SANTANA MINERALS LIMITED

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1 July 2025

BENDIGO-OPHIR GOLD PROJECT
UPDATED PRE-FEASIBILITY STUDY (PFS)

HIGHLIGHTS:

Very Strong Fiscal Outcomes

- After tax net present value $NPV_{6.5}$ of A$1.52 billion at a current gold price of A$4,950/oz (below 90-day average).
- After tax $NPV_{6.5}$ of A$780 million at a base-case gold price of A$3,500/oz (30% below current price).
- Internal Rate of Return (IRR) of 65% at current gold price (39% IRR at base-case).
- Profit before tax of A$3.5 billion at current gold price (A$1.90 billion at base-case).
- Total cash operating cost of A$1,741/oz[1] at current gold price (A$1,559/oz[1] at base case).
- All-in-Sustaining Cost (AISC) of A$1,842/oz[1] at current gold price (A$1,660/oz[1] at base case).
- All-in-Cost (AIC) of A$2,132/oz[1] at current gold price (A$1,950/oz[1] at base case).
- PFS detailed cost estimates to bankable standards with +/- 15% accuracy.

Refinement to Nov. 2024 PFS - Lower Pre-Production Capital Requirements

- Total construction and establishment pre-production costs of A$277 million including 10% contingency. Key areas include:
  - Infrastructure and services: A$64 million.
  - Process plant (1.2Mtpa) and Tailings Dam: A$119 million.
  - Mine Establishment and pre-strip costs: $94 million.

Expanded Initial Mine Life with Strong Production Profile

- Gold production of 1.25 million ounces over an initial 13.8-year mine life (LoM).
- Targeted annual gold output of ~120,000oz in peak mining periods.
- Production from open pits for >13 years and underground for 7 years (with further underground potential, post future resource conversion).

Refined Development Strategy

- Selective open pit staging and cutbacks focused on near-surface ore to reduce pre-strip requirements.
- A progressive build-up of gold production.
- Plant design changed from closed circuit semi-autogenous-grinding (SAG) mill to three-stage crushing and ball milling at a reduced rate of 1.2Mpta (with expansion capacity to enable a future increase to 1.8Mtpa).
- A commencement mining Reserve of 15Mt @ 2.58g/t Au for 1.24Moz.

ESG

- Low-emission power potential sourced from grid-connected hydroelectricity, supporting a low-carbon development.
- The most comprehensive environmental baseline study ever undertaken in the region covering ecology, water, land-use, and mine closure rehabilitation strategy, underpinning the project's ensuing approvals process. Full reports to become publicly available post submission of the Company's Fast Track Approvals Application.
- A modern processing plant with state-of-the-art detoxification and closed-circuit water treatment to substantially manage contamination risks with storge of plant wastes in low-risk engineered landforms.
- A project plan designed to generate significant regional employment opportunities, with strong industry multiplier effects and a clear focus on local hiring and workforce training.
- A project that will generate substantial economic output to the region and the Crown over its initial ~14-year life with a high potential to sustain output beyond the initial mining term.

[1]higher due to 10% Accounting Profit (APR) Crown royalty applying over 2% NSR.

Strategic Advancements in the Updated PFS

The following table specifies changes to modifying factors and planning inputs since the initial PFS announced on 15 November 2024. All modifying factors that have been reviewed and consequently remodelled for this Updated PFS are listed below.

Study Area	PFS - NOV. 2024	UPDATED PFS - JUN. 2025
Mineral Resources		
Indicated mineral resource	1.45Moz	1.54Moz
Deposits included	RAS & SRX	RAS & SRX
Processing		
Process plant	1.5Mtpa	1.2Mtpa (option to expand to 1.8Mtpa)
Process circuit	Single Stage Crush, SAG Mill	3 Stage Crush, Ball Mill
RAS avg. metallurgical recovery	92.4%	Regression derived based on head grade (93.2% average)
SRX avg. metallurgical recovery	68.3%	81.8%
Tailings Storage Facility	Shepherds Creek design	No change
Open Pit Mining		
Selective mining unit (SMU)	SMU 12.5m x 12.5m x 2.5m	SMU 7.5m x 7.5m x 2.5m
Geotechnical parameters	See November 24 PFS	Updated
Pre-strip to first ore	15Mbcm	6.5Mbcm
Open pit stages	5	7
Open pit - years	9.2 (incl. SRX)	13yrs at RAS, 1.5yr at SRX
Average mining cost	NZD$3.74/t	NZD$4.61/t
Reserve cut-off grade	0.3g/t	0.5g/t
Underground Mining		
Development	Twin decline from Shepherds Creek	No change.
Stope design	Top drive in TZ3 waste	Top drive in TZ4 ore
Mining method	Long hole stoping with paste fill	No change
Mining period	Year 3 to Year 8	Year 7 to Year 13
Average all-in mining cost	NZD$93.0/t	NZD$98.7/t
Reserve cut-off grade	1.7g/t	No change
Other Financial Inputs		
Royalties	Private royalties average 2.25% Crown royalty 10% APR	No change
Base case gold price	A$2,894/oz	A$3,500/oz
Power	Grid power at 13c/kWhr	No change
Tax credits	Not included	A$36M Included
Discount rate	8%	6.5% reflects project de-risking

Table 1 Changes in Modifying Factors

Key Updated PFS Data

The key financial outcomes of the Updated PFS are summarised in the following tables and charts with full cost and input information in the *Updated Pre-Feasibility Study – June 2025* attached. Financial projections are presented using a conservative three-month average spot gold price of US$3,138/oz (~A$4,950/oz at a foreign exchange rate of USD:AUD 0.63). A robust base-case study using US$2,220/oz (~A$3,500/oz at a foreign exchange rate of USD:AUD 0.63) is also presented in Table 3.

Total mining physicals underpinning all financials can be seen below in Table 2.

Key Project Mining Physical Targets and Assumptions		Nov 24. PFS	Updated 25. PFS
Mine life	Years	9.17	13.8
Ave plant throughput	ktpa	1,835	1,184
Open pit mill feed	kt	14,404	12,591
Open pit mill feed grade (incl SRX)	Au g/t	2.19	2.53
Open pit contained gold	kOz	1,014	1,024
Open pit recovered ounces	kOz	935	953
Underground mill feed	kt	2,413	3,788
Underground mill feed grade	Au g/t	2.99	2.60
Underground contained gold	kOz	232	316
Underground recovered ounces	kOz	215	295
Total mill feed	kt	16,817	16,379
Au grade - mined	g/t	2.30	2.54
Total contained gold	koz	1,245	1,340
Overall average plant recovery	%	92.38%	93.14%
Gold production	**kOz**	**1,151**	**1,248**

Table 2 Key Mining Physicals

The processing plant has been initially sized at 1.2Mtpa to align with the higher-grade ore feed from the RAS open pit, particularly in the early years of operation, and to support a smoother transition to underground mining. The plant layout has been designed with the flexibility to expand to 1.8Mtpa, providing the potential to increase throughput and bring forward gold production within the mine life. Figure 1 below illustrates the projected ore feed at 1.2Mtpa from the RAS open pit, RAS underground, and SRX deposits over the life of mine.

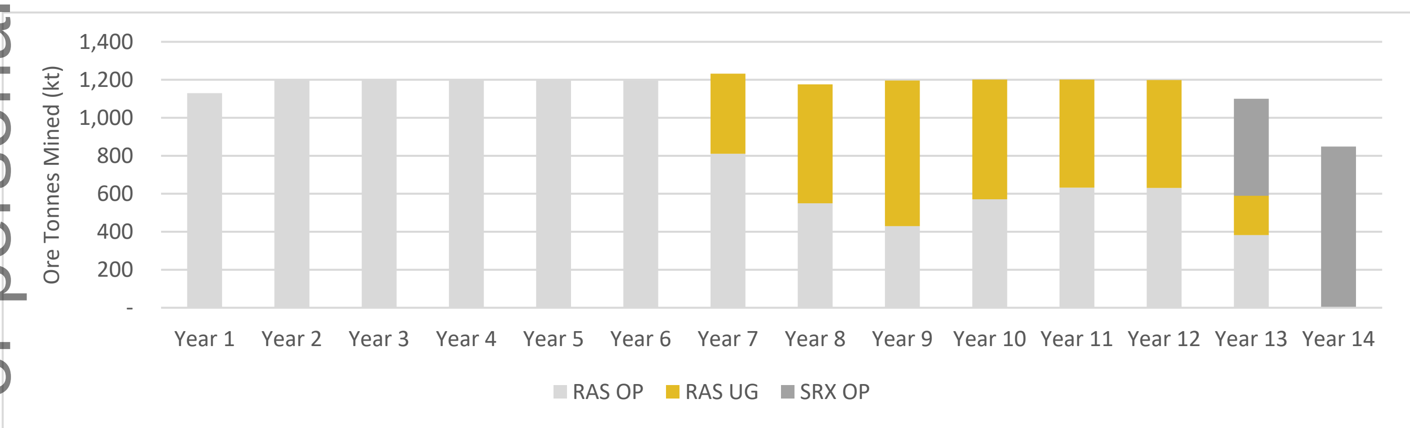

Figure 1 Ore Feed Schedule 1.2Mtpa

Figure 2 below shows the production profile per annum with gold being produced from the RAS open pit for 13 years, supplemented by ore from the RAS underground mine in Year 7 to Year 13, and SRX open pit in years 12 and 13.

