SANTANA MINERALS LIMITED

ASX RELEASE
ASX:SMI
NZX:SMI

15 November 2024

BENDIGO-OPHIR GOLD PROJECT
PRE-FEASIBILITY STUDY (+/-15%)

Long life, high margin gold development with low capex and significant upside

PRE-FEASIBILITY (PFS) HIGHLIGHTS:

- The project produces **147,000oz per annum** (first 3 years of production) and an average of 125,000oz per annum from an initial 9.2 years of Reserves, from Rise & Shine (RAS) and Srex (SRX) deposits only.
- The PFS enables a Probable Mining Reserve of 15.5Mt at 2.37g/t Au for 1.18Moz of gold to be reported. Open pit mining produces approximately one million ounces whilst the remainder comes from the initial phase of underground mining.
- A further ~770,000oz of Inferred resources at RAS and ~150,000oz at SRX and Come-in-Time (CIT) offer further known potential growth.
- The PFS estimates total production of **1.15Moz of gold** from open pit and underground mining at an average All-in-Sustaining-Cost **(ASIC) of A$1,416/oz,** at the current spot price of gold (A$4,000/oz)**.**
- A conventional 1.5Mtpa CIL process plant is designed to achieve an average 92.4% metallurgical recovery.
- Tailings are neutralised and stored in a waste-rock buttressed dam with closed circuit, process-water recirculation.
- The project generates **revenue of A$4.60 billion** at the current spot price of gold (A$4,000/oz) with an EBITDA of A$3.05 billion and **free cash flow of A$1.78 billion** after tax and royalties**.**
- Capex for plant and all associated mine infrastructure estimated at A$208M, whilst pre-production capex for the enlarged 39.5Mt pre-strip, to enable higher initial gold production, is A$132M. Total negative cash drawdown is estimated at A$340M.
- At the current spot price of gold (A$4,000/oz), an after-tax Net Present Value **(NPV_8) of A$1.06 billion** is generated over the initial term with an **IRR of 68%** and a simple **payback period of less than one year** from the commencement of production.
- At the base case price of gold (A$2,894/oz), an after-tax Net Present Value (NPV_8) of A$0.535 billion is generated over the initial term with an IRR of 42% and a simple payback period of 1.7 years from the commencement of production.
- At the current spot price of gold (A$4,000/oz) the New Zealand (NZ) government royalty payments are estimated at A$296M (NZ$325M) with corporate tax payments (28%) of a further A$728M (NZ$800M). These exclude payroll taxes and other indirect taxes which add up to show the fiscal significance of the project to NZ and the Central Otago regional economy.

Santana CEO, Damian Spring said:

"*We are pleased that our Pre-Feasibility Study with a higher level of accuracy has confirmed the robustness of the Bendigo-Ophir Gold Project previously outlined in our Scoping Study.*

We have upscaled the initial years of gold output commensurate with a deliberate decision to enlarge the mine pre-strip. Our detailed geotechnical works have recommended more conservative pit wall slopes than our scoping study resulting in higher strip ratios. However, our high-grade deposit combined with strong gold prices gives effect to a vastly improved post-tax NPV, valuing the project at multiples of the current market cap. We still believe we can improve the project from here and will be working to that end whilst advancing permitting.

Our team has worked tirelessly with detailed and diligent technical and baseline studies in line with previous Resource Management Act consenting. We are pleased that the certainty and significance of this project have now secured an opportunity to participate in the Fast-track Approvals process laid out by the New Zealand government. While we are confident we have met—and will continue to meet—all previously expected standards, the overwhelmingly positive economics of our proposed development add significant weight to our inclusion in this process and highlight the many benefits it brings to both the region and the nation. With around 40% of the company owned by Kiwis and nearly all of our employees residing in New Zealand, this project is truly shaping up to be a home-grown success story."

Development timetable post PFS:

Based on the robustness and large financial headroom as estimated in the PFS, the Board has elected to move onto a detailed construction plan with commencement of financing discussions.

Activities related to resource consents and mine permitting will continue with a view to applying into the Fast-track Approval process in February 2025.

Webinar

The Company's CEO Damian Spring will be hosting a webinar to present the outcomes of the PFS to investors at **10:30am (AEDT)/12:30pm (NZDT) on Friday, 15 November.** Registration is required prior to entry into the webinar, which can be accessed by following this link: **WEBINAR LINK**.

Ore Reserve Statement

The Santana Board is pleased to announce an Ore Reserve Estimate (ORE) at the wholly owned Bendigo-Ophir Gold Project (BOGP) in New Zealand. The BOGP JORC 2012 compliant ORE is 15.5 million tonnes @ 2.37g/t Au for 1.181 million ounces of gold. This ORE is based on a Mineral Resource Estimate (MRE) of 40.3 million tonnes @ 1.9g/t Au for 2.46 million ounces reported at a 0.25g/t cut-off grade. The July 2024 MRE of 36.8 million tonnes @ 2.1g/t Au for 2.46 million ounces was reported at a cut-off grade of 0.5g/t. The lower cut-off grade at 0.25g/t reflects the economic outcomes of this PFS.

The BOGP ORE is tabled below:

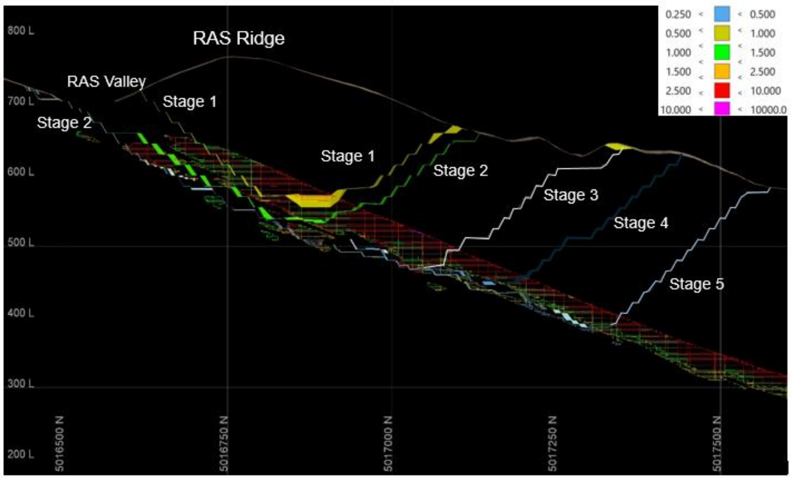
Area	Proven		Probable		Total		
	Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au koz
RAS (open pit)	-	-	11.9	2.42	11.9	2.43	928
RAS (Underground)			2.3	3.03	2.3	3.03	223
SRX			1.3	0.70	1.3	0.70	30
Total	-	-	15.5	2.37	15.5	2.37	1,181

Ore Reserve Statement

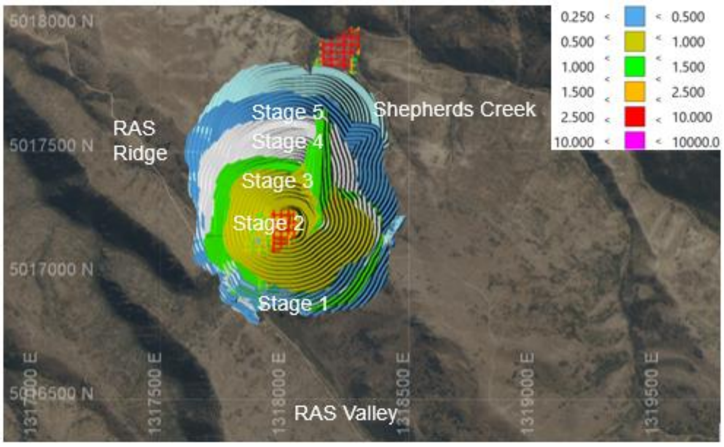
Note 1: RAS Open pit cut-off grade 0.3 g/t at $US1,650/oz Au price
Note 2: RAS Underground cut-off grade 1.75 g/t at $US1,650/oz Au price
Note 3: SRX Open pit cut-off grade 0.35 g/t at $US2,100/oz Au price
Note 4: Underground Reserves are from the quoted Open pit Resources area
Note 5: The effective date of the Mineral Reserve is 1 November 2024, estimated by Rodney Redden (MAusIMM and CP-Mining), a contractor to Santana Minerals Ltd.
Note 6: Approved consents and required permits are yet to be granted to enable mining of the RAS and SRX deposits.

Cautionary Statement

Of the Mineral Resources planned for extraction under the PFS production model approximately 94% is within the Indicated Resource category, with the balance (6%) being classified within the Inferred Resources category. 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 Indicated Mineral Resources or that the production target itself will be realised.



RAS Long Section Looking North - Open Pit Stages 1 to 5



RAS Plan View – Open Pit Stages 1 to 15

SANTANA MINERALS LIMITED

Key PFS Data

The key outcomes of the PFS are summarised in the following tables and charts with full cost and input information in the *PFS Summary* attached. Financial projections are presented using a spot gold price of A$4,000/oz. A robust base-case study using A$2,894/oz is also presented and compared in Table 3.

Total mining physicals underpinning all financials can be seen below in Table 1.

Key Project Mining Physical Targets and Assumptions		
Mine Life	Years	9.17
Plant Throughput	ktpa	1,835
Open Pit Ore Mined	kt	14,404
Open Pit Mill Feed	kt	14,404
Open Pit Mill Feed Grade	Au g/t	2.19
Open Pit Contained Gold	kOz	1,014
Open Pit Recovered Ounces	kOz	935
Underground Ore Mined	kt	2,413
Underground Mill Feed	kt	2,413
Underground Mill Feed Grade	Au g/t	2.99
Underground Contained Gold	kOz	232
Underground Recovered Ounces	kOz	215
Total Ore Mined	kt	16,817
Total Mill Feed	kt	16,817
Au Grade - Mined	g/t	2.30
Total Contained Gold	koz	1,245
Overall Plant Recovery	%	92.38%
Gold Production	**kOz**	**1,151**

Table 1. Key mining physicals

Table 2 below shows the 'per tonne', 'per ounce', and total combined cost for open pit and underground mining, with processing costs, G&A, selling costs, royalties, and sustaining CAPEX, to show the average project C1 cash costs and average AISC costs per ounce when using the spot price for gold of A$,4000/oz.

Operating Costs breakdown		AUD '000	AUD /T Milled	AUD /Oz Produced
Mining Cost	'000	771,984	45.9	671
Processing Costs	'000	288,943	17.2	251
General and Admin Costs	'000	55,633	3.3	48
C1 Cash Cost	'000	**1,116,559**	**66.4**	**970**
Selling Cost	'000	8,357	0.5	7
Royalties - Govt	'000	296,305	17.6	258
Royalties - Others	'000	123,887	7.4	108
Closure Capex (see note 1)	'000	-	-	-
Sustaining Capex	'000	83,663	5.0	73
All-in Sustaining Cost (AISC)	'000	**1,628,771**	**96.9**	**1,416**

Note 1: Conceptual mine closure costs netted to zero against mine salvage value.

Table 2. Gold production costs in C1 and AISC, per tonne, per ounce, and total A$.

Table 3 below follows the Scoping Study cash flow template (announced 17 April 2024), splitting out open pit mining costs and underground mining costs and building up totals to show a Total Production Cost per ounce, which **includes all pre-production CAPEX**. The Total Production Cost per ounce in the PFS, at spot gold prices is A$1,818/oz, as compared to the Scoping Study of $1,265/oz. The increase includes a government royalty of $186/oz, based on the higher profitability of the project at current spot gold prices, and by applying the royalty rate at the higher 10% of accounting profits (previously 2% NSR rate applied in the Scoping Study).

A base-case scenario underpinned the PFS at an Australian dollar gold price of A$2,894/oz, a ~28% discount to current spot gold prices, as at 12 November 2024. Financial projections at the spot gold price of A$4,000/oz are also reported in NZD and USD:

Key Financial Assumptions		Base Case AUD	Spot AUD	Spot NZD	Spot USD
Gold Price	$/oz	2,894	4,000	4,406	2,626
Exchange Rate	USD:$	0.66	0.66	0.60	1.00
Key Project Metrics					
Gold Produced	Oz		1.15 million		
Initial Mine Life	Yr(s)		9.17		
Gold Revenue	'000	**3,330,018**	**4,602,435**	**5,069,319**	**3,021,314**
Open Pit Mining Cost	'000	619,237	619,237	682,054	406,504
Underground Mining Cost	'000	152,747	152,747	168,242	100,272
Processing Costs	'000	288,943	288,943	318,254	189,679
General and Admin Costs	'000	55,633	55,633	61,276	36,521
Selling Cost	'000	8,357	8,357	9,204	5,486
Royalties - Govt	'000	170,173	296,305	326,363	194,512
Royalties - Others	'000	89,636	123,887	136,454	81,327
Total Cash Operating Cost	'000	**1,384,725**	**1,545,108**	**1,701,848**	**1,014,301**
Total Cash Operating Cost per Ounce	$/oz	**1,203**	**1,343**	**1,479**	**881**
Project EBITDA	'000	1,945,292	3,057,327	3,367,471	2,007,013
Depreciation and Amortisation (exc Rehab PPE)	'000	546,067	546,067	601,462	358,471
Total Production Cost (incl. all CAPEX)	'000	**1,930,793**	**2,091,175**	**2,303,310**	**1,372,773**
Total Production Cost per Ounce	$/oz	**1,678**	**1,818**	**2,001**	**1,193**
Net Profit Before Tax (NPBT)	'000	1,399,225	2,511,260	2,766,009	1,648,541
Tax Payable (28.0%)	'000	(424,010)	(728,094)	(801,954)	(477,965)
After Tax Profit	'000	**975,215**	**1,783,166**	**1,964,055**	**1,170,577**
Capital					
Initial Development Capex (inc. OP & Capitalised Opex)	'000	340,609	340,609	375,161	223,596
Underground Initial Development Capex	'000	121,795	121,795	134,151	79,954
Sustaining Capex	'000	83,663	83,663	92,151	54,922
Closure Capex (see note 1)	'000	-	-	-	-
Total CAPEX over Mine Life	'000	**546,067**	**546,067**	**601,462**	**358,471**
DCF Outcomes					
Initial NPV (unleveraged and after-tax) @8.00%	'000	534,975	1,058,104	1,165,441	694,603
IRR	%	41.66%	68.23%	68.23%	68.23%
Payback Period from production start (unleveraged and after-tax)	years	1.67 Yr(s)	0.92 Yr(s)	0.92 Yr(s)	0.92 Yr(s)

Note 1: Conceptual mine closure costs netted to zero against mine salvage value.

Table 3. Base Case vs Spot Gold and Currency Values

The chart below shows the gold production profile and AISC in the main production years, after the pre-production and commissioning phase, applying the spot gold price scenario. Approximately 150koz is produced from years one to three, bolstering upfront cash flows:

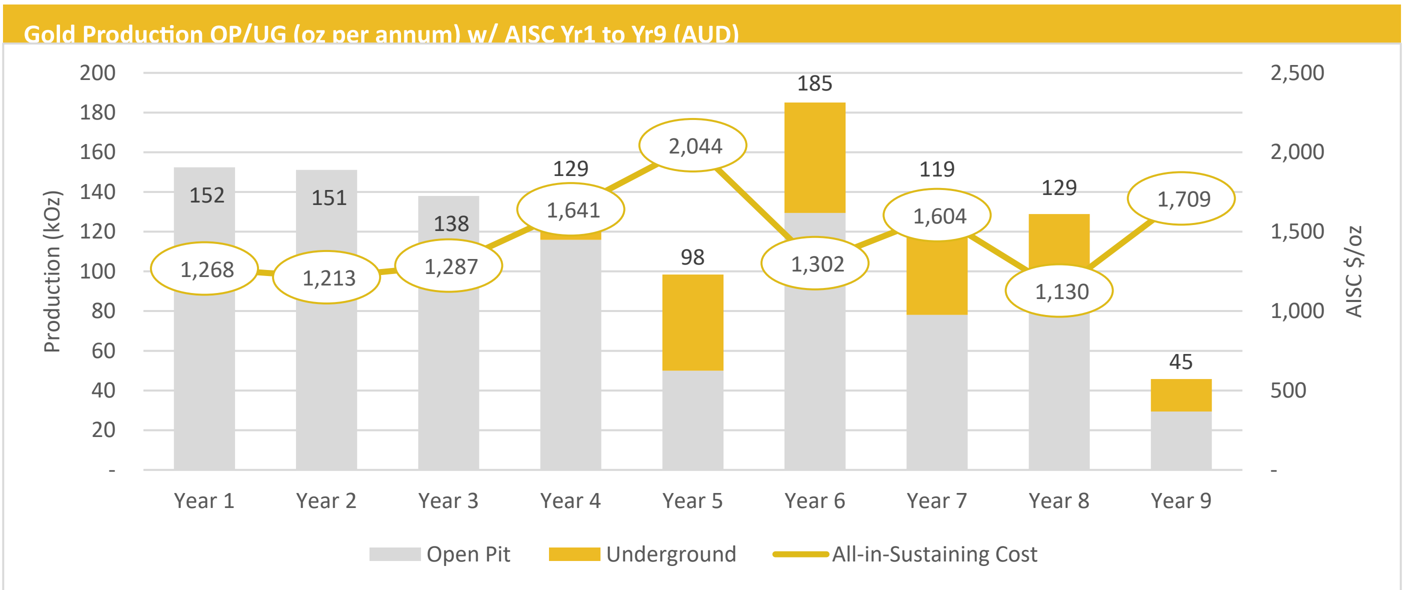


Figure 1. Production Profile OP/UG w/AISC at Spot Gold Price

At the spot gold price of A$4,000/oz, the project generates nearly A$1.8 billion in free cash flow over the initial mining term. The chart below shows the max cash draw down in the pre-production period, followed by very high returns in the first years of gold production, allowing a <1yr payback from first production.

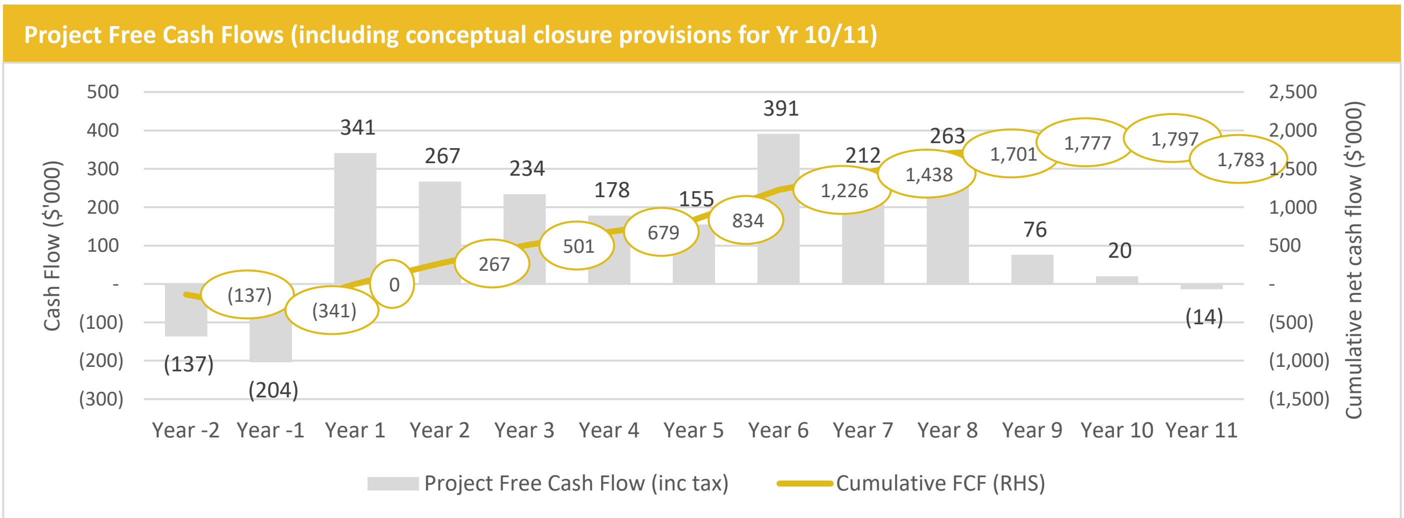


Figure 2. Project Free Cash Flows at Spot Gold Price

The PFS targeted Indicated resources to forecast the mine's economics. Approximately 94% of tonnes constituting ore feed are in the Indicated resource category, as seen in Figure 3 below:

Indicated vs Inferred Ore Feed (94% Indicated resources into PFS schedule)

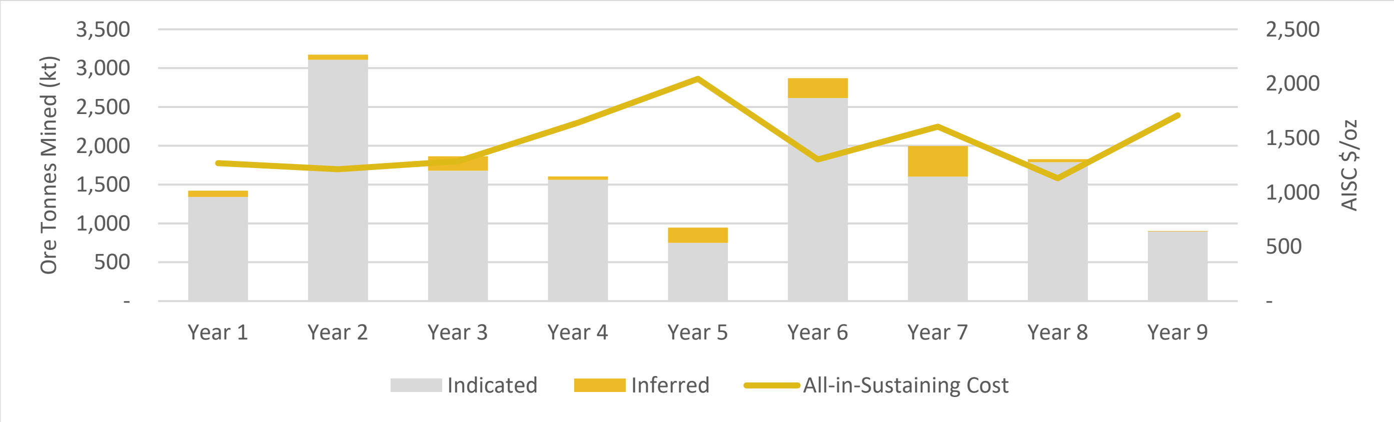


Figure 3. Indicated vs Inferred Ore Feed

The tornado chart below shows the NPV sensitivity analysis at the Base-Case gold price scenario.

NPV$_8$ Sensitivity analysis ($'000) based on the Base-Case gold price scenario at A$2,894/oz

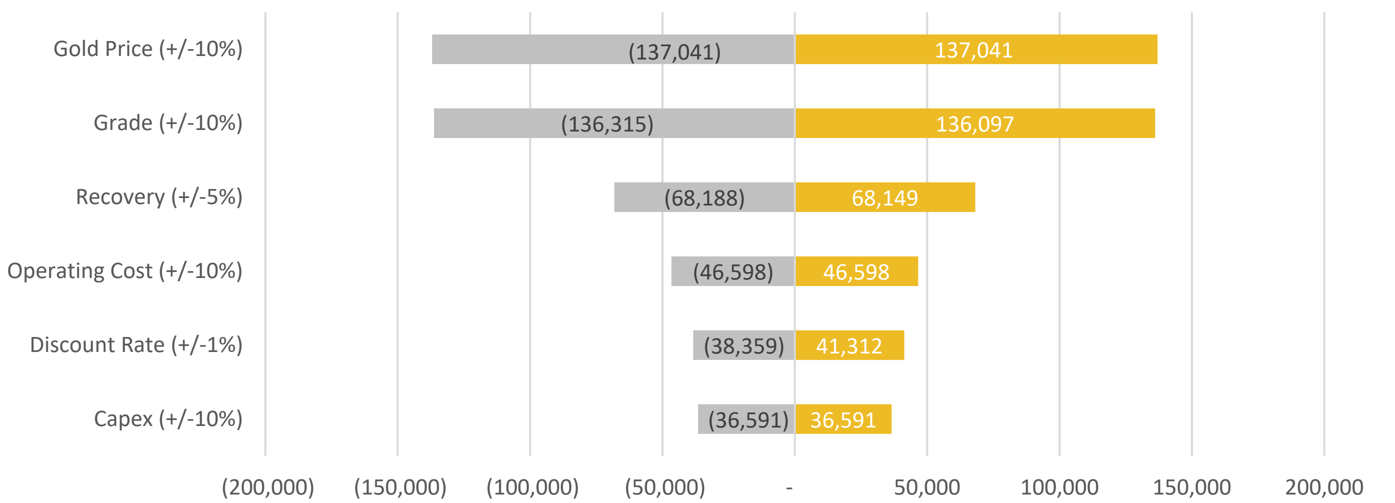


Figure 4. NPV$_8$ sensitivity analysis on the Base-Case gold price scenario at A$2,894/oz

The table below shows the NPV, IRR and Payback metrics at price steps above and below the Base-Case and Spot price scenarios applied in the PFS:

	-A$200/oz	Base-Case	Spot Price	+A$200/oz
	A$2,694/oz	A$2,894/oz	A$4,000/oz	A$4,200/oz
NPV$_8$	$440M	$535M	$1.06b	$1.15b
IRR	36%	42.00%	68%	73%
Payback	1,83Yrs	1.67Yrs	0.92Yrs	0.83Yrs

Table 4. Sensitivities on NPV, IRR and Payback metrics based on gold price movements.

Please see the PFS Executive Summary appended below for more detail. This announcement has been authorised for release by Santana's Board of Directors.

Enquiries:

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Cautionary Statement

The Preliminary Feasibility Study (PFS) discussed in this ASX announcement has been conducted to assess the potential development of the Bendigo-Ophir Gold Project in New Zealand. Of the Mineral Resources planned for extraction under the PFS production model, about 94% are categorized as Indicated, with the remaining 6% classified as Inferred over the 9.17-year assessment timeframe. The Company believes it has a reasonable basis to disclose a production target that includes some Inferred Mineral Resources as the Inferred Resources are not a determining factor in the viability of the Project. However, it acknowledges that there is a low level of geological confidence associated with Inferred Resources and that there is no certainty that further exploration will result in the determination of Indicated Mineral Resources or that the production target itself will be realised. Importantly, the feasibility of the development scenario outlined in the PFS does not hinge on the Inferred Mineral Resources. Additionally, Ore Reserves are based solely on Indicated resources.

This announcement includes forward-looking statements. The Company has concluded that it has a reasonable basis for those forward looking statements, including the production target set out in the PFS and the financial information on which it is based. This basis is detailed throughout the release, with all critical assumptions, including the JORC modifying factors, on which the forward looking statements rely, is fully disclosed in this release. Nonetheless, several variables could cause actual outcomes to vary significantly from those suggested by the forward-looking statements. Given these uncertainties, investors are cautioned against making investment decisions based purely on the PFS findings.

To achieve the range of outcomes anticipated in the PFS, the PFS estimates that financing on the order of A$340 million will be required. Santana believes that there are reasonable grounds for the assumptions it has made in satisfying itself that the requisite funding for the development of the Project will be available when required. However, Shareholders and prospective investors should be aware that there is no guarantee Santana will be able to secure this funding as required, and it is possible that the terms available may be dilutive or adversely otherwise impact the value of Santana's current shares. Additionally, Santana may explore alternative value-creating strategies, such as divesting some or all potential revenue streams from precious metals or a full or partial sale of its interest in the Bendigo-Ophir project.

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 "Bendigo-Ophir Gold Resources Increased 155% to 643k Oz" dated 28 September 2021
- ASX announcement titled "1.3m ounces upgraded to Indicated category from RAS drilling" dated 16 February 2024
- ASX announcement titled "Outstanding Economics - RAS Scoping Study (First 10 Years)" dated 17 April 2024
- ASX announcement titled "Infill drilling increases RAS Indicated category to 1.45Moz" dated 2 July 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 referenced above and, in the case of the Mineral Resource estimates, that all material assumptions and technical parameters underpinning the Mineral Resource estimates in the relevant announcements continue to apply and have not materially changed. The Company confirms that the form and context in which the Competent Person's findings are presented have not been materially modified from the original market announcements.

Current Disclosure - Competent Persons Statement

The information in this report that relates to the July 2024 RAS Mineral Resource Estimates (MRE) and to this November 2024 SRX and SRE MRE, is based on work completed by Mr Kerrin Allwood, a Competent Person (CP) who is a Member of The Australasian Institute of Mining and Metallurgy (AusIMM). Mr Allwood is a Principal Geologist of GeoModelling Limited, Petone, New Zealand and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which is being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the “Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves”. Mr Allwood consents to the inclusion in this report of the matters based on his information in the form and context in which it appears. Mr Allwood and GeoModelling Limited are independent of Santana Minerals Ltd.

The information in this report that relates to the prior 2021 Mineral Resource Estimates (2021 MRE) for the CIT deposit completed by Ms Michelle Wild (CP) (ASX announcement on 28 September 2021) continue to apply and have not materially changed.

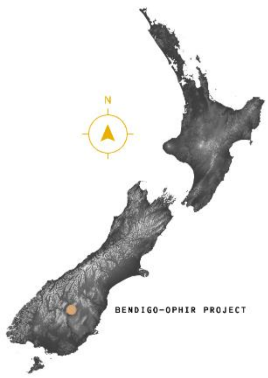
The estimated Ore Reserves underpinning the production target set out in this Announcement have been prepared by a Competent Person as defined in the 2012 Edition of the “Australasian Code of Reporting of Exploration Results, Mineral Resources and Ore Reserves”, estimated by Rodney Redden (MAusIMM and CP-Mining), a contractor to Santana Minerals Ltd.

The information in this report that relates to the Ore Reserves for Rise and Shine (RAS), Srex (SRX) and Srex East (SRE) is based on and fairly represents information and supporting documentation compiled by Mr Rodney Redden. Mr Redden is an Associate of Redden Mining Limited, a full time contractor to Matakanui Gold Limited (a wholly owned subsidiary of Santana, and is a Chartered Professional Mining Engineer of the Australian Institute of Mining and Metallurgy. Mr Redden has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity currently being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the “Australasian Code of Reporting of Exploration Results, Mineral Resources and Ore Reserves”. Mr. Redden consents to the inclusion in this report of the matters based on the information in the form and context in which it appears. The Company confirms that the form and context in which the Competent Persons’ findings are presented have not been materially modified.

Forward Looking Statements

Forward-looking statements in this announcement include, but are not limited to, statements with respect to Santana’s plans, strategy, activities, events or developments the Company believes, expects or anticipates will or may occur. By their very nature, forward-looking statements require Santana to make assumptions that may not materialize or that may not be accurate. Although Santana believes that the expectations reflected in the forward-looking statements in this announcement are reasonable, no assurance can be given that these expectations will prove to have been correct, as actual results and future events could differ materially from those anticipated in the forward-looking statements. Accordingly, viewers are cautioned not to place undue reliance on forward-looking statements. Santana does not undertake to update publicly or to revise any of the included forward-looking statements, except as may be required under applicable securities laws.

BENDIGO-OPHIR PRE-FEASIBILITY STUDY SUMMARY



Disclaimer

All information contained in this presentation is of a general nature. Potential investors are cautioned against using the content of this presentation, in isolation, for making investment decisions and should also refer to Santana Minerals Limited ('Santana') Annual Reports and ASX:SMI releases. For further information about Santana visit our website at www.santanaminerals.com.

Best efforts have been made to ensure the accuracy of information contained (at the time of preparation). Where forward targets and/or assumptions have been included – all such instances are indicative only and subject to alteration and/or cancellation as and when the management of Santana determines.

Research and advice of a qualified financial advisor or accountant are strongly recommended to anyone considering investing in listed company securities, including those of Santana.

The Prefeasibility Study, including the production target and the forecast financial information derived from the production target, referred to in this Presentation (PFS) was released to the ASX on 15 November 2024. This Presentation includes summary excerpts from the PFS and does not purport to be all-inclusive or complete.

Forward-Looking Statements

This Presentation contains various forward looking statements. Forward-looking statements in this presentation include, but are not limited to, statements regarding the production target, financial information based on that production target and statements statements with respect to Santana's future plans, strategy, activities, events or developments the Company believes, expects or anticipates will or may occur. By their very nature, forward-looking statements require Santana to make assumptions that may not materialize or that may not be accurate. The Company has concluded that it has a reasonable basis for providing these forward-looking statements, including the production target and the forecast financial information included in this Presentation. The detailed reasons for these conclusions are outlined throughout the ASX releases dated 15 November 2024. However, no assurance can be given that these expectations will prove to have been correct, as actual results and future events could differ materially from those anticipated in the forward-looking statements. Accordingly, viewers are cautioned not to place undue reliance on forward-looking statements. Santana does not undertake to update publicly or to revise any of the included forward-looking statements, except as may be required under applicable securities laws.

To achieve the range of outcomes anticipated in the PFS, the PFS estimates that financing in the order of A$340 million will be required. Santana believes that there are reasonable grounds for the assumptions it has made in satisfying itself that the requisite funding for the development of the Project will be available when required. However, Shareholders and prospective investors should be aware that there is no guarantee Santana will be able to secure this funding as required, and it is possible that the terms available may be dilutive or otherwise adversely impact the value of Santana's current shares. Additionally, Santana may explore alternative value-creating strategies, such as divesting some or all potential revenue streams from precious metals or a full or partial sale of its interest in the Bendigo-Ophir project.

Cautionary Statement – Inferred Resources Included in Production Target

Of the Mineral Resources planned for extraction under the PFS production model approximately 94% is within the Indicated Resources category and is classified Probable Reserves, with the balance (6%) being classified within the Inferred Resources category. 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 Indicated Mineral Resources or that the production target itself will be realised.

Competent Persons Statement

The production target and the forecast financial information derived from the production target set out in this presentation were first contained in a public announcement released to the ASX on 15 November 2024. The Company confirms that all material assumptions underpinning the production target and the forecast financial information derived from it continue to apply and have not materially changed.

The information in this report that relates to Mineral Resources is based on information contained in the following public announcements:

15 November – ASX Announcement titled "Bendigo-Ophir Pre-Feasibility Study"

2 July 2024 – ASX Announcement titled "Infill drilling increases RAS Indicated category to 1.45Moz"

16 February 2024 – ASX Announcement titled "1.3M ounces upgraded to Indicated category from RAS drilling:"

28 September 2021 – ASX Announcement titled "Bendigo-Ophir Gold Resources Increased 155% to 643k Oz"

The information in this report that relates to Ore Reserves is based on information contained in the public announcement made to the ASX on 15 November 2024.

The information in this report that relates to Exploration Results is based on information contained in the following public announcement:

22 August 2022 – ASX Announcement titled "MDD054 Jewellery Box Re-Assays to 1400 g/t Gold"

A copy of these announcements are available to view on the Santana Minerals Limited website www.santanaminerals.com or on the ASX platform www.asx.com.au.

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 referenced above and, in the case of the Mineral Resource estimates, that all material assumptions and technical parameters underpinning the Mineral Resource estimates in the relevant announcements continue to apply and have not materially changed. The Company confirms that the form and context in which the Competent Person's findings are presented have not been materially modified from the original market announcements.

