ASX RELEASE
ASX:SMI

17 April 2024

OUTSTANDING ECONOMICS - SCOPING STUDY FIRST 10 YEARS, FROM RAS ONLY

Cautionary Statement

This Scoping Study (Study) is conceptual to determine the open pit and underground gold mining metrics as a pre-cursor to further detailed open pit and underground mining studies.

The Study is based on a JORC compliant Mineral Resource Estimate (MRE) for the Rise and Shine gold deposit (ASX release 16 February 2024). The MRE underpinning the Study has been prepared by a Competent Person in accordance with the requirements of the JORC Code (2012). All material assumptions and technical parameters underpinning the MRE continue to apply and have not materially changed.

The Study is preliminary and not intended as a feasibility study. It should be understood by the reader that this announcement reports on initial outcomes of early stage open pit and underground optimisation for the Rise and Shine (RAS) deposit only. The Study includes preliminary economic analysis and is based on a number of Production Targets and material assumptions and other relevant factors estimated by a Competent Person to have an accuracy range of approximately ±25%. The Study findings are indicative only and subject to assumptions outlined in this announcement, and market and operating conditions. They should not be construed as guidance and are subject to further studies and all necessary approvals, permits, internal and regulatory requirements. While Santana considers that all the material assumptions are based on reasonable grounds, there is no certainty that they will prove to be correct or that the range of outcomes indicated by this Study will be achieved.

The constraining of the resource model and the addition of dilution parameters to provide plant feed estimates for preliminary economic analysis should not purport to represent a formal indication of Reserves for the Project or parts of it at this stage. As such, no Ore Reserve has been declared. While each of the Modifying Factors was considered and applied, there is no certainty of eventual conversion to Ore Reserves or that the Production Targets will be realised. Further evaluation work and appropriate studies are required before Santana is in a position to estimate an Ore Reserve or to provide any assurance of an economic development case. As such, the Study outcomes and forecast financial information referred to in this announcement is based on accuracy levels and technical and economic assessments that are insufficient to support estimation of Ore Reserves.

The open pit described in the report captures a portion of the RAS MRE and include variable amounts of Indicated and Inferred Mineral Resources. The total amount of Indicated ounces used in the production target is 84%, with Inferred being 16%. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of additional Indicated Mineral Resources or that the outcomes from open pit optimization studies will be realised. Santana confirms that the potential financial viability of producing gold from the Project is not dependent on the inclusion of Inferred Resources in the various Production Targets.

This announcement has been prepared by Santana. This document contains contextual information current as at the date of this announcement. This document provides a summary of the Study and does not purport to be all-inclusive or complete. Project development assumes the completion of a Definitive Feasibility Study (DFS). There is no certainty that the Company will be able to source the required development funding if and when required. The Company considers that there is a reasonable expectation that a project of this scale will be able to be funded with a combination of debt and equity at the appropriate time. It is also possible that such funding may only be available on terms that may be dilutive to or otherwise affect the value of the Company shares. It is also possible that Santana could pursue other 'value realisation' strategies such as a sale, partial sale or joint venture of the project. If it does, this could materially reduce the Company's proportionate ownership of the project. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the Study.

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Outstanding Economics - Scoping Study - First 10 Years from RAS Only:

Santana Minerals Limited (ASX: SMI) ("Santana" or "the Company") is pleased to provide the outcomes from its initial Scoping Study on the 100% owned Rise and Shine (RAS) discovery at this current snapshot-in-time. The RAS deposit remains open down plunge and infill drilling is continuing to upgrade Inferred resources to the Indicated resource category. RAS is just one of a number of discoveries within the overall Bendigo-Ophir Gold Project in the Central Otago region of New Zealand's South Island.

Key metrics at the conservative study price (Base Case) and spot prices in various currencies are provided for comparison[1]:

Key Financial Assumptions	Unit	Base Case NZD	NZD	AUD	USD
Gold Price Assumed	$/oz	$2,705	$3,900[2]	A$3,545[2]	US$2,340[2]
Exchange Rate	USD:$	US$0.61	US$0.60	US$0.66	US$1.00
Key Project Metrics					
Gold Produced	Oz	1.12 million			
Initial Mine Life		10 years of mine production			
Gold Revenue	**$M**	**$3,030M**	**$4,368M**	**$3,971M**	**$2,621M**
Mining Costs	$M	$530	$530	$481	$318
Processing Costs	$M	$228	$228	$207	$137
General and Admin Costs	$M	$42	$42	$39	$25
Royalty - Government	$M	$61	$87	$79	$52
Royalty - Other	$M	$82	$118	$107	$71
Total Cash Operating Cost	**$M**	**NZ$943M**	**NZ$1,005M**	**A$914M**	**US$603M**
	$/oz	*NZ$841/oz*	*NZ$897/oz*	*A$816/oz*	*US$538/oz*
Project EBITDA	**$M**	**NZ$2,087M**	**NZ$3,363M**	**A$3,057M**	**US$2,018M**
Depreciation and Amortisation	$M	$554	$554	$503	$332
Total Production Cost	**$M**	**NZ$1,496M**	**NZ$1,559M**	**A$1417M**	**US$935M**
	$/oz	*NZ$1,336/oz*	*NZ$1,392/oz*	*A$1265/oz*	*US$835/oz*
Net Profit Before Tax (NPBT)	$M	$1,534	$2,809	$2,554	$1,686
Tax Payable (28%)	$M	$438	$805	$732	$483
After Tax Profit	**$M**	**NZ$1,096M**	**NZ$2,005M**	**A$1,822M**	**US$1,203M**
Capital					
Capital Plant and Infrastructure	$M	$143	$143	$130	$86
Working Capital for pre-strip and mine set-up.	$M	$113	$113	$103	$68
Sustaining Capital Stripping and UG Development	$M	$297	$297	$270	$178
Total CAPEX over Mine Life	**$M**	**NZ$554M**	**NZ$554M**	**A$503M**	**US$332M**
DCF Outcomes					
Initial Project $NPV_{10\%}$	$M	$486	$937	$852	$562
IRR	%	49%	75%	72%	72%
Simple Payback (from start of production)	Years	1.4	1.0	1.0	1.0

[1] Any discrepancies in totals are due to rounding.
[2] Spot price as at 9th April 2024.

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Study Highlights:

- The Scoping Study (+/- 25% accuracy) applied a conservative base-case of NZ$2,705/oz gold price (US$1,650/oz at a USD/NZD exchange rate of 0.61) to open pit and underground optimisation studies which concluded in strong, positive financial outcomes (refer to body of report).
- A mining and processing rate of 1.5Mtpa was identified as the optimal size to balance waste-stripping and consistent production from the proposed open pit (12.1Mt at 2.5g/t) and underground mines (2.3Mt at 3.1g/t) from the RAS deposit.
- The study, which focused on RAS only, concludes an average annual gold production rate of 110,000oz per annum producing a total of 1.12Moz of gold over an initial 10-year mining operation.
- At a spot price of NZ$3,900/oz (A$3,545/oz, US$2,340/oz), gold is produced at a Cash Operating Cost (incl. royalties) of NZ$897/oz (A$816/oz, US$539/oz) and a Total Production Cost (cash operating plus depreciation & amortisation) of NZ$1,392/oz (A$1,265/oz, US$835/oz).
- Process Plant CAPEX is estimated at NZ$103.9M with an additional estimate of NZ$39.4M required for infrastructure and services. Given the geometry of the RAS deposit an amount of NZ$113.0M is required for pre-production working capital, which is dominated by the initial pre-stripping of waste to enable steady state production (total is NZ$256.3M).
- The staged development requires additional capex of NZ$297M to sustain production which is funded from internal cash flows.
- The average strip ratio for the open pit operation (post initial pre-strip is 9.8:1 (waste:ore) tonnes) over the 10 year period (12.5:1 overall if including the pre-strip).
- Initial optimisation of an underground mining add-on suggested that a viable underground mining operation post open pit was strongly economic. Only a small portion of the underground resource was considered due to its current JORC 2012 classification. The total production targets are made up of 84% Indicated category and 16% Inferred category, with the Inferred category material mined only as-a-consequence of mining the Indicated resource.
- At the spot price of gold used in the Study, initial NZ government royalty payments of NZ$87M plus Corporate Tax payments (28%) of NZ$805M provide an indication of the significance of this project to the NZ government.

The Scoping Study only addresses approximately 50% of the Total Mineral Resource at RAS. Drilling is continuing to upgrade more of the Inferred category to Indicated category so that it can be used to further enhance the project.

Santana CEO, Damian Spring said:

> "*We are thrilled to reveal some preliminary economics for this initial part of the RAS orebody, albeit across just part of the known mineralisation which is continuing to be defined.*
>
> *The initial financial metrics for the project at this scoping study level are compelling and we have no doubt of the importance of the discovery to our shareholders, and New Zealand in both a fiscal and economic output relevance.*
>
> *We are rapidly advancing the project toward production within the statutory guidelines which are presented and firmly believe we are on the cusp of a significant project with sustainable economic and employment outcomes for the region, with minimal environmental, but strong positive social impact.*
>
> *Whilst our short term focus is on commercialisation of the discovery, our eyes remain focussed on the huge potential we believe exists to enhance, enlarge and extend the project within the region."*

Please see the full Scoping Study appended below for more detail. This announcement has been authorised for release by Santana's Board of Directors.

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Previous ASX Disclosures - 2012 JORC Code

Information relating to Mineral Resources, Exploration Targets and Exploration Data associated with the Company's projects in this announcement is extracted from the following ASX Announcements:

- ASX announcement titled "More high grades from RAS Infill drilling" dated 4 April 2023
- ASX announcement titled "New Gold assays and metallurgical results from RAS" dated 24 April 2023
- ASX announcement titled "High grade intercept from infill drilling south of RAS ridge" 2 June 2023
- ASX announcement titled "RAS high grade zones expand with drilling results" dated 22 June 2023
- ASX announcement titled "Infill drilling at RAS continues to grow confidence" dated 13 July 2023
- ASX announcement titled "High grade zones strengthen ahead of RAS MRE Update" dated 27 July 2023
- ASX announcement titled "New results extend potential for upcoming RAS MRE" dated 30 August 2023
- ASX announcement titled "Drill results confirm and extend high grade mineralisation" dated 8th September 2023
- ASX announcement titled "Strong RAS and regional drill results" dated 23 October 2023
- ASX announcement titled "More High Grade Gold from Rise and Shine Prospect" dated 23 November 2023
- ASX announcement titled "Bendigo-Ophir Exploration and Project Update" dated 04 January 2024
- ASX announcement titled "High-Grade Intercepts Close out Resource Drilling at RAS" dated 24 January 2024
- ASX announcement titled "1.3m ounces upgraded to Indicated category from RAS drilling" dated 16 February 2024
- ASX announcement titled "Shiny Outcomes from Latest Metallurgical Test Work at RAS" dated 02 April 2024

A copy of such announcements are available to view on the Santana Minerals Limited website www.santanaminerals.com. The reports were issued in accordance with the 2012 Edition of the JORC Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The Company confirms that it is not aware of any new information or data that materially affects the information included in the original market announcements. The Company confirms that the form and context in which the CompetentPerson's findings are presented have not been materially modified from the original market announcements.

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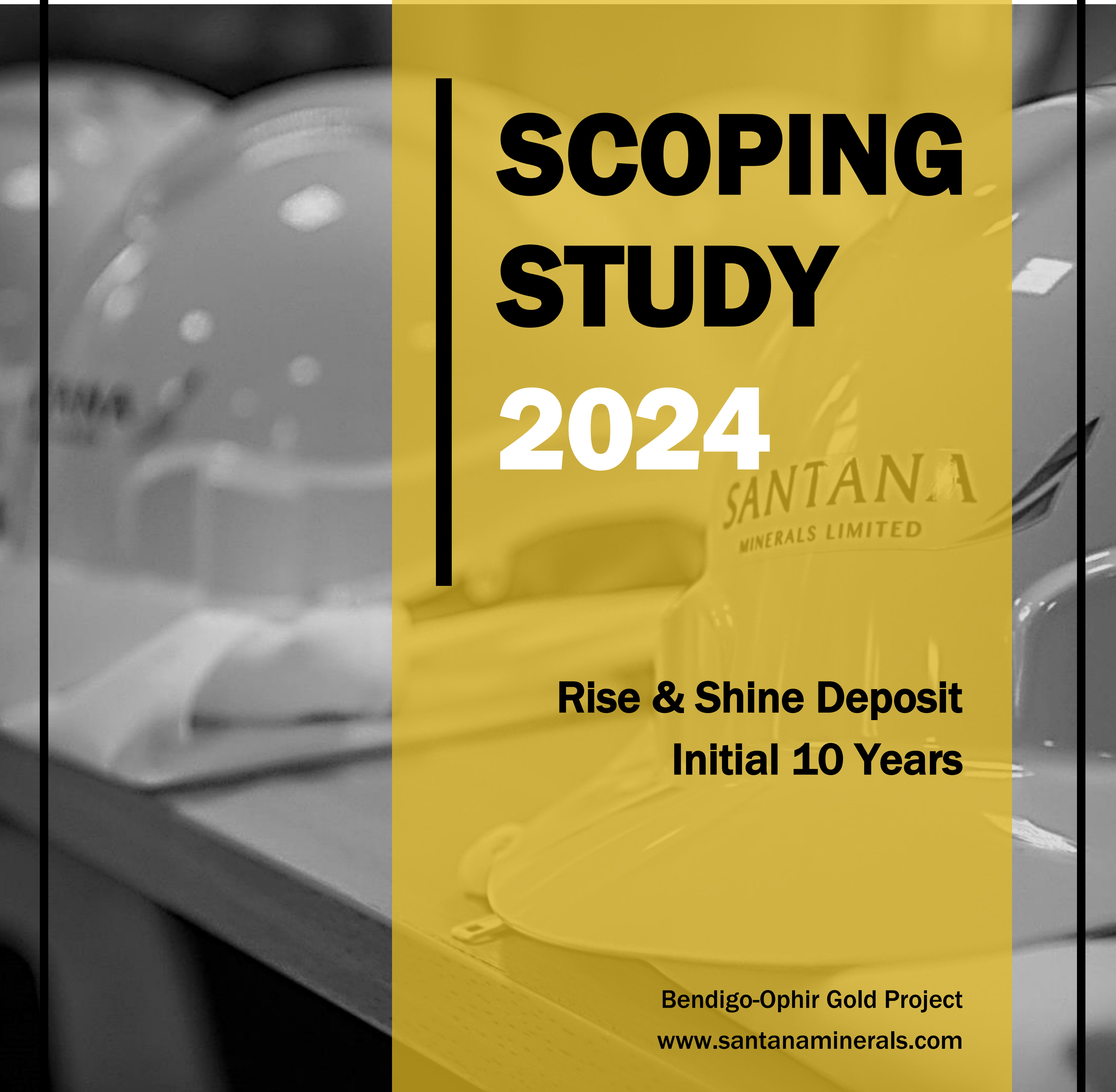

SCOPING STUDY 2024

Rise & Shine Deposit
Initial 10 Years

Bendigo-Ophir Gold Project

www.santanaminerals.com

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EXECUTIVE SUMMARY

CURRENT SPOT GOLD PRICE RESULTS

NZ$3,900/oz[1]

AUD$3,545 USD$2,340

AVG. GOLD PRODUCTION/YR	LIFE OF MINE (YRS)	TOTAL GOLD RECOVERED
110,000oz	**10**	**1.12Moz**

REVENUE (NZ$)	PRE-PROD & WORKING CAPITAL (NZ$)
$4.4 BILLION	**$256 MILLION**

TOTAL CASH OPERATING COST (NZ$)	TOTAL PRODUCTION COST (NZ$)
$897/oz	**$1,392/oz**

NET PROFIT AFTER TAX (NZ$)	AFTER TAX NPV_{10} (NZ$)	AFTER TAX IRR
$2.0 BILLION	**$937 MILLION**	**75%**

EST. TIME TO PERMITTING[2]	EST. TIME TO CONSTRUCTION[2]	SIMPLE PAYBACK
6 MONTHS	**12 MONTHS**	**1 YEAR**

1. Please see Table 23 - LOM financial results summary which includes detail and definitions
2. Please see section on Permitting and Key Risks in the study report for more detail

Introduction

Project Location and Ownership

The Bendigo-Ophir Gold Project (BOGP) sits within Mineral Exploration Permit (MEP) 60311 as shown in the figure below (Figure 1). MEP 60311 is owned by Matakanui Gold Ltd (MGL) (NZBN 9429041420614) which is a New Zealand (NZ) registered company and will be the mine operator. MGL is a wholly-owned subsidiary of Santana Minerals Ltd (SMI) (ACN 161 946 989), a publicly listed company on the Australian Securities Exchange (ASX:SMI).

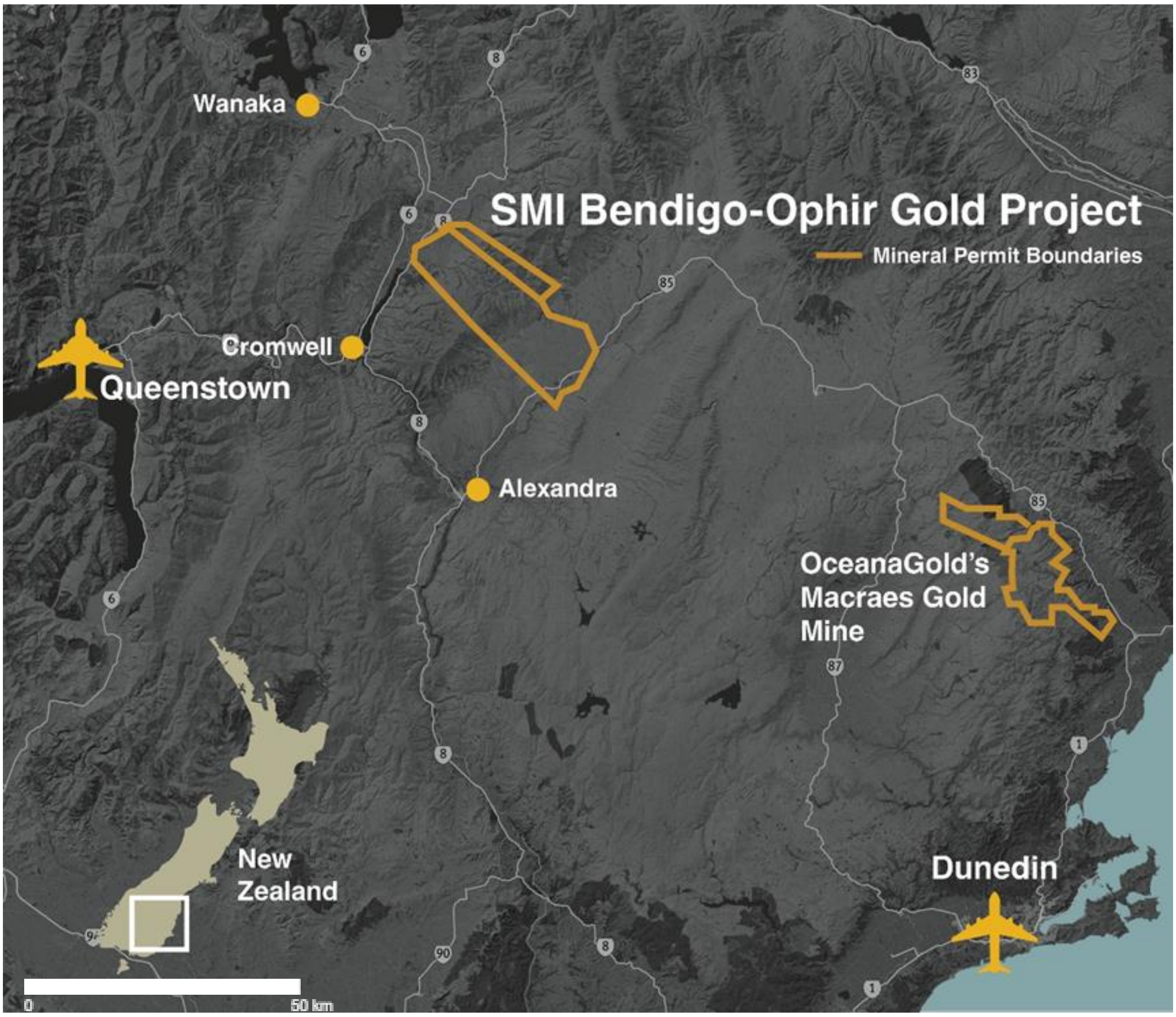

Figure 1 General location of the Bendigo-Ophir Gold Project

The BOGP is sited in the Dunstan Mountains of Central Otago New Zealand within areas administered by the Central Otago District Council (CODC) and Otago Regional Council (ORC).

The nearest main centre is the town of Cromwell, 26km by road south via State Highway 8 comprising 14kms along State Highway 8 and a further 12km via Bendigo Loop Road and the four-wheel drive Thomson Gorge Road (TGR) to the site of the Rise and Shine (RAS) Deposit.

The project area sits on semi-arid grazing land with moderate topography. The mine site and proposed plant site are located between incised valleys cut with Otago schist rock, and hidden within the terrain. The proposed active mining area rises from the Bendigo terraces at 370mRL to the top of the RAS future pit crest at approximately 770mRL.

Further south, the head of RAS valley peaks at 970mRL creating a watershed divide from the Matakanui catchment to the east which will be unaffected by the RAS mining project.

At a local scale, the proposed RAS mine sits between two incised valleys. The Rise and Shine Creek valley to the south trends the contact between the unconforming and overlying schist rocks (TZ3 schist) and the underlying TZ4 schist which is the host to mineralisation, with the TGR tracking it. To the north east the deposit is bisected by the Shepherds Creek valley. Shepherds Creek is an ephemeral stream with intermittent flow and sits approximately 300m above RAS mineralisation where it transects. Refer to Figure 2 below.

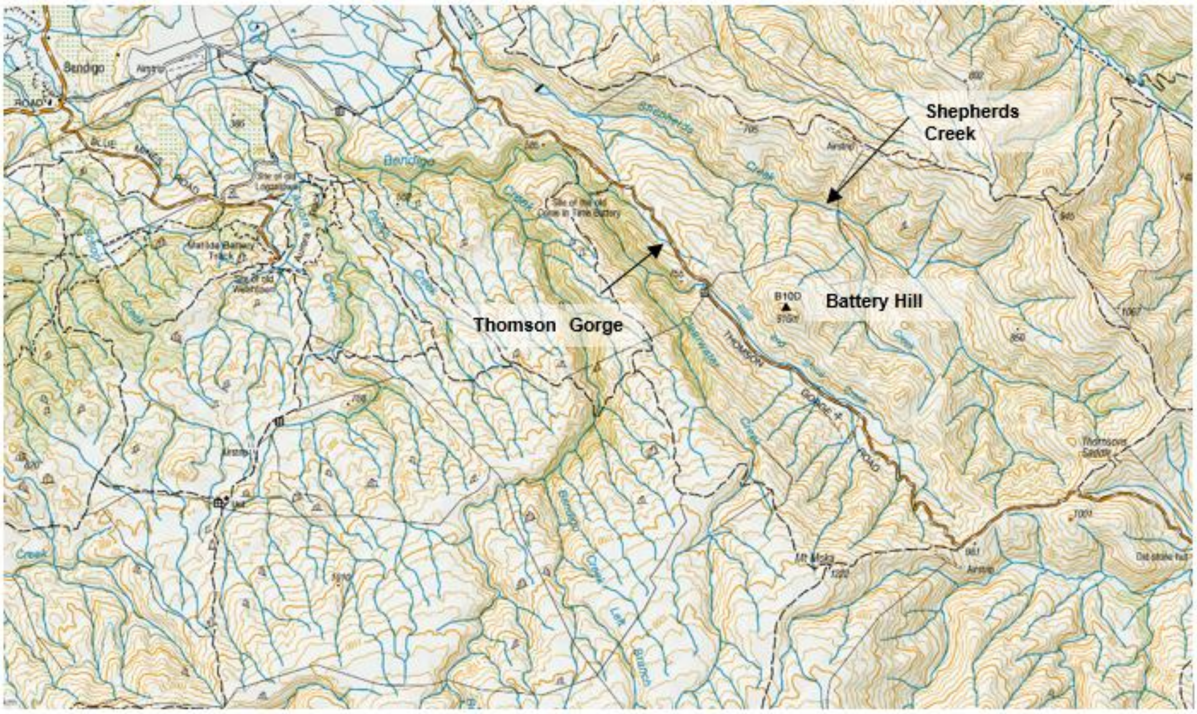

Figure 2 NZ Topo50 map of the BOGP

Land ownership across the project study area is freehold, private land with Bendigo Station to the SW and Ardgour Station to the NE. Otherwise, the study area is adjacent to Crown Land being the Bendigo Historic Reserve to the west, and the Ardgour Conservation Reserve to the east, both administered by the Department of Conservation (DoC) (Figure 2); and to Matakanui Station which is leasehold land administered by Land Information New Zealand (LINZ).