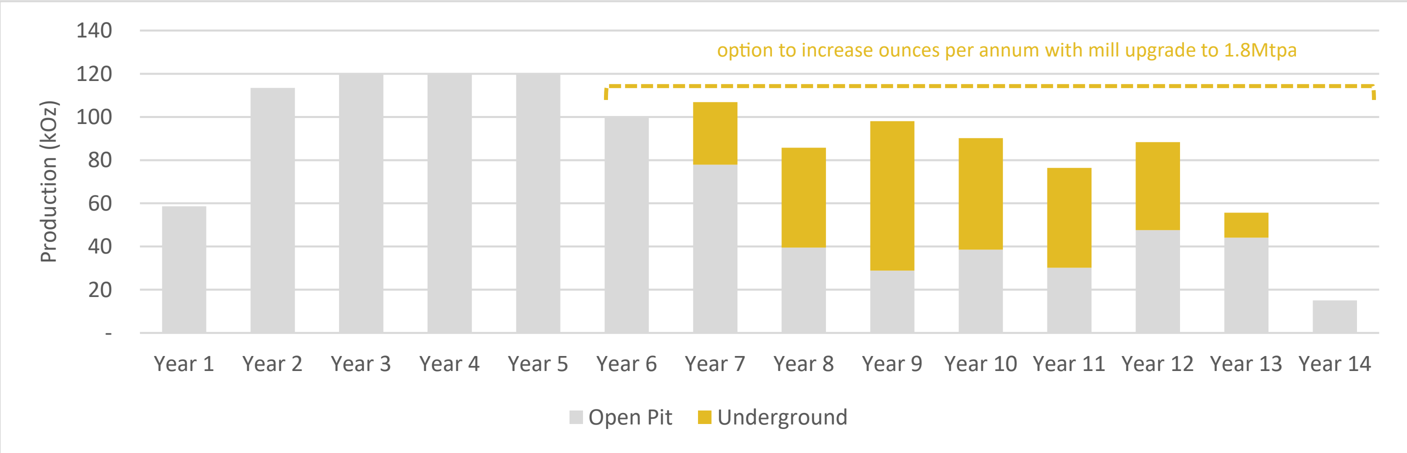

Figure 2 Annual Ounce Production Profile

Table 3 below shows the financial inputs that result in a 'Total cost per ounce' (AIC) which includes all pre-production CAPEX.

Key Financial Assumptions		Base-Case AUD	3-Month AUD	3-Month NZD
Gold price	$/oz	3,500	4,950	5,410
Exchange rate	USD:$	0.63	0.63	0.58
Initial Life of Mine Metrics				
Gold sales	Oz		**1.248 million**	
Initial mine life	Yr(s)		13.8	
Gold revenue ($'000)	**AUD 'mil**	**4,367**	**6,177**	**6,751**
Initial life of mine operating costs				
Total open pit mine operating costs	AUD 'mil	777	777	849
Total underground mine operating costs	AUD 'mil	246	246	269
Total ore processing operating costs	AUD 'mil	416	416	455
Total general and admin costs[2]	AUD 'mil	158[2]	158[2]	172[2]
Crown royalties (higher of 2% NSR or 10% annual profit)	AUD 'mil	232	410	448
Third party royalties – (3 other)	AUD 'mil	117	166	181
Total cash operating cost	**AUD 'mil**	**1,946**	**2,173**	**2,375**
Total cash operating surplus *(EBITDA)*	**AUD 'mil**	**2,422**	**4,004**	**4,376**
Non-cash costs				
Life of mine depreciation and amortisation	AUD 'mil	480	480	524
Total cost of sales	**AUD 'mil**	**2,425**	**2,652**	**2,899**
Historical PP&E	AUD 'mil	36	36	39
Net profit before tax (NPBT)	**AUD 'mil**	**1,906**	**3,489**	**3,813**
Corporate tax payable (28.0%)	AUD 'mil	(546)	(983)	(1,074)
Estimated net profit after tax (NPAT)	**AUD 'mil**	1,360	2,506	2,739
$NPV_{6.5}$ (unleveraged and after-tax)	**AUD 'mil**	780	1,521	1,662
Internal rate of return (IRR)	%	39%	65%	65%
Capital Expenditure Requirements				
Pre-production capital (incl. 10% contingency)	AUD 'mil	**277**	**277**	**302**
Sustaining capital expenditure (funded from cash flow)				
Plant & infrastructure	AUD 'mil	48	48	52
Waste stripping	AUD 'mil	78	78	85
Underground mine plant & infrastructure (year 6)	AUD 'mil	85	85	93
Closure capex (off-set against salvage value of PP&E)	AUD 'mil	(0)	(0)	(0)
Total capex over mine life	AUD 'mil	**487**	**487**	**533**
Comparative Metrics (rounded)				
Total cash operating cost per ounce	AUD / Oz	**1,559**	**1,741**	**1,903**
All in cost (AIC)	AUD / Oz	**1,950**	**2,132**	**2,330**

[2]Includes closure OPEX and site-specific G&A discussed in General and Administration section below

Table 3 Financial Projections, Base-Case and Current Gold Pricing

At a gold price of A$4,950/oz, the project generates over A$2.5 billion in free cash flow over the initial mining term. The chart below shows the max cash draw down in the pre-production period of 14 months, followed by high returns in the first years of gold production, allowing a 1.5yr payback from first production.

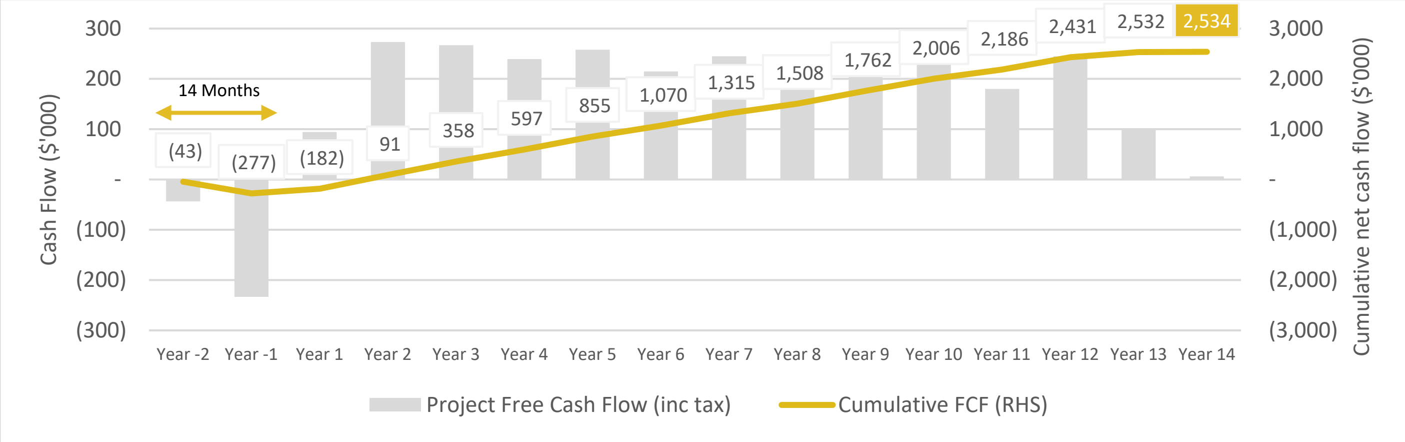

Figure 3 Project Free Cash Flows

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

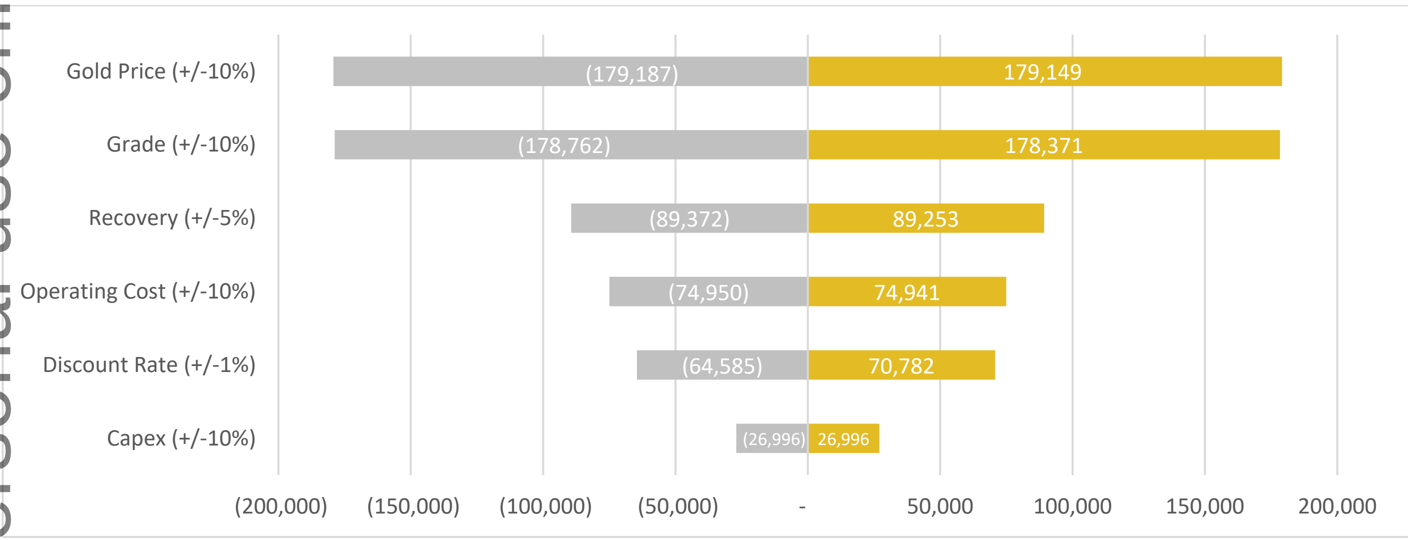

Figure 4 NPV Sensitivities on the Base-Case Scenario, Post Tax ('000 A$)