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EXECUTIVE SUMMARY

Based on a spot gold price of A$4,000/oz

ROBUST RETURNS

A$1.06 billion NPV_8 (after tax)
68% IRR (after tax)
<1yr payback (from production)

HIGH GRADE

>3g/t from RAS in early stages
2.3g/t on average for project
150koz pa in Yr1 & Yr2

HIGH MARGIN

>A$1,500/oz margin after-tax
After all costs and full depreciation of CAPEX

RESPONSIBLE

Strong environmental focus
Avoid, mitigate, offset strategy
Mining consents in line with strict NZ criteria

LOW HURDLE

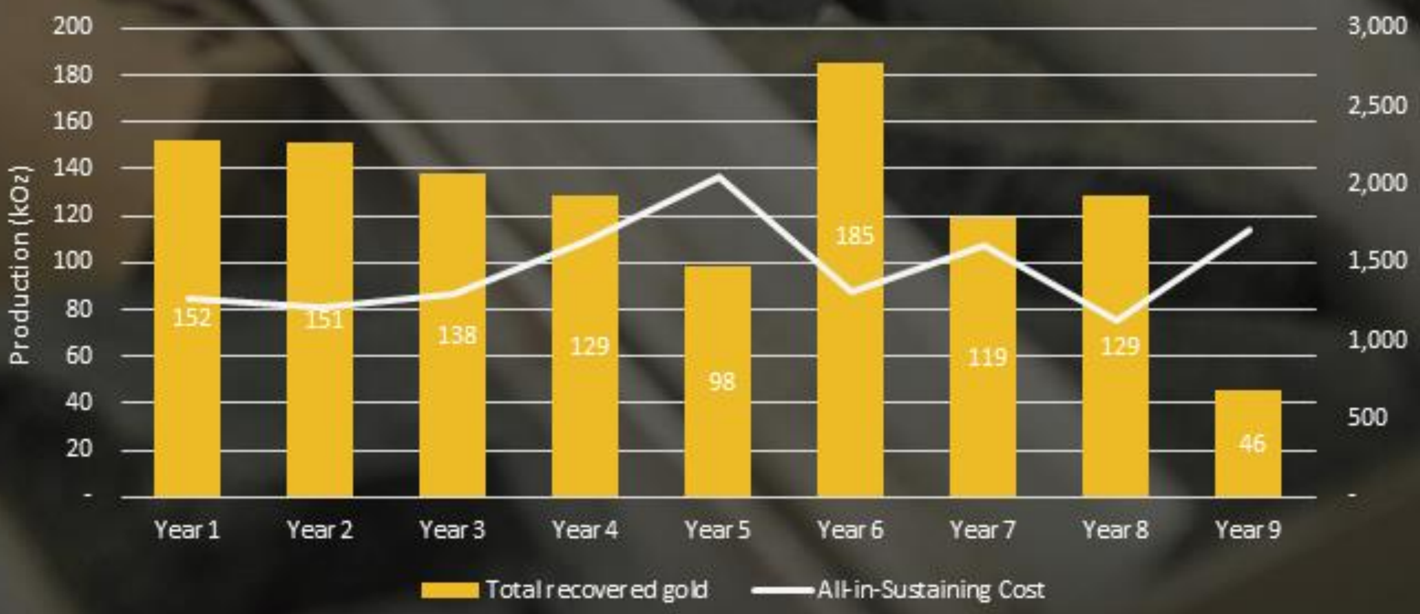
A$340M max cash drawdown
A$1,416/oz Au AISC
Surrounded by existing workforce & services

SIGNIFICANT

Designated by NZ Government
Nationally & regionally significant
Fast Track Approval eligibility

The Bendigo-Ophir Gold Project PFS demonstrates a highly profitable gold mine with an initial nine years of production, starting at 1.5Mtpa throughput, moving to 2.1Mtpa throughput, for an average of 125koz of gold produced per annum.

Annual Gold Production



Executive Summary

ES 1. Introduction, Background and History

ES 1.1 Introduction

The Bendigo Ophir Gold Project (BOGP) as presented in the Preliminary Feasibility Study (PFS) will be an open pit and underground mining operation based on the Rise and Shine (RAS) orebody with satellite feed from the Srex (SRX) orebody.

The study has been prepared by selecting a base-case scenario modelled on a conservative gold price of US$1,900/oz. To forecast more current economics, a spot gold price scenario was also modelled at US$2,626/oz. The key inputs, including foreign exchange rates, mining and processing costs, mining parameters, and all associated financial outcomes of the study can be found in section *ES14 Financial evaluation*.

At the current spot price of gold (A$4,000/oz), an after-tax Net Present Value (NPV8) of A$1.06 billion is generated over the initial term with an IRR of 68% and a simple payback period of less than one year from the commencement of production.

At the base case price of gold (A$2,894/oz), an after-tax Net Present Value (NPV8) of A$0.535 billion is generated over the initial term with an IRR of 42% and a simple payback period of 1.7 years from the commencement of production.

The primary RAS orebody underpins the PFS with grades and thicknesses that support an underground mining operation beneath the economic limits of the open pit. The processing rate will initially be 1.5Mtpa, expanding to 2.1Mtpa when the underground is brought into production in parallel with on-going open pit feed sources. The total processing inventory is 16.8Mt at 2.3g/t for 1,245koz. 94.9% of the feed ounces are from Indicated resources. Inferred resources mined are only as a consequence of mine designs optimised to target Indicated resources only.

The RAS orebody is estimated to have a total recovery to doré of 93% of the contained metal. SRX recovery is 68% through the process flow sheet which is optimised on RAS mineralisation.

Significant Inferred resources remain west of and down dip at RAS, and also at SRX and Come-in-Time (CIT) deposits that have not been included in this PFS.

Infill drilling at CIT deposit was incomplete prior to the most recent Mineral Resource Estimate (MRE) update.

The processing flowsheet is a simple single stage crush, grind, gravity concentrate, carbon-in-leach (CIL), elution to a conventional wet tailings storage facility (TSF). Doré will be the final product. The TSF will be completely buttressed during and post operations by the engineered landform (ELF) comprising waste material from the RAS pit.

There is nearby access from the national road network, along with fresh water and high voltage power, all readily accessible from just outside the project area. An initial construction camp will be established to cater for temporary personnel involved in establishing the project. Operations staff will be based in any of the multiple local communities within an hour’s drive of the project.

The overall site layout is shown in Figure ES 1.

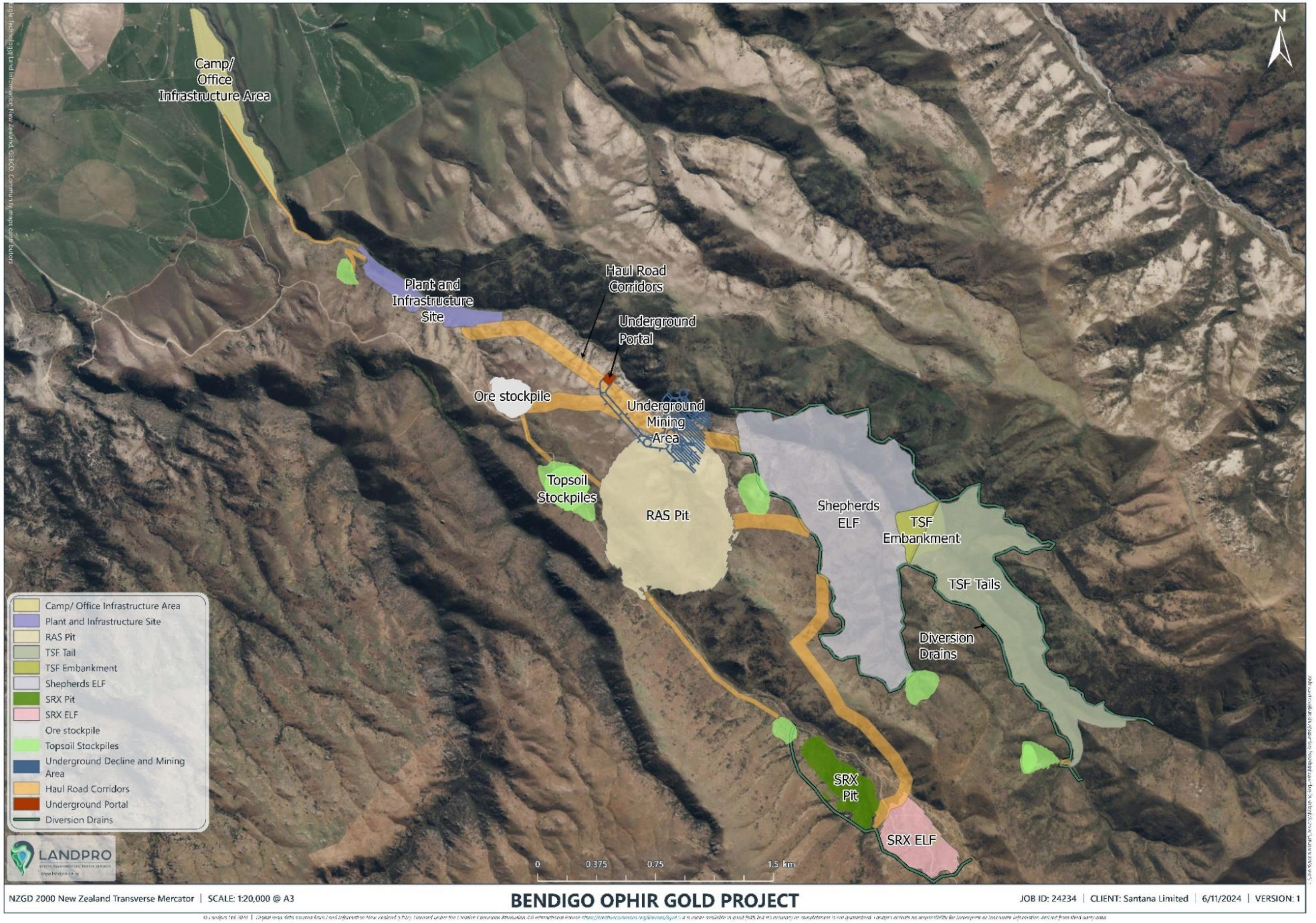


Figure ES 1: BOGP general site layout

Figure ES 2: BOGP Location

The BOGP is sited in the Dunstan Mountains of Central Otago, South Island, New Zealand (NZ) (see Figure ES 2) within the territorial authorities of Central Otago District Council (CODC) and Otago Regional Council (ORC).

The local area is known as Bendigo, named in the 1860s by miners arriving from the Australian (Bendigo, Victoria) goldfields. The Dunstan Mountains comprise large pastoral holdings. Vineyards and cherry orchards are developed on terraces flanking the northwestern margin of the Dunstan Mountains.

The preferred route to the site from Cromwell, the nearest main centre, is via State Highway 8 (SH8) to the Ardgour Road turn-off (24km from Cromwell), then via Ardgour Road, before travelling along Thomsons Gorge Road (TGR) for 6.5km. Finally, a new road into Shepherds Valley will provide access to the process plant site.

ES 1.3 Background and History

The area's mining activity dates back to the Otago gold rush of the 1860s, when the discovery of gold in Central Otago attracted miners from around the world. Gold mineralisation is widespread within the Otago Schist, with over 5 million ounces of hard-rock gold, and 8 million ounces of alluvial gold, being won from Otago goldfields.

The Bendigo field quickly became one of the most significant quartz reef mining areas in Otago, with approximately 300,000 ounces of gold mined from the Bendigo goldfield and its surrounding areas up to the 1940s.

In 2012 Depot Corporation Ltd, led by Santana Director, Kim Bunting, began regional exploration in the RAS valley using track and trail regolith mapping, rock chip sampling, and portable XRF soil geochemistry. In 2014 Depot Corporation vended its exploration permits to Matakanui Gold Limited (MGL) and drilling operations were conducted at CIT, RAS and SRX in 2018 and 2019.

In 2020, ASX-listed Santana Minerals Ltd bought MGL with initial diamond drilling commencing immediately thereafter. Drilling continued in 2021, when the RAS discovery hole (MDD007) was drilled at 40.3m @ 2.05g/t gold.

ES 2. Current Permits and Land access

The BOGP sits within Mineral Exploration Permit (MEP) 60311.

MEP60311 is owned by Matakanui Gold Limited (MGL) (NZBN 9429041420614) which is a NZ-registered company, and a fully owned subsidiary of Santana Minerals Limited (SMI) (ACN 161 946 989), an Australian-registered company listed on the Australian Stock Exchange (ASX.SMI) and the New Zealand Stock Exchange (NZX.SMI).

Land ownership across the project area is freehold, private land, with Bendigo Station to the southwest (SW) and Ardgour Station to the northeast (NE). Agreements are in place with both Bendigo and Ardgour stations that allow the project to proceed, subject to project consents and company approval, through a mix of agreements relating to:

- Purchase;
- Lease agreements; and
- Mining terms with royalty structures.

ES 3. Geology

Gold mineralisation occurs along the Rise and Shine Shear Zone (RSSZ) within the Otago Schist (Figure ES 3). The Otago Schist is formed from sedimentary and minor intermediate volcanics and volcaniclastics of the Caples and Rakaia tectono-stratigraphic terranes. Schist protolith rocks were deposited in a forearc setting along the paleo-Pacific convergent margin of Gondwana between ca. 250 and 200 Ma.

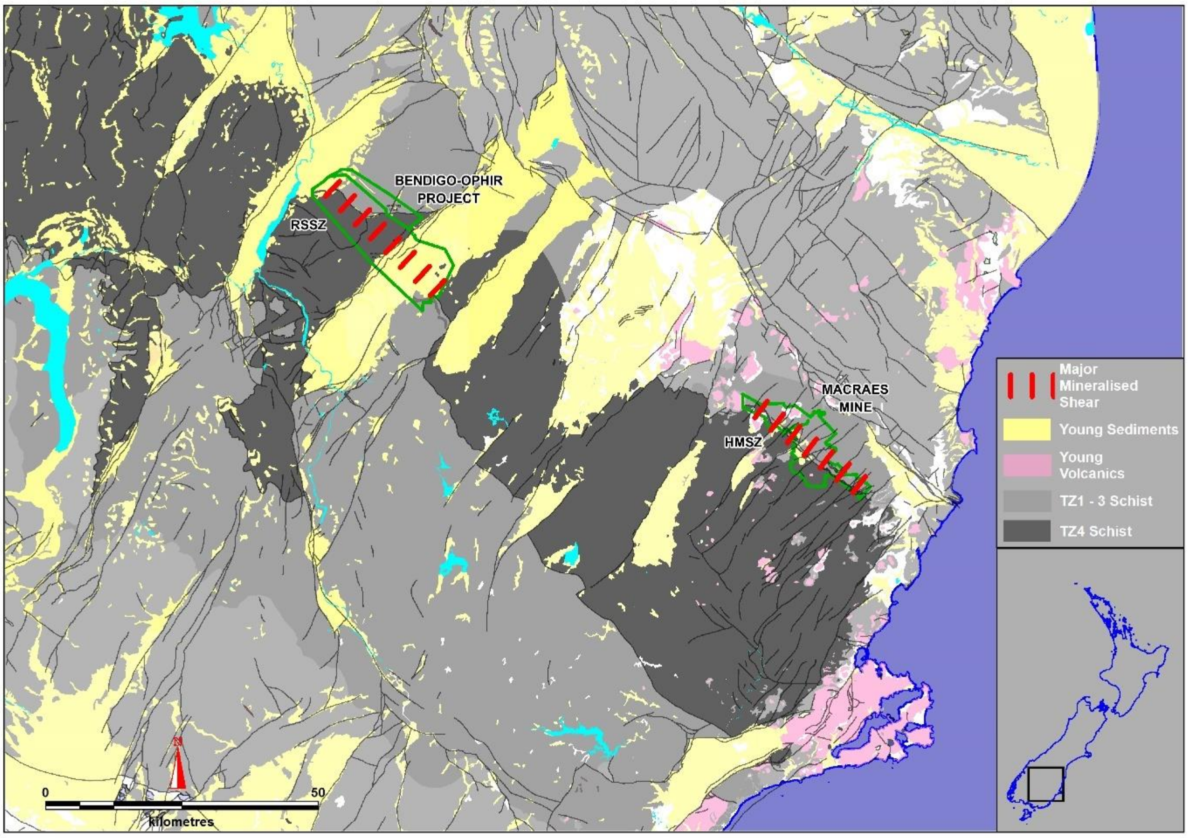


Figure ES 3: Regional Setting of the Bendigo-Ophir Project Area (BOGP)

ES 3.1 Local Geology

The Dunstan Mountains are an uplifted block of the Otago Schist, tilted towards the northwest, with remnants of a Cretaceous peneplain preserved on its northwest slope.

The largely coplanar RSSZ and younger Thomsons Gorge Fault (TGF) cut across the axis of the Dunstan Mountains. The TGF juxtaposes lower greenschist facies TZ3 and mid to upper greenschist facies TZ4 schists. Both the TGF and the RSSZ strike northwest and dip approximately 25° towards the northeast.

The TGF across the RSSZ is an unmineralised 0.3m – 12m thick cataclastic, fault zone separating biotite zone schists in the southwest (footwall, TZ4) from chlorite zone schists in the north-east (hanging-wall, TZ3). The TGF is locally extremely planar. Later NE-striking faults (i.e. Norms Fault) displace the TGF by 10's to 100's of metres.

The RSSZ occurs in TZ4 schists of the footwall of the TGF and is a zone up to 200m thick of low-angle, late-metamorphic, silicified brittle shears within greenschist facies psammitic, pelitic and meta-volcanic rocks of the Mesozoic Otago Schist Group. The RSSZ silicified shears are termed silica breccias (SBX).

The RSSZ dips 20-30° to the north-east and generally crosscuts the metamorphic foliation at a low angle.

ES 3.2 Deposit Geology and Mineralisation

Gold mineralisation is concentrated in four deposits along the shear zone. The deposits known along the RSSZ are: Come in Time (CIT), Rise and Shine (RAS), Srex (SRX), all approximately 1km apart, and Srex East (SRE), located about 200m east of SRX.

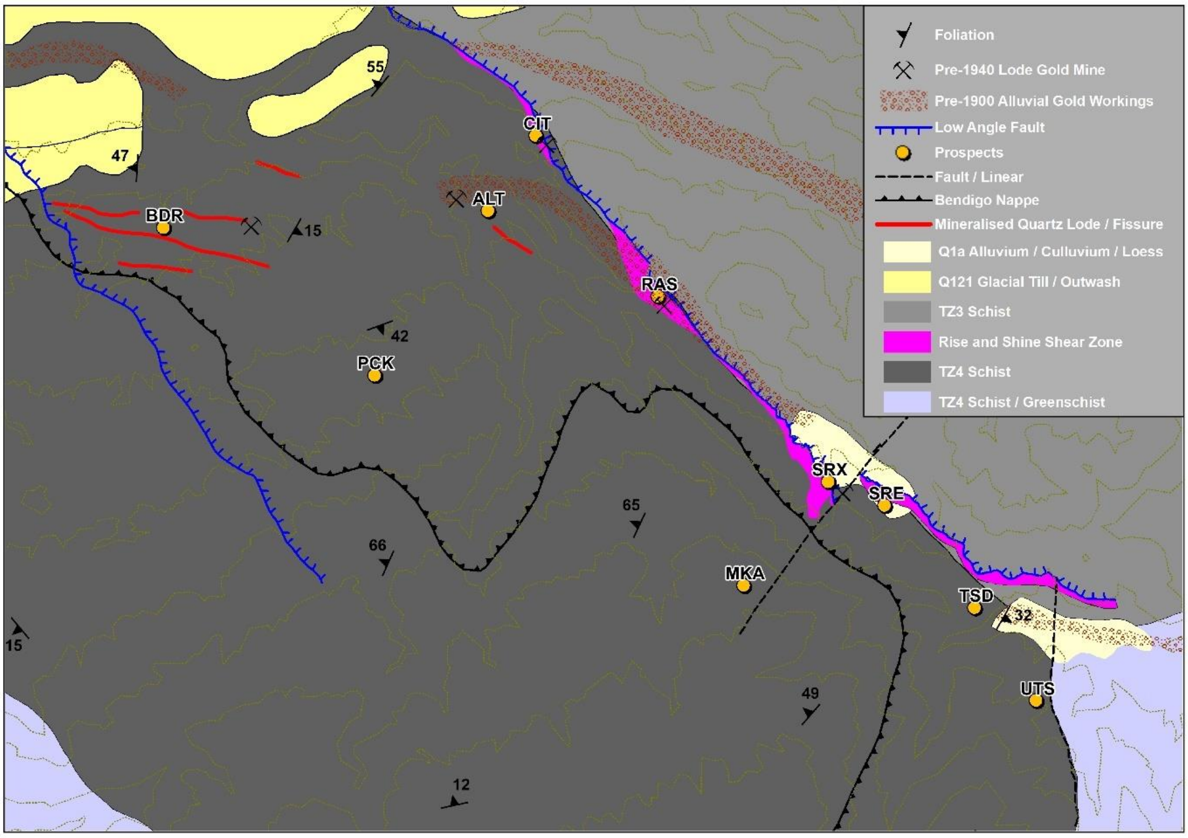


Figure ES 4: Main Prospects Along the RSSZ

RAS, SRX and SRE all outcrop in the base of the Rise & Shine Valley and dip north. The CIT deposit outcrops near the top of the southern face of the Shepherd's Creek Valley. The gold deposits identified to date in the RSSZ form 150m to 500m wide shoots plunging to the north. Mineralisation is generally hosted in breccias or veins.

ES 3.3 Rise and Shine

Mineralisation at RAS has been traced over a width of 350m and down plunge (25 degrees to the NNE) length of 1.7km. Mineralisation extends up to 90m below the TGF, however is typically 30-40m. Refer to Figure ES 5, Figure ES 6, and Figure ES 7.

Within the wider zone of mineralisation at RAS, a higher-grade core of approximately 150-200m width contains most of the gold. The high-grade core is a cataclasite (brecciated) network of anastomosing, post-metamorphic quartz (SBX), these occur with minor sulphide veins in a halo around 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.

The RAS deposit is primarily all fresh rock with subsurface oxidation variably extending from 5-20m depth.

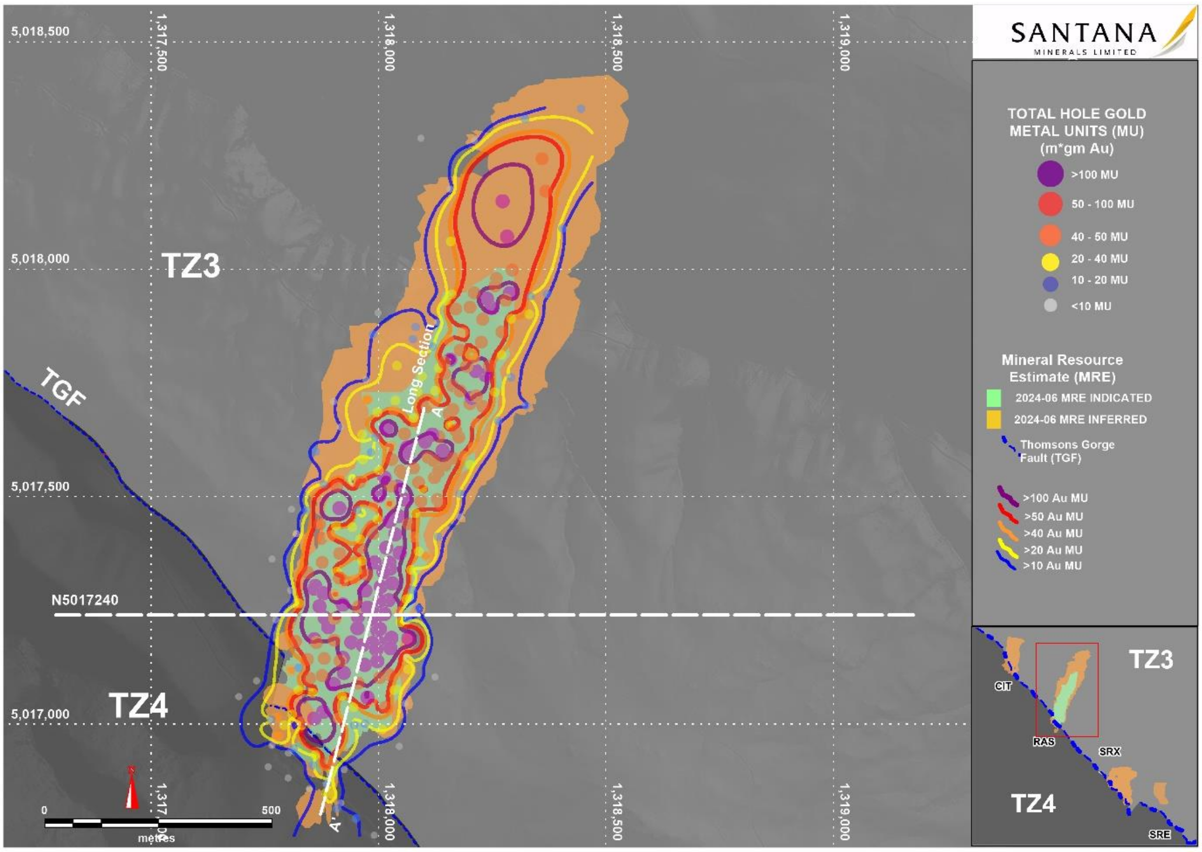


Figure ES 5: RAS deposit map

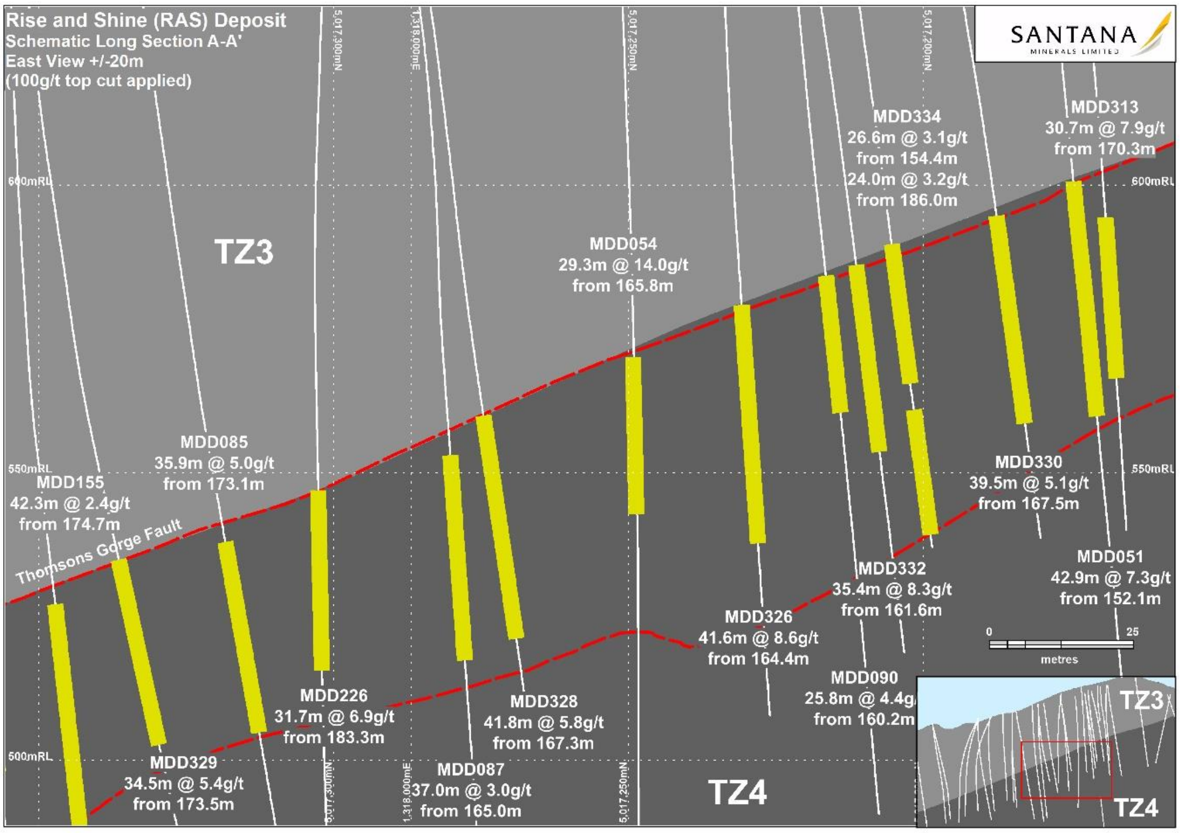


Figure ES 6: RAS deposit long section

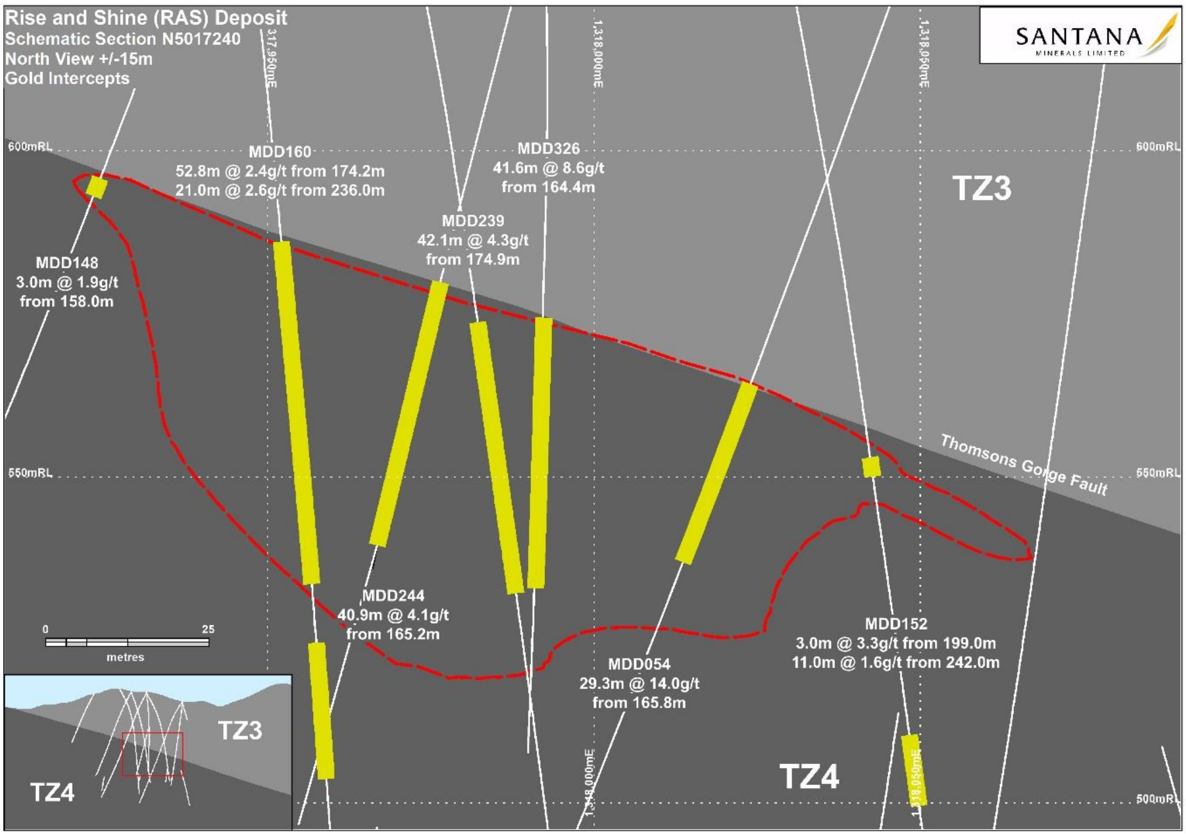


Figure ES 7: RAS deposit cross section

ES 3.4 SRX

The SRX and SRE deposits are located in the upper part of Rise and Shine Creek 1.5 to 2kms upstream of the much larger RAS deposit. Currently the known mineralisation at SRX covers an area of 470m x 470m (see Figure ES 8 and Figure ES 9). SRE is offset by 200m from SRX and covers an area of 350m x 450m. Total mineralised thicknesses are typically 5-15m. The total thickness is made up of smaller lenses containing economic grades surrounded by sub-grade material. The mineralised system at SRX is still open to the north and northwest. Micaceous-carbonate breccia (MCBX} forms the thickest and most extensive zone of strongly deformed rock within the wider RSSZ at SRX implying it is the principal strand of the RSSZ in this location. The dominance of MCBX over SBX at SRX implies there was less fluid flow and less intense alteration at SRX compared with RAS. The abundance of unaltered metamorphic mica in the matrix of MCBX, unlike in SBX, is consistent with this interpretation.

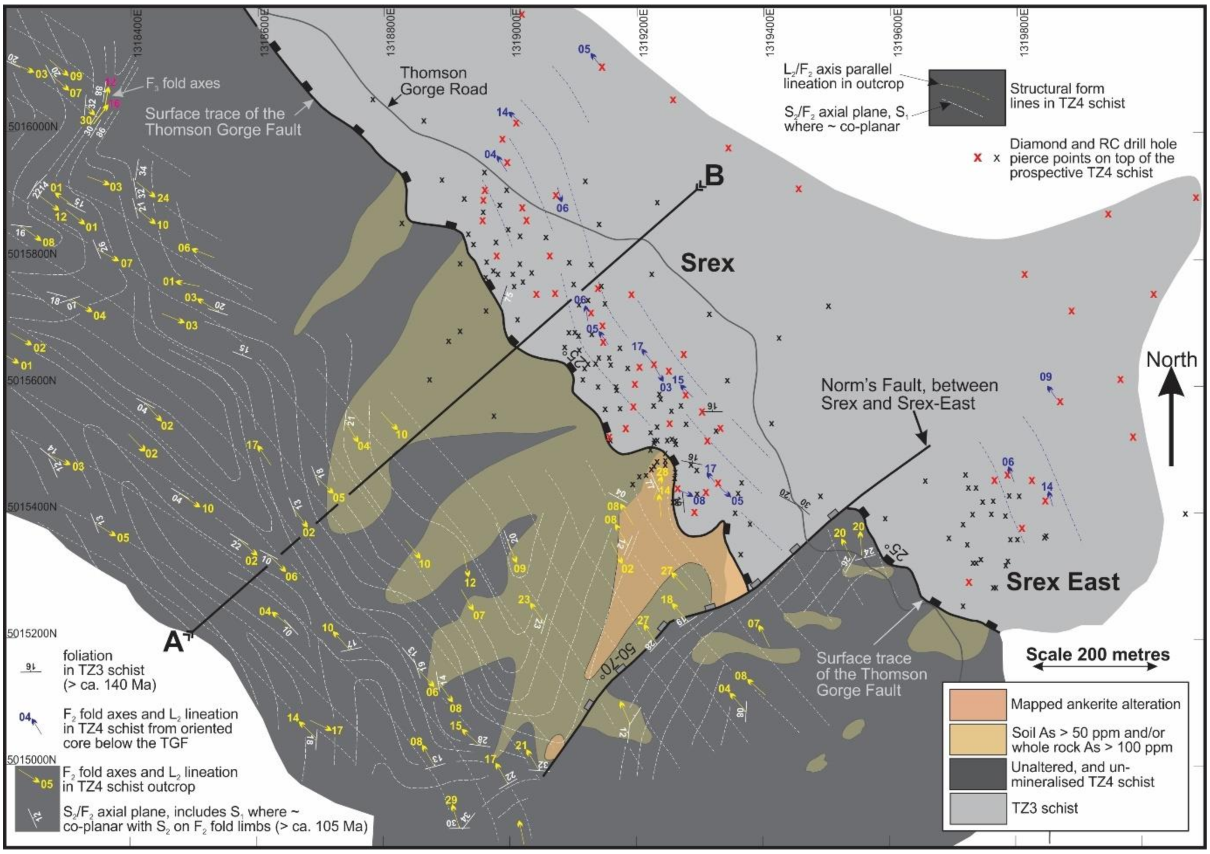


Figure ES 8: SRX deposit geological map

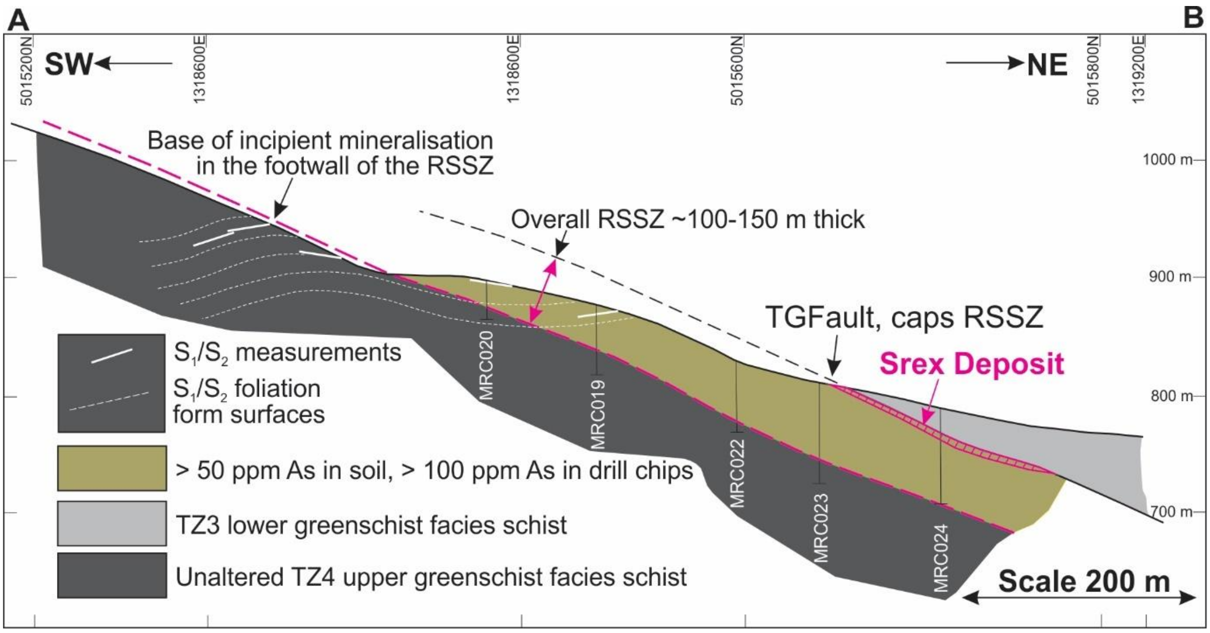


Figure ES 9: SRX deposit cross section

ES 4. Resources

The Total Mineral Resource Estimate (MRE) for the Bendigo-Ophir Gold Project using a 0.25 g/t cut-off for open pit and 1.5 g/t for underground is 40.3 Mt at 1.9 g/t for 2.46 Moz comprised of RAS open pit, RAS underground, CIT and SRX/SRE resources. All mineral resource estimates were prepared by an external consultant using data provided by Santana Minerals. The lower 0.25g/t cut-off grade than the 0.5g/t used in the RAS July 2024 MRE is due to the robust economics reported in this PFS, additional metallurgical testing and increase gold price. Other than reporting at a lower cut-off grade there are no other changes from the RAS July 2024 MRE.