Study Team

The Study Team for the Scoping Study drew on a number of expert consultants and specialists in their respective fields. The Scoping Study was managed and collated by Santana's internal personnel. The following table outlines the key consultants contributing to the study:

Study Team		
Area		Completed by
Geology		
Mineral Resource Estimate		GeoModelling Ltd
Drillhole Database Management		Skerten Holdings Ltd

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Study Team		
Structural Review		Rodinian (NZ) Ltd
Mining Technical		
Geotechnical Engineering		PSM
Open Pit and Underground Optimisations		AMC Consultants
Open Pit and Underground Designs		AMC Consultants
Open Pit Schedules		BCPANZL
Underground Schedules		In-house
Metallurgy and Processing		
Metallurgical Testwork		IMO
Process Plant Design		MACA Interquip
Cost Modelling		
Power Supply Costing		Total Utilities
Processing Plant		MACA Interquip
Tailings Storage Facility		Engineering Geology Limited
Mining – Open Pit		In-house
Other Site Infrastructure		In-house
Site Administration		In-house
Environment		
Flora and Fauna		Alliance Ecology
Hydrogeology		Kōmanawa Solutions Ltd
Heritage		New Zealand Heritage Properties
Environmental Geochemistry		Mine Waste Management Ltd

Geology and Mineral Resource

The Project Area is located within the Otago Schist belt comprising Permo – Triassic metasedimentary and metavolcanic rocks metamorphosed to greenschist facies with peak metamorphism in the Cretaceous period. Gold mineralization is widespread within the Otago Schists with over 5 million ounces of hard-rock gold and 8 million ounces of alluvial gold being won from Otago Goldfields.

Regional Geology

The Dunstan Mountains is an uplifted block of the Otago Schists tilted to the northwest with remnants of a Cretaceous peneplain well preserved on its northern slopes.

The Manuherikia Basin to the southeast is infilled by Cenozoic sediments, and the fluvio-glacial Tarras Terraces lap the north western margin of the Dunstan Range.

The Otago Schist is formed from sedimentary and minor intermediate volcanics and volcaniclastics of the Caples and Torlesse tectono-stratigraphic terranes. Greenschist facies rocks of the Otago schist are sub-divided into four textural zones based on mineralogy and mineral textures.

Peak metamorphic grades in the Otago Schist occurred during the Jurassic when the Zealandia micro continent formed the outboard subduction complex of the Gondwana continental margin.

Deposit Geology

The Rise and Shine Shear Zone (RSSZ) is a late metamorphic low angle shear zone dipping 20 – 30 degrees northeast and generally crosscutting the metamorphic foliation at a low angle. From its outcrop in the Rise and Shine Valley, it has been traced for 1.7km north-northeast beneath the unconforming TZ3 cover rocks with the bulk of the mineralisation sitting beneath 150-300m of the benign cover rock.

The flat lying and flat plunging deposit sits within a zone up to 400m wide and can be up to 90 metres in thickness (typically 30-40m).

This study deals with the Rise and Shine deposit only which is one of several zones of highly anomalous gold mineralisation exposed by the Thomson Gorge Fault (TGF) which currently makes up the Bendigo-Ophir Gold Project (refer to Figure 3). Rise and Shine is the most advanced and most intensely drilled target over approximately seven (7) kilometres of strike of the unconformity.

Gold mineralisation has been defined at multiple locations along the RSSZ below the TGF, and all have been associated with historic alluvial and elluvial mining from the historic gold rush of the late 1800's.

Other deposits in their infancy of definition will likely add to the Rise and Shine Mine as the project expands in time. These include: the Come in Time (CIT) prospect, the Rise and Shine (RAS) deeps, the Shreks (SHR) prospect, the Shreks East (SRE) prospect, and a series of strong gold-arsenic pathfinder anomalies at Thomson Saddle (TSD) and Upper Thomson (UTS).

On total project scale the TGF or unconformity is inferred to extend 30 kilometres along the length of the project area most of which has not been subjected to modern exploration, but has had various phases of alluvial and elluvial prospecting in the gold rush of the late 1800's

On a mine scale, the TGF is a post metamorphic, post mineralisation cataclastic fault zone developed more or less along the hanging wall of the RSSZ. It separates chlorite rich, textural zone 3 (TZ3) schists in the hanging wall from biotite rich textural zone 4 (TZ4) schists in the shear zone and foot wall.

Within the 500m wide zone of NNE trending mineralisation at RAS, a higher grade core approximately 150-200m wide contains the majority of gold metal. The RAS deposit is primarily all fresh rock with subsurface oxidation variably extending from 5-20m depth.

The main mineralisation at RAS is associated with silica-siderite/ankerite alteration with minor arsenopyrite sulphides associated with the gold. In some areas a cataclastite (brecciated) network of anastomosing, post-metamorphic quartz, occur with minor sulphide veins in a halo of the core mineralisation.

Locally, a number of splay faults are interpreted coming off the main structure which give a sense of structural control. These are also mineralised and are traceable for 10s to 100s of metres.

Gold occurs as free gold particles, typically up to 400µm but with some coarser visible gold.

A minor component occurs associated with the arsenopyrite grains, but typically not in solid solution, giving rise to the free milling and highly gravity recoverable components expressed by metallurgical testworks.

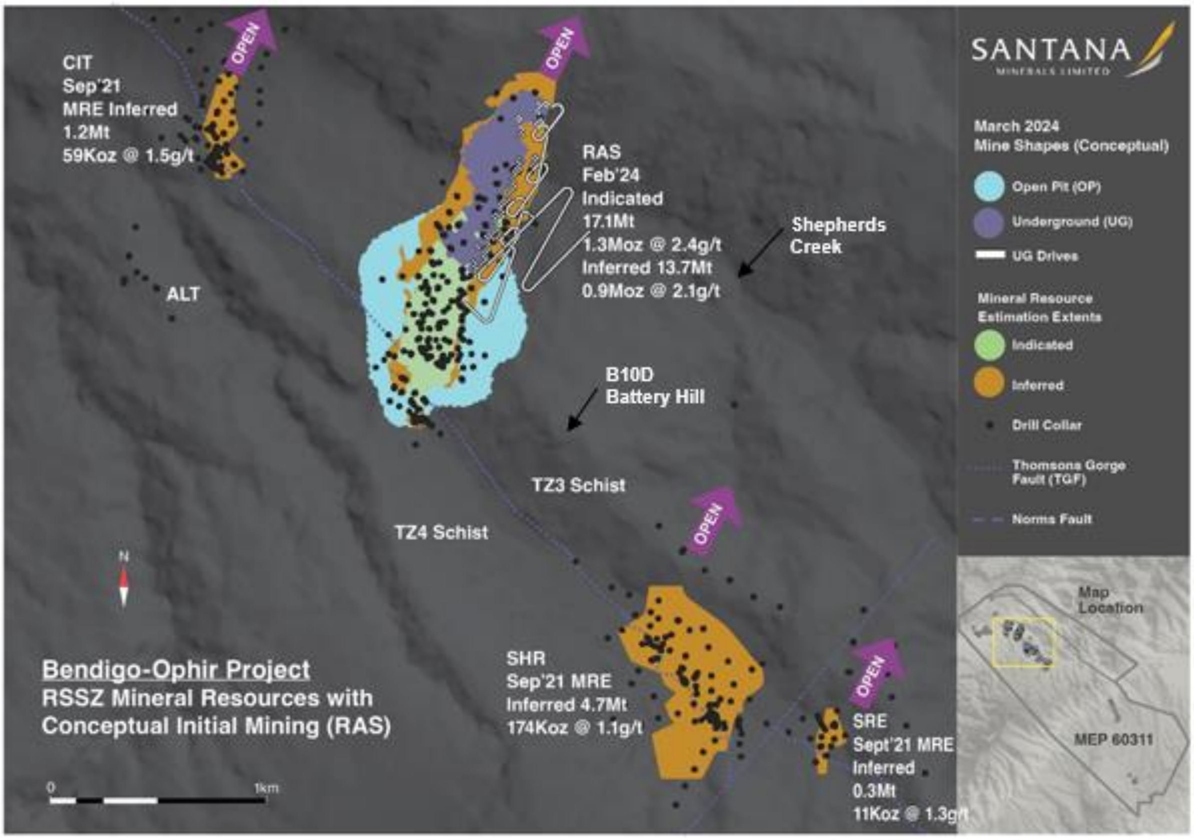

Figure 3 BOGP Location of Deposits contained with the Mineral Resource Estimate

Resource Estimation

The latest Mineral Resource Estimate (**MRE**) for RAS was prepared by independent consultants GeoModelling Limited in accordance with the JORC Code (2012 Edition) and was reported on 15 February 2024 (**2024 Feb MRE**) in ASX announcement *"1.3 million ounces upgraded to Indicated category from RAS infill drilling"*. The Project contains a Mineral Resource Estimate (MRE) calculated at a cut-off grade of 0.5 g/t Au with top cuts applied, as at February 2024 as follows:

Table 1 Bendigo-Ophir Gold Project Mineral Resource Estimate at 0.5 g/t

Deposit	Category	tonnes (Mt)	Au grade (g/t)	Contained Gold (koz)
RAS[1]	Indicated	17.1	2.4	1,293
	Inferred	13.7	2.1	923
RAS Total	Indicated and Inferred	30.8	2.2	2,216
CIT[2]	Inferred	1.2	1.5	59
SHR[2]	Inferred	4.7	1.1	174
SRE[2]	Inferred	0.3	1.3	11
RSSZ Total	Indicated	17.1	2.4	1,293
	Inferred	19.9	1.8	1,168
RSSZ Total	**Indicated and Inferred**	**37.0**	**2.1**	**2,462**

[1] 2024 RAS Mineral Resource Estimates completed by Mr Kerrin Allwood, (ASX announcement "1.3 million ounces upgraded to Indicated category from RAS infill drilling on 15 February 2024).

[2] 2021 Mineral Resource Estimates (2021 MRE) for CIT, SHR and SRE deposits completed by Ms Michelle Wild (CP) (ASX announcement on 28 September 2021).

Geotechnical and Groundwater

Mining study geotechnical inputs were based primarily on available geological and geotechnical parameters. Weathering wireframes were used and reviewed along with parameters from the nearby Macraes mining operation which has similar geology.

Drilling to support estimation of geotechnical constraints for RAS has been completed and will need to be adjusted for further stages of study based on the results of the related analysis.

Geotechnical

A dedicated geotechnical material property testing program was designed by PSM to capture information pertinent to characterising and understanding the mechanical behaviour of the different materials expected to be encountered during mining activities.

The following summarises the distribution and quality of the data available for geotechnical analysis:

- Approximately 100,000m of drill meterage across 627 boreholes.
- At the RAS deposit, drilling comprises of approximately 70,500m across 277 boreholes.
    - 2,000m reverse circulation (RC).
    - 68,500m diamond coring.
- Borehole spacing is approximately 30 – 35m providing good coverage of the deposit.
    - There is limited borehole coverage of the material behind the pit slopes as most boreholes are targeted/angled towards mineralisation.
- Geotechnical logging of exploration boreholes across the Project amounts to approximately 30,000m of logging data, of which 26,400m is within the RAS deposit. Logged geotechnical characteristics include:
    - Rock Quality Designation (RQD).
    - Fractures per metre.
    - Field Estimated Strength.
- A total of four fully cored geotechnical diamond drill holes located in the vicinity of RAS' planned pit walls totalling 1,019m.

The following points summarise the reviewed data for the geotechnical model:

- Rock mass:
    - The dominant rock mass unit is fresh schist, with a typical intact rock strength of R4 – high strength rock (50 to 100 MPa).
    - Thickness of weathering is variable across the site:
        - Base of Complete Oxidisation ranging from 4 to 25 mbgl.
        - Top of fresh rock ranging from 15 to 30m below base of complete oxidation.
    - Modelled thickness of cover sequences (alluvium/colluvium) of approximately 1 to 2m across the site. Thicker intervals of cover sequence are likely to be found in incised valleys.
- Geological structure:
    - Foliation is sub-parallel to mineralisation (dips 25 to 30° to the northeast).
    - The major fault orientation strikes northwest to southeast with a shallow dip of 20 to 30°.
    - Minor faults trend north and northeast and are thought to control the gold mineralisation.

The confidence level of the geotechnical data provides a good understanding of the implications for pit design and execution specific to the deposits.

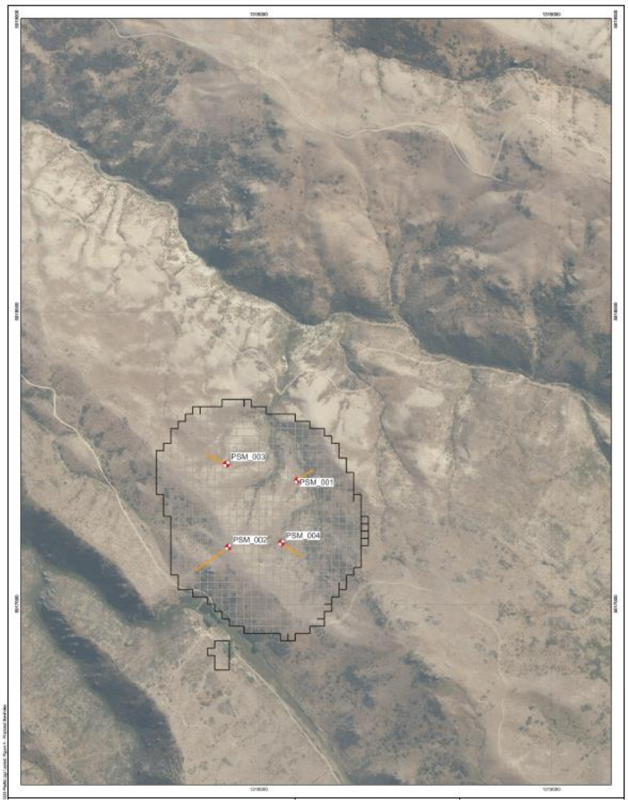

Figure 4 Location of geotechnical specific diamond drill holes at RAS

Groundwater

Depth to groundwater is variable across the site. Although it generally follows topography, ranging near surface within the base of the valley, to approximately 50m below ground level near ridges.

Some measurements describe artesian groundwater flows. Artesian conditions indicate high head pressures within the rock mass being released along the borehole, a possible result of a tight fractured rock aquifer associated with faults and possible compartmentalisation.

Modelling is being completed by Kōmanawa Solutions Ltd which has extensive experience in Otago including the nearby Lindis River scheme and Macraes gold mine.

Mining Optimisation, Design and Schedule

An open pit optimisation was undertaken using the Lerchs-Grossman (LG) algorithm. Underground optimisations using the Mineable Shape Optimiser (MSO) tool were undertaken in parallel. The outputs viewed together have informed the open pit to underground mining transition point as well as the mine designs.

The underground was further assessed for applicable mining methods as part of the MSO process.

Open pit stage designs were completed, and underground mining panels were defined.

A combined mining and processing schedule has been developed.

Pit Optimisation

Pit Optimisation

Pit Optimisation Input Parameters

Slope Sets

To determine the pit slope parameters to be used for the scoping study, available geological and geotechnical parameters and weathering wireframes were used, available core photos viewed and reviewed against nearby mining operations with similar geology.

OceanaGold's nearby Macraes operation is geologically very similar to RAS, therefore historic pit slope design parameters from Macraes can be a guide to the proposed open pit.

Table 2 Proposed overall open pit wall angles

Unit	Wall	Overall wall angle (Range)
TZ3	All	40°(+2°,-5°)
RSSZ and TZ4	North/South	47°(+2°,-5°)
RSSZ and TZ4	East/West	43°(+2°,-5°)

Mining Dilution and Recoveries

The minimum sub-cell size in the XY plane is 3.125m x 3.125m. Models were created on this grid using 2.5m and 5m high benches. Any waste in a bench above the mineralisation greater than 0.25g/t was counted as mill feed.

The following figures show the Mineral Resource, then the area selected for treatment when it is consolidated with 0m, 1.5m and 2.5m of dilution added.

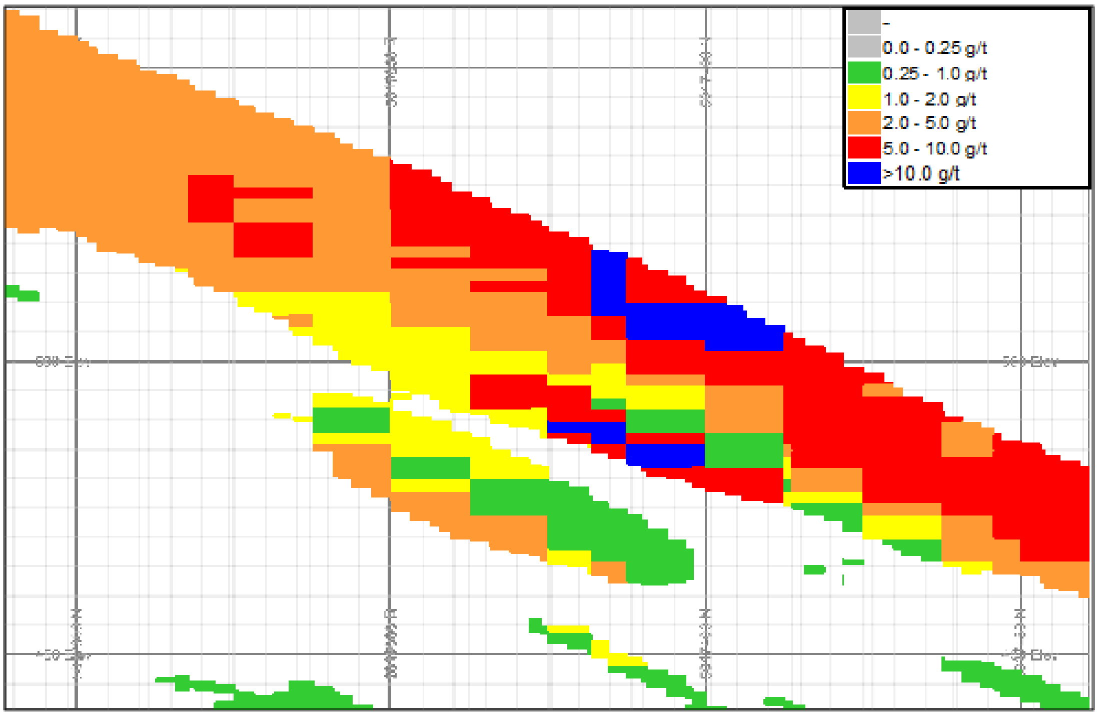

Figure 5 The undiluted in-situ resource

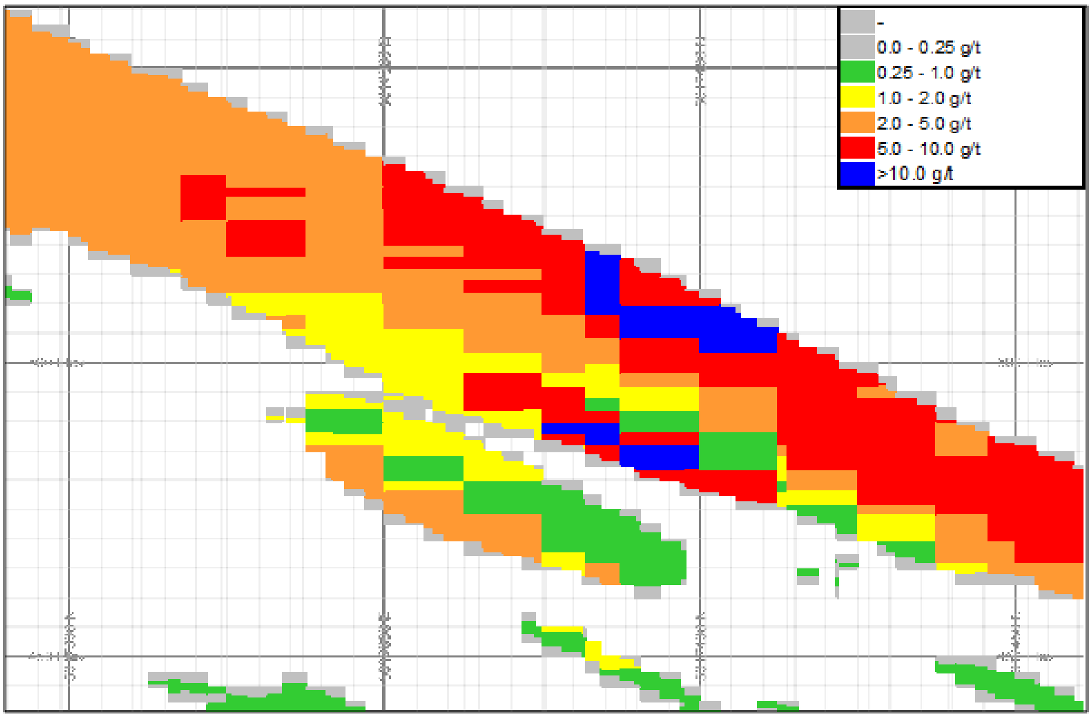

Figure 6 The mineral resource consolidated on a 2.5m bench

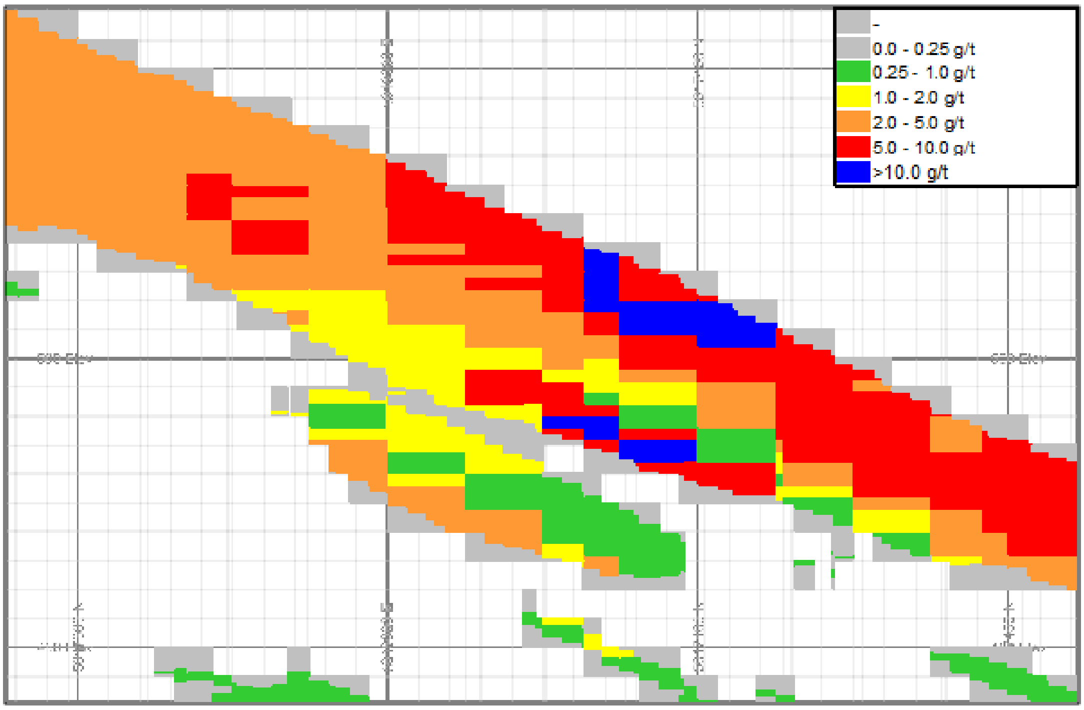

Figure 7 The mineral resource consolidated on a 5.0m bench

A summary of the amount of dilution based on each case is shown in Table 3. The 2.5m bench model is deemed appropriate to use for this Scoping Study.

