacity for the grinding circuit (at a plant feed at 0.25 Mtpa).

Crushed ore will be withdrawn from the mill feed bin by duty/standby variable speed apron feeders. The apron feeders will discharge onto the ball mill feed conveyor, which will convey the crushed ore to the ball mill feed chute. The ball mill feed conveyor will be fitted with a weightometer, used for controlling the speed of the apron feeders (mill feed rate) and for mass accounting of feed presented to the grinding circuit.

Any spillage generated within the reclaim area will be mechanically recovered using a bobcat and transported to the mill feed conveyor spillage reclaim hopper. Supernatant liquor will be pumped to the spillage thickener.

A manual belt cut point on the mill feed conveyor will be allowed for, to assist with calibration of the mill feed weightometer and metal accounting (head grade determination).

17.4.4 Ball Milling and Classification

The grinding circuit will operate 24 hours per day, 365 days per year with an availability of 91.3%. This is equivalent to operating about 8,000 hours per annum at a nominal throughput rate of 31.3 tph. The circuit will be designed to process secondary crushed ore with an F_{80} of 15 mm to produce a flotation feed product with a P_{80} of 106 µm.

The grinding circuit will comprise a single 10.5 ft × 15.5 ft effective grinding length (EGL) grate discharge ball mill, with 800 kW of installed power. Under nominal operating conditions, the mill will operate at 72% of critical speed with a 30 wt.% operating ball charge, to draw 626 kW at pinion (674 kW at motor). A grate discharge mill is considered to potentially reduce the overgrinding of brittle minerals such as stibnite.

The ball mill will operate in closed circuit with a cyclone cluster comprised of four 250 mm cyclones. The classification circuit has been selected to achieve the desired product size P_{80} of 106 µm, while handling a 300% circulating load. The circulating load has been selected to accommodate the hard ore being processed in conjunction with the coarse ball mill feed (P_{80} ~15 mm) originating from the secondary crushing circuit.

Ore reclaimed from the mill feed bin will be conveyed to the ball mill via the ball mill feed box. Dilution water and grinding media will also be added via the ball mill feed box. Cyclone underflow will return to the ball mill via the mill feed box. Ball mill discharge will pass through the ball mill trommel screen, with trommel undersize gravitating to the mill discharge hopper. Cyclone overflow constitutes flotation feed.

Oversize scat material will be collected in a scats bunker, from which it will periodically be removed by a front-end loader (FEL). From the ball mill discharge hopper slurry will be pumped by the variable speed cyclone feed pumps to the cyclone feed distributor. Dilution water will be added to the mill discharge hopper for density control.

Sump pumps will be installed in the grinding area to recover spillage. Any spillage generated in the grinding area will be returned to the spillage thickener.

Grinding media (graded charge of 80/40 mm balls) will be added to the ball mill via the ball mill feed.

17.4.5 Bulk Rougher Flotation

A scale-up factor of 2.5 was applied to the laboratory bench-scale flotation data to determine the design flotation circuit residence time. Consequently, the bulk rougher flotation circuit will comprise six mechanically agitated flotation cells, providing a circuit residence time of 18 minutes. Six cells have been selected to provide the requisite circuit volumetric capacity, prevent valuable mineral loss through short-circuiting, and to ensure that the sufficient froth surface area and launder lip length is available to recover the valuable components to the froth phase for subsequent downstream processing.

Given the low concentrate mass yield, the launder lip loading and froth carry rate is well below accepted industry norms. At a mass pull of 6% (mill feed), the resulting launder lip loading is 0.07 t/m/h while the resulting froth carry rate is 0.19 t/m²/h, substantially lower than the accepted maximum carry rate of 1.5 t/m²/h. This low froth to mineral volume ratio will necessitate the use of froth crowders to assist in removing the froth from the flotation circuit, thereby sustaining rougher recovery.

Given the high annual rainfall experienced at Auld Creek, and the fact that the processing plant is an open air design, localised covers will be installed over the flotation cells to mitigate the impact of excessive rainfall on flotation circuit performance.

Overflow from the grinding circuit classification cyclones at a solids content of 30 wt.% solids will constitute the feed to the flotation circuit. Cyclone overflow will report to a trash screen to mitigate the ingress of trash (plastic, fibre, woodchips) into the downstream circuits. The trash screen will be fitted with a 0.63 mm aperture screen. Trash screen oversize will report to a trash bin. Trash screen undersize will gravitate via a multi-stage, multi-fin metallurgical sampler to the flotation conditioning tank. The sample collected by the rougher feed sampler will be pumped to the on-stream analyser (OSA) unit for real-time on-stream analysis. At the OSA, a portion of the samples will also be filtered and submitted for assay on a shift composite basis.

Flotation reagents (PAX collector, lead nitrate $Pb(NO_3)_2$ activator) will be added to the slurry in the rougher conditioner tank. The conditioned slurry will gravitate from the rougher conditioner tank to a bank of six 5 m³ forced-air, rougher flotation cells operating in series. Frother (W24) will be added to the feed box of the first rougher flotation cell as well as the third rougher cell, if required.

Sulphide concentrate from the bulk rougher flotation circuit will be collected in a concentrate launder and gravitate to the flotation concentrate pump box via a static, dual-fin launder sampler, from which it will be pumped to the cleaner flotation conditioning tank. The sample collected by the launder samples will be pumped to the OSA for on-stream analysis. At the OSA, a portion of the samples will also be filtered and submitted for assay on a shift composite basis.

Tails from the bulk rougher flotation circuit will gravitate to the rougher flotation tails pump box, where it will be pumped to the final tails thickener feed box via a multi-stage, multi-fin metallurgical sampler. The sample collected by the rougher tails sampler will be pumped to the OSA unit for real-time on-stream analysis. At the OSA, a portion of the samples will also be filtered and submitted for assay on a shift composite basis.

Low-pressure air will be supplied to each flotation cell by a low-pressure blower system. The airflow to each cell will be controlled by vendor-supplied instrumentation. Similarly, cell levels will be controlled by vendor-supplied instrumentation. Dual internal dart valves on the outlet of each cell will control the pulp level in the cells. The action of the valves will be governed by the output from a ball-float, target-plate and ultrasonic level detector combination. In the event of a power failure, the valves will fail closed to prevent the cells from draining.

The rougher flotation circuit will be contained within a dedicated bunded area, and all spillages will be directed to a centralised sump pump and subsequently pumped to the rougher conditioning tank or spillage thickener.

17.4.6 Cleaner Flotation

The cleaner flotation circuit will comprise four stages of cleaner flotation. A scale-up factor of 2.5 was applied to the laboratory bench-scale flotation data to determine the design cleaner flotation circuit residence time. Consequently, the cleaner flotation circuit comprises:

- Cleaner 1: three 1.5 m³ trough cells providing a residence time of 13 minutes;
- Cleaner 2: three 0.5 m³ trough cells providing a residence time of 8 minutes;
- Cleaner 3: two 0.5 m³ trough cells providing a residence time of 5 minutes; and
- Cleaner 4: two 0.5 m³ trough cells providing a residence time of 5 minutes.

The cleaner flotation cells will be arranged in series, such that tails from the following stage gravitates to the preceding stage while concentrates will be pumped from the preceding stage to the following stage.

Concentrate from the bulk rougher flotation circuit will be pumped to the feed box of the first cleaner cell. Frother (W24 or H27) will be added to the feed box of the first cleaner flotation cell, as required. Likewise, frother will be added to the feed box of each of the following cleaning stages, as required.

Concentrate from the Cleaner 1 flotation circuit will be collected in a concentrate launder and gravitate to the Cleaner 1 concentrate pump (SALA pump or equivalent). This concentrate will be pumped, via in-line pressure sampler, to the feed box of the Cleaner 2 flotation bank. Tails from the Cleaner 1 flotation circuit will gravitate to the cleaner flotation tails hopper, from where it will be pumped to the Au concentrate thickener feed box via a multi-stage, multi-fin metallurgical sampler. The sample collected by the cleaner sampler will be pumped to the OSA unit for real-time on-stream analysis. At the OSA, a portion of the samples will also be filtered and submitted for analysis, on a shift composite basis.

Concentrate from the Cleaner 2 flotation circuit will be collected in a concentrate launder and gravitate to the Cleaner 2 concentrate pump (SALA pump or equivalent). This concentrate will be pumped, via in-line pressure sampler, to the feed box of the Cleaner 3 flotation bank. Tails from the Cleaner 2 flotation circuit will gravitate to the feed box of the Cleaner 1 flotation circuit.

Concentrate from the Cleaner 3 flotation circuit will be collected in a concentrate launder and gravitate to the Cleaner 3 concentrate pump (SALA pump or equivalent). This concentrate will be pumped, via in-line pressure sampler, to the feed box of the Cleaner 4 flotation bank. Tails from the Cleaner 3 flotation circuit will gravitate to the feed box of the Cleaner 2 flotation circuit.

Concentrate from the Cleaner 4 flotation circuit will be collected in a concentrate launder and gravitate to the Cleaner 4 concentrate pump (SALA pump or equivalent). This concentrate will be pumped to the stibnite concentrate thickener feed box, via a multi-stage, multi-fin metallurgical sampler. The sample collected by the Cleaner 4 sampler will be pumped to the OSA unit for real-time on-stream analysis. At the OSA, a portion of the samples will also be filtered and submitted for analysis, on a shift composite basis. Tails from the Cleaner 4 flotation circuit will gravitate to the feed box of the Cleaner 3 flotation circuit.

Low-pressure air will be supplied to each flotation cell by a low-pressure blower system. The airflow to each cell will be controlled by vendor-supplied instrumentation. Similarly, cell levels will be controlled by vendor-supplied instrumentation. Dual internal dart valves on the outlet of each cell will control pulp level in the cells. The action of the valves will be governed by the output from a ball-float, target-plate, and ultrasonic level detector combination. In the event of a power failure, the valves will fail closed to prevent the cells from draining.

The cleaner flotation circuit will be contained within a dedicated bunded area, and all spillages will be directed to a centralised sump pump and subsequently pumped to the rougher conditioning tank or spillage thickener.

All samples collected within the cleaner flotation circuit will be collected in dedicated sample hoppers. From the sample hoppers, the samples will be pumped to the OSA. Sample return from the OSA will gravitate back to the flotation circuit.

17.4.7 Stibnite Concentrate Thickening

In the absence of specific settling test work, benchmark data for a fine concentrate settling duty (0.150 t/m²/h) were used to determine the stibnite concentrate thickener diameter.

At the design solids loading of 3.5 ktpa, that is, a nominal solids rate of 0.4 tph, the required thickener diameter to process the stibnite concentrate arisings in a flocculated, high-rate thickener is 1.9 m. The circuit design accommodates a 4 m diameter thickener, providing a design margin and commonality with the Au concentrate thickener.

Concentrate from the Cleaner 4 flotation circuit will be pumped to the flotation concentrate thickener feed box. Dilute flocculant will be added to the concentrate thickener feed box to enhance the settling properties of the solids. Overflow from the concentrate thickener will gravitate to the process water pond. Stibnite concentrate thickener underflow, at a solids content of 55–60% solids, will be pumped by dedicated, variable speed thickener underflow pumps to the mechanically agitated stibnite concentrate filter feed tank. The 11 m³ live capacity stibnite concentrate filter feed tank will provide 24 hours of storage capacity.

The stibnite concentrate thickening area will be serviced by a single spillage pump, with spillage returned to the concentrate thickener feed box.

17.4.8 Stibnite Concentrate Filtration and Bagging

In the absence of specific filtration test work, benchmark data for fine concentrate filtration duty (100 kg/m²/h) were used to determine the stibnite concentrate filter size. The target moisture content for the stibnite concentrate will be <12% moisture.

At the design solids loading of 0.003 Mtpa, that is, a nominal solids rate of 0.5 tph at a filtration circuit availability of 85%, the required filtration area is 4.4 m². Allowance is made for 20 off 600 mm × 600 mm plates being installed with the filter frame. The selected filter frame has the capacity to accommodate

60 plates, providing significant margin to accommodate lower filtration rates, decreased cake moistures, and unforeseen impacts from slimes and clays, if required.

From the stibnite concentrate filter feed tank, slurry will be pumped to a single, dedicated, stibnite concentrate filter. Filtered product will gravity discharge to the stibnite concentrate storage bunker immediately below the filter while filtrate from the concentrate filter will be returned to the stibnite concentrate thickener feed box.

From the stibnite concentrate storage bunker, stibnite concentrate will be loaded into bags, via a dedicated stibnite concentrate bagging plant. The bagged concentrate (1 t) will be individually weighed and manually sampled for metallurgical accounting purposes.

The stibnite concentrate filtration circuit will be serviced by a single sump pump. Any spillage within this area will be returned to either the concentrate filter feed tank or stibnite concentrate thickener feed box.

Likewise, the stibnite concentrate bagging circuit will be serviced by a single sump pump. Any spillage within this area, originating from the bagging activity and truck tyre wash, will be returned to the stibnite concentrate thickener feed box.

17.4.9 Gold Concentrate Thickening

In the absence of specific settling test work, benchmark data for a fine concentrate settling duty (0.150 t/m²/h) were used to determine the Au concentrate thickener diameter.

At the design solids loading of 9.5 ktpa, that is, a nominal solids rate of 1.2 tph, the required thickener diameter, to process the Au concentrate arisings in a flocculated, high-rate thickener is 3.5 m. The circuit design accommodates a 4 m diameter thickener, providing a 20% design margin.

Tails from the Cleaner 1 flotation circuit will be pumped to the flotation concentrate thickener feed box. Dilute flocculant will be added to the concentrate thickener feed box to enhance the settling properties of the solids. Overflow from the concentrate thickener will gravitate to the process water pond. Gold concentrate thickener underflow, at a solids content of 55–60% solids, will be pumped by dedicated, variable speed thickener underflow pumps to the mechanically agitated Au concentrate filter feed tank. The 40 m³ live capacity Au concentrate filter feed tank provides 24 hours of storage capacity.

The Au concentrate thickening area will be serviced by a single spillage pump, with spillage returned to the concentrate thickener feed box

17.4.10 .Gold Concentrate Filtration and Bagging

In the absence of specific filtration test work, benchmark data for fine concentrate filtration duty (100 kg/m²/h) were used to determine the Au concentrate filter size. The target moisture content for the Au concentrate is <12%.

At the design solids loading of 0.01 Mtpa, that is, a nominal solids rate of 1.5 tph at a filtration circuit availability of 85%, the required filtration area is 14.3 m². Allowance is made for 40 off 600 mm x 600 mm plates being installed with the filter frame. The selected filter frame has the capacity to accommodate 60 plates, providing significant margin to accommodate lower filtration rates, decreased cake moistures, and unforeseen impacts from slimes and clays, if required.

From the Au concentrate filter feed tank, slurry will be pumped to a single, dedicated, stibnite concentrate filter. Filtered product will gravity discharge to the stibnite concentrate storage bunker immediately below the filter while filtrate from the concentrate filter will be returned to the Au concentrate thickener feed box.

From the Au concentrate storage bunker, Au concentrate will be loaded into bags, via a dedicated Au concentrate bagging plant. The bagged concentrate (1 t) will be individually weighed and manually sampled for metallurgical accounting purposes.

The Au concentrate filtration circuit will be serviced by a single sump pump. Any spillage within this area will be returned to either the concentrate filter feed tank or Au concentrate thickener feed box.

Likewise, the Au concentrate bagging circuit will be serviced by a single sump pump. Any spillage within this area, originating from the bagging activity and truck tyre wash, will be returned to the Au concentrate thickener feed box.

17.4.11 Flotation Tails Thickening

In the absence of specific tails settling test work, benchmark data for a tailings settling duty (0.600 $t/m^2/h$) were used to determine the flotation tails thickener diameter.

At the design solids loading of 0.237 Mtpa, that is, a nominal solids rate of 29.6 tph, the required thickener diameter to process the flotation tails in a flocculated, high-rate thickener is 7.9 m. The circuit design accommodates a 10 m diameter thickener, providing a 20% design margin.

Tailings from the bulk rougher flotation circuit final tails pump box will be pumped to the tails thickener feed box by the variable speed flotation tails pumps.

Dilute flocculant will be added to the tails thickener feed box to enhance the settling properties of the solids. Overflow from the tails thickener will gravitate to the process water pond. Tails thickener underflow, at a solids content of 55–60% solids, will be pumped by dedicated, variable speed thickener underflow pumps, to the mechanically agitated tails filter feed tank. The 750 m^3 live capacity tails filter feed tank provides 24 hours of storage capacity.

The tails filtration circuit will be serviced by a single sump pump. Any spillage within this area will be returned to either the tails filter feed tank or tails thickener feed box.

17.4.12 Flotation Tails Filtration and Load-Out

The duty/duty flotation tails filters will be designed to operate 365 days per year with an operating utilisation of 85% (90% availability, 94% utilisation). This equates to 7,446 operating hours per annum, which is equivalent to a nominal capacity of 31.8 tph.

Thickened tailings will be pumped from the tails filter feed tank to the duty/duty tails filters. Filtered tails will be gravity discharged to the bulk storage area immediately below the filters. Filtrate from the tails filter will gravitate to the tails filter filtrate tank, where it will be pumped to tails thickener feed box.

A front end loader will be used to recover the filtered tails from beneath the filtered tails stockpile and load the trucks, which will transport the filtered tails to dry-stacked tailings storage or to the paste plant filtered tails stockpile for paste mixing.

The tails handling area will be bunded and any spillage originating in this area will be hosed to the tails disposal area sump pump, from where it will be returned to the filter feed tank or tails thickener feed box.

17.4.13 Paste Backfill

A paste plant was initially allowed for during the PEA, as it was anticipated that mine waste rock would be used for plant and IWF construction and there would consequently be a shortfall of waste rock for mine backfill. The paste plant was subsequently deleted from scope at the conclusion of the PEA (as a late change) and has therefore been removed from the cost estimates but is still shown in engineering deliverables such as site layouts.

Mine backfill strategy and the need for a paste plant will be further assessed and optimised for the PFS.

17.4.14 Reagents

The major reagents used within the processing plant will include:

- potassium amyl xanthate (PAX) collector for flotation;
- frother (W24/H27) for flotation;
- lead nitrate ($Pb(NO_3)_2$) activator for flotation; and
- flocculant for thickening.

17.4.14.1 Potassium Amyl Xanthate

The primary collector (PAX) mixing circuit will compromise a 'stacked' tank design, with the mixing tank mounted directly above the storage tank. PAX will be delivered to site in 185 kg drums. The drum will be lifted to the PAX feed chute, mounted above the PAX mixing tank, to direct the solids flow into the mixing tank.

Raw water will be added to the mixing tank to achieve a solution with the desired concentration (10 wt.%). The mixing tank will be mechanically agitated to assist with solids dissolution. Once the mixing sequence is complete, the solution within the mixing tank will gravitate to the storage tank compartment, which provides storage buffer for subsequent makeup cycles.

Dedicated variable speed dosing (metering) pumps will deliver the reagent to the required locations within the flotation circuit.

The collector mixing and storage areas will be serviced by a common sump pump. Any spillage generated within this area will be pumped to the tails thickener feed box.

17.4.14.2 Frother (W24/H27)

Frother (W24/H27) will be delivered in 1,000 L intermediate bulk containers (IBCs). The IBC will be connected to a standpipe, with the IBC free draining to the standpipe, providing storage buffer when an empty IBC needs to be replaced.

Dedicated variable speed dosing pumps will extract frother from the standpipe and deliver the reagent to the required locations within the flotation circuit.

17.4.14.3 Lead Nitrate ($Pb(NO_3)_2$)

The lead nitrate mixing area will comprise a stacked tank design. Lead nitrate will be delivered to site in 1,000 kg bulk bags. The bulk bag will be lifted by the lead nitrate area hoist to the bulk bag splitter mounted above the lead nitrate storage hopper. Lead nitrate will be released from the bulk bag by the bag splitter.

Raw water will be added to the mixing tank to achieve a solution with the desired lead nitrate concentration (20 wt.%). The mixing tank will be mechanically agitated to assist with lead nitrate dissolution. Once the mixing sequence is complete, the solution within the mixing tank will gravitate to the storage tank compartment, which provides storage buffer for subsequent makeup cycles.

Dedicated variable speed dosing (metering) pumps will deliver the reagent to the required locations within the flotation circuit.

The lead nitrate mixing and storage areas will be serviced by a common sump pump. Any spillage generated within this area will be pumped to the tails thickener feed box.

17.4.14.4 Flocculant

Flocculant powder will be delivered to site in 25 kg bags and mixed in a proprietary mixing system, comprised of a bulk dry hopper, screw feeder, flocculant blower, mixing tank and storage tank. The flocculant plant will mix flocculant powder with raw water to achieve the required storage concentration (0.25 wt.%).

Flocculant will be withdrawn from the storage hopper by the flocculant screw feeder. The screw feeder will convey flocculant to the flocculant eductor, from which the flocculant powder will be pneumatically conveyed to the flocculant mixer by the flocculant blower. Raw water will be added to the mixer to hydrate the flocculant powder prior to discharging into the agitated flocculant mixing tank. Upon completion of the mixing cycle, the flocculant will be transferred to the flocculant storage tank by the flocculant transfer pump.

From the holding tank, flocculant will be distributed to the Sb concentrate thickener, Au concentrate thickener, and tail thickener (via in-line mixers) by the respective flocculant dosing pumps.

The flocculant area will be serviced by a sump pump. Any spillage generated within this area will be pumped to the tails thickener.

17.4.15 Water Services

17.4.15.1 Process Water

Process water will predominantly consist of Sb concentrate thickener overflow, Au concentrate thickener overflow, and tails thickener overflow plus the filtrate from the concentrate and tails filters. Process water will primarily be used for process stream dilution (density control), process sump level control, and general hose-down.

From the process water pond, process water will be reticulated throughput via a ring-main and dedicated the process water pumps, with off-takes supplied for the predominant user points, namely:

- grinding – ball mill feed, trommel screen sprays, gravity circuit feed, and cyclone feed dilution;
- flotation – launder sprays; and
- thickening – concentrate and tails thickening feed dilution and flocculant dilution.

The total of thickener overflows and filter filtrates will be insufficient to meet the entire process water demand. Additional process makeup water will be sourced from the raw water circuit.

17.4.15.2 Fresh/Raw Water

Raw water for the processing plant will be sourced from the Devils Creek water extraction area and transferred to the raw water storage tank. Raw water from this tank will be reticulated throughput the plant via a ring-main and dedicated raw water pumps, to the predominant user points, namely:

- crusher circuit dust suppression;
- reagent makeup;
- process water makeup;
- on-stream analyser; and
- potable water treatment plant.

17.4.15.3 Gland Service Water

Gland service water for the processing plant will be sourced from the process water circuit to maintain the plant water balance and mitigate the positive process water balance attributed to the filtered tailings disposal. Process water will be pumped through a sand filter, with filtered water reporting to the gland service water storage tank. Sand filter backwash will return to the tails thickener.

From the gland service water storage tanks, a dedicated low-pressure gland water system (pump) will supply gland water to the processing plant gland water duties while a dedicated, high-pressure gland water system (pump) will supply gland water to the tails filter feed pumps.

17.4.15.4 Fire Water

The plant raw water tank will provide a combined raw water and firewater reserve. Firewater will be supplied from the plant raw water storage tank, via a dedicated suction manifold. The firewater system will comprise:

- an electric jockey pump;
- an electrical firewater pump; and,
- a diesel standby firewater pump.

The firewater system pressure will be maintained by the jockey water pump. An electric firewater pump will automatically start on a drop in line pressure. The diesel firewater pump will automatically start if the line pressure continues to drop below the target supply pressure or during a power failure.

17.4.15.5 Potable Water

Raw water for potable water generation and for use within the processing plant will be sourced from the raw water facility.

The raw water will be subjected to water treatment by reverse osmosis (RO) for calcium, magnesium, and chloride removal. Treated water from the RO plant will be stored within a dedicated potable water storage tank. From the potable water storage tank, potable water will be distributed to the processing plant OSA for human consumption and to the safety showers and eye-wash stations.

17.4.16 Air Services

17.4.16.1 Low-Pressure Air

Low-pressure air for the flotation circuit will be distributed to the flotation circuit by low-pressure blowers. Two blowers will be provided, operating in a duty–standby configuration. Low-pressure air will be reticulated via the low-pressure air header, with air pressure regulated by adjusting the speed of the variable speed blowers.

17.4.16.2 High-Pressure Air

Plant air at 700 kPag will be provided by two high-pressure air compressors operating in a duty–standby configuration. The entire high-pressure air supply will be dried. Dried air will be distributed to the required plant areas from the plant air receiver.

Filtration air at 1,000 kPag will be provided by two high-pressure air compressors operating in a duty–standby configuration.

17.5 Benchmarking of Other Antimony Producers

Regarding Sb production, the following parts of the process are specific to stibnite-Au processing and are therefore benchmarked here:

- selective flotation to produce a stibnite concentrate; and
- recovery of Au from Sb ores, both to gravity concentrates and to stibnite concentrates.

Flotation of stibnite is well described in (Anderson, 2012; Segura-Salazar and Brito-Parada, 2021; Özer, 2022). General observations are:

- Optimal stibnite flotation is achieved in the pH range 3–6, while alkaline or oxidising conditions are typically unfavourable. Stibnite can still be floated at neutral or slightly alkaline conditions but requires higher activator and collector doses.
- Stibnite flotation is activated by using lead nitrate or copper sulphate. Lead nitrate is typically considered superior, but one study found that copper sulphate may be advantageous when ores have higher arsenopyrite content.
- Stibnite has some tendency to float naturally, but flotation recovery is improved by using long-chain xanthates. There is some work showing dithiophosphates or dithiocarbamates may improve selective flotation from complex ores with arsenopyrite.
- Frothers that are commonly used include pine oil, polyglycol frothers (such as Dowfroth 250), and methyl isobutyl carbinol (MIBC).
- Sodium cyanide has been successfully used to depress pyrite while floating stibnite but is problematic at acidic conditions due to HCN gas evolution and high cyanide consumption.
- A survey of stibnite flotation performance across the world (summarised in Segura-Salazar and Brito-Parada (2021)) suggests that Sb recovery will typically be 80–95% and concentrate grade will be 50–65% Sb in high-grade deposits where ROM ore is >2% Sb.

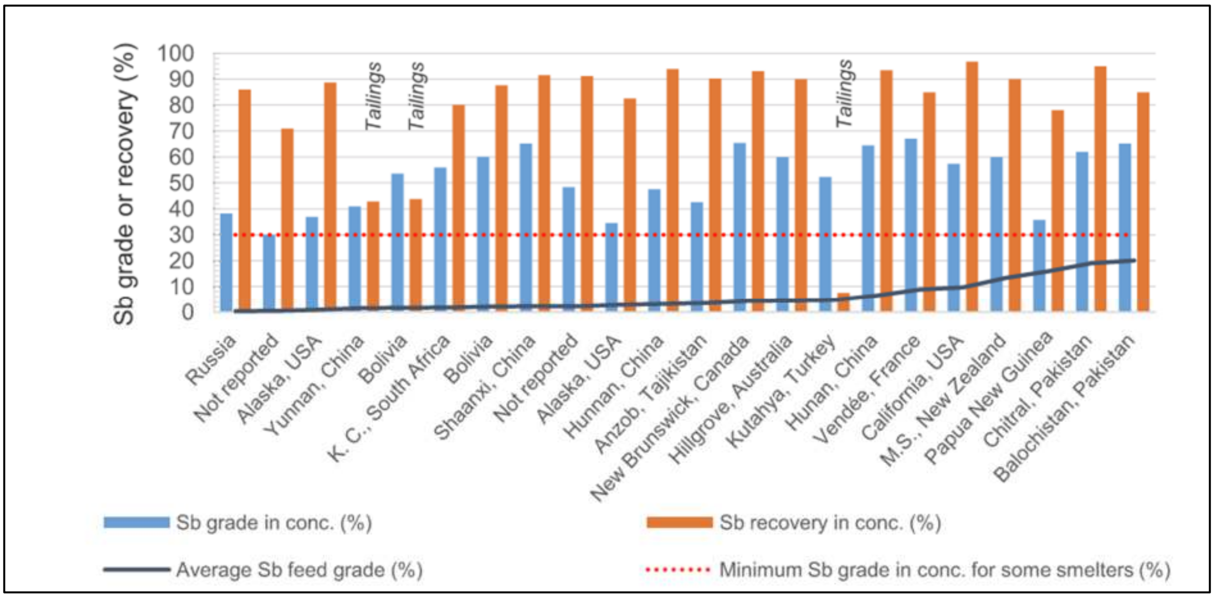


Figure 17-3: Stibnite flotation grade and recovery at variations operations, after Segura-Salazar and Brito-Parada (2021).

World production of antimony in 2023 was ~83,000 metric tonnes, with more than 75% of this production originating from China, Tajikistan, and Russia (January 2024). Lack of diversity in the supply chain is partially why Sb has been identified as a critical mineral, but it also leads to a scarcity of site-specific benchmark information. Other countries such as Bolivia, Turkey, and Vietnam are also important producers, but again there is a lack of published information about operations in these countries. The best available benchmark information is summarised below for sites in Australia, South Africa, and USA.

17.5.1 Hillgrove Mine (near Armidale, New South Wales, Australia)

The orebody (which contains Au, Sb, and W) has been mined since the 1870s and has produced over 50,000 metric tonnes of Sb (Swtzer et al., 2004). Antimony is present as stibnite, and the orebody geology has similarities to other orogenic Au-Sb orebodies such as in New Zealand.