A summary of the material information used to estimate the mineral resources for Rise and Shine (RAS) and Srex (SRX) and Srex East (SRE) is presented in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the 'JORC Code') 2012 edition (refer appended JORC Table 1). For details on the Come in Time (CIT) MRE, please refer to the announcement dated 28 September 2021.

ES 4.1 Data

The RAS MRE is based on assay sample results from 22 Reverse Circulation Drill (RC) holes (2,004.5m) and 256 Diamond Drill (DD) holes (71,640.4m). The SRX and SRE MRE is based on 154 RC holes (9,797.3 m) and 66 DD holes (9,445.3 m).

RC samples were collected in the sample box at the base of the cyclone and released into the riffle splitter at every metre. All diamond core samples were drilled using triple tube methods. Half-core sampling was completed at predominantly 1m intervals.

All samples were assayed by 50g fire assay at the SGS Laboratories in Westport, Waihi and Macraes. Routinely, a small number of samples were re-assayed by screen fire assay (SFA), photon analysis (PA) or accelerated cyanide leach (BLEG) of a 500g sub-sample of the original coarse rejects as QAQC checks on the original fire assays. The results of these re-assays were ranked higher than the original fire assay and were used in the MRE

QAQC procedures were completed as per industry standard practices, including blanks, certified standards, field duplicates, laboratory QAQC and umpire checks.

ES 4.2 Interpretation

At RAS an implicit wireframe model was generated based on a 0.2g/t cut off using surfaces interpreted from the structural SBX model to control the anisotropy. At the deposit scale no natural grade breaks have been identified to inform domain boundaries.

At SRX and SRE sectional interpretation was used to generate seven domain wireframes at a 0.25g/t cut off. The nominal interpretation grade was selected because histograms and cumulative probability plots of the un-domained SRX data showed no natural lower cutoff that could be used to define mineralisation.

Cross sections for CIT were interpreted on east-west sections at 40m intervals. An assay boundary of 0.1g/t Au and composite interval using a 0.1g/t Au cut-off grade were used to delineate the halo of mineralisation on each section. Points were snapped to drillholes, channels, trenches and underground adit sample points. Subsequent to the mineralised halo wireframing a higher grade shoot was domained as a discrete zone.

Four oxidation domains were interpreted across the entire project area. The oxidation domains are intended for use in estimating bulk density and mine planning. The domains include a soil boundary, notionally 1 m below topography, an oxide domain which varies from 5 to 20 m thick, a transitional domain which also varies from 5m to 20m thick and is underlaid by a fresh domain. The bulk of the mineralisation is located within the fresh domain.

ES 4.3 Estimation

High grade outlier analysis has been completed on 2m composites for each individual mineralization domain at RAS and SRX. CIT samples were composited to 1m intervals. The effects of the highest-grade composites on the mean grade and standard deviation of the gold dataset for each of the estimation domains have been investigated. An upper cut for each dataset was chosen coinciding with a pronounced inflection or increase in the variance of the data. RAS outliers were managed by using a distance restriction at a particular grade threshold to restrict the influence of high-grade values. At RAS gold values greater than the cut off were restricted to 25 m during the estimation process. At CIT, SRX and SRE, top cuts were used to cut the composited data.

Wireframed mineralisation domains were used as "hard boundaries" for estimation. Oxide and transitional mineralisation were estimated together with the fresh mineralisation.

Ordinary Kriging (OK) was selected as the method for estimating the gold grade. Samples at RAS and SRX were composited to 2m for grade estimation while CIT composites were 1m. At RAS a single pass was used to interpolate all blocks. Two interpolation passes were used at SRX and SRE as there are distinct densely and sparsely drilled zones there. CIT used 3 estimation passes to account for local changes in orientation.

At RAS and SRX block size of 12.5mE by 12.5mN by 2mRL was selected as the appropriate parent block size given the drill spacing with appropriate sub-celling to ensure adequate volume representation. CIT parent blocks were 10mE by 20mN by 5mRL.

Variography for the main domains indicate a moderate nugget of 50% with a maximum range of 125m, a semi-major range of 55m and a minor range of 35m for RAS and a high nugget of 80% with a short range of 30m for Srex. CIT had a lower nugget of 31% and a major range of 50m.

Elliptical search neighbourhoods within domains were applied. The search ellipsoids were oriented parallel to the variogram model axes and the ratios of the search ellipsoid axis lengths approximate the ratios in the variogram models. In the main domain at RAS the search ellipsoid major and semi-major axes were both 150m, reflecting the spatial distribution of the drilling. The search ellipsoid minor axis was 30 m, the variogram range in that direction. In the steep domain the semi-major axis of the search ellipsoid was reduced to 75 m to reflect the anisotropy in the variogram model. The search used a minimum of 4 and a maximum of 15 composites. At SRX and SRE pass 2 was similar to RAS whereas pass 1 used smaller search axes with a minimum of 10 and a maximum of 20 composites.

Bulk density was interpolated in all block models from density measurements into blocks of the fresh oxidation domain using inverse distance squared weighting. The fresh oxidation domain was used as a hard boundary. In blocks where bulk density was not interpolated, bulk density was assigned to the block model by oxidation domain.

ES 4.4 Classification

Resource categorisation reflects confidence in the estimation of gold grades and is based on input data quality, geological interpretations, distance to the nearest composite used to interpolate a block, the average distance to all composites used to interpolate a block and the kriging slope of regression. The resource estimates have been classified as Indicated and Inferred Mineral Resources.

ES 4.5 Reasonable Prospects

Reported resources have all been assessed for Reasonable Prospects of Eventual Economic Extraction (RPEEE) based on site specific criteria. RPEEE assessments were carried out sequentially for open pit mining followed by underground mining. Note that detailed Reserve assessments may move the mining method crossover point based on the economic outcomes.

The portion of the resource considered amenable to open cut mining is reported at lower cutoff grade of 0.25g/t Au within a RPEEE pit shell. This lower cut-off grade than the 0.5g/t used in the July 2024 MRE is due to the robust economics reported in this PFS, additional metallurgical testing and increase gold price. Other than reporting at a lower cut-off grade there are no other changes from the July 2024 MRE. Outside of this pit shell underground resources are reported at a lower cutoff of 1.5g/t Au at RAS only. Refer to Table ES 1.

To assess reasonable prospects of economic extraction for mineralisation that potentially could be extracted by open pit mining methods, the resources have been constrained within a conceptual Whittle pit shell for each of the deposits.

The conceptual open pit shells for RAS and SRX are based on the following input parameters:

- Gold price NZ$3,250/oz,
- Exchange Rate 0.60 NZD/USD,
- Metallurgical Recovery of 90%,
- Pit slope angles of 45° for all material; and
- Industry correlated mining, ore processing and administration operating costs.

For CIT the following inputs were used:

- Gold price NZ$2,500/oz
- Metallurgical Recovery of 70%
- Processing, NZD$15.00/t

Note that the economic assessment for CIT was at a higher level than RAS or SRX and was based on earlier extraction investigations.

Table ES 1: Collated Mineral Resource Estimates

Deposit	Mining method	Category	Cutoff (Au g/t)	tonnes (Mt)	Au (g/t)	koz
RAS	open pit	Indicated	0.25	19.6	2.3	1,452
		Inferred		9.9	2.0	634
		Total		29.5	2.2	2,086
	underground	Indicated	1.5	0	1.9	0
		Inferred		2.1	2.2	145
		Total		2.1	2.2	145
	RAS Total	Indicated		19.6	2.3	1,452
		Inferred		12	2.0	779
		Total		31.6	2.2	2,231
SRX	open pit	Indicated	0.25	2.6	0.7	59
		Inferred		2.4	0.9	73
		Total		5.0	0.8	132
SRE	open pit	Indicated	0.25	0.4	0.7	10
		Inferred		0.1	0.9	3
		Total		0.5	0.8	13
CIT	open pit	Inferred	0.25	3.2	0.8	81
Total	**combined**	**Indicated**		**22.6**	**2.1**	**1,521**
		Inferred		**17.7**	**1.6**	**936**
		Total		**40.3**	**1.9**	**2,457**

Note: totals may not add due to rounding

ES 5. Geotechnical

A mining focussed site investigation was conducted in early 2024, comprising of four geotechnical purposed boreholes targeting the proposed RAS pit walls.

The typical rock mass conditions at RAS are summarised as follows:

- A thin layer (1 to 2m) of alluvium and colluvium is present across the site. Alluvium located within the natural streams. This surficial layer is not anticipated to impact the open pit, nor underground mine designs.
- Weathered schist (Textural zone 3 - TZ3) is located within the top 20m across the site. Weathering at SRX is deeper than at RAS.
- The highly foliated (Textural zone 3 - TZ3) chlorite schist rock located above the TGF is generally weak to moderately strong rock (~20 MPa) and is dominated by foliation shears as well as foliation parallel shear zones, which become more abundant within 30m of the TGF.
- TGF comprises of fractured and fragmented schist within a matrix of clay gouge. The TGF thickness varies between 0.5 to 3m within the RAS deposit.
- Textural zone 4 (TZ4) situated below the TGF comprises mineralised biotite schist. The rock mass is generally medium to strong rock (50 to 90MPa).

ES 5.1 Open Pit

The recommended pit slope design sectors and slope configurations for RAS and SRX are presented respectively in Table ES 2 and Table ES 3. The level of data available for SRX is scoping level only.

Table ES 2: RAS PFS Level Recommended Pit Slope Design

<table>
<tr><th>Wall</th><th>Aspect (1) (°)</th><th>Unit</th><th>IRA (2) (°)</th><th>BFA (3) (°)</th><th>Berm Width (m)</th><th>Bench Height (m)</th><th>Controlled By</th></tr>
<tr><td>Southwest</td><td>350 to 065</td><td>All</td><td>30</td><td>50</td><td>9</td><td>15</td><td>Foliation/foliation shears dipping towards northeast</td></tr>
<tr><td rowspan="2">West</td><td rowspan="2">065 to 160</td><td>TZ3</td><td>35</td><td>50</td><td>11</td><td>15</td><td>Planar sliding along the obliquely dipping TGF (4)</td></tr>
<tr><td>TZ4</td><td>45</td><td>60</td><td>6.5</td><td>15</td><td rowspan="2">Planar failure along faults and shears identified behind the pit wall</td></tr>
<tr><td>Northeast</td><td>160 to 235</td><td>All</td><td>45</td><td>60</td><td>6.5</td><td>15</td></tr>
<tr><td rowspan="2">East</td><td rowspan="2">235 to 350</td><td>TZ3</td><td>40</td><td>60</td><td>9</td><td>15</td><td>Planar sliding along the obliquely dipping TGF (4)</td></tr>
<tr><td>TZ4</td><td>47</td><td>70</td><td>7.5</td><td>15</td><td>Planar failure along faults and shears identified behind the pit wall</td></tr>
</table>

(1) Slope aspect measured as the direction the wall dips towards.
(2) Inter-ramp angle.
(3) Batter face angle.
(4) Opportunity to steepen IRA based on future 3D stability analyses and/or mapping data of the TGF.

Table ES 3: SRX Scoping Level Recommended Pit Slope Design

Wall	Aspect (°)	IRA (°)	BFA (°)	Berm Width (m)	Bench Height (m)	Controlled by
Southwest	350 to 065	30	50	9	15	Foliation/foliation shears dipping towards northeast
West, Northeast, and East	065 to 350	45	60	6.5	15	Probable Planar failure along faults and shears identified behind the pit wall

ES 5.2 Underground

The empirical assessments conducted assess the potential dimensions of each stope face. Based on the assessment, the end walls of the stopes are not expected to control the stope width. Where the stability of the backs is able to be controlled by cable bolting, the requested 15m stope width is expected to be feasible.

Table ES 4: Assessed Hydraulic Radii for Stope Walls

Stope Wall	Hydraulic Radii (Recommended Range, typical conditions)
	Matthews / Potvin Assessment
Hanging and footwall (TZ4)	4.6 – 5.5
Stope backs (when in TZ3)	1.4 (1)
Stope backs (when in TGF)	0.6 (1)
Stope backs (when in TZ4)	3.2 – 3.8
Stope end wall (TZ4)	5.0 – 5.9

(1) Unsupported.

Table ES 5: Recommended Stope Dimensions with Cable Bolting in the Backs

Depth (mbgl)	Stope Height (1) (m)	Maximum Stope Length (2) (m)	Maximum Stope Width (3) (m)	Notes and Limiting Wall Mechanism
250	20	25	15 (4)	Assumes heavy support of the backs is practical and economic Potential for footwall planar slide
	25	20		
400	20	20		
	25	15		

(1) Vertical height.
(2) Along strike.
(3) Across Strike
(4) Stope width is expected to be controlled by the ability to support the backs.

Development ground support in the TZ3 will be shotcrete with rock-bolts and in the TZ4 it will be mesh with rockbolts.

ES 6. Hydrogeology

The BOGP sits astride a ridge dividing the Shepherds Creek and Rise and Shine Creek on the eastern flanks of the Dunstan Mountains, Central Otago. Shepherds Creek is a minor tributary of the Lindis River, while Rise and Shine Creek is a small tributary of Bendigo Creek, both diminish and cease flowing as the creeks pass off the hard-rock schist onto the more permeable gravel deposits forming alluvial aquifers across the valley floors. Regionally, the Clutha River catchment makes up 67% (⅔) of the Otago region, and the mean flow of the Clutha passing the Bendigo area is approximately 271 cubic metres per second (23,414 megalitres per day). There is a strong pluviographic (rainfall) gradient from extremely high runoffs in the Southern Alps headwaters to specific discharge only a sixth of the Alps in the Bendigo catchments. The gradient is the result of a drop-off in Alps spill-over precipitation, plus the effect of rain shadowing by inland Otago ranges.

Shepherds and Bendigo Creeks become intermittent and drain into the ground before meeting their respective downstream main stems, due to soakage into the creek bed gravel alluvium.

However, groundwater under the mining complex is found within fractured rock comprising Otago Schist (including textural zone 3, 4, and the Rise and Shine Shear Zone), with very low permeabilities compared to the alluvial aquifers found on the valley floors. The excavation of the RAS surface mining pit, and the underground, will each induce the seepage of small volumes of water from the surrounding rock, requiring it to be pumped away from the working faces. The relatively low fracture permeability of all parts of the schist rockmass constrains the rates of seepage and makes the discharge a relatively small part of the mining complex water balance. Current model estimates of the ultimate RAS pit inflows lie between 14 and 28 litres per second. Corresponding RAS underground workings' groundwater seepage rates approach 24 litres per second at peak.

The planned mining and processing complex requires a water supply of up to 97 litres per second for the process plant, mainly for plant make-up water and dust suppression. This would be obtained from two bores in the Bendigo Aquifer, located 7 kilometres to the west of the mining complex. Mine water would also be used in a supplemental role in processing and dust suppression, to offset the need to discharge or use clean water. When in full operating cycle, a significant amount of water will be recycled from the tailing storage facility. Otherwise, water mainly leaves the site through evaporation. During and after infrequent rain events, mine influenced water will be recycled or treated to meet water quality limits before any discharge.

A production bore test in July 2024 has demonstrated the required capacity in addition to inducing off-site effects, that are in all respects acceptable (i.e., water levels, impacts on surface water and availability of allocation).

ES 7. Metallurgical testwork

Various historical testwork programmes (Stages 1 through 4) have been completed for the BOGP, primarily by Kappes Cassidy and Associates (KCA) and ALS Laboratory, during 2018 to 2022.

The PFS testwork programme had the following objectives:

- Composite master sample selection to represent the expected Life of Mine (LOM) ore blend for the RAS deposit.
- Variability sample selection for RAS to provide spatial variability data.
- Determination of comminution characteristics for the master composite and variability samples.
- Gravity recovery and intensive leaching of gravity concentrate on all samples.
- Flotation sighter testing on master composite.
- Cyanide leach grind optimisation, reagent optimisation and CIL testing on master composite.
- Cyanidation response based on optimised flowsheet for the variability samples.

As the testwork program proceeded the following steps were included:

- Cyanide destruction testwork on master composite.
- Arsenic removal on master composite.
- Diagnostic leaching of optimised CIL of master composite.
- Thickening testwork.

The SRX deposit was subsequently added to the testwork program, with initial testwork based on the RAS optimised program.

The following conclusions can be drawn from the current and previous metallurgical and comminution testwork programmes:

- The RAS ore is a moderately abrasive and competent ore with moderately high comminution energy requirements.
  - An Abrasion index of 0.3077, indicating the ore exhibits moderate abrasivity.
  - Crusher Work index reported a range from 2.96 to 15.24 kWh/t and an average work index value of 5.15 kWh/tonne.
  - A Bond Ball Work index of 19.0 kWh/t (range 17.4 to 21.0 kWh/t) categorising the ore as moderately hard.
  - SAG Circuit Specific Energy values of 9.18 kWh/tonne (range 8.52 to 10.75 kWh/tonne), indicating a medium to high power input is required for grinding.
- The RAS ore is 'free-milling' with a high gravity - recoverable free gold component and high gold recovery from the gravity tails by cyanidation leach with moderate reagent consumptions.
  - Gravity gold recoveries ranged from 45.4% to 76.3%.
  - Testwork at this scale will over account for gravity recovery, and in design a value of 32% has been adopted.
  - Overall gold recoveries ranging from 86.0% to 97.8%.
  - Final leach residue grades ranging from 0.07g/t to 0.46g/t.
- A grind optimisation study was undertaken to evaluate the effect of grind size on project economics. The results of the grind optimisation study indicated:
  - The increase in gold revenue (recovery) with fineness of grind is offset by the increase in operating costs to achieve the finer grind sizes. The net revenue (gold revenue – operating costs) varies by less than 1% between 75 µm and 106 µm.
  - The optimum grind for the RAS ore is 106 µm.
- While leaching is effectively completed in 8 hours preg-robbing behaviour occurs with increased residence time and so are not suited to a hybrid leach-CIL. Cyanide in the process water should be minimised as much as possible.
- The RAS ores have demonstrated amenability to cyanide destruction using the air/SO2 system.
- Arsenic is leached in the leach process, and an arsenic removal step is required on the tailings. Initial testwork has indicated that the ores are amenable to arsenic removal using ferric chloride precipitation of a ferric arsenate.
- Anticipated lime and cyanide consumptions are typical of operations conducted with good quality water treating primary ores with a small refractory component. Cyanide consumptions ranged from 0.33 to 0.56kg/t throughout the tests using tap water. Further testing using site water is recommended given the high quality.

For the Srex deposit the following conclusions can be drawn:

- The Srex ore is similar in comminution characteristics to RAS, an abrasive, moderately competent ore with above average comminution energy requirements:
  - Abrasion Index – 0.3152;
  - Crusher Work index reported a range from 1.80 to 10.70 kWh/t and an average work index value of 5.25 kWh/t;
  - A Bond Ball Work index of 16.9 kWh/t; and
  - SAG Circuit Specific Energy values of 8.83 kWh/tonne (range 7.72 to 9.16 kWh/tonne), indicating a medium to high power input is required for grinding.

- The Srex ore is less free milling than RAS, showing a refractory component that is not leachable through fine (20 µm) grinding. The Srex deposit shows greater sensitivity to grind size than RAS.
- Flotation testwork indicated a higher recovery to flotation concentrate than seen with RAS.
- Gravity recovery for the Srex deposit is lower than that seen for the RAS material, ranging from 22-23%.
- As with RAS ores, Srex shows some preg robbing behaviour and is not suited to a hybrid leach-CIL. Overall gold recoveries ranging from 66.8% to 68.7%.

The metallurgical treatment route selected has been based on the results of this test programme with consideration of results from previous programmes and can be summarised as follows:

- Single stage crushing;
- Single stage SAG milling;
- Gravity concentration;
- Carbon-in-leach (CIL);
- AARL stripping circuit;
- Cyanide destruction; and
- Arsenic removal.

The Srex ore will be treated through the processing route optimised for RAS ore.

ES 8. Mining

The RAS deposit is planned to be mined by staged open pit and underground methods. The two operations are independent and can be developed in parallel as the topography allows a low elevation underground portal position, not requiring the open pit to reach a similar location. The SRX deposit is planned to be mined by staged but shallow open pit only.

ES 8.1 Introduction

Following open pit optimisation analysis on RAS, a final maximum pit size was selected. This was tested against mining deeper portions of the open pit from underground. This test concluded the final maximum open pit was preferable to underground mining as gold price increased above the base case.

ES 8.2 Pit Optimisations

ES 8.2.1 Mineral Resource Models

Open pit optimisations were completed using Whittle Optimisation Software (Whittle®) on the Indicated Category of the mineral resource estimates (MRE). A third-party consultant group, Geomodelling Ltd performed the estimates with the RAS outcome for RAS and SRX.
Table ES 6 lists the files utilised in the optimisation process.

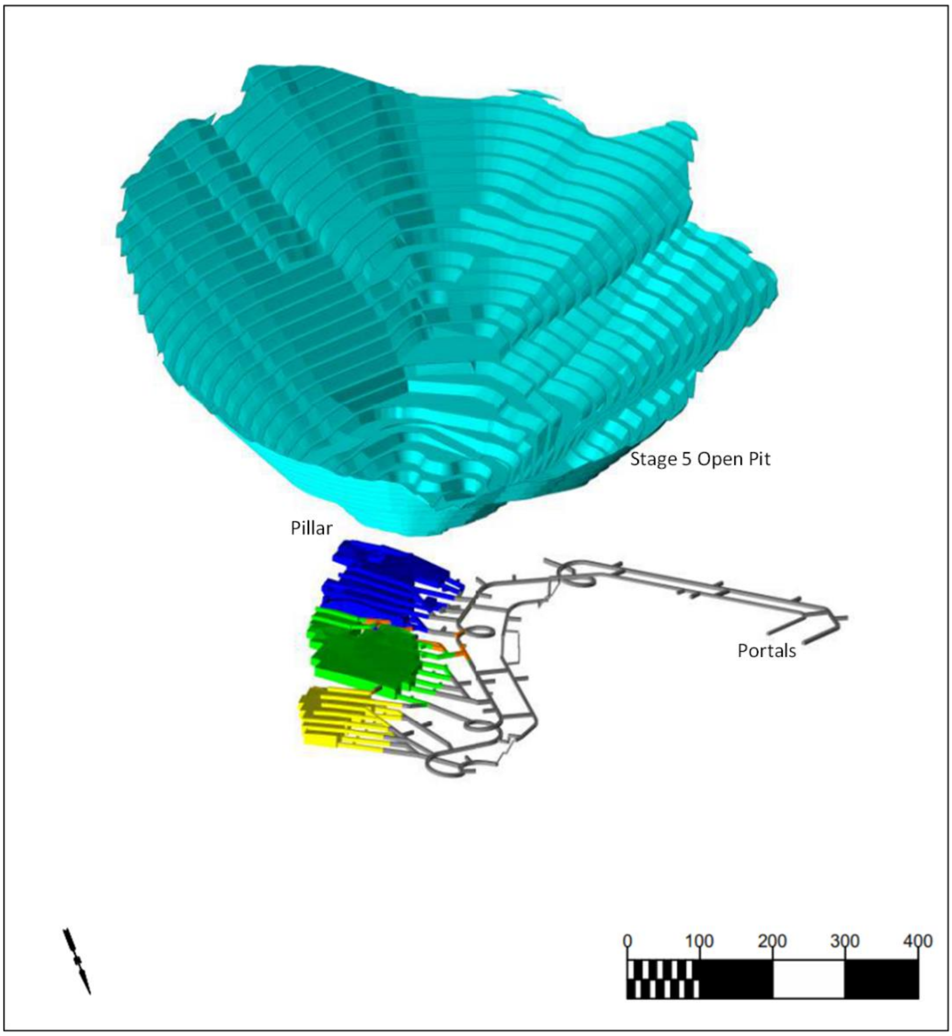


Figure ES 10: An orthogonal view looking south of the RAS final pit and Underground workings

Table ES 6: Block model and topography files

Item	File name	Description
Topography	TOPO_2021 Lidar_major_contours_filtered.00t	2021 Lidar topography over the Bendigo-Ophir Gold Project
	20240625 RAS PFS Model ext.bmf	Extended RAS model bmf file to cover enlarged model extents.
	20240830 SHRE model.bmf	SRX Block model

ES 8.2.2 Open Pit Optimisation Input Parameters

Whittle optimisation was completed utilising input parameters shown in Table ES 7. The parameters were derived from first principles, and the optimisation was completed on an operating cost basis. Capital costs, interest, income tax, depreciation, amortisation and closure costs were not included.

Table ES 7: Optimisation input parameters

Parameter	Units	SRX Value	RAS Value
Gold price	USD /oz	2,100	1,650
Exchange rate	NZD:USD	0.64	0.64
Overall metallurgical recovery	%	68	93
Gold price	NZD/oz	3,281	2,578
Royalties (varies, average applied)	%	3.5	3.5
Transport/Refinery cost	NZD/oz	8	8
Discount rate	%	7.5	7.5
Processing cost	NZD/t processed	17.73	17.73
Tailings storage facility cost	NZD/t processed	1.42	1.42
Crusher feed	NZD/t processed	0.87	0.87
Ore overhaul cost	NZD/t processed	1.5	-0.5
General and Admin cost	NZD/t processed	3.21	3.21

The resource block models applied are recoverable models with block sizes set as panels that have an allowance for mining recovery and dilution. A mining recovery of 100% and 0% dilution was assumed in the pit optimisations for the two deposits due to the block model being re-blocked to a 2.5m high block from 2.0m high.

Table ES 8: Optimisation mining cost and adjustment factors

Parameter	Units	SRX Value	RAS Value
Base mining cost	NZD /t	2.21	3.09
Bench mining cost adjustment factor	NZD /t	6.0978-0.00047* Bench mRL	8.086-0.0062*Bench mRL
Dilution*	%	0	0
Mining recovery*	%	100	100
Processing throughput	Mtpa	1.5	1.5
Mining capacity	Mtpa	3	20
Estimated Mine Life	Years	1	10

* Dilution and mining recovery or losses accounted for by the block model being re-blocked to a 2.5m

ES 8.2.3 RAS Open Pit Optimisation Results

Optimisation results analysis was completed on the RAS deposit. Pit shell 32 which corresponds to the revenue factor 0.99 shell, at NZD2,552 /oz (USD 1,633 /oz), has the highest cashflow for the best case and was selected as the ultimate pit shell.

Figure ES 11 shows pit by pit tonnes, pre-capex cashflow and pre-capex discounted cashflow values for the best- and worst-case scenarios. Hatched bars show selected shells for pit stage designs.

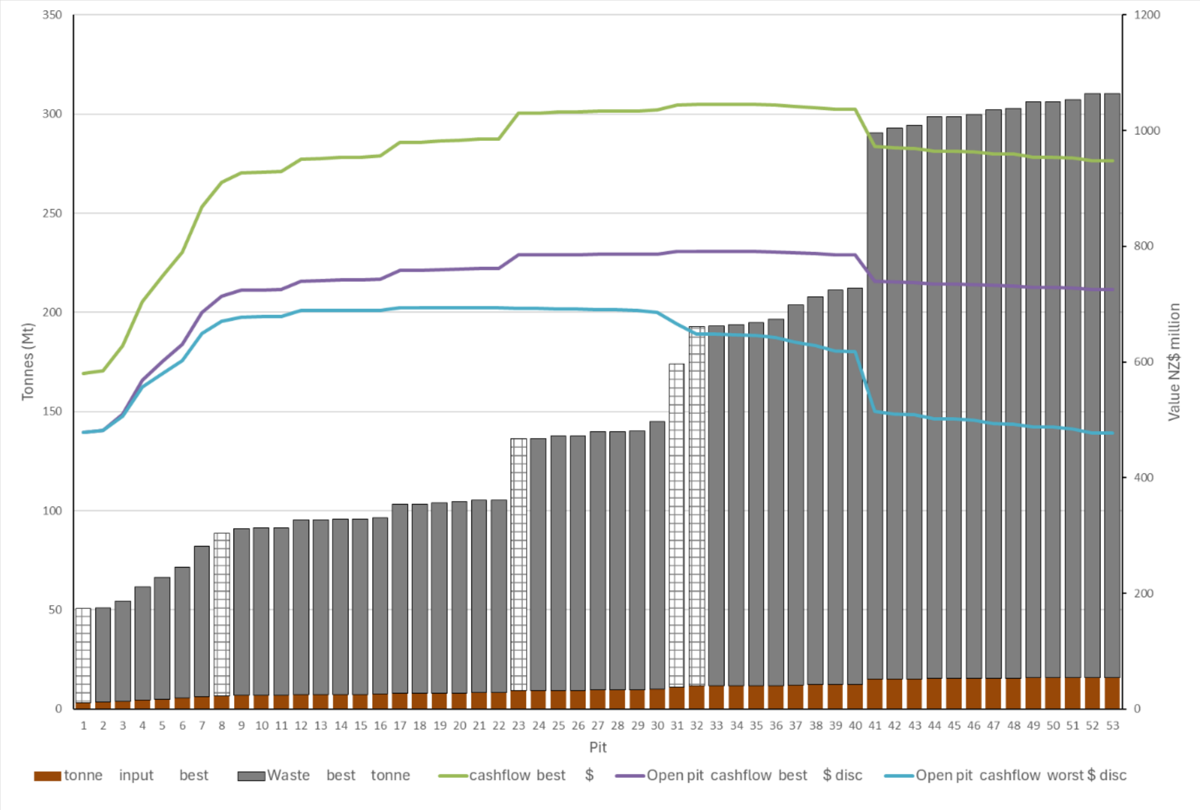


Figure ES 11: RAS pit by pit optimisation results

ES 8.2.4 SRX Open Pit Optimisation Results

Optimisation results analysis was completed on the SRX deposit. Pit shell 33 which corresponds to the revenue factor 1 shell, at NZD3,281 /oz (USD 2,100 /oz), has the highest cashflow for the best case and was selected as the ultimate pit shell.

Figure ES 12 shows pit by pit tonnes, pre-capex cashflow and pre-capex discounted cashflow values for the best- and worst-case scenarios. Hatched bars show selected shells for pit stage designs.

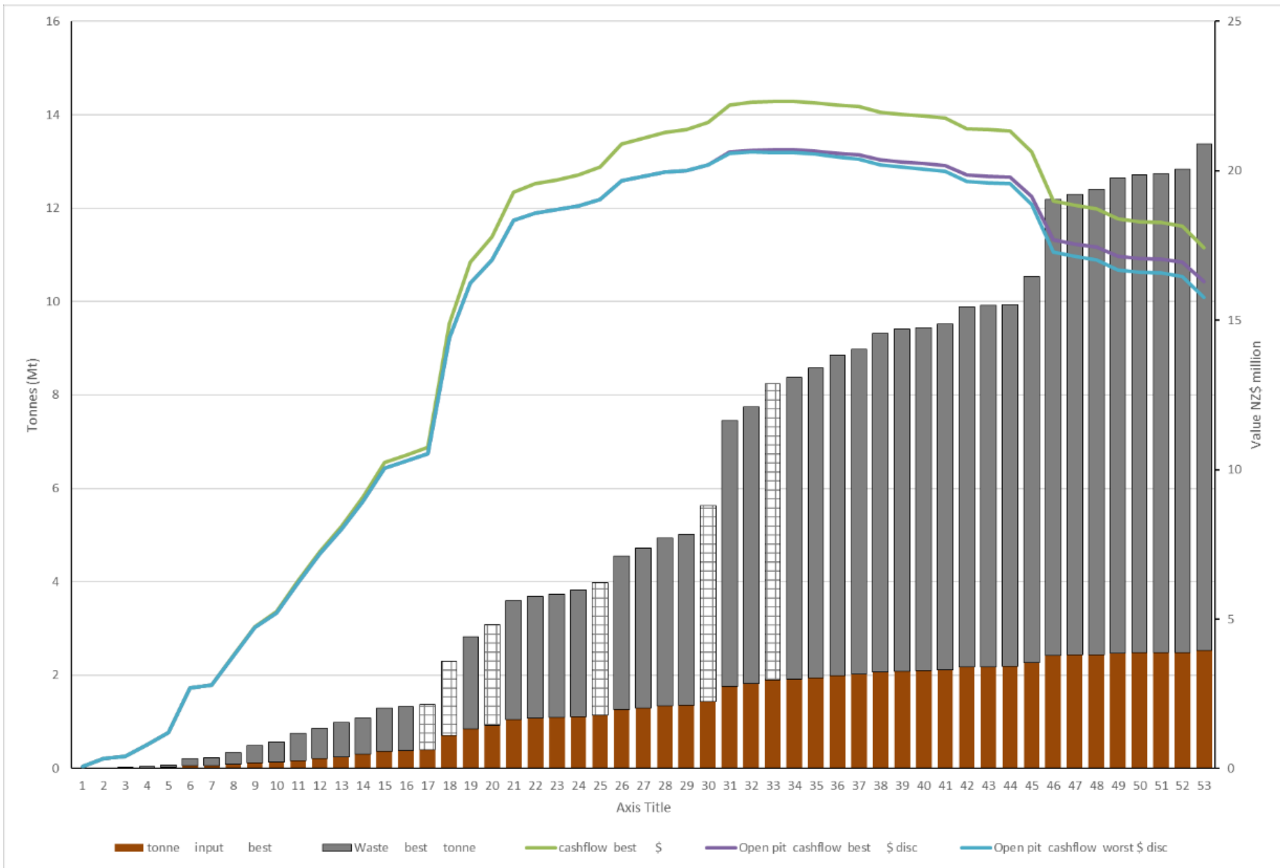


Figure ES 12: SRX pit by pit optimisation results

ES 8.2.5 Sensitivity Analysis

Sensitivity analysis has been completed on the RAS deposit by varying:

- Gold price;
- Metallurgical recovery;
- Pit slope angles;
- Processing cost;
- Mining cost
- Mining dilution; and
- Mining recovery.

Table ES 9 shows results of sensitivity analysis completed by varying the sensitivity parameters in Whittle. The results show that the project is highly sensitive to gold price and metallurgical recovery. An increase in gold price or recovery of 10% results in an increase in pre-capex discounted cashflow of 20%. A decrease in gold prices or recovery of 10% results in a decrease in pre-capex discounted cashflow of 18%. Mining recovery also has a significant impact with a decrease of 10% mining recovery resulting in a 15% decrease in pre-capex discounted cashflow. Pit slope angle is another significant factor with a 10% change in slope resulting in a 12% increase or 10% decrease in pre-capex discounted cashflow. Results of sensitivity analysis ranking are shown in Table ES 9 for the top three parameters.

Table ES 9: Sensitivity analysis top three parameters ranking

Parameter	Rank	Change in Parameter %	Parameter Value	Ore tonnes (Mt)	Au grade (g/t)	Contained ounces (koz)	Pre-Capex discounted cashflow (NZD m)
Price/Met. Rec	1	20%	USD1,980	12.6	2.40	969	1,106
		10%	USD1,815	11.7	2.47	930	947
		Base	USD1,650	11.6	2.49	925	791
		-10%	USD1,485	9.6	2.59	794	650
		-20%	USD1,320	8.1	2.67	692	511
Mining Recovery	2	Base	100%	11.6	2.49	925	791
		-5%	95%	9.1	2.57	755	732
		-10%	90%	8.7	2.57	716	676
		-15%	85%	8.2	2.57	676	622
		-20%	80%	7.6	2.59	629	562
Slope (°)	3	10%	Multiple Sectors	12.0	2.47	948	887
		5%	Multiple Sectors	11.5	2.49	924	834
		Base	Multiple Sectors	11.6	2.49	925	791
		-5%	Multiple Sectors	9.7	2.57	796	752
		-10%	Multiple Sectors	9.7	2.57	796	712

The project's pre-capex discounted cashflow is less sensitive to mining cost, processing cost and dilution compared to the parameters shown in Table ES 8. A change of 10% in dilution, mining costs and processing costs results in a 1-7% change in pre-capex discounted cashflow.

Sensitivity analysis spider graph is shown in Figure ES 13.