Table 3 Summary of dilution applied

			2.5m Flitch	5.0m Flitch
Undiluted				
	Quantity	Mt	12.4	12.4
	Grade	g/t	2.5	2.5
	Metal	koz	1,004	1,004
Diluted				
	Quantity	Mt	14.0	15.8
	Grade	g/t	2.2	2.0
	Metal	koz	1,004	1,004
Dilution		%	**13%**	**28%**

Lerchs-Grossman (LG) Optimisation Inputs

The key inputs used for the analysis were:

Mineral Resource – The modified model described above has been used including all Indicated and Inferred Mineral Resources.

Gold Price – A base case US$1,650/oz was selected to determine mineable inventories (see Table 4). This gold price was selected after analysis of various prices between US$1,000/oz and US$2,000/oz as depicted in the following graph analyses (Figure 8, 9, 11, and 12).

NZD:USD Exchange Rate – medium to long term rate of 0.65 for the optimisation only.

Mining Cost – NZ$4.00/t @ 700mRL increased by NZ$0.05/t/10m increment.

Processing Cost – NZ$19.19/t processed.

General and Administration – NZ$5.00/t processed.

Metallurgical Recovery – Assumes a 0.35g/t tails grade which leads to a variable recovery.

Slope Design – 40° for all walls.

Optimisation Results

The results of the RAS LG analysis are shown in Figure 8 and Figure 9.

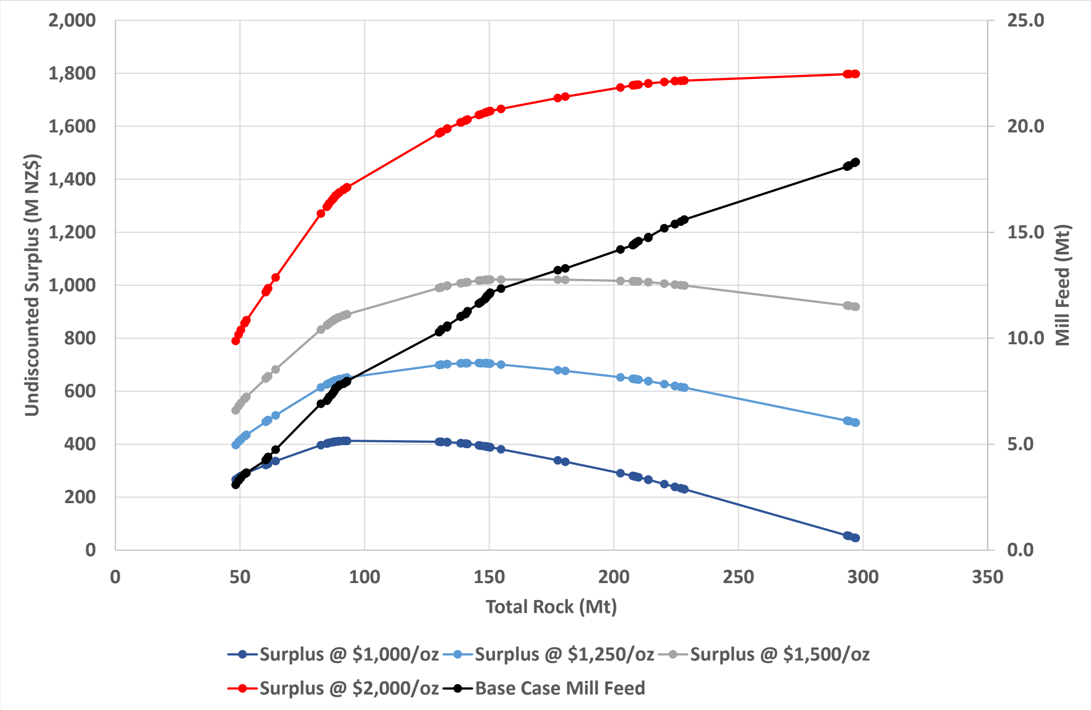

Figure 8 RAS LG Analysis – Undiscounted surplus and Mill feed.

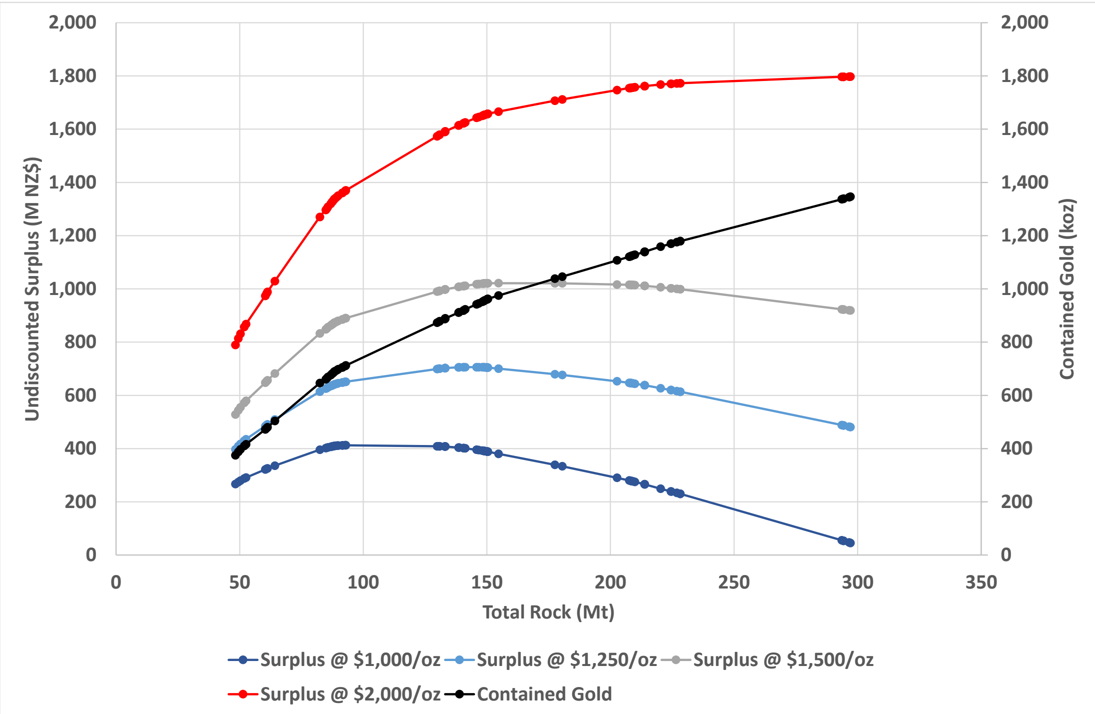

Figure 9 RAS LG Analysis – Undiscounted Surplus and Recovered Gold

The analysis showed distinct groups of shells:

- An initial set with around 50Mt of total rock, of which about 3.5Mt is mill feed delivering approximately 400koz of recovered gold. This material has a relatively high strip ratio, around 14:1, due to the initial mineralisation being located beneath the crest of the hill.
- This mineralisation has a relatively good grade of 3.5g/t. In the base case, this set of shells generates more than half the potential maximum undiscounted cash surplus.
- The second set of shell contains around 80Mt of total rock of which approximately 7.5Mt is mill feed delivering approximately 700koz of gold. The increment stepping here from the initial shell has a strip ratio of 9:1 but with a lower grade of 2.2g/t. In the base case this set of shells generates close to 25% of the potential maximum undiscounted cash surplus.
- The third set of shells contains around 140Mt of rock of with approximately 11.5Mt of mill feed delivering a little over 900koz of gold. This increment has a strip ratio of approximately 14:1 delivering a grade of 2.1g/t. In the base case this set of shells generates around to 15% the potential maximum undiscounted cash surplus.
- The fourth set of shells contains around 220Mt of rock, of which 15Mt is mill feed delivering around 1,100koz of gold. This increment has a strip ratio of 18:1, but the grade drops to about 1.8g/t. In the base case this set of shells generates no undiscounted cash surplus and given the high strip ratio it is likely to have a negative effect on the discounted surplus.
- The final set of shells contains a little under 300Mt of rock of which approximately 18Mt is mill feed delivering 1,350koz of gold. But this increment has a strip ratio of 24:1, and while the grade improves slightly to 1.9 g/t, in the base case this set of shells has a negative effect on the undiscounted cash.

The long section in Figure 10 shows selected shells from the groups identified.

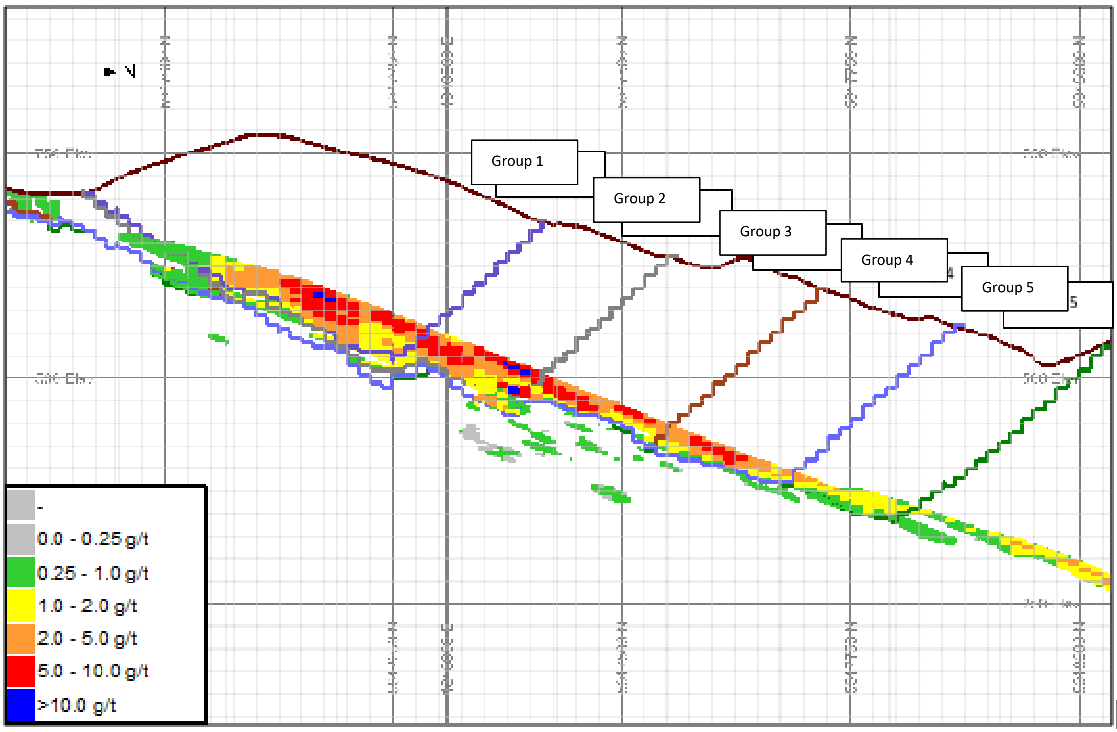

Figure 10 Long-section of the diluted model and shells.

The initial mineralisation under the top of the hill is higher grade and thicker than the deeper mineralisation.

This allows the initial high strip ratio required to access the mill feed to be mined and still generate a positive cash surplus. But then as the mineralisation dips toward the bottom of the valley, both the grade and the thickness reduce and then underground mining methods are considered.

The reduction in grade and thickness at depth reduces the quantity of gold and increase the strip ratio, leading to a reduced cash surplus when open pit mining.

RAS Sensitivity Analysis

Sensitivity analysis tested the effect on the results of changes in cost and slope angle. The results of the cost and slope sensitivities are shown in Figure 11 and Figure 12.

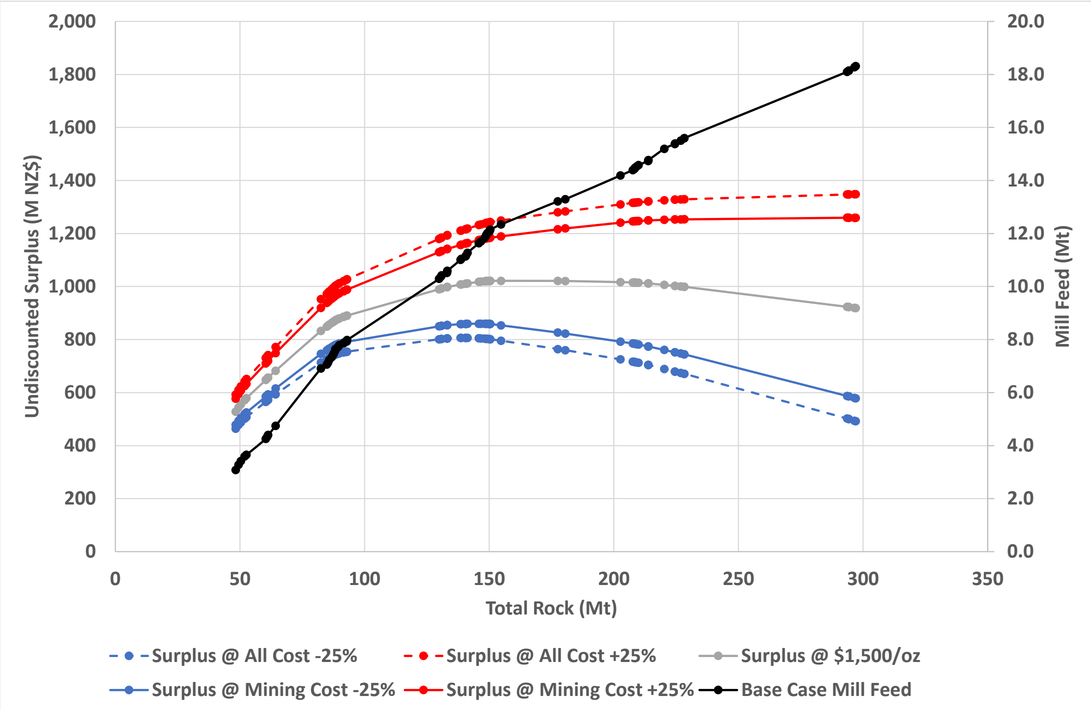

Figure 11 RAS LG analysis – Cost Sensitivity

As would be expected, the result is much more sensitive to changes in mining cost than treatment cost. With a strip ratio of over 10:1, mining cost per tonne of mill feed will be well more than $35/t compared to a processing cost of $24.19 (including G&A).

In the sensitivity analysis of the pit optimisation, pushing from a shell with 140Mt of rock to one with 220Mt adds little value, in the elevated cost case going to the larger shell clearly destroys value. But if lower costs can be achieved there is potential that the larger shell could become economically viable.

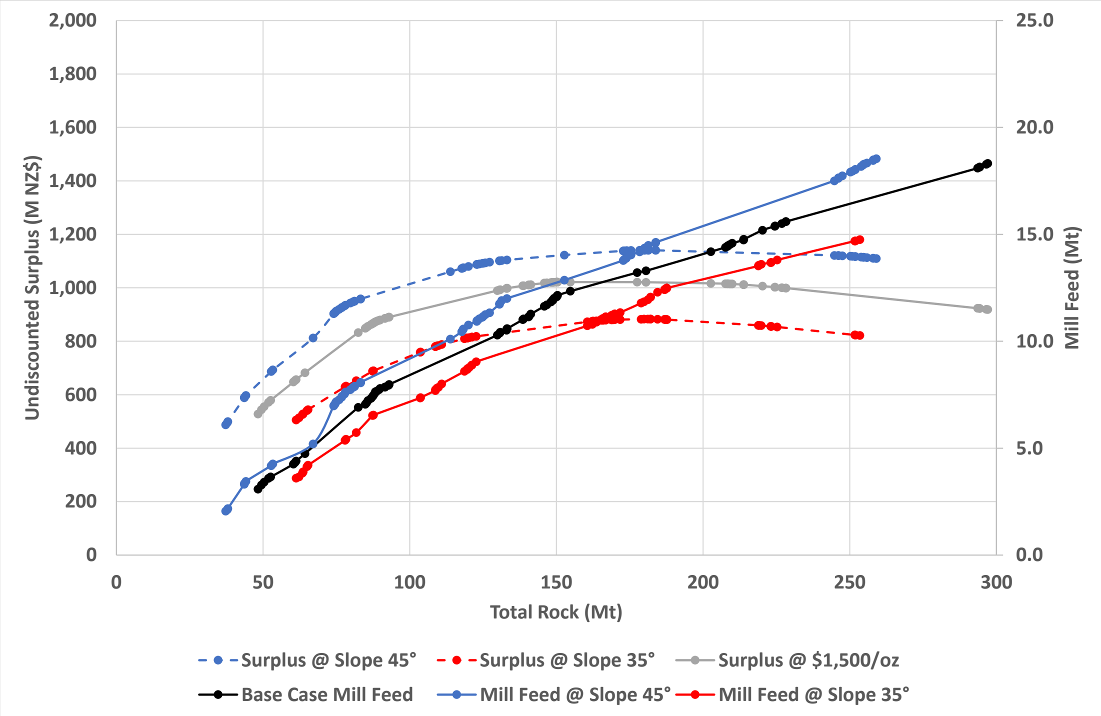

Figure 12 RAS LG Analysis – Slope Sensitivity

The slope angle graph indicates that the strip ratios for a pit including 20Mt of mill feed for the different slope angles are:

- 35° 14.0:1 t:t
- 40° 11.6:1 t:t
- 45° 10.1:1 t:t

Steepening of the slopes by 5° reduces the strip ratio by 8% potentially increasing the undiscounted surplus by over NZ$100M.

RAS LG Summary

The RAS results show that there is clearly a potential pit in the range of 140Mt to 150Mt of rock with about 12Mt of mill feed which is economically attractive and relatively insensitive to input gold prices. The expansion to 220Mt of rock with 15Mt of mill feed is possible, and the potential will be enhanced by efficient mining operations and good quality design and excavation of walls to allow the steepest, safest slopes possible to be mined.

Pit Designs

A series of four conceptual stages which approximately align with the 140Mt to 150Mt shells have been designed for use in scheduling. These are shown in Figure 13 to Figure 16.