While owned by New England Antimony Mines (1969–2004), the Hillgrove concentrator circuit consisted of crushing, milling, spirals, tables, flotation, and carbon-in-pulp (CIP) to produce three products (Johns et al., 2002):

- A gravity Au concentrate which recovered ~10% of feed Au. Intensive cyanidation was then used to process this through to doré.
- A stibnite concentrate containing >90% of the Sb and 10–20% of the feed Au.
- An arsenopyrite concentrate, which also contained 35–50% of the feed Au (but with Au credits being paid at less than half of the Au value).

In 1999, a pressure oxidation autoclave was commissioned to process arsenopyrite concentrate to recover the 35–50% of associated Au.

Larvotto Resources Ltd recently announced its Definitive Feasibility Study for the restart and upgrade of the Hillgrove Mine (2025). Plant design was supported by extensive test work and mineralogy. The selected flowsheet (illustrated in

Figure 17-4) expands the existing plant (in black) to sequential selective flotation with regrind and Jameson cleaner flotation for Sb and Au. Both flotation circuits use Knelson concentrators and intensive leaching to recover gravity Au.

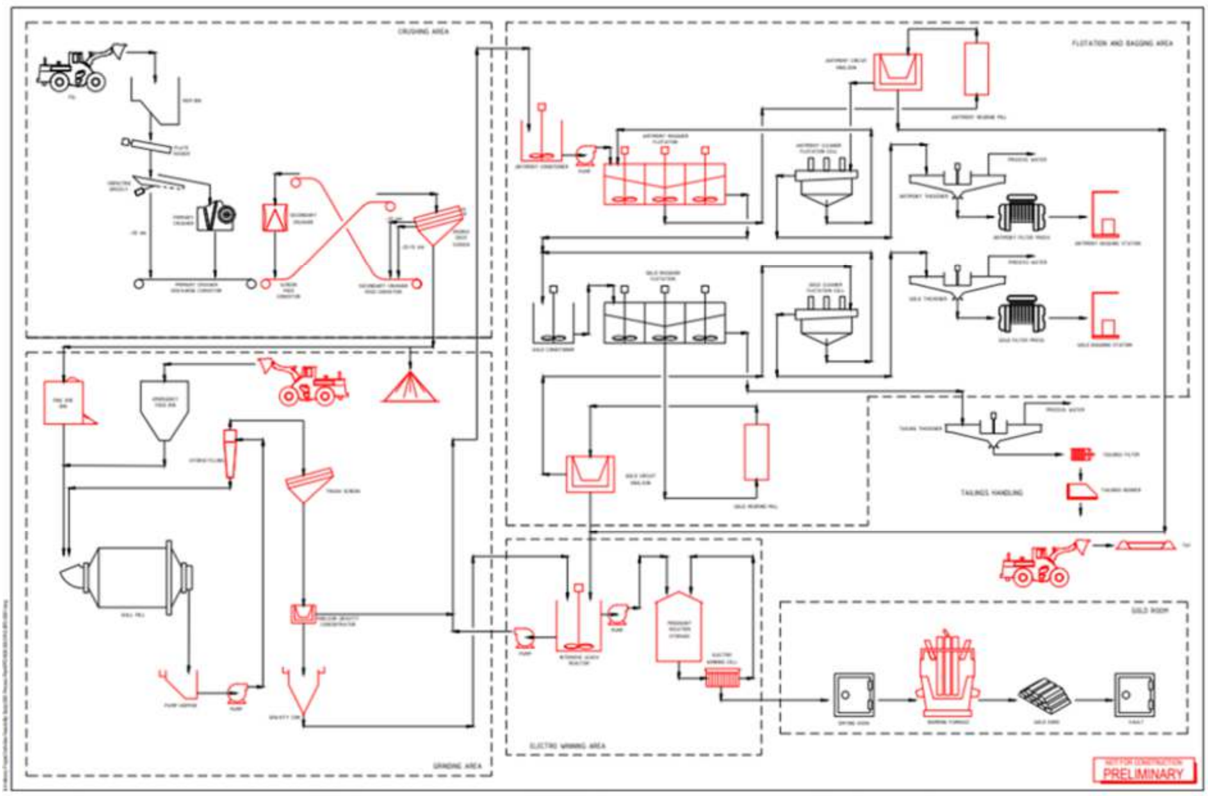


Figure 17-4: Hillgrove processing plant upgrades proposed by Larvotto Resources (2025).

Key outcomes were:

- LOM feed grade of 1.2% Sb and 5.9 g/t AuEq.
- Primary grind size target of 80% passing 180 µm.
- Antimony recovery of ~87% and concentrate grade of ~52.5% Sb.
- Gold recovery of ~84% and concentrate grade of ~46 g/t Au.
- These results were claimed to be better than historical 2015 plant data, which had 85% Sb and 83% Au recovery.
- Tailings is filtered before disposal.

- Existing pressure oxidation (POX) facilities, Sb solvent extraction/electrowinning (SX/EW) circuit and Sb casting plant are made redundant.
- Processing costs of AUD 54/t of ore milled.

17.5.2 Augusta Mine (Costerfield, Victoria, Australia)

The Augusta Mine is located in a region that has been mined for Au and Sb since the 1860s. The Costerfield/Augusta deposit was explored and mined by several operators. It has been mined by Australian Gold Development (AGD) since 1981.

Processing as of 2008 is summarised below (McCarthy et al., 2008):

- 75 ktpa ROM ore processed with ~6% stibnite and 15 g/t Au.
- Crushing and ball milling to 80% passing 106 µm.
- Gravity Au circuit recovers ~30% of feed Au.
- Two-stage cleaner flotation produces a stibnite concentrate containing at least 52% stibnite and 60 g/t Au.
- Concentrate is filtered and bagged for shipment to China where it is refined.

17.5.3 Consolidated Murchison Ltd (Northeastern Transvaal, South Africa)

Consolidated Murchison started production in 1936, and by 1986 was responsible for 18% of world Sb production (Davis et al., 1986). The processing plant evolved and grew over this period. By 1986, the concentrator consisted of two-stage crushing, closed-circuit ball milling and the flotation circuit (Figure 17-5).

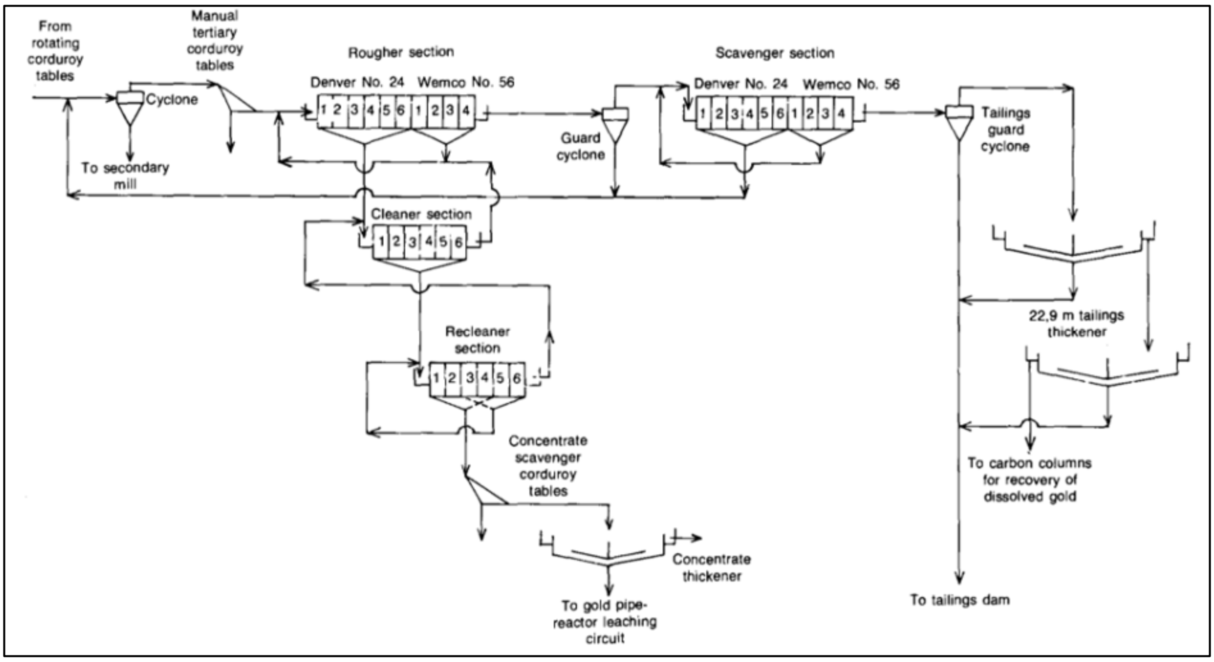


Figure 17-5: Consolidated Murchison Ltd's flotation circuit, after (Davis et al., 1986).

Key aspects of the process were (Davis et al., 1986):

- Stibnite was the primary Sb mineral, but a range of other secondary antimony minerals were present.
- Primary grinding to 80% passing 400 µm.
- Gravity Au recovery from cyclone underflow.
- Antimony recovery at flotation of 80–88% to produce a concentrate with Sb >56% grade. In addition, 30–40% of the Au is recovered to concentrate with a grade of 25–40 g/t. Stibnite was

fast-floating with high recovery. Antimony losses to tails were driven by slow-floating secondary Sb minerals such as berthierite.

- The limit on arsenic in concentrate was 0.25%. Arsenic minerals were depressed by using reduced pH and sodium cyanide (which also depresses stibnite and dissolves some Au).
- Lead nitrate was used as an activator for Sb flotation, and slightly acidic flotation conditions were used to improve recovery of the secondary Sb minerals.
- Antimony concentrate was further processed in a low-alkalinity high-pressure cyanidation lead to recover Au.
- Stibnite concentrate was roasted at 1,100–1,200°C to Sb oxide in rotary kilns.

17.5.4 Sunshine Mine (Couer d'Alene district of Idaho, USA)

The Sunshine Mine has operated since the 1940s and was the largest Sb producer in USA in ~1991. Antimony is present as argentiferous tetrahedrite ($Cu_2S.Sb_2S_3$) and so hydrometallurgical processes were used to recover antimony (Anderson and Krys; Nordwick and Anderson; Anderson et al., 1991). This operation is therefore not a useful benchmark when considering stibnite recovery using flotation.

17.5.5 Other Antimony Projects in Development

Recent focus on critical mineral supply chains has sparked a range of other Sb projects. Projects listed in Table 17-2 are under development but have not yet reported any metallurgical test work results.

Table 17-2: Other Sb projects in development.

Country	Project Name	Company	Minerals of Interest
United States (Montana)	Thompson Falls Antimony Project	Red Mountain Mining (ASX: RMX)	Sb, Au
United States (Montana)	Stibnite Hill	United States Antimony Corporation (NYSE: UAMY)	Sb
United States (Alaska)	Estelle / Stibium Antimony-Gold Project	Nova Minerals (ASX: NVA)	Sb, Au
Slovakia	Trojarova Project	Military Metals (CSE: MILI)	Sb, Au
United States (Nevada)	Last Chance Antimony Project	Military Metals (CSE: MILI)	Sb
Australia (WA)	Mt Clement Antimony & Gold Project	Black Cat Syndicate (ASX: BC8)	Sb, Au, Ag, Pb
Australia (NSW)	Halls Peak Project	Black Cat Syndicate (ASX: BC8)	Sb
United States (Idaho)	Stibnite Gold-Antimony Project	Perpetua Resources (TSX: PPTA)	Sb, Au
Canada (British Columbia)	New Polaris Gold-Antimony Project	Canagold Resources (TSX: CCM)	Sb, Au
Canada (Newfoundland)	Beaver Brook	New Age Metals (TSX.V: NAM)	Sb, Au
United States (Alaska)	Treasure Creek Antimony Project	Felix Gold (ASX: FXG)	Sb, Au

18 Project Infrastructure

18.1 Overall Site

The Project site layout has been developed for the PEA and is presented in Figure 18-1 and Figure 18-2. These layouts incorporate input from Mining One for mining scope and input from KPPL for the IWF. Layout for water treatment facilities is nominal and is based on a recent project with similar scope.

The preliminary layout has been developed with consideration to:

- Minimisation of ore and waste haulage.
- Separation of mine and haul routes from other site roads and foot traffic.
- Separation of regular site deliveries from the processing plant and mine areas (where practical).
- Location of the plant on relatively flat ground to minimise cut-and-fill quantities.
- Potential need to store waste rock from mining for periods before this waste rock is then used in construction (such as embankment raises for the IWF).
- RUA selection of processing plant and IWF locations to reduce visual, noise, and dust effects on local communities.
- Future expansion or duplication of the processing plant to increase throughput as further reserves are identified.

The QP (Marius Phillips) recommends the following work for the PFS to further develop the site layout:

- Geotechnical investigations should be conducted in the processing plant area to establish founding conditions, foundation designs, and costs.
- Potential for flooding of site areas by Devils Creek needs to be investigated and may affect decisions about the location and design of flood mitigation measures.
- The location of the main mine portal can potentially be improved to reduce haul distances.
- Haul road alignment from the upper mine portal should be reviewed by a road design consultant.
- The need for waste rock in construction should be quantified, and schedule for waste rock production and use should be mapped out. This will help clarify the capacity needed for waste rock stockpiles and may also drive decisions about plant location to optimise cut-and-fill requirements.
- Site runoff and water management design should be developed further to define drainage features and water treatment systems.

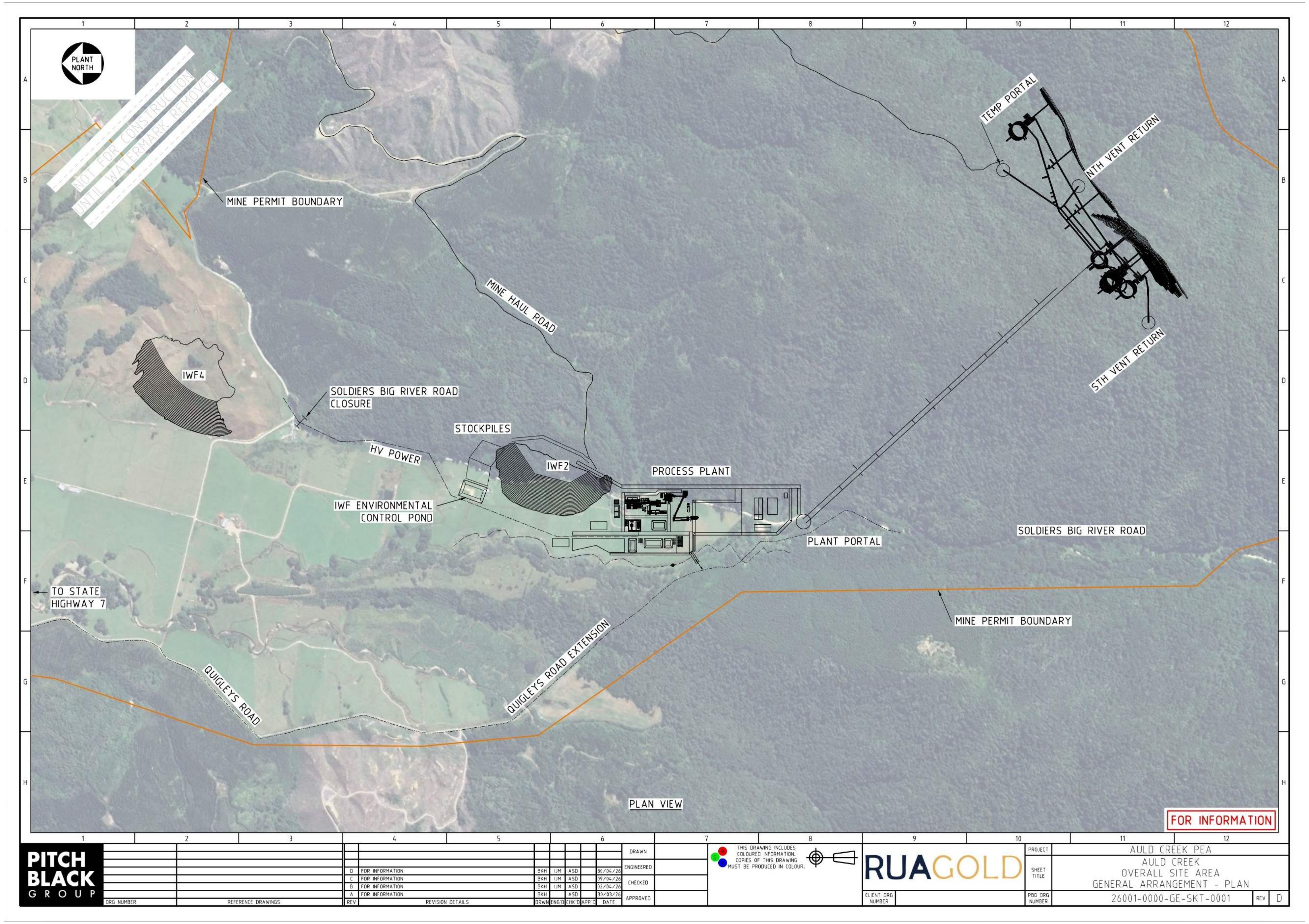


Figure 18-1: Overall layout of the Project site.

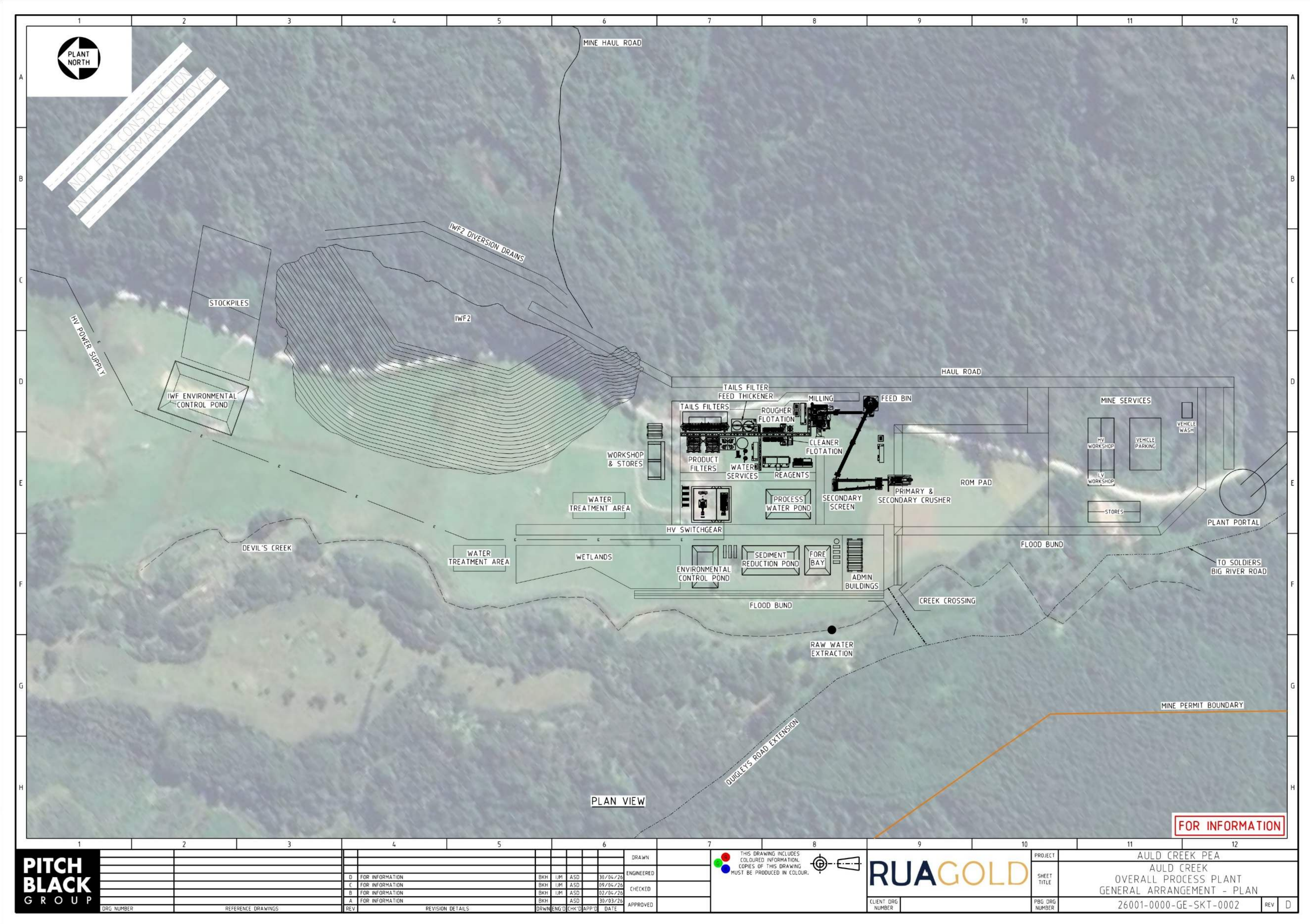


Figure 18-2: Layout of the processing plant and nearby surface infrastructure.

18.2 Roads

18.2.1 Access to Site and Diverting of Public Traffic

Public roads around the Project are illustrated in Figure 18-3. Mark-ups relate to project scope for public roads which are described below this figure.

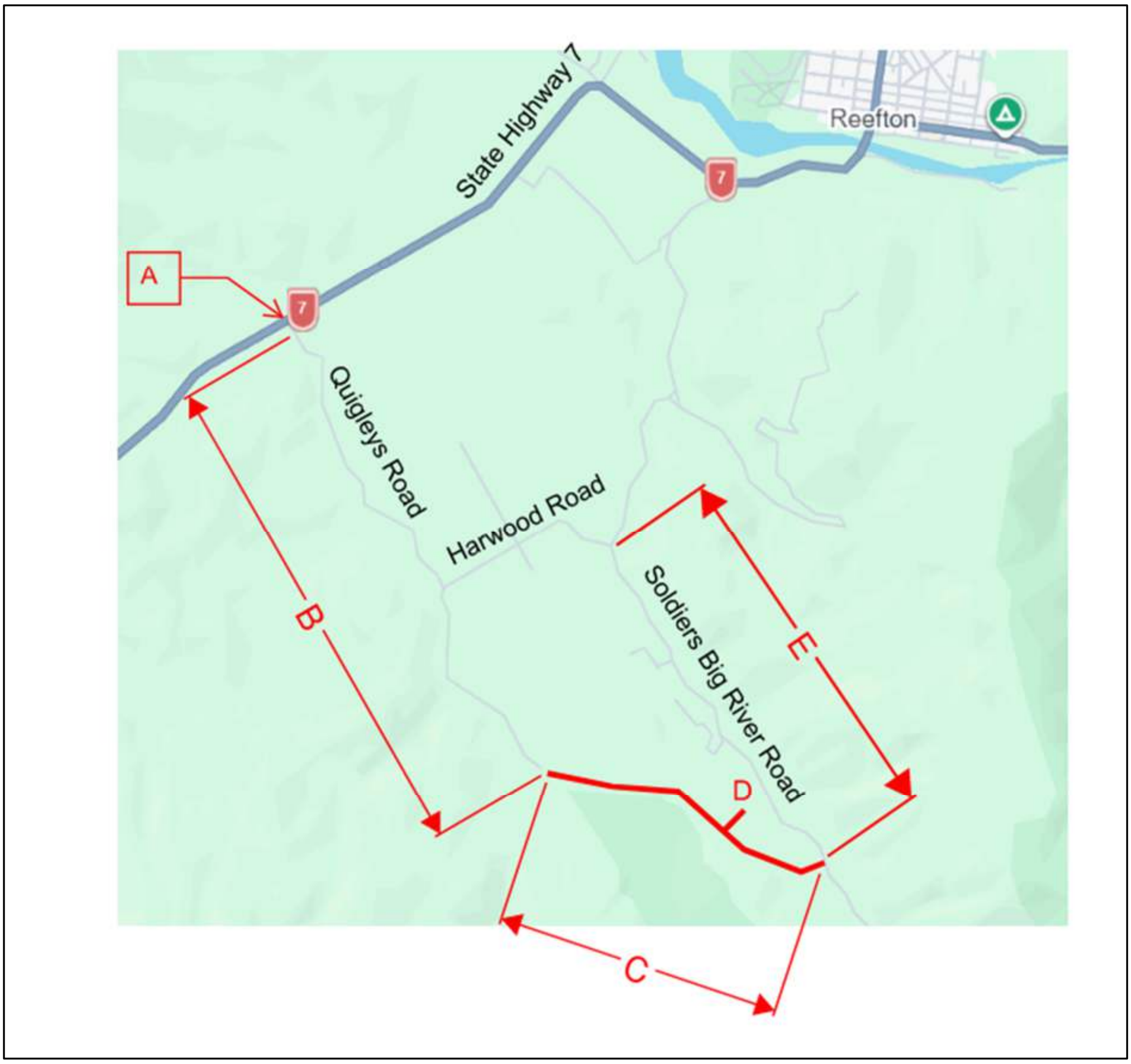


Figure 18-3: Public roads near the Auld Creek Gold-Antimony Project.

The main mine portal and upper mine portal will lie to the east of Soldiers Big River Road, while the processing plant and integrated waste storage facilities will lie to the west. The section of Soldiers Big River Road that runs through the Project will be unsafe for public road traffic, and so it is proposed to close this section and to bypass public traffic around the Project via Stage Highway 7 and Quigleys Road. The main site access would also be provided from this bypass.

Key scope is listed below, and location is indicated in Figure 18-3:

A. Seal and widen the intersection of State Highway 7 and Quigleys Road:
   - Upgrade of roadways using PPM Diagram E Access Treatment guide
   - Underground/relocation of power supply lines or protection
   - Public signage and safety upgrades.

B. Widen the existing 2.5 km length of Quigleys Road.

C. Extend Quigleys Road southeast by adding 1.3 km of new roadway to intersect with Soldiers Big River Road (and thereby create a public bypass around the Project).
D. Create the main access road (200 m length) to the Project by branching off the Quigleys Road extension and crossing Devils Creek.
E. Close a section of Soldiers Big River Road to public access (with public using the new alternative route via Quigleys Road).

18.2.2 Project Site Roads

Plant internal roads from public road access will provide access between the administration area, processing plant facilities, mine services area, and accommodation camp. These roads will typically be 6 m wide and will be constructed flush with bulk earthworks pads to ensure that storm water sheet flow is achieved across the Project, thereby avoiding the need for deep surface drains and culvert crossings within the plant area.

18.2.3 Mine Haul Roads

Mine activity vehicles will travel via internal heavy vehicle haul roads. These roads will be segregated from other access roads for light vehicles. These roadways will be constructed to access mine infrastructure such as the portal, ROM pad, IWF, and mine services areas.

18.2.4 Access Tracks

Access tracks will provide light vehicle access to features such as raw water supply at the Devils Creek water extraction station, fire breaks, and animal control fencing. The access tracks will be cleared and graded natural earth tracks. Routes will be determined during later phases of the Project to best fit local terrain and vegetation density.

18.3 Power

18.3.1 Power Supply

Power will be supplied from the Reefton high-voltage (HV) power supply network. The power supply will supply at HV which will be stepped down to 11 kV at a central switchyard with substations. 11 kV power will be distributed through the processing plant area via buried cables and/or rack-run cables, and will also be distributed to infrastructure outside the processing plant via overhead pole lines. The power supply has been sized at 10.0 MW connected load to accommodate a peak load of 8.0 MW and an average running load of 5.5 MW.

The Project power loads are summarised in Table 18-1. Mining One has provided preliminary electrical loads for the mine and mine services.

The plant and mine services emergency loads will be backed up by diesel generators.

The ball mill at the processing plant will be the largest motor load. The ball mill has been specified with a variable speed drive to reduce the load surge during start-up and to minimise the impact to the HV power supply network.

Table 18-1: Estimated site electrical loads (including mining scope).

Area	Connected Load (kW)	Operating Load (kW)
Processing Plant		
Crushing and Ore Storage	395	270
Ball Milling and Classification	1,025	770
Bulk Antimony-Gold Rougher Flotation	140	85
Cleaner Flotation	80	40
Antimony Concentrate Thickening & Filtration	300	140
Gold Concentrate Thickening & Filtration	300	140
Tailings Thickening & Filtration	1,360	690
Reagents	85	40
Water Services	510	180
Air Services	270	110
Spillage Handling	15	10
Miscellaneous (lighting, air conditioning, mill motor cooling, office and workshop, losses)	1,300	800
Sub-Total	**5,780**	**3,275**
Paste Plant	800	400
Infrastructure	350	350
Supply to Mine	3,000	1,500
TOTAL	**9,930**	**5,525**

18.3.2 Electrical Distribution

The electrical system will be based on a 11.0 kV distribution and 400 V, 50 Hz working voltage. The 11.0 kV feeder from the power supply HV switchgear will feed the site distribution 11.0 kV switchboard. For the processing plant the 11.0 kV supply will be stepped down from 11.0 kV to 400 V at each switchroom using separate 11.0 kV/400 V distribution transformers fed from the HV distribution board.

The power supply to the mine portals will have 11.0 kV/1,000 V distribution transformers.

The following switchrooms will be provided in the plant:

- crushing area;
- mill area;
- flotation, reagents and water/air services area;
- paste plant area;
- mine services area; and
- mine portal areas.

Processing plant switchrooms will house 400 V motor control centres (MCCs), area variable voltage variable frequency (VVVF) drives, plant control system cabinets, plant lighting transformers, various distribution boards and uninterruptible power supply (UPS) power distribution.

11.0 kV underground and overhead powerlines will provide power to various remote facilities (IWF pumps, raw water pump, etc.). Pole-mounted transformers will step down the voltage at each location and supply an outdoor 400 V switchboard local to each equipment area.

Larger power demands of 11.0 kV power will be provided to the mine services area and the mine portal for further transformers and substation distributed.