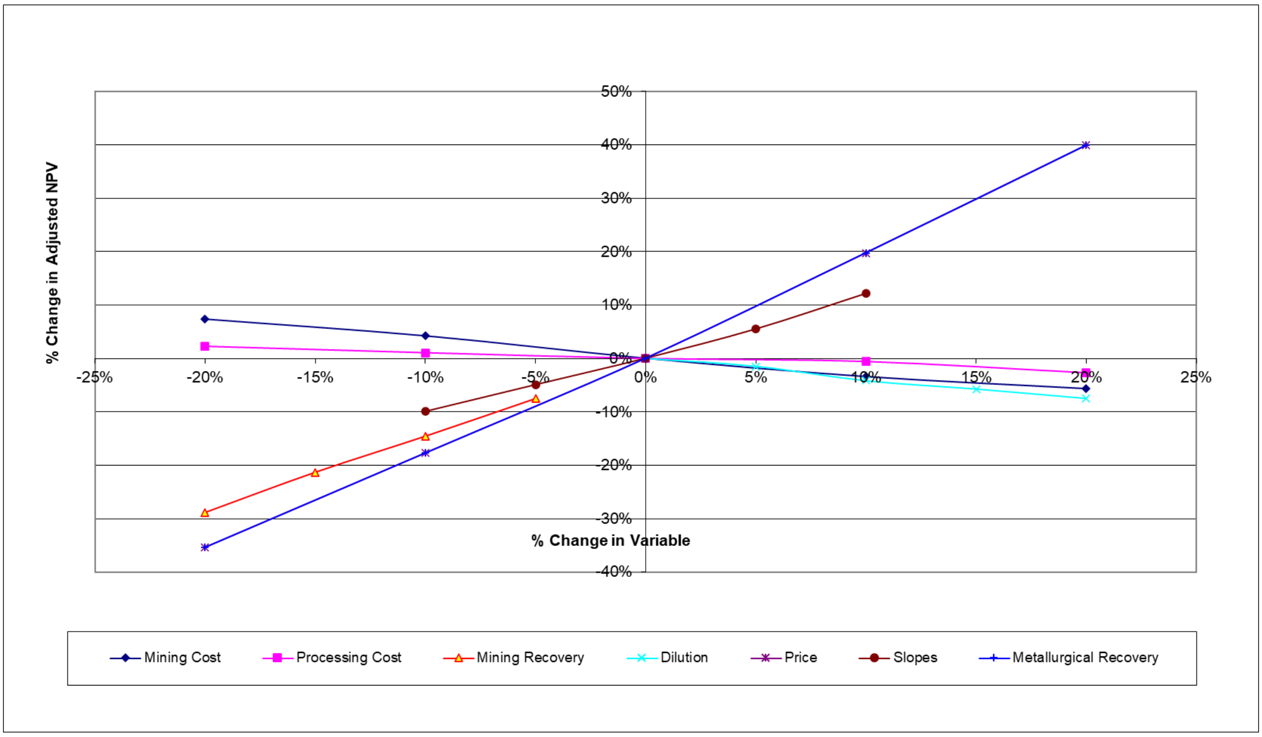


Figure ES 13: RAS sensitivity analysis spider graph

ES 8.3 Pit Design

ES 8.3.1 RAS pit

The ultimate pit design was based on shell 32, the highest pre-capex discounted cashflow shell.

The final pit design based on shell 32 is approximately 200 m deep at the highwall, approximately 1,000 m long in a roughly north-south direction and approximately 900 m wide (Refer to Figure ES 14 and Figure ES 15). The transition from shell to actual workable pit design using recommended geotechnical and ramp design parameters resulted in a 14% increase in overall volume, a 3% increase in ore mined and a 3% lower overall grade resulting in slightly more ounces than the pit shell. A summary of the comparison is shown in Table ES 10.

Table ES 10: Final pit design versus shell 32

Pit Stage	Total tonnes	Ore tonnes	Ore grade	Cont. Ounces	Waste tonnes
	(Mt)	(Mt)	(g/t)	(koz)	(Mt)
Shell 32	187.2	11.6	2.49	926	175.6
Final Design	214.0	11.9	2.42	928	202.1
Variance	**14%**	**3%**	**-3%**	**0.23%**	**15%**

Table ES 11 summarises individual pit stage inventories. Pit stage 1 has the highest grade at 4.06 Au g/t. The grade in subsequent stages decreases with the final stage being the lowest at 1.89 Au g/t.

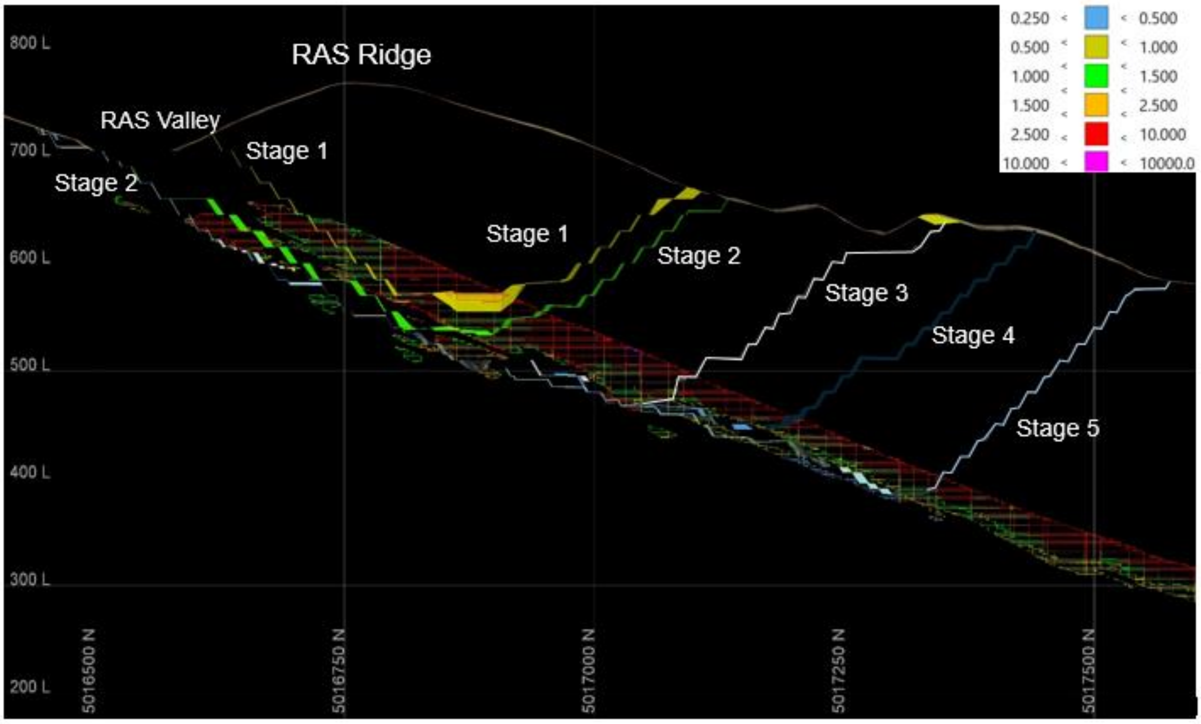


Figure ES 14: RAS staged pit designed long section

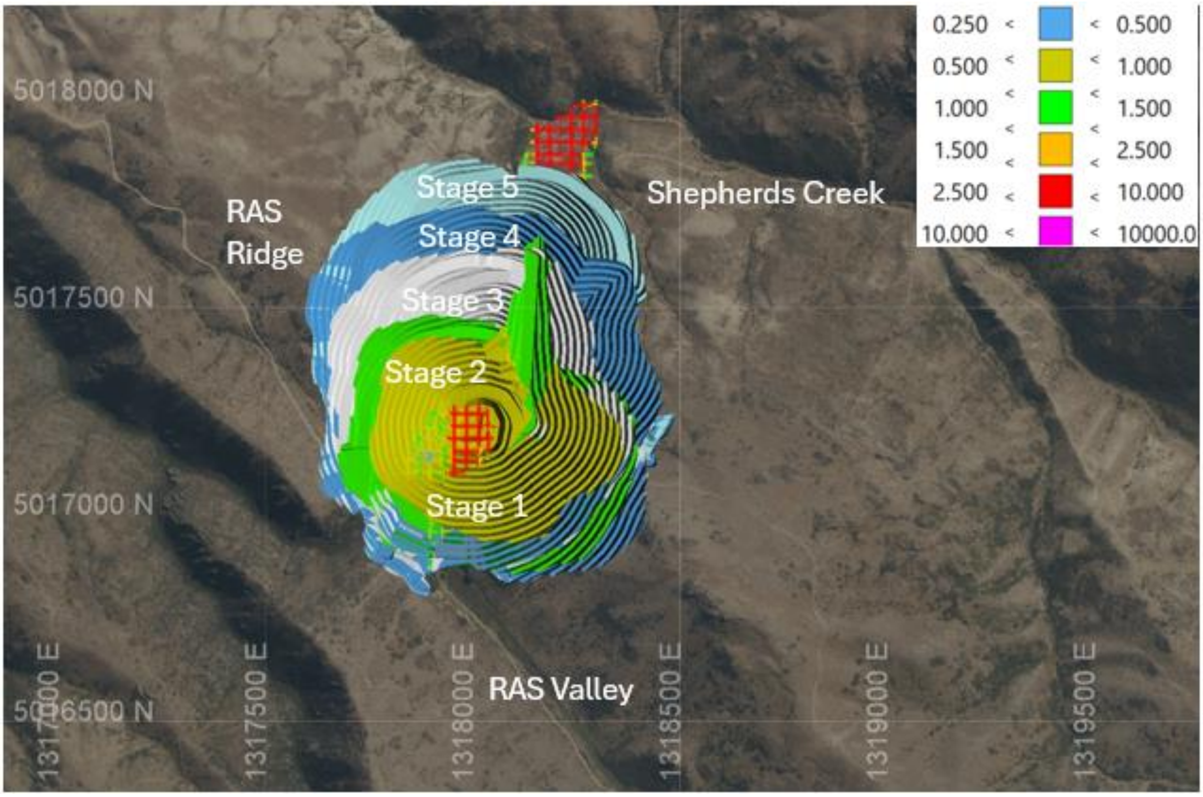


Figure ES 15: RAS pit stages in plan view

Table ES 11: Pit Stage Inventories

Pit Stage	Total tonnes	Total volume	Ore tonnes	Ore grade	Contained Ounces	Ore volume	Ore bulk density	Waste tonnes	Waste volume	Waste bulk density
	(Mt)	(Mbcm)	(Mt)	(g/t)	(koz)	(Mbcm)	t/bcm	(Mt)	(Mbcm)	t/bcm
Stage 1	46.8	17.8	1.4	4.06	186	0.5	2.73	45.4	17.3	2.63
Stage 2	29.2	10.9	3.0	2.35	228	1.1	2.73	26.2	9.8	2.66
Stage 3	40.4	15.2	3.0	2.32	226	1.1	2.71	37.4	14.1	2.65
Stage 4	57.6	21.6	2.4	2.13	162	0.9	2.72	55.2	20.7	2.67
Stage 5	40.0	15.2	2.1	1.89	125	0.8	2.72	37.9	14.4	2.63
Total	**214.0**	**80.7**	**11.9**	**2.42**	**928**	**4.4**	**2.72**	**202.1**	**76.3**	**2.65**

Note: These do not include Inferred mined as a consequence of mining the Indicated blocks. The inferred resource mined as a consequence of mining Indicated categories totalled 1.1Mt at 1.54 g/t containing approx. 55koz).

ES 8.3.2 SRX Pit

The final SRX pit design based on shell 33 is approximately 88 m deep, 650 m long and 210 m long. The shell to workable pit design transition resulted in a 11%reduction of both ore and waste and 9% higher grade resulting in a 5% reduction in ounces compared to the Whittle shell. A summary of the comparison is shown in Table ES 12.

Table ES 12: SRX final pit design versus shell 33

Pit Stage	Total tonnes	Ore tonnes	Ore grade	Cont. Ounces	Waste tonnes
	(Mt)	(Mt)	(g/t)	(koz)	(Mt)
Shell 33	8.2	1.9	0.69	42	6.3
Final Design	7.3	1.4	0.68	31	5.9
Variance	**-11%**	**-11%**	**9%**	**-5%**	**-11%**

ES 8.4 Open Pit Production Scheduling

ES 8.4.1 RAS

An open pit mine production schedule for the RAS deposit has been produced using the pit stages, final pit and waste rock dump designs.

Table ES 13 below shows the resulting pit inventory which forms the basis of the production schedule.

Table ES 13: RAS - Pit Design Inventory

	Unit	Quantity
Total Rock	**(kt)**	**213,972**
Total Waste	**(kt)**	**200,996**
TZ3 Tonnes	(kt)	179,886
TZ4 Tonnes	(kt)	18,089
Soil	(kt)	3,021
Ore	**(kt)**	**12,976**
Gold Grade	(Au g/t)	2.36
Contained Gold	(oz Au)	982,832

All the fresh Indicated and Inferred material above a cutoff grade of 0.3 g/t Au is considered as ore and planned to be sent either directly to the run-of-mine (ROM) pad or placed on a temporary ROM rehandling stockpile. The ROM material has been further subdivided into the following grade bins:

- High Grade (HG) >= 2.6 g/t Au
- Run-of-Mine 2 (ROM2) 1.6 – 2.6 g/t Au
- Run-of-Mine 1 (ROM1) 0.6 – 1.6 g/t Au
- Low Grade (LG) 0.3 – 0.6 g/t Au (primarily processed at the end of the project and makes up only 2.1% of the metal feed)

The open pit processing break-even grade (BECOG) was determined utilising the 2.1Mtpa processing and G and A costs plus a rehandle and crusher feed cost to determine what material mined would at the pit exit contribute to a positive cashflow if treated rather than sent to the waste dump. (Refer Table ES13A)

For RAS a conservative gold price of NZ$2,578 (US$1,650 and 0.64 NZD:USD exchange rate) per ounce was used for project evaluation and optimisation works. For SRX a less conservative gold price of NZ$3,281 (US$2,100 and 0.64 NZD:USD exchange rate) was used due to its small size and hence pre-stripping commitment.

A break-even cut-off grade of 0.3 g/t was ultimately applied for RAS and 0.35 g/t for SRX (rounding up to the nearest 0.05g/t from the base calculations.

Table ES 13A: Process BECOG calculation

Metric	Units	RAS	SRX	Comments
Mining cost	$/t	$1.11	$1.11	Stockpile rehandle and crusher feed
Processing Recovery (Au)	%	93.0%	68.0%	
Processing cost	$/t ore	$17.75	$17.75	Plant at 2.1mtpa
G&A	$/t ore	$3.08	$3.08	Plant at 2.1mtpa ($6.46M/year)
Total operating Cost	$/t	$21.94	$21.94	
Gold Price USD	$/Oz	US$1,650	US$2,100	
NZD:USD		0.64	0.64	
Gold price USD	$/oz	$2,578	$3,281	
Royalty		4.5%	4.5%	Crown (2%), Landowners (varies - assumed 1%), Venders (1.5%)
Selling cost	$/Oz	$8.0	$8.0	
Effective Gold price		$2,454	$3,126	
Value 1 gram recovered	$/g	$78.64	$100.23	
Value 1 gram in feed	$/g	$73.14	$68.16	
COG to process plant	**g/t**	**0.30**	**0.32**	

The scheduling parameters forming the basis for the mine scheduling are as follows.

- Maximum mill feed of 1.5 Mtpa.
- Mill feed commences with an allowance for commissioning ramp-up:
  - Month 1: 87.5 kt;
  - Month 2: 119 kt; then
  - Month 3 (full production) at 125 kt per month for the first two years of production.
- Mill feed includes both the Probable Reserves mined and the Inferred Resource mined as a consequence of mining the reserves.
- A scale back of mill head grade to a maximum of 3.5 g/t Au has been applied in the processing plant schedule. It is achieved by blending down the grade in early years.
- 100% of mill feed is re-handled at the ROM pad for crusher feed purposes, ensuring the head grade not exceeding 3.5 g/t Au.
- Mining generally progresses across multiple benches within the cutbacks, rather than strictly following a bench-by-bench sequence allowing for quicker ore exposure.
- All the transitional and soil materials are planned to be sent to their own dedicated stockpile areas.
- Mineralisation below 0.3 g/t Au is sent to the waste rock dump.

ES 8.4.2 Material Movements

Total material movements are shown in Figure ES 16 and Figure ES 17. The annual mining schedules are shown in Table ES 14. The first 100 kt of ore will be exposed after mining roughly 35 Mt of waste around month 15 of mining, which will be stockpiled. The plant is expected to be brought into its commissioning phase around this period after which a sustainable supply of ore is available to start feeding the mill by month 17 of mining.

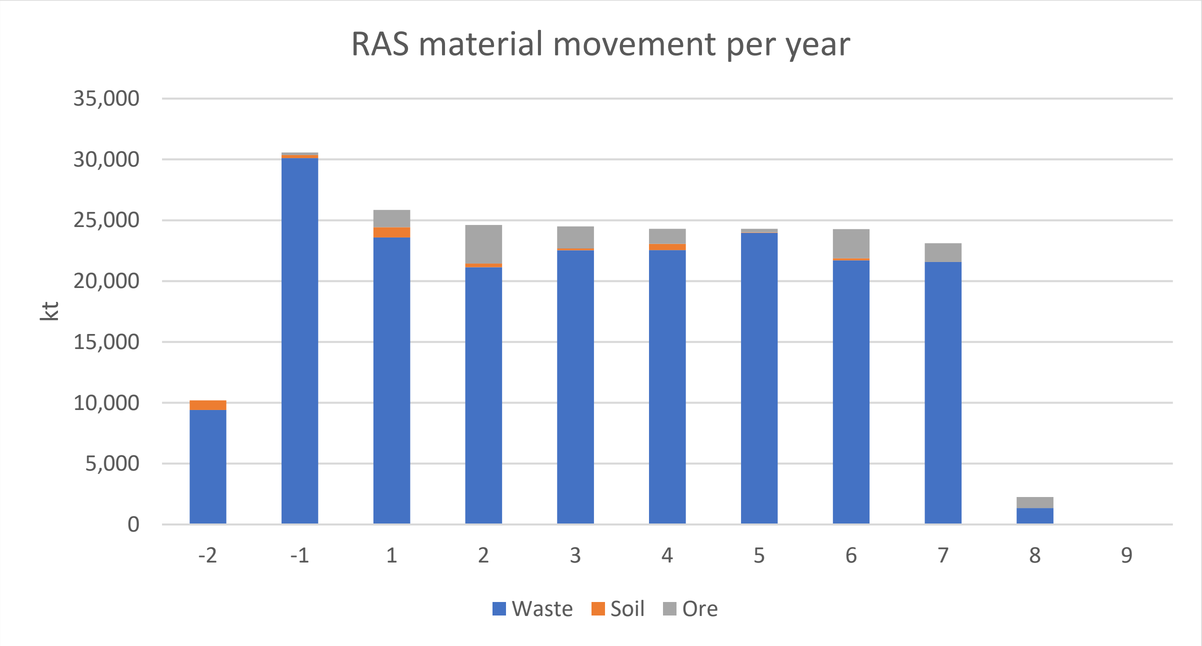


Figure ES 16: Total material mined per year at RAS open pit

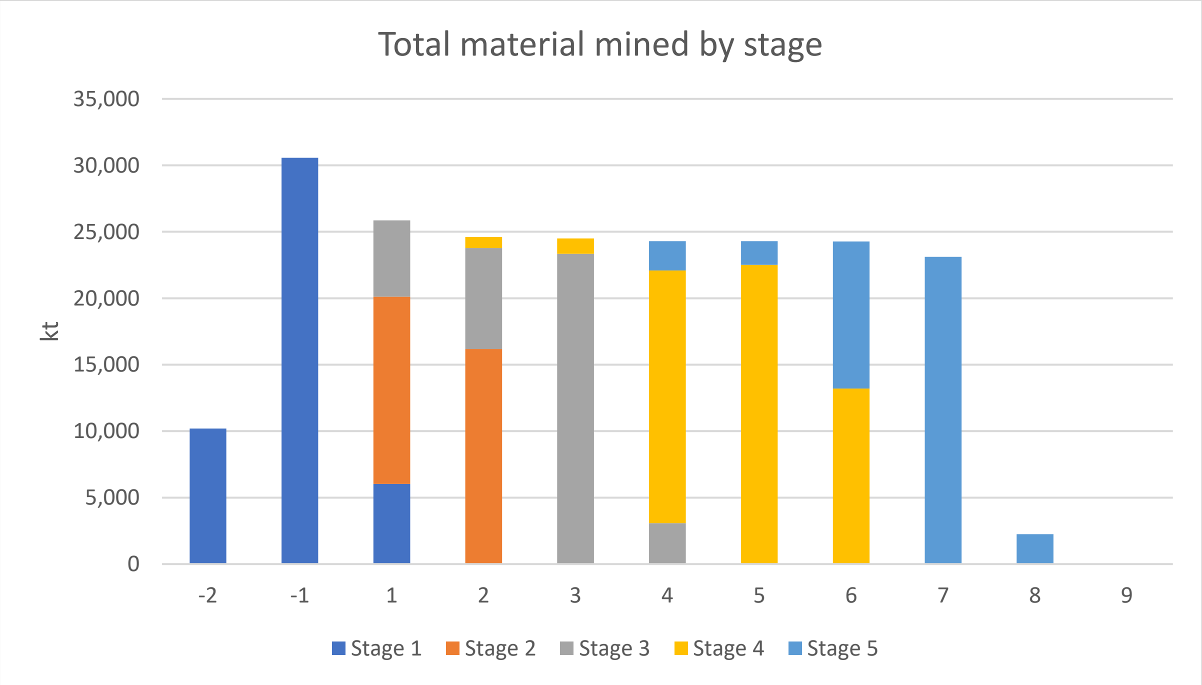


Figure ES 17: Total material mined at RAS open pit per stage per year

Table ES 14: RAS OP Mining Schedule

			Year									
			-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
Total Rock Mined	**(kt)**	213,972	10,200	30,572	25,859	24,612	24,493	24,300	24,300	24,267	23,112	2,258
Total Waste	**(kt)**	**200,996**	10,200	30,363	24,437	21,438	22,680	23,060	24,007	21,880	21,575	1,355
TZ3 Tonnes	**(kt)**	179,886	9,417	29,967	20,456	16,804	19,717	20,939	22,636	19,530	19,401	1,019
TZ4 Tonnes	**(kt)**	18,089	0	149	3,139	4,344	2,815	1,613	1,339	2,184	2,171	336
Soil	**(kt)**	3,021	783	247	842	291	148	508	33	166	3	0
Ore	**(kt)**	**12,976**	0	209	1,422	3,173	1,813	1,240	293	2,387	1,537	902
Gold Grade	**(Au g/t)**	**2.36**	0.00	0.00	3.50	3.37	3.08	2.59	1.11	2.89	1.63	1.67
Contained Gold	**(koz Au)**	**983**	0	9	185	246	125	98	11	166	80	63
Stripping Ratio	**(t:t)**	15.5	0.0	145.5	17.2	6.8	12.5	18.6	81.9	9.2	14.0	1.5

Note: ROM mined includes Inferred 1.1Mt at 1.54 g/t, therefore 94.4% of the RAS open pit mill feed is from Probable Reserves.

ES 8.4.3 SRX

An open pit mine production schedule for the nearby SRX deposit has been produced using the pit stages, final pit and waste rock dump designs. Table ES 15 below shows the resulting pit inventory which forms the basis of the production schedule.

Table ES 15: SRX - Pit Design Inventory

	Unit	Quantity
Total Rock Mined	**(kt)**	**7,344**
Total Waste	**(kt)**	**5,916**
TZ3 Tonnes	(kt)	4,376
TZ4 Tonnes	(kt)	710
Soil	(kt)	626
Transition Mineralisation (> 0.3 g/t Au)	(kt)	204
Ore	**(kt)**	**1,428**
Gold Grade	(Au g/t)	0.68
Contained Gold	(oz Au)	30,674
Stripping Ratio	(waste t: ore t)	4.1

Scheduled material is at a 0.3 g/t cut-off grade. Reserves where quoted are at a 0.35 g/t cut-off grade.

Table ES 16: SRX (monthly) Mining Schedule

			Months																						
	Unit	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Total Rock Mined	**(kt)**	**7,344**	**209**	**209**	**209**	**309**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**350**	**108**
Waste Rock																									
Total TZ3	(kt)	4,377	33	79	84	185	209	253	202	214	211	277	199	247	289	162	227	292	211	234	226	205	212	104	21
Total TZ4	(kt)	606	47	42	21	19	28	4	22	14	44	13	17	37	10	29	42	7	28	38	16	30	20	63	14
Total Economical WST (>0.30g/t Au)	(kt)	103	0	0	4	2	8	3	9	0	8	1	1	3	4	5	10	2	0	4	2	6	7	18	4
ROM Mined (>0.30g/t Au Fresh IND and INF)	**(kt)**	**1,428**	**0**	**15**	**26**	**34**	**47**	**9**	**75**	**32**	**84**	**27**	**73**	**60**	**38**	**95**	**70**	**47**	**82**	**74**	**94**	**99**	**111**	**164**	**69**
Gold Grade	Au g/t	0.68	0.00	0.62	0.76	0.69	0.81	1.03	0.81	0.77	0.67	0.61	0.58	0.69	0.68	0.61	0.63	0.70	0.64	0.57	0.64	0.61	0.68	0.72	0.74
Contained Gold	koz	31	0	0	1	1	1	0	2	1	2	1	1	1	1	2	1	1	2	1	2	2	2	4	2
Stockpile Materials																									
Total Soil Mined	(kt)	625	80	51	49	61	36	59	30	64	3	30	44	3	10	54	0	2	28	0	11	10	0	0	0
Total Transition Mineralisation (>0.30g/t Au)	(kt)	204	49	21	25	8	21	21	11	25	0	3	15	0	0	5	0	0	1	0	0	0	0	0	0

ES 8.5 Mining Fleet and Requirements

ES 8.5.1 Mining Fleet

A mining cost estimate based on an owner mining model has been produced. The cost model is based on a conventional open pit Drill and Blast, Load and Haul mining method.

ES 8.5.2 Drilling and Blasting

Open pit drilling operations are divided into two distinct zones. These are ore and waste bulk waste zones.

Ore zone drilling has been modelled on a 7.5 m bench. This zone comprises of ore and inter-burden waste. There is no separate grade control drilling. Ore blast holes are also used for grade control sample. Drilling in the ore zone is performed by Atlas Copco L8 RC drills.

Bulk waste drilling has been modelled on 15 m benches and will be carried out by Sandvik D45KS equivalent drill.

Table ES 17 shows drill and blast parameters for ore and waste zones.

Table ES 17: Open pit drill and blast parameters

Parameter	Units	Ore zone	Waste zone
Drill		Atlas Copco L8 RC	Rotary (Sandvik D40KS)
Hole diameter	mm	102	200
Sampling frequency	t/sample	128	Nil
Bench height	m	7.5	15.0
Burden x Spacing	m	4.7 x 4.0	7 x 8
Powder factor	kg/bcm	0.32	0.52

ES 8.5.3 Loading

The cost model assumes an equivalent of a 260-t Hitachi EX2600 hydraulic backhoe excavator for digging in ore and waste (17m^3 bucket capacity). A 120-t Hitachi EX1200 excavator is included to selectively mine those parts of the orebody that need a balance between productivity, dilution and recovery.

ES 8.5.4 Hauling

A fleet of 150t class rear-dump trucks, Cat 785 equivalent, have been used in the modelling for all mine haulage activities. These trucks match with the 17m^3 hydraulic excavators for approximately five to six passes per truck.

Crusher Feed

A Cat 988 wheel loader will be used to re-handle ore from ROM stockpiles into the crusher bin.

Ancillary Equipment

A fleet of support equipment has been modelled. This fleet supports primary production equipment in the pit, at the waste rock stacks, haul roads, ROM pad and stockpiles. The fleet consists of:

- Cat 777 water trucks;
- Cat 834 wheel dozer;
- Cat 16 motor graders; and

- Cat D10 dozers.

Table ES 18 shows the maximum mining fleet composition during the peak mining periods.

Table ES 18: Mining fleet composition

Description	Number	Activity
260-t Backhoe Excavator	2	Ore and waste loading
120-t Backhoe Excavator	1	Rehandling ore stockpiles and backup for primary excavators
Cat 785 150t Rigid Body Dump Truck	22	Ex-pit and stockpile Ore and waste hauling
Cat D10 Tracked Dozer	3	Mining, waste rock tip management, rehabilitation and other site dozing requirements
Cat 16H Motor Grader	2	Haul road and work area maintenance
Cat 834 Wheel Dozer	1	Haul road and work area maintenance
Cat 777 Water Cart	2	Haul road and work area dust suppression
988 Front End Loaders	1	Feeding ore crusher bin
Atlas Copco L8 drills	1	Ore (7.5m)
D40KS Rotary Drill	1	Waste drilling on 15m bench height

ES 8.6 Engineered Landform (ELF)

Overburden waste rock will be stored in the Shepherds Creek engineered landform (ELF).

Several potential waste storage areas have been considered to store the planned waste rock from the RAS deposit. Areas have been identified that are suitable for use for this PFS study, excluding those impacted by social and foundational terrain issues, as well as the potential sterilization of nearby satellite deposits. The selected area is located just downstream of the location of the tailings storage facility (TSF). Effectively, the waste rock is being planned to act as a buttress of this TSF to enhance stability downstream of the dam while providing sufficient storage capacity to store all the planned waste from the pit.

Table ES 19 shows the required and designed capacities in bank cubic metres (bcm) and loose cubic metres (lcm) for the waste dump. A total swell factor of 25% after compaction (based on the majority of fresh rock) has been assumed.

Table ES 19: Waste Rock Dump Design Capacity

Material	Description	Requirements ('000 bcm)	Swell Factor (lcm:bcm)	Requirements ('000 lcm)
TZ3	block model code DTZ = 3 Fresh and Transitional (excluding Soil), < 0.3 Au g/t	67,621	1.25	84,526
TZ4	block model code DTZ = 4 Fresh and Transitional (excluding Soil), < 0.3 Au g/t	5,555	1.25	6,944
Mineralized Waste	block model code RSCAT = 0, >= 0.3 Au g/t	1,088	1.25	1,361
Total Waste Rock	**Total Material to be stored on waste rock dump**	**74,264**	**1.25**	**92,830**

A waste rock dump with a total design capacity of 103.6 million loose cubic metres (LCM) of which 3.2 million LCM from the TSF dam has been produced. The design quantity has a contingency of roughly 12% which would account for possible changes in the swell factor or the compaction ratio following further analysis on the actual site-specific parameters.

Waste landform sequencing has been undertaken based on the assumption that background levels of arsenic are higher in the ore host rock(TZ4) and for precautionary reasons it is planned to be encapsulated and capped with low/non-arsenic waste rock (TZ3) to mimic its natural occurrence. A base layer and encapsulating layer of inert material is typically required, with a core of non-inert material. For this reason, the dump has a base layer of 3 metres which would only consist of TZ3 material. The TSF dam embankment will also only consist of TZ3 material. Finally, a 20 m thick capping layer is added consisting only of the TZ3 material. The core will allow for all material types to be stored. Figure ES 18 shows a cross-section of the dump.

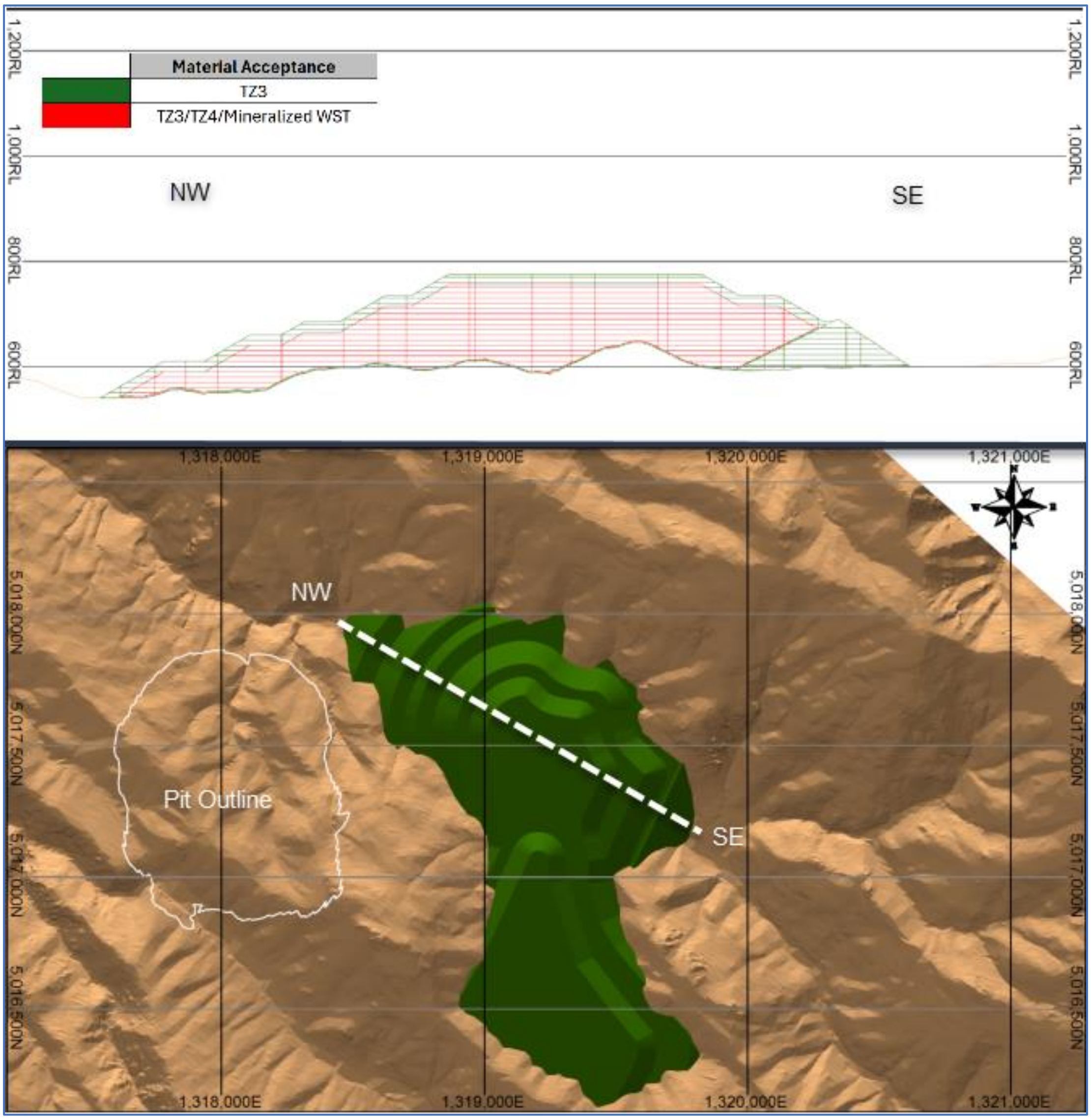


Figure ES 18: Northwest-Southeast Cross-section of the Waste Rock Dump

ES 8.7 Rise and Shine Underground

The RAS underground targets the continuation of the orebody down plunge and beneath the open pit. Only the Indicated resource category has been evaluated by the initial underground mine plan. There remains extensive lower class Inferred mineralisation down-dip that if converted after infill drilling would allow the underground to:

a. Extend its life; and
b. Increase the annual production rate.

Furthermore, the ore system remains open at depth and deeper extensions are considered likely to eventuate with deeper drilling.

ES 8.7.1 Mineable package targeting

A conservative gold price of NZ$2,578 (US$1,650 and 0.64 NZD:USD exchange rate) per ounce was used for project evaluation and optimisation works. Note: The breakeven cutoff grade (BECOG) was determined at the start of the underground mining process, it assumed a 1Mtpa operation that is mining stand-alone.

A break-even cut-off grade of 1.70 g/t was ultimately applied for underground mine scheduling.

Table ES 20: Break Even Cut off Grade Calculation

Metric	Units		Comments
Mining cost	$/t	$80.00	SMI estimate
Re-load and haul to plant	$/t	$2.00	
Processing Recovery (Au)	%	93.0%	
Processing cost	$/t ore	$26.60	MIQ at 1Mtpa
G&A	$/t ore	$4.81	MIQ at 1Mtpa
Total operating Cost	$/t	$113.41	
Gold Price USD	$/Oz	1650.0	
NZD:USD		0.64	
Gold price USD	$/oz	$2,578	
Royalty		4.5%	2% crown, 1.5% vendors, 1% landowners
Refining and Selling cost	$/Oz	$8.00	Includes freight
Value 1 gram in recovered		$78.90	
Value 1 gram in ground		$73.38	
COG to process plant	**gpt**	**1.55**	
COG insitu	**gpt**	**1.72**	an insitu grade target allowing 10% dilution

At first the Mineable Shape optimiser tool (MSO) was used to create automated mineable shapes, that were then used as a guide to prepare manual designs for the stoping outlines as it was found the MSO tool could not replicate a realistic mining selection package.