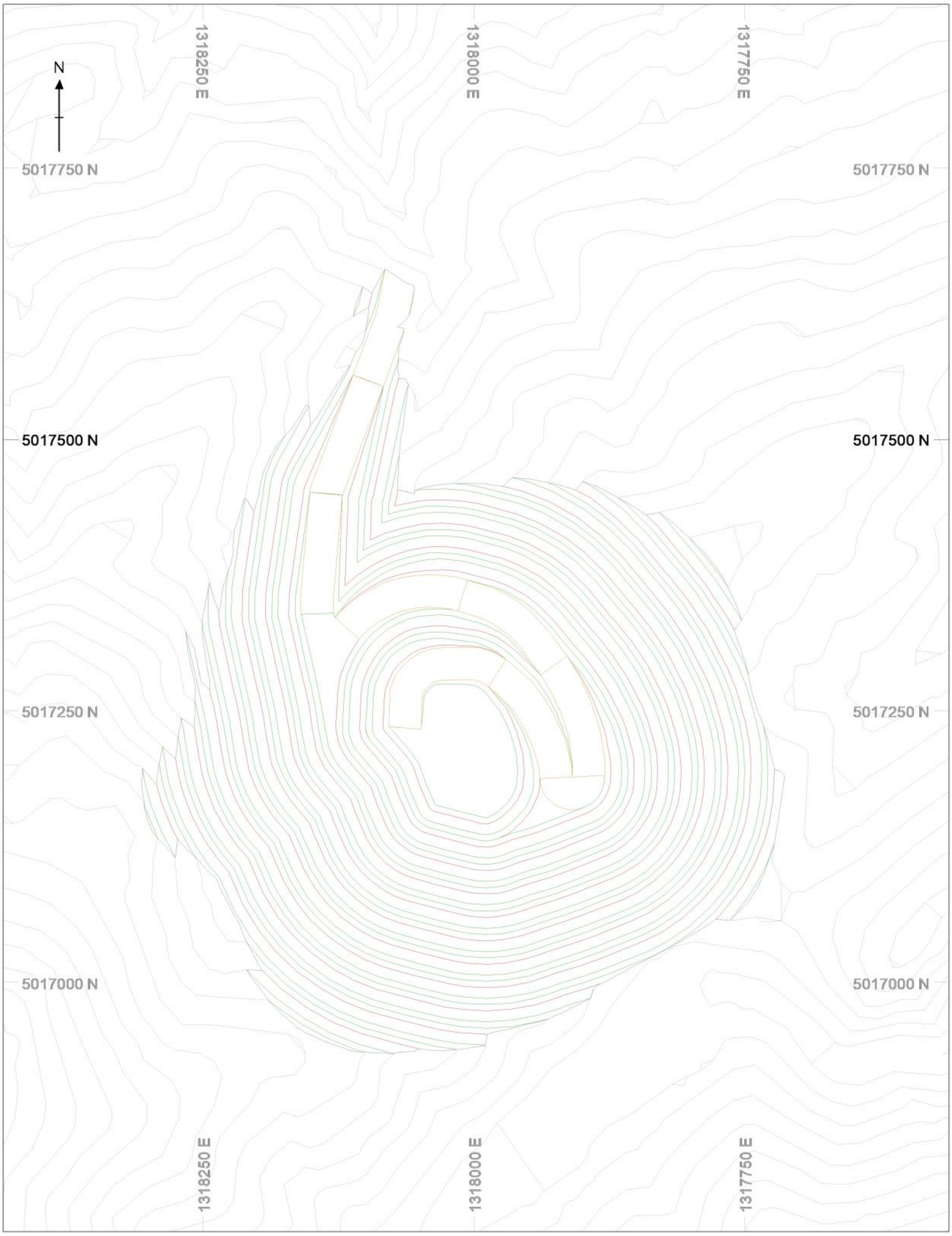

Figure 13 Stage 1

Figure 14 Stage 2

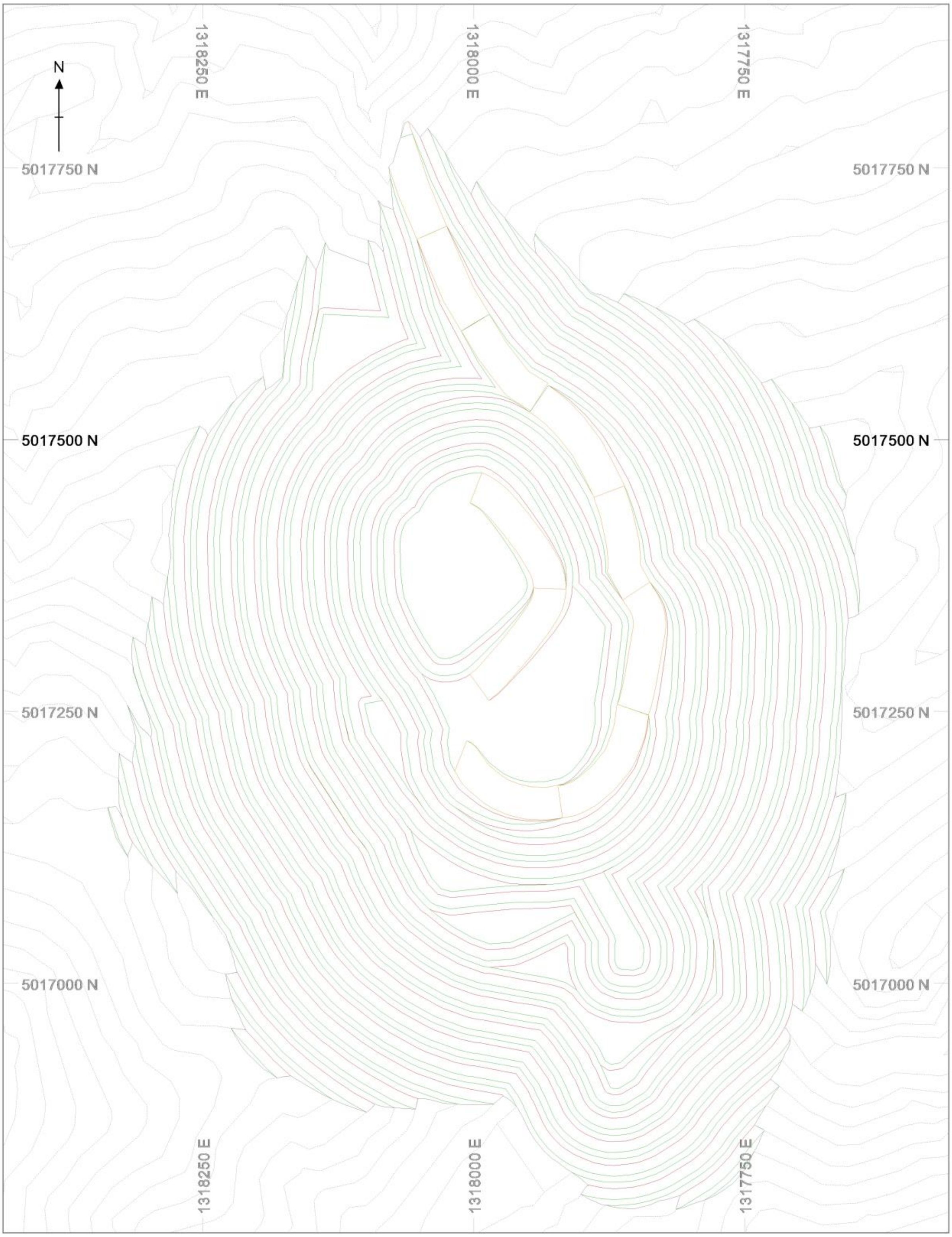

Figure 15 Stage 3

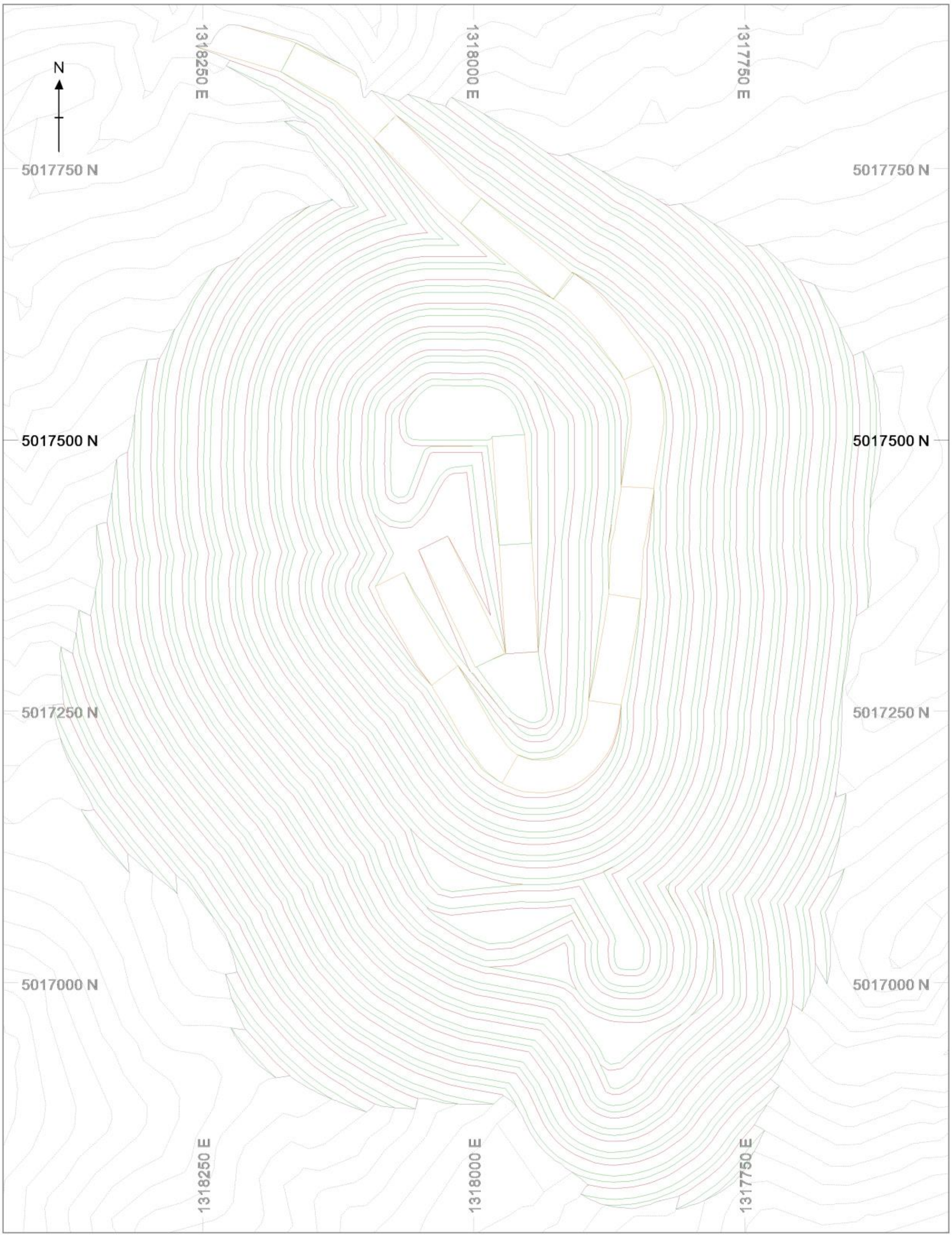

Figure 16 Stage 4

To minimise the waste movements all the pit exits are to the North, close to the lowest point on the pit rim.

The graph in Figure 17 compares the stage contents to the LG results graph, and long-section in Figure 18 compares the stage designs to Shell 51 which was used to guide the ultimate pit.

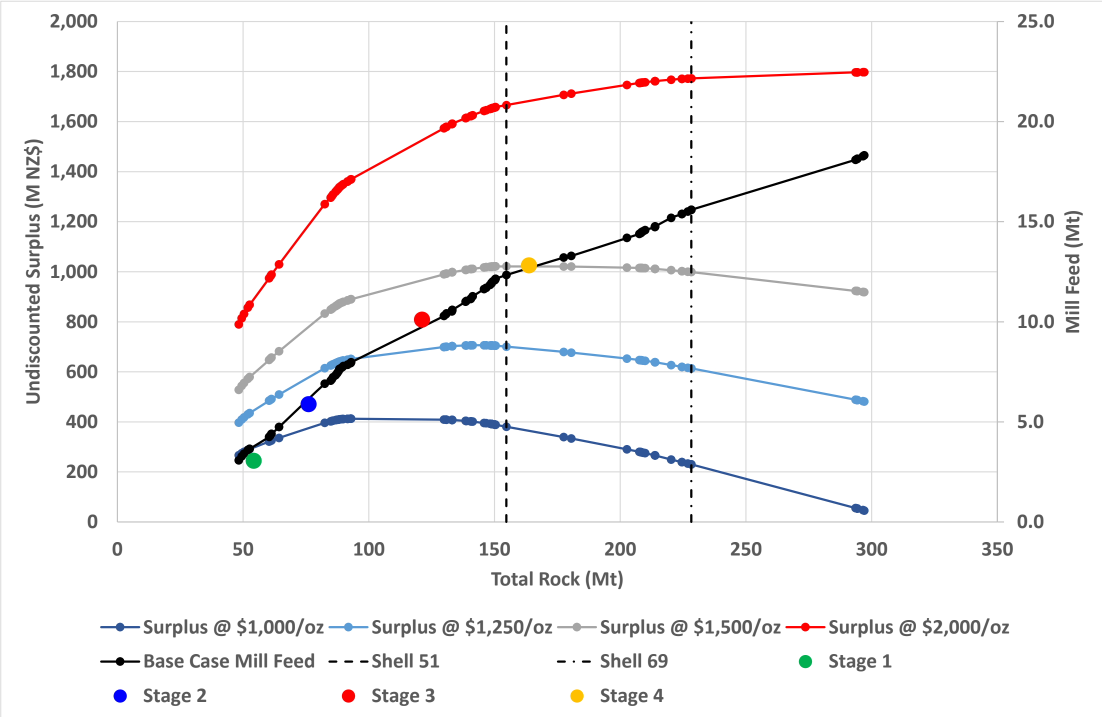

Figure 17 Designs vs LG Results

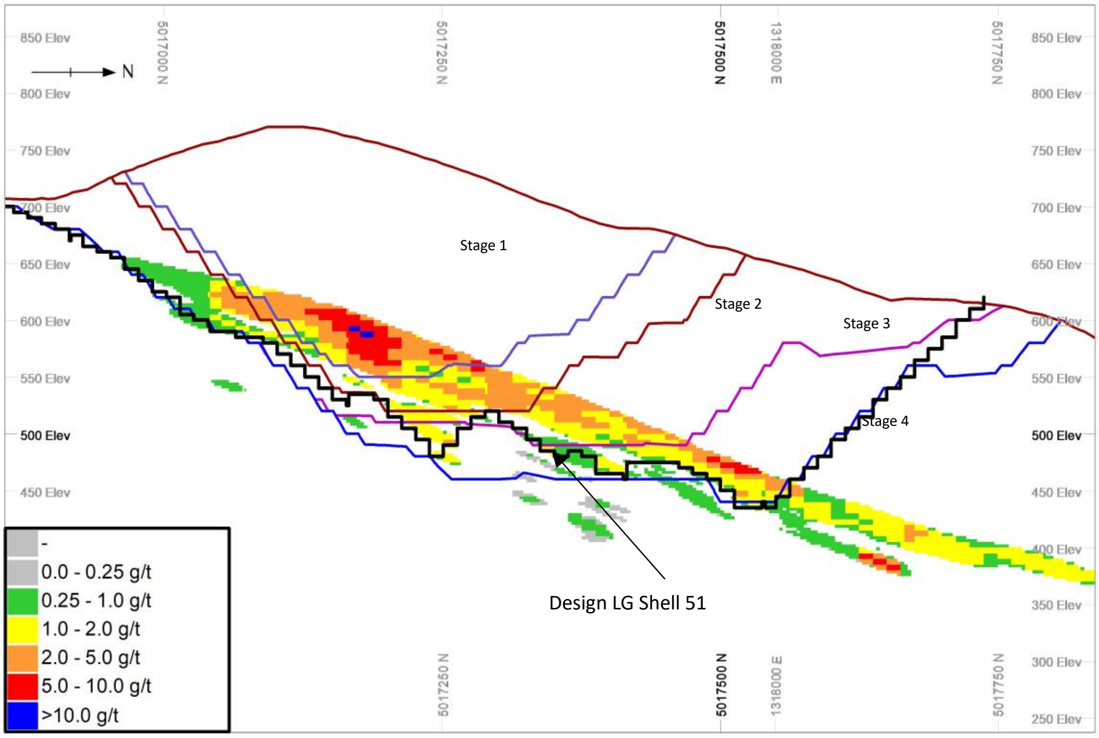

Figure 18 Long section through RAS showing Staged Pit Designs vs LG Shell 51

The stage designs agree acceptably with the LG results and are appropriate for this level of study.

Scheduling Inventories

The quantities contained in the stage designs and essentially in shell 51 are based on a 0.6g/t cut-off grade as shown in Table 4 with quantities broken down by Resource classification.

Table 4 Open Pit Scheduling Inventories

	Total Rock	Waste	Mill Feed	Strip Ratio[1]	Grade	Gold	Indicated	Inferred
	Mt	Mt	Mt		g/t	koz	% of Gold	
Increment								
Waste Pre-Strip	32.0	32.0						
Stage 1	20.9	17.9	3.0	6:1	3.3	321	89%	11%
Stage 2	21.9	19.2	2.7	7.1:1	2.3	202	84%	16%
Stage 3	45.1	41.2	3.9	10.6:1	2.1	263	78%	22%
Stage 4	42.6	40.1	2.5	16:1	2.3	185	84%	16%
Total[1]								
Stage 1	20.9	17.9	3.0	6:1	3.3	321	89%	11%
Stage 2	42.8	37.1	5.7	6.5:1	2.8	523	87%	13%
Stage 3	87.9	78.3	9.6	8.2:1	2.5	786	84%	16%
Stage 4	130.5	118.4	12.1	9.8:1	2.5	971	84%	16%

[1]Excludes 32Mt removed during the pre-strip

Table 4 shows that of the material contained in the initial four stages 16% of the gold is based on Inferred mineralisation which is mined only as a consequence of mining the Indicated resources material. Note that Stage 1 excludes the 32Mt pre-strip required to expose the mineralisation that can then be mined at a regular rate to supply the mill over the subsequent years. Including this in the total waste movement results in an overall strip ratio of 12.5:1.

The pictures in Figure 19 illustrate how the Inferred Resource is around the periphery of the mineralisation and a structure on the western flank of the deposit.

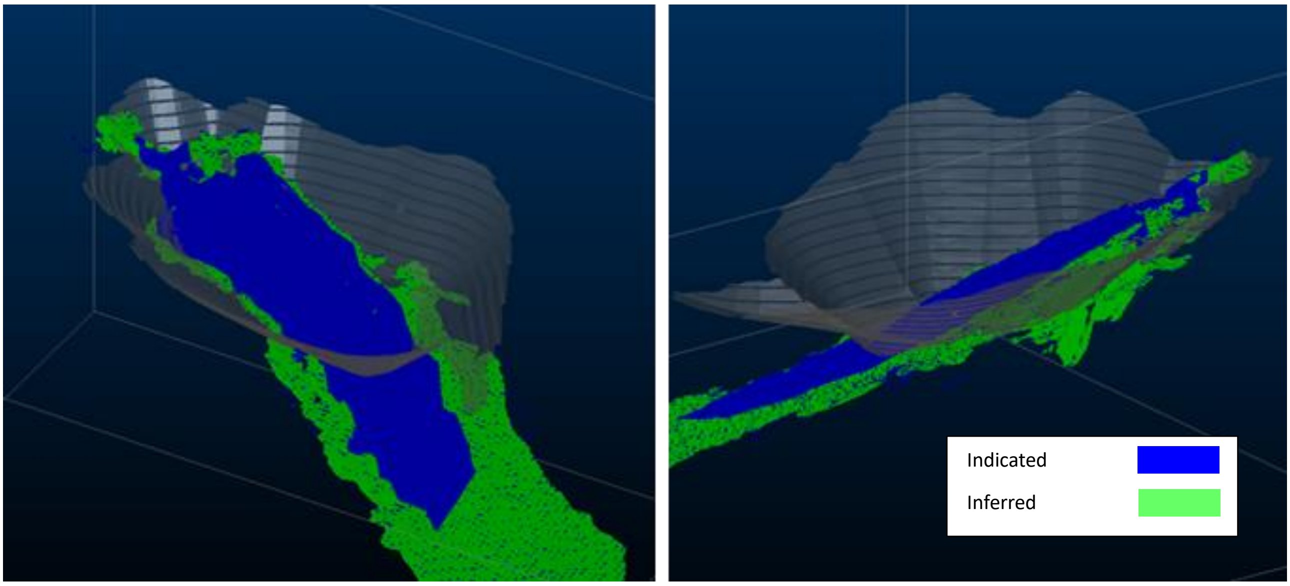

Figure 19 Resource classification against the ultimate pit design

Open Pit Mining Fleet

Contract mining has been assumed with the expected main fleet configuration shown in Table 5.

Table 5 RAS open pit mining fleet unit class

Primary Fleet	
Waste Excavator	EX2500
Ore Excavator	EX1200
Dump Truck	Cat 785
Dump Truck	Cat 777
Bulldozer	Cat D10
Front End Loader	Cat 995
Blasthole Rig	Pantera Top Hammer
Grader	Cat 16M
Water Truck	Cat 777

Underground Optimisation and Designs

Studies indicate underground mining is technically and economically feasible beyond the likely open pit economic limits.

The preferred mining method is Longhole Open Stoping (LHOS) with cemented paste backfill, mined in either a transverse orientation or longitudinal orientation.

The use of paste backfill increases mill feed recovery by minimising pillars required for an underground design, and mitigates unplanned dilution caused by over-break. It is considered to be economically viable for the preferred LHOS mining method.

The economic cut-off grade is estimated at 1.75g/t Au with cement paste backfill.

The open pit optimisation LG results indicate there is negligible incremental value using undiscounted operating surplus between the currently selected open pit size and any larger LG shell scenarios. This indicates the current pit limits may be the optimal transition from open pit to underground mining.

To fully understand the long-term potential of the underground operation, both Indicated and Inferred mineral resources were evaluated below the open pit. The selected base case for the underground mining option is:

- Mining method — LHOS with backfill
- In situ mining inventory — 7.7Mt @ 3.27g/t Au (811koz)
- Production inventory — 6.5Mt @ 3.12g/t Au (655koz)

Further analysis on the Indicated mineral resources only, concluded a viable underground mine still possible. Figure 20 illustrates the optimal mining scenario via open-pit mining method and transitioning to an underground operation.

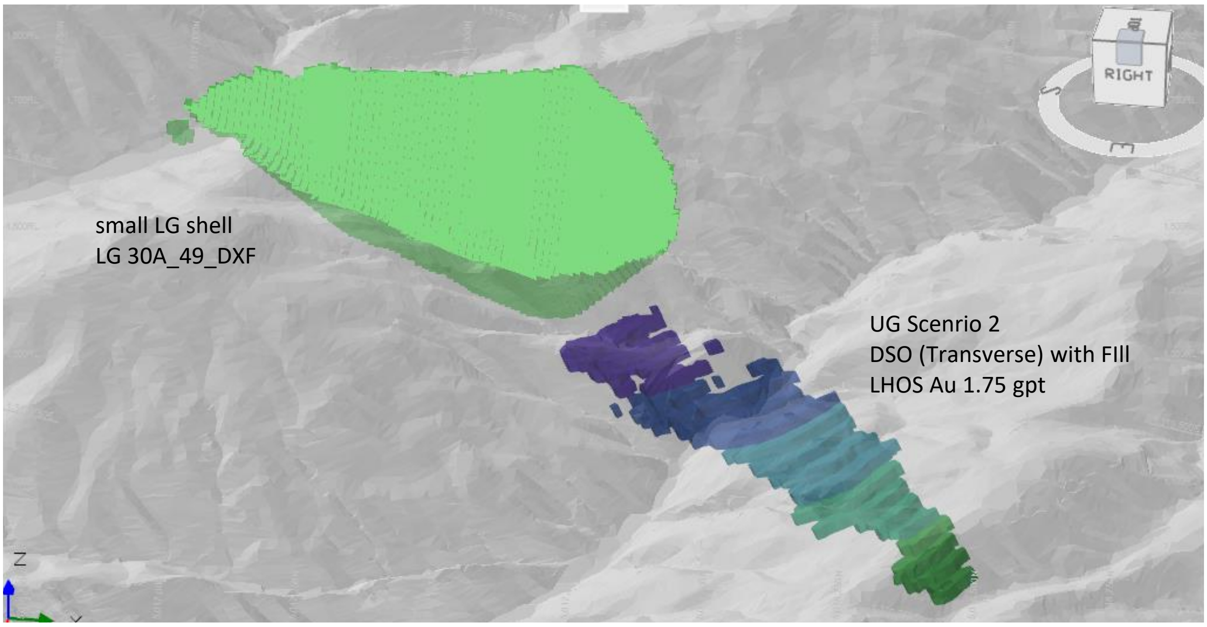

Figure 20 Schematic mining scenario with open pit transitioning to underground

Decline trucking is considered the most appropriate option for material movement due to deposit geometry and practical access.

Mining Method Analysis

In general, the rock mass conditions are not anticipated to have a material impact on the mining methods selected. A brief mining method ranking is summarized in 6.

Table 6 Underground mining methods applicability to RAS

Mining Methods	RAS Deposit	Suitability ranking
Caving Methods		**5 = highest**
Block Caving	Not suitable.	N/A
Sub-level Caving	Not suitable due to deposit geometry, i.e. shallow 23-degree dip.	N/A
Core and Shell	Not suitable.	N/A
Open Stoping Methods		
Sub-level Open Stoping	Where the deposit is vertical to subvertical, sublevel stoping will be considered in preference due to lower cost. Not suited for deposit geometry.	1
Longhole Open Stoping	Feasible - moderately dipping deposit (single lift with / without fill).	4
Benching / Avoca / Modified Avoca	Potentially technically feasible - moderately dipping deposit. Not ideally suited for deposit geometry.	2

Mining Methods	RAS Deposit	Suitability ranking
Shrinkage / Vertical Retreat	Labour intensive, intrinsically unsafe. Not suited for deposit geometry.	N/A
Drifting Methods		
Room and Pillar	Suitable but high cost. This method is suitable for flat to sub-horizontal deposits.	3
Post Pillar Cut-and-fill	Suitable and high cost. This method is suitable for sub-horizontal to moderate deposits.	3
Mechanized Cut-and-fill	This method is suitable for vertical through to flat deposits	3
Drift and Fill	This method is suitable. High unit mining cost and low productivity.	3

The mining methods technically feasible for consideration for a thick flat-to-moderate dipping deposit like RAS, would comprise of:

- Long hole open stoping with cemented paste fill (bottom-up, primary-secondary).
- Long hole open stoping without backfill (top-down, uphole retreat with pillars); and
- Mechanized cut-and-fill with cemented paste fill.

The deposit geometry is very similar to the Frasers Underground mine at Macraes in NZ just 90km from the BOGP. This operation has successfully used longhole open stoping but without backfill. The lower grade of that deposit meant that ore is left in yielding pillars versus the higher cost of introducing cemented paste backfill.

The higher grade of RAS is likely to support cemented paste backfill introduction to achieve a higher resource recovery.

There is no potential benefit likely via mechanised cut and fill over the open stoping options with or without fill as used currently at the Macraes operation.

Cut-off Grade Estimate

The underground mining breakeven cut-off grade estimates by mining method, for the method assessment, are summarised in Table 77.

Table 7 Breakeven cut-off grade estimate by mining method

Item	Unit	LHOS with Fill	LHOS without Fill	Source
Mining cost	A$/t	83.2	75.2	AMC Benchmark
Milling cost	A$/t	20	20	Estimate, LG input
Admin cost	A$/t	4.7	4.7	Estimate, LG input
Total site cost	A$/t	107.9	99.9	-
Milling recovery	%	88	88	Assumption, LG input
Au price	US$/oz	1.500	1,500	Assumption, LG input
Exchange rate	$A:US	0.66	0.66	-
Au price	A$/oz	2,273	2,273	-

Item	Unit	LHOS with Fill	LHOS without Fill	Source
Conversion	oz/g	31.1	31.1	-
Au price	$/g	73.07	73.07	-
Return at gate	%	95	95	Royalty 4.5%, TC/RC assume 0.5%
Return at gate	A$/g	69.42	69.42	-
Cut-off grade	g/t Au	1.77	1.64	-
Cut-off grade (Use)	g/t Au	1.75	1.65	-

The breakeven cut-off grades for the selected mining methods were estimated to be:

- LHOS with Fill 1.75 g/t Au. Sensitivity ±0.25 g/t Au (±15%)
- LHOS without Fill (Pillars) 1.65 g/t Au. Sensitivity ±0.20 g/t Au (±15%)

Production Rate Assessment

Using the nearby Frasers Underground as an analogy, the production rate with a similar deposit geometry and tonnes per vertical metre is estimated for this study at 800ktpa.

Mineable Shape Optimisation

A 20m offset (exclusion) zone was applied on the working MSO resource model to provide for a crown pillar between the open pit and the underground.

The MSO parameters used in shape generation are summarized in Table 8.

Table 8 MSO parameters by selected mining method

Parameter	Unit	LHOS with Fill	LHOS with Fill	LHOS with Fill	LHOS with Fill
Orientation		Transverse	Longitudinal	Transverse	Longitudinal
Cut-off	g/t	1.75	1.75	1.65	1.65
Geometry					
Level Spacing	m	15	variable	15	variable
Length	m	variable	20	variable	25
Width	m	15	15	20	20
Height	m	15	variable	15	variable
Minimum width	m	5	7.1	5	7.1
Maximum width	m	200	100	200	100
Split	m	15	N/A	15	N/A
Waste Pillar	m	10	5	10	5
Dilution					
HW Dilution	m	0.5	0.5	0.5	0.5

Parameter	Unit	LHOS with Fill	LHOS with Fill	LHOS with Fill	LHOS with Fill
FW Dilution		0.25	0	0.25	0
Trough Position	m	N/A	Left	N/A	Left

The results of the MSO (physicals including Inferred resources) and an economic evaluation showed a clear advantage of introducing cemented paste backfill to increase the conversion of insitu inventory to production inventory.