Table 4 below shows the NPV, IRR and Payback (from production) metrics at price steps above and below the Base-Case and current price scenarios applied in the Updated PFS:

	50% of Spot Gold price	-10%/oz	Base-Case	Spot Price	+10%/oz
	A$2,475/oz	A$3,150/oz	A$3,500/oz	A$4,950/oz	A$5,446/oz
$NPV_{6.5\%}$	A$255M	$601M	$780M	$1.52b	$1.77b
IRR	18%	32%	39%	65%	73%
Payback	4.9Yrs	3.0Yrs	2.6Yrs	1.7Yrs	1.6Yrs

Table 4 Gold Price Scenarios

Development timetable post PFS Update:

With the Updated PFS now complete, the Company is advancing several parallel workstreams to position the Project for development. The Company expects to submit its Fast-track Approval (FTA) application in the next quarter, following detailed engagement with relevant authorities. Based on current guidance, a permitting decision is anticipated within six months of submission.

Detailed engineering is underway in parallel with the FTA application to support project readiness and construction planning. Financing discussions are also progressing with multiple potential lenders, with the aim of aligning funding execution with the expected approval timeline. These coordinated activities are intended to enable a seamless transition into early works, and project construction upon receipt of approvals.

Ore Reserve Estimate (ORE)

The BOGP JORC 2012 compliant ORE is 15.0 million tonnes @ 2.58g/t Au for 1.242 million ounces of gold. This ORE is based on the March 2025 Mineral Resource Estimate (MRE) of 34.3 million tonnes @ 2.1g/t Au for 2.34 million ounces reported at a 0.5g/t cut-off grade.

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	-	-	10.5	2.78	10.5	2.78	937
RAS underground			3.2	2.66	3.2	2.66	275
SRX			1.3	0.70	1.3	0.70	30
Total	**-**	**-**	**15.0**	**2.58**	**15.0**	**2.58**	**1,242**

Table 5 Ore Reserve Statement

Note 1: RAS Open pit cut-off grade 0.5g/t at $US2,000/oz Au price
Note 2: RAS Underground cut-off grade 1.7g/t at $US1,650/oz Au price
Note 3: SRX Open pit cut-off grade 0.30 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 30 June 2025, estimated under the supervision of Damian Spring (FAusIMM).
Note 6: Approved consents and required permits are yet to be granted to enable mining of the RAS and SRX deposits.

Estimation Methodology

Ore Reserves have been estimated using conventional pit and stope optimisation software, applying economic and geotechnical constraints consistent with the Updated PFS parameters described herein. The estimates incorporate modifying factors such as mining dilution, ore loss, metallurgical recovery, operating costs, and gold price assumptions.

Classification Criteria

Only Indicated Resources were converted to Probable Ore Reserves in accordance with JORC Code guidelines. The classification reflects the confidence in the geological model, data spacing, and continuity of mineralisation, as well as the modifying factors applied.

Cautionary Statement

The Updated Pre-Feasibility Study June 2025 (PFS) presented in this announcement assesses the potential development of the Bendigo-Ophir Gold Project in New Zealand, based on updated Mineral Resource Estimates as of March 2025, revised pit staging and a re-optimised process plant design. Of the Mineral Resources planned for extraction under the production schedule, approximately 93% are classified as Probable Ore Reserves, with the remaining 7% comprising Inferred Resources over a mine life of 13.8 years.

The Company considers that it has a reasonable basis for disclosing a production target that includes a minor proportion of Inferred Resources mined as a consequence of mining the Indicated Resources. These Inferred Resources are not material to the overall economic viability of the Project, and the feasibility of the development case does not depend on their inclusion. Nonetheless, it is acknowledged that there is a lower level of geological confidence associated with Inferred Resources and no certainty that further exploration will convert them to Indicated or Measured categories, or that the production target will ultimately be achieved.

All Ore Reserves underpinning the production target are based solely on Indicated Mineral Resources, and have been estimated in accordance with the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code).

This announcement contains forward-looking statements, including statements regarding production targets, project development, and financial forecasts. Santana Minerals Ltd considers it has a reasonable basis for these statements, supported by the modifying factors and assumptions set out throughout this PFS update. However, actual results may differ materially due to various risks and uncertainties, and investors are cautioned not to place undue reliance on these statements.

To achieve the outcomes outlined in the Updated PFS, a capital investment of approximately A$276 million is estimated. Santana believes it has reasonable grounds to assume that the required funding will be available when needed, based on indicative engagement with potential debt and equity providers. However, there is no certainty that such funding will be secured on favourable terms or at all. The Company may also consider alternative financing strategies, including the potential sale or joint venture of part of its interest in the Project or the monetisation of future revenue streams.

This announcement has been authorised for release by Santana's Board of Directors.

Enquiries:

Damian Spring
Exec. Director & CEO
dspring@santanaminerals.com

Sam Smith
Exec. Director Corp Affairs & IR
ssmith@santanaminerals.com

Webinar

A conference call will be held at 11am AEST/1pm NZST on Tuesday 1 July 2025 for Executive Director & CEO Damian Spring and Executive Director Sam Smith to present the outcomes of the study. To listen live please follow the link below at the specified time:

https://events.teams.microsoft.com/event/c9aa85b8-e0c1-4aed-84ef-a7cc2aa55c75@398b2b9c-83ee-4b15-8c8b-040598f65e24

Updated Pre-Feasibility Study – June 2025

Since the release of the initial Pre-Feasibility Study (PFS) in November 2024, the Company has undertaken a re-evaluation of its development approach for the Bendigo-Ophir Gold Project. This update is underpinned by the March 2025 Mineral Resource Estimate, which significantly improved geological confidence in the high-grade (HG1) domain through refined interpretation and tighter geological controls. As a result, the project has shifted from a higher-tonnage mining strategy to a more selective and value-driven project.

The November 2024 PFS design assumed a larger-scale mining operation with an aggressive pre-strip of approximately 15 million bank cubic metres (bcm) of overburden as the critical path to gold production. This approach necessitated substantial early capital deployment and generated a large low-grade stockpile, creating downstream challenges in maintaining mill feed quality and balancing waste and ore throughout the mine life.

In contrast, the Updated PFS adopts a selective and staged mining strategy, comprising manually designed open pits with progressive cutbacks that remain within the November 2024 PFS ultimate pit shell. This approach enables a more efficient sequence of material movement, reducing pre-strip volumes by 8.5 million bcm, and improves strip ratio balance across mining periods. The project life is extended by four years while still delivering approximately 120,000 ounces of gold per annum during the peak mining term. The revised mine plan also better aligns mining activity with near-surface portions of the deposit, allowing for earlier cash flow generation and a smoother operational ramp-up. While cash flow is generated in month 15 after the commencement of construction (as opposed to month 23 in the November 2024 PFS) the payback is slightly longer at 33 months as opposed to 30 months if the same current gold price is applied to the November 2024 PFS scenario.

To support this refined approach, the process plant has been redesigned from a 1.5Mtpa circuit in the November 2024 PFS, to a 1.2Mtpa configuration featuring three-stage crushing and a single ball mill. This revised design is better suited to the selectively mined, higher-grade ore from the HG1 domain and reduces comminution sensitivity to variable ore hardness, while also lowering capital and operating complexity. The plant layout has been designed to retain optionality for a future expansion to 1.8Mtpa by the addition of an extra ball mill to allow for any strategic need to accelerate gold production.

The revised strategy also allows the underground mine to be deferred to Year 7 of production. Deferring the underground extends the mine life and reduces pressure on the project during its ramp-up phase. Open pit grades continue to be comparable to the underground up to this time.

This Updated PFS represents the next phase of the Company's development strategy, incorporating both open pit and underground mining as part of the integrated mine plan. It provides the technical and economic basis to support financing discussions and marks a key milestone in advancing the Bendigo-Ophir Gold Project toward execution early in 2026.