18.3.3 Electrical Buildings

Electrical buildings will be prefabricated modular buildings fitted out with all electrical equipment off-site to minimise on-site labour. Buildings will be elevated ~2 m above ground level (on a structural framework) to allow for bottom entry of cables into electrical cabinets. The electrical buildings will be installed with air-conditioners and will be suitably sealed to prevent ingress of dust.

18.3.4 Transformers and Compounds

All 11.0 kV/400 V distribution transformers will be of ONAN cooling configuration and vector group Dyn11. Fire-rated concrete walls will be constructed around the pad-mounted transformers.

18.4 Fuel Supply

Bulk fuel supply will be provided by an on-site fuel storage facility and will store diesel for the mine vehicles, light vehicles, and other users at the processing plant. Day storage tanks will be provided at the mine services area, which will be localised for all users. Bulk fuel will be supplied from local fuel suppliers as required.

The fuel storage facility will include 14 days storage of diesel. Diesel fuel dispensing bowsers will provide secure and monitored fuel to mine vehicles and light vehicles across all departments.

18.5 Sewage and Solid Waste Management

18.5.1 Sewage Treatment

Effluent from all water fixtures in the processing plant, mine services area, and administration areas will be pumped to a common sewage treatment plant vendor package located near the processing plant. Treated effluent will be discharged to the environment control pond. Treatment plant sludge will be suitable for municipal contractor disposal.

18.5.2 Solid Wastes

Wastes will be sorted and re-used or recycled as much as possible. Waste lubricating oils and general non-hazardous solid wastes will be collected at a general waste facility for removal by local contractors. Hazardous waste will be collected and stored separately before being transferred to a suitable permitted facility, either on-site or off-site, depending on the specific materials and requirements.

18.6 General and Processing Plant Buildings

Site buildings will be fit for purpose industrial-type structures. Workshops, offices, amenity buildings warehouses, and reagent storage sheds will be constructed using a concrete slab on the ground, with structural steel frame and metal cladding. Smaller buildings may be modular. Buildings will be equipped with ventilation and heating as needed to suit the climate and to meet occupational health & safety regulations.

The following buildings and facilities will be provided at the processing plant area:

- laboratory (dry and wet);
- main warehouse and office;
- outdoor storage area for consumables;
- administration/mining services building, including first aid room;
- reagents storage;
- short-term concentrate storage sheds;
- processing plant control room;
- plant workshop and maintenance yard;
- plant workshop yard;

- crib room;
- male and female ablutions; and
- plant security gatehouse.

18.7 Processing Plant Geotechnical

A geotechnical investigation will be conducted to assess the founding conditions at the proposed processing plant location. This investigation will include drilling of rotary cored boreholes, test pit excavation, in situ testing, and laboratory testing.

A more detailed site layout will be developed at the PFS stage. This layout will guide the further geotechnical investigation by identifying the locations to be tested and the type of testing needed.

18.8 Water

18.8.1 Water Balance

Water supply and treatment requirements for the Project have been assessed using the conceptual water balance. This balance includes the processing plant, concentrate export, mining areas, ROM pad, IWF, environmental control dam (ECD), and water treatment system. The water balance block diagram is presented in Figure 18-4.

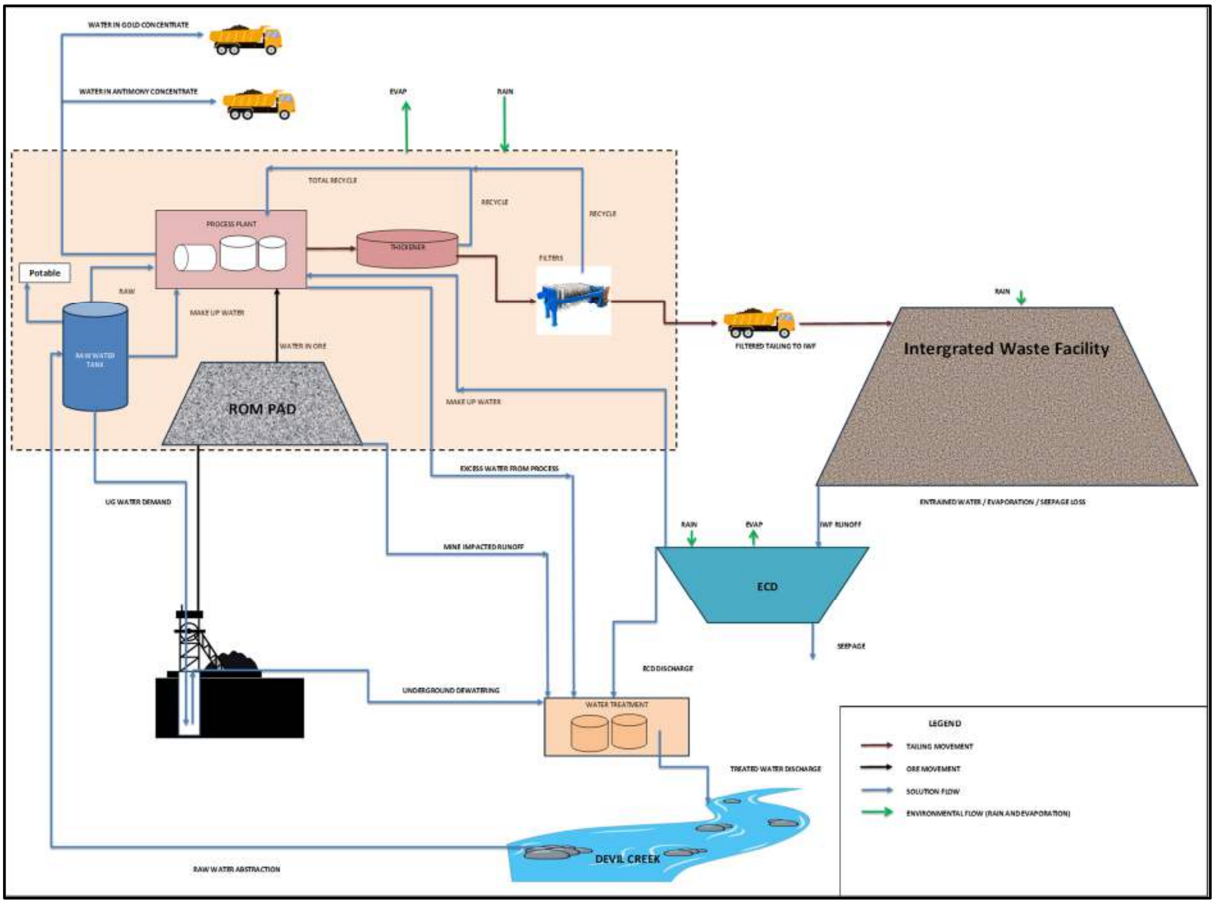


Figure 18-4: Water balance model schematic.

The assessment provides an overview of site water behaviour under average climatic conditions and confirms that sufficient water is available to support processing, while excess contact water can be safely managed through controlled treatment and discharge. The following subsections summarise the water balance methodology, water demands, water excess, water supply sources, and the treatment and disposal strategy.

A conceptual water balance was developed to support the preliminary sizing of the environmental control dam (ECD), assess site raw demand (river water), and assess the required capacity of the water treatment facility to manage contact water. The model incorporates long-term monthly rainfall and evaporation data from the National Institute of Water and Atmospheric Research (NIWA) Reefton Electronic Weather Station, operational water inputs, and catchment runoff. Runoff was calculated using coefficients of 0.95 for lined areas and 0.70 for the exposed tailings surface, with contributing areas updated as tailings deposition progresses. Evaporation losses were applied using lake-surface evaporation values derived from Penman data and adjusted using the Stanhill correction factor. The ECD stage-storage curve was used to determine monthly pond area and available storage, allowing the model to calculate net water accumulation after accounting for inflows, evaporation, and the controlled 4 L/s treatment discharge. This monthly timestep approach demonstrates that the ECD maintains adequate freeboard throughout operations and can accommodate extreme rainfall events within the available flood storage.

Water supply for the Project will be provided through a combination of recycled process water and raw water sourced from Devils Creek. Recycled water contributes ~70–74% of flotation demand, with the remaining balance supplied as raw water. The average water abstraction rate from Devils Creek estimated for the PEA is ~18.5 L/s.

Water demands for the Project are driven by filtered tailings production, concentrate export, and site services. Key demands include water entrained in filtered tailings (disposed at 85% solids), potable water, underground water demands, and raw water required to supplement flotation demand. These inputs were applied on a monthly basis in alignment with the production schedule to reflect changes in plant throughput and cumulative tailings placement.

The processing plant will have a low demand for raw water makeup: all concentrates and tailings are filtered, and the filtrate is recycled to the grinding circuit. Rainfall and hose-down in process areas will also be collected, treated in a dedicated spillage handling facility, and metered back into the process. Other site runoff waters are presently of unknown quality, and so the assumption is that these will be treated and discharged rather than being used in the process. The potential to use other runoff will be investigated in future phases.

The QP (Gary Davison) has allowed for an average of 25 L/s of mine dewatering to the environmental control pond at the processing plant. Further hydrogeology study is required at future phases to confirm the dewatering rate.

Excess water is generated primarily from high regional rainfall, runoff from lined and tailings surfaces in the IWF, and underground dewatering, which exceed evaporation, concentrate export, and entrainment losses. The conceptual water balance shows that the ECD will maintain adequate freeboard under average climatic conditions, with pond level controlled by discharge to the water treatment plant. Sensitivity testing confirms that the ECD can also accommodate the 1-in-1,000 year, 72-hour storm event (379 mm) within the available flood storage. Runoff from the mining and process area will report to the water treatment plant, along with underground dewatering flow.

All contact water generated within the IWF, mining areas and processing areas will be directed to the water treatment plant for disposal. The average water treatment demand estimated for the PEA is in the order of 35 L/s. It is recommended that when a better understanding of the quality of the key water excesses (IWF runoff and underground dewatering) is gained, an assessment of the viability of re-using some or all of this water as process makeup water or underground drilling and dust suppression water be undertaken to reduce the water treatment rates. Release from the water treatment plant will be monitored for quality and quantity to ensure compliance with licence conditions. Scope of water treatment will be developed during the PFS.

18.8.2 Water Supply and Distribution

Raw water demands at site will be met from the creek extraction station which is located in Devils Creek adjacent to the processing plant. The creek extraction station will include submerged intake strainer, submerged concrete pump well complete with internal access ladder and davit for pump maintenance, and submersible pump including flexible cables/piping.

A common potable water system will be provided for the processing plant and mine services usage. It will be located at the processing plant. A vendor package modular potable water treatment plant including filtration, ultraviolet sterilisation, and chlorination will be installed. Raw water will be delivered via a reticulation system using a constant pressure variable flow pump system. The pump skid will include an ultraviolet disinfection unit to provide additional security against biological contamination.

18.8.3 Management of Excess Water and Sediment

The Project site experiences high annual rainfall, and rainfall on adjacent mountains will be channelled through the site. These water flows cannot be stored indefinitely on site or recycled, and so will need to be released to the local waterway. Three types of water have been identified, based on the level of treatment they need before release:

- Clean (uncontaminated) water that is diverted around facilities so that it can be discharged without treatment.
- Water carrying sediments, which will be managed using sediment control structures before release.
- Contaminated water (carrying dissolved species, oils, sediments, etc.) that needs to be treated to acceptable levels before release. This would include mine dewatering streams. It is likely that excess process water and runoff from the IWFs would contain dissolved As and Sb, and would therefore need treatment.

Water and sediment management system scope and design is preliminary, and further work is required at future phases. Work is ongoing to quantify the local geochemistry and the leaching potential of the tailings and waste rock.

The ECDs will be constructed as a single zone, low-permeability embankment with the upstream batter and basin lined with a 1 mm-thick density polyethylene (HDPE) liner to minimise seepage losses from the facility and reduce the potential seepage and embankment integrity. The ponds will be equipped with a floating submersible pump and pipelines to pump water to the water treatment plant.

Allowance has been made for an active water treatment system. This will draw water from various environmental ponds, treat this water and then discharge to an artificial wetland that drains to Devils Creek.

18.9 Mining Facilities

The QP (Gary Davison) considers that the following major infrastructure will be required on the surface:

- tailing storage facility (~1 Mt);
- processing plant;
- buildings and offices;
- workshop (HV and LV);
- waste dump/ROM pad;
- new roads; and
- portal infrastructure.

Surface infrastructure preliminary sizing estimates are tabulated in Table 18-2.

Table 18-2: Surface infrastructure sizing estimates.

Surface Infrastructure	Size	BOE
LV Workshop	15 m × 20 m	3 m × 5 m LV bays 20 m deep
HV Workshop	25 m × 50 m	3 m × 10 m HV bays (25 m deep) with some space in between for walkways and offices
Hardstand	30 m × 50 m	For HV parking
Washdown Bay	15 m × 10 m	About the size of a truck with room either size
Stores	20 m × 50 m	Assumed based on experience
Minesite Buildings	100 m × 200 m	Based on average size mines
Processing Plant	120 m × 300 m	Mining One preliminary estimate.
LV Workshop	15 m × 20 m	3 m × 5 m LV bays 20 m deep
Surface Stockpiles		
ROM	140 m × 90 m	ROM based on 2 weeks of storage at 250 ktpa production (4,500 m^3 = 14-day pile) and then split into three separate stockpiles (HG, MG, LG) and allowing room for a loader to work.
Plant Waste Dump	75 m × 75 m	Rill angle of 37°. Based on physicals from Deswik schedule and 12 m-high stable dump with allowance for some working berms.
Temporary Waste Dump	40 m × 40 m	Rill angle of 37°. Based on physicals from Deswik schedule and 12 m-high stable dump with allowance for some working berms.

18.9.1 Surface Layout

A preliminary layout for mining facilities is suggested as per Figure 18-5. Note that positions will not be finalised until RUA's PFS partners have completed engineering work.

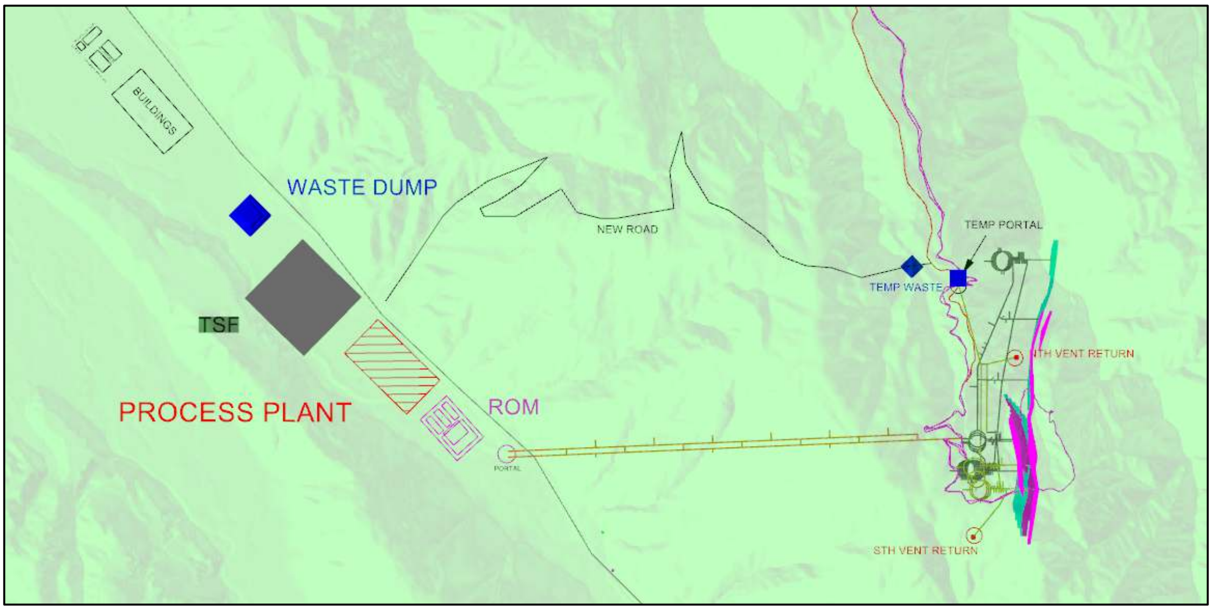


Figure 18-5: Surface infrastructure layout.

18.9.1.1 Portal Layout

Note that at each portal (Temporary and Plant), ~40 m × 40 m of clearance will be required to establish critical services. The QP (Gary Davison) suggests the preliminary layout illustrated in Figure 18-6.

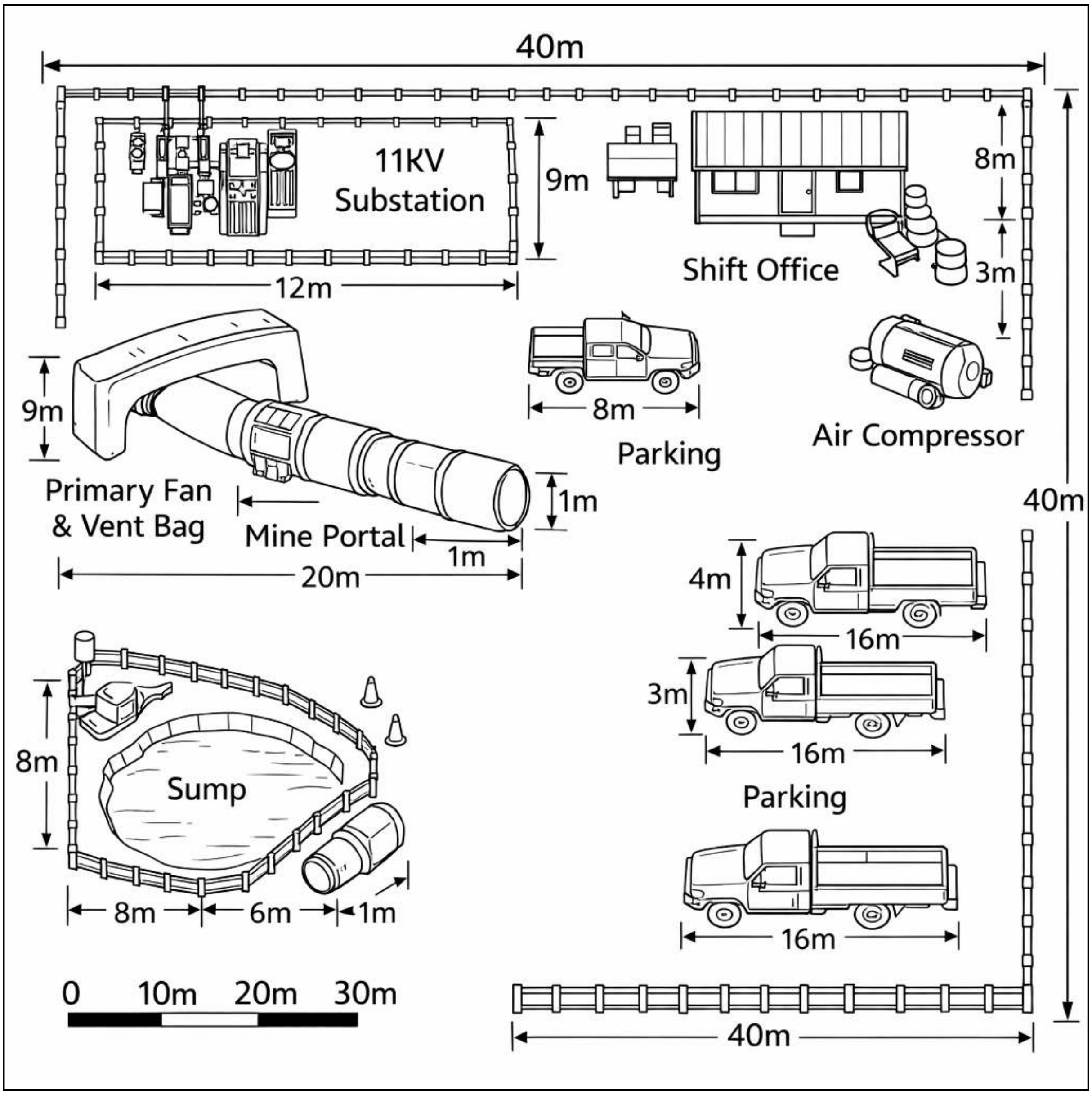


Figure 18-6: Portal infrastructure and layout.

18.9.2 Services (Water and Power)

18.9.2.1 Water

Compressed air, water, and power must be supplied for the underground fleet at both the temporary portal and the plant portal. The QP (Gary Davison) estimated mine demand for water supply and dewatering (

Table 18-3) based on an existing operating narrow-vein gold mine in Victoria, Australia, and allowing for some contingency. Dewatering will only be fully understood once hydrogeological data for the construction of a hydrogeological model (PFS level) are available.

Table 18-3: Mine services likely demand.

Key Assumption	Amount	UoM
Water Demand for UG fleet	15	L/s
Dewatering Demand	25	L/s

18.9.2.2 Power

Power was estimated based on underground fleet and rated motor power. A summary is captured in Table 18-4. Power demand for the underground fleet was derived from original equipment manufacturer (OEM) specifications. Fan selection was based on the Mining One ventilation design and fan specifications, assuming shaft power converted directly to electrical supply. Pumps and compressors were estimated based on the deposit depth and available pumps (Challenge Mono pumps and compressors that are typically used in mines). Note that, at this concept level, The QP (Gary Davison) determined that power demand by mining will be ~2.2 MW.

Table 18-4: Power demand estimation.

Equipment	Peak Quantity	kW	Utilisation	Total Demand (kW)
MUKI FF Jumbo	9	55	0.5	248
Production Drills	0	120	0.0	0
Twin-Boom Jumbo	2	124	0.6	150
Subtotal Cabled Mining Fleet				**400**
Primary Fans	4	90	1.0	360
Secondary Fans	10	45	0.95	428
Compressors	2	250	1.0	500
Mono Pumps	2	110	0.8	176
Flygt (Submersible) Pumps	10	2	1.0	20
Workshop	1	300	1.0	300
TOTAL				**2,180**

18.9.3 New Roads

The QP (Gary Davison) suggests the new (temporary) haul road be constructed in the mountainous region (Figure 18-7). The new road will require permitting for land clearance. The road will also require crushed waste rock (road base) for construction. The estimated quantity is given in Table 18-5.

The QP (Gary Davison) has allocated a preliminary cost for all roads in the economic model, but suggests the designs be refined at the PFS level.

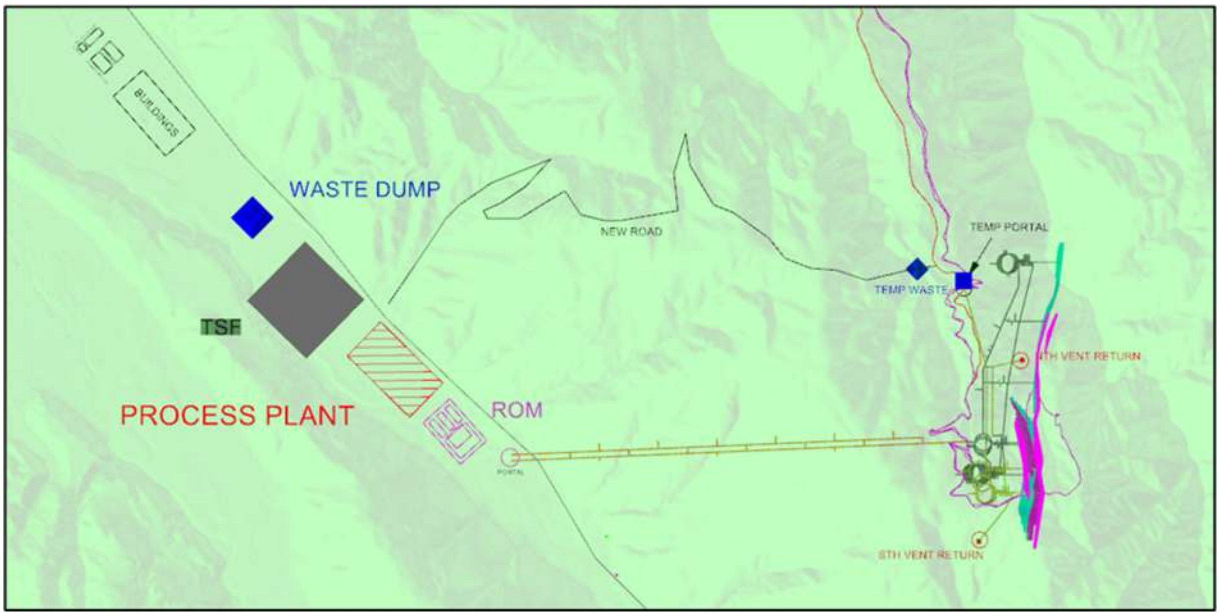


Figure 18-7: New (temporary) haul road.

Table 18-5: Waste rock requirements for new haul road.

Key assumption	Amount	UoM
Road length	4,000	m
Waste rock (road base) depth	200	mm
Road width	3.0	m
Required waste rock	2,400	m^3

18.9.4 Surface Disturbance on DOC Land

RUA has indicated that surface disturbance requirements essential for mining operations on DOC land must be clearly outlined as part of this PEA. These requirements should also be reported in "tennis courts" of land clearance required. For reporting purposes, a standard tennis court is taken as 23.77 m (length) × 10.97 m (width), or ~261 m^2.

The mine design requires five areas of DOC land disturbance:

- temporary portal access (will remain for LOM);
- temporary waste dump (can be removed once plant decline connects to main decline);
- southern ventilation exhaust (will remain for LOM);
- northern ventilation exhaust (will remain for LOM); and
- land clearance for crown pillar mining (to be mined and rehabilitated within 2 years of project commencement).

The areas of disturbance are presented in Figure 18-8 and their area of surface disruption is summarised in

Table 18-6. Space requirements for the portal configuration were displayed previously based on infrastructure requirements, ventilation adit land requirements were assumed as 10 m × 10 m to provide room for some barricading (fans would be mounted underground), and the crown pillar disruption was governed by the economical shapes.

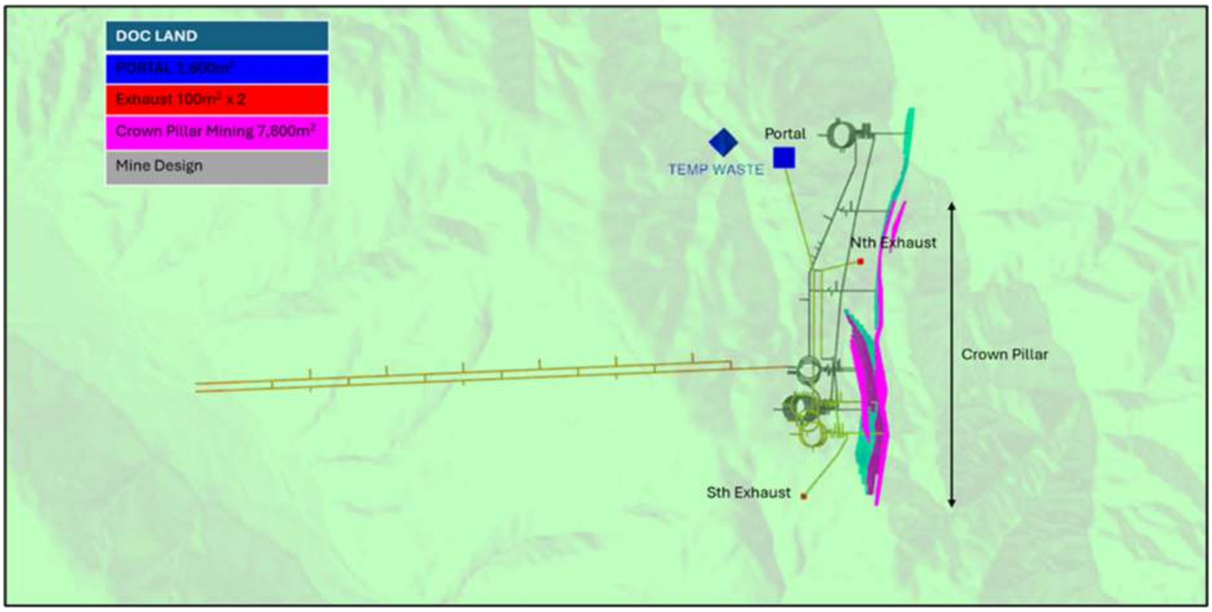


Figure 18-8: Land disturbance on DOC land.

Table 18-6: Land disturbance reported in tennis courts.

Land disturbance area name	Area (m^2)	Area in number of tennis courts
Temporary portal	1,600	7
Northern exhaust	100	1
Southern exhaust	100	1
Crown pillar mining	7,800	30
Temporary waste dump	1,600	7

18.10 Integrated Waste Facility

The Project will manage filtered tailings and limited volumes of surplus waste rock within an IWF. The IWF concept has been developed to address the site's high rainfall, steep terrain, and high seismicity, while providing a robust and practical solution for filtered tailings disposal. The facility comprises a filtered tailings stack, together with surface water management systems and an ECD designed to safely collect and store contact water.

18.10.1 Disposal Technology

Filtered tailings has been selected as the preferred disposal technology based on its reduced geotechnical risk in a high-seismicity environment, smaller footprint relative to conventional impoundments, lower runout potential in the unlikely event of a breach, and improved water management due to the minimal presence of free water. Tailings will be dewatered using filters to a target of 85% solids. The filtered tailings will be transported to the IWF by trucks and placed in controlled lifts. Compaction will be undertaken where practicable; however, given the region's high rainfall, maintaining rigorous compaction control at all times cannot be guaranteed. For this reason, the IWF incorporates a defined structural zone to ensure long-term stability.