Table 9 Physicals and cashflow surplus for the short-listed options

Parameter	LHOS with Fill	LHOS with Fill	LHOS with Fill	LHOS with Fill
	Transverse	Longitudinal	Transverse	Longitudinal
Insitu Inventory	7.7Mt @ 3.27g/t for 811koz	6.3Mt @ 3.14 g/t for 639koz	7.0Mt @ 3.47 g/t for 780koz	4.2Mt @ 3.41 g/t for 459koz
Production Inventory	6.5Mt @ 3.12g/t for 655koz	5.3Mt @ 2.93 g/t for 502koz	5.9Mt @ 3.3 g/t for 629koz	3.5 Mt @ 3.17 g/t for 361koz
Undiscounted Surplus	$A362M	$A248M	$A375M	$A162M

Preferred option

At this stage of the Study, the option selected is LHOS with cemented paste backfill in a transverse layout as it provides a higher level of certainty in layout geometry and ore-drive gradients. Subsequent to the method selection, only Indicated resources were included in the final financial analysis. Some Inferred resources (16% of the total) are mined as a consequence of accessing the underground Indicated resources.

Figure 21 illustrates the LHOS with backfill (transverse) MSO shapes at the selected cut-off (1.75g/t Au) against two LG optimisation shells.

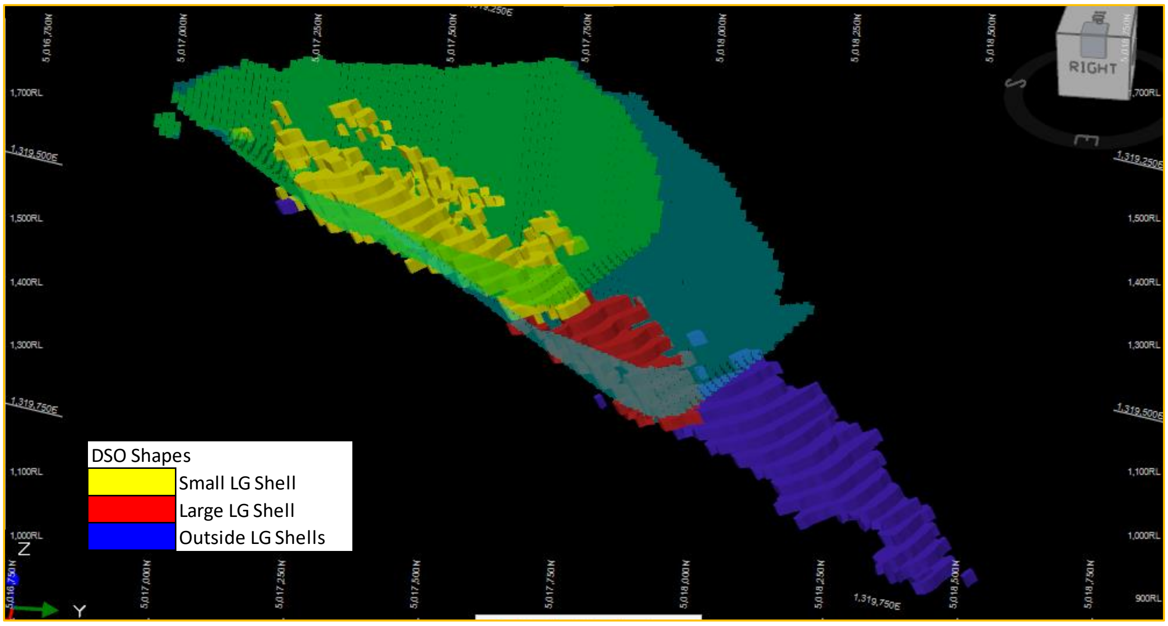

Figure 21 Schematic of the MSO Shapes - Isometric – LHOS with Fill (Transverse) 1.75gpt Au (red and blue shapes)

A haulage decline via portal access in the Shepherds valley will provide access to the underground operation.

Figure 22 shows a long-section view of the open pit and the down dip underground mining.

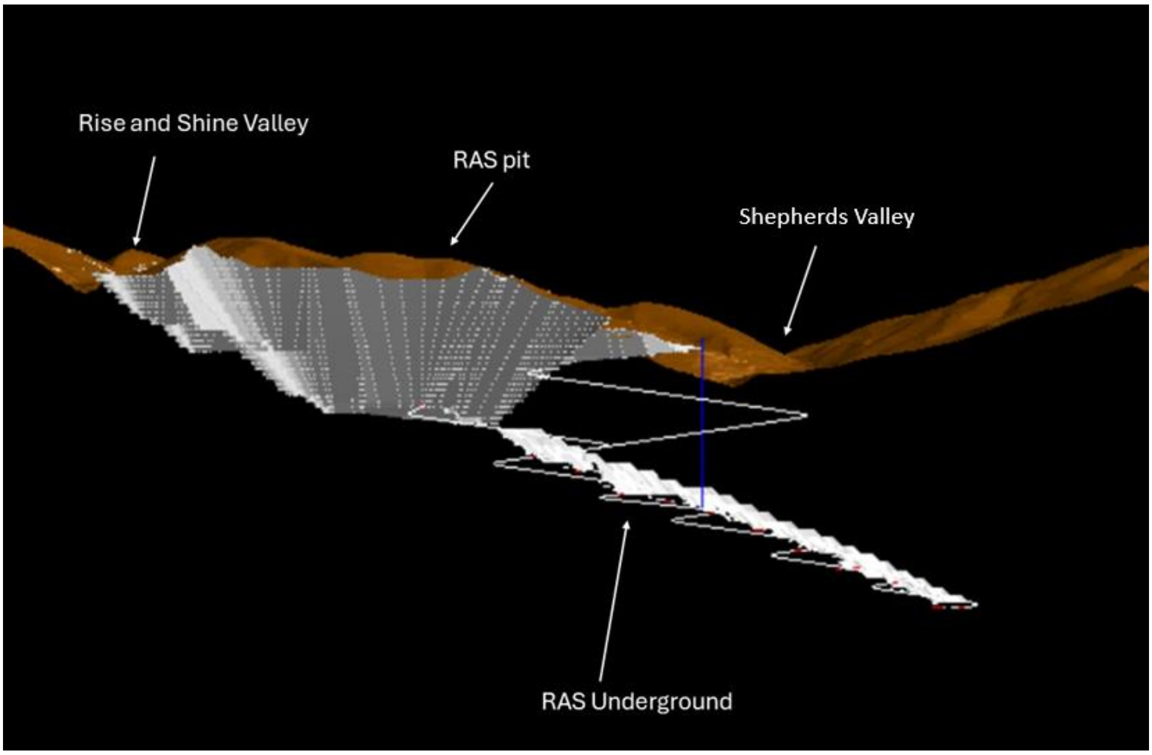

Figure 22 Schematic long section view of the open pit and underground

Heading Sizes and Fleet

The heading sizes chosen are to suit 50 tonne class articulated haul trucks and 17 tonne capacity Load-Haul-Dump (LHD) units and are as shown in Table 10.

Table 10 Key underground development profiles

Profile	Size	Type	Examples
A	5.5mw x 5.5mH	Lateral	Decline, Level Access, Stockpiles
B	4.5mw x 4.5mH	Lateral	FWD, X-cuts, vent drives, ore-drives
C	3.5m Dia	Vertical	RAR, FAR
D	0.75m Dia	Vertical	Egress
E	0.15m Dia	Vertical	Services

The corresponding mining fleet is as per Table 11.

Table 11 RAS underground mining fleet unit class

Primary Fleet	
Loader	Sandvik LH517
Haul Truck	Sandvik TH551
Development Drill	Sandvik DD422i
Production Drill	Sandvik DL431
Ancillary Fleet	
Cablebolt Drill	Sandvik DS421
Explosive Charging Unit	Normet Charmec MF605D
Shotcrete Sprayer	Normet Spraymec 6050W
Agitator Truck	Normet Utimec LF700 Agitator
Service Truck	MacLean BT3
Water Truck	Elphinstone WF810
Grader	CAT 12M
Light Vehicle	4x4 pick-up trucks
Integrated Tool Carriers	CAT 930K

Underground Physicals

Table 12 Underground mining schedule

		Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Total
Development							
Lateral Development							
Decline	metres	320	426	426	426	506	2,104
Level Access	metres	345	460	460	460	546	2,271
FWD Xcut	metres	-	220	661	979	979	2,839
Ore Xcut	metres	-	332	997	1,477	1,477	4,283
Total	metres	665	1,439	2,544	3,342	3,508	11,497
Vertical Development							
RAR	metres	45	60	60	60	71	296
Egress	metres	79	71	60	60	71	341
Services	metres	45	60	60	60	71	296
Total	metres	169	191	180	180	214	933
Production							
Tonnes	kt	-	180	540	800	800	2,320
Grade	g/t Au	-	2.56	3.00	3.33	3.16	3.13

Life of Mine (LoM) Schedule

The mining schedule commences with open pit supply feeding at a processing rate of 1.5Mtpa followed by underground mining at 800ktpa.

The open pit mining rate has a maximum movement of 20Mtpa of total rock. All open pit mining is completed by year 9. Figure 23 shows the mining rate by year, by stage.

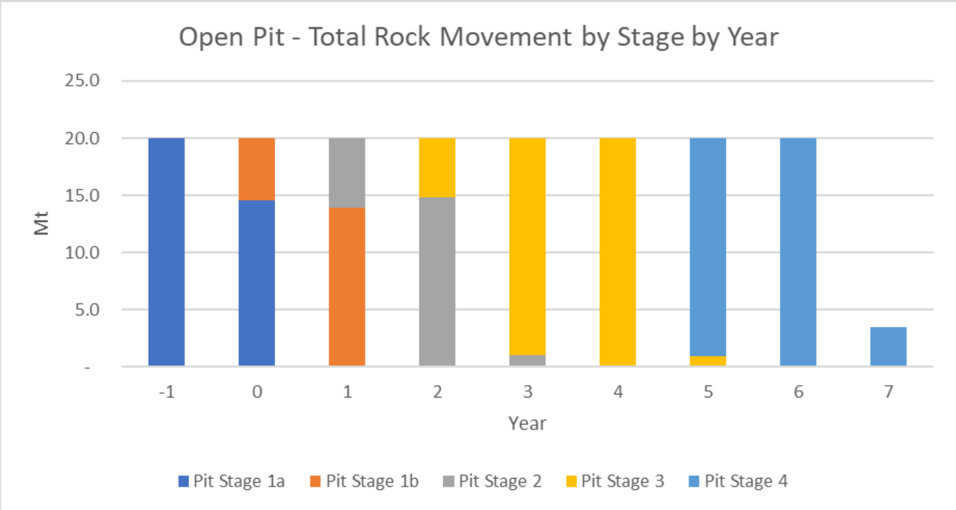

Figure 23 Total open pit rock mined by stage by year

The underground operation is staged to commence production as the open pit is nearing completion. Figure 24 shows the open pit from years 1-7 and underground mill feed commencing in year 7 till 10.

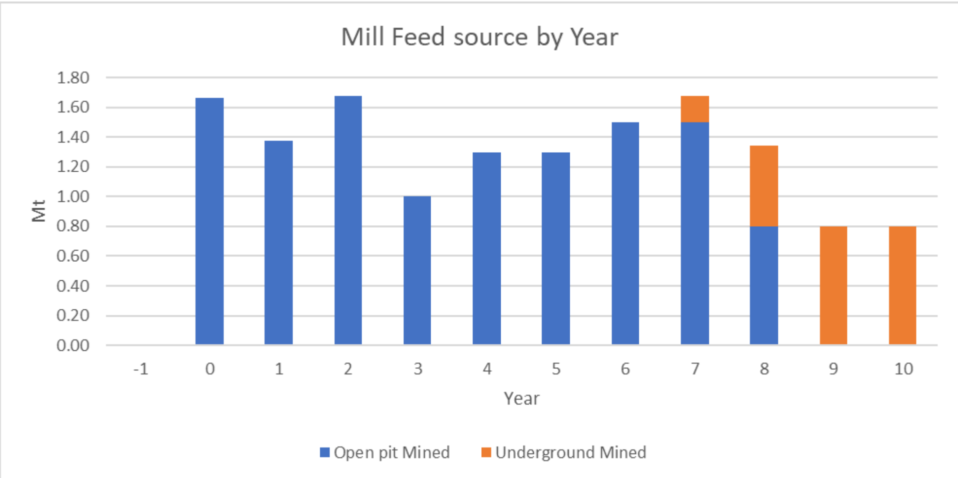

Figure 24 Mineral Resource mining by source

84% of the ounces mined from the open pit are from Indicated resources. The full life of mine schedule is shown in Figure 25.

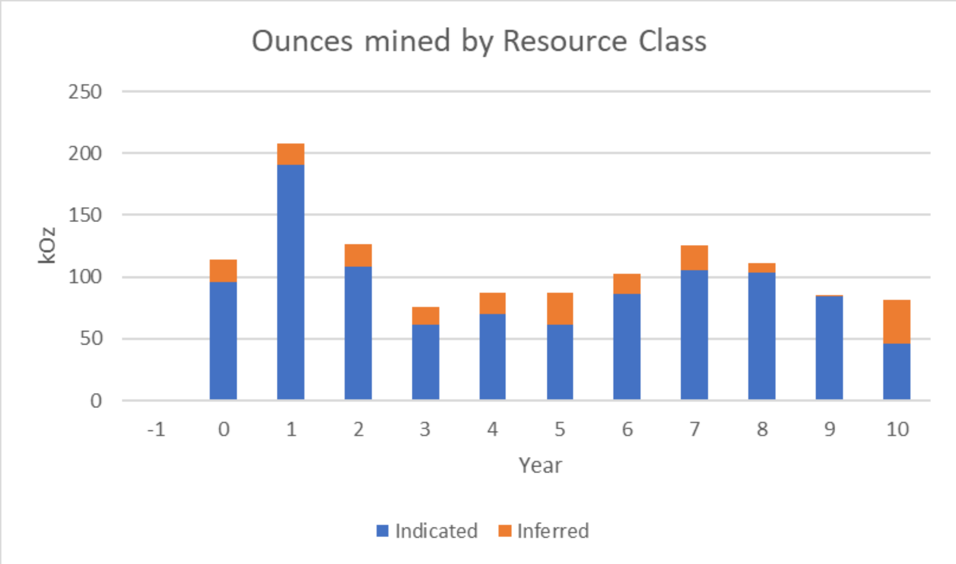

Figure 25 Ounces mined per year by resource classification

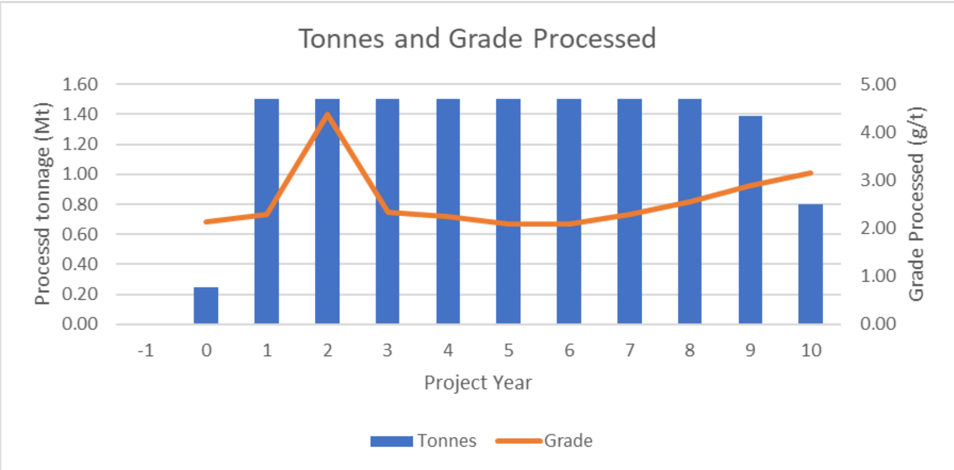

Figure 26 Processed tonnes and grade by year

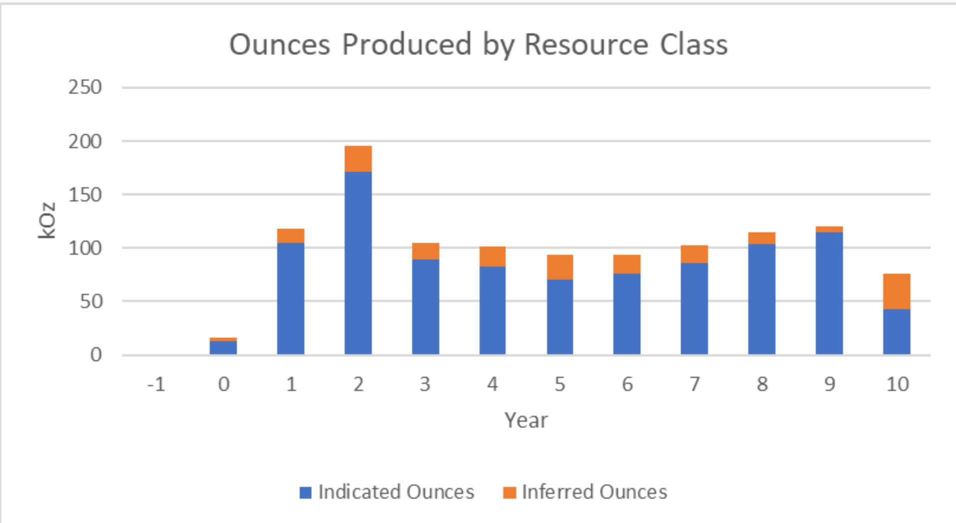

Figure 27 Ounces Produced by Resource Class per year

Table 13 Summary LOMP schedule physicals

Mining	Units	LOMP Total	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Open Pit														
Total Rock Movement	Mt	**163.5**	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	2.3	1.2	-	-
Mill feed tonnage- Mined	Mt	**12.1**	0.00	1.66	1.38	1.68	1.00	1.30	1.30	1.50	1.50	0.80		
Mill feed grade- Mined	g/t	**2.49**	0.00	2.13	4.70	2.34	2.34	2.09	2.09	2.12	2.30	2.30		
Underground														
Mill feed tonnage - Mined	Mt	**2.3**									0.18	0.54	0.80	0.80
Mill feed grade- Mined	g/t	**3.13**									2.56	3.00	3.33	3.16
Waste rock development	Mt	**0.5**	-	-	-	-	-	-	-	-	0.10	0.17	0.22	0.24
Processing														
Tonnes Processed	Mt	**14.4**	0.0	0.3	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.4	0.8
Grade Processed	g/t	**2.59**	-	2.13	2.28	4.36	2.34	2.25	2.09	2.09	2.28	2.55	2.90	3.16
Ounces - feed	koz	**1,204**	0	17	110	210	113	109	101	101	110	123	129	81
Ounces - recovered	koz	**1,120**	0	16	102	196	105	101	94	94	102	115	120	75

Metallurgy and Processing

Metallurgy

Test work samples of each lithology type were selected from mineralised intersections located within the RAS deposit.

The Master Composite targeted a gold grade of 2.50g/t Au with mixed lithology taken from 12 intervals of 2.5m each.

Table 14 Master Composite Summary

Hole ID	From	To	Interval	Weight	Au	As
	m	m	m	kg	g/t	ppm
MDD204	19	21.5	2.5	8.65	2.47	5,226
MDD171	174	176.5	2.5	8.30	5.63	3,421
MDD178	159	161.5	2.5	8.10	3.39	2,757
MDD138	163.5	166	2.5	8.80	1.13	6,657
MDD061	151.5	154	2.5	7.40	1.12	3,663
MDD081	164	166.5	2.5	8.15	6.29	2,435
MDD081	180	182.5	2.5	9.45	2.97	1,368
MDD086	159	161.5	2.5	9.95	1.16	5,297
MDD086	169	171.5	2.5	7.75	2.95	1,855
MDD080	196	198.5	2.5	7.90	1.73	4,758
MDD160	175	177.5	2.5	8.85	1.16	0
MDD165	144.5	147	2.5	9.10	1.21	1,897
Master Composite			30	102.4	2.56	3,284

Ten variability samples were selected to represent spatial and grade variability. Lithology was mixed with seven intervals of 2.5m each selected for each sample.

The test work for these samples includes:

- Head assay analysis.
- Comminution including SMC, Crusher Work Index (CWi), Bond Ball Mill Work Index (BWi), Bond Abrasion Index (Ai) for Master Composite and variability samples.
- Gravity and gravity tails direct cyanidation on Master Composite for three grind sizes (150, 106 and 75 µm) and for reagent optimisation;
- Sighter Flotation test work on Master Composite for pyrite flotation.

The location of the diamond drill holes used in the metallurgical test work are shown in Figure 28.

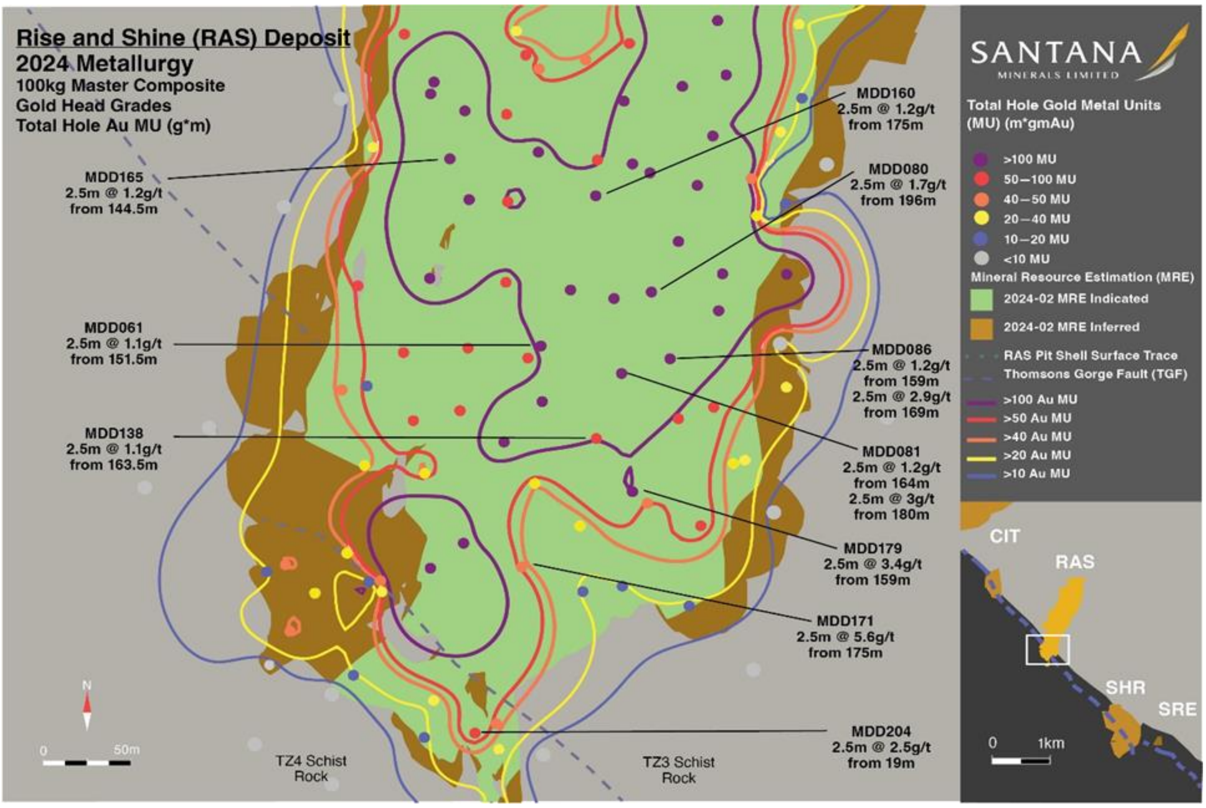

Figure 28 Plan of RAS showing locations of drill holes intervals that formed the Master Composite.

Key observations from the testwork are noted in the following sections.

Head Assay

- Head assays in the variability samples ranged from 0.44 to 13.71g/t Au.
- The Master Composite sample assayed between 2.61 and 2.91g/t Au, compared to design LoM grade of 2.70g/t Au.
- Significant variation in gold assays indicates coarse gold in the sample. This is verified by high gravity recovery.
- Arsenic and sulphides concentrations show a strong correlation, however gold and sulphide grade do not, suggesting that the gold is not associated with sulphide and therefore unlikely to be refractory. This is confirmed by the flotation and cyanidation results.
- The total carbon tested on the samples are at a level where preg-robbing may occur, this is confirmed by the leach testwork.