The total recoverable Probable Mining Reserve is 2,288 kt at 3.03 g/t containing 223 koz. As a consequence of extracting this, a further Inferred resource of 124 kt at 2.2 g/t containing 9 koz is mined. This results in a total underground mine inventory of 2,413 kt.

In total, 96.1% of the underground plant feed is from Probable Mining Reserves generated form the Indicated Resource blocks with modifying factors applied.

The underground is designed as a mechanised operation with ramp haulage. Extraction is via open stoping with cemented paste backfill. The relatively constrained Indicated resources at this point in time constrains the eventual overall annual production rate. If this material can be converted to indicated category, the underground mining rate could be increased.

Twin portals service a twin ramp development, which leads to a simple haulage (intake) and egress/return network system.

The underground workings stand off from the selected final Stage 5 Pit by 30m.

ES 8.7.2 Mining Layout, Panels, Design

The mine design layouts are shown in Figure ES 19 and Figure ES 20.

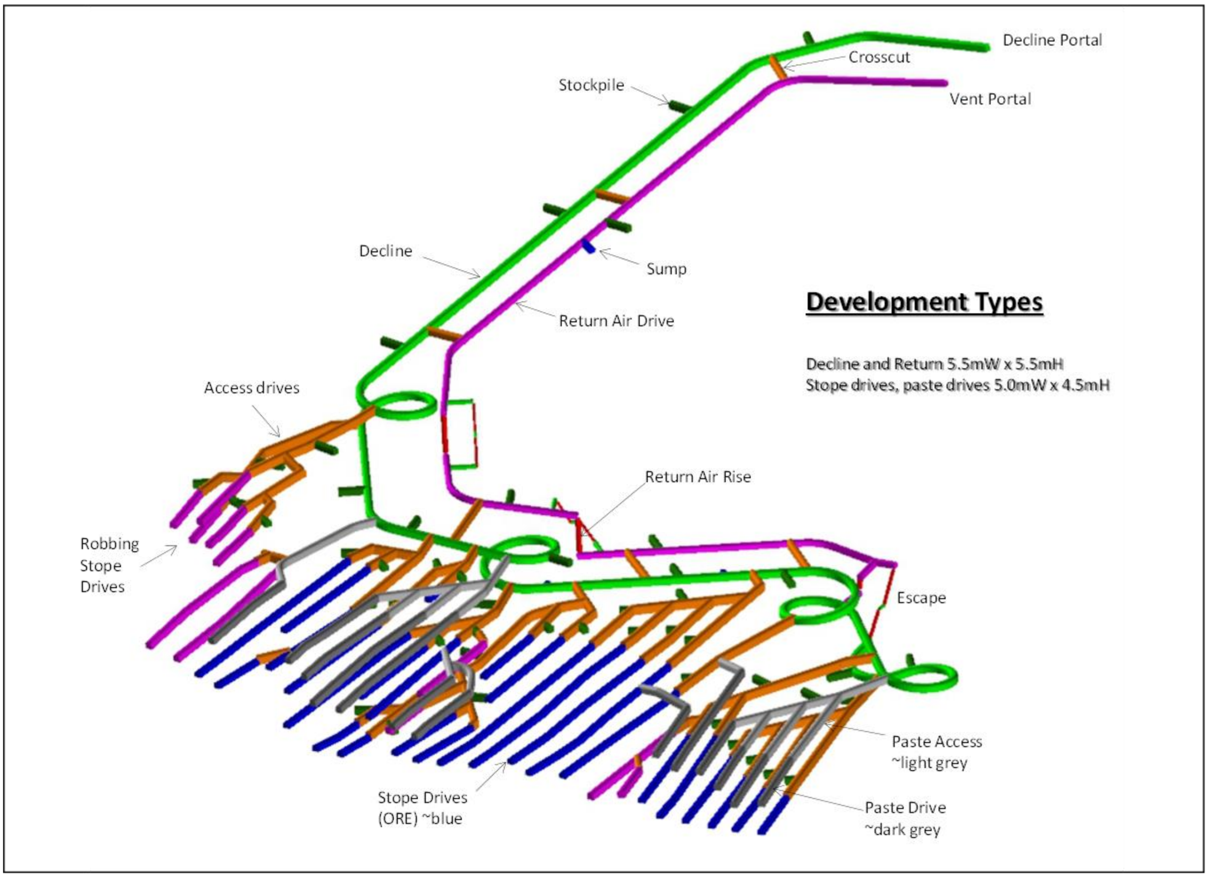


Figure ES 19: Development Design

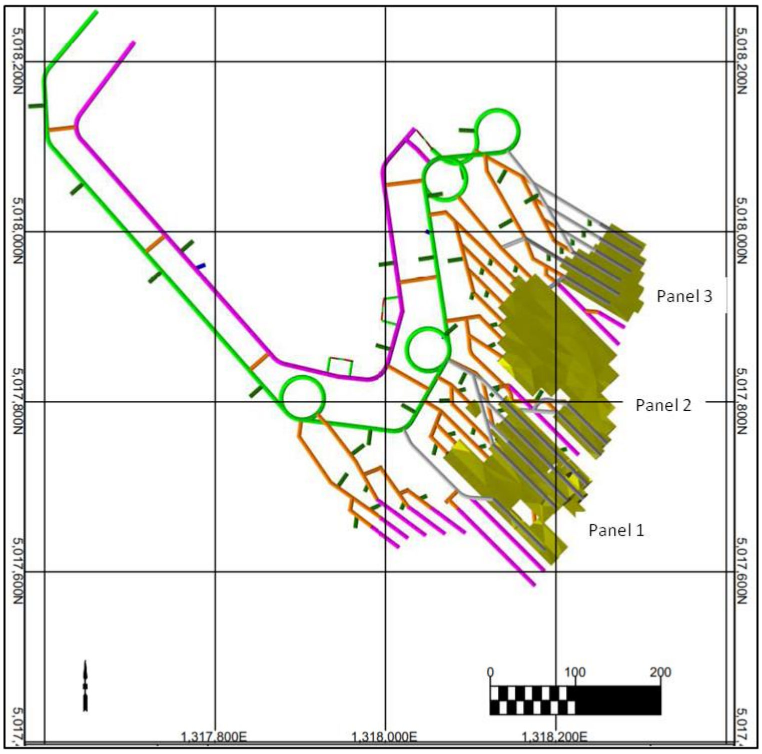


Figure ES 20: Panel layout

Table ES 21: Ore details and Waste by year

Processing Year		Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Indicated Ore								
Devt Ore (kt)	**253**	-	76	104	73	-	-	-
Devt g/t	**1.85**	-	1.79	1.73	2.08	-	-	-
Devt (koz)	**15**	-	4	6	5	-	-	-
Stope Ore (kt)	**2,035**	-	92	422	421	483	420	198
Stope g/t	**3.17**	-	1.97	3.02	3.18	2.90	3.11	4.86
Stope (koz)	**208**	-	6	41	43	45	42	31
Inferred								
Inferred Dev (kt)	**110**	-	13	53	43	-	-	-
Inferred Dev g/t	**1.83**	-	1.24	1.65	2.23	-	-	-
Inferred Dev (koz)	**6**	-	1	3	3	-	-	-
Inferred Stope (kt)	**15**						15	
Inferred Stope g/t	**4.82**						4.82	
Inferred Stope (koz)	**2**						2	
Total Mill feed - mined (kt)	**2,413**	-	181	579	537	483	435	198
Ore g/t	**2.99**	-	1.84	2.66	2.96	2.90	3.15	4.86
Ore (koz)	**232**	-	11	49	51	45	44	31
Waste(kt)	**565**	132	180	179	75	-	-	-
Total Movement (kt)	**2,974**	132	362	757	611	483	435	198

Note 1: Numbers may not add due to rounding.
Note 2: Devt Ore includes 146kt @ 1.4 g/t low grade (between 1.0 g/t and 1.7 g/t)
Note 3: Mined may not correspond to processed schedule timings.

Table ES 22: Development by Type

	Total	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Development TYPE								
Decline 5.5mW x 5.5mH	**1,845**	791	472	564	17	-	-	-
Stockpile 5.5mW x 5.5mH	**246**	87	70	87	-	-	-	-
Level Access 5.5mW x 5.5mH	**2,718**	36	679	1,189	814	-	-	-
Return Air Drive 5.5mW x 5.5mH	**961**	566	131	260	4	-	-	-
Vent access 5.5mW x 5.5mH	**229**	85	91	54	-	-	-	-
Sump 5.0mW x 4.5mH	**19**	11	4	4	-	-	-	-
Level Stockpile 5.5mW x 5.5mH	**153**	-	30	73	50	-	-	-
Remote Stockpile 5.5mW x 5.5mH	**244**	-	82	79	83	-	-	-
Paste Access 5.5mW x 5.5mH	**878**	-	466	199	213	-	-	-
Paste Drives 5.0mW x 4.5mH	**1,118**	-	591	297	230	-	-	-

Airleg Drive 3.0mW x 3.0mH	**71**	18	36	17	-	-	-	-
Escape Drive 2.0mW x 2.0mH	**19**	-	7	7	6	-	-	-
TOTAL LATERAL DEVT	**8,501**	1,594	2,660	2,831	1,416	-	-	-
Return Air Rise 4.0mW x 4.0mW	**59**	-	28	31	-	-	-	-
Escape Ladder 2.0mW x 2.0mH	**120**	-	38	36	46	-	-	-
TOTAL VERTICAL DEVT	**179**	-	66	67	46	-	-	-
Stope Drives 5.0mW x 4.5mH	**2,536**	-	825	1,357	354	-	-	-
Robbing Stope 5.0mW x 4.5mH	**758**	-	24	103	631	-	-	-
TOTAL STOPE DRIVES	**3,294**	-	849	1,461	985.	-	-	-
TOTAL LATERAL DEVT	**11,795**	1,594	3,509	4,291	2,401	-	-	-

Note: Numbers may not add due to rounding

Table ES 23: Paste, cabling, trucking, shotcrete

	Total	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Total Devt	11,974	1,594	3,575	4,358	2,447	0	0	0
Total Ore (kt)	2,413	0	181	579	537	483	435	198
Paste (kt)	1,893	0	75	339	372	387	437	283
Cable metres	36,818	900	12,731	9,681	7,170	2,246	2,277	1,813
Shotcrete (m3)	16,750	2,782	7,403	3,468	3,097	0	0	0
TKMS	4,507,634	80,833	459,203	1,128,293	1,057,336	845,851	673,002	263,116

Note: Numbers may not add due to rounding

ES 8.8 Mining Reserves

A Mining Reserve estimate was completed for the RAS and SRX deposits.

The following sections explain the estimation processes for the part of the deposits which can be economically mined and has had necessary mine planning work completed. Proved and Probable reserves are based on Measured and Indicated resource blocks. Inferred blocks are not included in the reserve estimate. Where Inferred blocks are within the pit outlines, they represent potential minable inventory if confirmed by grade control drilling.

Similarly in the underground mine plan, some Inferred mineralisation is encountered as part of the mine development sequence.

ES 8.8.1 Open pit

RAS and SRX open pit reserve tonnages and grade are reported from pit designs that are guided by Whitte 4X Optimisation Software (Whittle®).

A local currency adjusted gold price from a base of USD1,650/oz was applied for RAS estimates and similarly USD2,100/oz for SRX. The projected mining and processing costs, pit slope parameters, processing recovery, a 2% ad valorem royalty payable to the New Zealand Government and between 2.5 and 3.5% other royalties, and refining and handling charges at NZD 8/oz, have been used in the optimisation

Dilution, mining losses and recovery were incorporated into the mining block model by re-blocking the resource block model from 2m high blocks to 2.5m high blocks. This is the height of the planned ore flitches, incorporating adequate dilution and provides a reasonable estimate of mined tonnage and grades.

Pit optimisation and design inputs and methodologies are discussed in ES 8.2 Pit Optimisations.

ES 8.8.2 Underground

The underground mineable portion was based on the development and stope designs created manually.

A gold price of USD1,650/oz for RAS, projected mining and processing costs, processing recovery, a 2% ad valorem royalty payable to the New Zealand Government and 2.5% other royalties, and refining and handling charges at NZD 8/oz have been used in the optimisation.

A global 10% dilution was applied to achieve the final cut-off grade.

ES 8.8.3 Reserve Estimate

The open pit Mineral Reserves summarised in Table ES 24 are reported at the cut-off grades listed below the table.

Table ES 24: BOGP Mineral reserve estimate as at 1 December 2024

Area	Proven		Probable		Total		
	Mt	Au g/t	Mt	Au g/t	Mt	Au g/t	Au koz
RAS (open pit)	-	-	11.9	2.42	11.9	2.43	928
RAS (Underground)			2.3	3.03	2.3	3.03	223
SRX			1.3	0.70	1.3	0.70	30
Total	-	-	15.5	2.37	15.5	2.37	1,181

Note 1: RAS Open pit cut-off grade 0.3 g/t at $US1,650/oz Au price
Note 2: RAS Underground cut-off grade 1.70 g/t at $US1,650/oz Au price
Note 3: SRX Open pit cut-off grade 0.35 g/t at $US2,100/oz Au price
Note 4: Underground Reserves are from the Open pit Resources area

ES 8.9 Relevant Factors

- The effective date of the Mining Reserve Estimate is 1 November 2024, estimated by Rodney Redden (MAusIMM and CP-Mining), a contractor to Santana Resources Ltd.
- There are no approved consents and not all required permits are in place to enable mining of the RAS and SRX deposits.

ES 9. Plant feed schedule – all sources

Mining commences at RAS open pit and begins supplying ore from month 15 so that from month 17 processing starts on RAS open pit ore.

The RAS underground commences development in month 19 of processing (36 months after RAS open pit commencing). Initial underground ore is stockpiled and is timed to commence feed to the process plant as the processing rate expansion to 2.1Mtpa is complete.

Due to the lower economics, the SRX open pit is mined and processed last F.

All the fresh, Indicated and Inferred material above a cutoff grade of 0.3 g/t Au is considered as ore and planned to be sent either directly to the run-off mine (ROM) pad or placed on a temporary ROM rehandling stockpile. The ROM material has been further subdivided into the following grade bins:

- High Grade (HG) >= 2.6 g/t Au
- Run-of-Mine 2 (ROM2) 1.6 – 2.6 g/t Au
- Run-of-Mine 1 (ROM1) 0.6 – 1.6 g/t Au
- Low Grade (LG) 0.3 – 0.6 g/t Au

Material is fed to the processing plant in the above priority order, whilst maintaining a maximum 3.5 g/t head-grade.

Table ES 25: Annual Processing Schedule

			YEAR	1	2	3	4	5	6	7	8	9	10
RAS OP	Probable Reserves	kt	11,864	1,375	1,466	1,401	1,381	1,414	1,372	1,359	1,639	456	-
		g/t	2.43	3.66	3.38	3.09	2.70	1.08	2.97	1.58	1.73	0.49	-
		koz	928	162	159	139	120	49	131	69	91	7	-
	Inferred	kt	1,112	81	34	99	119	86	128	240	112	213	-
		g/t	1.54	0.86	2.84	2.83	1.28	1.65	2.02	1.91	0.77	0.78	-
		koz	55	2	3	9	5	5	8	15	3	5	-
	Total OP	kt	12,976	1,456	1,500	1,500	1,500	1,500	1,500	1,599	1,751	670	-
		g/t	2.36	3.50	3.37	3.08	2.59	1.11	2.89	1.63	1.67	0.58	-
		koz	983	164	162	148	125	54	139	84	94	12	-
RAS UG	Probable Reserves	kt	2,288	-	-	-	200	600	600	488	327	73	-
		g/t	3.03	-	-	-	2.11	2.70	3.10	2.76	4.11	4.60	-
		koz	223	-	-	-	13,582	52,135	59,840	43,301	43,195	10,812	-
	Inferred	kt	124	-	-	-	0	0	0	13	22	90	-
		g/t	2.19	-	-	-	0.00	0.00	0.00	1.82	1.82	2.33	-
		koz	9	-	-	-	0	0	0	757	1,268	6,720	-
	Total UG	kt	2,413	-	-	-	200	600	600	501	349	163	-
		g/t	2.99	-	-	-	2.11	2.70	3.10	2.74	3.97	3.35	-
		koz	232	-	-	-	13,582	52,135	59,840	44,058	44,463	17,532	-
SRX	Probable Reserves	kt	1,340	-	-	-	-	-	-	-	-	1,216	124
		g/t	0.70	-	-	-	-	-	-	-	-	0.70	0.70
		koz	30	-	-	-	-	-	-	-	-	27,226	2,790
	Inferred	kt	20	-	-	-	-	-	-	-	-	10	10
		g/t	0.48	-	-	-	-	-	-	-	-	0.48	0.48
		koz	0	-	-	-	-	-	-	-	-	152	152
	Total SRX	kt	1,360	-	-	-	-	-	-	-	-	1,226	133
		g/t	0.69	-	-	-	-	-	-	-	-	0.69	0.69
		koz	30	-	-	-	-	-	-	-	-	27,378	2,942
Total Inferred			7%	6%	2%	7%	7%	4%	6%	12%	6%	15%	7%

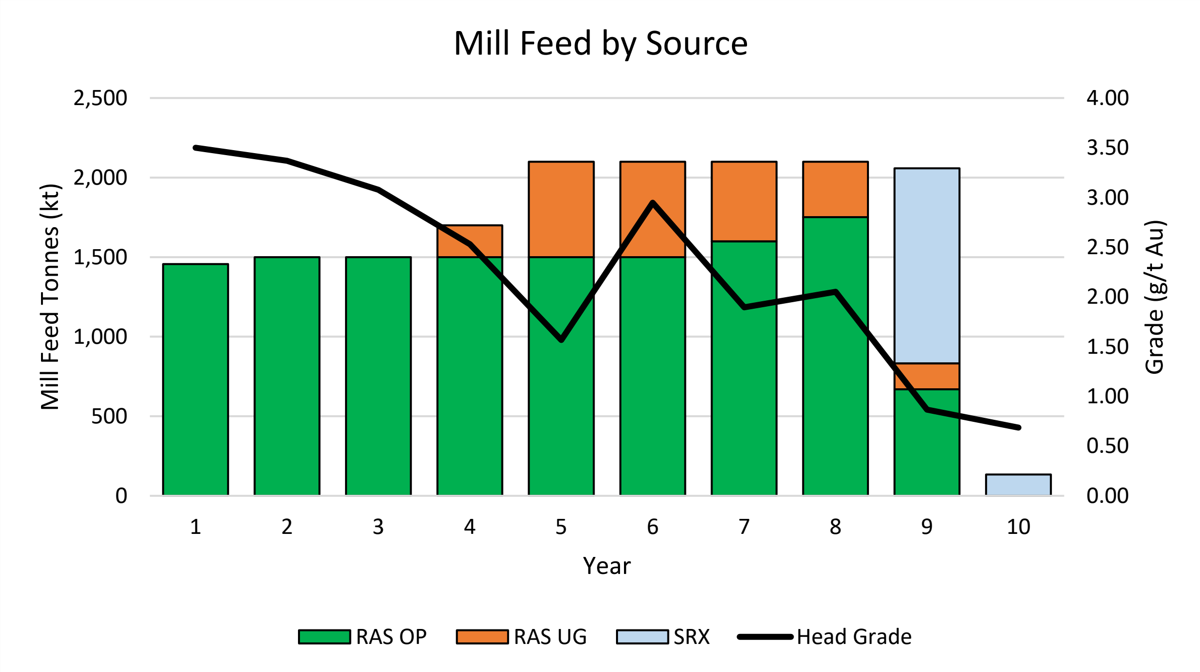


Figure ES 21: Mill feed by source

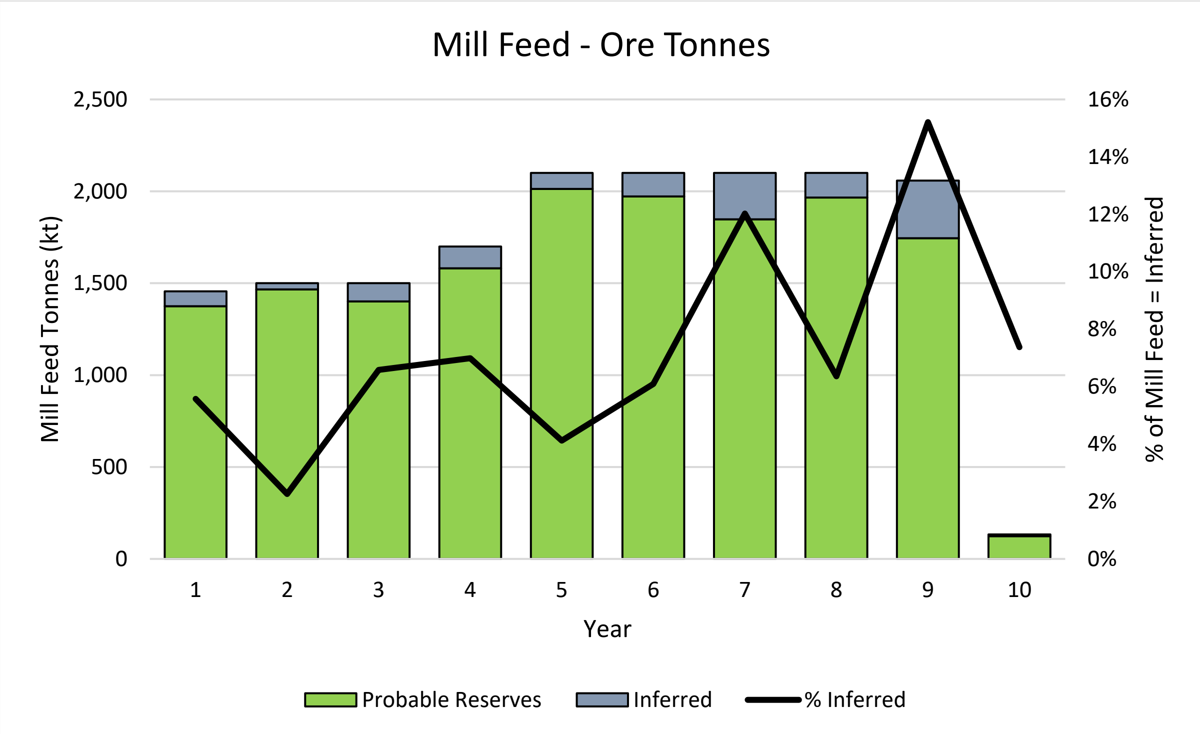


Figure ES 22: Mill feed tonnes by resource class

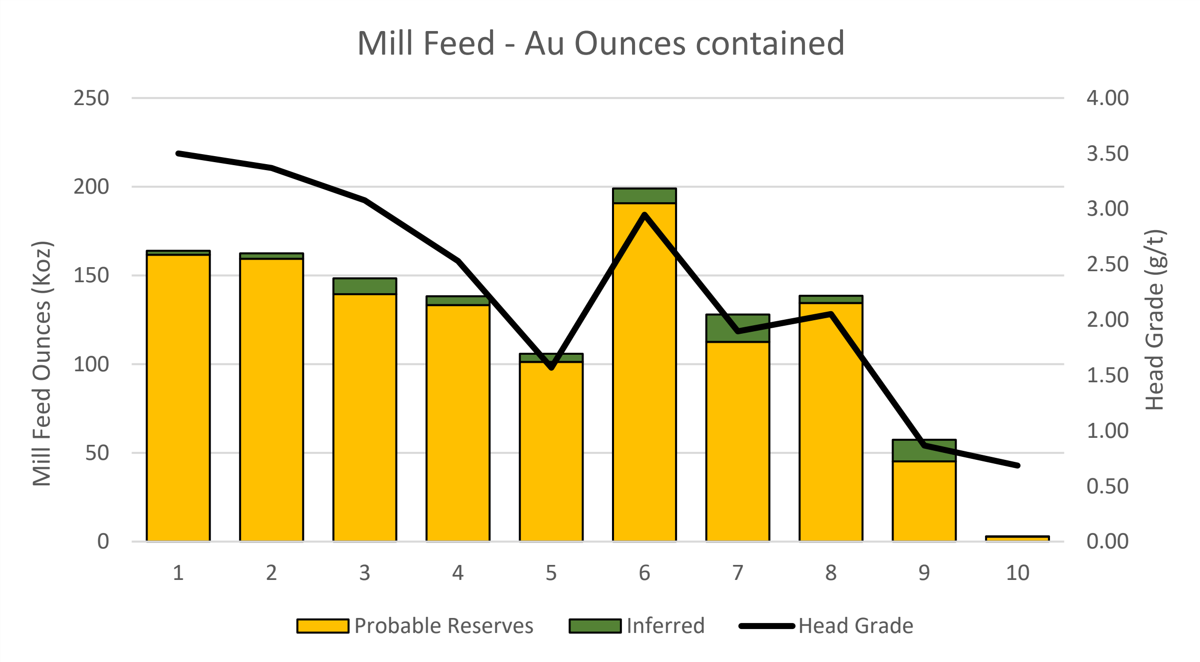


Figure ES 23: Mill feed ounces by resource class

ES 10. Processing

The process plant design for the RAS gold deposit is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is based upon unit operations that are well proven in industry.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a compact footprint that will minimise construction costs.

The key project and ore specific criteria that the plant design must meet are:

- 1.5Mtpa of primary ore, easily expandable without plant downtime to 2.1Mtpa;
- mechanical availability of 91.3% supported by crushed ore storage, standby equipment in critical areas and grid based power supply; and
- sufficient automated plant control to minimise the need for continuous operator interface and allow manual override and control when required.

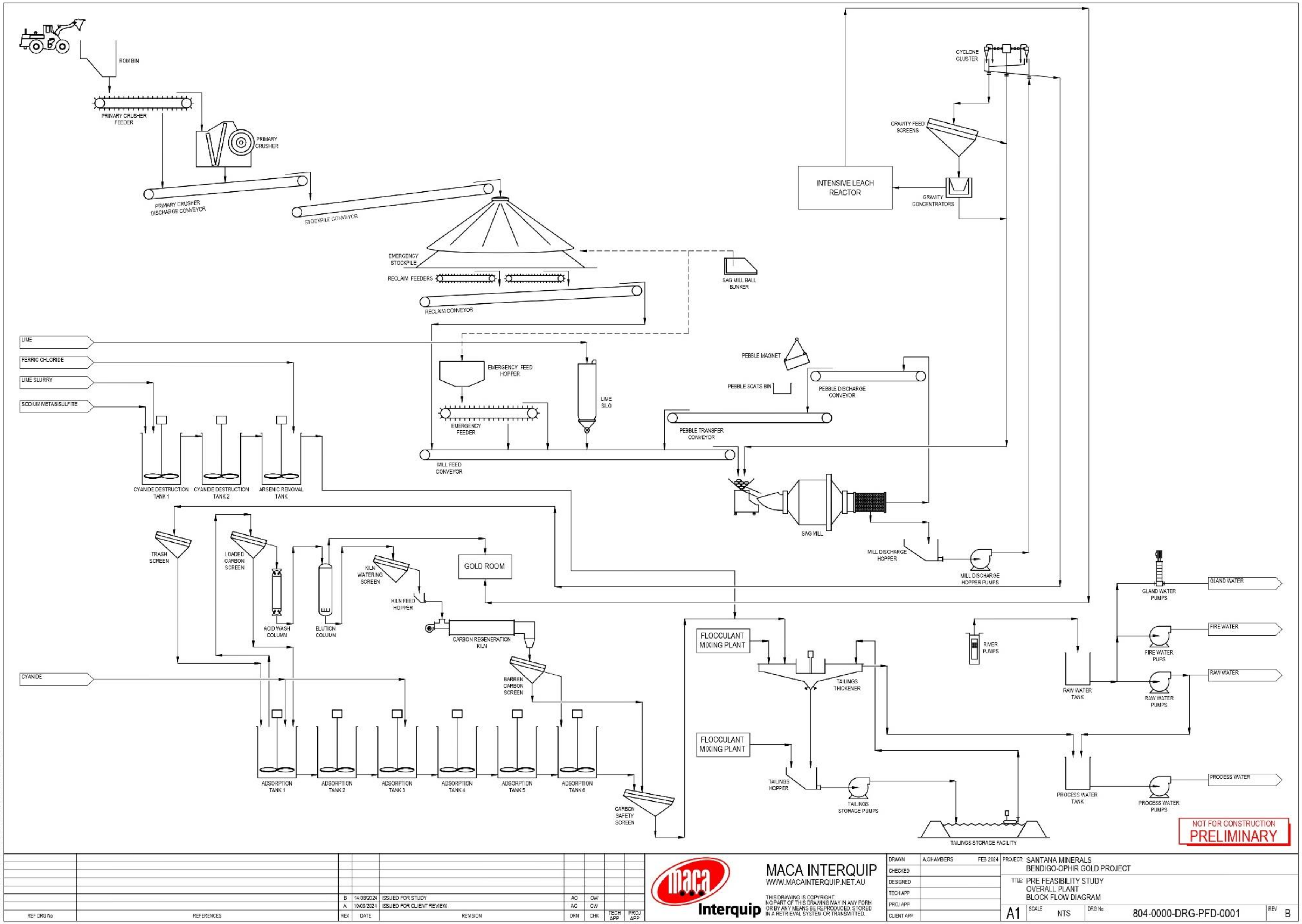


Figure ES 24: BOGP Process plant – process flow diagram

ES 10.1 Run-of-Mine (ROM) Pad and Crushing Circuit

The ROM pad will contain up to 14 days of stockpiled ore to provide a buffer between the mine and the plant. This will allow the crushing plant to only operate for 12 hours per day.

The primary jaw crushing circuit has been sized based on operating 12 hours per day at 80% utilisation at a feed rate 128% above the mill feed rate. Ore will be stockpiled so that the downstream plant can continue operating during the periods that the crushing plant is not operational.

A run of mine (ROM) grizzly aperture of 700 mm has been selected to minimise oversize material entering the ROM bin and causing down-stream blockages.

An apron feeder has been selected to draw material from the ROM bin.

A single toggle jaw crusher has been selected for the primary crushing duty due to the moderate UCS values and the moderately abrasive nature of the ore. In addition, the single toggle crusher has a higher capacity than an equivalently sized double toggle crusher.

Ore discharged from the jaw crusher will be conveyed to the covered stockpile. An overhead magnet located on the crusher discharge conveyor will remove magnetic tramp material that may be present with the ore.

Ore will be withdrawn from the crushed ore stockpile using a variable speed apron feeder that will discharge ore onto the mill feed conveyor.

ES 10.2 Milling

A single stage SAG mill has been selected to reduce crushed product to the nominal circuit P_{80} size of 106 µm.

ES 10.3 Classification

Relatively large diameter cyclones (380 mm) have been selected for the classification duty to minimise wear and reduce the potential for spigot blockages occurring from coarse ball mill discharge material.

ES 10.4 Gravity Concentration

Testwork indicates that relatively high gravity gold recoveries (up to 32%) can be expected from the primary ores, with high gold recoveries and rapid leach kinetics achieved on the gravity tails material.

The gravity circuit has been designed with two installed centrifugal concentrators. The gravity concentrator will be fed with cyclone underflow slurry.

The gravity concentrate periodically discharging from the concentrators will be directed to a storage hopper located in the gold room. The concentrates will be processed through a vendor supplied Intensive Leach Reactor and dedicated electrowinning cell to recover gold and silver.

Selection of the Intensive Leach Reactor unit has been based on the expected high gravity gold recovery and the potential impact of returning conventional gravity / shaking table tails to the leach circuit. The Intensive Leach Reactor will utilise high intensity cyanidation to maximise the recovery of gold and silver values from the concentrate stream.

A separate pregnant solution tank, pump and electrowinning cell will be provided for the Intensive Leach Reactor to assist with metallurgical accounting and eliminate any potential impact on the operation of the carbon elution and electrowinning circuit.

ES 10.5 Trash Screening

A vibrating trash screen has been selected to prevent oversize particles from entering the downstream leach and adsorption circuit. Although minimal trash is expected from the primary ore, good trash screening will be essential for good carbon management.

ES 10.6 Leach and Adsorption Circuit

Metallurgical test work indicated that the primary ore shows minor "preg-robbing" characteristics. The initial leach kinetics for primary ore are fast, with approximately 85% to 90% of total CIL gold extraction being achieved in the first 4 to 8 hours of leaching. On this basis, a circuit configuration utilising six adsorption tanks has been adopted. Six adsorption tanks are the minimum number required to maintain reasonable overall stage efficiencies.

Due to the minor preg-robbing behaviour a hybrid leach cannot be adopted, reducing the solution tenor of the first adsorption tank, and driving slightly lower loaded carbon grades than can be achieved in a hybrid circuit. Lower carbon loadings will increase the batch size and frequency of carbon elution.

The adsorption tanks will be identical in size with a total circuit residence time of 24 hours at 45% w/w density in the tanks. The tanks will be arranged such that any one tank can be taken offline for maintenance without affecting the remainder of the circuit.

ES 10.7 Elution

An AARL elution circuit has been selected based on excellent raw water supply and quality. An AARL elution circuit separates elution and electrowinning, allowing more batches to be processed if required. An eight-tonne batch size has been nominated and based on the calculated carbon movements required to meet the grade variability that is currently predicted by the production plan. Under nominal conditions a total of six elution cycles are required each week. The gold room will operate 7 days a week.

Three parallel 12 cathode electrowinning cells are proposed for the gold room to provide a high pass efficiency (greater than 90%) and ensure a low gold tenor in the spent electrolyte returning to the strip solution tank.

The high cell pass efficiency will ensure a near barren solution is returned to the strip solution minimising the gold returned to the column and minimising the number of elution cycles required to achieve the target barren carbon grade. It is anticipated that the elution electrowinning cycle will be completed in 8 to 12 hours. This will allow additional strips to be conducted during the week, if required.

A sludging cell design has been adopted for electrowinning to simplify the cathode handling process. Sludge will be filtered in a vacuum pan filter and dried in an oven prior to smelting to produce doré.

ES 10.8 Cyanide destruction

An air/SO_2 circuit has been selected for cyanide destruction based on the relatively lower operating cost of these circuits, the less hazardous reagents required in comparison to Caros acid and the amenability shown in the testwork of the ore to this form of cyanide destruction. The circuit will reduce the weakly acid dissociable cyanide to less than 30 ppm at discharge of the TSF spigot.

ES 10.9 Arsenic removal

Ferric chloride precipitation of solubilised arsenic as a ferric arsenate has been selected based on the amenability of the ore to this removal method, and the anticipated stability of the arsenate species generated.

ES 10.10 Tails Thickening

A high-rate tails thickener treating the plant tailings has been selected to thicken tails to a density of 50% w/w solids. This is intended to minimise the volume of tailings being pumped to the TSF and volume of decant requiring return, as well as the mass of solubilised species entering the TSF.

ES 10.11 Tailings Pumping

Thickener underflow will be pumped to tailings hopper and will be pumped to the tailings storage facility using two centrifugal pumps operating in series.

ES 10.12 Throughput expansion

The plant throughput will be expanded from 1.5Mtpa (base case) to 2.1Mtpa as open pit grades from RAS decline. To facilitate this the following approach has been adopted:

1. Orway Mineral Consultants (OMC) have reviewed the comminution circuit design, and made the following recommendations:
    a. The base case crushing plant design can provide the required feed, however the stockpile size is too small (residence time will be reduced to 8 hours).
    b. During expansion a ball mill with installed power of 2,000 kW should be installed.
    c. The cyclone cluster was found to be adequately sized for the increased throughput.
2. Initial installation of slurry and service pumps will allow for the expanded case.
3. Initial screen selections will allow for the expanded case.
4. No change will be made to the CIL tank sizes; however, this must be reviewed once kinetic data is available for the circuit. Residence time will decrease from 24 hours (allowed in the base case due to lack of kinetic data) to 17 hours.
5. The period of increased throughput will coincide with decreased head grade, and the selection of column and electrowinning sizes for plant feed grade variability means that these circuits will be adequately sized in the initial design.
6. Initial electrical design will allow for the inclusion of the later Project phase.

The decision to utilise equipment sized for the larger case will reduce plant downtime during the expansion, with the only significant construction works required being the inclusion of the ball mill. Room has been allowed in the base case design and implementation plan for this to be done.

The Project execution will therefore proceed in two phases – the initial installation (Phase 1), with a schedule the same as the base case design but with increased capital cost, and Phase 2, where the ball mill will be procured, design work undertaken for its installation, installation of the mill and commissioning.