18.10.2 Site Selection Study

The multi-account analysis, completed in accordance with Environment and Climate Change Canada's "Guidelines for the Assessment of Alternatives for Mine Waste Disposal" (2013) identified IWF-2 as the preferred location for the IWF at the PEA level. This option achieved the highest overall merit score due to its favourable balance of technical, financial, environmental, and social performance. Key advantages include short haulage distance, moderate embankment length, and a compact liner footprint. Although the foundation comprises alluvial soils, this constraint is shared with most upstream sites and is outweighed by IWF-2's strong operational efficiency and lower overall cost profile. Figure 18-9 illustrates the five siting alternatives considered in the assessment.

IWF-4 is considered the second-ranked alternative, offering a strong combination of short embankment length, good expansion potential, and reduced interaction with existing infrastructure. IWF-5 ranks third, supported by favourable foundation conditions and minimal infrastructure conflicts; however, its longer haulage distance and its closer proximity to the Reefton community reduce its overall suitability. IWF-1 and IWF-3 performed significantly lower due to greater construction complexity, limited expansion potential, and higher social and environmental impacts. A site visit on 12 March 2026 confirmed the desktop findings. Final site selection will be confirmed following geotechnical investigations, hydrological studies, and stakeholder engagement.

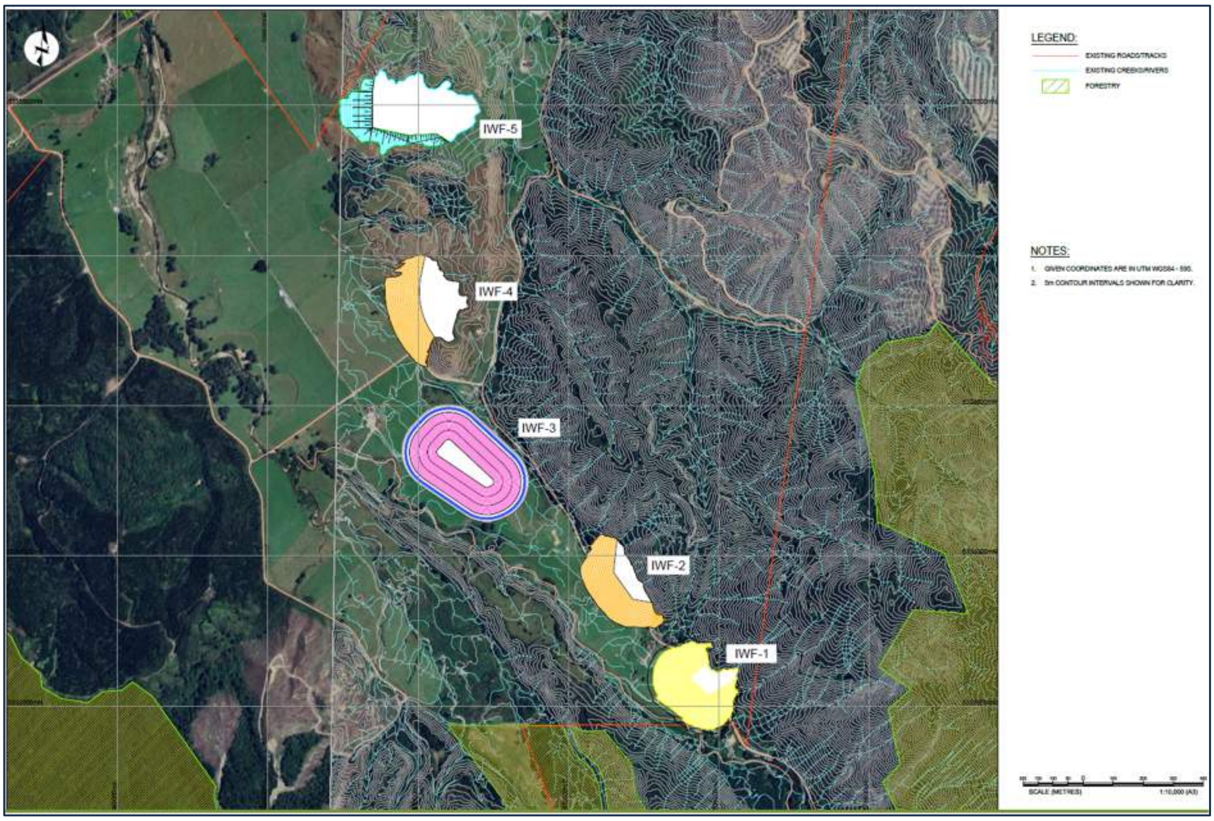


Figure 18-9: IWF siting options assessed in the study.

18.10.3 Design Parameters

The preliminary design parameters for the IWF have been developed using the current understanding of the Project's metallurgical characteristics, geotechnical conditions, seismic environment, and hydrological setting. These parameters establish the basis for sizing the facility, defining its operational requirements, and assessing its long-term performance and stability. They reflect the available laboratory test work, production forecasts, and regional environmental data, and will be refined as additional site investigations, geochemical testing, and engineering studies are completed in subsequent Project phases.

Tailings Production:

- annual tailings production: ~235 ktpa
- LOM tailings mass: ~1.33 Mt
- tailings allocated to underground backfilling: ~0.3 Mt
- tailings requiring surface storage: ~0.9 Mt
- surface storage volume: ~520,000 m³.

Tailings Properties:

- particle size distribution: P_{80} ≈ 106 µm
- solids specific gravity: 2.86–2.91
- expected geochemical behaviour: non-acid forming to low-potentially acid forming, with elevated As and Sb requiring monitoring.

Seismic Design:

- operating basis earthquake: 1 in 475 years (PGA ≈ 0.46 g)
- safety evaluation earthquake: 1 in 5,000 years (PGA ≈ 0.93 g)
- post-seismic stability to be assessed using residual shear strength parameters.

Hydrological Design:

- average annual rainfall: ~2,057 mm
- potential evapotranspiration: ~749 mm
- design storage allowance: 1 in 100-year average recurrence interval (ARI) wet season
- extreme storm storage: 1 in 1,000-year ARI 72-hour event.

18.10.4 Facility Design

The IWF design addresses the site's high rainfall, steep terrain, and elevated seismicity, all of which strongly influence the selection of construction materials, operational strategies, and long-term stability requirements. The design approach prioritises robustness, simplicity, and resilience, recognising that filtered tailings placement in such an environment presents practical limitations that must be explicitly addressed in the engineering basis. The design has been developed with ANCOLD Guidelines (2019) as the primary technical reference for dam safety, stability, and performance expectations.

From a tailings design perspective, achieving consistent and rigorous compaction of filtered tailings to densities that would achieve a dilative state parameter and reliably prevent liquefaction is unlikely under the high-rainfall conditions at Auld Creek. Frequent wetting, variable moisture content, and operational constraints reduce the feasibility of maintaining the degree of compaction required for the tailings to perform as a structural material. For this reason, the IWF will incorporate a structural embankment with sufficient size and strength to ensure stability under both rainfall and seismic loading. The current design assumes construction of this structural zone using imported quarry rock, with final volumes dependent on material availability from the mine plan and refined as the design advances. If quarry rock is prohibitively costly or unavailable, an alternative approach involving a cemented tailings mixture may be adopted to reinforce the batter slopes and provide a structural zone with higher shear strength.

At this stage, ~520,000 m³ of filtered tailings are expected to be placed in IWF-2 during the first 6 years of operation (with a further 520,000 m³ allocated to IWF-4 for potential mine expansion), after accounting for the volumes directed to underground backfilling. These estimates will continue to be refined as the design progresses and as additional geotechnical, hydrological, and operational data become available.

The conceptual configuration of the IWF is presented in Figure 18-10 for IWF-2 and Figure 18-11 for IWF-4, illustrating the overall layout and embankment geometry for each phase. Representative internal geometry and waste placement profiles are provided in Figure 18-12, which presents a typical cross-section for both facilities. Together, these figures define the spatial arrangement and structural zones that underpin the facility design.

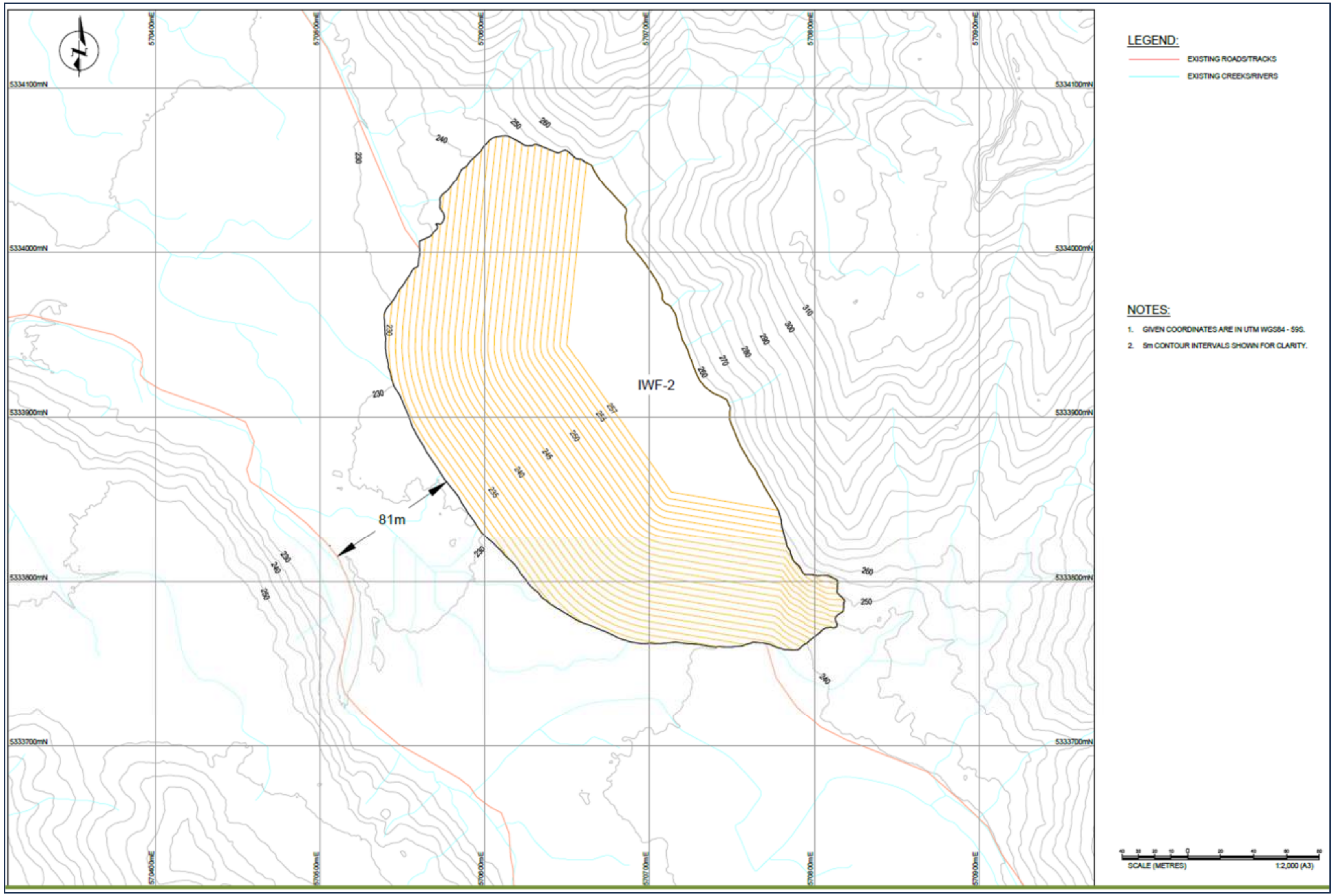


Figure 18-10: Conceptual layout of IWF-2.

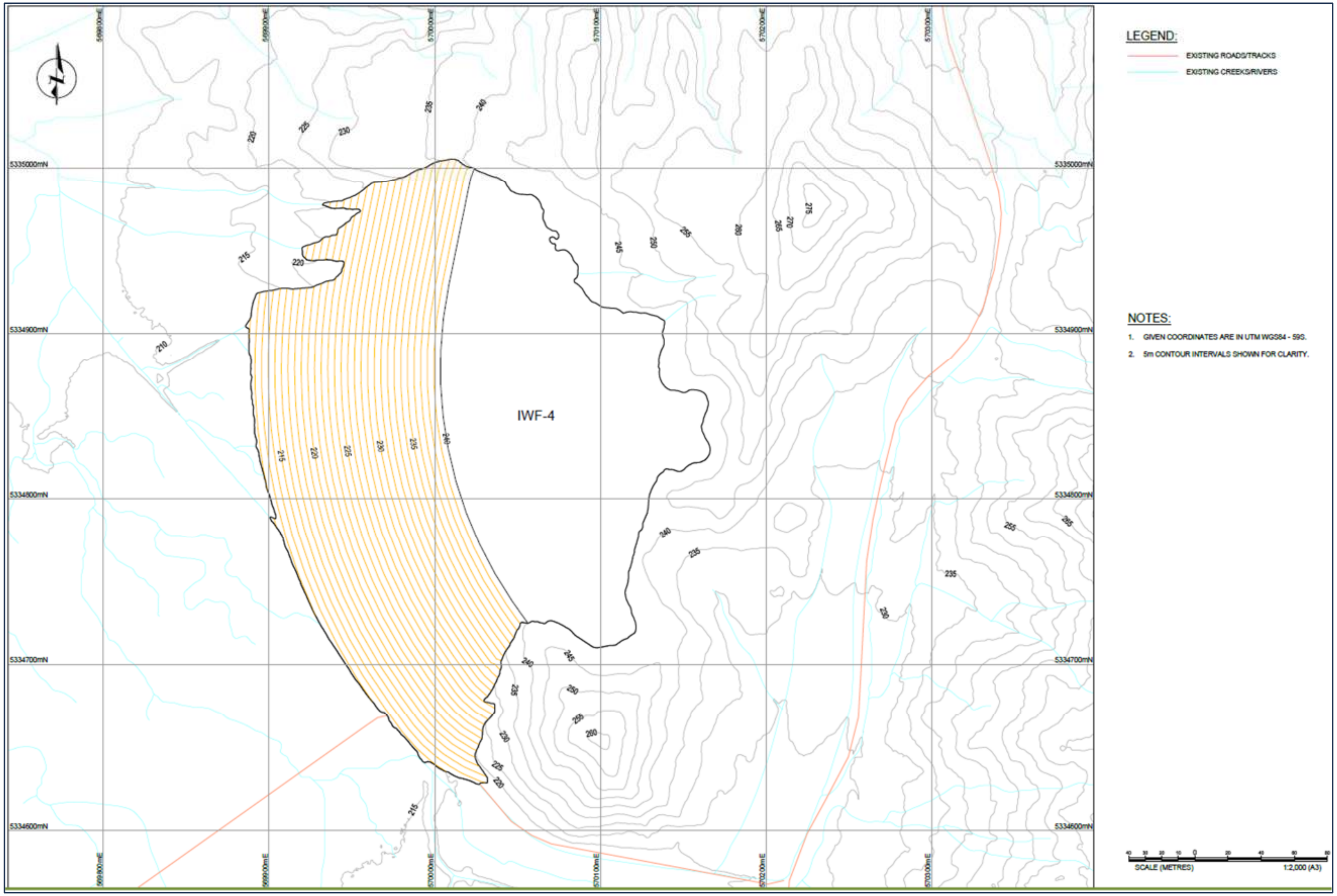


Figure 18-11: Conceptual layout of IWF-4.

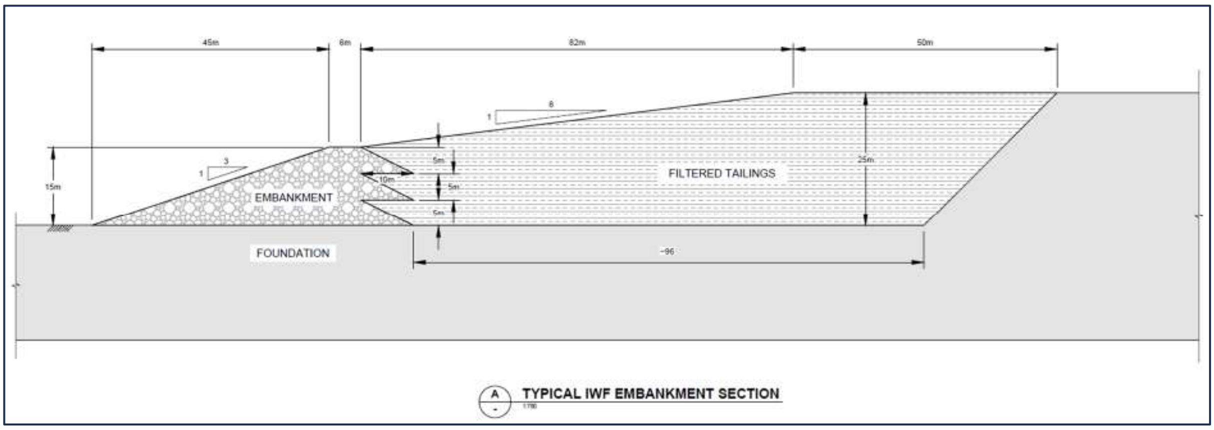


Figure 18-12: Conceptual cross-section of IWF-2 and IWF-4.

18.10.5 Haul Access

Haul access to the IWF is designed to provide safe and efficient movement of filtered tailings and construction materials throughout operations. The routes follow natural terrain where possible to minimise earthworks, maintain suitable gradients, and reduce environmental disturbance. Road geometry, platform widths, and drainage are developed to ensure all-weather performance in the region's high-rainfall conditions and to meet Australian National Committee on Large Dams (ANCOLD)-aligned safety expectations. The haul roads will accommodate either rigid body or articulated dump trucks with appropriate passing bays and traffic-management controls. As operations transition from IWF-2 to IWF-4, the haul network will be extended and upgraded to maintain efficient haul distances and operational continuity.

18.10.6 Environmental Control Dam

The ECD will function as the primary containment facility for all contact water generated within the IWF, including runoff from lined and tailings surfaces, seepage released during consolidation, and percolation of rainfall through the tailings.

The ECD will be a fully lined basin with 3H:1V internal batters and a storage capacity of ~38,000 m³ at full supply level, corresponding to a surface area of ~8,650 m². For the PEA, the lining has been assumed to be high-density polyethylene, but geotechnical investigation will be required to refine this selection.

The ECD has been sized based on long-term rainfall and evaporation datasets from the Reefton climate station, which provide the basis for estimating precipitation inputs, evaporation losses, and seasonal variability affecting the facility.

The conceptual water balance indicates that the ECD will maintain adequate freeboard under average climatic conditions with a discharge rate to the water treatment plant of ~4 L/s. The required discharge rate for higher rainfall conditions will be defined at later design stages through probabilistic modelling. Sensitivity testing demonstrates that the facility can also accommodate the 1-in-1,000-year, 72-hour storm event of 379 mm within the available flood storage above the normal operating level.

Overall, the ECD is considered appropriately sized for the PEA stage. The QP (Marius Phillips) recommends further refinement of hydrological inputs, operational assumptions, and storage requirements as the IWF engineering, water management strategy, and processing plant design are advanced during subsequent study phases.

18.10.7 Preliminary Closure Plan

A preliminary closure concept has been developed for the IWF and ECD, consistent with the level of engineering completed for the PEA. Closure of the IWF will focus on establishing a stable, low-permeability landform that minimises infiltration, controls erosion, and supports long-term revegetation. The final tailings surface will be covered with a compacted soil layer or geosynthetic layer to reduce percolation, followed by a mulching layer to promote vegetation growth and protect against surface erosion. Once the cover system is placed, a surficial drainage network will be constructed to direct runoff towards designated channels and prevent ponding on the closed landform. Additional closure measures, including slope stabilisation, erosion-resistant armouring where required, and long-term monitoring of water quality and landform performance, will be refined during the PFS as geotechnical, hydrological, and operational inputs are further developed.

19 Market Studies and Contracts

The Project is expected to produce Au and Sb in the form of concentrates. Metal price assumptions applied in this PEA were developed using a combination of historical price data and long-term consensus forecasts from mining industry analysts. The Au and Sb prices used in this Report were provided by RUA and are based on price assumptions used in the February 2026 Independent Technical Report (Whaanga, 2026): the 36-month average spot price for Sb (USD 25,000/t) and 24-month average spot price for Au (USD 3,000/oz). RUA then weighted the price to closer align with the long-term market consensus pricing analysts of USD 25,000–32,000/t for Sb and USD 3,600/oz for Au.

Final long-term metal prices adopted for the study are:

- Gold: USD 3,300/oz; and
- Antimony: USD 27,000/t.

RUA has initiated preliminary discussions with a number of potential off-take counterparties for both concentrates. As at the effective date of this Report, no binding terms or agreements have been finalised. For the purposes of this PEA, a payability assumption of 95% has been applied to both Au and Sb concentrates. This assumption is based on high-level market analysis and reflects typical industry expectations, including allowances for minor variability in concentrate specifications and the presence of penalty elements within acceptable limits.

As at the effective date of this Report, no material contracts, off-take agreements, or hedging arrangements are in place. RUA has not committed any portion of future production and retains full exposure to market pricing. RUA considers the market and commercial assumptions applied in this study to be appropriate for a PEA, noting that further work will be required at subsequent study stages to confirm concentrate specifications, payability terms, and commercial agreements.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

This section summarises environmental, permitting, and social considerations relevant to the Project, based on available information and the preliminary level of engineering associated with this PEA. The level of detail is appropriate for a concept-stage study and is subject to refinement through future studies.

20.2 Environmental Setting and Baseline Information

New Zealand's principal environmental legislation is the *Resource Management Act 1991* (RMA), which regulates the impacts of activities on land, water, and air (see Section 4).

Environmental baseline information for the Project area is currently limited and has been derived from publicly available information and targeted studies completed to support exploration activities.

Specific studies completed to date include:

- a bat survey undertaken by Wildlands (Giejsztowt et al., 2023); and
- an archaeological assessment undertaken by Southern Archaeology (Petchey, 2022).

These studies were completed to support exploration access arrangements and provide an initial understanding of environmental and cultural values within the Project area.

No comprehensive baseline monitoring programmes (e.g. surface water, groundwater, geochemistry, flora, or fauna) have been completed at this stage. Such programmes will be required to support future permitting.

Land within the Project area comprises predominantly conservation land forming part of the Victoria Conservation Park, administered by the Department of Conservation (DOC), with smaller areas of freehold land and forest land administered by Timberlands West Coast (Section 4).

20.3 Environmental Compliance and Liabilities

To the best of the knowledge of RUA/RRPL and the QP (Abraham Whaanga), no breaches of the RMA or any other environmental legislation have been committed. Neither RRPL nor RUA has been the subject of enforcement proceedings for breaches of environmental obligations (Section 4).

No material environmental liabilities have been identified based on available information. However, this conclusion is preliminary and will require confirmation through further investigations, including:

- geochemical characterisation of mineralised and waste materials;
- baseline water quality and hydrological studies; and
- ecological surveys.

20.4 Permitting and Regulatory Framework

20.4.1 Minerals and Land Access

Mineral rights in New Zealand are governed by the CMA, administered by NZ Petroleum and Minerals (NZP&M) on behalf of the Minister of Resources (Section 4).

RRPL is the 100% holder and operator of exploration permit EP 60648, which covers an area of 40.64 km² and was granted on 19 March 2021 for a 5-year term (Section 4). An EOD was granted for an additional 5-year term, expiring on 18 March 2031.

An exploration permit provides the holder with the exclusive right to explore for specified Crown-owned minerals within the permit area, subject to compliance with permit conditions and applicable legislation.

20.4.2 Land Access and DOC Requirements

As outlined in Section 4, the granting of a permit under the CMA does not confer a right of land access beyond minimum impact activities. Access agreements with landowners or occupiers are required for activities exceeding minimum impact thresholds. Access to conservation land requires approval from DOC, including:

- a concession for minimum impact activities (MIA); and
- an access arrangement (AA) for higher-impact activities such as drilling.

RRPL previously held an MIA concession (102174-MIA) for EP 60648 and an AA (93190-AA), both of which expired in March 2026.

A variation to extend the AA has been submitted, and RRPL is currently operating under an interim authority to enter and operate (AEO) while the application is being processed (Section 4).

20.4.3 Resource Management Act Requirements

Exploration and mining activities must comply with the RMA, either as permitted activities or through the granting of resource consents by regional or district councils (Section 4).

The RMA classifies activities into categories ranging from permitted to prohibited, which determines whether resource consent is required and the level of assessment.

Activities are subject to a general obligation to avoid, remedy, or mitigate adverse environmental effects.

20.4.4 Permitting Status

RRPL holds the necessary permits under the CMA for current exploration activities and has secured, or is in the process of securing, the necessary land access arrangements (Section 4).

Based on the QP’s (Abraham Whaanga) review of EP 60648 and associated agreements, no significant risks have been identified with respect to RRPL holding sufficient permits and surface rights to allow effective exploration of the permit area as at the effective date of this Report (Section 4).

20.5 Environmental and Permitting Risks

Key environmental and permitting risks include:

- the requirement for additional baseline environmental data to support future approvals;
- permitting complexity associated with activities on conservation land;
- potential for public notification and appeals under the RMA; and
- evolving regulatory requirements.

These risks are typical of projects at the exploration and PEA stage and are considered manageable with appropriate studies and stakeholder engagement.

20.6 Social and Community Considerations

Mining in New Zealand is a sensitive and political subject, and projects may be subject to opposition from environmental and community groups.

However, as noted in Section 4, the West Coast region (including the Buller and Grey districts) has historically demonstrated stronger support for mining compared to other parts of New Zealand.

The Reefton area has an extensive mining history, and mining has made a significant contribution to local employment and economic activity. This contributes to a comparatively favourable social licence environment.

Recent consenting outcomes in the region, including the Snowy River mine processing plant and the Barrytown mineral sands project demonstrate that mining projects can obtain regulatory approval, although typically subject to conditions and, in some cases, appeals.

Community and stakeholder engagement undertaken by RUA and its consultants to date indicates generally positive feedback. Engagement has informed project planning and measures to reduce potential environmental and social impacts.

A Social Impact Assessment (SIA) is currently underway to identify potential impacts and benefits to the local community and to define management measures required to support a social licence to operate.

20.7 Mana Whenua

The Project area lies within the rohe of mana whenua. As noted by the QP (Abraham Whaanga), RUA engages with iwi and hapū whose interests may be affected by the Project (Section 4).

Engagement with mana whenua will remain an important component of Project development and future permitting processes, including:

- identifying cultural values;
- assessing potential impacts; and
- supporting consent applications.

20.8 Waste and Water Management

Waste and water management considerations are at a conceptual stage consistent with the level of engineering in this PEA.

Future work will include geochemical characterisation of mineralised and waste material, development of water management strategies, and assessment of potential acid rock drainage and metal leaching. These studies will inform future design and environmental assessment.

20.9 Mine Closure and Reclamation

Closure planning is conceptual at this stage and based on standard industry practices.

Key considerations include rehabilitation of disturbed land, long-term management of waste materials, and post-closure water quality. Closure costs have been estimated at a preliminary level and included in the economic analysis.

20.10 Regulatory Outlook and Opportunities

As outlined in Section 4, the Fast-track introduces a coordinated approvals regime for projects of national or regional significance.

This framework has the potential to streamline approvals across multiple statutes, including the RMA, CMA, and Conservation Act. While applicability to the Project cannot be confirmed at this stage, it represents a potential opportunity to reduce future permitting timelines.

21 Capital and Operating Costs

21.1 Capital Cost Estimate

21.1.1 General

Capital cost estimates have been prepared by Mining One, PBG, and KPPL for their respective scopes, as described in sections 21.1.2, 21.1.3 and 21.1.4. These have then been consolidated into a Project capital cost estimate in Section 21.1.5.

Similarly, each consultant has prepared an operating cost for their respective scope, and then these are consolidated into a Project operating cost estimate in Section 21.2.4.

The following approach has been used:

- All cost estimates are presented in US dollars at a base date of Q2 2026. No escalation beyond the base date has been applied.
- Capital cost estimates are for Project execution only. Costs prior to execution are excluded (such as resource drilling campaigns, further test work, feasibility studies, permitting, land purchases and community engagement).
- RUA is responsible for the owner's cost estimate, typically 3% of Project direct costs.
- RUA and the consultants contributing to this PEA report have agreed to allow project contingency at 30% of all other pre-production costs (including direct, indirect, Engineering, Procurement, and Construction Management (EPCM), and owner's costs). This is appropriate for the maturity of the Project, and the availability of recent cost data for projects with similar scope. A 5% contingency was applied to mining sustaining capital costs, including underground development. No other contingencies were allocated.
- Closure costs were included in the capital cost estimate.

21.1.2 Mining Scope

Cautionary Statement: the capital and operating cost estimates presented in this report are preliminary in nature and have been prepared at a conceptual level of accuracy consistent with a PEA under NI 43-101. The estimates are based on a combination of conceptual mine designs, benchmarked costs from comparable operations, budgetary quotations, and engineering judgement. As such, cost estimates are subject to a high degree of uncertainty. There is no certainty that the estimated costs will be realised, and actual costs may differ materially due to changes in Project scope, market conditions, mining and processing assumptions, permitting requirements, labour availability, and other factors. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA will be realised.

Capital and operating costs were derived from:

- values supplied by RUA;
- review of Mining One's previous studies on similar mines and escalation for inflation;
- Mining One personnel's experience working in similar operations; and
- request of quotes or estimates from known suppliers and mines.