Comminution Testwork

SMC and Bond Work index tests were carried out on the Master Composite and variability samples. The results indicate that the mill feed has the following characteristics:

- An Abrasion index of 0.3077, indicating the mill feed exhibits moderate abrasivity.
- Crusher Work index reported an average work index value of 5.15 kWh/tonne.
- A Bond Ball Work index of 19.0 kWh/t, categorising the mill feed as hard.
- SAG Circuit Specific Energy values ranging from 8.52 to 10.75 kWh/tonne, with the Master Composite having a value of 9.18 kWh/tonne, indicating that medium to high power input is required for grinding.

Grind Optimisation

Grind optimisation was carried out on the Master Composite sample at grind sizes of 150, 106 and 75µm. The deposit showed increasing recovery at finer grind sizes, with 24-hour gold recoveries of 92.6%, 93.6% and 93.7 %, respectively.

A grind optimisation exercise resulted in the selection of 106µm as the optimal grind size to maximise project economics.

Gravity Gold Recovery

Gravity gold recovery at laboratory scale has indicated that gravity recoverable gold is present with recoveries of between 57.0% and 64.7%. The flowsheet will be designed to maximise the recovery of gravity gold ahead of the leaching process.

Gravity recovery of 31.3% has been selected as the design value, with the leach circuit design able to accommodate zero gravity recovery.

Leach Behaviour

The leach behaviour over 48 hours has indicated that a preg-robber may be present, this is currently being tested. Leach recoveries decline between 0.5% and 1% after 8 hours, and up to 48 hours, a carbon-in-leach environment will typically prevent this style of preg-robbing. A preliminary CIL indicated a recovery of 93.3 % after 24 hours at 150µm.

Overall gold recovery for the purposes of this scoping study is 93.0%.

Flotation Recovery

The Master Composite was tested in a standard pyrite flotation regime to establish if a flotation concentrate could be generated. Gold recoveries of 45.6% and 52.3% were achieved in the two tests. No further flotation work is planned.

Reagents

The Master Composite testwork value for cyanide consumption is 0.38 kg/t plant feed. The plant design will allow up to a 50% increase on this value.

The Master Composite testwork value for lime consumption for leaching is 0.08 kg/t plant feed. The testwork was done using Perth tap water, which is not representative of site water, however the site water is fresh and has low salinity.

Oxygen Uptake

Oxygen uptake tests have not been conducted, one leach test at 105µm indicated that oxygen addition did not have a substantial effect. The presence of preg-robbers together with the apparent low oxygen consumption suggests that oxygen sparging is not required.

Metallurgical Testwork Gaps

The following further testwork covering all lithology domains and deposit depths throughout the entire mine pit shell is recommended for progress to a definitive feasibility level of study:

- CIL optimisation at 106µm.
- Cyanide concentration optimization 106µm.

- Pulp viscosity testing for agitator sizing.
- Thickener flocculant screening and dynamic settling tests in site water are required for sizing of the tailings thickener.
- Filtration testwork to assess the viability of dry stack tailings.
- Tailings solution cyanide speciation and potential cyanide detoxification.
- Tailings characterisation.

Processing Design Criteria

Design criteria have been prepared to provide the key design parameters for equipment selection and engineering for a single-stage crush, single-stage grinding option. The design criteria incorporates the main details for the mill feed and the processing plant. A summary of the key design criteria is provided in Table 15.

Table 15 Processing key design criteria

Parameter	Units	Value
Operating Schedule		
Annual Throughput	t/a	1,500,000
Crusher Plant capacity	t/h	428.1
Grinding Plant capacity	t/h	187.5
Design Feed Grade - Gold	g/t	2.70
Design Gold Recovery	%	95.6
Design CIL Recovery	%	93.6
Design Gravity Recovery	%	31.3
Nominal Gold Production	kozpa	83.8
Physical Mill Feed Characteristics		
Mill feed Source		RAS open pit and underground
Bond Ball Work Index - design	kWh/t	19.8
Crushing		
Circuit Type		Primary crushing
Primary Crusher		Jaw
Feed Size F100	mm	700
Product Size P80	mm	121
Grinding		
Circuit Type		Single Stage SAG Mill
Feed Size F80	mm	121
Product Size P80	µm	106
Grinding Mill Power Installed	kW	5,100
CIL Circuit		
No of Tanks	#	6
Adsorption Circuit volume total	m^3	7,440
Adsorption Circuit residence Time	h	24
Elution and Electrowinning		
Carbon Elution Process		Split AARL
Design Capacity (Carbon)	t	6.0
Carbon Regeneration		

Parameter	Units	Value
Reactivation Kiln Type		Electric
Capacity	kg/h	350

Process Description

The Bendigo-Ophir processing plant has been designed based on processing 1.5Mtpa of gold mill feed at a feed grade of 2.70g/t Au.

The design crushing throughput rate is 428t/h, equating to 80% availability (day shift operation only).

Design milling rate is 187t/h based on availability of 91.3%. The following process plant description is based on the Process Design Criteria and flowsheets. The processing circuit includes the following major equipment areas:

- Primary jaw crusher.
- Cyclone classification.
- Milling and gravity separation.
- Hydrocyclone classification.
- Gravity concentration and intensive leaching of gravity concentrate.
- Leach and adsorption.
- Cyanide destruction.
- Elution circuit and carbon regeneration.
- Tails thickening.
- Services and reagents.

A summary process flow diagram is presented in Figure 29:

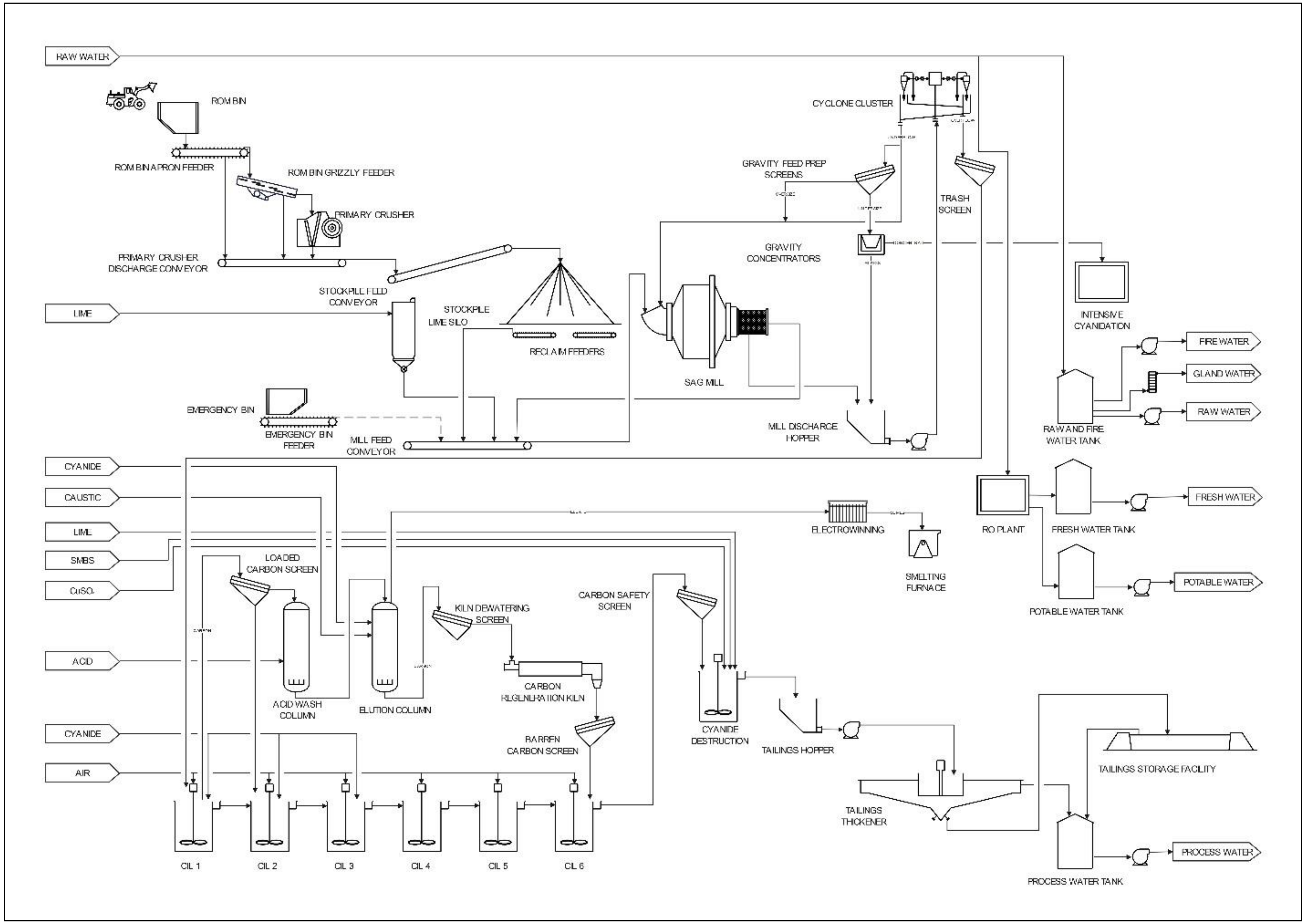

Figure 29 Summary flowsheet for Bendigo-Ophir Gold Plant

Crushing

The single-stage crushing circuit will produce a crushed product of nominally 80% passing (P80) of 121mm as feed for the Single Stage SAG (SSSAG) milling circuit.

The crushing circuit will only operate on dayshift, and coarse crushed mill feed will be stored on a stockpile with 12-hour live capacity.

All conveyors in the crushing and reclaim circuit will be fixed speed and equipped with:

- belt scrapers.
- belt ploughs.
- belt under speed detection.
- belt drift detection.
- local emergency stop button.
- local isolation.
- conveyor pull wires.
- belt rip detection.

All conveyor transfer chutes will be fitted with blocked chute detectors which will be interlocked to stop the respective conveyor drive if a high-level alarm is triggered.

Primary Crushing

Mill feed will be fed by front-end loader to the run of mine (ROM) bin which will be fitted with parallel grizzly scalping bars to prevent oversize entering the ROM bin.

The ROM bin will be equipped with dump/no dump lights to regulate mill-feed to the bin. The ROM bin dump point will incorporate a concrete pad with integrated tyre bump stops.

Mill feed will be withdrawn from the ROM bin at a controlled rate by the variable speed ROM Bin Apron Feeder, feeding the Vibrating Grizzly to remove any undersized material before feeding the oversize to the Primary Crusher. The Primary Crusher will be controlled from the Crusher Control Room.

Crushed mill feed will fall onto the heavy-duty Primary Crusher Discharge Conveyor. This conveyor will be fitted with an overhead Primary Tramp Metal Magnet to remove tramp metal. The Primary Crusher Discharge Conveyor will discharge to the Stockpile Feed Conveyor.

Dust suppression sprays will mitigate dust release at all transfer points, the ROM Bin and the Primary Crusher.

Stockpile and Reclaim

The Stockpile Feed Conveyor will discharge crusher mill feed onto the twelve-hour live capacity Stockpile. Dust mitigation at this point will be by the Stockpile Cover and water sprays.

Coarse crushed mill feed will be reclaimed from the stockpile by two variable speed Reclaim Feeders and deposited onto the Reclaim Conveyor feeding the Mill Feed Conveyor.

An emergency feeder will be located on the Mill Feed Conveyor to allow reclaim of any spillage, as a means of feeding the circuit if the stockpile is not available, and as a location to add mill balls. The Lime Silo will meter lime onto the Mill Feed Conveyor. The Mill Feed Conveyor Weightometer will allow control of the lime metering rate and of the Reclaim Feeders to manage the mill feed rate.

Bulk lime is delivered to site and pneumatically transferred to the Lime Silo.

SANTANA MINERALS LIMITED

Milling, Classification and Gravity Separation

Milling and Classification

The milling circuit will comprise a single stage 5,100kW SAG Mill, operating in closed circuit with cyclone classification to produce a leach feed sizing of P80 106µm.

The mill discharge density will be controlled by water addition to the mill feed chute, regulated using a flowmeter and flow control valve. The mill discharge slurry will flow through the trommel screen into the mill discharge hopper fitted with a level indicator. The slurry level in the mill discharge hopper will be maintained by a level control loop to the variable speed controller on the Cyclone Feed Pumps motors.

Scats generated by the SAG Mill will be transferred by the Pebble Discharge Conveyor, which will be fitted with an overhead Pebble Magnet to remove metal, to the Pebble Transfer Conveyor, and from there to the Mill Feed Conveyor. The Pebble Discharge Conveyor will be fitted with a Pebble Discharge Conveyor Weightometer to allow the scats mass to be monitored and accounted for.

The mill discharge slurry will be pumped to a cluster of 12 cyclones. Operation of the cyclones will be monitored by a slurry flowmeter, pressure measurement and a gamma density gauge. The cyclone feed density will be regulated by water addition to the mill discharge hopper using the slurry density reading provided by the gamma density gauge.

The cyclone underflow (coarse fraction) will flow to a Gravity Circuit Splitter Box feeding two Knelson Concentrators, the tails from which will gravitate back to the Mill Discharge Hopper.

Gravity Circuit

A fraction of the cyclone underflow slurry (maximum 38%) will be diverted from a splitter box to two horizontal vibrating Gravity Feed Screens. Oversize from the screens will be directed to the SAG Mill Feed Chute. Undersize from the Gravity Concentrator Feed screens (nominally minus 2 mm) will flow into two batch centrifugal concentrators. Two concentrators have been selected based on the high gravity recovery.

The gravity concentrators will maintain fluidised beds using raw water added at constant flow rates which will be controlled by flowmeters and flow control valves. The concentrators and feed screens can be bypassed if required for maintenance or operational purposes.

The gravity concentrators will be operated semi-continuously by local programmable logic controllers (PLC). Concentrate from the batch centrifugal concentrators will be periodically discharged to a secure hopper feeding the intensive gravity leach unit.

Tailings from the gravity concentrator and the gravity screen oversize will be discharged into the Mill Discharge Hopper. The milling area will be equipped with automatically operated sump pumps.

Leaching and Adsorption

The cyclone overflow will gravitate to a horizontal trash screen, which will remove any trash (plastics, wood). The oversize trash will fall into the Trash Screen Oversize Bunker and be periodically removed by front end loader for disposal.

Trash screen undersize will flow into the Leach Feed Distribution Box.

The leach circuit will comprise six Adsorption Tanks. Circuit selection has been based on testwork indicating that a preg-robber is present in the sample.

A pH probe will be installed in Adsorption Tank 1 (Tank 2 as an alternate) and will control the lime addition rate to the mill circuit to maintain a pH level above 10.5.

Cyanide levels will be monitored using a Cyanide Analyser, cyanide will be added to the Adsorption Tanks via a flow control valve to maintain a minimum free cyanide level in the various tanks.

Air will be added downshaft to the Adsorption Tanks via at a rate manually controlled using a flow control valve and monitored using a flowmeter. The air addition rate will be adjusted to target the required dissolved oxygen levels as measured by the dissolved oxygen probe installed in Leach Tank 1.

Barren eluate from the elution circuit will be returned to the Leach Feed Distribution Box in a controlled manner.

Carbon will be added to the adsorption tanks to adsorb the gold from solution and will be pumped counter current to the direction of slurry flow using recessed impeller Carbon Transfer Pumps. Carbon concentration in the adsorption tanks will typically be 10 - 15 g/L. Carbon will be retained within each tank by pumped type inter-tank screens as the slurry flows through the screens and overflow launders.

Barren carbon will be added to the last adsorption tank online and will be successively moved up the tank train using the Carbon Transfer Pumps.

The loaded carbon will be recovered from the first adsorption tank online by the loaded Carbon Transfer Pump and transferred to the horizontal vibrating Loaded Carbon Screen, where it will be washed by sprays. The carbon will overflow the screen to the acid wash circuit, while the underflow will be returned to the relevant adsorption tank.

The tailings slurry flow from the CIL will gravitate to a linear vibrating Carbon Safety Screen. This screen will collect any carbon in the 0.6mm - 0.8mm size range together with any coarser carbon if there is a leaking intertank screen on the last absorption tank. The undersize from the carbon safety screen will flow to the Cyanide Destruction Feed Distribution Box.

Cyanide Destruction

A cyanide destruction facility will be installed to reduce the weak acid dissociable cyanide (CNWAD) in the CIL discharge prior to tails thickening from a nominal level of 200ppm to 20ppm. Cyanide destruction will be carried out using the Inco air / SO2 method. The process utilizes SO2 and air in the presence of a soluble copper catalyst to oxidize cyanide to the less toxic compound cyanate (OCN-). The SO2 source will be Sodium Metabisulfite (SMBS).

Oxygen is also required in the reaction, and this is supplied by sparging air into the cyanide destruction tanks using dedicated blowers. The reaction is typically carried out at a pH of 8.0 to 9.0 in an agitated tank. Lime is added to neutralize acid (H+) formed in the reaction to maintain pH in this range. The circuit will consist of two cyanide destruction tanks operating in series. This configuration has been selected to ensure that free cyanide is completely bound by the copper added as part of the CuSO4 addition.

Slurry from the carbon safety screen undersize will discharge into the cyanide destruction feed distribution box. Hydrated lime will be fed from the cyanide destruction lime silo via a cyanide destruction lime feeder at a rate controlled by a pH meter installed in the first cyanide destruction tank into the cyanide destruction feed distribution box, together with copper sulphate (CuSO4) from the CuSO4 dosing pumps and SMBS from the SMBS dosing pumps. CuSO4 will be dosed at a fixed rate, while SMBS will be added based on the redox potential of the slurry measured by an ORP probe located in either cyanide destruction tank 1 or 2. SMBS can be added to cyanide destruction Tank 2.

Air will be added via spargers into the two cyanide destruction tanks.

Elution and Goldroom

The following operations will be carried out in the stripping and goldroom areas:

- Acid washing of carbon.
- Stripping of gold from loaded carbon using the split AARL method.
- Regeneration of carbon.
- Electrowinning of gold from pregnant solution.
- Smelting of electrowinning products.

The stripping and goldroom areas will normally operate 7 days per week. Most of the loaded carbon preparation and stripping will occur during day shift. The AARL stripping circuit will be automated and will contain separate acid wash and elution columns.

Acid Wash

Loaded carbon will be received into the 6-tonne capacity acid wash column. The facility to bypass acid wash and directly load the elution column will be provided. Transfer and fill operations will be controlled manually. All other aspects of the acid wash sequence will be automated.

During acid washing a dilute solution of hydrochloric acid will be pumped into the bottom of the acid wash column to remove contaminants, predominantly carbonates, from the carbon. This process improves elution efficiency and has the beneficial effect of reducing the risk of calcium magnesium slagging within carbon during the regeneration process.

After a 30 minute soak period, loaded carbon will be rinsed with water to displace acid solution and contaminants. Water rinsing will consist of pumping four bed volumes of water through the column. Dilute acid and rinse water will be pumped to the cyanide destruction facility for disposal.

Pre-Soak and Elution

Lean eluate will be pumped from the lean eluate tank through the inline recovery heat exchanger and primary heat exchanger and injected into the base of the elution column. Sodium hydroxide and sodium cyanide solutions will be pumped from the respective storage tanks and injected into the suction line of the duty strip solution pump. Loaded carbon will be pre-soaked in cyanide / caustic solution for 30 minutes to elute gold. Pregnant eluate will be rinsed from the carbon by initially drawing five bed volumes of recycled heated lean eluate drawn from the lean eluate tank to the pregnant solution tank for recovery by electrowinning. Once the lean eluate tank is empty, the remainder of the strip will draw from the stripping water tank and elute will be directed to the lean eluate tank in preparation for the next strip.

Electrowinning and Goldroom

Once elution is completed and the eluate directed to the two pregnant solution tanks, recovery of gold by electrowinning will proceed. Direct current will be passed through stainless steel anodes and stainless-steel mesh cathodes within the two parallel electrowinning cells and electrolytic action will cause gold in solution to plate out on the cathodes.

Electrowinning will take 8 - 12 hours. Solution discharging from the electrowinning cells will return by gravity to the appropriate eluate tank. The system will be configured to allow single or multiple pass electrowinning. Electrowinning will continue until the solution exiting electrowinning cells is depleted of gold. A poppet type sampler will be provided in the pipe feeding the pregnant solution tank. Barren eluate solution will be returned to the leach feed distribution box, allowing any residual gold to be recovered and pH and cyanide credits to be utilised in the CIL circuit.

Electrowinning cells will be of stainless-steel construction with lockable lids, sloping floors and will be located within the secure area of the goldroom.

Rectifiers, one per cell, will be in a caged area outside the goldroom allowing semi-restricted operation and maintenance access without going through full goldroom security. Rectifier remote ammeters and controls will be located adjacent to the cells in the goldroom to allow remote current adjustment.

An overhead crane will be provided to assist with handling of cathodes. Cathodes will be washed with high pressure spray water and gold slime will be recovered in a plate and frame filter press.

Gold sludge filter cake will be dried in a drying oven and direct smelted with fluxes in the electric arc Smelting Furnace to produce doré bars.

Fume extraction to remove any volatised heavy metals and toxic gases from the electrowinning cells and drying ovens will be provided. In addition to this, fresh air fans will be provided to ensure there is adequate ventilation inside the goldroom.

Carbon Regeneration

After completion of the elution process, barren carbon will be transferred to the carbon regeneration kiln. Carbon will be hydraulically transferred to the Carbon Dewatering screen prior to entering the kiln feed hopper. In the Kiln Feed Hopper any residual and interstitial water will be drained from the carbon before it enters the kiln.

Carbon will be heated to 650 - 750°C and held at this temperature for 15 minutes to allow effective regeneration to occur. Regenerated carbon from the kiln will discharge to the carbon quench tank. Quenched carbon will be pumped by the carbon quench tank pump (a recessed impeller pump) to the barren carbon screen. Sized carbon enters the CIL tanks while undersize carbon passes through the sizing screen and reports to the carbon safety screen.

Exhaust gas streams around the carbon regeneration kiln will be vented to atmosphere.

Tailings

Slurry exiting the cyanide destruction facility will be pumped to the tailings thickener feed box via a two stage tailings sampler. The tailings primary cross stream sampler cuts a sample which is sub sampled by a vezin tailings secondary sampler. The tailings sampler underflows recombine with the sampled stream.

A high-rate tailings thickener will thicken final tailings to recover water. Slurry will be de-aerated in the tailings thickener feed box where flocculant will be added prior to entry into the thickener. Flocculant will also be sparged into the thickener feed box to assist with thickening.

Thickener underflow will be pumped via the tailings pumps to the tailings hopper, from where it will be pumped to the tailings storage facility by the Train 1 and Train 2 tailings pumps 1 and 2.

Thickener overflow will report to the thickener overflow tank, and then be pumped to the process water tank.

Reagents

Quicklime

Quicklime will be delivered to site in bulk and transferred to the lime silo, located over the mill feed conveyor. Lime will be dosed from the lime silo by the lime silo rotary valve into the lime silo feeder and will drop onto the mill feed conveyor. The lime feed rate will be controlled via a variable speed drive in a control loop cascading from the leach tank pH and mill feed rate.

Hydrated Lime

Hydrated lime will be delivered to site in bulk bags and transferred to the cyanide destruction lime silo, located over the cyanide destruction feed distribution box, via a bag breaker and using the Lime hoist.

Lime will be dosed from the cyanide destruction lime silo by the cyanide destruction lime silo rotary valve into the cyanide destruction lime silo feeder and will drop into the cyanide destruction feed distribution box. The lime feed rate will be controlled via a variable speed drive in a control loop cascading from the cyanide destruction tank pH.

Cyanide

Solid sodium cyanide will be delivered to site by truck. The trailers will be unloaded, and solids mixed in the cyanide mixing tank, which will be filled with raw water prior to the cyanide delivery truck arriving on site.

Mixed cyanide from the cyanide mixing tank will be periodically transferred to the cyanide storage tank by the cyanide transfer pump as required to maintain its level.