Mineral Resource Estimate (MRE)

The Updated PFS incorporates the March 2025 Mineral Resource Estimate for the Rise and Shine (RAS) deposit, which replaces the previous MRE used in the Nov PFS. This revised estimate is based on an additional 7,060 metres of infill drilling and improved domaining, resulting in a 7% increase in Indicated grade (from 2.35g/t to 2.52g/t Au) and a 6.4% increase in contained ounces. In particular, the improved domaining focussed open pit stage design on the high-grade domain ore as a priority while extracting ore from other domains as a consequence. The updated geological interpretation and more selective domain modelling has enabled a higher-confidence resource, supporting detailed mine planning. The March 2025 MRE for RAS is reported below at 0.5g/t Au cut-off grade.

Deposit	Category	tonnes (Mt)	Au grade (g/t)	Contained Gold (koz)
RAS	Indicated	18.9	2.5	1,538
	Inferred	7.6	2.2	542
RAS Total	**Indicated and Inferred**	**26.5**	**2.4**	**2,080**

Table 6 RAS March 2025 MRE

Study Team

Minecomp Pty Ltd was engaged to develop a series of manually designed pit stages, with a primary focus on minimising pre-strip requirements, and extracting the high-grade HG1 domain.

SRK Consulting supported this process by conducting pit shell optimisations to validate the manual pit designs against economic shell models. SRK also provided technical input on dilution modelling and assisted in the scheduling of the final pit stages to ensure consistency with optimisation outcomes and practical mining considerations.

Frumla Ltd updated the underground mine designs that were originally developed for the November 2024 PFS, incorporating changes based on the March 2025 Mineral Resource Estimate.

Other key consultants contributing to the study include:

- MACA Interquip Mintrex (process plant design and cost estimate).
- Peter O'Bryan and Associates (geotechnical).
- Engineering Geology Ltd (tailings storage facility).
- Model Answer (financial modelling).

All cost inputs that follow are disclosed in New Zealand dollars (NZD).

Open Pit Reserve Estimation

The following technical inputs were updated prior to estimating the Ore Reserve:

Mining Methodology

The RAS deposit will be mined using conventional truck and backhoe excavator open pit methods, incorporating the following staged approach:

- Implementation of environmental controls and clearances.
- Clearing, grubbing, and stripping of near-surface material from the pit area, Tailings Storage Facility (TSF), and proposed engineered landform (ELF) areas, with suitable material stockpiled for later use in site rehabilitation.
- Pioneering works with a smaller fleet of articulated trucks in high-relief terrain to establish safe and efficient working benches for the production fleet.
- Grade control drilling to refine ore boundaries.
- Drill and blast operations in waste, blasting to 15m batter faces in two 7.5m blast benches, excavated in two flitches by a 250-tonne class excavator.
- Drill and blast operations in ore will include 7.5m blasts benches excavated in three flitches by a 120-tonne class excavator.
- Load and haul of ore using 120-tonne class excavators and 90-tonne rigid or 60-tonne articulated trucks, mining in 2.5m flitches to maintain ore selectivity.
- Haulage of ore to the Run-of-Mine (ROM) pad for stockpiling and subsequent crushing.

SMU Selection and Dilution Modelling

To support robust mine design and scheduling, the Mineral Resource model was regularised to a Selective Mining Unit (SMU) of 7.5m x 7.5m x 2.5m, balancing mining selectivity with operational efficiency for the revised fleet size. SMU sizes ranging from 5m to 15m were evaluated during the study, with the selected configuration representing an optimal trade-off between ore recovery and dilution control under expected operating conditions.

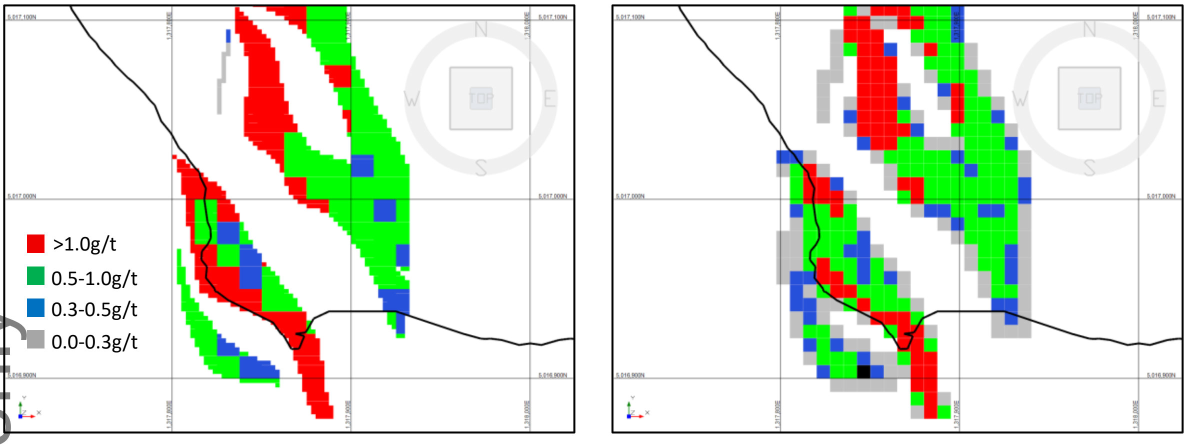

Figure 5 Geological Block Model Slice vs Regularised Model to 7.5m x 7.5m x 2.5m SMU

Dilution and ore loss assumptions were then applied consistently across the block model using the marginal cut-off grade (free on surface) of 0.3g/t as calculated in the November PFS, achieving the following loss and dilution results.

Model	Unit	Cut-Off Grades		
		0.3g/t	0.5g/t	1.0g/t
Resource Model				
Tonnage	(Mt)	12.0	11.5	8.3
Grade	(g/t Au)	2.81	2.90	3.74
Contained Metal	(koz Au)	1,078	1,071	996
7.5m x 7.5m x2.5m Diluted Model				
Tonnage	(Mt)	11.8	11.2	8.3
Grade	(g/t Au)	2.63	2.75	3.46
Contained Metal	(koz Au)	1,001	993	923
Dilution	(%)	6.6	5.6	8.2
Recovery	(%)	92.9	92.7	92.7

Table 7 Ore Loss and Dilution, Interrogation on Block Model

Due to the geometry and high-grade nature of the deposit, ore loss and dilution are relatively insensitive to changes in cut-off grade as seen in Table 7. Additional dilution studies were conducted in-house by the Company's geology team using flitch-by-flitch analysis to validate the final SMU selection. The chosen SMU size delivered moderate dilution levels considered appropriate for gold recovery while preserving overall resource integrity.

Pit Optimisation Parameters

Pit optimisations were conducted using the regularised and diluted block model, applying updated assumptions shown in Table 8. Topographic constraints were also included to ensure the ultimate pit shell did not encroach into Shepherds Creek where mine infrastructure is located, where underground mining in planned, and where incremental strip ratio increases dramatically due to rising topography to the north of the creek.

The optimisations helped define the economic limits of the deposit and validated manual pit designs. A base-case pit shell at a revenue factor of 0.75 (RF0.75) delivered the highest pre-capital NPV and was closely aligned to the manually designed pits and therefore used as the reference case for validating the final pit designs.

Parameter	Unit	Rate
Gold price	USD/oz	$2,000
Exchange rate	USD NZD	US0.56
NZ gold price	NZD/oz	$3,585
Overall met recovery	%	93%
Royalties (average)	%	8
Transport/refining	NZD/oz	8
Discount rate	%	8
Mining (MCAF adjusted)	NZD/t	3.75
Processing	NZD/t	24.5
TSF	NZD/t	1
Crusher feed	NZD/t	1.25
G&A	NZD/t	4

Table 8 RAS Pit Optimisation Inputs

Figure 6 below shows strong alignment between the optimised ultimate pit shell (RF0.75) and the manually designed pits. At RAS South, opportunities remain where recent drilling and updated geological interpretation have improved confidence in the structural controls on lower-grade mineralisation, which may be economically extracted beneath Stage 1.