Note that, at the PEA level, cost estimates are expected to be accurate to an order of magnitude ±50%. These costs will be further refined at the PFS level. RUA requested that the economic model be provided in USD, and hence costs are generally displayed in USD unless otherwise stated. Pre-production capital costs were incurred starting in May 2027 (date provided by RUA). This was used as the date to discount value to in the economic model.

21.1.2.1 Underground Fleet

Underground fleet was not considered for capital cost as RUA has engaged Terra Firma (a New Zealand mining contractor) to operate the mine. Mining costs were calculated based on industry-standard estimates and an overhead cost was applied.

21.1.2.2 Capital Development

Development advance rate costs were estimated using a 2022 Australian mining contract and then scaled back to reflect the smaller development profiles for Auld Creek (number of bolts and surface supports per cut) and then escalated by 12% to reflect inflation since 2022. Table 21-1 lists the advance rates per profile used in the economic model. A narrow-vein Au mine in Victoria with known costs per development metre was also used as a comparison point, particularly for mineralised drive development rates, where the ground support and mining method are expected to be similar.

Table 21-1: Development costs per metre.

Profile	Cost (USD/m)
3.5 mW x 4.0 mH	3,200
3.5 mW x 3.5 mH	3,100
3.5 mW x 4.5 mH	3,400
3.0 mW x 3.5 mH	1,700
1.4 mD Raise	1,600
3.5 mD Raise	3,400

Total capital development cost was calculated as USD 52.4 million.

21.1.2.3 Resource Drilling

Exploration drilling cost was conservatively estimated using:

- 25 m × 25 m spacing for resource drilling; and
- 15 m × 15 m spacing for grade control drilling.

The QP (Gary Davison) used deposit geometry and expected vertical advance per year to calculate approximate drilling demand per month. The results are presented in Table 21-2.

Table 21-2: Resource drilling cost estimation.

Item	Estimate	UoM
Drilling cost	130	USD/m
Resource drilling	960	m/mo
Grade control drilling	1,030	m/mo
Total resource drilling cost	9.4	USD million
Total grade control drilling cost	9.1	USD million

21.1.2.4 Mine Haul Roads

An allowance was made for construction of the mine haul road (excluding road base material acquisition) and is summarised in Table 21-3. Note that other on-site roads and upgrades to public roads are discussed in the PBG scope.

Table 21-3: Road capital cost.

Item	Cost Estimate	UoM
Mine Haul Road Capital Cost	632,000	USD

21.1.2.5 Portal costs

Other capital costs considered are summarised in Table 21-4.

Table 21-4: Portal costs.

Item	Cost estimate	UoM
Primary fans (4 × 90 kW fans)	498,000	USD
Primary fan civil work	194,000	USD
Power, water and services[1]	4,000,000	USD
Portal clearance and establishment	210,000	USD
TOTAL	**6,702,000**	**USD**

Notes:

1. Services estimation was derived from a previous Mining One study for a similar mine, and is inclusive of establishment of air, water, and power to the entire site.

21.1.2.6 Mining Scope Closure Costs

Closure costs (including rehabilitation of the mine portals and ROM pad) were approximated using a similar Mining One study and escalated to suit (Table 21-5). This was applied in the last month of the economic model.

Table 21-5: Closure costs.

Item	Cost estimate	UoM
Closure cost	2,300,000	USD

21.1.2.7 Total Capital Cost for Mining Scope

Total capital cost for the mining scope is USD 110 million (Table 21-6). Owner's costs have been applied as 3% of direct/indirect costs. Contingency has been applied as 30% of direct/indirect and owner's costs.

Table 21-6: Total capital cost for mining scope at a base date of Q2 2026.

Item	Direct and Indirect Costs (USD million)	Owner's Cost (USD million)	Contingency (USD million)	Total Capital Cost (USD million)
Underground fleet	N/A			
Capital development	52.4	2.62	16.5	71.5
Resource drilling	9.4	0.47	3.0	12.8
Grade control drilling	9.1	0.46	2.9	12.4
Portal costs	6.7	0.34	2.1	9.2
Mine haul roads	0.6	0.03	0.2	0.9
Closure	2.3	0.12	0.7	3.1
TOTAL	**80.5**	**4.0**	**16.5**	**109.9**

21.1.3 Processing Plant and Site Infrastructure

21.1.3.1 Introduction

This capital cost estimate for the processing plant and site infrastructure for the Project PEA is based on the following:

The processing plant design detailed in Section 17.4 and depicted in Figure 17-1

Figure 17-1. The processing plant will process 0.25 Mtpa using a two-stage crushing and ball milling circuit to produce a suitable feed for the flotation circuit which produces two saleable concentrate products. Filtered tailings disposal is adopted, with a portion of the filtered tailings used for paste backfill production, to support the underground mining operation. Concentrate products are filtered and bagged for sale.

- The site infrastructure described in Section 18. This does not include tailings and waste rock storage (which is reported separately by KPPL) or mining infrastructure (which is reported separately by Mining One).

At a PEA level of definition, the capital cost estimate developed is a factored estimate based on the following supporting inputs and criteria:

- a tailored processing plant flowsheet which has been developed from recent (historical) metallurgical test work results;
- preliminary mechanical equipment list, using appropriately sized and recognised industry equipment;
- historical quotes for major equipment items from vendors, sourced from recent projects and the PBG in-house equipment pricing database;
- other mechanical equipment items pricing obtained from historical pricing from other recent or similar in-country projects;
- concept-level design for major buildings, pads, roads and electrical distribution infrastructure; and
- cost allowances for site drainage and water management.

At a PEA level of accuracy, the anticipated level of accuracy of the capital estimate is anticipated to be ±35% as per the American Association of Cost Engineers (AACE) cost guidelines (Table 21-7).

Table 21-7: AACE Classification guidelines for capital cost estimates (RP 47R-11).

Class of Estimate	Purpose	Accuracy		Level of project completion
		Lower limit	Upper limit	
Class 5 Order of Magnitude Estimate	Scoping Study	-50% to -20%	+30% to +100%	0% to 2%
Class 4 Study Estimate	Prefeasibility Study	-30% to -15%	+20% to 50%	1% to 15%
Class 3 Definitive Estimate	Feasibility Study	-20% to -10%	+10% to 30%	10% to 40%
Class 2 Detailed Estimate	FEED/EPC Bid Preparation	-15% to -5%	+5% to 20%	30% to 70%
Class 1 Check Estimate	Tender	-10% to -3%	+3% to 15%	50% to 100%

21.1.3.2 Basis of Estimate

This capital cost estimate allows for the elements listed in Table 21-8 (while noting that there is little or no quantification of the sub-elements at this PEA phase):

Table 21-8: Elements of the capital cost estimate.

Cost Element	Sub-Elements
Direct Costs	Mechanical equipment supply and installation, including: • Mechanical equipment in the process flowsheet • Maintenance equipment such as mill reline equipment, cranes, jibs, hoists, and mechanical equipment in workshops • Equipment for supply of utilities and services such as compressors, receivers and batching systems • Water treatment systems • Mobile equipment such as vehicles, forklifts, and mobile cranes • First set of wear liners for crushers and mills

Cost Element	Sub-Elements
	Platework supply and installation, including: • Field-fabricated and shop-fabricated steel tanks • Chutes and launders • Bins and hoppers
	Electrical equipment supply and installation, including: • Powerlines • Transformers, power correction and harmonic conditioning • Electrical switch rooms complete with air conditioning and fire protection • All switchgear and drives • Process control system and networking • All field instrumentation and control valves (modulating and on-off) • On-stream analyser (OSA) • All site communications systems • All power and control cabling • Uninterruptible power supplies and backup generators • Earthing and lightning protection • Site lighting
	Piping supply and installation, including: • All process piping and piping fittings • All manual valves • All pipe supports • Pressure testing and leak testing • Pipe marking
	Earthworks and concrete supply and installation, including: • Bulk and detail earthworks • Construction of pads, embankments, ponds, and retaining structures • Roads, culverts, drainage, trenching and general excavations • Ground improvement measures such as earth replacement, compaction, and piling • Concrete foundations, reinforcing, slabs, and all embeds • Paving and surfacing • Pond liners and geofabrics • Site security fencing
	Structural steel supply and installation, including: • Structural steel fabrication and painting • Procurement of all bolts and fasteners • On-site steel erection • All roofing and cladding • All building features such as doors, windows, stairs, elevated working areas, internal walls, internal fit-out, roof ventilators, roller doors • Prefabricated and modular buildings
Indirect Costs	EPCM costs to: • Complete the Project design (from pre-feasibility through detailed design) • Procure equipment, materials and bulk commodities • Secure contracting services needed to construct the Project • Manage the Project construction • Assist with commissioning and ramp-up • Deliver construction completion documents such as vendor documentation and as-builts
	Construction indirect costs, including: • Common distributable costs such as hire of cranes, scaffolding, formwork, temporary fencing, temporary construction offices, and similar shared construction costs • Freight • Customs • Tools, PPE and consumable items • Construction crew supervision and training

Cost Element	Sub-Elements
	Operational readiness, including: • First fills • Spares and minor equipment • Vendor support during commissioning
Owner's Costs	Owner's costs, including: • Owner's team costs through to first production (such as recruitment, staffing, travel, expenses and training) • Cost of consultants and contractors engaged directly by the owner • Operating costs through commissioning up to the first production milestone. There are a myriad of other costs incurred by an owner (such as insurances, legal support, permitting costs, project accounting, community engagement, and land purchases). The owner needs to define which costs are included and excluded in owner's costs.
Contingency	Contingency is allowed as a lump sum which is calculated as a percentage of all above costs. This contingency allows for the uncertainty in the capital cost estimate at the PEA level due to relatively low level of definition of scope, quantities, and costs.

For the processing plant, major items of processing plant equipment were sized based on the process design criteria and then a major mechanical equipment list (MEL) was specifically developed for the Project. Supply of each major equipment item was costed, based on either a recent/historically quoted, benchmark or PBG's database price of recent similar projects. Electrical equipment, bulk commodities (such as piping, cable, steel and concrete) and installation labour were estimated by factoring based on mechanical equipment supply costs. Factors used are based on PBG's experience with recent, similar projects in remote locations of industrialised countries. Where possible, those estimates in a cold-climate application have been given priority. At the PEA level, plant layout is still conceptual and so there has been no quantity estimate for bulk commodities.

Both concentrate filters have been sized to achieve <9% cake moisture, to reduce the chance of freezing/agglomeration of bagged concentrates. Loading of filtered concentrate into 1-tonne bags has been adopted as the preferred concentrate shipping method. Consequently, the capital estimate includes allowance for the purchase of a bespoke concentrate bagging plant.

An allowance for in-plant buildings has been factored based on the number and size of buildings from similar sized mineral processing plants. The exact functions to be conducted on site will determine the nature of the buildings required, workshop functions to be conducted and on-site warehousing requirements.

The capital cost estimate has been developed on the following basis.

- The capital cost estimate is expressed in USD at a base date of Q2 CY 2026. The costs described will be incurred at future unknown dates, but forward escalation has not been applied to the estimate. Project schedule, timing of cashflows and forward escalation will be considered at later phases of this project. Future inflation and changes in exchange rate have not been considered.
- The capital cost estimate is an unbiased estimate of the scope as it is currently defined, so that it is neither conservative nor optimistic.
- For a PEA study (where a project is at <2% completion) there are typically many client requirements that have not yet been defined (such as level of automation, equipment sparing philosophy, degree of modularisation, supplier preferences). The requirements RUA has provided to date are included. Where no specific requirement has been defined, the capital cost estimate is factored from other projects and so allows for typical client requirements.
- The capital cost estimate is typically based on a stick-build construction strategy with limited off-site construction of modular facilities. The main exception is the electrical switchrooms,

which are costed on the basis that they are fully fabricated, fitted out, and tested off-site and then transported to site as modules. Other skid-mounting and modularisation opportunities will be considered at later phases of the Project.

- The following exchange rates have been used to convert to USD from other currencies:
  - 1 NZD = 0.58 USD
  - 1 AUD = 0.70 USD.
- No allowance has been made in this capital cost for future expansion of the processing plant, haulage of ROM ore from other mines, or extension of the Auld Creek mine life through an increase in reserves.
- A general allowance has been made for capital and operating spares, but spares requirements have not yet been defined. Specific high-cost capital spares such as mill motors and transformers, have not been identified or allowed for.
- No allowance has been made in this cost estimate for force majeure events such as supply chain disruptions, material shortages, labour shortages, epidemics, wars/conflicts, or disruptions of global finance.
- Geotechnical conditions for processing plant and infrastructure locations are unknown. The capital cost estimate assumed that geotechnical conditions are reasonable and therefore earthworks and foundation costs are similar to other PBG projects. An allowance for piling has not been included.
- Geochemical behaviour of tailings and waste rock is being assessed by others. Potential for acid generation and contamination of runoff water is currently unknown. The capital cost estimate makes a general allowance for active treatment of runoff, but scope and cost may change in the future once geochemical testing has progressed.

21.1.3.3 Exclusions

The following items were excluded from this capital cost estimate and will need to be provided by others:

- Mining and mine infrastructure (to be provided by Mining One) such as all development works, temporary waste storage stockpiles, haul roads, ventilation, mine service facilities and pads, explosives magazines, and mobile equipment. PBG's battery limits with regards to mining are:
  - distribution of power from switchrooms located at the main portal and upper portal;
  - supply of raw water to the entrance of each mine portal;
  - receipt of mine dewatering flow at each portal; and
  - pumped backfill paste at the paste plant.
- IWL facilities (to be provided by KPPL) such as tailings embankments, liners, water management, environmental ponds, access roads, pumps, and support facilities.
- Construction camp costs, construction messing and accommodation, mobilisation and demobilisation fees.
- Further metallurgical testing.
- Geotechnical investigation and soil testing.
- Further orebody and mining studies.
- Environmental studies and monitoring.
- Land purchase or leasing costs including mining leases.
- Community assistance programs, donations, and sponsorships.
- Insurances, including those during construction, motor vehicle, public and professional liability.
- Operator training and ramp-up costs.
- Pre-production operating costs.
- Escalation beyond the estimate base date.
- Allowances for variation in exchange rates.
- GST and other taxes.

- Capitalised spare parts except for commissioning spares.

21.1.3.4 Estimate Detail

The methodology for estimating quantities and costs is further detailed in Table 21-9.

Table 21-9: Estimation methodology.

Commodity or Scope	Details
Mechanical equipment Supply & Install	Supply of mechanical items, as detailed in the MEL, has been individually priced, with the equipment pricing sourced from database figures for concentrators of a similar scale and process configuration. Installation hours have not been estimated from first principles but a benchmarked installation factor of 30% of the mechanical equipment supply cost has been adopted. Benchmark information from a range of concentrators suggests that this installation factor ranges from 18–30%, with a mid-point of 25%. The upper range factor was adopted due to the relatively small nature of the equipment compared to some of the equipment in the Project database and to accommodate installation of certain processing plant aspects within an enclosed building (e.g. concentrate bagging).
Platework Supply & Install	Platework supply and install cost was determined by using a benchmarked factor of 15% of mechanical equipment supply cost, based on similar concentrator projects. Platework typically benchmarks with a range of 8–20%, with the upper end of the range selected to accommodate for the use of bins for crushed ore storage (rather than on-ground stockpiles) and tankage for water storage (instead of lined earthen ponds).
Piping Supply & Install	Piping supply and install cost has been determined by using a benchmarked factor of 25% of mechanical equipment supply cost. Piping typically benchmarks with a range of 15–30%. The higher end of this range is used because the Project piping does not provide efficiencies of scale.
Electrical and Instrumentation Supply & Install	Electrical supply and install have been factored at 31% of the mechanical equipment supply cost. Instrumentation supply and install costs have been factored at 10% of the mechanical equipment supply cost. Based on benchmarking data of similar concentrator projects, as well as recent projects within the same geographical location, this combined factor varies from 30–50% of the mechanical equipment supply cost, depending on the desired level of monitoring, control, and automation. A mid-range factor was adopted on the basis that a moderately high level of instrumentation, in line with current industry best practices, and the low labour complement envisaged, is the preferred strategy for Auld Creek. Also, critical instrumentation (OSA) has been captured within the MEL and individually priced, further supporting the use of a mid-range factor.

<table>
<tr><th>Commodity or Scope</th><th>Details</th></tr>
<tr><td>Earthworks and Concrete Supply & Install</td><td>For the purposes of the capital estimate, it was assumed that the selected plant site is essentially flat with minimal bulk earthworks requirements and little opportunity to use topography for materials movement and flow. A single level pad will be created for the processing plant, and associated cut-and-fill quantities were estimated. The following unit costs were allowed for civil works:
• AUD 5 per m^3 of cut excavation
• AUD 3 per m^3 for supply of rough fill
• AUD 35 per m^3 for supply of structural fill
• AUD 50 per m^3 for ground improvement (including excavation, geomembranes, replacement fill, and compaction)
• AUD 5 per m^3 of placement and compaction.
Civil works at the processing plant and ROM pad allow for cut of 45,000 m^3 and supply/placement of 96,000 m^3 of structural fill at a direct capital cost of USD 3.4 million.
Geotechnical data are not yet available, but the ground is potentially alluvial with poor bearing strength. Allowance was made for ground improvement under heavy equipment in the processing plant (for a total area of 4,000 m^2 and to a depth of 1.2 m). Direct capital cost for this ground improvement is USD 170,000.
Concrete works has been factored using a benchmarked factor of 39% of the mechanical equipment supply cost based on similar-scale concentrator projects. The factor is considered adequate for the available flat site, assumed reasonable geotechnical conditions and small size of the plant equipment. Earthworks have been factored at 31% of mechanical equipment supply cost based on similar concentrator projects. Geotechnical test work is needed at future phases to confirm earthworks/foundations design and cost.</td></tr>
<tr><td>Structural Steel Supply & Install</td><td>Structural steel benchmarks at 15–30% of the mechanical equipment supply costs. A factor of 28% of the mechanical equipment supply cost has been adopted based on similar concentrator projects, to accommodate for the relatively small equipment required within the flowsheet, the required seismic design loadings and on the basis that steel will be sourced from local suppliers.</td></tr>
<tr><td>Infrastructure</td><td>Infrastructure scope and cost vary widely between projects, and so it is not appropriate to estimate cost by factoring. Instead, infrastructure capital cost has been built up on the following basis:
• Road construction and upgrade costs have been provided by RUA:
  o NZD 0.8 million per km for widening of sealed roads
  o NZD 2.2 million per km for new sealed roadways
  o NZD 30,000 per metre length for creek crossing fords
• Overhead powerlines have been costed at USD 200,000 per km
• Buildings have been costed using the following all-in cost metrics:
  o NZD 2250 per m^2 area for offices and control room
  o NZD 500 per m^2 area for shed and warehouse type structures
  o Adjustments have been applied to these basic metrics according to the degree of fit-out required.
• An allowance of USD 750,000 has been made for an active water treatment plant based on a similar nearby project.
• Other costs are based on the PBG's database of historical projects.</td></tr>
<tr><td>Indirect Costs</td><td>EPCM costs are benchmarked as in range 12–18% of direct costs across a wide range of mining projects. PBG has allowed EPCM costs at 18% of direct costs due to the relatively small scale of the Project.
Other indirect costs for construction and operational readiness have been estimated by factoring, and are the sum of the following:
• 12% of processing plant direct costs
• 12% of paste plant direct costs
• 5% of infrastructure direct costs.
Construction/operational readiness indirect costs for processing plants are typically in the range 8–12% of direct costs.</td></tr>
</table>

Commodity or Scope	Details
Owner's Costs	Owner's costs have been estimated by factoring, and are the sum of the following: • 3% of direct costs for the processing plant • 3% of direct costs for the paste plant • 1.5% of direct costs for infrastructure. Note that estimation of owner's costs is outside of PBG's scope and should be provided by RUA in future phases.
Contingency	A contingency allowance of 30% on direct + indirect + owner's costs has been included to account for uncertainties in scope, quantities, and costs at the current level of engineering design. This is within the expected accuracy range of an AACE Class 5 cost estimate (as shown in Table 21-7).

21.1.3.5 Capital Cost Estimate

The AACE Class 5 capital cost estimate for the Project PEA, based on the scope described in Section 1.1 and Section 1.14, is estimated to be **USD 105 million** inclusive of owner's costs and contingency. A summary of this capital cost estimate is provided in Table 21-10. Note the following late changes to this estimate at the conclusion of the PEA:

- Deletion of the paste plant.
- Use of quarried material (rather than mine waste rock) as a construction material. This change of construction material does not incur additional indirect, EPCM or owner's costs but contingency has been allowed because of quantity and price uncertainties.

Table 21-10: Capital cost estimate at base date of Q2 CY 2026.

Scope Area	Direct Cost (USD million)	Indirect Cost (EPCM) (USD million)	Indirect Cost (other) (USD million)	Owner's Costs (USD million)	Contingency (USD million)	Total Capital Cost (USD million)
Processing Plant	44.6	8.0	5.3	1.3	17.8	77.1
Paste Plant	Removed from scope					
Quarry Rock Import	3.6	0	0	0	1.1	4.7
Infrastructure	14.4	2.6	0.7	0.2	5.4	23.3
TOTAL	**62.5**	**10.6**	**6.1**	**1.6**	**24.2**	**105.0**

The build-up of direct costs for the processing plant is summarised in Table 21-11, and for infrastructure in Table 21-12.

Table 21-11: Direct costs for the processing plant at base date of Q2 CY 2026.

Area	Mechanical Equipment Supply (USD million)	Mechanical Equipment Install (USD million)	Earthworks Supply & Install (USD million)	Concrete Supply & Install (USD million)	Steelwork Supply & Install (USD million)	Platework Supply & Install (USD million)	Piping Supply & Install (USD million)	Electrical Supply & Install (USD million)	Instrumentation Supply & Install (USD million)	TOTAL (USD million)
Crushing and Ore Storage	2.71	0.81	0.84	1.06	0.76	0.93	0.68	0.84	0.27	**8.90**
Ball Milling and Classification	1.59	0.48	0.49	0.62	0.44	0.24	0.40	0.49	0.16	**4.90**
Bulk Antimony-Gold Rougher Flotation	2.29	0.69	0.71	0.89	0.64	0.34	0.57	0.71	0.23	**7.08**
Cleaner Flotation	1.16	0.35	0.36	0.45	0.32	0.17	0.29	0.36	0.12	**3.57**
Antimony Concentrate Thickening and Filtration	0.96	0.29	0.30	0.38	0.27	0.14	0.24	0.30	0.10	**2.98**
Gold Concentrate Thickening and Filtration	1.04	0.31	0.32	0.40	0.29	0.16	0.26	0.32	0.10	**3.20**
Tailings Thickening and Filtration	2.28	0.68	0.71	0.89	0.64	0.42	0.57	0.71	0.23	**7.12**
Reagents	0.66	0.20	0.20	0.26	0.18	0.10	0.16	0.20	0.07	**2.03**
Water Services	0.69	0.21	0.21	0.27	0.19	0.10	0.17	0.21	0.07	**2.14**
Air Services	0.39	0.12	0.12	0.15	0.11	0.06	0.10	0.12	0.04	**1.20**
Spillage Handling and Utilities	0.47	0.14	0.14	0.18	0.13	0.07	0.12	0.14	0.05	**1.44**
TOTAL	**14.23**	**4.27**	**4.41**	**5.55**	**3.98**	**2.74**	**3.56**	**4.41**	**1.42**	**44.6**

Table 21-12: Direct costs for infrastructure at base date of Q2 CY 2026.

Item	Direct Cost (USD million)
Upgrades to Public Roads	4.7
On-Site Roads and Tracks	0.4
Power Supply and Distribution	4.0
Non-Process Switchrooms	0.7
Site Buildings (Process and Non-Process)	1.2
Raw Water and Potable Water Supply and Distribution	1.0
Water and Sediment Management	1.6
Other Utilities and Services	0.8
TOTAL	**14.4**

21.1.3.6 Capital Cost Benchmarking of the Processing Plant (PBG)

Capital cost of the processing plant is benchmarked against other recent projects and studies to validate this part of the estimate. Note that infrastructure scope varies greatly from project to project, and so cannot be usefully benchmarked.

Figure 21-1 summarises the PBG benchmark information for processing plants on 33 projects (eight of similar capacity), including cold-climate projects. Of these 33 projects, at least 12 have been constructed in recent years while the others are at various stages of study, ranging from scoping study through to feasibility study. Note that the benchmark data provided in Figure 21-1 excludes the non-process infrastructure (NPI) cost, mining cost, paste plants, and contingency. The benchmarked range of capital cost for a 0.25 Mtpa processing plant (excluding contingency) is **USD 52–72 million**.

The processing plant for the Project is estimated to have a capital cost (excluding contingency) of **USD 59 million** and therefore lies in the middle of the benchmark range and is considered to be a reasonable estimate at AACE Class 5 accuracy level.

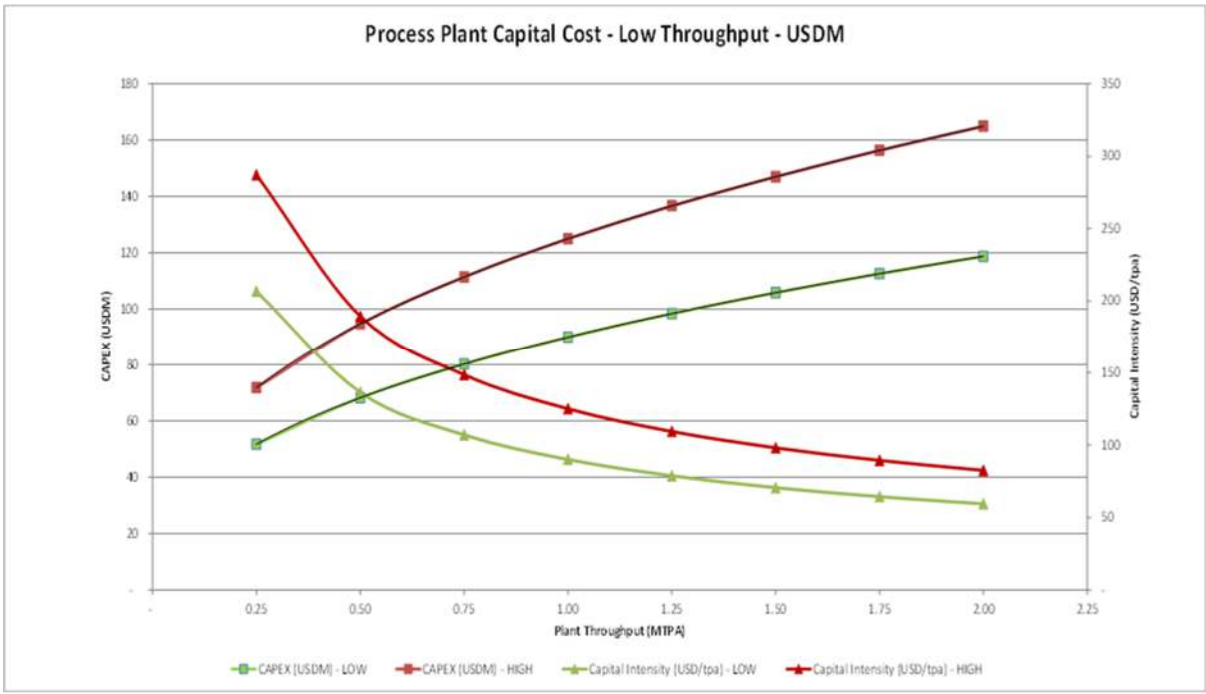


Figure 21-1: PBG benchmark capital cost curves.

21.1.4 Integrated Waste Facility

Capital costs for the IWF and ECD were determined for the initial development of IWF-2 and ECD-2. Costs include earthworks, lining systems, drainage infrastructure, road relocations, and engineering services.

The estimate is presented in Q2 2026 USD, which is the base date for all costs. No escalation beyond this base date was applied. The estimate assumes an exchange rate of 1 USD = 1.65 NZD, consistent with the average for Q2 2026.

The conceptual IWF design incorporates embankment raises, and the capital cost estimate includes the full embankment construction requirements associated with these raises. This assumes the raises will be conducted during early-stage construction when waste rock is available from the portal and decline development and prior to processing commencing; however, exact timing will be dependent on the final waste rock mining schedule.

The upfront direct capital costs for IWF-2 and ECD construction are estimated at USD 5.53 million. These costs cover clearing and foundation preparation of 57,000 m², underdrainage and basal drainage systems, geotextile protection, installation of a 2.0 mm-thick HDPE liner, 200,000 m³ of rock embankment, bulk haulage, diversion channels, instrumentation, and 500 m of road relocation of the existing Soldiers Road. Rock embankments will be constructed from quarried rock, and cost allowance has been made for 25% use of structural fill and the remainder of rough fill. Supply of this fill contributes USD 1.55 million direct cost. Bulk embankment haulage assumes an average haul distance of 400 m. No allowance has been made for rehandling from other areas of the site.

The road relocation allowance reinstates Soldiers Road strictly to its existing condition as an internal mine access road; the road remains unpaved, and no upgrades, improvements, or sealing to public road standards are included.

ECD-2 construction contributes USD 1.3 million of the USD 5.53 million upfront cost, and includes clearing, foundation preparation, geotextile protection, HDPE lining, and 40,000 m³ of cut-and-fill earthworks.

Detailed engineering and QA/QC services contribute USD 0.80 million of the USD 5.53 million upfront cost. No costs for engineering studies or designs prior to detailed design were included in this estimate.

Capital costs for the IWF-4 were not included in the cost estimate for the PEA as they relate to expansion beyond the currently identified resources.

Closure costs were estimated for IWF-2 and the ECD-2. Costs include earthworks but exclude closure plans, permitting, closure trials or ongoing maintenance and monitoring. Cost for the earthworks component of closure were estimated at USD 1.52 million.