The process plant General arrangement together with the run of mine (ROM) pad and other site infrastructure is shown in Figure ES 25.

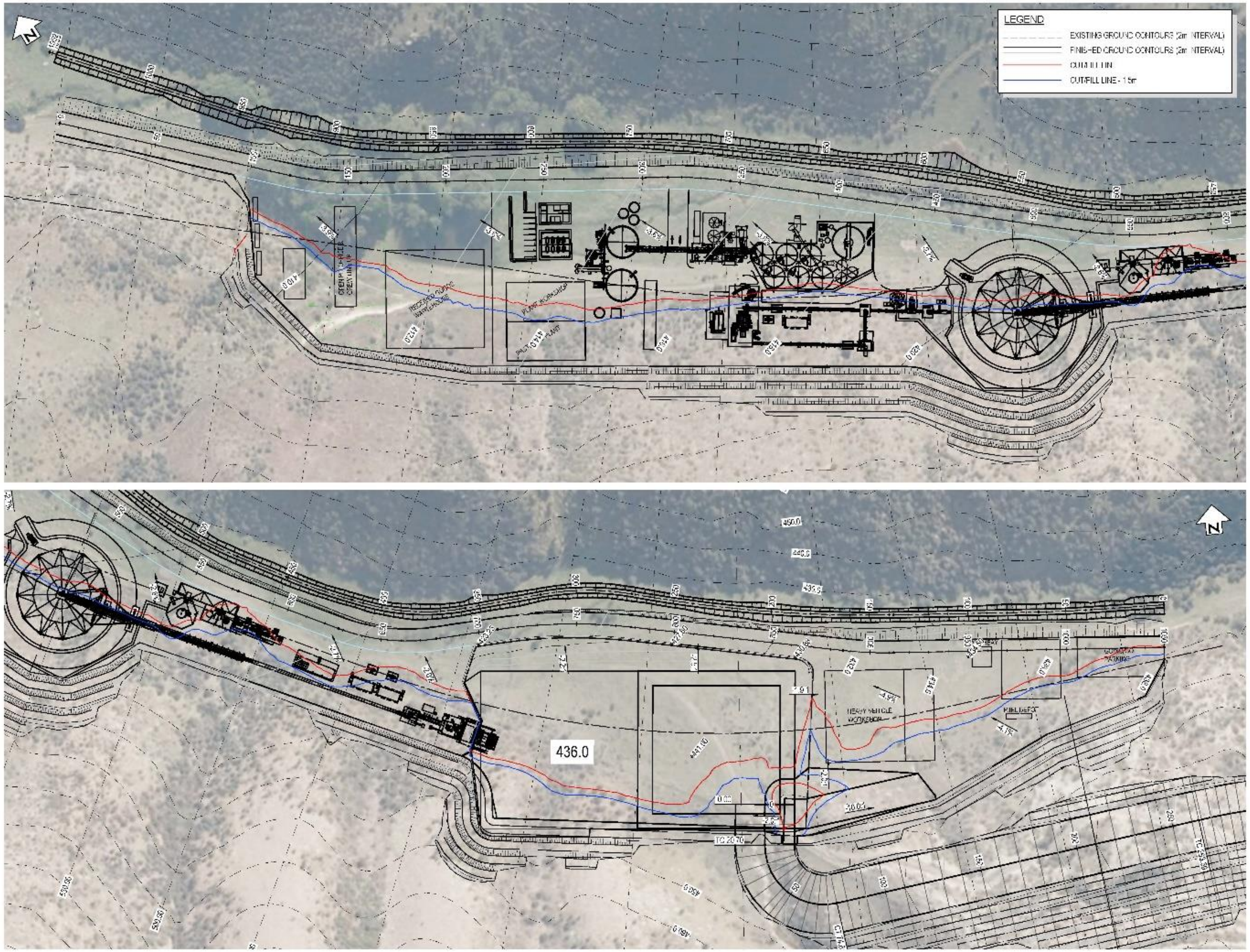


Figure ES 25: Process plant – general arrangement

ES 11. TSF

The proposed Shepherds TSF is in the upper reaches of Shepherds Valley with the proposed Shepherds ELF located immediately downstream (Figure ES 26). The final ELF height will match, and in places exceed, the TSF embankment and extends downstream of the TSF by 1.4 km.

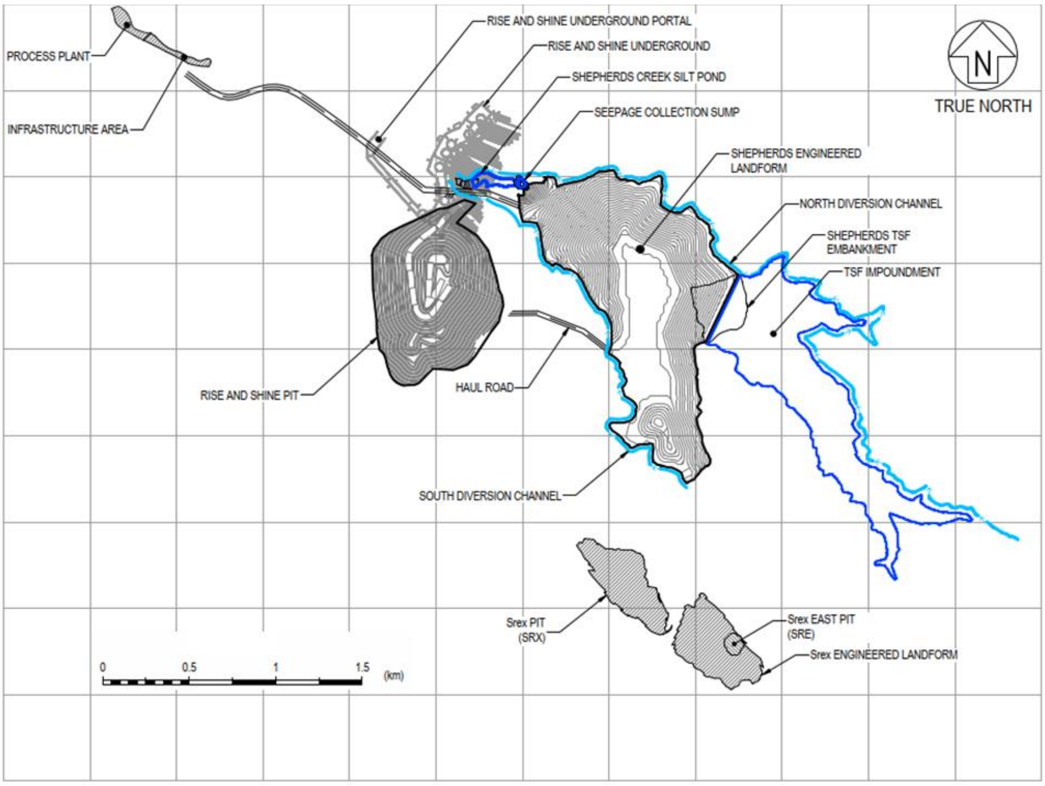


Figure ES 26: shows the location of the TSF and ELFs

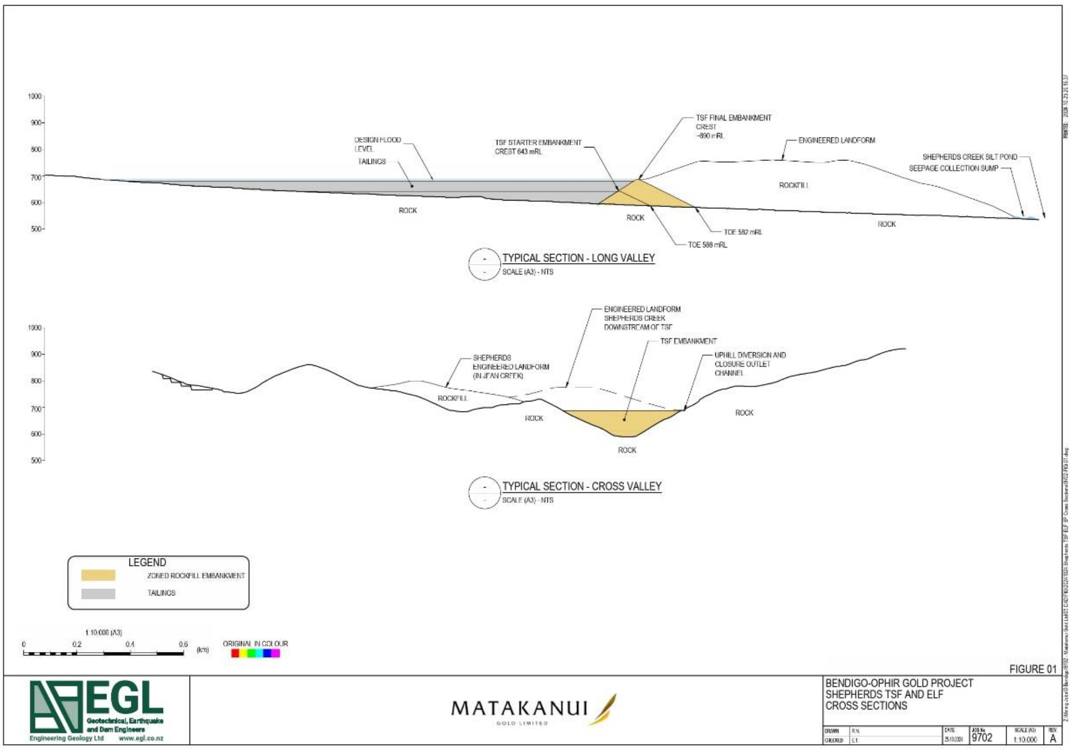


Figure ES 27: TSF and ELF sections

ES 11.1 Design

The Shepherds TSF is to be formed using a downstream construction embankment using rockfill from the RAS open pit. The downstream construction embankment is proposed to have a 1 vertical to 2 horizontal downstream slope and 1 vertical to 1.5 horizontal upstream slope. It is proposed that the tailings are delivered as a conventional slurry from the process plant and discharged using a combination of spigots and end pipe methods.

The Shepherds TSF is designed to meet the New Zealand Dam Safety Guidelines. The proposed final crest level allows for the TSF to be managed as a full containment facility (supernatant water managed onsite). This includes allowance to manage a normal operational decant pond and inflows from a 72-hour probable maximum precipitation (748 mm depth), with 1m freeboard for wave action. The uphill diversion channels are conservatively assumed not to function for this design condition.

Consideration of a probable maximum precipitation event meets the design criteria for a High Potential Impact Classification Dam under the New Zealand Dam Safety Guidelines. A High Potential Impact Classification is the highest dam classification under the New Zealand Dam Safety Guidelines which are in accordance with international practice. For comparison the average annual rainfall at the top of the TSF catchment is estimated to be approximately 540 mm including allowance for increased rainfall depths due to topographical effects.

The Shepherds TSF will safely contain tailings when subjected to potential future extreme earthquakes. It will be designed to withstand a 1 in 10,000 year earthquake including aftershocks. This includes withstanding a potential rupture on the Alpine Fault or any of the other active faults in the region. The proposed design has the tailings contained behind the downstream rockfill embankment, that will also be buttressed by a large volume of rockfill placed in the Shepherds ELF. The proposed design will provide safe and robust tailings storage solution for both operation and post closure of the site.

ES 12. Infrastructure

ES 12.1 Mining Operations and Processing Plant Site & Infrastructure

The Mine Site and supporting infrastructure are located in the lower Shepherds Creek valley, positioned across two areas. The first is 14 ha in area for the processing plant and mining operations within the valley that is secluded from the Bendigo and Ardgour Terraces and the second area of 10 ha positioned outside the valley on the Ardgour Terraces providing non-operational infrastructure including administration building, high voltage substation and construction activity support.

To establish the processing plant and mining operations area the valley will be widened to a site platform of 130m x 1,050m, engineered to accommodate the plant and supporting infrastructure and buildings. A 7-month period is planned to complete the earthworks establishment of the lower Shepherds Creek site.

The mining operations area includes heavy vehicle workshops and service infrastructure, a future paste plant, goods warehouse and office and administration areas to support the open pit and future underground operations. All supporting infrastructure including power reticulation, water treatment, stormwater management, parking and circulation are included.

The Ardgour site provides for a temporary 82-person construction camp for the duration of the construction phase and allocation for the mine administration building and carparking, high voltage substation, interim heavy vehicle workshop and construction lay down areas.

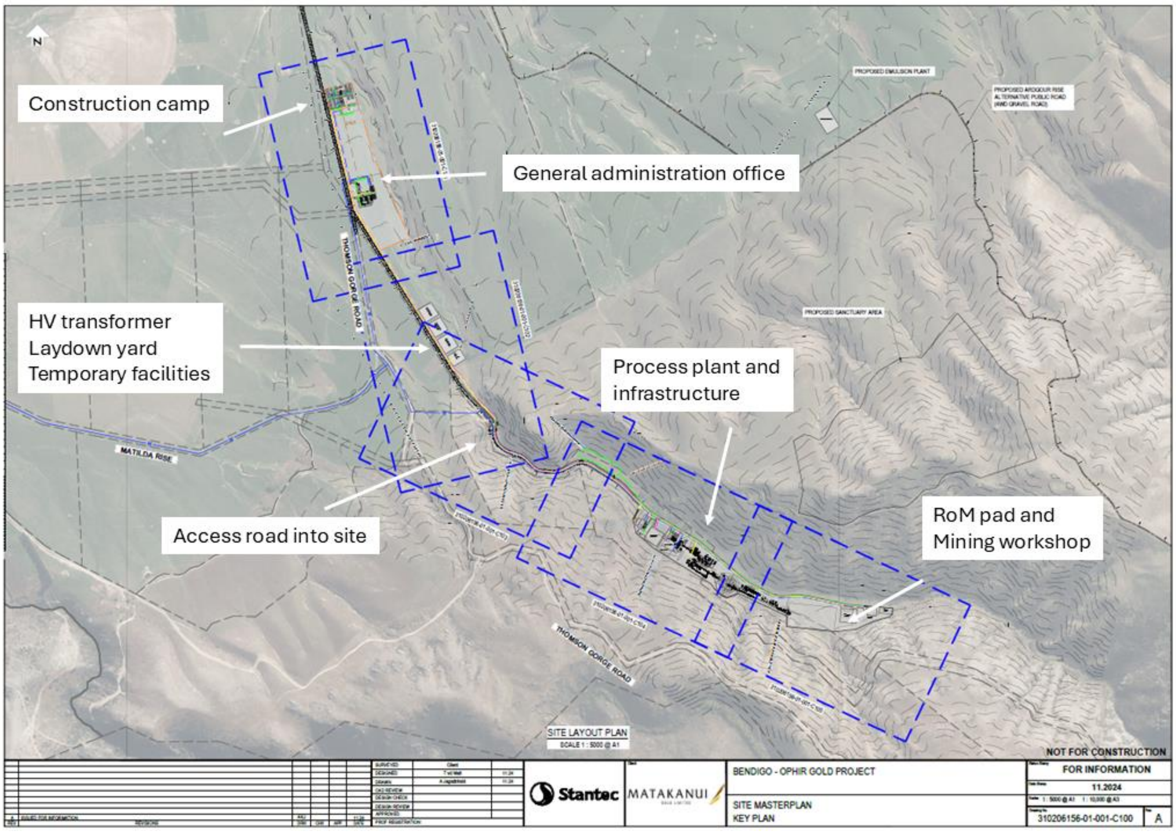


Figure ES 28: Main BOGP Infrastructure area

ES 12.2 Project Water Supply

A water supply is required to support project operations, a total supply peak of 97 l/s, or 8,381 m^3/day at project start-up with future demand declining to a range of 50-70 l/s as more recycled water becomes available from the processing and mining operations.

A reliable water source from the Bendigo aquifer has been developed, with the first of two bores constructed in close proximity to the Bendigo Creek and the exploration base, off Bendigo Loop Road. A dedicated pipeline (315mm HDPE) is to be constructed using in-bore pumps, and a booster pump station on Matilda Rise, to lift the water approximately 200m vertically and 6.5km, to the mine site in the lower Shepherds valley. A water permit will be applied for to secure water rights.

ES 12.3 Power Supply

The electrical load requirements for the new mine site have been modelled based on the known loads of the mine for the initial stages of operation, 13.8MVA and expected future loads for the underground phases of the mine's lifecycle, to a total 19.5MVA.

The new supply distribution is currently connecting to the Aurora Energy owned, Upper Clutha Network via the Lindis Crossing Substation, installing a new 66kV overhead network approximately 10km north of the mine, utilising road reserve for the above ground network to a single 66 /11kV, 24 MVA transformer located at the entrance to the lower Shepherds Creek gorge, approx. 500m from the process plant and mine operations site. The wider site and plant will be connected by multiple 11kV networks.

ES 12.4 Site Access and alternate route for Thomson Gorge Road

ES 12.4.1 Site Access

New and upgraded road access is required to provide for personnel and visitor access, delivery of construction and mine operation equipment, plant and vehicles.

Following an options assessment the preferred route is based on reduced impact to the current roading network and to neighbouring property owners and businesses. The access will be from SH8 (Cromwell – Tarras), approximately 24km from Cromwell, via Ardgour Road, then Thomson Gorge Road (TGR) to the mine site in the lower Shepherds valley, a 7.3km route from the SH8 intersection. The final leg is a 540m extension through existing road reserve in the lower Shepherds valley gorge.

ES 12.4.2 Thomson Gorge Alternate (TGR) Route – Ardgour Rise

Proposed mining activities will likely disrupt normal use of Thomson Gorge Road.

An alternate 4WD drive diversion access road will be established that follows Ardgour Station ridge line commencing in the west from the existing TGR in the Lindis Valley and to the east re-joining close to Matakanui Station.

Chorus telecommunication's network has a fibre optic cable located in the TGR that will be reinstated to maintain connectivity within the new Ardgour rise road alignment.

ES 13. Costs

ES 13.1 Operating Cost Estimates: Open pit

ES 13.1.1 Basis of Estimate

Operating cost estimates have been completed from first principles, based on an owner mining model generated by the competent person. Physicals are used as drivers and unit rates are based on demonstrated rates from suppliers and similar style operations such as OceanaGold's nearby Macraes Gold Mine. Cost estimates accuracy are expected to be in the range of +/-15% and are expressed as 2024 NZD.

ES 13.1.2 Mining Operating Costs

The mining unit cost averaged over life-of-mine (LOM) is approximately NZ$3.74 /t and includes:

- Ore and waste drill and blast;
- Ore and waste load and haul;
- Grade control sampling and assaying;
- Crusher feed and stockpile rehandling;
- Ancillary equipment for supporting mining activities;
- Mine management and technical services costs;
- Leasing, maintenance and servicing of all mining and ancillary equipment;
- Pit dewatering and Services (Lighting, work area maintenance, signage, haul road and access road maintenance); and
- Top soil stripping and surface access haul roads maintenance.

The mining fleet ownership costs are included.

Table ES 26: LoM open pit mining costs

Operating Costs	Amount NZD m	Unit cost ($/t moved ex-pit)
Loading	76.1	0.34
Hauling	291.0	1.31
Drilling	87.2	0.39
Blasting	51.3	0.23
Grade control	1.70	0.01
Crusher feed	10.3	0.05
Ancillary	109.9	0.50
Mining operations overheads	189.7	0.86
Stockpile rehandle	7.6	0.03
Pit clear and grub	3.7	0.02
Total Mining Costs	**828.4**	**3.74**

ES 13.2 Operating Cost Estimates: Underground

ES 13.2.1 Basis of Estimate

Operating cost estimates have been completed from first principles, based on an owner mining model. Physicals are used as drivers and unit rates are based on demonstrated rates from suppliers and similar style operations such as OceanaGold's Macraes Gold Mine. Cost estimates accuracy are expected to be in the range of +/-15% and are expressed as 2024 NZD.

ES 13.2.2 Mining Operating Costs

The total operating costs are estimated at $224.4M and mining unit cost averaged over life-of-mine (LOM) at NZ$93.01 /t processed and includes:

- Development and stoping direct costs. Ground support, explosives, mine services, assaying, etc. Driven by metres and tonnes
- Mobile fleet operating costs. Maintenance and running costs for drills, LHD's, trucks and all auxiliary fleet
- Infill diamond drilling.
- Power.
- Labour. Management, Mining operations, maintenance and technical services
- Paste. Cement and plant maintenance
- Miscellaneous. Miners tools, technical services, fan and pump maintenance

Post the underground development phase (months 1-24) the unit costs are $78.64/ tonne processed including on-going mine development.

Table ES 27: LoM Underground mining operating costs

	Total LOM		Month 25+ (post mine development)	
	$NZD M	Unit cost ($/tonne processed)	*$NZD M*	Unit cost ($/tonne processed)
Direct Operating Cost per metre (consumables)	20.7	8.57	12.8	5.32
Fleet costs	37.1	15.38	28.9	11.96
Infill Diamond drilling	3.5	1.45	2.5	1.04
Power	16.0	6.63	13.4	5.55
Labour	107.0	44.36	79.4	32.91
Paste Plant Operations (Directs)	37.2	15.40	35.9	14.87
Miscellaneous	2.9	1.22	2.6	1.06
Total	224.4	93.01	175.4	78.64

ES 13.3 Operating Cost Estimates: Processing

The Processing costs were estimated based on the processing flowsheet for 1.5Mtpa throughout and 2.1Mtpa throughput. With a breakdown by fixed and variable cost elements.

The LOM unit cost per tonne processed is estimated at $18.92 from a total $318.3M.

Table ES 28: LoM Processing operating costs

	Rate	$NZD M
1.5Mtpa - fixed annual	$11,637,438	42.7
1.5Mtpa - variable ($/t processed)	$11.40	62.2
2.1Mtpa - fixed annual	$13,151,033	72.3
2.1Mtpa - variable ($/t processed)	$11.49	130.5
Met-lab sampling (annual 1.5Mtpa)	$925,480	10.5
Total		318.3
Unit cost - LOM	$18.92	

ES 13.4 Operating Costs: General and Administration

The General and Administration costs have been estimated and are broken into three broad categories:

- Labour. For steady state operations a 25-person support team including General Manager, Commercial and Administration manager accountants, accounts payable, environmental, HR, payroll, nurse, OHS.
- General site support. Worker transport, community relations, consultants, light vehicles, office expenses, legal fees, recruitment and auditing
- Specific site support. Land leases, Site water supply (power), Ecology sanctuary, water treatment and closure bonding.

Over the LOM this equals $74.0M or $4.40 tonne processed.

Table ES 29: General and Administration operating costs

Cost area	$NZD M	Unit cost ($/tonne processed)
Operations labour	27.2	1.62
Other - general	20.7	1.23
Other - specific	26.1	1.55
Total	74.0	4.40

ES 13.5 Capital Cost Estimates

Capital cost estimates include project capital, sustaining and closure

ES 13.5.1 Site establishment

The majority of these costs are project (establishment) costs and are summarised below.

Owners team: The labour and support costs for the owners construction team is estimated at $4.4M.

Land and Landowner costs, which include:

- Land purchases; and
- Payments to landowners on project start-up including up-front royalties.

The estimated costs are $7.2M

Site infrastructure:

- HV power connection to the site from the local 66kV network including a 66kV to 11kV transformer. ($12.8M)
- Road access upgrades and new roads connecting from the state highway, plus a local public road diversion (which is a sustaining cost item). ($13.9M)
- Service water system establishment. Borefield, pumps and pipelines. ($5.7M)
- The construction camp establishment and running cost. ($9.9M)
- Site facilities. Administration and mining offices, mobile fleet workshop, refueling, washdown, warehouse, associated facilities and theservices reticulation. ($23.3M)

Establishment earthworks are required across the terrain. These include:

- Diversion drains. Clean water diversion drains around the TSF and main Shepherds ELF ($1.0M);
- Sediment ponds. This most notably includes a pond in the mid Shepherds valley to contain mine impacted waters from the upper Shepherds and Jeans Creek basins and would be a final collector from the RAS pit stripping also ($1.7M); and
- The site haul road network which will involve cuts from the mining locations to the ROM pad, ore-stockpile and ELF ($13.8M).

Table ES 30: Site establishment costs breakdown

Area	Specifics	$NZD M
Owners costs		4.4
Land and landowner costs		7.2
Site Infrastructure	Construction camp	9.9
	Roads	13.9
	Power	12.8
	Service water	5.7
	Site Facilities	23.3
Establishment earthworks	Diversion drains	1.0
	Sediment ponds	1.7
	Haul roads	10.5
	ELF stripping	2.8
	Total	**93.2**

ES 13.5.2 Process plant

Total process plant costs including preparing the Shepherds valley earthworks is estimated at $135.2M. With further a further breakdown:

- Shepherds valley earthworks to strip unsuitables, train the creek to the northern bank, drill and blast in TZ4 to get hard rock foundations for the crusher and milling circuit and building back up the valley with engineered fill - $5.4M
- The initial 1.5Mtpa process plant build as an EPCM (with minimum disruption later expansion to 2.1Mtpa) - $128.9M
- Fit out of the laboratory - $0.9M

Table ES 31: Further detailed breakdown of the 1.5Mtpa plant EPCM cost

Description	$NZD M
Contractor Indirects	3.4
Preliminary & General	0.3
First Fills & Initial Consumables	1.8
Equipment Spares	0.5
EPCM	18.2
Commissioning	1.5
Crushing	15.5
Milling & Classification	23.6
Pebble Crushing & Conveying	1.4
Leach & Adsorption	13.9
Gold Recovery	7.8
Reagents	2.4
Water Services	2.0
Piping	7.6
E&I	17.0
Construction Overheads	3.6
Plant & Equipment	6.2
Plant Buildings	2.4
Total	**128.9**

The process plant expansion to 2.1Mtpa is $30.3M

ES 13.5.3 TSF

Total TSF construction and closure costs are $58.1M excluding bulk zone fill materials which are included in the mining costs.

Table ES 32: Full TSF capital costs

Description	$NZD M
TSF - Starter	9.5
TSF - Raises	33.5
TSF - Closure	15.1
Total	**58.1**

ES 13.5.4 Underground

The capital underground costs are $82.2M, not including capitalised operating costs. The significant cost contributors are the mining fleet including light vehicles ($35.3M) and the paste plant and associated paste infrastructure ($33.1M).

Table ES 33: Underground capital

Description	$NZD M
Mining Fleet	35.3
Ventilation	1.3
Dewatering Pumps	1.1
Paste plant and associated infrastructure	33.1
Mine support Infrastructure	8.5
Safety	1.7
Miscellaneous	1.1
Total	**82.2**

ES 13.5.7 Other

Other specific capital costs are for:

- Water treatment at closure when the site will need to start releasing water ($8.3M);
- Light vehicles not captured elsewhere;
- Provision for ecological offsets; and
- General closure rehabilitation costs not already capture by the TSF or water treatment

Table ES 34: Other capital costs

Description	$NZD M
Water treatment	8.3
Light vehicles	0.6
Ecological offsets	3.6
Closure (plant-site, infrastructure removal, general)	35.0

ES 14. Financial evaluation

The Financial Evaluation was prepared on the following basis:

- A real discount rate of 8% was applied. This was based on an internal weighted average cost of capital calculation and a peer analysis of comparative projects.
- All estimated costs are nominal (not adjusted for inflation).
- A corporate tax rate of 28% has been applied, without allowances for New Zealand tax losses.
- All pre-production capital has been capitalised up until the point of commercial production.
- Conceptual mine closure costs have been netted to zero with provisional project salvage values.

Key financial outputs from the study are as shown in Table ES 35 and Table ES 36.

Table ES 35: Key Project Metrics

Description	Unit	Base Case Scenario
Key Project Mining Physical Targets and Assumptions		
Mine Life	Years	9.17
Plant Throughput	ktpa	1,835
Open Pit Ore Mined	kt	14,404
Open Pit Mill Feed	kt	14,404
Open Pit Mill Feed Grade	Au g/t	2.19
Open Pit Contained Gold	koz	1,014
Open Pit Recovered Ounces	koz	935
Underground Ore Mined	kt	2,413
Underground Mill Feed	kt	2,413
Underground Mill Feed Grade	Au g/t	2.99
Underground Contained Gold	koz	232
Underground Recovered Ounces	koz	215
Total Ore Mined	kt	16,817
Total Mill Feed	kt	16,817
Au Grade - Mined	g/t	2.30
Total Contained Gold	koz	1,245
Overall Plant Recovery	%	92.38%
Gold Production	koz	1,151

Modelling of the government or Crown royalty modelling has resulted in the 10% accounting profits being applied which is higher than the 2% ad valorem.

Table ES 36: Key Financial Metrics

Key Financial Assumptions		Base Case AUD	Spot AUD	Spot NZD	Spot USD
Gold Price	$/oz	2,894	4,000	4,406	2,626
Exchange Rate	USD:$	0.66	0.66	0.60	1.00
Key Project Metrics					
Gold Produced	Oz	1.15 million			
Initial Mine Life	Years)	9.17			
Gold Revenue	'000	**3,330,018**	**4,602,435**	**5,069,319**	**3,021,314**
Open Pit Mining Cost	'000	619,237	619,237	682,054	406,504
Underground Mining Cost	'000	152,747	152,747	168,242	100,272
Processing Costs	'000	288,943	288,943	318,254	189,679
General and Admin Costs	'000	55,633	55,633	61,276	36,521
Selling Cost	'000	8,357	8,357	9,204	5,486
Royalties - Govt	'000	170,173	296,305	326,363	194,512
Royalties - Others	'000	89,636	123,887	136,454	81,327
Total Cash Operating Cost	'000	**1,384,725**	**1,545,108**	**1,701,848**	**1,014,301**
Total Cash Operating Cost per Ounce	$/oz	**1,203**	**1,343**	**1,479**	**881**
Project EBITDA	'000	1,945,292	3,057,327	3,367,471	2,007,013
Depreciation and Amortisation (exc Rehab PPE)	'000	546,067	546,067	601,462	358,471
Total Production Cost (incl. all CAPEX)	'000	**1,930,793**	**2,091,175**	**2,303,310**	**1,372,773**
Total Production Cost per Ounce	$/oz	**1,678**	**1,818**	**2,001**	**1,193**
Net Profit Before Tax (NPBT)	'000	1,399,225	2,511,260	2,766,009	1,648,541
Tax Payable (28.0%)	'000	(424,010)	(728,094)	(801,954)	(477,965)
After Tax Profit	'000	**975,215**	**1,783,166**	**1,964,055**	**1,170,577**
Capital					
Initial Development Capex (inc. OP & Capitalised Opex)	'000	340,609	340,609	375,161	223,596
Underground Initial Development Capex	'000	121,795	121,795	134,151	79,954
Sustaining Capex	'000	83,663	83,663	92,151	54,922
Closure Capex (see note 1)	'000	-	-	-	-
Total CAPEX over Mine Life	'000	**546,067**	**546,067**	**601,462**	**358,471**
DCF Outcomes					
Initial NPV (unleveraged and after-tax) @8.00%	'000	534,975	1,058,104	1,165,441	694,603
IRR	%	41.66%	68.23%	68.23%	68.23%
Payback Period from production start (unleveraged and after-tax)	years	1.67 Yr(s)	0.92 Yr(s	0.92 Yr(s)	0.92 Yr(s)

Note 1: Conceptual mine closure costs netted to zero against mine salvage value.

Summary operating costs are shown in Table ES 37.

Table ES 37: Summary operating costs

Operating Costs (Production)	NZD '000	NZD /t Milled	NZD /oz Produced
Mining Cost	850,296	50.6	739
Processing Costs	318,254	18.9	277
General and Admin Costs	61,276	3.6	53
Selling Cost	9,204	0.5	8
Royalties - Govt	326,363	19.4	284
Royalties - Others	136,454	8.1	119
Total Cash Operating Cost	**1,701,848**	**101.2**	**1,479**
Initial Development Capex (inc. OP & Capitalised Opex)	375,161	22.3	326
Underground Initial Development Capex	134,151	8.0	117
Sustaining Capex	92,151	5.5	80
Closure Capex	-	-	-
Total Production Cost (inc Closure)	**2,303,310**	**137.0**	**2,002**

Summary capital costs are shown in Table ES 38.

Table ES 38: Capital cost summary

Capital Cost Requirement Estimates		
Pre-Production Capital: Open Pit & Initial Dev		
Owners Team	NZD '000	3,829
Land	NZD '000	7,225
Infrastructure	NZD '000	56,115
Site Establishment	NZD '000	11,633
TSF	NZD '000	9,473
Process plant	NZD '000	135,218
Open Pit	NZD '000	2,182
Others	NZD '000	562
Capitalised OpEx - Open Pit Mining	NZD '000	146,339
Capitalised OpEx - G&A	NZD '000	2,585
Total Initial OP Pre-Production Cost Estimates	NZD '000	**375,161**
Underground Infrastructure	NZD '000	78,007
Capitalised OpEx - Underground Mining	NZD '000	56,144
Total UG Pre-Production Cost Estimates	NZD '000	**134,151**
Sustaining & Closure Capital – Life of Mine Estimates		
Underground	NZD '000	4,194
Open Pit	NZD '000	87,956
Closure Capex	NZD '000	-
Total Sustaining & Closure Capital	NZD '000	**92,151**
Total Capital Cost	NZD '000	**601,462**

Figure ES 29 below, shows the undiscounted cashflows on an annual basis at the spot gold price scenario (NZ$4,406/oz).

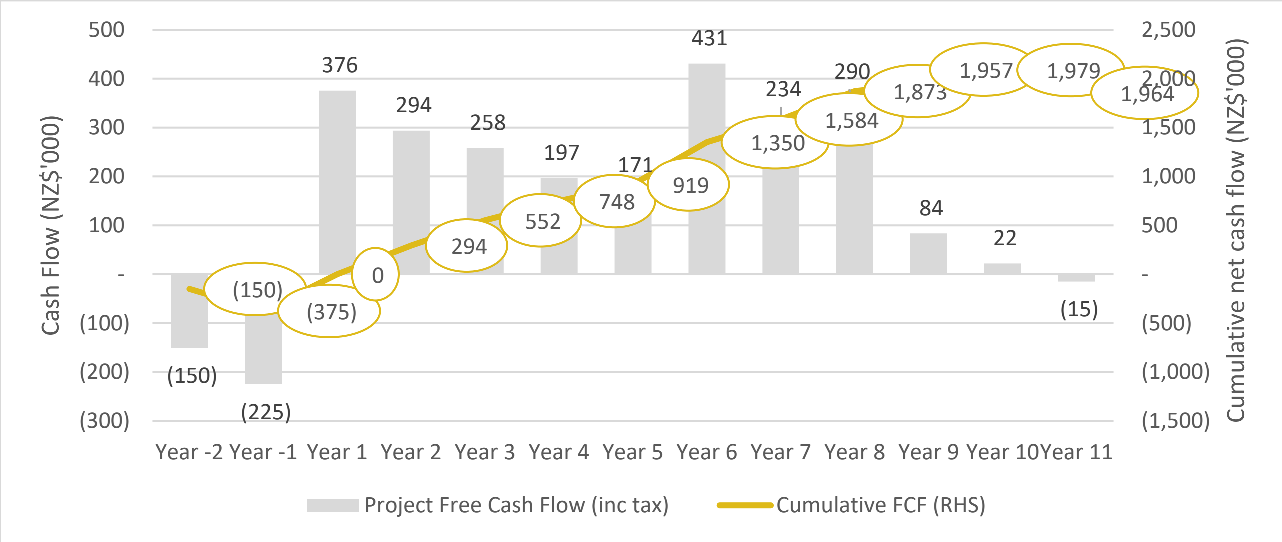


Figure ES 29: Project Free cash flow (inc Tax, NZD'000)

Sensitivities for gold price, capital and operating costs, discount rate and metallurgical recoveries at relevant ranges are shown in Figure ES 30 below.