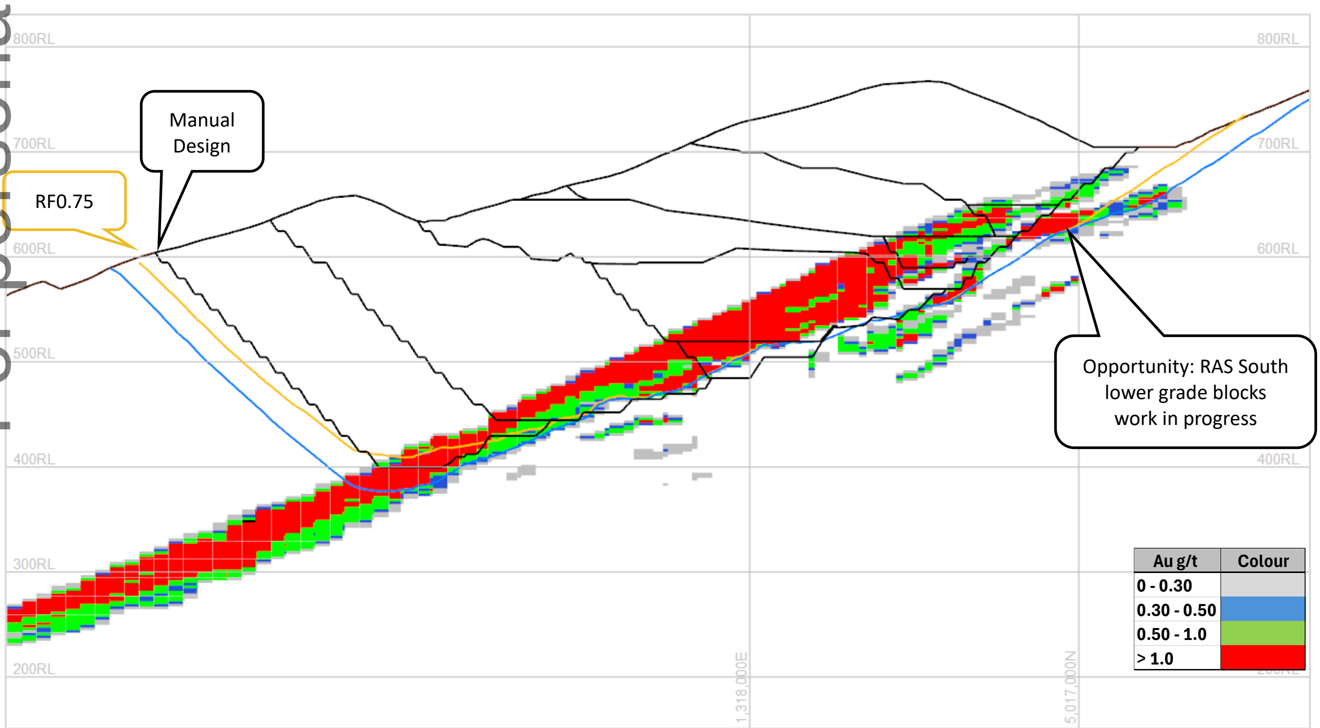

Figure 6 Manual Pits Design vs Optimisation (View East)

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Geotechnical Parameters

Geotechnical conditions influencing wall stability in the proposed open pit were assessed by Peter O'Bryan and Associates, based on current geological interpretations, drill core data, and their experience in geotechnical assessment and peer review in comparable geological settings. Design recommendations for the RAS open pit are:

Wall	Aspect (1) (°)	Unit	IRA (2) (°)	BFA (3) (°)	Berm Width (m)	Bench Height (m)
Southwest	350 to 065	All	36.8	50	7.5	15
West	065 to 160	All	34.8	50	9	15
Northeast	160 to 235	All	45.7	60	6	15
East	235 to 350	All	42.5	60	7.5	15

(1) Slope aspect measured as the direction the wall dips towards.
(2) Inter-ramp angle.
(3) Batter face angle.

Table 9 RAS Geotech Parameters

RAS Manual Pit Design and Validation

The updated RAS pit stages were manually designed by Minecomp Pty Ltd with a focus on minimising pre-strip by targeting near-surface ore at the southern end of the deposit. While this area carries a lower average grade than the high-grade HG1 domain approximately 100m to the northeast, it enables the early establishment of sustainable ore production and supports a progressive ramp-up in gold output. This approach allows subsequent bulk stripping to occur in parallel without disrupting productivity. The designs incorporate updated geotechnical parameters, including minor wall angle improvements achieved since the November 2024 PFS. Designs also considered haulage efficiency and operational practicality. The reported diluted inventories presented below are derived from interrogation of the pit shell solids at 0.5g/t cut off.

Stage	Total Diluted Inventory Rounded Totals			% Indicated	Waste Volume
	Tonnes (t)	Grade (g/t)	Ounces (oz)		(bcm)
Stage 1	747,000	1.43	34,400	93%	7,930,000
Stage 2	817,000	2.14	56,300	100%	4,117,000
Stage 3	-	-	-		7,350,000
Stage 4	730,000	3.86	90,500	100%	5,135,000
Stage 5	3,573,000	3.46	397,700	96%	6,010,000
Stage 6	3,768,000	2.46	298,000	85%	30,895,000
Stage 7	1,596,000	2.29	117,000	93%	14,138,000
Total	**11,231,000**	**2.75**	**993,500**	**93%**	**75,575,000**

Table 10 RAS Mineral Inventory by Stage (totals include rounded figures)

Figure 7, Figure 8 and Figure 9 below show the seven pit stages containing the total gold inventories outlined in Table 10 above. Notable improvements in the revised ultimate pit shell exist to the southeast where the higher elevation of Battery Hill had previously increased stripping costs.

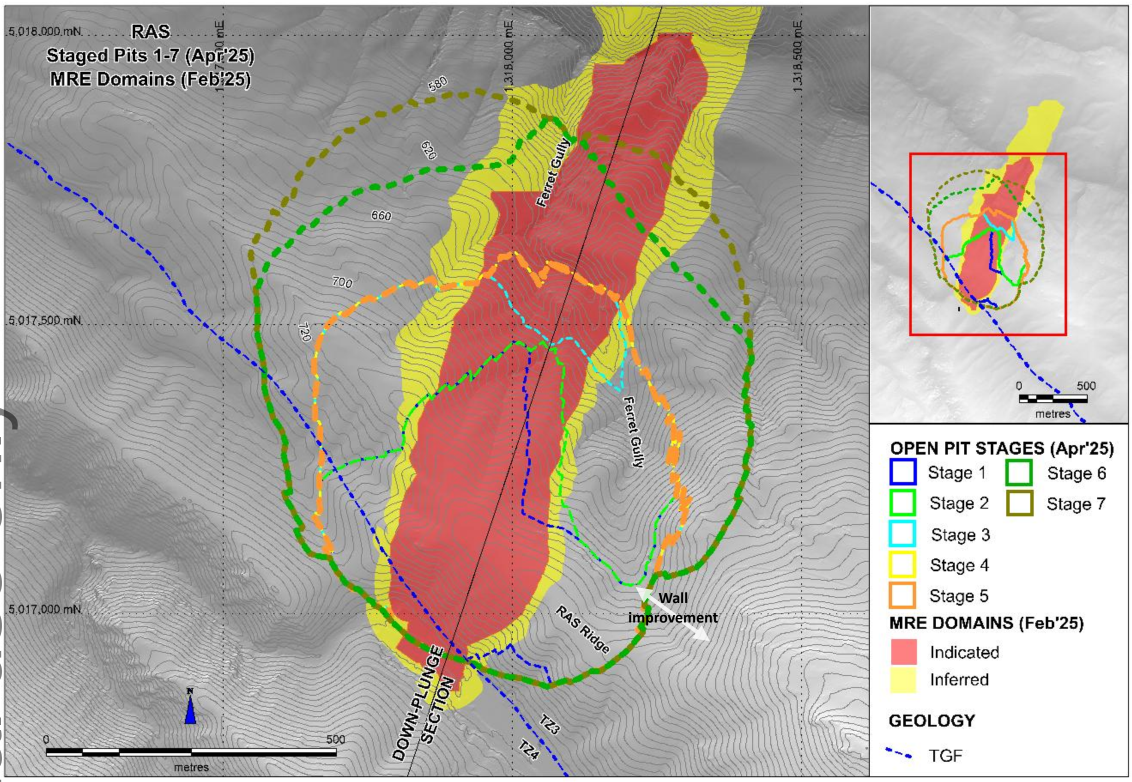


Figure 7 Plan View of Seven RAS Pit Stages vs Nov 25 PFS Ultimate Shell

Figure 8 below shows the seven pit stages in long section, with Stage 1 now focusing on Indicated, near surface ore at RAS South, with subsequent cut-backs to the north vectoring on HG1, as shown in Figure 9 (purple domain). The pre-strip volume required to access sustainable ore feed is now 6.5 million bcm.