Total capital cost of the IWF scope is summarised in Table 21-13.

Table 21-13: Capital cost estimate for the IWF scope at base date of Q2 2026.

Item	Direct and Indirect Costs (USD million)	Owner's costs (USD million)	Contingency (USD million)	Total Capital Cost (USD million)
IWF-2 Construction	3.38	0.10	1.04	4.52
ECD-2 Construction	1.35	0.04	0.42	1.81
Supply of Quarry Material	1.55	0	0.47	2.02
IWF-2 Closure	1.52	0.05	0.47	2.04
Engineering and QA/QC Services	0.8	0	0.24	1.04

Item	Direct and Indirect Costs (USD million)	Owner's costs (USD million)	Contingency (USD million)	Total Capital Cost (USD million)
TOTAL	**8.6**	**0.2**	**2.6**	**11.4**

The capital cost estimate has been prepared in accordance with an AACE Class 5 estimate, consistent with the level of definition available at the PEA stage. The expected accuracy range for a Class 5 estimate is -35% to +50%, reflecting the conceptual nature of the design, limited site investigation data, and reliance on benchmarked unit rates.

21.1.5 Project Capital Cost Estimate

The AACE Class 5 estimate of total project capital cost is presented in Table 21-14. Based on the weighted accuracy of each major scope item, the overall estimate accuracy is ±43%.

Table 21-14: Project capital cost estimate at base date of Q2 2026.

Item	Direct and Indirect Costs (USD million)	Owner's Costs (USD million)	Contingency (USD million)	Total Capital Cost (USD million)	Accuracy
Mining	80.5	4.0	25.3	109.9	± 50%
Processing Plant [1]	61.5	1.3	18.9	81.7	± 35%
Site Infrastructure	17.7	0.2	5.4	23.3	± 35%
IWF	8.6	0.2	2.6	11.4	-35% +50%
TOTAL	**168.3**	**5.8**	**52.2**	**226.3**	**± 43%**

Notes:

1. *Processing plant includes late change cost of using imported quarry material for construction.*

21.2 Operating Cost Estimate

21.2.1 Mining Scope

The mining cost breakdown is summarised in Table 21-15. These costs were estimated using a similar contractor structure and mine size as the basis of estimate.

Table 21-15: Mining cost breakdown.

Item	Amount	UoM
Operating Development Cost	71.0	USD/t
Haulage Cost (TKM Based)	1.2	USD/t
RF Backfill Cost	8.0	USD/t
CRF Backfill Cost	20.0	USD/t
DST Backfill Cost	5.0	USD/t
Cement Cost	1.3	USD/t
G&A	7.0	USD/t
Contractor Overhead	16.0	USD/t
Treatment of Mine Dewatering	1.0	USD/t
Total Mining Cost	**130.5**	**USD/t**

21.2.2 Processing Plant and Site Infrastructure

21.2.2.1 Introduction

The 0.25 Mtpa processing plant operating costs (operating expenditure; OPEX) were calculated based on material costs, unit costs supplied by suppliers, data from existing operations or developed from various external sources, and test work data available at the time of writing. They include the following.

- test work data from the historical test work programs;
- typical labour structure and labour complement, annual unit labour rates, and on-costs as agreed with RUA;
- unit cost for reagents and consumables provided by international suppliers and delivered to Auld Creek;
- unit cost for services provided by local suppliers and delivered to Auld Creek; and
- PBG's database of similar-scale projects.

The processing plant operating cost estimate includes all areas, from the ROM crushing circuit through to the filtration of tailings and discharge to in-plant stockpiles (ready for haulage to the IWF). Most of the process operating cost is associated with the following cost centres:

- Labour
- Power
- Reagents
- Operating consumables (including mobile equipment)
- Media
- Maintenance
- Laboratory.

The overall accuracy of the OPEX estimate is ±25% and no accuracy limits or contingency elements are applied. All reported costs are in USD, with a base date of April 2026. Exclusions to this operating cost estimate are as follows:

- All operating costs associated with mining (to be provided separately by Mining One) and IWF (to be provided separately by KPPL)
- Sunk costs and ongoing project costs (business interruption insurance, project finance costs, political risk insurance, withholding tax)
- General & Administration (G&A) cost
- Exploration, mining, waste dump management, haul/access road maintenance, and ongoing mine geology
- Tailings haulage from the processing plant to IWFs
- Off-site costs associated with transport, insurance and refining
- Local community engagement initiatives, donations, and subsidies
- Rehabilitation or closure costs
- Government monitoring and compliance programs
- Replacement capital
- Taxes, royalties or duties
- Dividend payments
- Escalation or exchange rate variations
- Contingency.

Consumable costs quoted in alternate currencies, including Australian dollars (AUD), New Zealand dollars (NZD), and Euro (EUR), were all converted to the USD base case currency through the application of the projected exchange rates.

21.2.2.2 Basis of Estimate

The operating cost estimate was developed from several sources. Cost determinations have been based on fixed and variable components relating to ore throughput and ore characteristics. The sources of data for the basis of estimate are detailed in Table 21-16.

Table 21-16: Operating cost basis of estimate.

Cost Centre	Source Data
Power	Consumption from MEL load estimate and power cost unit rates (NZD 0.17 per kWh) as advised by RUA.
Labour	Manning schedules based on labour list. Labour structure and on-cost rates from benchmarking of similar-scale operations and advice from RUA.
Reagents	Consumptions from historical test work and unit prices from international suppliers or available from benchmarking. Transport cost allowance of 2% of reagent supply cost applied.
Consumables	Consumptions predicted from comminution test work and unit prices from global suppliers.
Maintenance	Estimated as a factored percentage of the capital cost estimate.
Laboratory	Estimated using unit costs for similar-scale operations.
Mobile Vehicles	Estimated using annual usage and unit costs for similar-scale operations.

21.2.2.3 Operating Cost

The processing plant operating costs were determined for five discrete cost centres and were compiled from a variety of sources. Operating costs were developed using the ore-specific plant parameters and resulting mass balance that are presented in the process design criteria (document 26001-PR-PDC-0001). The overall process operating costs are summarised in Table 21-17.

Table 21-17: Operating cost summary (with rounding applied).

Cost Centre	Annual Operating Cost (USD)	Operating Cost in USD Per Tonne Dry ROM Ore (Dry Basis)	Cost Centre as % of Total Operating Cost
Labour	1,940,000	7.77	24%
Power	3,350,000	13.38	42%
Reagents and Consumables	1,570,000	6.29	20%
Maintenance and Mobile Equipment	960,000	3.84	12%
Laboratory	160,000	0.66	2%
TOTAL	**7,980,000**	**31.9**	**100%**

The total operating cost of the 0.25 Mtpa processing plant is **USD 31.9 per tonne** of ROM ore (dry basis).

21.2.3 Integrated Waste Facility

Operating costs were estimated for the IWF-2 operating period. Costs reflect routine filtered tailings placement, compaction, environmental management, equipment operation, and labour.

For IWF-2, operating costs total USD 7.25 million over 6 years (or average of USD 1.2 million per year, which is equivalent to USD 4.83 per tonne dry ROM ore). Major components include tailings haulage over 400 m, dozer spreading and compaction of 520,000 m³ of tailings, liner maintenance over 57,000 m², labour for three operators and one supervisor, environmental control, equipment maintenance, fuel, consumables, and consultancy support.

Operating costs for the IWF-4 were not included in the cost estimate for the PEA as they relate to expansion beyond the currently identified resources.

Operating cost assumptions are consistent with the capital cost basis described above, including the Q2 2026 base date, no escalation, and reliance on benchmarked unit rates appropriate for an AACE Class 5 estimate.

21.2.4 Project Operating Cost Estimate

The AACE Class 5 estimate of total project operating cost is presented in Table 21-18.

Table 21-18: Project operating cost estimate at base date of Q2 2026.

Area	Annual Cost (USD million)	Cost Per Tonne ROM Ore (Dry Basis) (USD)	% of Total Operating Cost
Mining	32.6	130.5	78%
Processing Plant and Site Infrastructure	8.0	31.9	19%
IWF	1.2	4.8	3%
TOTAL	**41.8**	**167.3**	**100%**

22 Economic Analysis

Cautionary Statement: the economic analysis and associated financial modelling presented in this PEA are preliminary in nature and are based on conceptual mine designs and production schedules, assumed capital and operating costs, forecast metal prices, and other modifying factors that have not been demonstrated at a feasibility level. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA will be realised.

Economic modelling and analysis were undertaken as part of the PEA to evaluate the Project viability based on a 5% discount rate. All values are in USD unless stated otherwise, and key model assumptions are included in Section 21. Commodity payability was estimated by Mining One at the start of the study, based on an internal database of previous concept-level studies. The QP (Gary Davison) anticipates that in the next stage of study (PFS), payability estimates will be refined as understanding of concentrate composition increases. Tax rate and royalties payable were provided by RUA and then benchmarked and agreed upon by Mining One.

Capital costs were split into pre-production (Figure 22-1) and post-production (Figure 22-2) costs. Operating costs are summarised in Figure 22-3.

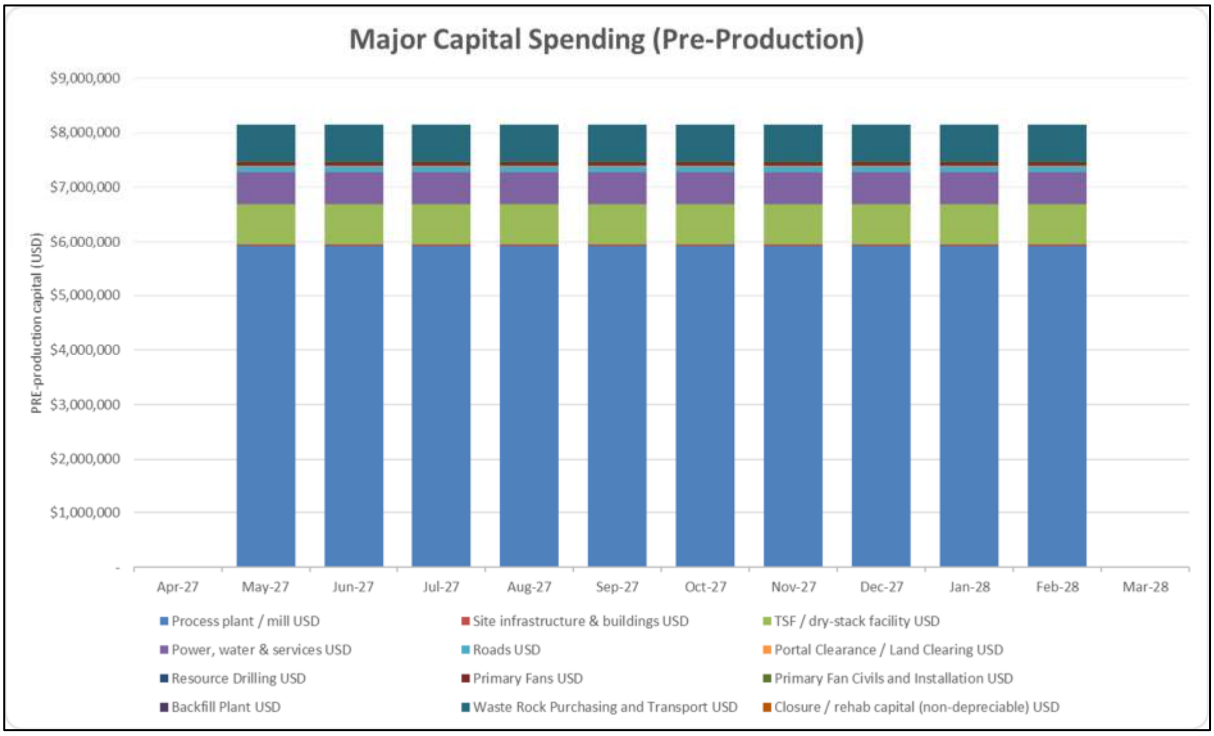


Figure 22-1: Breakdown of pre-production capital costs.

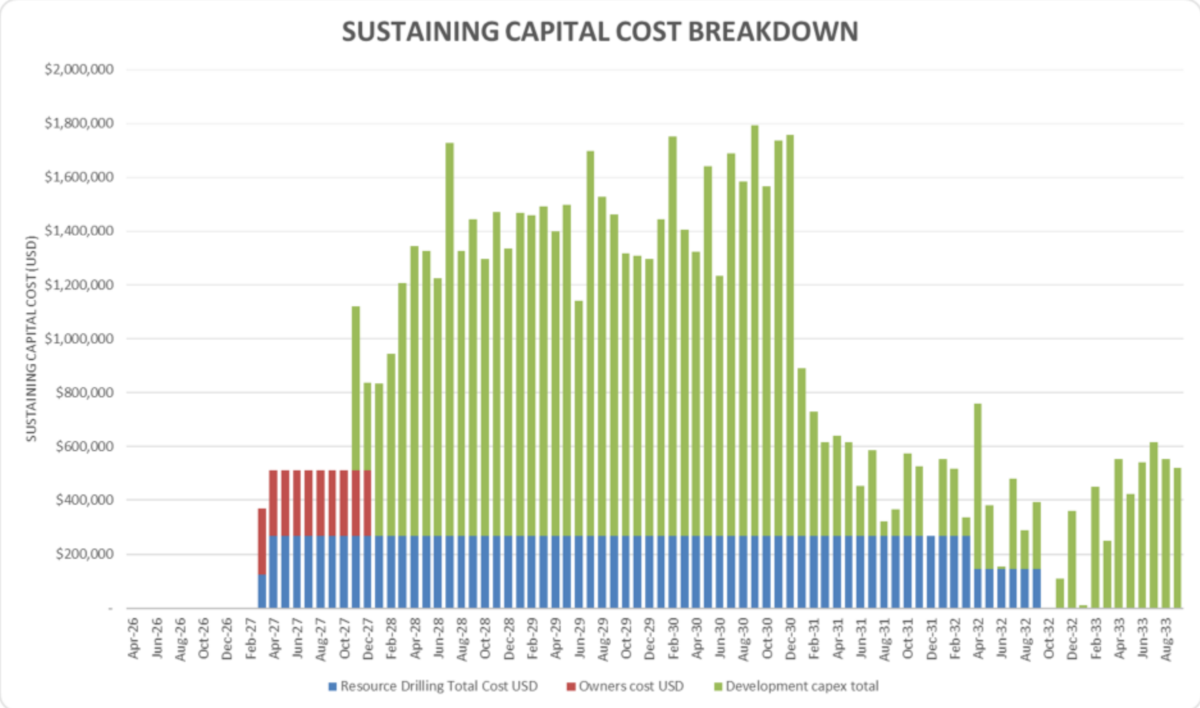


Figure 22-2: Breakdown of post-production (sustaining) capital costs.

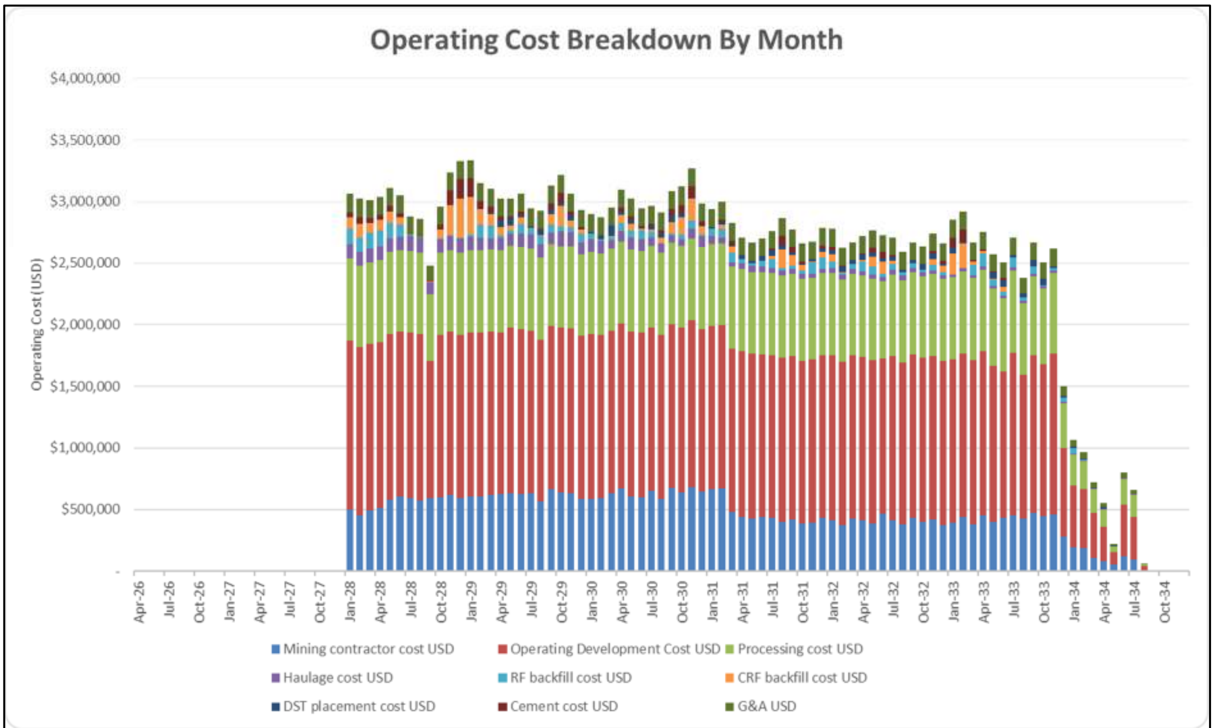


Figure 22-3: Breakdown of operating costs.

22.1 Summary of Results

Highlights of the economic model are included in Table 22-1. Figure 22-4 presents the cashflow analysis for the LOM.

Table 22-1: Economic analysis highlights base case (USD).

Economic Model KPI	Amount	UoM
Gold Price	3,300	USD/oz
Antimony Price	27,000	USD/t
Discount Rate	5.0	%
Undiscounted Cashflow	~70	USD million
Discounted Cashflow (NPV)	~42	USD million
Internal Rate of Return (IRR) [1]	~17	%
Payback Period	~40	Months
Total Operating Cost	~144	USD/t
Pre-Production Capital	~133	USD million
Post-Production Capital	~64	USD million
Total Capital Cost	~197	USD million
Net Revenue After Royalty	~486	USD million
Max Drawdown	~(135)	USD million
Operating Efficiency Ratio	0.22	%
Recovered Au	~84.5	koz
Recovered Sb	~9.0	kt
LOM Grade	3.36	g/t AuEq
Calculated COG [2]	1.79	g/t AuEq
Cash Cost	~1,399	USD/oz mined
AISC	~1,835	USD/oz mined

Notes:
1. *The IRR was annualised from monthly project cashflows using compound conversion.*
2. *The calculated COG refers to the COG back-analysed from the economic model using realised Au price and more precisely calculated costs (than compared to the first-pass COG calculation). The resultant COG is 1.77 g/t AuEq, slightly higher than the PEA COG of 1.5 g/t AuEq. This COG could be used at the PFS level.*

Auld Creek plays a pivotal role in a broader series of prospective mines in the Reefton area. The Project is robust enough to wear the capital expenditure of the mill, which has the upside of being used to process material from multiple sources. The QP (Gary Davison) suggests that the economic benefits of the Project are not fully realised by this stand-alone study, and that the broader implications to RUA's assets are important to consider.

Annual cashflow is summarised in Table 22-2 and a graphical depiction of discounted cashflow and free cashflow is presented in Figure 22-4.

Table 22-2: Annual cashflow summary from economic model.

Line Item	UoM	FY27	FY28	FY29	FY30	FY31	FY32	FY33	FY34
Mined Ore	t	-	124,788	246,273	249,905	249,455	249,878	217,986	18,731
Recovered Au	oz	-	7,819	17,657	17,515	15,089	13,360	11,886	1,157
Recovered Stibnite	t	-	617	1,764	1,845	1,707	1,590	1,337	134
Recovered AuEq Ounces	oz AuEq	-	12,084	29,849	30,269	26,891	24,350	21,129	2,087

Line Item	UoM	FY27	FY28	FY29	FY30	FY31	FY32	FY33	FY34
Gold Equivalent Grade AuEq g/t	g/t AuEq	-	3.0	3.8	3.8	3.4	3.0	3.0	3.5
Revenue After Royalty	US$M	-	39.5	98.6	100.2	89.3	81.0	70.1	6.9
Total Operating Cost	US$M	-	18.9	37.6	37.4	35.7	33.5	29.7	2.4
EBITDA	US$M	-	20.6	61.0	62.8	53.5	47.6	40.4	4.5
Pre-Production Capital	US$M	25.4	107.2	-	-	-	-	-	-
Post-Production Capital	US$M	-	1.4	17.8	18.5	13.2	5.4	5.4	2.4
AISC	US$M	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Capital	US$M	25.4	108.6	17.8	18.5	13.2	5.4	5.4	2.4
Free Cashflow	US$M	-25.4	-88.0	37.8	35.3	35.1	39.4	34.1	2.0
Discounted Cashflow	US$M	-25.4	-86.5	34.9	31.1	29.4	31.4	25.9	1.5

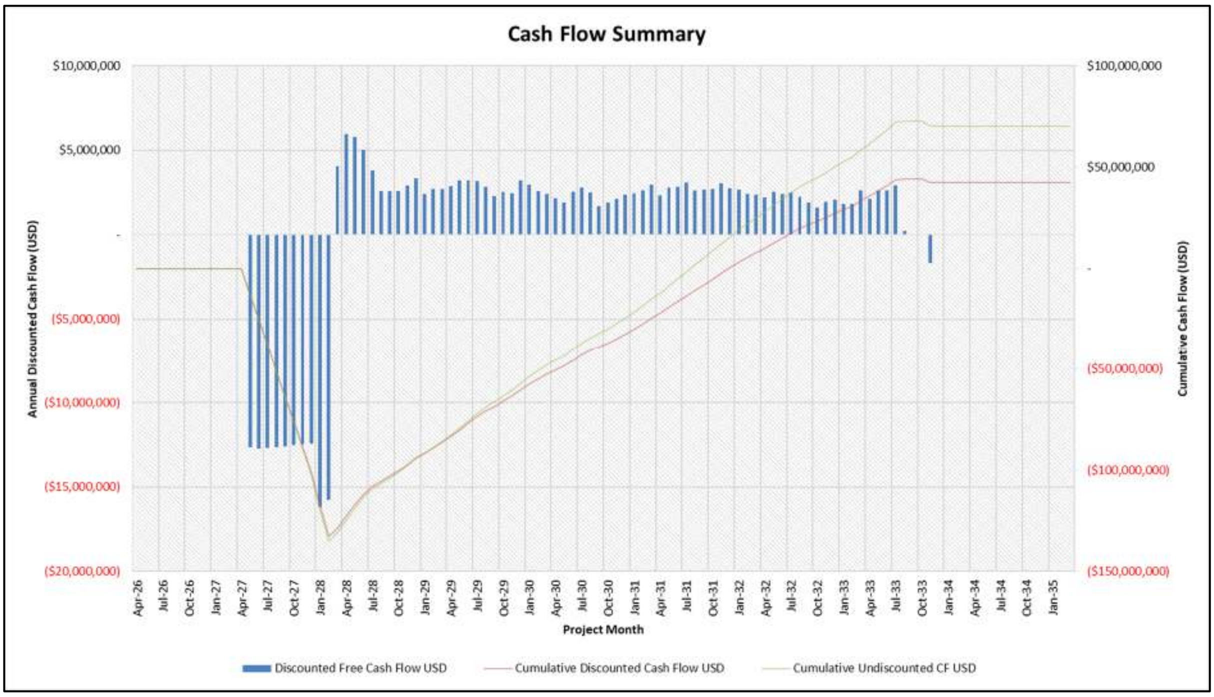


Figure 22-4: Cashflow summary.

22.2 Sensitivity

The QP (Gary Davison) has undertaken sensitivity analysis on the principal drivers of value for the Project (Figure 22-5) to understand the Project's resilience to increasing costs and fluctuating market conditions.

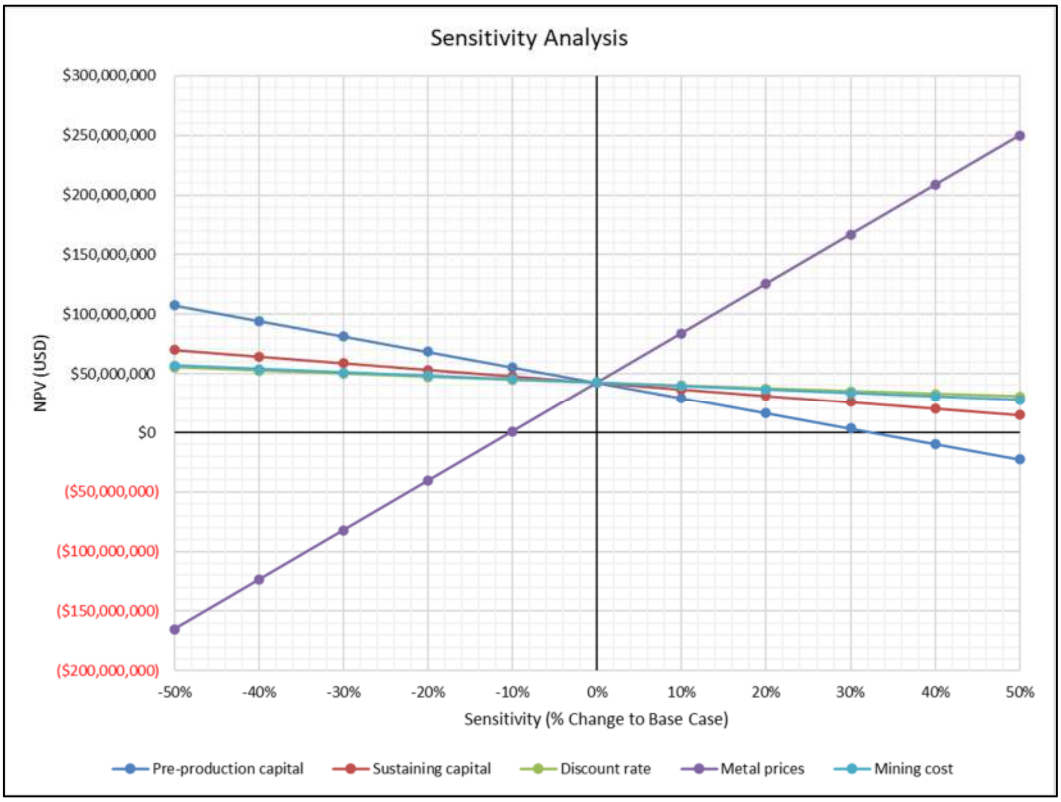


Figure 22-5: Sensitivity analysis – base case.

For revenue, the Project breaks even when the Au price drops by 10%, which equates to a long-term price of USD 2,970/oz. At the time of writing, the Au price was USD 5,170/oz, meaning that the Project has an element of conservatism built in by metal price.

For costs, the Project is most sensitive to capital costs. Capital costs would need to increase by ~30% for the Project to be break-even. The QP (Gary Davison) notes that the Project is not especially sensitive to discount factor given the short mine life, but this is subject to change with the expected resource expansion.

The QP (Gary Davison) considers that, at the PEA stage, the Project is economically viable, with a return on investment albeit sub-optimal. However, the Project has substantial potential. The QP (Gary Davison) suggests that upside exists in the exploration of the deposit to create a more centralised (and hence less development-intensive) mine design.

22.3 Opportunities to Increase Value and Offset Waste Deficit

Mining One has provided economic models for the following cases in addition to the base case:

- Base case (Scenario 1, discussed in previous section).
- Scenario 2: waste fill will not be used underground (except for the early mining pre-processing plant commissioning where tailings are not yet available as a backfill medium). Backfill will completed almost entirely using dry-stacked tailings and cemented tailings.
- Scenario 3: paste plant will be used for backfill.
- Scenario 4: higher (2.5 g/t AuEq) COG used.

Scenarios 2, 3 and 4 aim to demonstrate Project potential and provide some alternatives to offset the waste deficit. All scenarios use OHCAF as the mining method.

22.3.1 Scenario 2: Dry-Stacked and Cemented Tailings Fill

Scenario 2 addresses a potential solution to the waste deficit by filling predominantly with dry-stacked tails (DST) and cemented dry-stacked tails (CDST) underground. This approach will involve geotechnical numerical modelling and test work to test the feasibility of this mining method.

This scenario has the added benefit of reducing the TSF required from 1.0 Mt (base case) to ~400 Mt. The production profile for this scenario is depicted in Figure 22-6 and the new economic outcomes are summarised in Table 22-3. This scenario potentially allows as much as 500 kt of waste material to be used on surface.