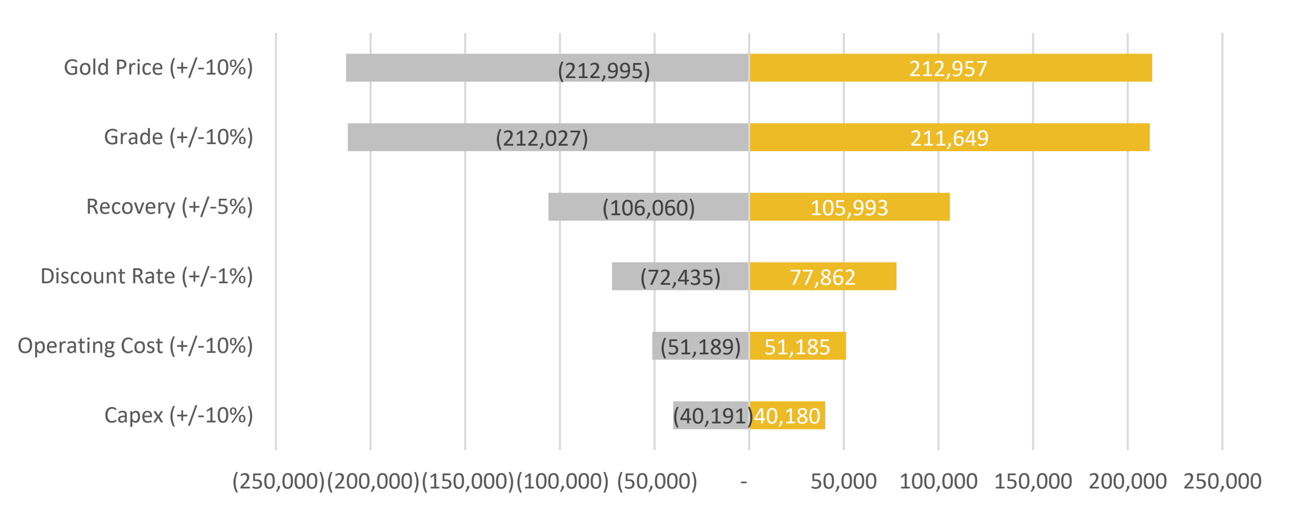


Figure ES 30: Project NPV Sensitivities at spot gold price of NZ$4,406/oz (NZD'000) at 8% Post-Tax, Real Discount Rate

Additional sensitivity analysis was undertaken at the base case gold price of NZ$3,204/oz and is shown in below.

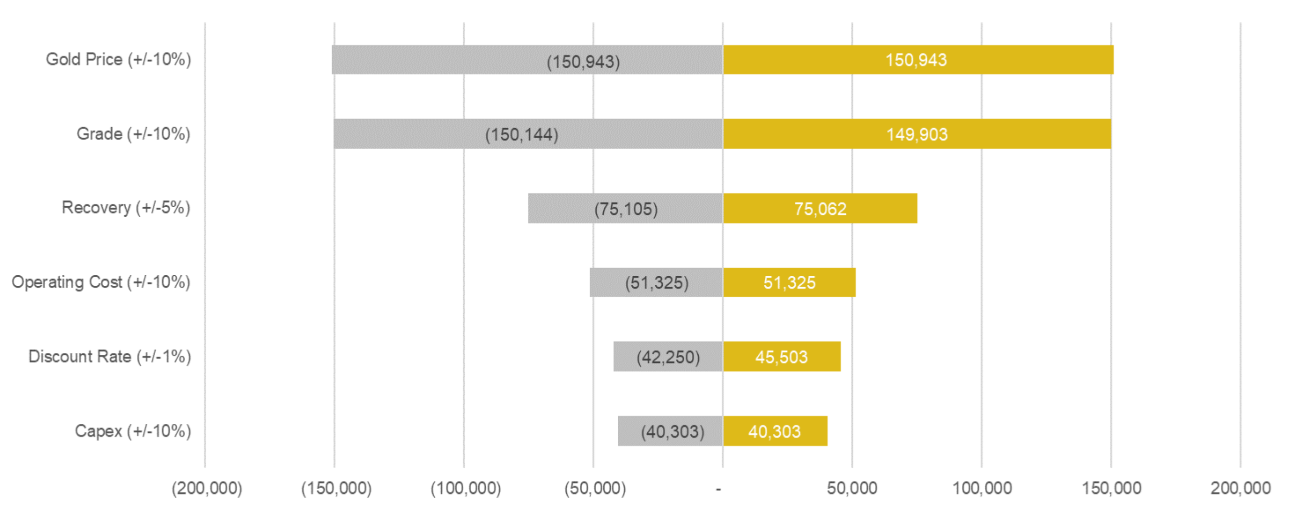


Figure ES 30A: Project NPV Sensitivities at base case of NZ$3,204/oz ($NZD'000) at 8% Post-Tax, Real Discount Rate

Figure ES 31 shows the overall annual gold production profile and AISC at the spot gold price scenario (NZ$4,406/oz) for the primary production years.

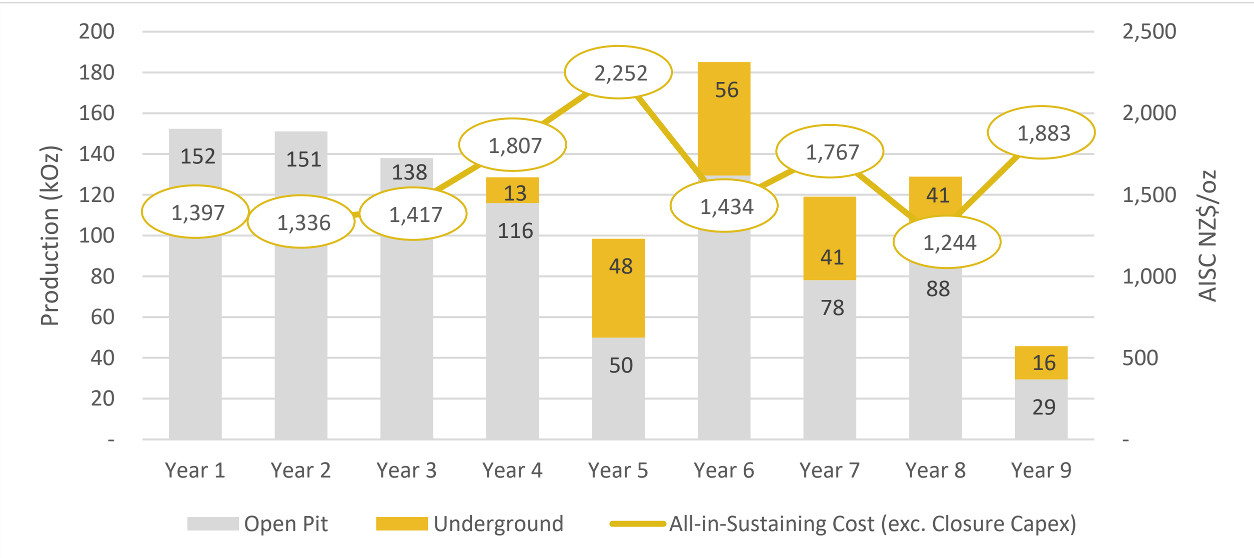


Figure ES 31: Production Profile OP/UG w/AISC at Spot Gold Price (NZ$4,406/oz)

ES 15. Environment

A comprehensive set of baseline studies have been commissioned to understand the existing environment across the project area and surrounding landscape. As the project description has developed the assessment of effects on the environment has also progressed along with associated considerations of opportunities to address potential negative effects as far as practical.

Environment related studies include ecology, waterways and wetlands, ground and surface water, geochemistry, noise, air quality, heritage, closure, visual effects, recreation and traffic.

The project is located in the 90,000 ha Dunstan Ecological District. It is dominated by a mosaic of native and exotic scrub, native tussockland, mixed depleted herbfields and exotic pasture. The Ecological Study Area (ESA) for the project covers approximately 5,000 ha. The Project Footprint (PF) is defined as the area of direct impact of approximately 550 ha. Adjacent to these areas is land administered by the Department of Conservation (DOC) including the Bendigo Historic Reserve (459ha) to the west which is contiguous with Bendigo Scenic Reserve (628ha) further to the west; the Bendigo Conservation Area (1,973ha) to the southwest; and the Ardgour Conservation Area (303ha) to the east which is contiguous with the Neinei i kura Conservation Area (1,643ha) to the north. Thus, a total land package of 5,006 ha is under DOC management adjacent to the project.

A conservation covenant was established over Bendigo Station when it passed from leasehold into freehold and the DOC areas were established. The conservation objectives broadly seek to protect the natural character of the land and the ecological character and maintain the landscape and historic values. The covenant also provides for prospecting or mining of minerals with the Minister of Conservation’s approval having regard for the objectives. Matakanui Gold holds the necessary approvals from the Minister to undertake exploration.

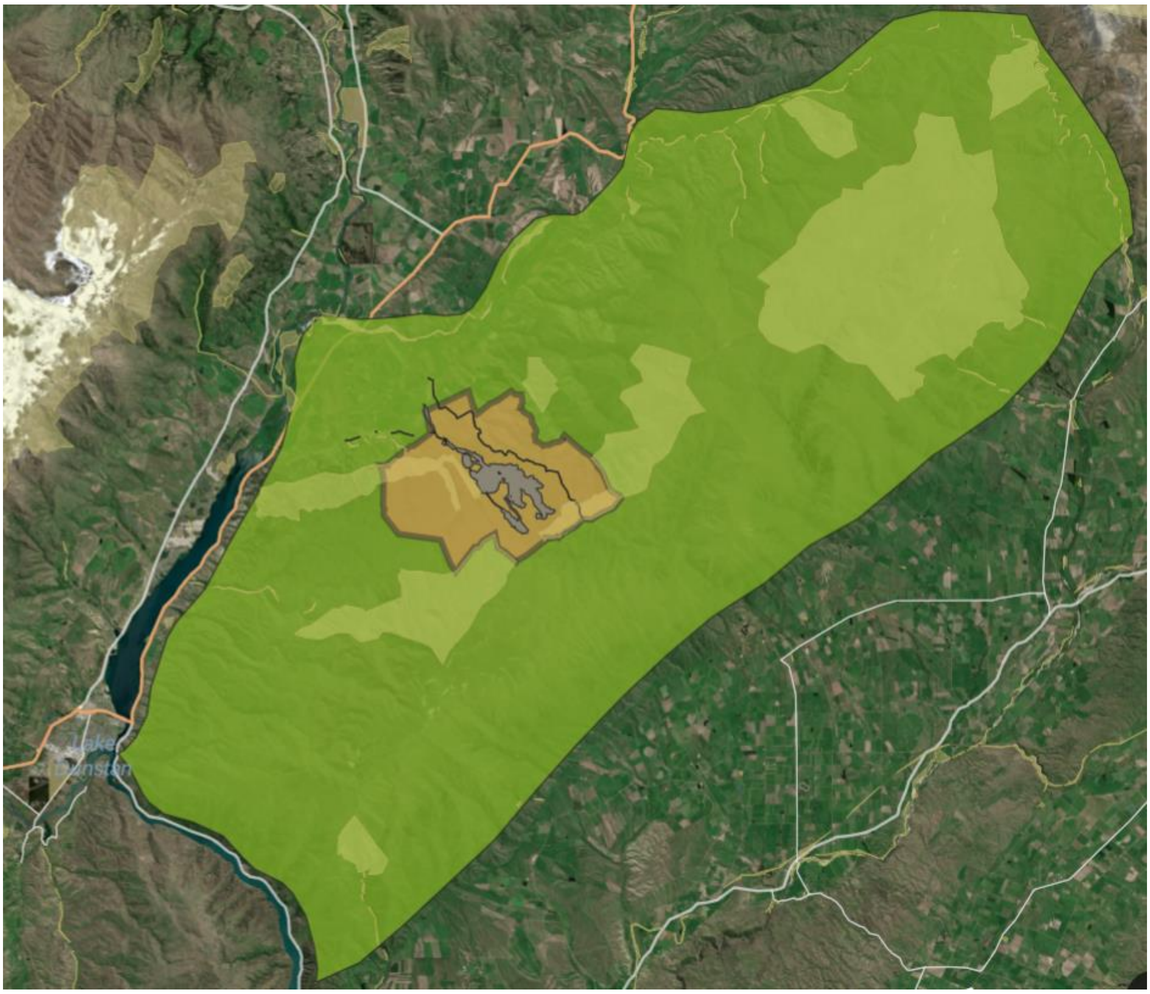


Figure ES 32: Project Footprint, Ecological Study Area, Dunstan Ecological District and DOC administered conservation areas

The BOGP is located across two freehold working pastoral stations. The indigenous biodiversity in the ESA is heavily modified by past and current land use practices, along with the ongoing effects of invasive weeds, introduced mammalian predators, and browsers.

Despite the past and current land use practices the PF contains some areas of high terrestrial value based on the presence of native-dominated habitats and nationally 'threatened' or 'at-risk' flora and fauna.

Fifty 'threatened' or 'at-risk' flora and fauna species (including 16 invertebrates not listed or assessed) have been identified in the ESA to date. Further surveys to confirm the extent of these species and search for additional flora species will be undertaken in spring 2024.

Freshwater ecological values of the area are associated with the overall stream habitat and the macro-invertebrates present. No freshwater fish values were identified within Shepherds nor Rise and Shine Creeks and macroinvertebrate communities exhibited degradation in areas but were otherwise considered fair.

A mammalian pest survey of the ESA confirmed the presence of 10 mammalian pests including deer, pigs, rabbits, hares, ferrets, cats, rats, mice, goats and possums.

Potential effects on the ecology have been identified. Opportunities to avoid and mitigate negative effects are an integral part of the project design process.

Lengths of waterways and associated streams in Shepherds Creek will be covered by the tailings storage facility (TSF) and engineered landform (ELF) or realigned to accommodate the process plant footprint. Loss of waterways will be mitigated by construction of freshwater diversion drainage channels in the Shepherds valley at or just above the final height of the TSF and ELF. The channels will be designed to replicate similar flow characteristics to mountain creeks and streams as far as practical to promote naturalisation over the project life and the return of native vegetation and invertebrates. Seeding of native species and management of grazing animals will facilitate naturalisation.

An ecological effects management strategy has been developed which includes rehabilitation for ecological and pastoral (grazing) outcomes on direct disturbed areas. To offset or compensate for residual effects that cannot be avoided, minimised or rehabilitated a combination of broad scale and targeted habitat restoration and enhancement measures will be employed to deliver improved biodiversity management and uplift.

This approach aims to ensure that at a broad level biodiversity benefits outweigh impacts in general accordance with regional and national policy requirements. Offsetting and compensation for residual ecological effects are being considered via land management programs.

The Central Otago District Plan identifies the Dunstan Mountains as an Outstanding Natural Landscape (ONL). An ONL requires protection from inappropriate subdivision, use and development as a matter of national importance in accordance with Section 6(b) of the Resource Management Act. Landscape and visual effects of the operation are being assessed by experienced landscape architects considering the physical, perceptual and associative values of the landscape. Project design work aims to balance the overall disturbance effects across the landscape values, including ecological, historical and visual values. Potential visual effects are substantially reduced by locating much of the disturbance within the confines of the Shepherds and Rise and Shine Valleys, of which there is limited visibility from key vantage points in the Clutha / Mata-Au Valley.

The project area has been surveyed for archaeological sites via pedestrian survey, LiDAR imagery, historical photographs and documents and previous archaeological investigations to establish detailed maps of archaeological sites. The sites have been further evaluated to understand their archaeological values. Thirty-three archaeological sites have been mapped in the project footprint. The majority of the sites are associated with the intensive historic alluvial, elluvial and hard rock mining in the survey area. These sites include footings of a stamper battery, tailing areas. water races, huts, dams, culverts, and drains. The predominant archaeological features are remains of 10 hut sites, of which the majority are associated with mining activity although several may have also been used by musterers. The heritage structures within the Rise and Shine Valley are mostly ruinous and at least partially obscured by topography and/or vegetation, with the exception of the small dam in Rise and Shine valley. Whereas the wider Bendigo historic areas exhibit residual and visible huts and other buildings which are much more intact and the mining features are easily identified by the public. No Māori archaeological sites have been identified in the project area.

Conservative noise modelling has shown that district council noise limits can be readily achieved. The noise model will be refined as more information comes to hand during detailed design.

Lighting design aims to minimise night glow and light spill while providing for safe operation of the plant. The location of the process plant and infrastructure in Shepherds valley greatly reduces the potential effects of light pollution.

Air quality effects assessment is in progress and indicates that nuisance dust can be managed within the project footprint.

ES 16. Geochemical

The BOGP gold deposit is located within the Otago Schist and is associated with the mineralised Rise and Shine Shear Zone (RSSZ) which juxtapose lower greenschist facies Textural Zone 3 (TZ3) and mid to upper greenschist facies Textural Zone 4 (TZ4) schists in their hanging walls and footwalls respectively. This mineralisation is dominated by elevated sulphate (SO_4) and arsenic (As) (e.g., the mineral arsenopyrite) with other trace metals also being potentially elevated but at much lower concentrations (e.g., cobalt, (Co), copper (Cu), chromium (Cr), antimony (Sb), and zinc (Zn)).

The outcropping mineralisation associated with the BOGP area has contributed to baseline water quality being elevated in some metals, which has been exacerbated by historical legacy mining activities, leading to streams in the project area containing slightly higher contaminants of potential concern (COPC) that include for instance, As, cadmium (Cd), Cu, Cr, manganese (Mn), and Zn leaching at faster rates from exposed mineralisation.

Some soils within the Rise and Shine Creek catchment are elevated in arsenic due to natural outcropping mineralisation and historic mining spoils. These soils will require management to avoid adverse effects if they are disturbed (e.g., dust management, stockpile management).

Studies indicate that the rocks associated with the project (TZ3, TZ4, and RSSZ) will not generate acid rock drainage with >350 samples tested by industry accepted acid base accounting (ABA) techniques (e.g., AMIRA, 2002). This is a function of the high acid neutralisation capacity (ANC) of the rocks associated with carbonate minerals (e.g., dolomite) and a low sulfide mineral content (e.g., arsenopyrite, pyrite) that can generate lesser acidity. The overall ABA assessment indicates that the rocks are classified as non-acid forming (NAF).

Data for waste rock indicates that the TZ4 and RSSZ lithologies contain ~97.7% of arsenic and 37.2% of sulfur yet represent only 18% of the waste rock that will be disturbed. Hence, appropriate management of waste rock to reduce sulfide mineral oxidation and the release of arsenic is a critical step to minimise any potential deleterious effects of mining, i.e., manage 18% of the waste rock well to mitigate 97.7% of the arsenic risk in the Engineered Landform ELF) that will contain the waste rock.

Nitrogenous compounds such as nitrate are also expected to be elevated in seepage from blasted rock due to the use of ANFO, an ammonium-nitrate fuel oil explosive. This is not an uncommon problem in the mining industry.

It is expected that mining of the BOGP will affect waters within the project area and these effects will include:

- Elevated total suspended solids (TSS)
- Neutral metalliferous drainage (NMD) with elevated sulfate and the certain COPCs such as As, Fe, and potentially lesser amounts of trace metals such as Co, Cu, Cr, Sb, Zn; and
- Nitrate-rich drainage due to the use of ANFO.

Collectively these waters are referred to as mine impacted water (MIW) to acknowledge the different contributions to poor water quality within the project area. The management of MIW will involve several engineering controls to minimise the effects on the downstream environment. These engineering controls have been accounted for in the mine plan, including:

- Materials management and the construction of an Engineered Landform (ELF) to minimise contaminant loads from the waste rock; and
- Water management and treatment as necessary.

ES 17. Community and iwi

The Bendigo-Ophir Gold Project (BOGP) is located on private farmland in the Central Otago region, with the closest towns being Tarras and Cromwell. The project is likely to be resourced by the townships of Cromwell, Tarras, Alexandra, Queenstown, Wanaka, Clyde and Omakau. Access agreements are in place with both of the station landowners that the project covers, Bendigo Station and Ardgour Station.

Engagement to date has included regular drop-in sessions in both the Tarras and Cromwell communities, attendance at community events, community sponsorships, presentations to community and business groups, site visits, a monthly newsletter, and project information on Santana Minerals website.

A cultural impact assessment is being undertaken to understand the potential impact on mana whenua cultural values as identified by iwi in their 2018 statement.

A Social Impact Assessment study is being carried out to assess the social consequences and community concerns, interviews are currently underway.

ES 18. Permitting

ES 18.1 Mining Permits

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.

ES 18.2 Resource Consents

Environmental approvals are predominantly 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.

Resource consent and building consent are required under New Zealand legislation to construct the TSF and large dams. This will include the Shepherds Creek Silt Pond. Only resource consent is required to construct the ELFs. Resource consents relating to environmental effects will be applied for as part of the wider project application. Building consents for the TSF and Shepherd Creek Silt Pond will be separately applied for through the building consent authority for dams in the area.

The NZ Government has introduced a new legislation, Fast-track Approvals Bill (FAB). This Bill provides a streamlined decision-making process to facilitate the delivery of infrastructure and development projects with significant regional or national benefits. On 04/10/2024 it was announced that the Santana Minerals Bendigo-Ophir gold project is included within the list of projects eligible to access the fast-track consenting framework under the proposed FAB.

The Bill is intended to be a "one stop shop" for consenting projects which would otherwise require consents under multiple different regimes including resource consents under the RMA, concessions under the Conservation Act 1987, wildlife permits under the Wildlife Act 1953, archaeological authorities under the Heritage Pouhere Taonga Act 2014, and land access provisions of the Crown Minerals Act 1991. The "one stop shop" approach marks a significant change in project approvals in New Zealand, and it is hoped that this will significantly reduce consenting costs, uncertainty, and timeframes.

ES 18.3 Application

Santana Minerals NZ subsidiary Matakanui Gold Ltd is preparing a substantive application for the Bendigo-Ophir project. The strategy is for the application to include all the necessary information, from leading technical experts to a high standard to enable robust decision making. Planned lodgement is Q1 2025.

ES 19. Closure

A conceptual Mine Closure Plan is under development to be submitted as part of the consent application for the project. The overarching objective of the mine closure plan is a final landform and site that is safe, stable, and non-polluting at a standard that is acceptable to stakeholders and regulators. Technical experts have been engaged to develop the conceptual Plan.

The conceptual closure plan will consider the following:

- Knowledge base
- Legal obligations
- Closure risks
- Post mining land uses
- Closure objectives and completion criteria, and
- Community consultation.

The development of the plan will bring together knowledge from a range of experts considering:

- Geochemistry
- Geotechnical
- Mine planning and schedule, including final landform designs
- Ecology
- Rehabilitation
- Visual effects, and
- Stakeholder requirements.

ES 20. Project Implementation

The Project implementation critical path timeline, from receipt of consent and financing, is two months for mobilisation and site establishment, four months pioneering work to establish at RAS haul roads, ELF and first mine benches, then sixteen months of bulk-stripping of 39.5Mt through to commencement of processing.

In October 2024 the BOGP was designated as a ‘Fast-track Project’ under Schedule 2 of the NZ Government's Fast-track Approvals Bill (FAB). The Bill is anticipated to be enacted by year-end.

An Assessment of Environmental Effects (AEE) is currently in progress, as per the current requirements under the Resource Management Act. The designation of the BOGP as a fast-track project does not change the specifics or level of detail being prepared. It is anticipated the application will be lodged for the required consenting documentation into the FAB by the end of February 2025.

Whilst the draft FAB legislation allows for decisions within a shorter timeframe, the pathway is yet to be utilised, and timeframes are still relatively unknown. As a result, the company expects up to six months for approval.

ES 21. Risks and Opportunities

Opportunities are presented by:

- Adding resources to the production schedule through RAS downdip (Inferred extension).
- Adding production schedule ounces through further gold price escalation.
- Ongoing exploration to increase resource and reserve base.
- Improved understanding of geological domains leading to conversion of further resource to reserves.
- Conversion of inferred resources to indicated at CIT and generation of viable mine plan.
- Existing opportunities within Central Otago to meet short-term accommodation needs thus avoiding the need for or the reducing the size of the construction camp.

As part of the PFS, a specific risk assessment has been undertaken across all disciplines covered in this report. From this work, 108 risks were identified, then assessed and ranked, to allow further targeted investigations. These risks were grouped as either High, Significant, Moderate or Low.

The risks were further categorised as to what stage of the project they related to, these groups being:

- Design,
- Consent,
- Construction, and
- Operations.

And the responsible departments of:

- Project Engineering,
- Implementation Manager,
- Operational Readiness Manager,
- Environment Manager, and
- Corporate.

Generally, the main grouping of risks related to:

- The Fast-track Approval Bill is yet to be passed into law so the final form of the Act may need further consideration,
- The project start-up timeframe,
    - Ability to mobilise the main mining fleet in a timely manner; and
    - Ability to connect the high voltage supply to the site.
- The ramp up of pre-strip volume requirements in the pre-strip (critical to first positive cashflows); and
- Ability to secure financing.

ES22. Funding

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

- The PFS shows strong economics associated with Project development, including a very strong return on capital and robust cashflows, even at a base case gold price of US$1,900/oz. 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 9-year mine life generating significant free cash flow relative to the development capital required.
- The PFS 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 high 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.

JORC Code, 2012 Edition – Table 1

Section 1: Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	*Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* *Aspects of the determination of mineralisation that are Material to the Public Report.* *In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	This Mineral Resource Estimate (MRE) is estimated from drilling samples collected by reverse circulation and diamond drilling. 'Blasthole', surface trench and underground channel samples were used as an aid for geological interpretation and domaining but not for grade estimation. Diamond drill (DD) core samples for laboratory assay are typically 1 metre samples of diamond saw cut ½ diameter core. In the rare cases where the core was friable or unconsolidated, the sample was collected from one side of the core using a scoop. Where distinct mineralisation boundaries are logged, sample lengths are adjusted to the respective geological contact. RC samples were sub-sampled at 1.0 m intervals using either a riffle splitter or a cone splitter mounted below the cyclone. The splitter produced 2 x 12.5% splits and 1 x 75% split. The two 12.5% splits were used as primary sample and field duplicate (if submitted) with the 75% split used for logging and then stored at the MGL core yard. Samples are crushed at the receiving laboratory to minus 2mm (85% passing) and split using a rotary splitter to provide 1kg for pulverising in a ring mill to -75um. Pulps are fire assayed (FAA) using a 50g charge with AAS finish. Prior to 2019 only 200g of the crushed material was pulverised. 877 samples were assayed this way. Certified standards, blanks and field replicates are inserted with the original batches at a frequency of ~5% each for QAQC purposes. All pulps and crush reject (CREJ) are returned from the laboratory to MGL for storage on site. Of these returned samples, a further ~5% are re-submitted as QC check samples which involve pulp FAA re-assays by the original and an umpire laboratory and CREJ re-assayed by 500-gram (+ & -75mu) screen fire assay (SFA), 1kg BLEG (LeachWELL) and 2*500-gram Photon analysis (PHA) for gold. Where multiple assays exist for a single sample interval, larger samples are ranked in the database: PHA > BLEG > SFA > FAA. All returned pulps are analysed for a suite of 31 elements by portable XRF (pXRF). The sampling, sub-sampling and assaying methods are appropriate to the geology and

Criteria	JORC Code explanation	Commentary
		mineralization being reported.
Drilling techniques	*Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	Diamond (DD) and reverse circulation (RC) drilling has been used to inform the MREs being reported here. All diamond coring was PQ3 size triple tube for holes MDD001 to MDD016. The DD coring in since MDD016 has all been HQ3 size triple tube. Where PQ3 core size (83mm diameter) is commenced this is maintained throughout the DD hole until drilling conditions dictate reduction in size to HQ3 core (61mm diameter). DD pre-collars are drilled open hole through un-mineralised TZ3 schist to within about 15 m of the mineralisation hangingwall at which point diamond coring commences. RC drilling was only carried out where the mineralisation target was less than about 150m downhole and used a face sample bit with sample collected in a cyclone mounted over a riffle or cone splitter producing 2 x 12.5% splits and 1 x 75% split. The two 12.5% splits were used as primary sample and field duplicate (if submitted) with the 75% split used for logging and then stored at the MGL core yard. Drillholes are oriented to intersect known mineralised features in a nominally perpendicular orientation as much as is practicable. A small number of holes are oriented in other directions to resolve areas of ambiguous geological interpretation. All drill core is oriented to assist with interpretation of mineralisation and structure using a Trucore orientation tool.
Drill sample recovery	*Method of recording and assessing core and chip sample recoveries and results assessed.* *Measures taken to maximise sample recovery and ensure representative nature of the samples.* *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	DD core sample recoveries are recorded by the drillers at the time of drilling by measuring the actual distance of the drill run against the actual core recovered. The measurements are checked by the site geologist. DD core recovery averages 94.2% within the gold estimation domains. When poor core recoveries are recorded the site geologist and driller endeavour to immediately rectify any problems to maintain maximum core recoveries. DD core logging to date indicate ~97% recoveries. RC sample recovery is visually estimated and averages 96.5%. All RC samples logged as wet were omitted from use in this MRE. Of the RC samples used in these MREs, 94.7% were logged as dry and 4.9% logged as moist. Sample grades were plotted against drilling recovery by drilling method and no relationship was

Criteria	JORC Code explanation	Commentary
		established. Wet RC samples do show higher grades than dry RC samples. This may be due to wet RC samples coming from higher grade zones or sampling bias due to the loss of fines in wet samples. Whatever the cause, this bias was the reason that wet RC samples were omitted from use in this MRE.
Logging	*Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* *The total length and percentage of the relevant intersections logged.*	All DD holes have been logged for their entire length below upper open hole drilling (nominally 0-450 metres below collar). Data is recorded directly into AcQuire database with sufficient detail that supports Mineral Resource estimations (MRE). Logging is mostly qualitative but there are estimations of quartz and sulphide content and quantitative records of geological / structural unit, oxidation state and water table boundaries. Oriented DD core allows alpha / beta measurements to determine structural element detail (dip / dip direction) to supplement routine recording of lithologies / alteration / mineralisation / structure / oxidation / colour and other features for MRE reporting, geotechnical and metallurgical studies. All RC chips were sieved and logged for lithology, colour, oxidation, weathering, vein percentage and sulphide minerals. All core is photographed wet and dry before cutting. Sieved RC chips are also photographed. 100% of all relevant (within the gold grade domains) intersections were logged. The logging is of sufficient quality and detail for resource estimation.
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.* *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* *Measures taken to ensure that the sampling is representative*	DD core drill samples are sawn in ½ along the length of the core on cut lines marked by geologists' perpendicular to structure / foliation or to bisect vein mineralisation for representative samples whilst preserving the orientation line. One half is dispatched to the laboratory for assay and the other half retained in core trays at MGL's core storage facility. Intervals required for QAQC checks are nominated by geologists and the crushed sample being split by the laboratory with the two replicated samples then assayed. QA procedures used to maximise the representivity of sub-samples include the use of a riffle splitter on the RC rig and cutting DD core perpendicular to the regional foliation. QC procedures to assess the representivity of sub-sampling include field duplicates, pulp duplicates, standards, and blanks at a frequency of ~5%. In addition approximately 5% of the mineralised samples are

Criteria	JORC Code explanation	Commentary
	of the in situ material collected, including for instance results for field duplicate/second-half sampling. *Whether sample sizes are appropriate to the grain size of the material being sampled.*	periodically re-submitted to the primary laboratory and umpire laboratory for re-assay by fire assay (50g), screen fire assay (200g), BLEG (LeachWELL, 1000g) and photon assay (500g). The larger re-assay methods provide a check on sub-sampling at the laboratory. The mass proportion of every 10th sample passing 75um is reported by the laboratory and monitored to ensure sample preparation quality. Calculations based on Pitard (1993) show that sub-sample masses are appropriate to gold particle size and grade, if the size and shape of the gold particles are reduced in the ring mill in a similar way to the gangue particles.
Quality of assay data and laboratory tests	*The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	FA, BLEG, SFA and PHA are all total gold assays and are appropriate to the RSSZ mineralization. DD core and RC chip samples for gold assays undergo sample preparation by SGS laboratory Westport. Sample preparation involves drying and crushing of the entire sample to 2 mm followed by milling of a 1000g sub-sample to 75um. The sample is then sent to SGS laboratory Waihi where a 50 g sub-sample is assayed by fire assay with an AAS finish (SGS method FAA505 DDL 0.01ppm Au or FAD505 DDL 1ppm Au & FAD52V DDL 500ppm Au). Other SGS laboratories at Macraes and Townsville and the ALS laboratory in Townsville, are used from time to time and follow the same processes. Prior to 2019 the 75um sub-sample was only 200g. For laboratory QAQC, samples (certified standards, blanks and field replicates) are inserted into each laboratory batch at a frequency of ~5% respectively. A selection of 5% of retained lab pulps across a range of grades are sent for re-assay and to an umpire laboratory for cross-lab check assays. Portable XRF (pXRF) instrumentation is used onsite (Olympus Innov-X Delta Professional Series model DPO-4000 equipped with a 4 W 40kV X-Ray tube) primarily to identify arsenical samples (arsenic correlates well with gold grade in these orogenic deposits). The pXRF analyses a 31-element suite (Ag, As, Bi, Ca, Cd, Cl, Co, Cr, Cu, Fe, Hg, K, Mn, Mo, Nb, Ni, P, Pb, Rb, S, Sb, Se, Sn, Sr, Th, Ti, V, W, Y, Zn, Zr) utilising 3 beam Soil mode, each beam set for 30 secs (90 secs total). pXRF QAQC checks involve regular calibration (every 20 samples) and QAQC analyses of SiO_2 blank, NIST standards (NIST 2710a & NIST 2711a), & OREAS standards. pXRF QAQC checks involve regular calibration (every 20 samples) and QAQC analyses of SiO_2 blank, NIST standards (NIST 2710a & NIST 2711a), & OREAS standards.

Criteria	JORC Code explanation	Commentary
		No geophysical tools have been used in this MRE.
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel.* *The use of twinned holes.* *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* *Discuss any adjustment to assay data.*	Significant gold assays and pXRF arsenic analyses are checked by alternative senior company personnel. Original lab assays are initially reported and where replicate assays and other QAQC work require re-assay or screen fire assays, the larger sample results are adopted. To date results are accurate and fit well with the mineralisation model. Twinned data is available where DD core holes have been sited adjacent to previous RC drillholes and where DD redrills have occurred. pXRF multi-element analyses are directly downloaded from the pXRF analyser as csv electronic files. These and laboratory assay csv files are imported into the database, appended and merged with previous data. Since October 2022 all logging has been directly entered into the Acquire database using tablets. All collar surveys, downhole surveys and assay results are provided digitally and directly imported into the database. On import into the database validation checks are made for: interval overlaps, gaps, duplicate holes, duplicate samples and out of range values. The AcQuire database is stored on a cloud server and is regularly backed up, updated and verified by an independent qualified person. The only adjustment made to the data on import to the database is to convert below detection results to negative the detection limit. Samples with multiple Au results are ranked by assay method (SFA > FA > other) and on export only the highest ranked method is exported. Prior to import into Minesight software for resource estimation the data is further validated as above plus checks on the highest and lowest values. Negative below detection results are converted to half the detection limit on import into Minesight.
Location of data points	*Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in* *Mineral Resource estimation. Specification of the grid*	All drillhole collar locations are accurate (+/- 50mm) xyz coordinates when captured by an experienced surveyor using RTK-GPS equipment. All drill holes reference the NZGD2000 NZTM map projection and collar RLs the NZVD2016 vertical datum.

Criteria	JORC Code explanation	Commentary
	system used. *Quality and adequacy of topographic control.*	DD down hole surveys are recorded continuously with a Precision Mining and Drilling "North-seeking" Gyro downhole survey tool. RC holes are surveyed at 12m intervals using a Reflex multi-shot camera in a non-magnetic stainless steel rod behind the hammer. There are very minor historical adits and shafts at RAS. No surveys of these voids exist, although at least one adit is still accessible. Historical production records total 630.5 tons of ore crushed. Such small volumes are not material to this MRE. Topographic control is provided by LiDAR topographic surveys in 2018 and 2021 covering the entire project area. These are very accurate and suitable for resource estimation.
Data spacing and distribution	*Data spacing for reporting of Exploration Results.* *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* *Whether sample compositing has been applied.*	Drill collar locations in steep terrain are dictated to some degree by best access along contour tracks and gradients that allow safe working access. Drillhole designs take into account this variation to achieve evenly spaced intercepts at the hangingwall of the mineralisation. Drillhole intersection spacing on the hangingwall of the mineralisation at RAS is typically 30 m (EW) by 30 m (NS) but varies from 20 m (EW) by 20 m (NS) in closely spaced areas to 120 m (EW) by 100 m (NS) in widely spaced (inferred) areas. At SRX and SRE drillhole intersection spacing varies from 20 m (EW) by 20 m (NS) to 100 m (EW) by 100 m (NS). These spacings are considered appropriate for determination of geological and grade continuity at the mineral resource categories reported. Some of the RC drilling was sampled as 4m composites and if the composite result exceeded a threshold later re-sampled. There are no composited samples within the gold grade estimation domains and so no composited samples were used in this MRE. Sampling and assaying are in one metre intervals or truncated to logged features.
Orientation of data in relation to geological structure	*Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and*	Drillholes are oriented to intersect known mineralised features in a nominally perpendicular orientation as much as is practicable. True widths are estimated perpendicular to mineralisation boundaries where these limits are known. As the deposits are tabular and lie at low angles, there is not anticipated to be any introduced bias for resource estimates.