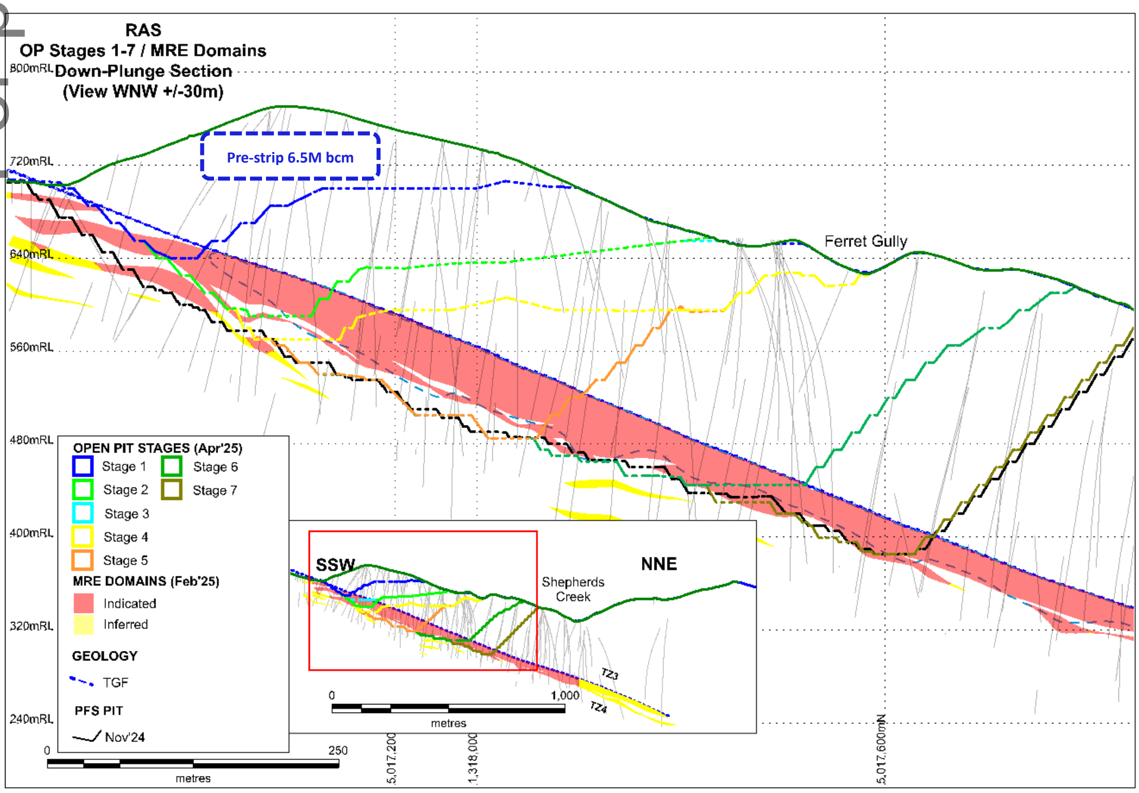


Figure 8 Long Section of RAS Pit Stages

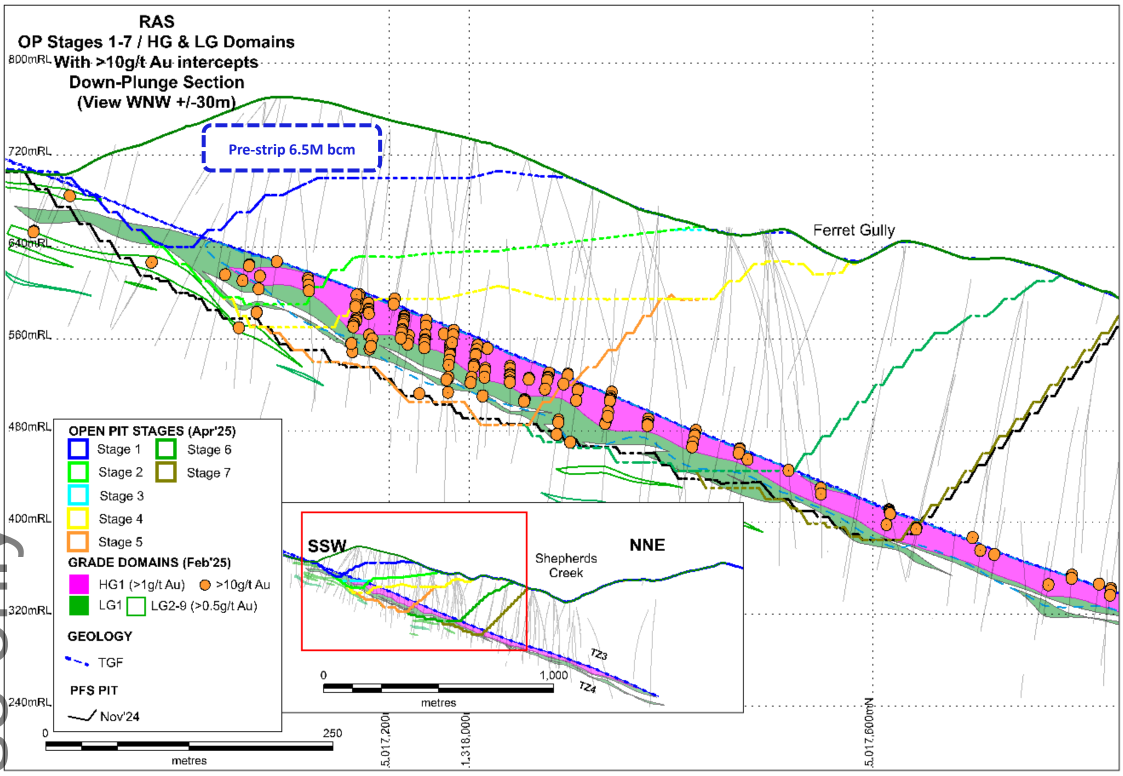
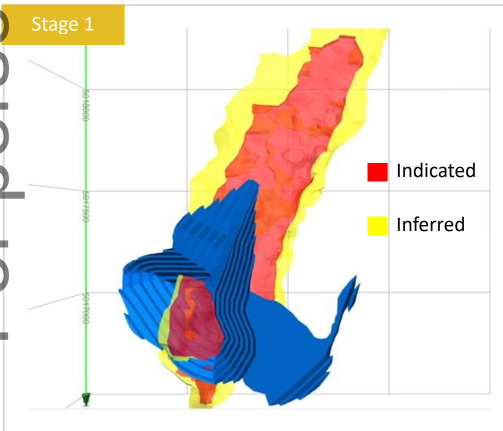


Figure 9 Long Section of RAS Pit Stages Proximal to HG1 Domain

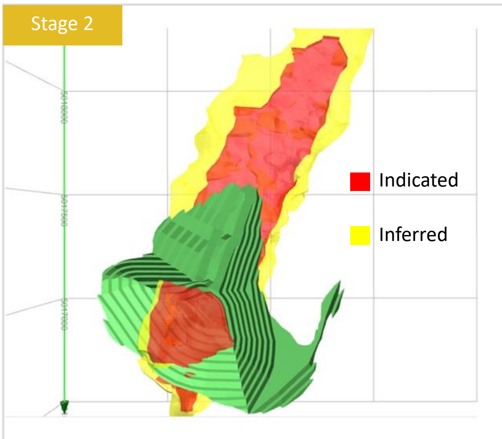
Figure 10 illustrates the seven pit stages in detail, including ramp designs for access, and their spatial relationship to Indicated (red) and Inferred (yellow) resources.



The Stage 1 Pit is designed to access near-surface resource blocks at a grade of ~1.4g/t at RAS South. This stage will be completed by excavating a total of ~7.9 million bcm of TZ3 waste (6.5 million bcm defined as pre-strip) to expose ore for commissioning the plant.

The TZ3 bulk waste overburden will be stripped off the deposit until exposure of the Thomson Gorge Fault (TGF) interface, then the ore zone beneath the TGF will undergo detailed grade control practices and then be selectively mined with 100t excavators and 90t rigid dump trucks or 60t articulated dump trucks.

The gold resource at RAS South inside Stage 1 includes 747kt at 1.43g/t of diluted mineral resources.



The Stage 2 Pit incorporates a new ramp to maintain access to the pit floor as the Stage 1 ramp is mined out.

The pit floor is mined down to the 590RL providing access to higher grade resource blocks. Stage 2 provides access to future pit stages (Stage 3 and 4).

Stage 2 contains 4.1M bcm of waste rock and 817kt at 2.14g/t of diluted mineral resource.

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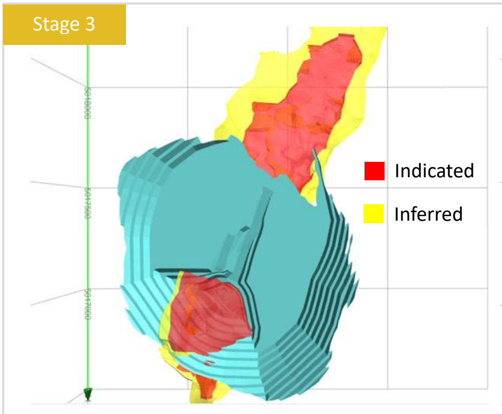
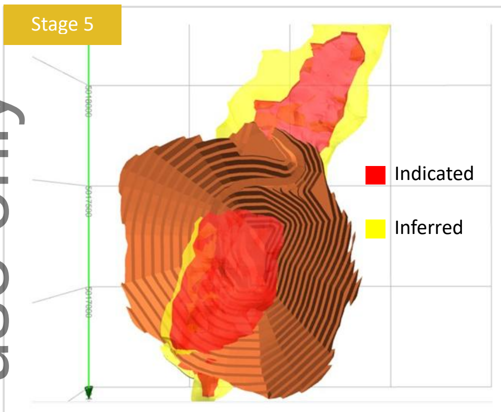


The Stage 3 Pit is a deferred striping activity excavated concurrently with Stage 2, with scheduling focussed on continuous exposure of ore.

The Stage 3 Pit is essentially the deferred strip out of the northern section area to its full dimension in preparation for exposing the HG1 as it dips away from the pit.

Waste stripping is focused on ultimately establishing a flat floor at the 655RL, to enable easy access into the Stage 4 pit.