The cyanide solution will be circulated in a ring main, using separate control valves and flowmeters to regulate the addition of cyanide to following locations:

1. Leach feed distribution box and adsorption Tanks 3 and 5.
2. Elution column.
3. Intensive leach reactor.

Hydrochloric Acid

Hydrochloric acid (32% w/w) will be delivered in bulk to the site storage tank. Acid will be dosed to the acid wash column via a variable speed pump where it will be diluted and mixed in-stream with fresh water to a concentration of 3% by weight for acid washing of the loaded carbon.

Sodium Hydroxide

Sodium hydroxide (49% w/w) will be delivered in bulk to the site storage tank. Sodium hydroxide will be circulated in a ring main, using separate control valves and flowmeters to regulate the addition of sodium hydroxide to following locations:

1. Elution column.
2. Intensive leach reactor.

Gold Room and Intensive Leach Reagents

Flux reagents for gold smelting will be delivered in powder form in 25kg bags, including silica sand, sodium nitrate, soda ash and borax.

Leachaid will be delivered in 20kg buckets for use in the intensive leach reactor.

Flocculant

Flocculant will be delivered to site in batches of 20 x 1t bulk bags. Dry flocculant will be hoisted and emptied into the dry flocculant storage hopper. When required the flocculant will be mixed in a vendor supplied automated mixing system and aged prior to transferring into the storage tank. Dedicated duty/standby dosing pumps will supply the flocculant to the thickener where it will be diluted 10-fold with process water prior to use in the thickener.

Sodium Meta Bisulphite (SMBS)

SMBS will be delivered in 1 t bulk bags. The bulk bags will be lifted by monorail hoist to a bag breaker above the SMBS mixing tank.

SMBS will be dissolved by mixing with raw water to a concentration of 20% w/v in the agitated mixing tank and will be pumped to the SMBS bulk storage tank. SMBS will be metered to the cyanide destruction tanks by positive displacement dosing pumps.

Copper Sulphate ($CuSO_4$)

CuSO4 will be delivered in 1 t bulk bags. The bulk bags will be lifted by monorail hoist to a bag breaker above the CuSO4 mixing tank.

CuSO4 will be dissolved by mixing with raw water to a concentration of 20% w/v in the agitated mixing tank and will be pumped to the CuSO4 bulk storage tank. CuSO4 will be metered to the cyanide destruction tanks by positive displacement dosing pumps.

SAG Mill Balls

SAG mill grinding media will be delivered in 200 litre drums and manually emptied in the SAG ball bunkers. SAG balls will be loaded into the emergency feeder hopper by front end loader.

Activated Carbon

Fresh activated carbon will be delivered in 500kg bulk bags. The bulk bags will be lifted by monorail hoist to the chute above the regenerated carbon transfer / quench tank. Fresh carbon will then be conditioned and pumped to the barren carbon screen. The carbon sizing screen will remove carbon fines from the fresh material before adding it to the circuit. The facility to recirculate carbon through the pump will be provided to allow additional preconditioning of the carbon if required.

Water Services

Process Water

Process water pumps (duty and standby) will draw from the process water tank to supply the following services:

- Grinding area density control.
- Gravity circuit for solids transfer.
- Elution area for carbon transfer from the kiln quench tank.
- Tailings for flocculant dilution and flushing water.
- Service points throughout the plant.

The process water tank will receive tailings thickener overflow which is inclusive of decant return water and raw water which is modulated to maintain the process water tank level.

Raw Water

Raw water will be drawn from a riparian borefield and pumped to the clean water storage dam in Shepherds creek. From here water will be transferred to the raw water tank and the RO feed tank. The raw water tank also will act as the fire water reserve.

The following pumps will draw from the Raw Water Tank:

- Duty/standby raw water pumps.
- Electric, diesel and jockey fire water pumps.
- Gland water pumps.
- Elution water pump (for acid rinse).
- Carbon reduction pump.

The fire water pumps will withdraw water from a suction located lower than the other pumps, to ensure that the fire water reserve is always maintained within the Raw Water Tank.

The Raw Water Pumps (duty and standby) will draw water from the Raw Water Tank and supply the following plant services:

- Crushing area dust suppression sprays.
- Gravity concentrator fluidising water.
- Trash, carbon safety, kiln dewatering and barren carbon screen spray water.
- Carbon regeneration kiln.
- Process water tank make-up.
- Gold room.
- Flocculant mixing.
- Cyanide mixing.
- SMBS mixing.
- $CuSO_4$ mixing.
- Hydrochloric acid storage.

Gland water pumps (duty and standby) will draw water from the Raw Water Tank to supply high pressure gland seal water to the cyclone feed pumps and tailings pumps.

Fire Water

The intake for the raw water pumps will be at the lowest point of the Raw Water Tank to allow for a fire water reserve to be maintained at all times. Fire water pumps (electric duty pump and diesel engine emergency standby) will draw water from the base of the raw water pond to supply the site fire water network which will include hydrants and hose reels.

Potable Water

Raw water from the borefield will be directed to a dedicated Reverse Osmosis (RO) Feed Tank. Water treatment will comprise RO and chlorine sterilisation with the permeate flowing to the Potable Water and Fresh Water Tank. Brine rejected from the RO plant will report to the Raw Water Tank.

Potable Water Pumps (duty and standby) will discharge into a ring main to supply the plant safety showers, site drinking water, ablution blocks, laboratory, crib room, store, and workshop. Pipe lagging and thermal relief valves will be used to avoid excessively high water temperature at the safety showers.

The Fresh Water Tank will provide water storage for the Elution Water Pump and the Fresh Water Pump. The elution pump will be dedicated to the elution sequence. The Fresh Water Pump will supply the following:

- The intensive leach reactor.
- Gold room pressure washer.
- Carbon regeneration kiln quench water.

Air Services

Plant and Instrument Air

Two screw Plant Air Compressors will provide dried and filtered compressed air (high pressure plant air) to the Plant Air Receiver, from which the air will be reticulated throughout the plant. An additional Mill Air Receiver will be located adjacent to the mill and be dedicated to the mill lubrication system. All air produced will be of instrument quality allowing for a single air distribution system throughout the plant. Plant air service points will be provided around the plant.

CIL and Cyanide Destruction Aeration

Duty and standby medium pressure CIL Air Blowers will supply CIL aeration requirements. Short term peaks in demand will be accommodated operating both blowers in parallel.

Duty and standby medium pressure Cyanide Destruction Air Blowers will supply cyanide destruction aeration requirements. Short term peaks in demand will be accommodated operating both blowers in parallel.

Power Supply

Aurora Energy Limited is the local statutory lines company. Aurora have provided an initial outline of supply from its Lindis Crossing Substation comprising of 8km of 33kV lines to a new transformer to be installed at the project. Supply from there around the Project will be managed by the Company. An alternative grid exit point from the Roxburgh - Twizel A (ROX-TWZ-A) 220kV transmission line is being investigated with the operator of New Zealand's national electricity grid, Transpower.

Confidential negotiations are in progress with several potential electricity retailers. Current long term supply contracts are in the range of $0.10 to $0.13 per kWh.

Other Infrastructure

Tailings Storage Facility (TSF) and Engineered Landforms (ELF)

The proposed RAS Tailings Storage Facility (TSF) is located in the Rise and Shine Valley, upstream of the proposed processing plant area, as shown in the site TSF and Engineered Land Form (ELF) layout plan in Figure 30. The proposed Rise and Shine ELF is for the final storage of non-auriferous sterile rock that and is located between the RAS TSF and the processing plant, and the proposed Jeans Creek ELF and Shepherds Creek ELF are to the north at a lower elevation.

The proposed embankments will be constructed with RAS Pit overburden.

The proposed TSF downstream construction embankments have a 1V to 2H downstream slope and 1V to 1.5H upstream slope with an 8m final crest width. It is proposed that the tailings are delivered as a conventional slurry. The TSF foundation is schist rock mass.

The schist rock is expected to form a low permeability foundation. A moderate thickness of soils in the valley floor and on some slopes will be stripped for embankment construction. Detailed investigation of the foundations is required. The proposed embankment is zoned, requiring compaction and conditioning of the upstream portion of the embankment, to achieve performance requirements. Chimney drains are proposed for the initial starter embankment for the TSF until an effective tailings beach can be established against the embankment. Upstream cut off drains are proposed to full embankment height to intercept natural seepage.

Foundation underdrains are proposed in the valley to collect tailings seepage which will be returned to the TSF or process plant. In closure the tailings surface is to be dry capped in overburden material and site won surficial soils. A final surface water open channel is proposed to convey surface water off the rehabilitated surface of the tailings.

The total potential tailings storage capacity of the RAS TSF scoping study design is estimated to be 19.8 million tonnes, which is sufficient for the proposed scoping study processing requirements. The TSF embankments require an estimated 17 million tonnes of overburden material for construction, which will come from the RAS open pit.

The proposed ELFs have a maximum final slope of 1V to 3H, providing final storage locations for the sterile, overburden rock from the pit. Much of the ELF surface will be much flatter than 1V to 3H.

Proposed diversion culverts beneath or alongside the ELFs will pass surface water in the creeks during construction. Final landforms are to blend into the natural environment. Once final landforms are established it is proposed that the culverts are disestablished, and final surface water channels constructed over or around the ELF. It is proposed the ELF is rehabilitated in site won surficial soils.

All TSF and ELFs are positioned on land with existing access rights.

Resource consent and building consent are required under New Zealand legislation to construct the TSF. Only resource consent is required to construct the ELFs.

Resource consents relating to the environmental effects will be applied for as part of the wider project application. Building consent for the TSF will be separately applied for through the building consent authority for dams in the area.

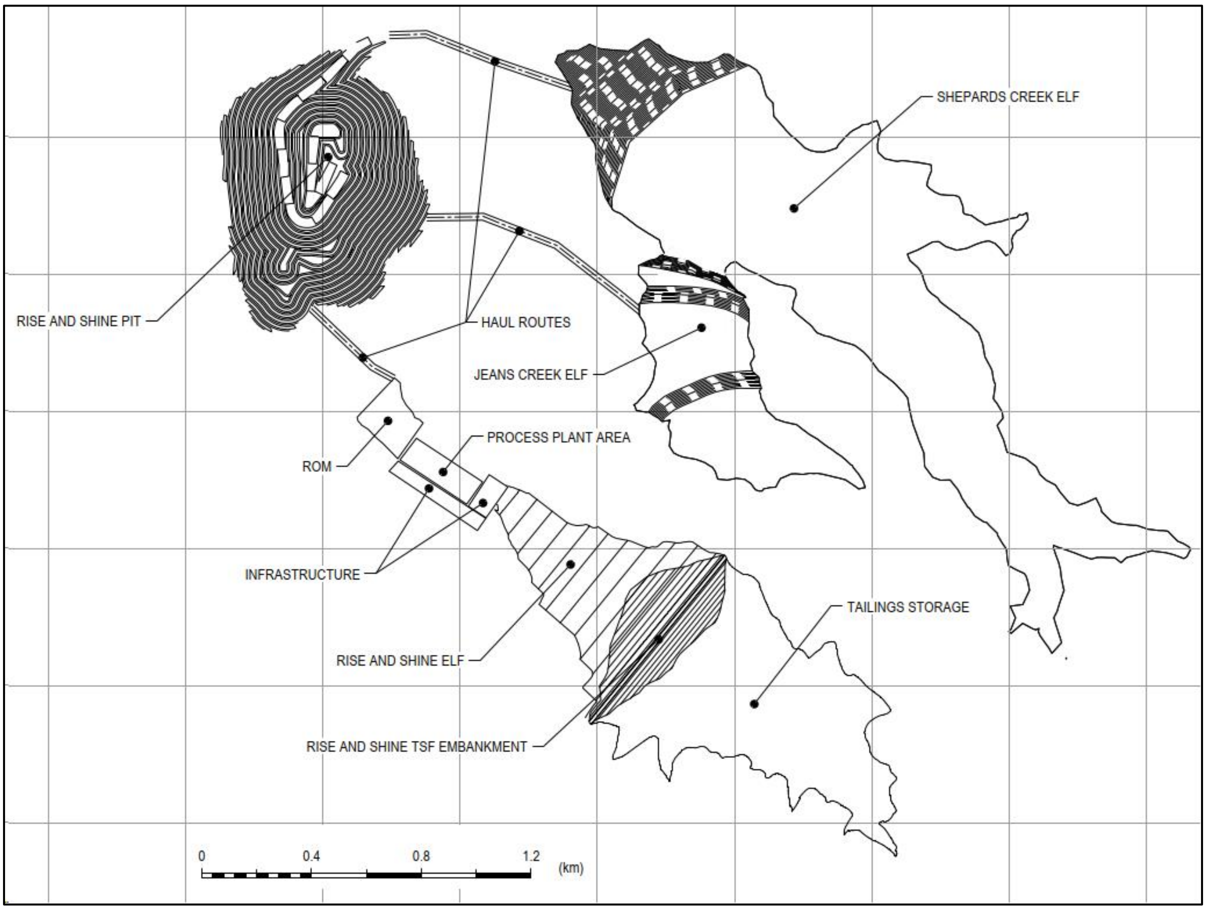

Figure 30 RAS Scoping Study Site Layout showing the location of the pit, ROM, Processing Plant, TSF and the ELFs with an area of approximately 350ha.

Site Roads and Access

The main access to site is via State Highway 8 (SH8), with a turn-off 14km north of Cromwell onto the Bendigo Loop Road. The main plant and infrastructure area is reached by a further 13km of paved and unpaved public road.

Road upgrades will be required to allow for construction and operational traffic.

A low usage high country public access road, the Thompson Gorge Road will be partially realigned as part of the project.

Water Supply and Storage Distribution

The process and dust suppression water requirements will be in the order of 50-100 litres per second (l/s) depending on the stage of the project's development and the season.

The project is likely to have a negative site water balance.

Project make up water could be sourced from a riparian borefield located in the Bendigo aquifer that is effectively recharged from the Clutha River. Water would be supplied via a 12km pipeline to be constructed over land with whom there are existing access arrangements, to the plant site.

Accommodation and Flights

The mine will be operated as a residential-only site with staff for the operational phase expected to live in the local area with sizeable communities of Cromwell (population 7,000 and 27km away), Alexandra (6,000 and 58km), Wanaka (12,000 and 48km), and Queenstown (29,000 and 82km), all within one hour drive of the site.

To manage fatigue, buses from the furthest locations are planned to be operated, collecting personnel along the way.

Regular daily international and domestic flights are available through Queenstown airport.

Non-Process Infrastructure

Other infrastructure that has been estimated for the project capital, both mining and non-mining related, are as follows

- Administration office.
- Open pit mining infrastructure:
    - Mine office (Management, technical and supervision).
    - Mine change-house, muster and crib facilities.
    - Mobile fleet workshop, fleet refuelling facility and wash down.
    - Warehouse and associated material laydowns; and
    - Explosives magazines.
- Additional Underground mining infrastructure:
    - Mine office (Management, technical and supervision).
    - Mine change-house, muster and crib facilities.
    - Mobile fleet workshop, fleet refuelling facility and wash down.
    - Primary exhaust fan; and
    - Surface settling sumps for underground dewatering.

Environmental

The project area is within the larger MGL Exploration permit (MEP60311), located in the Dunstan Mountains east of Lake Dunstan in the Rise and Shine area of the TGR.

Hydrogeology

Major drainages in the project area include (from SW to NE): Bendigo Creek, Rise & Shine, and Shepherds Creek.

These catchments drain into the fluvio-glacial sediments and groundwater aquifers of the Cromwell-Tarras valley. These groundwater aquifers in turn flow toward the Lindis and Clutha Rivers.

Surface water flow is intermittent across the valley. Surface flow connectivity to the Clutha River and subsequently Lake Dunstan approximately 5 kilometres to the north-west occurs only during large events.

Climate

The Central Otago climate is unique in New Zealand. The prevailing weather comes from the Tasman Sea with Central Otago lying in the rain shadow of the Fiordland mountains and Southern Alps resulting in a semi-arid climate. Summers are hot and droughts are common. Winters are characterised by hard frosts and regular snow in the high country.

Climate data is available for nearby population centres (Cromwell, Wanaka, Tarras) with average annual rainfall in the project area between 400 and 550mm that increases with elevation.

Late summer rainfall can be as low as 60mm (January to March). Median summer air temperatures for the area are 16–17 °C and winter median temperatures are 5-6 °C, cooling with increasing elevation.

Ecology

Detailed ecological surveys (flora, fauna, freshwater, invertebrates) commenced in September 2023 and will continue into 2024 to capture seasonal changes. The surveys are across the Project Area of about 1,500ha and the wider Ecological Study Area of 3,500ha located immediately adjacent to the Project. Figure 31 shows the extent of these two areas. The mine site layout shown in Figure 30 31 lies fully within the Project Area and covers an area of approximately 350ha.

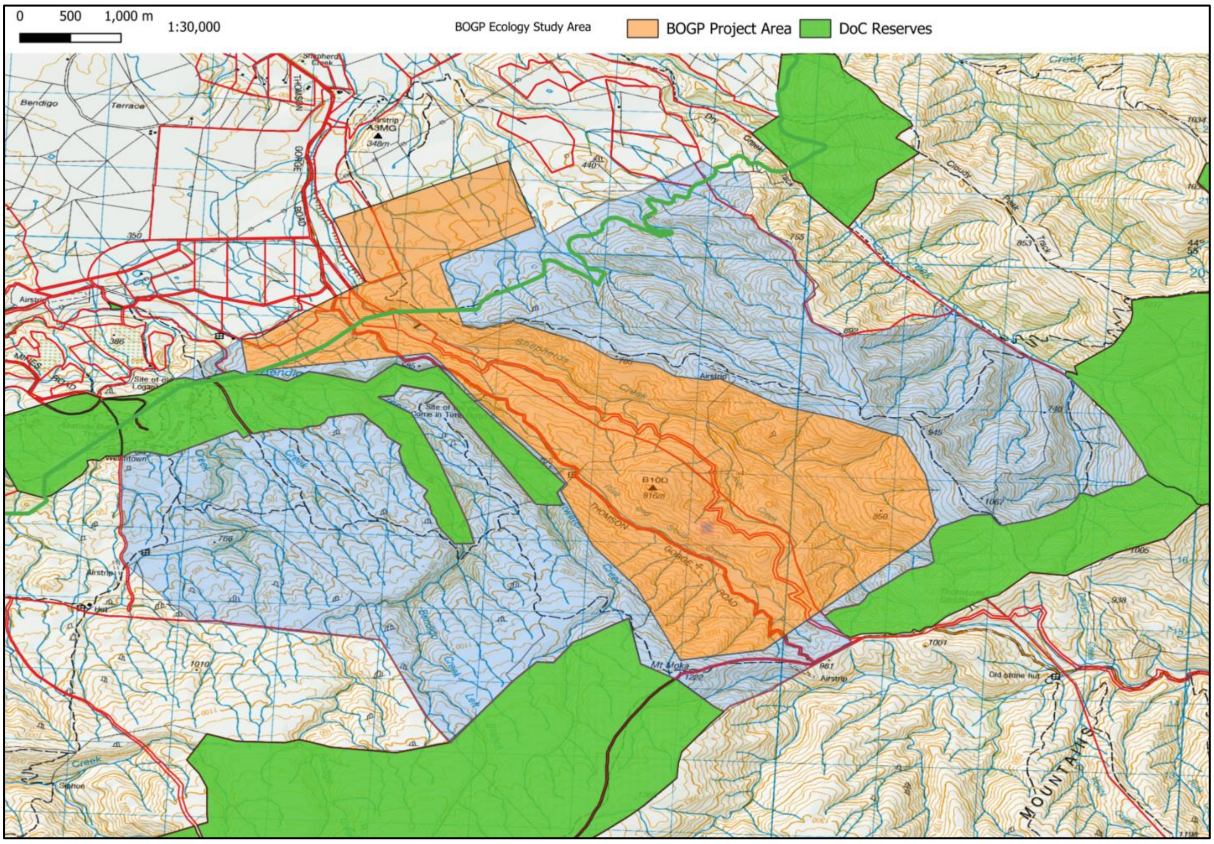

Figure 31 BOGP Project and Ecology Study Areas.

The Project Area does not include any notable ecological values that have been stated as significant in the Otago Regional Policy Statement or the Central Otago District Plan.

While the site has been heavily modified by historic and existing land use practices and the ongoing effects of invasive weeds and introduced mammalian predators, there are some areas of high terrestrial value based on the presence of native dominated habitats and of nationally 'Threatened' or 'At Risk' species. These values continue within the Ecology Study Area, outside of the Project.

Initial desktop reviews and survey results have identified the following ecosystem types in the Project and Ecology Study areas:

- Exotic dominated grassland/shrub mosaic.
- Native dominated Taramea herbfield.
- Native dominated Grey scrub.
- Native dominated Helichrysum, Melicytus shrubland/tussockland/rockland.
- Native dominated Manuka scrub forest (outside project area).
- Kanuka, Olearia scrub/treeland (outside project area).
- Native dominated Hall's totara, mountain celery pine, broadleaf forest (outside project area).
- Exotic dominated wetland complex.
- Indigenous dominated sedgeland wetland complex.
- Wetland seepages.
- Hard-bottom streams.

Wetlands are not prevalent with the Project Area except for some small (<1ha) areas of riparian wetland and seepage wetlands in conjunction with low to moderate value hardbottom streams. The wetlands are predominately exotic-dominated but some native-dominated wetlands are present.

Native plant diversity within the Project and Ecology Study areas is high, with at least 100 native vascular plant species known to be present, along with some non-vascular species (algae, bryophytes, fungi, lichens and mosses). Of the vascular plants 12 nationally "Threatened" or "At Risk" species are present in the project area. Ongoing surveys will identify if further threatened or at-risk species are present and the extent of their ecosystem types.

At the time of writing terrestrial and wetland bird values were assessed as low relative to other habitats present in Central Otago, whilst lizard values were considered to be high and invertebrates largely unknown. Ongoing surveys will further define these values.

A freshwater ecological assessment of Bendigo, Shepherds and Rise and Shine Creeks showed dynamic surface water features which are likely to fluctuate in flow due to their steep catchments and locality. No fish were identified during the assessment likely due to the ephemeral flow and perched culverts. Macroinvertebrate communities exhibited degradation in areas but were otherwise considered fair.

Further Work

Baseline surveys and effect assessments are continuing to further understand the existing environment and inform an assessment of environmental effects as required for a consent application including: ecology, noise, dust, traffic, water, environmental geochemistry, socio-economic, recreational, landscape, soil, waterways, wetlands, cultural and heritage.

Iwi

Engagement activity has been undertaken since 2017 directly through the iwi-owned consultancy service Aukaha, which is governed by five papatipu rūnaka:

- Te Rūnanga o Waihao
- Te Rūnanga o Moeraki
- Kāti Huirapa Rūnaka ki Puketeraki

- Te Rūnanga o Ōtākou
- Hokonui Rūnanga

Engagement included briefings on exploration activities and more recently the proposed Bendigo Ophir Gold Project.

Aukaha undertook an assessment and provided a cultural statement about the wider project area in 2018. Aukaha visited the site in April 2024 to understand the progress of the project.

A review of archaeological surveys will identify if any further survey and assessment work is required for consenting purposes.

Permitting

Mining Permit

The project is located within Minerals Exploration Permit (MEP) 60311. To develop the project, the Company will need to apply for a minerals mining permit (MMP) over the immediate area to New Zealand Petroleum and Minerals (NZPAM). This is part of the Ministry of Business, Innovation and Employment (MBIE) and administers the Crown Minerals Act (1991) (CMA). Section 23 of the CMA provides that the purpose of a minerals mining permit (MMP) is to authorise the permit holder to mine for the minerals specified in the permit. "Mining" is defined in the Act as meaning "to take, win, or extract, by whatever means, a mineral existing in its natural state in land, or a chemical substance from [that mineral]."

The Minister will ordinarily grant a mining permit if satisfied that:

> (a) the permit applicant has identified and delineated at least an indicated mineable mineral resource or exploitable mineral deposit, and
> (b) the area of the permit is appropriate, and
> (c) the objective of the mining permit is to economically deplete the mineable mineral resource or deposit to the maximum extent practicable in accordance with good industry practice.