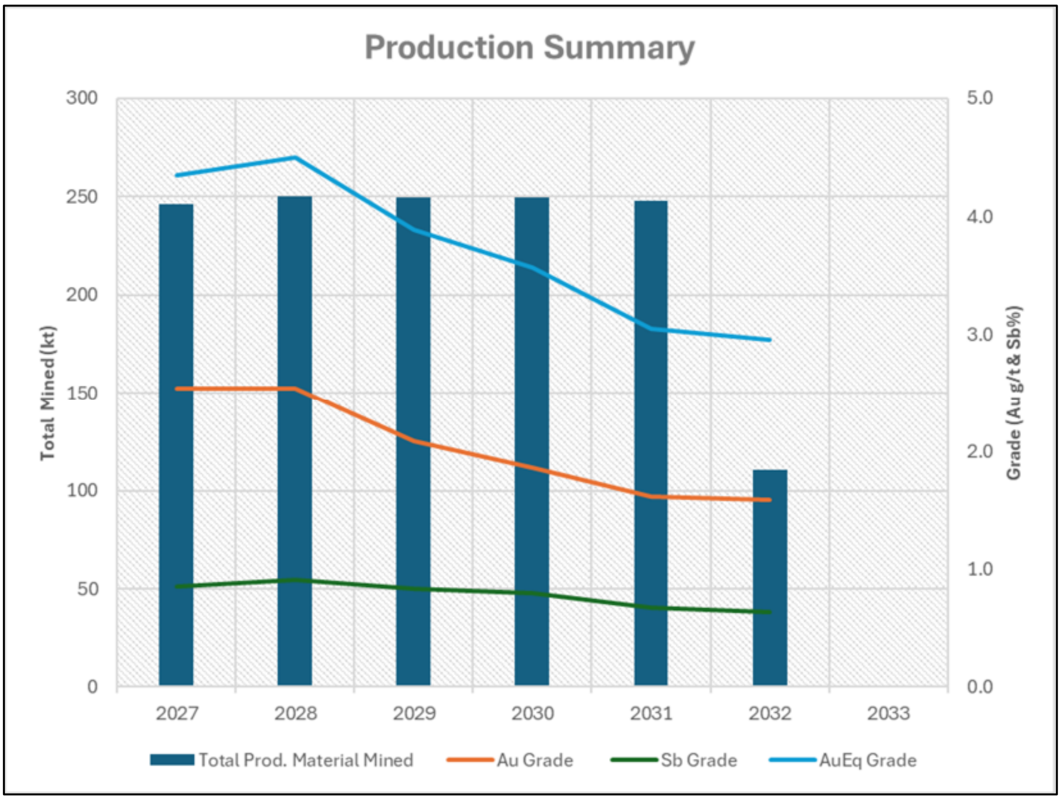


Figure 22-6: Production summary – Scenario 2: dry-stacked tailings.

Table 22-3: Economic summary – Scenario 2: dry-stacked tailings.

Economic model KPI	Amount	UoM
Gold Price	3,300	USD/oz
Antimony Price	27,000	USD/t
Discount Rate	5.0	%
Undiscounted Cashflow	~68	USD million
Discounted Cashflow (NPV)	~40.4	USD million
Internal Rate of Return (IRR)	~16	%
Payback Period	~41	Months
Total Operating Cost	~144	USD/t
Pre-Production Capital	~133	USD million
Post-Production Capital	~66	USD million
Total Capital Cost	~199	USD million
Net Revenue After Royalty	~484	USD million
Max Drawdown	~(135)	USD million
Operating Efficiency Ratio	~0.21	%

Economic model KPI	Amount	UoM
Recovered Au	84.2	koz
Recovered Sb	9.0	kt
LOM Grade	3.35	g/t AuEq
Calculated COG	1.81	g/t AuEq
Cash Cost	1,401	USD/oz mined
AISC	1,854	USD/oz mined

22.3.2 Sensitivity Analysis Scenario 2: Dry-Stacked Tailings

The QP (Gary Davison) conducted sensitivity analysis for this scenario to demonstrate the range of potential Project value, varied by key cost drivers (Figure 22-7).

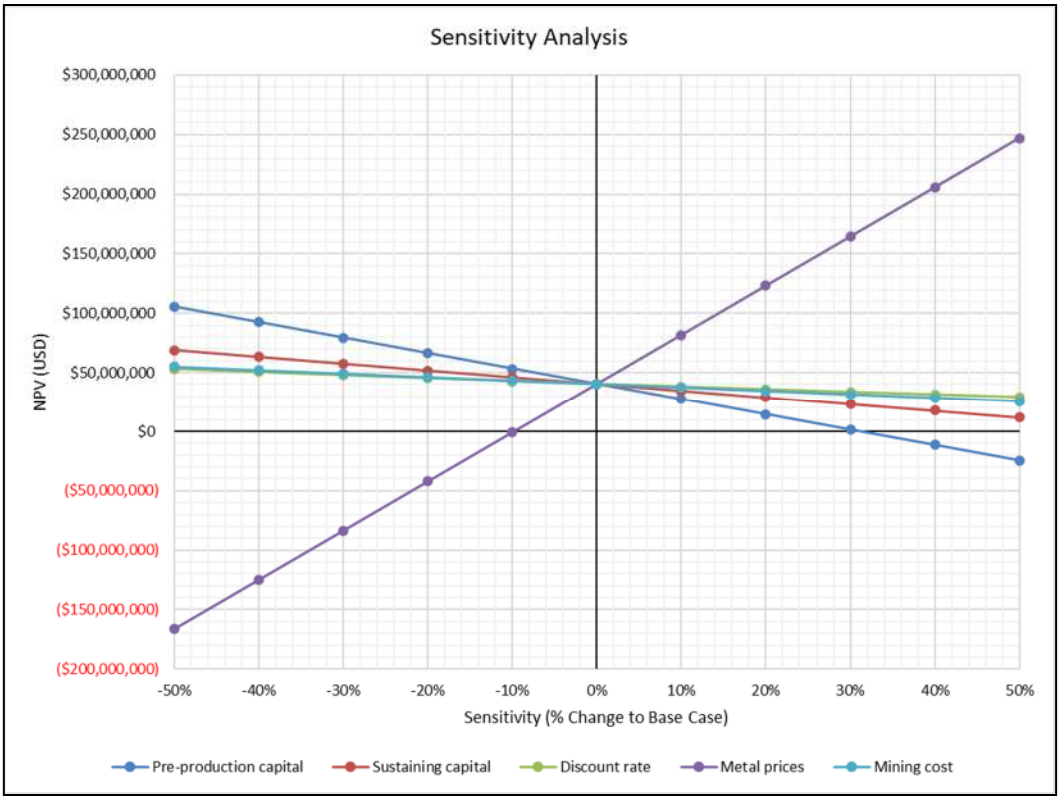


Figure 22-7: Sensitivity analysis – Scenario 2: dry-stacked tails backfill.

22.3.3 Scenario 3: Paste Plant

The QP (Gary Davison) implemented a paste plant into the schedule and economic model. The paste plant capital cost and operating cost estimate are presented in Table 22-4. The capital cost estimate was provided by PBG as part of its work on the PEA, and the operational cost was benchmarked by the QP (Gary Davison) based on previous studies.

A paste plant is a well-tested option for reducing the waste deficit issue. The QP (Gary Davison) considers that it is likely a paste plant will be required at some point in the Project life. However, further trade-off studies and test work are required at the PFS level of study to determine the best option for Auld Creek. The QP (Gary Davison) notes that a paste plant, while effective in providing tailings backfill underground, presents major challenges to the Project, including:

- the requirement to pump paste uphill to the top of the mine;
- lead time on backfill availability (waste rock would need to be used as backfill for early production, while the processing plant and paste plant are being commissioned); and

- the substantial capital cost of the plant itself.

The paste plant was added to the schedule assuming a paste placement rate of 1,000 m^3/day. This is a conservative estimate compared to industry best practice paste placement rates. The QP (Gary Davison) has been conservative to build in allowance for paste cure and wall building, which have not been explicitly scheduled.

Table 22-4: Paste plant cost estimates.

Item	Cost Estimate	UoM
Paste Plant CAPEX	11.7	USD million
Paste Plant Operating Cost	15.0	USD/t paste placed

The production profile for this scenario is depicted in Figure 22-8 and the new economic outcomes are summarised in Table 22-5.

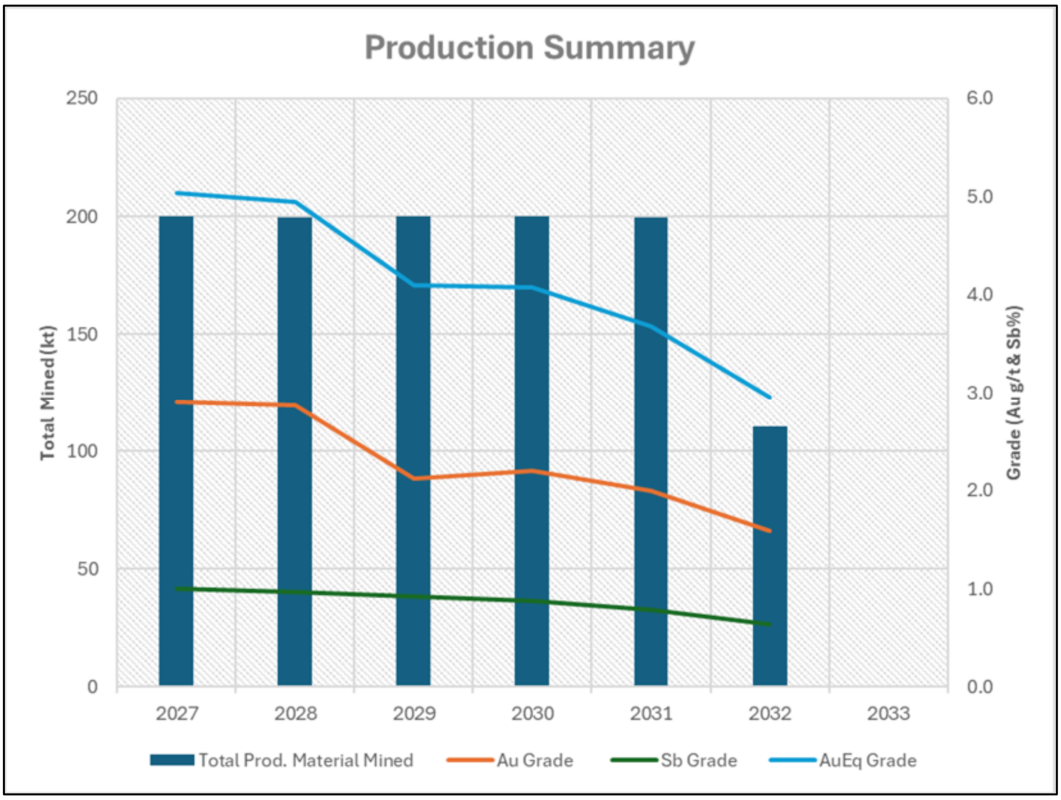


Figure 22-8: Production summary – Scenario 3: pastefill.

Table 22-5: Economic summary – Scenario 3: pastefill.

Economic model KPI	Amount	UoM
Gold Price	3,300	USD/oz
Antimony Price	27,000	USD/t
Discount Rate	5.0	%
Undiscounted Cashflow	54	USD million
Discounted Cashflow (NPV)	25.6	USD million
Internal Rate of Return (IRR)	11	%
Payback Period	47	Months

Economic model KPI	Amount	UoM
Total Operating Cost	153	USD/t
Pre-Production Capital	148	USD million
Post-Production Capital	67	USD million
Total Capital Cost	215	USD million
Net Revenue After Royalty	496	USD million
Max Drawdown	(152)	USD million
Operating Efficiency Ratio	0.12	%
Recovered Au	85.4	koz
Recovered Sb	9.3	kt
LOM Grade	3.41	g/t AuEq
Calculated COG	1.89	g/t AuEq
Cash Cost	1,461	USD/oz mined
AISC	1,907	USD/oz mined

22.3.4 Sensitivity Analysis Scenario 3: Pastefill

The QP (Gary Davison) conducted sensitivity analysis for this scenario to demonstrate the range of potential project value, varied by key cost drivers (Figure 22-9).

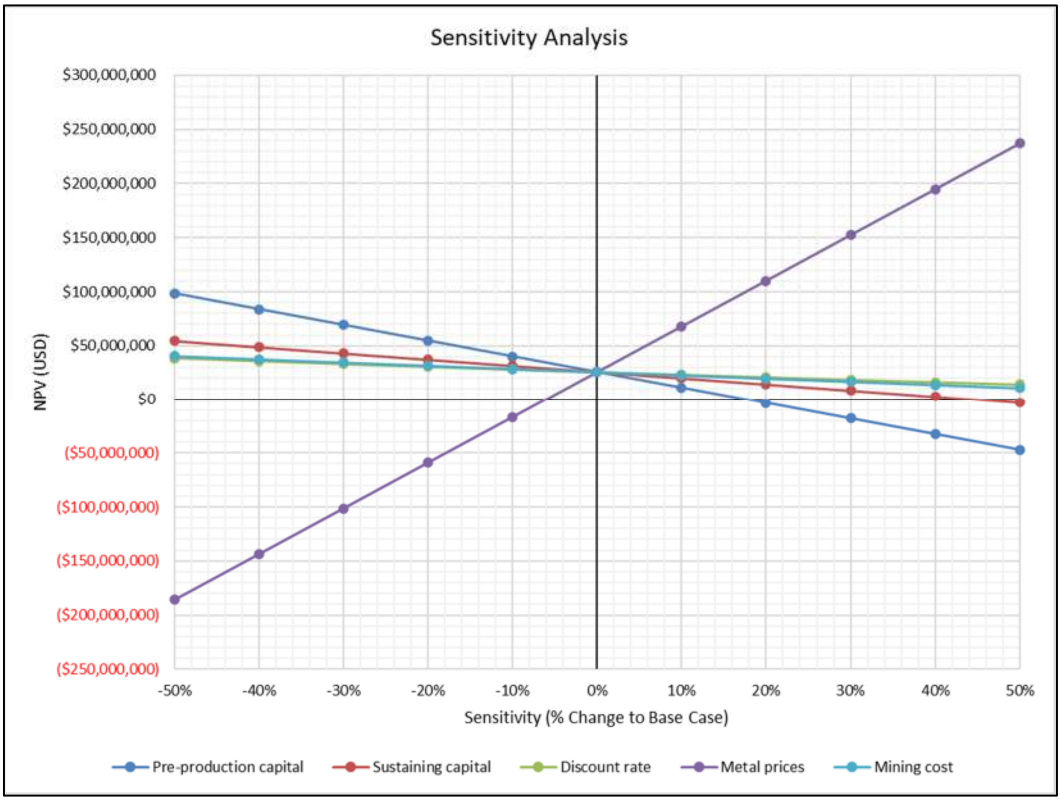


Figure 22-9: Sensitivity analysis – Scenario 3: pastefill.

22.3.5 Scenario 4: Higher COG

The QP (Gary Davison) compiled a schedule using COG = 2.5 g/t AuEq (compared to the study COG = 1.5 g/t AuEq). There is strong justification for increasing the COG, as all the economic models resulted in a calculated COG above the study COG.

The production profile in this case is reduced when compared to the base case (Figure 22-10) due to having less resource to mine. The economic summary in Table 22-6 suggests that this is a good scenario in terms of NPV; however, the QP (Gary Davison) suggests that this design will not encapsulate the full potential of the deposit.

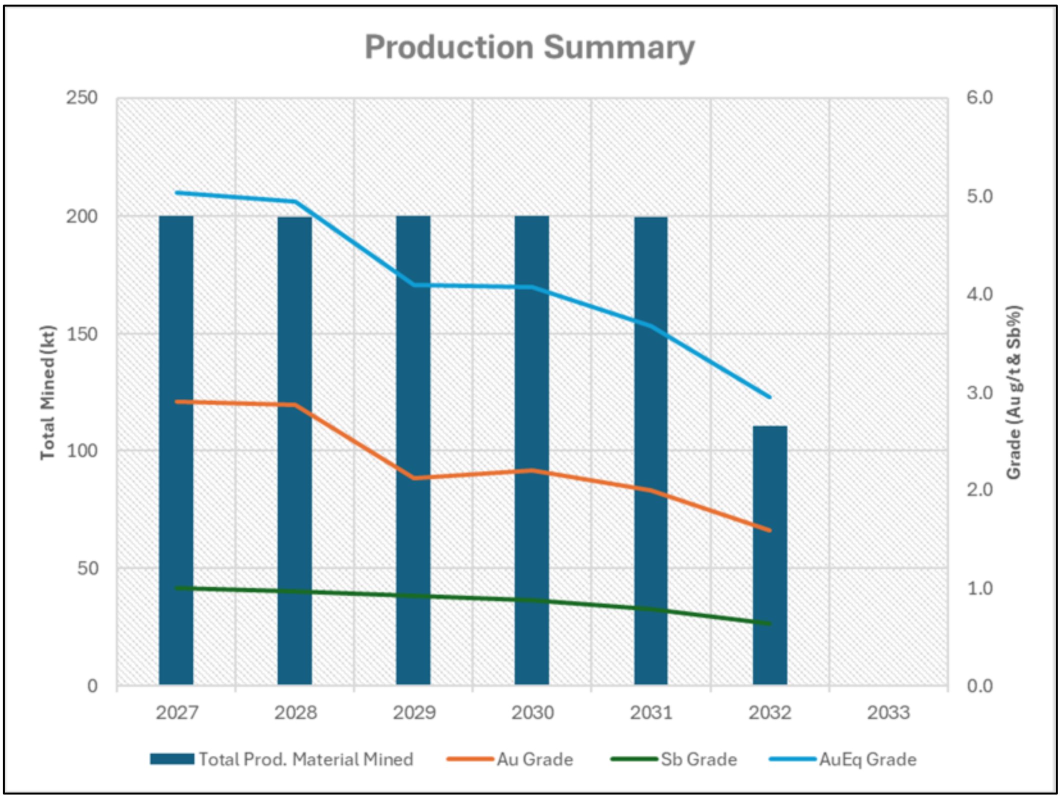


Figure 22-10: Production summary – Scenario 4: COG = 2.5 g/t AuEq.

Table 22-6: Economic summary – Scenario 4: COG = 2.5 g/t AuEq.

Economic Model KPI	Amount	UoM
Gold Price	3,300	USD/oz
Antimony Price	27,000	USD/t
Discount Rate	5.0%	%
Undiscounted Cashflow	54	USD million
Discounted Cashflow (NPV)	31.6	USD million
Internal Rate of Return (IRR)	15	%
Payback Period	41	Months
Total Operating Cost	147	USD/t
Pre-Production Capital	133	USD million
Post-Production Capital	68	USD million
Total Capital Cost	201	USD million
Net Revenue After Royalty	432	USD million
Max Drawdown	(134)	USD million
Operating Efficiency Ratio	0.20	%
Recovered Au	74.4	koz
Recovered Sb	8.1	kt
LOM Grade	3.80	g/t AuEq

Economic Model KPI	Amount	UoM
Calculated COG	1.98	g/t AuEq
Cash Cost	1,270	USD/oz mined
AISC	1,794	USD/oz mined

22.3.6 Sensitivity Analysis Scenario 4: COG = 2.5 g/t AuEq

The QP (Gary Davison) conducted sensitivity for this scenario to demonstrate the range of potential project value, varied by key cost drivers (Figure 22-11).

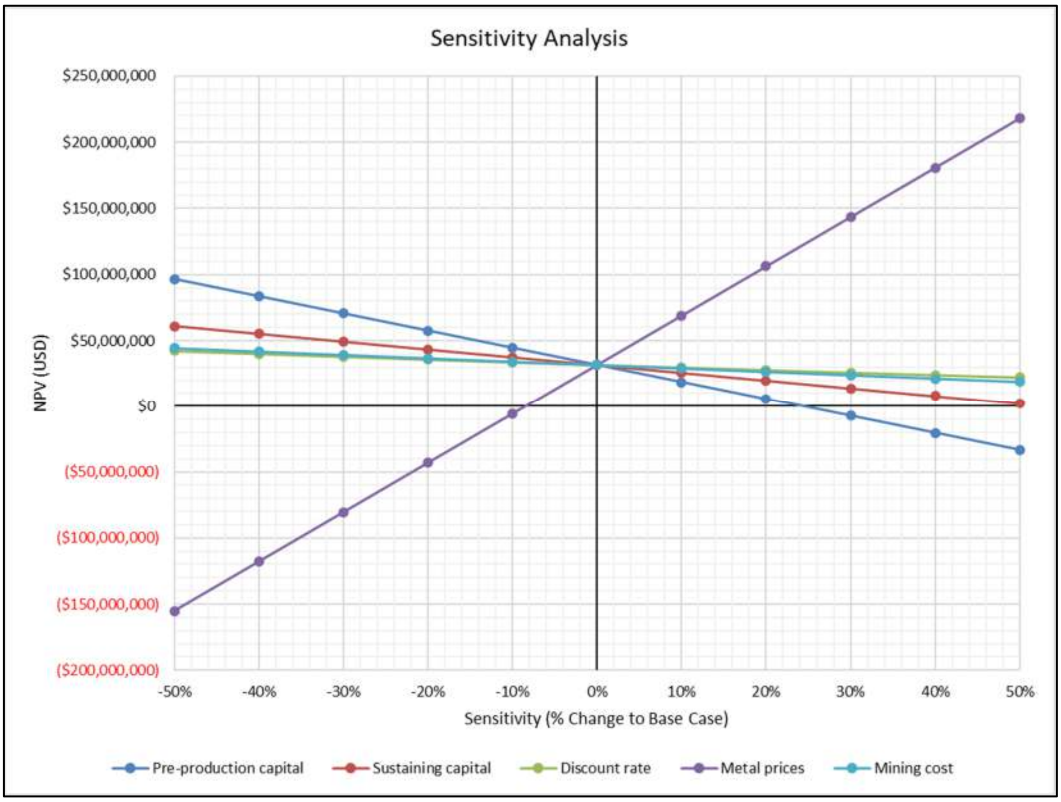


Figure 22-11: Sensitivity analysis – Scenario 4: COG = 2.5 g/t AuEq.

22.4 Summary of Scenarios

Table 22-7 summarises the NPV of each scenario as the key schedule outcomes.

Table 22-7: Summary of schedule scenarios for OHCAF.

Scenario	1 – Base Case	2 – DST / CDST	3 – Paste Plant	4 – COG 2.5 g/t	UoM
Production tonnes	1,350	1,350	1,350	1,100	kt
Recovered metal	85	85	85	74	Au koz
Recovered metal	9	9	9	8	Sb kt
Production rate	250	250	250	200	ktpa
Mine life	5.5	5.5	5.5	5.5	Years
Pre-Production Capital Cost	~133	~133	~148	~133	USD million
Sustaining Capital Cost	~64	~66	~67	~68	USD million
Operating Cost	~144	~144	~153	~147	USD/t
Project $NPV_{7\%}$	~42	~40	~26	~32	USD million

23 Adjacent Properties

There are numerous active permits adjacent to the Project area (Figure 23-1). EP 60648, in which the Project is located, forms part of RUA's wider Reefton Project. In addition to RUA's Reefton Project, there are two significant hard-rock Au properties in the Buller region: Endura Mining's Snowy River Project and the Reefton Restoration Project, which involves rehabilitation of the Globe-Progress Mine.

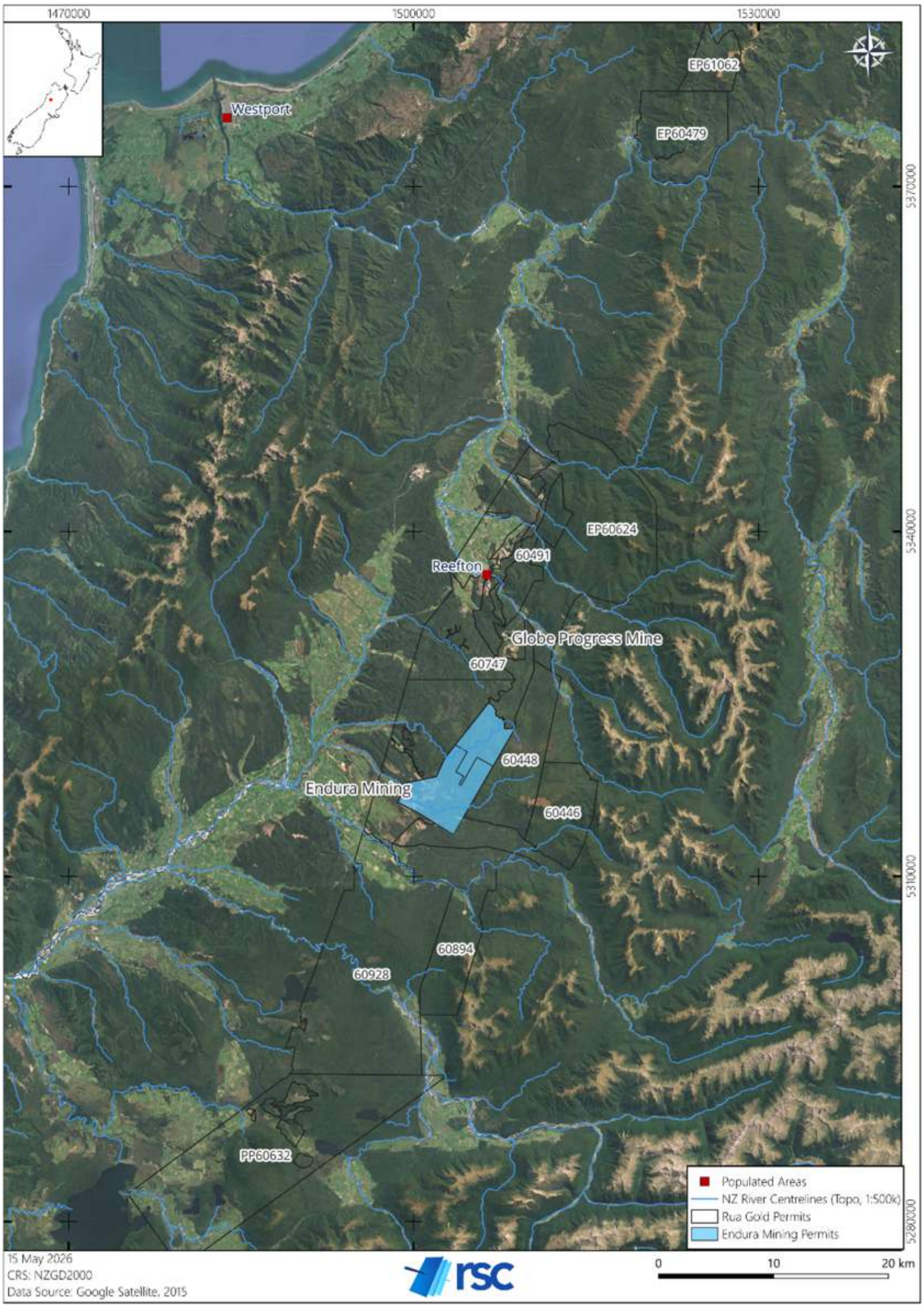


Figure 23-1: Significant adjacent properties in the Reefton area.

While, the QP (Abraham Whaanga) has verified the scientific and technical information related to RUA's adjacent properties (Reefton Project) discussed in Section 23.1, this information is not necessarily indicative of the mineralisation potential at the Project. The QP has not verified the scientific and technical information related to the other adjacent properties, and this is information is not necessarily indicative of the mineralisation at the Project.

23.1 RUA Reefton Project

The Reefton Project is located within a historically significant orogenic Au district that has produced ~2 Moz Au from high-grade underground operations. RUA holds permits in the district over an area of ~1,250 km^2, encompassing several historical mining camps, including Murray Creek, Crushington, Capleston, and Caledonian.

The Reefton Project permits are in various stages of exploration and development. In addition to the current MRE for the Auld Creek Gold-Antimony Project, RUA has previously reported MREs for the Alexander River, Big River, and Supreme deposits (Whaanga, 2026).

23.1.1 Alexander River

The Alexander River deposit is located within the Alexander River permit (EP 60446), which is located ~15 km southeast of the Auld Creek Gold-Antimony Project area. RUA previously reported an Inferred Mineral Resource of 156 koz Au at a cut-off of 1.5 g/t Au (Table 23-1) (Whaanga, 2026).

Table 23-1: Mineral Resources reported at the Alexander River deposit (Whaanga, 2026).

Domain	Classification	Tonnes (Mt)	Au (g/t)	Contained Au Ounces (koz)
LG McVicar West	Inferred	0.51	3.4	56
HG McVicar West	Inferred	0.20	4.1	27
LG Bull East	Inferred	0.26	1.6	13
HG Bull East	Inferred	0.06	3.7	7
Bruno 1	Inferred	0.05	5.5	9
Bruno 2	Inferred	0.01	5.9	2
Loftus-McKay	Inferred	0.20	5.5	35
McVicar East	Inferred	0.06	3.8	8
Total	**Inferred**	**1.35**	**3.6**	**156**

Notes:

1. *The definitions for Mineral Resources of the Canadian Institute of Mining, Metallurgy and Petroleum were followed.*
2. *The Mineral Resource was reported at a cut-off grade of 1.5 g/t Au.*
3. *The Mineral Resource was assessed for reasonable prospects for eventual economic extraction by re-blocking to a regular 2 mW × 4 mH × 4 mL minimum block dimension, converting to wireframe solids, and generating minimum mining units, commensurate with the anticipated minimum mining unit dimensions for a long-hole stoping operation.*
4. *Totals may vary due to rounding.*

Disclosure of Historical Estimates on Adjacent Properties

- The February 2026 historical estimate was reported in accordance with the CIM Definition Standards (May 2014) and included in a technical report compliant with NI 43-101, with an effective date of 27 February 2026, and filed on RUA's SEDAR+ profile (Whaanga, 2026).
- The February 2026 Alexander River historical estimate is considered reliable and relevant by the QP (Abraham Whaanga), as it was an updated resource estimate.
- In addition to the key assumptions, parameters, and methods described above, the February 2026 Alexander River historical estimate was reported at a cut-off grade of 1.5 g/t Au.
- The QP (Abraham Whaanga) has not done sufficient work to classify the February 2026 Alexander River historical estimate as current mineral resources, and RUA is not treating this

historical estimate as current mineral resources. The purpose of stating this historical estimate in the Report is to fully disclose nearby historical estimates.

- The QP (Abraham Whaanga) is not aware of any other recent historical estimates for the Alexander River prospect.