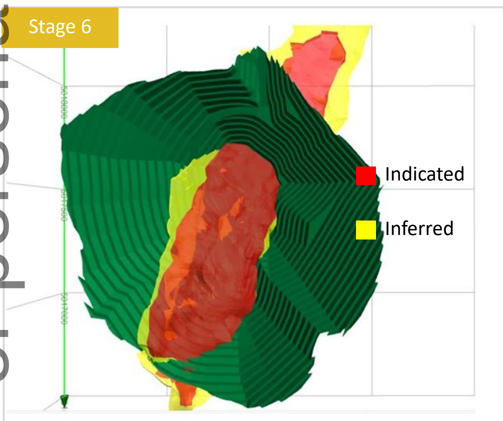
The Stage 3 Pit mines 7.3M bcm of waste rock to expose Stage 4 ore.



The Stage 5 Pit mines the western side of the high-grade domain.

Mined to the 520RL, Stage 5 contains 6.1M bcm of waste rock and exposes a diluted 3.5Mt at 3.46g/t of mineral resources.

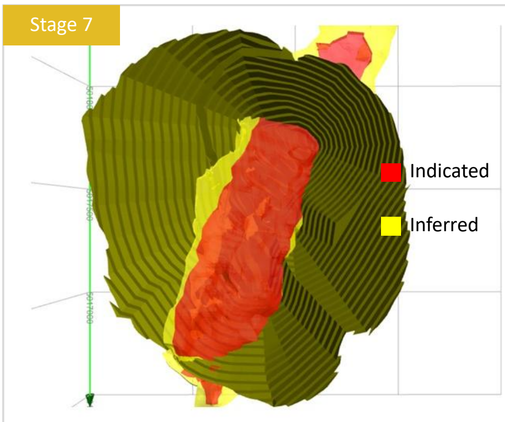
This ore will be progressively exposed and mined to fulfill ore processing requirements whilst additional waste stripping in future stages exposes additional open pit ore supply.



Stage 6 is the penultimate shell cutting back toward the northwest, preparing the relevant geometries and ramp accesses for the ultimate pit shell in Stage 7.

This shell mines 3.7Mt of ore at 2.46g/t. Total waste to be excavated is 31M bcm.

Stage 6 is mined down to a base level of 445RL.



Stage 7 is the ultimate open pit shell, mined down to the 385RL.

This shell is topographically constrained to the north by Shepherds Creek, where excavation of the adjacent ridgeline would result in a higher stripping ratio. As a result, future gold production beneath Shepherds Creek is expected to be sourced via underground mining. Further, Shepherds Creek has been nominated as the logical area for mine infrastructure and haul roads.

Stage 7 contains a diluted resource of 1.6Mt of ore at 2.29g/t, with 14M bcm of waste rock. Final ramp access is situated in the western wall, exiting the pit to the north, towards Shepherds Creek.

Figure 10 Pit Shells w/Ramps Showing IND/INF

RAS Open Pit - Total Mineral Inventory

An open pit production schedule for the RAS deposit has been developed based on the defined pit stages, final pit shells, and waste rock dump and ROM destinations. The resulting pit inventory, which underpins the schedule, is presented in Table 11 below.

Type	Unit	Quantity
Total rock	tonnes	211,500,000
Total waste	tonnes	200,250,000
Ore tonnes	tonnes	11,231,000
Average gold grade	g/t	2.75
Total contained ounces	**oz**	**993,330**

Table 11 RAS Pit Material Movements

Basis of RAS Reserve Estimation

In the November 2024 PFS, a marginal break-even cut-off grade of 0.3g/t Au was applied for Reserve estimation, reflecting the processing and operating costs at that time. For the Updated PFS, the same marginal cut-off grade of 0.3g/t Au was retained as it remains valid under current cost assumptions and supports comparability between studies. However, a more selective approach has been adopted in the mine schedule, with only material grading above 0.5g/t Au now classified and scheduled as ore.

Grade bins for ore destinations are defined as follows:
- Mill Feed = >0.5g/t sent to the ROM.
- Very Low-Grade = 0.3 – 0.5g/t sent to be stockpiled at the ELF (left unprocessed).
- Mineralised Waste = <0.3g/t sent to a stockpile at the ELF.

The parameters forming the basis for mine scheduling are as follows:
- Nameplate mill feed of 1.2Mtpa.
- Mill feed commences with an allowance for commissioning ramp-up.
- Stockpiles are managed at the ROM for optimisation of annual production.
- Higher grade underground mill feed is preferentially treated from Year 7.
- Mill feed includes both the Probable Reserves mined plus Inferred Resource mined as a consequence of accessing the Reserves (92% of all mill feed in classified as Reserves).
- All soil and brown rock materials are sent to their own dedicated stockpile areas for utilisation in rehabilitation activities.
- Mineralisation below 0.5g/t Au is separately stockpiled at the ELF.

RAS Material Movements by Stage

Total material movements by stage for RAS are illustrated in Table 12. Approximately 6.5 million bcm of waste will be mined over the first 11 months to expose the initial 100kt of ore, which will be stockpiled ahead of plant commissioning. The process plant is expected to enter commissioning in month 12 of mining, with a sustainable ore supply available to commence mill feed shortly thereafter (corresponding to month 15 from the start of construction).

	Waste Tonnes	Ore Tonnes	Strip Ratio
Pre-Strip	17,491,000		
Stage 1	3,168,000	747,000	4.2:1
Stage 2	10,998,000	817,000	13.5:1
Stage 3	18,988,000	-	Deferred stripping
Stage 4	13,719,000	730,000	18.8:1
Stage 5	16,299,000	3,573,000	4.6:1
Stage 6	82,031,000	3,768,000	21.7:1
Stage 7	37,552,000	1,596,000	23.5:1
Total Operating Strip	**163,767,000**	**11,231,000**	**14.6:1**

Table 12 Strip Ratios

Excluding the pre-strip and deferred waste stripping campaign in Stage 3, the operating strip ratio averages 14.6:1 over the life of production, carried by an average grade of 2.75g/t.

SREX (SRX) Open Pit – Basis of Reserve Estimate

The SRX satellite pit optimisation parameters, pit designs and schedule have not changed from the November 2024 PFS, with the same material movements and mineral inventory defined at a 0.3g/t cut-off grade and featured in Table 13 below. The SRX mining schedule is underpinned by Indicated Resources, which comprise 99% of the total mined ounces, providing a high level of confidence in the resource base.

	Unit	Quantity
Total rock mined	tonnes	7,344,000
Total waste	tonnes	5,916,000
Total ore	tonnes	1,428,000
Average gold grade	g/t	0.68
Total contained ounces	**oz**	**30,674**
Stripping Ratio	(waste t: ore t)	4.1

Table 13 SRX Open Pit Material

RAS Underground Mining – Basis of Estimate

Since the November 2024 PFS, the underground mine design has undergone refinement largely in response to the updated geological model released in March 2025 which features a redefined Indicated Resource boundary. These changes have enabled targeted improvements to the planned underground extraction strategy.

To update the mineable inventory for underground production, a break-even cut-off grade of 1.7g/t Au was applied in the Mine Shape Optimiser, consistent with the approach taken in the November 2024 PFS. This cut-off grade was originally derived based on underground mining and processing operating costs, metallurgical recovery assumptions, and a conservative gold price. Despite the Updated PFS adopting a higher base-case gold price of A$3,500/oz (NZD$3,844/oz) and reflecting slightly reduced processing costs due to the revised 1.2Mtpa throughput rate (previously factored at 1Mtpa at the optimisation stage), the 1.7g/t cut-off remains appropriate and conservative for defining mineable shapes and scheduling underground production.

RAS Underground Mining Method

The underground mining method is largely unchanged from the November 2024 PFS assumption and remains as a fully mechanised system utilising ramp haulage, with ore extracted via longhole stoping, with voids supported by cemented paste fill. Twin portals provide access to parallel ramp developments, creating a network for haulage, fresh air intake, and return ventilation.

Figure 11 below shows the underground mine designs with twin declines (green and pink) for primary ventilation and haulage, with portal establishment in Shepherds Creek. Ore drives in blue will be longitudinally stoped and filled with paste from top drive locations (grey) where accessible. Stope voids that do not have top drive access will be paste-filled through service holes drilled from the ore drive into the stope void.