Criteria	JORC Code explanation	Commentary
	reported if material.	
Sample security	*The measures taken to ensure sample security.*	Company personnel manage the chain of custody from sampling site to laboratory. DD drill core samples are transported daily from DD rig by the drilling contractor in numbered core boxes to the Company secure storage facility for logging and sample preparation. After core cutting, the core for assay is bagged, securely tied, and weighed before being placed in polyweave bags which are securely tied. Retained core is stored on racks in secure locked containers. RC samples are also place in polyweave bags and secured with zip ties. Polyweave bags with the calico bagged samples for assay are placed in plastic cage pallets, sealed with a wire-tied cover, photographed, and transported to local freight distributer for delivery to the laboratory. On arrival at the laboratory photographs taken of the consignment are checked against despatch condition to ensure no tampering has occurred.
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*	An independent Competent Person (CP) conducted a site audit in January 2021 and December 2022 of all sampling techniques and data management. No major issues were identified, and recommendations have been followed. In February 2023 Snowdon Optiro completed a desktop review of the assay methods and QC sample results and in its report concluded that the sampling and assaying methods are in line with standard industry procedures and that that the assay data in the supplied database is suitable to be used as the basis for a Mineral Resource.

Section 2: Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	Exploration is being currently conducted within Mineral Exploration Permit (MEP) 60311 ($252km^2$) registered to Matakanui Gold Ltd (MGL) issued on 13th April 2018 for 5 years. In 2023 the term of this permit was extended for a further 5 years until 12 April 2028. There are no material issues with third parties. MGL was granted Minerals Prospecting Permit (MPP) 60882 ($40km^2$) to the north of MEP60311 on 30 Nov 2023 for a term of 2 years. The tenure of the Permits is secure and there are no known impediments to obtaining a licence to operate. 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.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	Early exploration in the late 1800's and early 1900's included small pits, adits and cross-cuts and alluvial mining. Exploration has included soil and rock chip sampling by numerous companies since 1983 with drilling starting in 1986. Exploration in the 1990's commenced with a search for Macraes style gold deposits along the RSSZ. Drilling included 13 RC holes by Homestake NZ Exploration Ltd in 1986, 20 RC holes by BHP Gold Mines NZ Ltd in 1988 (10 of these holes were in the Bendigo Reefs area which is not part of the MRE area), 5 RC holes by Macraes Mining Company Ltd in

Criteria	JORC Code explanation	Commentary
		1991, 22 shallow (probably blasthole) holes by Aurum Reef Resources (NZ) Ltd in 1996, 30 RC holes by CanAlaska Ventures Ltd from 2005-2007, 35 RC holes by MGL in 2018 and a further 18 RC holes by MGL in 2019 prior to SML acquiring MGL.
Geology	• *Deposit type, geological setting and style of mineralisation.*	The RSSZ is a low-angle late-metamorphic shear-zone, presently known to be up to 120m thick. It is sub-parallel to the metamorphic foliation and dips gently to the north- east. It occurs within psammitic, pelitic and meta-volcanic schists. The hangingwall of the RSSZ is truncated by the post metamorphic and post mineralisation Thomsons Gorge Fault (TGF). The TGF is a regional low-angle fault that separates upper barren chlorite (TZ3) schist from underlying mineralised biotite (TZ4) schists. Gold mineralisation occurs in the RSSZ as 4 known deposits with Mineral Resource Estimates (MRE) – Come-in-Time (CIT), Rise and Shine (RAS), Srex (SRX) and Srex-East (SRE). The gold and associated pyrite/arsenopyrite mineralisation at all deposits occur as stockworks of brecciated / laminar quartz veinlets within the highly- sheared and silicified schist. The stockworks are centred on highly silicified shear zones and breccia (SBX) which control mineralisation with TGF parallel, moderately east dipping and very steeply east dipping structures all influencing gold distribution. The gold mineralisation in the oxide, transition and fresh zones is characterised by coarse free gold.

Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:*
	  o *easting and northing of the drill hole collar*
	  o *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar*
	  o *dip and azimuth of the hole*
	  o *down hole length and interception depth*
	  o *hole length.*
	
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does*
		Not applicable as no exploration results are being reported.

Criteria	JORC Code explanation	Commentary
	not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	Not applicable as no exploration results are being reported.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	Not applicable as no exploration results are being reported.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Not applicable as no exploration results are being reported.

Criteria	JORC Code explanation	Commentary
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Not applicable as no exploration results are being reported.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Not applicable as no exploration results are being reported.
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	DD infill drilling of existing inferred resources is continues along with minor programmes designed to resolve local geological interpretation uncertainties. A review of field mapping, soil sampling and geophysical surveys is in progress to determine new targets for drilling in the project area. Concurrent to the planned drilling outlined above, additional metallurgical test work, environmental, geotechnical and hydrological investigations are on-going to support the studies into a gold mining and processing operation.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*	Collar location surveys, downhole surveys and assay data are imported into the database from digital files provided by external providers. Geological logging, sample information and QAQC sample insertion data are entered directly using picklists into spreadsheets on mobile devices in the field. All source data is archived for later audits.

Criteria	JORC Code explanation	Commentary
	• *Data validation procedures used.*	All data is validated on import into the database with checks made for interval overlaps, gaps, duplicate holes, duplicate samples and out of range values. The database structure uses key fields to ensure there are no duplicate drillholes or samples.
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.*	Mr Allwood has visited the site on 7 occasions between January 2021 and May 2024, inspecting RC and DD drilling, logging, sampling, QC insertion practices and site geology. No major issues were identified. Some minor recommendations were made and these have since been implemented.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	There is good confidence in the large scale interpretation of the geology. The TGF is easily recognized in core and has a simple tabular geometry. Structural measurements of vein and fault orientations from oriented core allow good confidence in the geometry of mineralisation controlling faults. The drill spacing makes recognizing small scale (<10 m) variations in geometry, especially the internal grade geometries within the estimation domains difficult. The RAS gold grade domains were created using Leapfrog software (v 2023.1.0) which created a 50% probability iso-surface wireframe using a radial basis function (rbf) interpolation of an 0.2 g/t Au indicator of 2 m composites. The rbf used a ‘structural trend’ comprising an an-isotropy of 3:3:1 oriented parallel to the manually interpreted TGF and parallel to a manually interpreted very steeply east dipping, north striking zone identified in the west of the deposit. The TGF footwall and steep zone were manually interpreted from logged lithology and oriented structural data (specifically quartz veins). The gold domains were also restricted to below the footwall of the TGF. Some below indicator grade samples are included within the gold grade domains and some above indicator grade samples are excluded from the gold grade domains because the rbf estimates the probability of points in space being above or below the indicator grade. Manual grade orientation domains were used to split the RAS Leapfrog gold grade wireframe into an east dipping (roughly parallel to the TGF) domain and a steeply dipping domain. 94.5% of the samples are within the east dipping domain. The Srex (SRX) and Srex East (SRE) gold grade domains were interpreted on east-west sections at a nominal grade threshold of 0.25 g/t Au. The TGF and quartz vein orientations were used to guide the domain interpretations. A nominal interpretation grade was used because histograms

Criteria	JORC Code explanation	Commentary
		and cumulative probability plots of the un-domained SRX data showed no natural lower cutoff that could be used to define mineralization. The Au domain grade nominal criteria (0.2 g/t Au) was selected because it is sufficiently below the likely resource reporting cut-off grade (previously 0.25 g/t) that the resource would largely be constrained by block grade estimation rather than interpretations based on sample support. Most of the contained metal (67%) at SRX and SRE occurs in the SRX main domain which is parallel to, and immediately below the TGF. The SRX and SRE gold domains had a minimum width of 2 m downhole and in places included material not meeting the domain criteria to ensure geological and geometric continuity. While individual high grade samples occur throughout the deposit, the best gold grades generally occur immediately below the TGF in the east dipping domain. Further below the TGF gold grades are generally best in the core of the domains and weaken towards the margins. The geometry of the main zone immediately below the TGF is well defined, alternative interpretations of the gold mineralization geometry deeper (more than about 40 m) below the TGF and in the RAS steep domain are possible. The resource categorization reflects this with areas where alternative interpretations are likely classified as inferred, regardless of grade estimation quality measures. Oxidation domains were interpreted from logged oxidation.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	At RAS the east dipping domain has been defined by drilling 1,850m down plunge (-25° towards 025°) and is 300 m to 380 m wide. In plan, this equates to approximately 1,750 m NNE and 300 m to 380 m ESE. Mineralisation extends vertically in multiple zones over about 180 m. The thickest part of the east dipping domain is continuously mineralized over 50 m vertically below the TGF. Other zones range in thickness from 20 m to 2 m. The deepest part of the east dipping domain is at 180 RL or about 650 m below surface. The core of the east dipping domain is very continuous At SRX the main gold domain extends approximately 700 m along strike (NW), 150 m to 450 m down dip and is typically 4 m to 12 m thick. The other SRX domains are less extensive, having strike lengths of 100 m to 250 m, extending 50 m to 100 m down dip and being typically 2 m to 6 m thick. The mineralization at SRX is quite continuous, but there are rare un-mineralised holes within the domains.

Criteria	JORC Code explanation	Commentary
		Similarly, at SRE the gold domain extends approximately 100 m along strike (NW), 400 m down dip and is typically 2 m to 14 m thick. The mineralization at SRE is quite continuous, but there are rare un-mineralised holes within the domains.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.*	This MRE was made by interpolating gold assays composited to 2.0m by ordinary kriging into a sub-blocked model using Minesight v 16.1.0 software. Geostatistical analysis was carried out using Leapfrog Edge v 2023.1.0 software. Outlier grade limits were determined from log histograms and cumulative probability plots were used to restrict the spatial influence of extremely high grades by domain. At RAS the outlier grades were 40 g/t Au in the east dipping domain and 20 g/t Au in the steep domain. In the SRX main domain the CV was reduced to 1.3 after the application of the top cut (10 g/t Au). The same variogram model was used in all RAS domains. The variogram model was determined from experimental variograms of composites below the outlier limit grade (40 g/t Au) in the east dipping domain. There are insufficient data in the steep domain to create robust experimental variograms, therefore the east dipping domain variogram model was appropriately rotated to reflect the geometry of the steep domain. The variogram model had a relative nugget effect of 52% and two sills. The major axis was parallel to the intersection of the steep zone and the east dipping zone (19/357), the semi-major axis 16/093 and the minor axis 65/220. The total ranges were 125 m for the major axis, 55 m for the semi-major axis and 35 m in the minor axis direction. At RAS blocks were interpolated by ordinary kriging of the top cut composites using a minimum of 4 and a maximum of 15 composites from within a 150m by 150m by 50m ellipsoid oriented parallel to the variogram model. A maximum of 7 composites were used per quadrant from a minimum of two quadrants. Gold domain boundaries were treated as hard boundaries. Parent blocks were 12.5 m (E) by 12.5 m (N) by 5m (vertical), sub-blocked to 2.5 m by 2.5 m by 0. 5m. The block model parent blocks are approximately 25% of the typical drill spacing. The parent block size was selected as a compromise between honouring the domain geometry / volume and minimizing block grade estimation error. At SRX and SRE the variogram model was determined from experimental variograms of normal score transformed composites from the SRX main domain. The variogram model was back-transformed prior to use. The back-transformed variogram model had a relative nugget effect of 68% with one sill. The major axis (00/130) and the semi-major axis (25/040) have similar ranges and together define a plane parallel to the TGF. The minor axis was 65/220. The total ranges were

Criteria	JORC Code explanation	Commentary
	• *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	30 m for the major axis, 25 m for the semi-major axis and 4 m in the minor axis direction. The orientation of the variogram model and search ellipsoid was varied to be parallel to other domains as appropriate. At SRX and SRE blocks were interpolated by ordinary kriging of the top cut composites in two passes. The first pass used a minimum of 10 and a maximum of 20 composites from within a 40 m by 40 m by 6 m ellipsoid oriented parallel to the variogram model. A maximum of 7 composites were used per quadrant from a minimum of two quadrants with a maximum of 4 composites from each drillhole. The second pass was the same as the first pass except that it used a minimum of 4 and a maximum of 15 composites, no quadrant restriction and a 150 m by 120 m by 20 m ellipsoid. Gold domain boundaries were treated as hard boundaries. Parent blocks were 12.5 m (E) by 12.5 m (N) by 4 m (vertical), sub-blocked to 2.5 m by 2.5 m by 0.5 m. The block model parent blocks are approximately 25% of the typical drill spacing. The parent block size was selected as a compromise between honouring the domain geometry / volume and minimizing block grade estimation error. Check estimates were completed on the RAS MRE as follows: using top cuts at the outlier grade limits; outlier restriction at 12.5 m instead of 25 m; and no top cut. In addition, volume – variance analysis using an affine correction was completed to assess which variants best represented the theoretical grade – tonnage curve. Previous estimates of the gold MRE at RAS have been made in 2019, 2021, July 2022 and February 2023 and February 2024. At SRX and SRE previous estimates of the gold MRE were made in November 2021. There has been no production from the BOGP to allow reconciliation of the model. No by-products are assumed. pXRF Arsenic grades have been estimated in the block models for use in characterizing waste. The block model parent blocks are approximately 25% of the typical drill spacing. The parent block size was selected as a compromise between honouring the domain geometry / volume and minimizing block grade estimation error. Open pit mining is assumed with a likely smallest mining unit (SMU) of about 5m by 5m by 5m. Underground mining is also possible, albeit at a higher cut-off grade (around 1.5 g/t Au). No assumption is made of correlation between variables.

Criteria	JORC Code explanation	Commentary
		The MRE is geologically controlled by the use of domains interpreted with reference to the geological model. At RAS the influence of outlier grade composites was restricted to 25 m. At SRX and SRE top cuts were applied to the composites prior to grade interpolation as described above. The block model was validated against drilling grades visually in section and in plan, by the use of swath plots, and by comparison of the block model volumes to domain wireframe volumes. No reconciliation data is available as mining has not commenced.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages are estimated on a dry basis. Assays are reported as weight proportion of oven (110°C) dried samples. Bulk densities were determined from air dried core by immersion.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The reporting cut-offs (0.25 g/t) for ‘open pittable’ resources and 1.5 g/t for underground resources are based on metallurgical recovery indicated by gravity / CIL test work, processing, mining and G & A costs from comparable projects and revenue from a gold price of USD$1,830/oz escalated by 30% to allow for the reasonable prospects test. Reporting using the lower 0.25g/t cut-off grade than the 0.5g/t used in the RAS July 2024 MRE is due to the robust economics reported in this PFS, additional metallurgical testing and increase gold price. Other than reporting at a lower cut-off grade there are no other changes from the RAS July 2024 MRE.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous.*	No allowance has been made for mining dilution or mining recovery except that domains were interpreted with a minimum width of 2 m.

Criteria	JORC Code explanation	Commentary
	Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	Metallurgical test work investigating a gravity – CIL process has resulted in combined recoveries ranging from 86.0% to 97.8% and averaging over 90%. Further work is underway to determine full processing parameters and economics.
Environmental factors or assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should*	It is assumed that all permits necessary for commercial gold production will be obtained. Baseline studies are well advanced including: • surface water flow and quality • aquatic ecology • ecology including geckos, skinks, bats, birds, pests and flora • geochemistry • hydrology • socio-economic Other studies have commenced as mine studies advance including noise, traffic, lighting and visual.

Criteria	JORC Code explanation	Commentary
	be reported with an explanation of the environmental assumptions made.	
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	Bulk density was interpolated by inverse distance squared weighting into the fresh and partial oxidation domains from 2,202 bulk density measurements. There was insufficient data in the oxide domain to allow interpolation. Bulk density was assigned to un-interpolated blocks by oxidation domain based on the median values of the bulk density samples in each oxidation domain. No difference was found in the median value of bulk density data between mineralised and un-mineralised samples. Bulk density was measured by core immersion. The core was not routinely coated, allowing water to penetrate voids, however the rocks have very low porosity due to metamorphism. 100 samples of fresh (unweathered) core were tested by the routine method and by wax coating to check for the effect of the water ingress on the bulk density measurements. There was no difference in the average value or the CV of the two methods. Therefore, MGL continues to use un-coated core for density determinations.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	Input data quality, confidence in the geological interpretations, average distance to composites used, distance to the nearest composite used and the kriging slope of regression (a function of grade continuity and data (drilling) configuration), and for SRX and SRE, interpolation pass number were all considered when classifying the model. In general, indicated resources are reported from continuous zones of un-ambiguous geological interpretation and in block grades where the nearest composite was less than 25 m away, the average composite distance was less than 40 m, kriging slope of regression was greater than 0.6 and at SRX and SRE interpolated in pass 1. Resource categorization is based on confidence in the estimation of gold grades only. The resource classification appropriately reflects the Competent Person's view of the deposit.

Criteria	JORC Code explanation	Commentary
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates*	An earlier iteration of the RAS MRE was reviewed by AMC Consultants who concluded that the MRE is an adequate representation of average grade and grade trends but with a degree of local variability not able to be accurately represented in the model.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource 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 resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that 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.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	The relative accuracy and confidence in the MRE is reflected in the resource classification. No quantitative assessment of errors has been made. The RAS MRE is a global estimate intended to give the best global grade – tonnage relationship, suitable for use in long term planning but not for local (block scale) estimates. No production data are available for reconciliation as mining has not commenced.

Section Four: Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

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 Ore Reserve estimate is prepared from the following Mineral Resources reported by Santana Minerals:

Deposit	Mining method	Category	Cutoff (Au g/t)	tonnes (Mt)	Au (g/t)	koz
RAS	open pit	Indicated	0.25	19.6	2.3	1,452
		Inferred		9.9	2.0	634
		Total		29.5	2.2	2,086
	underground	Indicated	1.5	0	1.9	0
		Inferred		2.1	2.2	145
		Total		2.1	2.2	145
	RAS Total	Indicated		19.6	2.3	1,452
		Inferred		12	2.0	779
		Total		31.6	2.2	2,231
SRX	open pit	Indicated	0.25	2.6	0.7	59
		Inferred		2.4	0.9	73
		Total		5.0	0.8	132
SRE	open pit	Indicated	0.25	0.4	0.7	10
		Inferred		0.1	0.9	3
		Total		0.5	0.8	13
CIT	open pit	Inferred	0.25	3.2	0.8	80
Total	**combined**	**Indicated**		**22.6**	**2.1**	**1,521**
		Inferred		**17.7**	**1.6**	**935**
		Total		**40.3**	**1.9**	**2,456**

- The Mineral Resources are reported inclusive of Ore Reserves
- The block models used were
  - 20240625 RAS PFS Model ext.bmf (Extended model bmf file to cover enlarged model

Criteria	JORC Code explanation	Commentary
		extents.) ○ 20240830 SHRE model.bmf (SRX Block model)
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.*	• The Ore Reserve estimate was completed by Rodney Redden who is the project study manager and has been to site multiple times since February 2024
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 viable, and that material Modifying Factors have been considered.*	• The Reserves are supported by the completion of a pre-feasibility study undertaken by Santana Minerals – PFS (this study).
Cut-off parameters	• *The basis of the cut-off grade(s) or quality parameters applied.*	• Estimated site operating costs, royalty payments, processing recoveries and an underlying gold price assumption were used to calculate the cut-off grades • For the underground estimate a dilution % was also factored. • Cut-off grades applied to select material for inclusion in the ore reserves were: ○ RAS Open pit: 0.3 g/t ○ SRX Open pit: 0.35 g/t ○ RAS Underground 1.70 g/t
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 resource block models as received were re-blocked for open pit mining assessment to simulate the assessed minimum mining unit (12.5m x 12.5m x 2.5m). Which only required the existing block size to be increased in the z direction from 2m to 2.5m. • The Lerch Grossman algorithm (LG) was used to create sets of possible open pit mining shells for both RAS and SRX. This process was performed on all Indicated resources as well as a high grade – high confidence core wireframe at RAS. The high confidence core scenarios were particularly used for guiding the initial mining stages (stages 1 and 2)

Criteria	JORC Code explanation	Commentary
	• *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.*	• For underground analysis, an insitu cut-off grade was used to target mineralization for mining. Both an automated (MSO) process followed by a more thorough manual targeting process were utilised. • The open pit mining base assumption is that all material will be dug off in 2.5m high flitches (this is in line with optimized practices at the nearby Macraes mine which has a very similar ore geometry). The pre-strip at RAS is significant as it outcrops under the RAS ridge. The selected fleet sizing is based on 150 tonne haul trucks loaded by 300 tonne excavators. • The Underground method selected is longitudinal open stoping with paste backfill. The scoping study examined a number of different mining methods with open stoping with paste backfill an obvious preferred option. • The grade of the underground mineralization supports the increased costs of backfilling compared with the resource loss without using backfill • Paste backfilling has significant benefits over other filling methods due to ○ Quick filling time; and ○ Ability to tight fill to support the stope backs • The geotechnical parameters recommended and applied for RAS open pit are:

Wall	Aspect [1] (°)	Unit	IRA [2] (°)	BFA [3] (°)	Berm Width (m)	Bench Height (m)	Controlled By
Southwest	350 to 065	All	30	50	9	15	Foliation/foliation shears dipping towards northeast
West	065 to 160	TZ3	35	50	11	15	Planar sliding along the obliquely dipping TGF [4]
		TZ4	45	60	6.5	15	Planar failure along faults and shears identified behind the pit wall
Northeast	160 to 235	All	45	60	6.5	15	
East	235 to 350	TZ3	40	60	9	15	Planar sliding along the obliquely dipping TGF [4]
		TZ4	47	70	7.5	15	Planar failure along faults and shears identified behind the pit wall

(1) Slope aspect measured as the direction the wall dips towards.

Criteria	JORC Code explanation	Commentary
		(2) Inter-ramp angle. (3) Batter face angle. (4) Opportunity to steepen IRA based on future 3D stability analyses and/or mapping data of the TGF.

- The geotechnical parameters recommended and applied for SRX open pit are:

Wall	Aspect (°)	IRA (°)	BFA (°)	Berm Width (m)	Bench Height (m)	Controlled by
Southwest	350 to 065	30	50	9	15	Foliation/foliation shears dipping towards northeast
West, Northeast, and East	065 to 350	45	60	6.5	15	Probable Planar failure along faults and shears identified behind the pit wall

- The geotechnical parameters recommended and applied for RAS underground are:

Depth (mbgl)	Stope Height [(1)] (m)	Maximum Stope Length [(2)] (m)	Maximum Stope Width [(3)] (m)	Notes and Limiting Wall Mechanism
250	20	25	15 [(4)]	Assumes heavy support of the backs is practical and economic Potential for footwall planar slide
	25	20		
400	20	20		
	25	15		

(1) Vertical height.
(2) Along strike.
(3) Across Strike
(4) Stope width is expected to be controlled by the ability to support the backs.

- Grade control drilling in the open pits will be done in conjunction with the blastholes. No separate grade control drilling program is planned. Ore-zone drilling will be based on 7.5m high packages and drilling is on a 4.0m x 4.7m pattern
- Underground grade control will be performed by a diamond drill from the lower ore-drives once they are in place to define the orebody hangingwall

Criteria	JORC Code explanation	Commentary

		• The re-blocked open pit model is a recoverable model with dilution and ore-loss accounted for in this process. No further dilution or ore-loss is then factored
		• For Underground, the applied dilution and recoveries are:
		  ○ Development dilution was calculated at 18%;
		  ○ Primary stope dilution of 8% and secondary stope dilution of 12% was used;
		  ○ Recovery of blasted material calculated at 95%; and
		  ○ Recovery of the final pillars was factored 60%.
		• The minimum cut-back applied to the open pits is 40m
		• The minimum stoping height assumed is the ore development drive height of 4.5m, which is not required
		• Inferred mineralization was not targeted for pit optimisations or designs, nor was it used as a guide for underground stoping designs.
		• Inferred that is mined in the schedule is only as a consequence of falling inside the pit design, stopes or development.
		  ○ RAS Open pit scheduled material is 94.4% Indicated
		  ○ SRX open pit scheduled material is 99.0% Indicated
		  ○ RAS UG scheduled is 96.2% Indicated
		• No Inferred resources are included in the ore reserves and their exclusion from the overall scheduled mill feed has a negligible effect.
		• The total site infrastructure requirements are discussed in “infrastructure” below.
		• The open pit mining specifically will require an explosive emulsion plant and magazines, fleet workshop, refuelling facility, mobile fleet workshop and washdown facilities, supported by mining offices and a crib-room/pre-start area.
		• The Underground requires a paste backfill plant installed on surface. A dedicated portal area is established for twin decline ramps that provide the primary ventilation circuit, secondary egress, main haulageway and for paste/services. The 11kv site system is extended to the

Criteria	JORC Code explanation	Commentary
		portal and then underground, eventually reduced to 11kv. The primary fan is on surface at the portal exhaust.

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 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?*
		• The final selected flowsheet is appropriate to the style on mineralization and involves:
		  ○ a single stage crush (121mm);
		  ○ single stage SAG mill (p80 106 micron), with the addition of a ball mill when the throughput rate is to be expanded;
		  ○ Cyclone classification;
		  ○ Gravity gold concentration;
		  ○ CIL leach and adsorption gold extraction of the gravity tails;
		  ○ Elution;
		  ○ Cyanide destruction;
		  ○ Arsenic removal;
		  ○ Tails thickening and tails pumping to a wet TSF facility
		• The technology is well tested. There is no novel technology involved.
		• The process flowsheet is supported by multiple rounds of metallurgical testwork.
		  ○ Stages 1-4 of testwork was completed from 2018-2022
		  ○ The most recent (stage 5 testwork) of which undertaken in 2024 has used a master composite for both RAS and SRX of 100kg each plus variability samples (10) for RAS and 8 for (SRX)
		The stage 5 (PFS) testwork programme had the following objectives:
		• Composite master sample selection to represent the expected Life of Mine (LOM) ore blend for the RAS deposit.
		• Variability sample selection for RAS to provide spatial variability data
		• Determination of comminution characteristics for the master composite and variability samples.
		• Gravity recovery and intensive leaching of gravity concentrate on all samples.
		• Flotation sighter testing on master composite.
		• Cyanide leach grind optimisation, reagent optimisation and CIL testing on master composite.

Criteria	JORC Code explanation	Commentary
		• Cyanidation response based on optimised flowsheet for the variability samples. As the testwork program proceeded the following steps were included: • Cyanide destruction testwork on master composite. • Arsenic removal on master composite. • Diagnostic leaching of optimised CIL of master composite. • Thickening testwork. The SRX deposit was subsequently added to the testwork program, with initial testwork based on the RAS optimised program. • The RAS master and variability testwork is complete. The variability samples supported the aster composite recoveries at 106 micron grind of 65.2% gravity and a 93.9% overall recovery. • The SRX master composite work is also complete and through the process route optimized for RAS it returned 22.9% gravity and a 68.3% overall recovery • No allowances have been made for deleterious elements • No bulk sample has been taken
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.*	• A comprehensive set of baseline studies have been commissioned to understand the existing environment across the project area and surrounding landscape. As the project description has developed the assessment of effects on the environment has also progressed along with associated considerations of opportunities to address potential negative effects as far as practical. • Environment related studies include ecology, waterways and wetlands, ground and surface water, geochemistry, noise, air quality, heritage, closure, visual effects, recreation and traffic. • Project waste rock characterization is well advanced. Studies indicate that the rocks associated with the project (TZ3, TZ4, and RSSZ) will not generate acid rock drainage with >350 samples tested by industry accepted acid base accounting (ABA) techniques (e.g., AMIRA, 2002). This is a function of the high acid neutralisation capacity (ANC) of the rocks associated with carbonate minerals (e.g., dolomite) and a low sulfide mineral content (e.g., arsenopyrite, pyrite) that can generate lesser acidity. The overall ABA assessment indicates that the rocks are classified as non-acid forming (NAF). Data for waste rock indicates that the TZ4 and RSSZ lithologies contain ~97.7% of arsenic and 37.2% of sulfur yet represent only

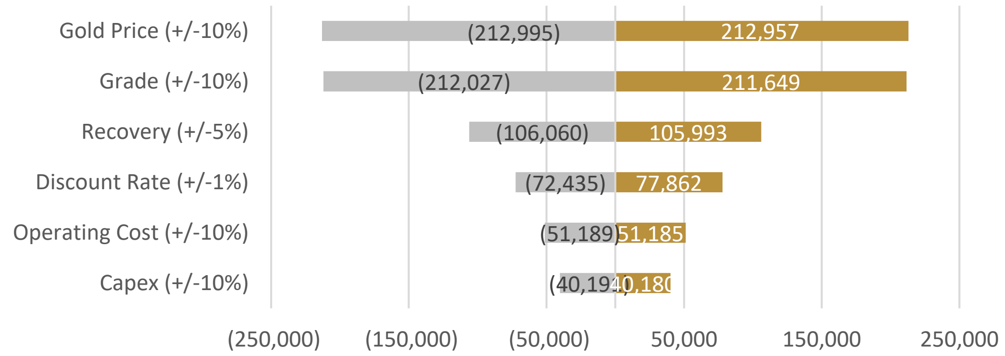
Criteria	JORC Code explanation	Commentary
		18% of the waste rock that will be disturbed. Hence, appropriate management of waste rock to reduce sulfide mineral oxidation and the release of arsenic is a critical step to minimise any potential deleterious effects of mining, i.e., manage 18% of the waste rock well to mitigate 97.7% of the arsenic risk in the Engineered Landform ELF) that will contain the waste rock. Nitrogenous compounds such as nitrate are also expected to be elevated in seepage from blasted rock due to the use of ANFO, an ammonium-nitrate fuel oil explosive. This is not an uncommon problem in the mining industry. • The management of MIW will involve several engineering controls to minimise the effects on the downstream environment. These engineering controls have been accounted for in the mine plan, including: ○ Materials management and the construction of an Engineered Landform (ELF) to minimise contaminant loads from the waste rock; and ○ Water management and treatment as necessary. • The main waste rock stack (Shepherds ELF) has been designed to enclose TZ4 and RSSZ materials in its core away from water and air ingress • The ELF will require a building consent from the local council. This would be subsequent to the FAB major consent decision • The TSF will require both a resource consent and building consent. • Waste rock stack approval. This would be subsequent to the FAB major consent decision
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.*	• Planned infrastructure includes: ○ An initial 1.5Mtpa processing facility expanding to 2.1Mtpa eventually ○ A Tailings Storage Facility (TSF) ○ A ROM pad ○ An Engineered Landform (ELF) to take the non processed materials ○ A 24MVa 66kv high voltage transformer and power supply from the local power grid ○ An all weather two lane road from the nearby state highway connection ○ A 100l/sec borefield and pipeline to site ○ Administration, mining and processing offices ○ A warehouse

Criteria	JORC Code explanation	Commentary
		○ Mobile fleet workshops for Open pit and Underground ○ Refuelling and plant washdown facilities ○ Metallurgical laboratory ○ An 80-person construction workforce camp • The company has agreements in place with the two main landowners that the project straddles to purchase or lease the required land for the project and all infrastructure.
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.* • *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.*	• Operating costs have been estimated by: Applying productivity, availability and utilisation to the mining and processing physicals (including derived activities) to calculate required quantities for equipment, personnel, consumables and power. • Input costs for equipment, personnel, consumables and power have been sourced from current administration costs, nearby operating sites, rates submitted by contractors and suppliers, updated budget pricing for consumables and advice from consultants. • Capital costs have been estimated by: ○ Engineering cost estimate by MACA Interquip Mintrex for processing plant and tailings pipeline, completed in October 2024. ○ TSF estimate by Engineering Geology Limited, October 2024 ○ Power costs estimate by ERGO consulting, October 2024 ○ Water servicing to site by Pattle Delamore Partners, October 2024 ○ Road upgrades by Stantec NZ Ltd, October 2024 ○ Other infrastructure by Performance Ltd ○ Mobile fleet purchase cost estimates from TerraCAT, Cable-price, Sandvik, Normet and Volvo • Capitalised operating costs for pre-production operations include: ○ Open pit mining costs ○ Site G&A costs • After Commercial Production, capital costs include:

Criteria	JORC Code explanation	Commentary
		○ Sustaining capital projects ○ TSF raises ○ Underground mine development – capital development only ○ Ecological offsets and water treatment facilities ○ Closure • No allowance has been made for deleterious elements. • Exchange rates are derived from current exchange rates. • The NZ government royalty rate is 2% Ad Valorem or 10% of Net Accounting profits (whichever is the higher) • Other Royalties are vendor and landowner. These vary from a minimum of 1.5% to a maximum of 3.5% Ad Valorem with the actual amount depending on: ○ Location of extraction; ○ Total ounces extracted from various locations; and • Land ownership arrangements
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.*	Metal prices assumed for economic test of the Ore Reserve estimate are: • RAS (Open pit and Underground) ○ Au Price: US$1,650/oz ○ NZ$:US$ exchange: 0.64 • SRX ○ Au Price: US$2,100/oz ○ NZ$:US$ exchange: 0.64 Metal prices assumed for Base Case of the Pre-feasibility are: • Au Price: US$1,900/oz • NZ$:US$ exchange: 0.596 and A$:US$ exchange: 0.66 • Metal Price and exchange rate assumptions have been benchmarked against industry peers (for Au)

Criteria	JORC Code explanation	Commentary
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.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *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.*	• For gold doré sales, there is a well-established and transparent market.

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.*
		Inputs to the financial model are:
		• Capital and operating cost estimates from the Study, estimated as described above (no escalation has been applied to costs);
		• Physicals schedule of saleable;
		• Gold prices assumed for Base Case of the Pre-feasibility Study (no escalation has been applied to selling prices);
		• The base case NPV applied:
		  ○ Au Price: US$1,900/oz
		  ○ NZ$:US$ exchange: 0.596
		• A discount rate of 8% has been applied to calculate NPV
		• The base case post tax NPV is $AUD535M
		• Sensitivities in AUD have been assessed at various selling prices for Au as follows

Criteria	JORC Code explanation	Commentary
		
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	• The company has established access agreements to the freehold land that is required for the project to be executed as per the PFS. • The company has been in frequent consultation with the Central Otago District Council and the Otago Regional Council, various state regulators and hold good standing with the local community. • The company will continue to communicate and negotiate in good faith with all stakeholders as part of the proposed development. It is not expected that there will be any significant impediments to development of the project.
Other	• *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.*	• Earthquakes are the single largest material naturally occurring risk. ○ The Shepherds TSF will safely contain tailings when subjected to potential future extreme earthquakes. It will be designed to withstand a 1 in 10,000 year earthquake including aftershocks. This includes withstanding a potential rupture on the Alpine Fault or any of the other active faults in the region. The proposed design has the tailings contained behind the downstream rockfill embankment, that will also be buttressed by a large volume of rockfill placed in the Shepherds ELF. The proposed design will provide safe and robust tailings storage solution for both operation and post closure of the site. ○ The processing plant and all infrastructure has been engineered to NZ building code

Criteria	JORC Code explanation	Commentary
	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.	standards relevant to the local region. • 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. • The NZ Government has introduced a new legislation, Fast-track Approvals Bill (FAB). This Bill provides a streamlined decision-making process to facilitate the delivery of infrastructure and development projects with significant regional or national benefits. On 04/10/2024 it was announced that the Santana Minerals Bendigo-Ophir gold mine is included within the list of projects eligible to access the fast-track consenting framework under the proposed FAB. The Bill is intended to be a “one stop shop” for consenting projects which would otherwise require consents under multiple different regimes including resource consents under the RMA, concessions under the Conservation Act 1987, wildlife permits under the Wildlife Act 1953, archaeological authorities under the Heritage Pouhere Taonga Act 2014, and land access provisions of the Crown Minerals Act 1991. The “one stop shop” approach marks a significant change in project approvals in New Zealand, and it is hoped that this will significantly reduce consenting costs, uncertainty, and timeframes.
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.*	• Material classified as Indicated Mineral Resources has been converted to Probable Ore Reserve

Criteria	JORC Code explanation	Commentary
	• *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	• The results described in the PFS appropriately reflects the Competent Person's view of the deposit. • There are no Probable Ore Reserves quoted from Measured Mineral Resources
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• No external audit or review of this Ore Reserve estimate has been undertaken.
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*	• The design, schedule and financial model for the BOGP has been completed to a Pre-feasibility standard with a +/-25% level of confidence. • A degree of uncertainty exists with the geological estimates used to estimate the Ore Reserve which is reflected in the Mineral Resource classification. • The Ore Reserve is best reflected as a global estimate. • There is a degree of uncertainty regarding estimates of modifying mining factors, geotechnical and processing parameters that are of a confidence level reflected in the level of the study. • There is a degree of uncertainty in the prices used. • The Competent Person is satisfied that the assumptions used to determine economic viability of the Ore Reserve are reasonable at time of publishing. • The Competent Person is satisfied that a suitable margin exists that the Ore Reserve estimate would remain economically viable with any negative impacts applied to these factors or parameters.

Criteria	JORC Code explanation	Commentary
	accuracy and confidence of the estimate should be compared with production data, where available.	