Resource Consents

Environmental approvals are administered under the Resource Management Act 1991 (RMA) the purpose of which is to promote the sustainable management of natural and physical resources. The Act is administered by the consent authority whose permission is required to carry out an activity for which a resource consent is required under this Act. In this case, resource consents will be required from both the Central Otago District Council and the Otago Regional Council.

The NZ government has introduced a new legislation, Fast Track Approvals Bill, which is currently being heard in parliament. This Bill seeks to provide a one-stop-shop for applications for projects of national/regional significance to be determined in an expedited timeframe.

Applications

The Company plans to submit the above applications to the relevant authorities on the completion of the prefeasibility study in 2024.

Operating Cost Estimate

The breakdown of operating costs is summarised in Table 16.

Table 16 Operating Costs Summary

	$NZ M	$/t milled	$/oz
Open Pit Mining	$377	$31.13	$375
Underground Mining	$152	$65.66	$701
Total Mining	$530	$36.68[1]	$473[1]
Processing	$228	$15.79	$204
G and A	$42	$2.94	$38
Royalties	$142	$9.86	$127
Total Cash Operating Costs	$943	$65.27	$841

[1]Weighted average of open pit and underground

Mining Costs

Open Pit

A bottom up first principles cost model has been built based upon resourcing 250t and 120t excavators matched with 140t and 90t capacity haul trucks.

Haul profiles have been created to reflect the over-burden haul to the ELF locations.

Indicative operating costs for all the major units were sourced from the OEMs and explosives and tyre costs provided by key suppliers.

The average LOMP cost per tonne of rock moved is estimated to be $3.90/tonne assuming current contract mining costs.

Underground

Underground mining costs were produced by our lead mining consultant and are in-line with a similar operation in NZ whilst also accounting for additional cost of cemented paste backfill.

The total mining operating costs are $152M or $65.66/t mill feed.

Processing Costs

Total processing costs are estimated to be $228M or $15.79/t processed.

The annual costs are based on $9.58M/year fixed and $8.95/tonne processed variable.

A further breakdown is provided in Table 17.

Table 17 Process operating costs (variable at 1.5Mtpa processing rate)

Cost Centre	Total Cost		% Fixed	Fixed	Variable	
	NZ$M/yr	NZ$/t		NZ$M/yr	NZ$M/yr	NZ$/t
Operating Consumables	$8.80	$5.87	30%	$2.64	$6.16	$4.11

Cost Centre	Total Cost		% Fixed	Fixed	Variable	
Maintenance	$1.97	$1.31	31%	$0.61	$1.35	$0.90
Process & Maintenance Labour	$4.51	$3.01	100%	$4.51	$0.00	$0.00
Power	$7.73	$5.15	23%	$1.82	$5.91	$3.94
Total	$23.01	$15.34[1]		$9.58	$13.43	$8.95

[1]Assuming 1.5Mtpa

General and Administrative Costs

G and A annual costs are estimated at $3.61M per year.

Table 18 G and A operating costs (variable at 1.5Mtpa processing rate)

Cost Centre	Total Cost		% Fixed	Fixed	Variable	
	NZ$M/yr	NZ$/t		NZ$M/yr	NZ$/yr	NZ$/t
General & Administration Costs	$1.87	$1.24	100%	$1.87	$ -	$ -
Administration Labour Costs	$1.75	$1.17	100%	$1.75	$ -	$ -
TOTAL	$3.61	$2.41[1]		$3.61	$ -	$ -

[1]Assuming 1.5Mtpa

Royalties

As gold is a Crown mineral, a royalty is payable to the Crown as either the higher of an ad valorem royalty of 2% of the net sales revenue or an accounting profits royalty of 10%.

The Project is subject to a 1.5% Net Smelter Royalty (NSR) on all production from MEP 60311 (and successor permits) payable to an incorporated, private company (Rise and Shine Holdings Limited) which is owned by the prior shareholders of MGL (NSRW Agreement) before acquisition of 100% of MGL shares by Santana Minerals Limited.

Access arrangements are in place with landowners that provide for current exploration and other activities, and any future decision to mine. As such, compensation is payable, including payments of up to $1.5M on a decision to mine, plus total royalties starting at 1% on the net value of gold produced, increasing to 1.5% and ultimately 2% dependent on location and total gold produced over the life of the mine. The royalties are also subject to pre-payment of up to $3M upon commencement of mining operations.

Capital Cost Estimate

All cost estimates are within +/-25% accuracy unless otherwise specified.

Total project capital is $143.2M including:

- Processing plant and associated infrastructure; and
- Non-processing site infrastructure

Table 19 Project capital

Item	Project Capital ($M)
Site Infrastructure and establishment	$39.4
Process plant and associated Infrastructure	$103.9
Total Project Capital	$143.2

Processing Plant

The processing plant capital is based on an EPCM style construction and is for a 1.5Mtpa conventional CIL facility. The cost estimate includes plant associated infrastructure including:

- Plant administration buildings.
- Plant workshop and warehouse; and
- Analytical laboratory.

The total processing plant and associated processing infrastructure cost is $103.9M. The detailed breakdown is shown in Table 20.

Table 20 Capital cost for the 1.5Mtpa CIL plant (NZ$ M)

Item	Material Cost	Labour Cost	Freight Cost	Sub-total
Direct				
Earthworks	$1.02	$2.03	$0.00	$3.05
Civil works	$4.34	$3.54	$0.01	$7.90
Mechanical equipment	$21.44	$1.76	$1.74	$24.94
Platework	$5.29	$1.90	$0.25	$7.44
Structural steel	$4.95	$1.75	$0.49	$7.20
Electrical installations	$9.04	$5.84	$0.70	$15.57
Buildings	$1.24	$0.19	$0.16	$1.59
Piping	$3.14	$2.63	$0.49	$6.26
Construction equipment	$4.40	$2.41	$0.05	$6.87
Total Direct	**$54.87**	**$22.05**	**$3.90**	**$80.82**
Indirect				
Temporary construction facilities	$0.60	$0.24	$0.12	$0.96
Supervision and Construction Management	$0.26	$4.36	$0.00	$4.61
Project and procurement management	$0.07	$2.89	$0.00	$2.96
Engineering design	$0.00	$6.17	$0.00	$6.17

Item	Material Cost	Labour Cost	Freight Cost	Sub-total
Vendor Commissioning	$0.02	$0.23	$0.00	$0.25
Commissioning	$0.06	$1.00	$0.00	$1.06
Mobilisation and demobilisation	$6.62	$0.13	$0.26	$7.01
Total Indirect	**$7.62**	**$15.03**	**$0.38**	**$23.03**
Total	$62.49	$37.08	$4.28	$103.85

Non-Processing Infrastructure Cost Breakdown

The significant non-processing infrastructure includes:

- Main access road, public road diversion and site internal roads.
- High voltage power grid connection to site.
- Service water extraction and connection to site.
- Water storage dam, impacted and non-impacted water diversions and dams.
- RAS TSF embankment.
- Environmental mitigation and;
- Open pit mining and G and A buildings, workshops and supporting infrastructure.

Included for sustaining capital is a 5% per annum cost of the total non-processing infrastructure.

Working Capital

Pre-production costs are $113.0M, due to the 32Mt pre-strip requirement to reach sustainable mill feed, averaging $3.53/t including near pit haulage.

Table 21 Working capital

Item	Working Capital ($M)	Comments
Pre-production	$113.0	Mining pre-strip
Total	**$113.0**	

Sustaining Capital

Underground Mining

Underground mining capital totals $108.6M and includes:

- Mining fleet as outright purchase.
- Surface infrastructure to support the operation such as offices, change-houses, workshops, power, roads, etc.
- Ventilation and dewatering infrastructure.
- Portal and all required development to reach mineralisation.
- Paste backfill plant.

Table 22 Total Underground Mining capital ($NZ M)

Capital	
Ventilation	$2.6
Dewatering	$1.1
Electrical	$1.2
Pastefill Plant	$18.0
Mine Infrastructure - Surface	$12.1
Capex (primary fleet, ancillary) over schedule	$48.4
Capex (Lat, Vert dev) - over schedule	$25.4
Total - Capex	$108.6

Mine Closure & Rehabilitation Costs

Total mine closure costs are estimated to be $31.1M (including proceeds of disposal of assets), with $10.3M specifically for the TSF, and the remaining for:

- ELF closure.
- Plant and other infrastructure removal; and
- Final impacted and non-impacted water diversions.

Project Economics – Financial Analysis and Outcomes

Financial Result

The base case financial model uses a gold price of NZ$2,705 per ounce (US$1,650 per ounce from NZD:USD exchange rate 0.61), which is lower than the gold spot price over the past eighteen months. At the spot price of NZ$3,900 as at 9th April 2024, the Project is forecast to generate a post-tax IRR of 75%, an NPAT of NZ$2,005 million and a post-tax $NPV_{10\%}$ of approximately NZ$937 million. The financial summary is presented in Table 23 below.

Table 23 LOM financial results summary [1]

Key Financial Assumptions	Unit	Base Case NZD	NZD	AUD	USD
Gold Price Assumed	$/oz	$2,705	$3,900[2]	A$3,545[2]	US$2,340[2]
Exchange Rate	USD:$	US$0.61	US$0.60	US$0.66	US$1.00
Key Project Metrics					
Gold Produced	Oz	1.12 million			
Initial Mine Life		10 years of mine production			
Gold Revenue	**$M**	**$3,030M**	**$4,368M**	**$3,971M**	**$2,621M**
Mining Costs	$M	$530	$530	$481	$318
Processing Costs	$M	$228	$228	$207	$137
General and Admin Costs	$M	$42	$42	$39	$25
Royalty - Government	$M	$61	$87	$79	$52
Royalty - Other	$M	$82	$118	$107	$71
Total Cash Operating Cost	**$M** *$/oz*	**NZ$943M** *NZ$841/oz*	**NZ$1,005M** *NZ$897/oz*	**A$914M** *A$816/oz*	**US$603M** *US$538/oz*
Project EBITDA	**$M**	**NZ$2,087**	**NZ$3,363M**	**A$3,057M**	**US$2,018M**
Depreciation and Amortisation	$M	$554	$554	$503	$332
Total Production Cost	**$M** *$/oz*	**NZ$1,496M** *NZ$1,336/oz*	**NZ$1,559M** *NZ$1,392/oz*	**A$1417M** *A$1265/oz*	**US$935M** *US$835/oz*
Net Profit Before Tax (NPBT)	$M	$1,534	$2,809	$2,554	$1,686
Tax Payable (28%)	$M	$438	$805	$732	$483
After Tax Profit	**$M**	**NZ$1,096M**	**NZ$2,005M**	**A$1,822M**	**US$1,203M**
Capital					
Capital Plant and Infrastructure	$M	$143	$143	$130	$86
Working Capital for pre-strip and mine set-up.	$M	$113	$113	$103	$68
Sustaining Capital Stripping and UG Development	$M	$297	$297	$270	$178
Total CAPEX over Mine Life	**$M**	**NZ$554M**	**NZ$554M**	**A$503M**	**US$332M**
DCF Outcomes					
Initial Project $NPV_{10\%}$	$M	$486	$937	$852	$562
IRR	%	49%	75%	72%	72%
Simple Payback (from start of production)	Years	1.4	1.0	1.0	1.0

[1] Any minor discrepancies in totals are due to rounding.
[2] Spot price as at 9th April 2024.

Sensitivity Analysis

Sensitivity analysis was completed by varying key parameters by 20%, except for recovery that was varied to 89% and 97%, being +/-4% representing the metallurgical study level of accuracy around a calculated recovery of 93%. The resultant changes to the base case after tax $NPV_{10\%}$ of NZ$486M is shown in Figure 32.

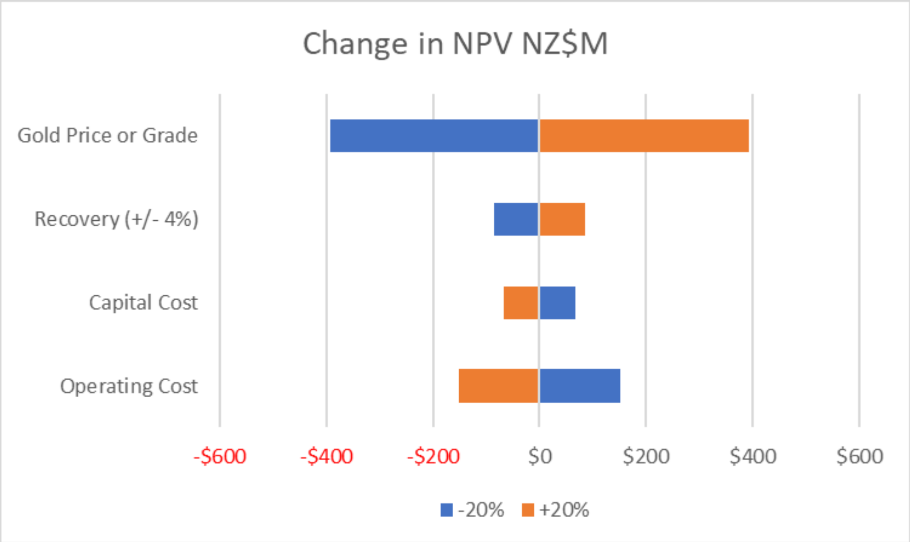

Figure 32 Change to base case of NZ$2,705/oz after tax $NPV_{10\%}$ of NZ$486M due to varying key project parameters

Further analysis was completed for the spot price of NZ$3,900/oz with an after tax $NPV_{10\%}$ of NZ$937M and is shown in Figure 33.

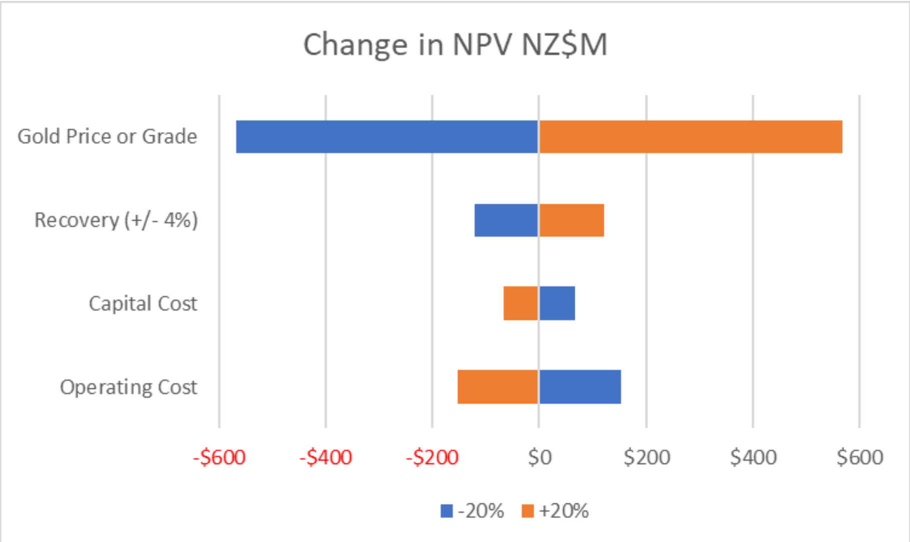

Figure 33 Change to spot price of NZ$3,900/oz after tax NPV10% of NZ$937M due to varying key project parameters

Funding

The Board reasonably believes funding will be available for development of the Project based on the following Study highlights, together with the following attributes of the Company and its Board's experience:

- The Study shows strong economics associated with Project development, including a very strong return on capital and robust cashflows, even at a base case gold price approximately US$690/oz below current spot gold prices. This provides a strong platform to source debt and equity funding.
- The Board of Santana has a strong track record in raising funds through debt and equity markets.
- The Project has a 10-year mine life generating significant free cash flow relative to the development capital required.
- The Study illustrates an NPV that is significantly value-accretive to current shareholder value.
- The Company has a tight capital structure and owns 100% of the Project, making potential financing arrangements uncomplicated.
- The Board has extensive experience in mine development and production in the resources industry which is attractive to potential financiers seeking certainty of project delivery.
- At record gold prices, global debt and equity finance availability for gold projects remains robust with several recent examples of funding being made available for ASX listed gold development projects.

There is, however, no certainty that the Company will be able to source funding as and when required. Typical project development financing would involve a combination of debt and equity. It is possible that such funding may only be available on terms that may be dilutive to or otherwise affect the value of the Company's existing shareholders.

Key Risks

The following key risks should be considered when reading this Scoping Study:

- Gold Price – downward fluctuations of the gold price, assuming a steady foreign exchange rate, can negatively impact project economics. Please see Figure 32 and 33 for sensitivity to the NPV when the gold price fluctuates.
- Permitting – the NZ government has introduced the Fast Track Bill which aims to reform the mine permitting process for projects of national significance, but there is no certainty that this legislation will be passed and there is no certainty that Bendigo-Ophir will be named as a project of national significance. This means the permitting process would likely default back to the provisions in the Resource Management Act 1991. Under this legislation the permitting process could be considerably protracted.

  Foreign Exchange – gold sales are denominated in the US dollar gold price, though project costs will be expensed in NZ dollars. Major changes in the USD:NZD exchange rate can impact project economics.

Mine Closure and Rehabilitation

In compliance with the Resource Management Act 1991 (RMA), our mine closure planning will prioritise robust closure and rehabilitation processes. Throughout feasibility and permitting phases, we will refine closure plans, integrating them into project design and securing necessary approvals.

Upon mining permit approval, implementation begins, focusing on land rehabilitation and post-closure monitoring as mandated by the RMA. Our commitment to regulatory compliance and environmental responsibility will ensure a sustainable legacy for the RAS gold project.

Conclusions and Next Steps

The Scoping Study provides preliminary validation that development of the RAS deposit results in a commercially viable, stand-alone mining operation. Therefore, the Board of Santana has approved the continuation of Pre-Feasibility Study work. Work will also continue on other identified opportunities that may enhance the project's economics.

Pre-Feasibility Study assessment work will focus on optimising the mining schedule, including firming up the site layout to minimise costs associated with haulage. Further studies will also look at minimising pre-production capital expenditure related to infrastructure and the pre-strip.

There is also strong potential to extend the proposed Study mine life through underground mining at RAS and by further defining other satellite deposits around RAS.

Reasonable Basis for Forward Looking Assumptions

No Ore Reserve has been estimated or declared for the Project. This document has been prepared in compliance with the JORC Code (2012) and the ASX Listing Rules. All material assumptions on which the Scoping Study production target and projected financial information are based have been included in this release and disclosed in the table below. The level of study does not support the estimation of Ore Reserves or provide any assurance that the Project will go ahead or be realised. The scoping study strongly supports progress to the next level of study being a preliminary feasibility study.

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	*Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	The Mineral Resource Estimate (MRE) on which the Scoping Study is based was announced to the ASX on 16th February 2024. No Ore Reserve has been declared as part of the Scoping Study.
Site Visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* *If no site visits have been undertaken indicate why this is the case.*	Hamish McLauchlan, the Competent Person for the reporting of exploration results is based on site. Kerrin Allwood, the Competent Person for the Estimation and Reporting of Mineral Resources visited the site in January 2021, in December 2022, and three times in 2023.
Study status	*The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* *The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically*	No Ore Reserve has been declared. The Study is a Scoping Study.

Criteria	JORC Code explanation	Commentary
	viable, and that material Modifying Factors have been *considered.*	
Cut-off parameters	*The basis of the cut-off grade(s) or quality parameters applied.*	Cut-off grade parameters are based on operating costs and site overheads.
Mining factors or assumptions	*The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).* *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).* *The mining dilution factors used.* *The mining recovery factors used.* *Any minimum mining widths used.* *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.* *The infrastructure requirements of the selected mining methods.*	No Ore Reserve has been declared. Refer to the Optimisation, Mine Design and Schedule Section of this report.
Metallurgical factors or assumptions	*The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* *Whether the metallurgical process is well-tested technology or novel in nature.* *The nature, amount and representativeness of metallurgical test work undertaken, the nature*	Refer to the Metallurgy and Processing Section of this report.

Criteria	JORC Code explanation	Commentary
	of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied. *Any assumptions or allowances made for deleterious elements.* *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.* *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	
Environmental	*The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	Refer to the Environmental Section of this report. Refer to the Permitting Section of this report.
Infrastructure	*The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided or accessed.*	BOGP is located approximately 20 kilometres north of the town of Cromwell, Central Otago which consists of 15km sealed highway and 5km of formed gravel road. BOGP is within an hour's drive from Queenstown to the southwest, and Alexandra to the southeast via Cromwell, and directly from Wanaka to the northwest. All towns have a range of accommodation and services available to support a residential workforce. Queenstown Airport has daily domestic and international flights direct to the east coast of Australia. Central Otago is connected via SH8 to Canterbury to the north, via SH8 to Dunedin, the provincial capital of Otago to the east, and via SH6 to the West Coast to the northwest and Southland to the southwest. Sufficient land is available within the project area to accommodate the infrastructure contemplated by this Scoping Study. Power and water requirements are discussed in this report.
Costs	*The derivation of, or assumptions made, regarding projected capital costs in the study.* *The methodology used to estimate operating costs.* *Allowances made for the content of deleterious elements.*	Capital costs: Project capital costs for the processing plant and non- process infrastructure were provided at ± 50% by experienced constructors for a 1,500 ktpa process plant option.

Criteria	JORC Code explanation	Commentary
	The source of exchange rates used in the study. *Derivation of transportation charges.* *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* *The allowances made for royalties payable, both Government and private.*	For the tailings storage facility, capital costs have been provided by experienced tailings storage facility engineers, and based on the mining and processing schedules. For all other capital costs including infrastructure and mining related costs have been estimated from similar NZ mining operations. Operating Costs: Operating costs have been provided by experienced NZ-based mining engineers for the planned scope of operations, with items estimated by company personnel derived from first principles and/or supplier quotes. Deleterious Elements: No allowance has been made for deleterious elements content on the basis that no deleterious elements have been detected. Exchange Rates: All costs were estimated in New Zealand dollars (NZD). Transportation Charges: It is assumed that gold doré will be transported from site for refining in Perth. Treatment and Refining: Treatment and refining charges in the financial model are based on market observations for similar products where available. Royalties: Refer to the Royalty Section of this report.
Revenue factors	*The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	The derivation of feed grades comes from the Mineral Resource estimates with the application of dilution modifying factors as outlined above. The product to be sold is gold in the form doré bars produced on site. A base case gold price of NZ$2,705/oz has been used and a spot price of NZ$3,900/oz.
Market assessment	*The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.*	Gold doré bars will be sold at spot price.

Criteria	JORC Code explanation	Commentary
	A customer and competitor analysis along with the identification of likely market windows for the product.	Not applicable
	Price and volume forecasts and the basis for these forecasts.	
	For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	
Economic	*The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	Refer to economic analysis, which assumes a discount rate of 10%, and nil inflation. Economic analysis includes a sensitivity analysis on various cost factors gold grade and gold price scenarios for both the base case and spot price scenarios.
Social	*The status of agreements with key stakeholders and matters leading to social licence to operate.*	Access arrangements are in place with the landowners upon where the project will be located. Consultation is ongoing with manu whenua and with the local community.
Other (incl Legal and Governmental)	*To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* *Any identified material naturally occurring risks.* *The status of material legal agreements and marketing arrangements.* *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre- Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	No Ore Reserve has been declared. No naturally occurring risks have been identified. The project is 100% owned by Santana Minerals Ltd via is 100%-owned NZ-subsidiary, Matakanui Gold Ltd and there are no marketing arrangements in place. All of the working area in the study is within the exploration permits with all work program obligations met. Refer to the Permitting Section of this report for the pathway to permitting. No Ore Reserve has been declared.
Classification	*The basis for the classification of the Ore Reserves into varying confidence categories.*	No Ore Reserve has been declared.
	Whether the result appropriately reflects the Competent Person's view of the deposit.	No Ore Reserve has been declared.
	The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	No Ore Reserve has been declared.
Audits or reviews	*The results of any audits or reviews of Ore Reserve estimates.*	No Ore Reserve has been declared.

Criteria	JORC Code explanation	Commentary
Discussion of relative accuracy/ confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the* *procedures used.* *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore.* *Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should* *be compared with production data, where available.*	No Ore Reserve has been declared. Metallurgical recoveries have been based on testwork data. Costs have been derived from both recent industry data and estimations from independent consultants and suppliers. Cost estimate accuracy for the Scoping Study is considered to be in the order of ±25%.