23.1.2 Big River

The Big River deposit is located within the Big River permit (EP 60448), which is located ~8 km south of the Auld Creek Gold-Antimony Project area. RUA previously reported an Inferred Mineral Resource of 101 koz Au at a cut-off of 1.5 g/t Au (Table 23-2) (Whaanga, 2026).

Table 23-2: Mineral Resources reported at the Big River deposit (Whaanga, 2026).

Domain	Classification	Tonnes (Mt)	Au (g/t)	Contained Au Ounces (koz)
Shoot 4 Upper	Inferred	0.25	3.41	28
Shoot 4 Lower	Inferred	0.56	2.98	53
Shoot A2	Inferred	0.34	1.79	20
Total	**Inferred**	**1.16**	**2.72**	**101**

Notes:

1. *The definitions for Mineral Resources of the Canadian Institute of Mining, Metallurgy and Petroleum were followed.*
2. *The Mineral Resource is reported at a cut-off grade of 1.5 g/t Au.*
3. *The Mineral Resource was assessed for reasonable prospects for eventual economic extraction by re-blocking to a regular 2 mW × 5 mH × 2.5 mL minimum block dimension, converting to wireframe solids, and generating minimum mining units, commensurate with the anticipated smallest mining unit dimensions for a long-hole stoping operation.*
4. *Totals may vary due to rounding.*

Disclosure of Historical Estimates on Adjacent Properties

- The February 2026 historical estimate was reported in accordance with the CIM Definition Standards (May 2014) and included in a technical report compliant with NI 43-101, with an effective date of 27 February 2026, and filed on RUA's SEDAR+ profile (Whaanga, 2026).
- The February 2026 Big River historical estimate is considered reliable and relevant by the QP (Abraham Whaanga), as it was an updated resource estimate.
- In addition to the key assumptions, parameters, and methods described above, the February 2026 Big River historical estimate was reported at a cut-off grade of 1.5 g/t Au.
- The QP (Abraham Whaanga) has not done sufficient work to classify the February 2026 Big River historical estimate as current mineral resources, and RUA is not treating this historical estimate as current mineral resources. The purpose of stating this historical estimate in the Report is to fully disclose nearby historical estimates.
- The QP (Abraham Whaanga) is not aware of any other recent historical estimates for the Big River prospect.

23.1.3 Supreme

The Supreme deposit is located within the Cumberland permit (EP 60747), which is located immediately west of EP 60648. RUA previously reported an Inferred Mineral Resource of 92 koz Au at a cut-off grade of 1.5 g/t Au (Table 23-3) (Whaanga, 2026).

Table 23-3: Mineral Resources reported at the Supreme deposit (Whaanga, 2026).

Domain	Classification	Tonnes (Mt)	Au (g/t)	Contained Au Ounces (koz)
Supreme	Inferred	1.46	1.96	92
Total	**Inferred**	**1.46**	**1.96**	**92**

Notes:

1. *The definitions for Mineral Resources of the Canadian Institute of Mining, Metallurgy and Petroleum were followed.*
2. *The Mineral Resource is reported at a cut-off grade of 1.5 g/t Au.*
3. *The Mineral Resource was assessed for reasonable prospects for eventual economic extraction by re-blocking to a regular 2.5 mW x 2.5 mH x 5 mL minimum block dimension, converting to wireframe solids, and generating minimum mining units, commensurate with the anticipated smallest mining unit dimensions for a long-hole stoping operation.*
4. *Totals may vary due to rounding.*

Disclosure of Historical Estimates on Adjacent Properties

- The February 2026 historical estimate was reported in accordance with the CIM Definition Standards (May 2014) and included in a technical report compliant with NI 43-101, with an effective date of 27 February 2026, and filed on RUA's SEDAR+ profile (Whaanga, 2026).
- The February 2026 Supreme historical estimate is considered reliable and relevant by the QP (Abraham Whaanga), as it was an updated resource estimate.
- In addition to the key assumptions, parameters, and methods described above, the February 2026 Supreme historical estimate was reported at a cut-off grade of 1.5 g/t Au.
- The QP (Abraham Whaanga) has not done sufficient work to classify the February 2026 Supreme historical estimate as current mineral resources, and RUA is not treating this historical estimate as current mineral resources. The purpose of stating this historical estimate in the Report is to fully disclose nearby historical estimates.
- The QP (Abraham Whaanga) is not aware of any other recent historical estimates for the Big River prospect.

23.2 Endura Mining: Snowy River (Blackwater) Project

In January 2024, Endura Mining (previously Federation Mining) agreed to exercise the option to buy the Snowy River Mine Project (formerly the Blackwater Mine) asset from OGL. The project is located 20 km south of Reefton (Figure 23-1), and Endura Mining's objective is to establish an underground mine at the site. Construction of two 3.3-km twin declines is complete and will provide sites for underground drilling. OGL previously reported an Inferred Mineral Resource of 700,000 oz Au, although no cut-off grade was applied (Table 23-4) (Madambi and Moore, 2013; Federation Mining, 2022). A 20-year mining permit has been granted, and the company aims to start production by the end of 2026 (Jones, 2025).

Table 23-4: Mineral Resources reported at Snowy River (Federation Mining, 2022).

Company	Project	Classification	Cut-off (g/t)	Tonnage (Mt)	Au Grade (g/t)	Au (koz)
OGL	Snowy River	Inferred	Not reported	0.9	23.0	700

Required disclosure under Section 2.4 of NI 43-101 (Disclosure of Historical Estimates)

- The 2022 Snowy River historical estimate was reported in accordance with the JORC Code (JORC Code, 2012) and included in a Competent Person's report with an effective date of 30 September 2022 (Grove and Binks, 2023).
- The 2022 Snowy River historical estimate is considered reliable and relevant by the QP (Abraham Whaanga), as it was the initial resource estimate for the Snowy River prospect.
- The 2022 Snowy River historical estimate was not reported at a cut-off grade and was geologically constrained.
- The 2022 Snowy River historical estimate uses similar categories to those set out in Section 1.2 of NI 43-101 but was classified using the JORC Code (2012), in which resource classifications are similar to the resource classifications under the CIM Definition Standards (May 2014).

- The QP (Abraham Whaanga) has not done sufficient work to classify the 2022 Snowy River historical estimate as current mineral resources, and RUA is not treating the historical estimate as current mineral resources. The purpose of stating this historical estimate in the Report is to fully disclose nearby historical estimates.
- The QP (Abraham Whaanga) is not aware of any other recent historical estimates for the Snowy River prospect.

23.3 Globe-Progress: Reefton Restoration Project

The former Globe-Progress mine is located 7 km southeast of the Reefton township, adjacent to the Cumberland and Golden Point permits (Figure 23-1). Commercial operations commenced in 2007, producing 610,000 oz Au over the 8-year life of the open-pit operation. The mine transitioned from operation to closure and rehabilitation in 2016 and is now known as the Reefton Restoration Project. Restoration has included a comprehensive closure and rehabilitation program, with works involving:

- removal of the processing plant and infrastructure;
- water treatment;
- waste rock reshaping and landscaping; and
- spreading topsoil and planting trees.

The Globe-Progress mine is the first modern large-scale Au mine in the South Island of New Zealand to transition to closure.

As of the effective date of this Report, RRPL had submitted EPA 61557.01 to NZP&M, covering the Globe-Progress Reserved Area (Figure 23-1).

24 Other Relevant Data and Information

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25 Interpretation and Conclusions

25.1 Geological and Mineralogical Scope

The Project comprises EP 60648, which is held by RRPL. RRPL is a wholly owned New Zealand subsidiary of RUA.

Previous exploration work carried out by RRPL includes stream sampling, soil sampling, mapping, geophysical surveys, trenching, 3D modelling, and diamond drilling. RRPL carried out extensive diamond drilling programmes at the Project, prior to which OGL carried out drilling across EP 60648. RRPL also completed a historical estimate for the Project and conducted initial metallurgical test work, indicating the potential for >90% Au and Sb recoveries.

Between July 2024 and February 2026, RUA conducted additional soil and rock-chip sampling across the Project area and mapped areas around Auld Creek. RUA completed a further 38 drillholes at the Project, for a total of 8,086.9 m. Drilling data from these holes were used as a basis for updating the MRE for the Project.

The QP (Abraham Whaanga) has conducted several site visits, collected validation samples, reviewed the SOPs, and independently assessed QC for diamond core sampling. Based on these reviews, the QP (Abraham Whaanga) considers the historical and recent exploration programmes, including sampling, preparation, and analytical data, to be fit for the purposes of classifying at least Indicated Mineral Resources in accordance with NI 43-101 and the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014).

The QP (Abraham Whaanga) has classified the Mineral Resource for the Project in the Inferred and Indicated categories, in accordance with NI 43-101 and the CIM as the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014). For the Indicated Mineral Resources, geological evidence is derived from adequately detailed and reliable exploration, sampling, and testing, and is sufficient to assume geological and grade or quality continuity between points of observation. For the Inferred Mineral Resources, geological evidence is sufficient to imply but not verify geological and grade continuity. The Mineral Resource is based on exploration, sampling, and assaying information gathered through appropriate techniques from trenching and drillholes. In assessing the RPEEE, the QP evaluated preliminary mining, metallurgical, and environmental parameters. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued drilling.

Uncertainties and risks related to informing data, modelling, and resource estimations are discussed in detail in Section 14.12. The QP (Abraham Whaanga) recommends making changes to the core cutting procedures to minimise the risk of introducing first-split selection bias and collecting first-split duplicates of core samples in any future resource delineation drilling programmes. CRMs were not matrix matched with the rock type and mineralisation. Future CRMs should be sourced from similar metasediments with elevated Sb and As. There is additional geological uncertainty in places, due to many drillholes intersecting the mineralisation at high angles and inconsistent data spacing.

Mining in New Zealand is a sensitive subject, and even though the West Coast region has stronger support for mining than the rest of New Zealand, the QP notes RUA must effectively monitor and address these issues in order to not lose its social licence to practice.

25.2 Mining Scope

This PEA indicates that the Auld Creek deposit has potential to support a small underground mining operation producing approximately 250 ktpa of production material. Selective mining methods are required due to the narrow-vein geometry of the mineralisation and the relatively poor rock mass conditions identified in the geotechnical assessment.

Overhand Cut-and-Fill (OHCAF) mining was selected as the preferred base-case method due to its ability to maintain selectivity and manage geotechnical risk. Conceptual mine design and scheduling indicate a potential mine life of ~6 years with annual production of ~1.4 Mt.

Capital and operating cost estimates were prepared using benchmark data and information supplied by study partners. At the PEA stage, these estimates have an expected accuracy range of ±50%.

The Project presents a modest $NPV_{5.0\%}$ = USD 42.4 million at a rate of return IRR = 17%. It is important to appreciate a broad context when evaluating the economics of this Project. The Auld Creek deposit proves revenue-positive, while also offsetting the capital cost of major infrastructure (such as the processing plant) which has the potential to be used for several prospective mines in the Reefton Goldfield.

25.3 Metallurgy

The metallurgical test work completed to date is sufficient to support a PEA level of assessment. The test work was completed on representative samples and concluded that:

- The Auld Creek ore is moderately hard with average comminution requirements. Additional test work is required to further refine understanding of ore hardness, competency and abrasivity.
- The Auld Creek ore contains little to no gravity recoverable gold. Ore gold grades are variable, with Au typically associated with arsenopyrite. Future work will refine the understanding of Au-arsenopyrite association.
- Arsenic head grades are high and arsenic deportment to final concentrate requires careful management since As is a penalty element within final Au concentrate products.
- Gold and stibnite are recoverable by froth flotation. To affect optimum stibnite recovery, activation via the addition of lead nitrate is required.
- Reverse cleaner flotation of the rougher concentrate, without the need for regrind, produces a separate, saleable, stibnite concentrate and Au concentrate.
- A stibnite recovery of 84.6% is achieved into concentrate containing 61.7% stibnite. Gold recovery to stibnite concentrate is equivalent to 5.1%, with the stibnite concentrate containing 12 g/t Au.
- An Au recovery of 90.1% is achieved into concentrate containing 95 g/t Au. Stibnite recovery to Au concentrate is equivalent to 12.6%, with Au concentrate containing 4.1% stibnite and 10% As.

26 Recommendations

26.1 Geological and Mineralogical Scope

A subsequent programme of works is recommended by the QP (Abraham Whaanga). In addition, the QP (Abraham Whaanga) makes the following recommendations.

26.1.1 Data Management

- Transfer all drilling data (collar, survey, assay, lithology, bulk density, recovery, geotechnical, etc.) from MS Excel workbooks for all prospects into the secure database containing Auld Creek data before any further drilling is undertaken.
- Collect all QC data, including duplicate measurements (e.g. from soil sampling, trenching, drilling, bulk density, and pXRF analysis) to allow quantitative assessment of data quality.
- Undertake a full core shed sample and core inventory.

26.1.2 Quality Assurance

26.1.2.1 Soil

- Revise the soil sampling SOP to provide specific instructions, including sample splitting.
- Develop an SOP for ionic leach sampling that is specific to RUA, including only relevant information and instructions.

26.1.2.2 Drilling

- Resurvey all collars surveyed by GPS using DGPS, as the GPS was prone to large errors (~5 m).
- Revise the bulk density SOP to provide tolerance thresholds and detail QC procedures.
- Revise the drilling SOP to clearly document the procedure to be followed in the event of poor core recovery, including guidelines on what is considered acceptable recovery.
- All core sizes (PQ and HQ) were half-core sampled. The QP recommends updating the SOP to include different procedures for cores with different diameters. NQ core should be sampled in full, rather than half-core.
- Mark all core with an orientation line and cut core a few degrees off the line to preserve it. Always collect the same half of the core to reduce sample selection bias.
- Update the core cutting SOP to document the process for determining the location of the cut line through mineralised or broken intervals.
- Update the core logging SOP to include regular check logging to ensure consistency of logging between geologists.
- Create an SOP covering sample transport and chain-of-custody details to fully capture the process once drilling details and logistics have been confirmed.

26.1.2.3 pXRF

- Update the pXRF SOP to include instructions on reviewing the QC data, including calibrating the pXRF data using the CRM results.
- Add photographs to the pXRF SOP to demonstrate the pXRF set-up with test stand, laptop, and sample cups.

26.1.2.4 Trenching

- Resurvey the trenches located by GPS by a professional surveyor, as the GPS was prone to large errors (~5 m).

26.1.3 Quality Control

26.1.3.1 Bulk Density

- Collect repeat/duplicate bulk density and moisture data from diamond drill core.
- When selecting bulk density samples, select core samples with a range of defects, and test alternative methods.

26.1.3.2 Drilling/Sampling

- Collect repeat GPS measurements for all collars and trench locations in order to assess the quality of the location data.
- Resurvey trench locations using DGPS.
- Increase the collection of first-split duplicate samples for any diamond drillholes that are to be included in future resource estimates.
- Collect duplicate trench samples during the first split to monitor the performance of the splitting stage.
- Collect second-split (coarse crush) repeat samples for any future resource delineation drilling programmes from the same samples used for core-split duplicates.
- Continue to monitor the QC data to ensure that any issues are identified and resolved.
- For pulp samples, the QP recommends instructing the relevant laboratory to homogenise samples before collecting sub-samples to avoid bias caused by settling during storage and transport.
- The QP (Abraham Whaanga) recommends twin drilling, particularly of significant intersections, as the Project progresses to higher-confidence resource definition.
- Undertake further investigations to identify the source of the check sample bias between the original half-core and the quarter-core check samples.

26.1.4 Analytical

- Re-assay original blanks that fail during a programme of re-analysis, rather than just the primary sample material.
- Analyse all intervals of interest for Sb using multi-element laboratory methods.
- Calibrate all pXRF data based on the CRM results.
- Source new, matrix-matched CRMs.
- Include blank samples after suspected high-grade intervals.

26.1.5 Metallurgical

- Conduct bespoke GRG test work in future test work programs.
- Future test work programs should explore whether the use of a coarser grind and lead nitrate activator would offer any further improvement in rougher flotation performance.
- Future test work should consider leaching of the flotation circuit tails as a means to improve Au recovery. Low pH test work along with the use of oxidants (lead nitrate) and pre-aeration should be tested.
- Consider measures to reduce As grade during cleaner optimisation test work in future works.

26.1.6 Geotechnical

- Laboratory testing: Conduct UCS and tensile strength testing to improve confidence in rock strength parameters for the PFS.
- Geotechnical model: Develop a high-level geological and geotechnical model, including bedding wireframes, major faults, and bedding intensity.
- Structural data: Improve structural confidence through enhanced logging and/or oriented core data.

- Numerical modelling: Once the mining method and schedule are defined, undertake detailed numerical modelling to refine stope stability, dilution, infrastructure placement, and CRF sill performance.
- Stress measurements: Consider in situ stress measurements at later stages when underground access is available.
- Design optimisation: Complete trade-off studies between stope span, dilution, and support requirements.

26.1.7 Mine Design

- OHCAF is recommended as the most appropriate mining method as it allows for higher productivity and lower costs compared with UHCAF.
- At the PFS level, UHCAF using pastefill could be investigated in a trade-off study, which could give the advantage of reducing tailings storage requirements.
- At the PFS level, the 1 km-long twin decline (to connect this portal location to the decline design) should be optimised.
- An innovative mining method known as Shotcrete Drift-and-Fill (SDAF) is recommended in areas of the deposit that are much wider than the deposit average width.
- At the PFS level, create a 3D schedule for the SLOS option to fully understand the trade-off between the OHCAF and SLOS schedule.
- At the PFS level, revisit other materials handling options (e.g. hoist or conveyor) for hauling material to the surface.

26.1.8 Other

- For Sb at Auld Creek, the QP (Abraham Whaanga) recommends reviewing the two estimation domains containing high- and low-grade populations and determining if two geological domains can be defined.

26.1.9 Phase 1

Following the review of historical and recent exploration undertaken at the Project, the QP (Abraham Whaanga) recommends a staged and success-driven exploration programme.

26.1.9.1 Exploration Target Interpretation

The QP (Abraham Whaanga) recommends undertaking a targeting programme over the Project. This work will require a comprehensive process of data compilation, data processing, and the creation of new interpretations and exploration targets for the Project area. This phase of work will require the compilation of all existing geological data in a Project-wide database and GIS workspace. This phase will also fulfil a number of CMA permit obligations, such as data compilation and targeting.

26.1.9.2 Geophysical Surveys

RUA has its own proprietary ultra-detailed magnetic surveying equipment (the UAV-based MagArrow system) that it plans to use extensively to assist in structural interpretation associated with specific target areas. The magnetic surveying will also fulfil the geophysical component of the CMA permit obligations across all the permits.

26.1.9.3 Drilling

Using the MRE completed for the Project, RUA plans to carry out additional comprehensive geological modelling, with ramp-up drilling being a priority.

26.1.9.4 Regional Exploration

On completion of the mineral systems evaluation and targeting of the whole Project area, a comprehensive surface geochemical and field geological mapping programme is envisaged to bring additional opportunities to provide a pipeline of exploration targets for modelling and drilling. This work will fulfil the CMA work programme obligations in the 2026 exploration program.

26.1.10 Phase 2

The Phase 2 exploration programmes will be dependent on the exploration success of the Phase 1 programmes. The QP (Abraham Whaanga) notes that the bulk of the Phase 2 expenditure will be associated with the diamond drilling in and around known Mineral Resources. The timing of these drilling programmes will vary based on exploration success and consenting for access.

26.1.11 Budget

The QP (Abraham Whaanga) has recommended budget and exploration tasks for the Phase 1 and 2 exploration programmes, which are presented in Table 26-1. Estimated costs are in Canadian dollars (CAD).

Table 26-1: Proposed exploration budget (CAD) for Phase 1 and 2 expenditures.

Category	Phase	Exploration task	Estimated Cost (CAD)
Prospecting and exploration expenditures	1	Targeting and Data Compilation	125,000
	1	Mapping	174,000
	1	Geochemistry	150,000
	1	Geophysics	97,000
	1	Drilling	4,219,000
Other expenditures	1	Consenting	332,000
	1	Administration	542,000
	1	Corporate	210,000
Total Phase 1			**5,840,000**
Prospecting and exploration expenditures	2	Data Compilation	42,000
	2	Mapping	74,000
	2	Geochemistry	77,000
	2	Geophysics	33,000
	2	Drilling	2,288,000
Other Expenditures	2	Consenting	124,000
	2	Administration	263,800
	2	Corporate	81,000
Total Phase 2			**3,566,000**

26.2 Mining Scope

Given the concept-level nature of this study, additional work will be required before the Project can advance to a PFS. Priorities for future work include the following.

- Substantial exploration drilling (which is already underway) for resource definition to facilitate a more optimised mine design, and for upgrade of resource classification in line with PFS requirements.
- Geotechnical and hydrogeological data collection to support advancement to PFS.
- Full investigation, including mine design and schedule, for a SLOS or hybrid (SLOS and OHCAF) mining method and trade-off study to select the most applicable mining method.

26.3 Infrastructure

A focused recommendation for infrastructure development is to advance the current conceptual designs through targeted geotechnical investigations of both the IWF and ECD sites, as well as the tailings and waste rock materials to be placed within them. These investigations should refine foundation conditions, material properties, embankment performance, and drainage requirements, ensuring that construction quantities, liner systems, and stability assessments are based on defensible data. Updated geotechnical inputs will support more accurate engineering, sequencing, and cost estimates as the Project progresses towards the PFS.

26.4 Water Balance

The water balance indicates that the site is strongly water-positive, with an excess of ~67 m^3/h (18.6 L/s). However, significant water is currently modelled as being extracted from Devils Creek. This apparent imbalance reflects the assumption that underground dewatering water is unsuitable for process use or for use in underground operations.

Further work at the PFS stage should focus on characterisation of underground water quality and assessment of its suitability for reuse. If suitable, this could significantly reduce raw water abstraction requirements and reduce water treatment demands significantly.

26.5 Metallurgy

Additional metallurgical test work is required to further refine the understanding of the metallurgical response, together with the physical characteristics of the orebody. Variability test work is required to assess variability in ore response and define the product quality achievable from the three main ore types. In addition to variability testing, the impact of water quality and metallurgical response is to be assessed. Additional metallurgical test work is scheduled to be completed during the PFS.

27 Certificate of Qualified Person (Abraham Whaanga)

I, Abraham Whaanga, BSc MAusIMM CP(Geo) of 2 Grenadier Lane, Waihi, Waikato, 3610, New Zealand, do hereby certify that:

- I am a Senior Resource Geologist at RSC Consulting Ltd, located at 245 Stuart Street, Dunedin, 9106, New Zealand.
- The Technical Report to which this certificate applies is titled "Technical Report on the Preliminary Economic Assessment for the Auld Creek Gold-Antimony Project, New Zealand", with an effective date of 19 June 2026.
- I graduated with a BSc from Victoria University of Wellington in 2000.
- I am a Member and Chartered Professional (CP) registered with the Australasian Institute of Mining and Metallurgy (AusIMM) in Australia (recognised overseas professional organisation) as member 304495, in good standing.
- Throughout my career, I have practiced continuously as an exploration geologist, underground mining geologist, geology manager, and consultant for mining and exploration firms in the following commodities: epithermal and orogenic gold, komatiite-hosted nickel sulphide, and iron ore. I have undergone continuing professional development with recognised courses and training seminars.
- I have read the definition of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
- I have undertaken site visits to the Reefton Project on 24–25 January 2025 and 19–21 January 2026.
- I am responsible for sections 1–12, 14, 19, 20, and 23–26 of this Technical Report.
- I am independent of the issuer, Rua Gold Inc., applying all of the tests in Section 1.5 of National Instrument 43-101.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.
- As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed this 19 June 2026 in Waihi, New Zealand.

(Original signed and sealed)

Abraham Whaanga, BSc MAusIMM CP(Geo)

Senior Resource Geologist, RSC Consulting Ltd

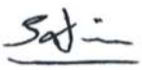
28 Certificate of Qualified Person (Gary Davison)

I, Gary Davison, Dip. Min. Eng M.Min. Ec FAusIMM(CP Min) of 15 Hannan St, Williamstown, 3016, Victoria, Australia, do hereby certify that:

- I am a Principal Mining Engineer and Director of Mining One Pty Ltd, Level 9, 50 Market Street, Melbourne, Victoria, 3000, Australia.
- The Technical Report to which this certificate applies is titled “Technical Report on the Preliminary Economic Assessment for the Auld Creek Gold-Antimony Project, New Zealand”, with an effective date of 19 June 2026.
- I graduated with a Diploma of Mining Engineering from the Royal Melbourne Institute of Technology (RMIT University) in 1978 and hold a MSc in Mineral and Energy Economics from Macquarie University NSW (1997).
- I am a Fellow registered with the Australasian Institute of Mining and Metallurgy (AusIMM) in Australia (recognised overseas professional organisation) as member 106195, in good standing.
- I have over 48 years of experience in underground and surface mining operations, mine management, mine design, feasibility studies, technical due diligence, and operational reviews across a range of commodities and mining jurisdictions. I have extensive experience in underground mining methods relevant to the Project, including cut-and-fill and long-hole stoping operations, and have supervised and reviewed numerous technical and economic studies from scoping through to feasibility level.
- I have read the definition of “Qualified Person” set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
- I have conducted a site visit to the Reefton Project.
- I am responsible for sections 1–4, 16, 18, 21, 22, and 24–26 of this Technical Report.
- I am independent of the issuer, Rua Gold Inc., applying all of the tests in Section 1.5 of National Instrument 43-101.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.
- As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed this 19 June 2026 in Melbourne, Victoria, Australia.

(Original signed and sealed)



Gary Davison, Dip. Min. Eng M.Min. Ec FAusIMM(CP Min)

Principal Mining Engineer and Director, Mining One Consultants

29 Certificate of Qualified Person (Marius Ward Phillips)

I, Marius Ward Phillips, NHD(ExMet) MAusIMM CP(Met) RPEQ of 18 Rosemont Court, Underwood, Queensland, 4119, Australia, do hereby certify that:

- I am a Technical Director Metallurgy and Process at Pitch Black Group Pty Ltd, located at 8/230 Brunswick Street, Fortitude Valley, Queensland, 4006, Australia.
- The Technical Report to which this certificate applies is titled "Technical Report on the Preliminary Economic Assessment for the Auld Creek Gold-Antimony Project, New Zealand", with an effective date of 19 June 2026.
- I graduated with a NHD Extractive Metallurgy from Technikon Witwatersrand (University of Johannesburg) in 1997.
- I am a Member and Chartered Professional (CP) registered with the Australasian Institute of Mining and Metallurgy (AusIMM) in Australia (recognised overseas professional organisation) as member 227570, in good standing. I am a Registered Professional Engineer of Queensland (RPEQ).
- Throughout my career, I have practiced continuously for 32 years as a metallurgist, metallurgical superintendent, metallurgical manager, process engineer, principal process engineer and process consultant for mining and engineering firms in the following commodities: base metals (copper, lead, zinc), precious metals (gold, silver) and PGMs. I have undergone continuing professional development with recognised courses and training seminars.
- I have read the definition of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
- I have undertaken a site visit to the Reefton Project.
- I am responsible for sections 1–4, 13, 17, 18, 21, and 24–26 of this Technical Report.
- I am independent of the issuer, Rua Gold Inc., applying all of the tests in Section 1.5 of National Instrument 43-101.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.
- As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed this 19 June 2026 in Underwood, Brisbane, Queensland, Australia.

(Original signed and sealed)

Marius Phillips, NHD(ExMet) MAusIMM CP(Met) RPEQ

Technical Director Metallurgy and Process, Pitch Black Group Pty Ltd

30 Certificate of Qualified Person (Timothy Rowles)

I, Timothy Rowles, MSc FAusIMM CP(Env) MAIG (RPGeo) RPEQ, of 2 Rees Street, Kelvin Grove, Queensland, 4059, Australia, do hereby certify that:

- I am Regional Manager at Knight Piésold Pty Ltd, located at 36 Cordelia Street, South Brisbane, Queensland 4101, Australia.
- The Technical Report to which this certificate applies is titled "Technical Report on the Preliminary Economic Assessment for the Auld Creek Gold-Antimony Project, New Zealand", with an effective date of 19 June 2026.
- I graduated from the Royal School of Mines, Imperial College, London, with a BSc in Environmental Geology in 1996 and from the University of Manchester with a MSc in Earth and Environmental Science in 1998.
- I am a Fellow and Chartered Professional registered with the Australasian Institute of Mining and Metallurgy (AusIMM) (recognised overseas professional organisation) as member 227249, in good standing. I am a member and Registered Professional Geoscientist with the Australian Institute of Geoscientists (recognised overseas professional organisation) as member 10317, in good standing. I am a Registered Professional Engineer of Queensland (RPEQ).
- Throughout my career, I have practiced continuously for over 25 years as a Geotechnical/Geo-environmental Engineer undertaking the design, construction, operational oversight and closure of tailings management systems, surface water management systems, and mine site infrastructure for a broad range of commodities including gold and antimony. I have undergone continuing professional development with recognised courses and training seminars.
- I have read the definition of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
- I have not personally visited the Project site.
- I am responsible for sections 1–4, 18.8, 18.10, 21.1.4, 21.2.3, 26.3 and 26.4 of this Technical Report.
- I am independent of the issuer, Rua Gold Inc, applying all of the tests in Section 1.5 of National Instrument 43-101.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read NI 43-101 and Form 43-101F1, and this Technical Report has been prepared in compliance with that Instrument and Form.
- As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed this 19 June 2006 in Brisbane, Australia.

(Original signed and sealed)

Timothy Rowles, MSc FAusIMM CP(Env) MAIG (RPGeo) RPEQ

Regional Manager (QLD), Knight Piesold Pty Ltd

RUAGOLD

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