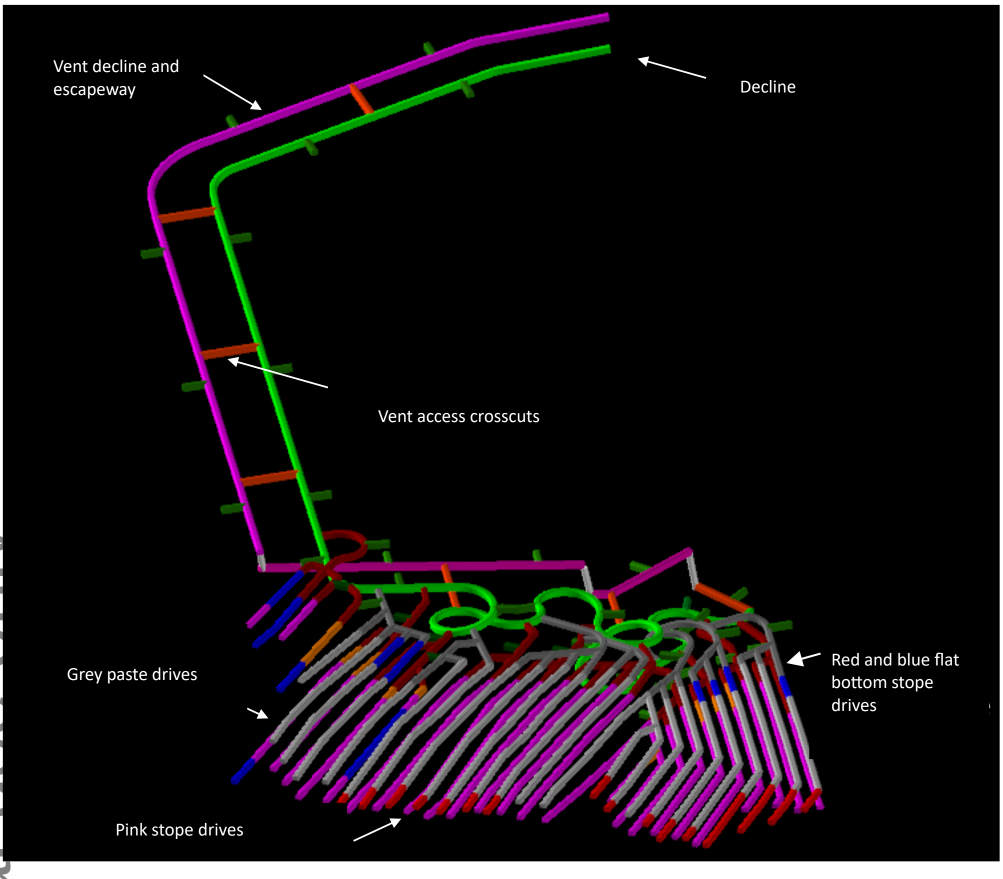


Figure 11 RAS Underground Designs

The underground mining operation is now scheduled to commence development in Year 6, with mill feed being preferentially fed in Year 7 of production.

Underground Mineral Inventory

Key design modifications influencing the increased mineral inventory include:

Adjustment of Paste Fill Infrastructure

As seen in Figure 12, the upper paste fill drives have been realigned from the original design in the TZ3 zone and repositioned into the TZ4 zone. This change is intended to enhance geotechnical control and paste delivery efficiency in light of revised stope sequencing and block model geometry.

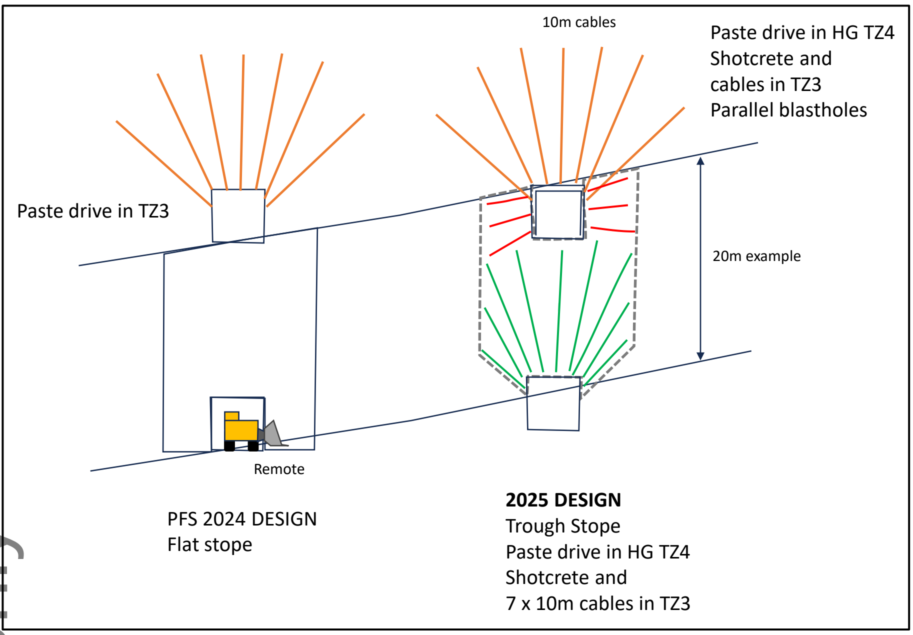


Figure 12 Top Drive (Paste Drive) Relocation from TZ3 to TZ4

Reprofiling of Stope Shapes

Also depicted in Figure 12, the base of the planned stopes have been modified from a flat floor to a trough-shaped configuration. This change improves ore recovery, reduces dilution risk, and better aligns with the updated geometry of high-grade mineralisation.

Updated Resource Model and Design Alignment

The new stope layout reflects both the updated block model and redefined Indicated Resource limits as shown in Figure 13. The revised design ensures all scheduled underground stoping occurs within higher grade ore blocks as shown in Figure 14.

Figure 13 Change in Indicated Resource Boundary

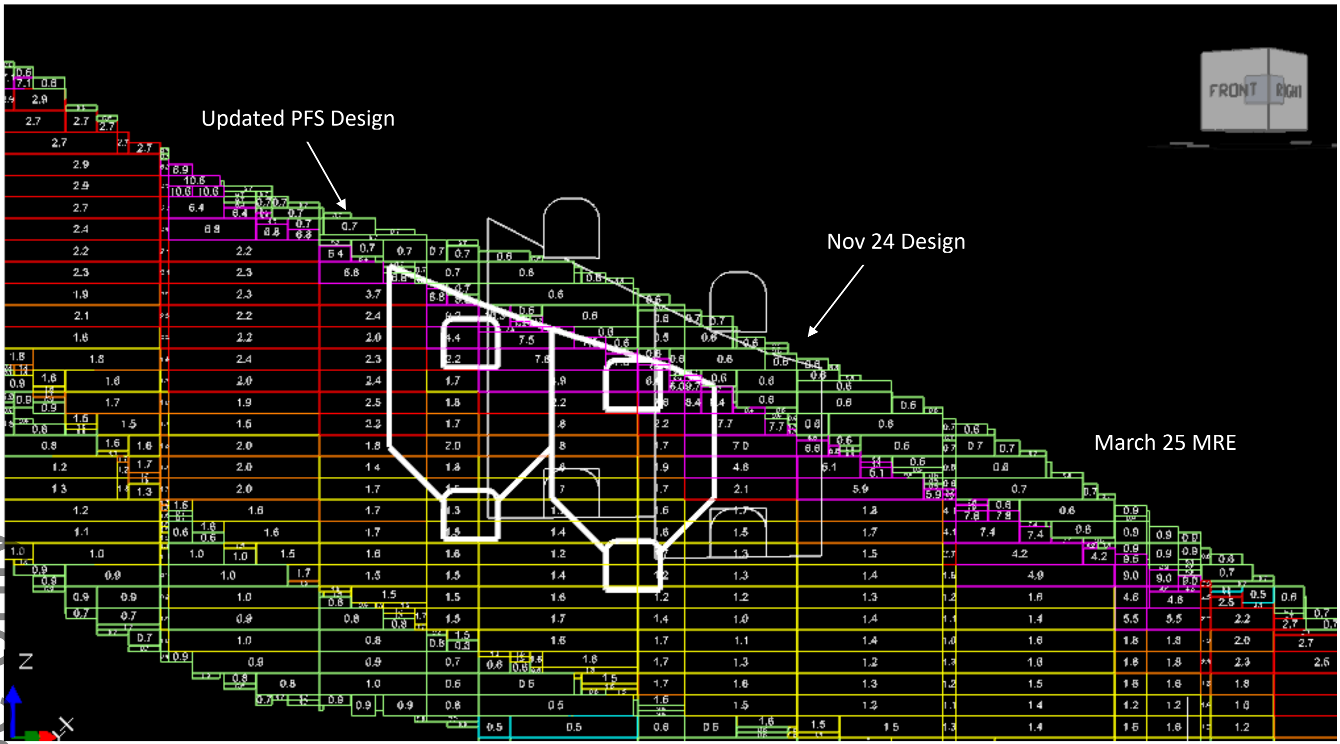


Figure 14 New UG Designs on March 2025 Geological Model

The mineral inventory and annual production schedule for the underground development is summarised in Table 14 below.

Year		Total	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7
Indicated									
Ore tonnes	t	3,217,966	400,200	597,876	691,111	575,518	185,321	567,663	200,277
Grade	g/t	2.66	2.36	2.51	3.12	2.84	3.13	2.40	1.87
Ore oz	oz	275,012	30,417	48,260	69,353	52,540	18,665	43,741	12,038
Inferred									
Ore tonnes	t	570,028	21,480	28,042	75,592	54,791	383,403	-	6,720
Grade	g/t	2.24	0.86	1.47	1.94	1.65	2.53	-	1.70
Ore oz	oz	41,133	597	1,328	4,706	2,903	31,231	-	368
Total									
Ore tonnes	t	3,787,994	421,679	625,918	766,704	630,309	568,724	567,663	206,997
Grade	g/t	2.60	2.29	2.46	3.00	2.74	2.73	2.40	1.86
Ore oz	oz	316,145	31,014	49,588	74,059	55,442	49,896	43,741	12,406
Total Indicated	**%**	**87%**	**98%**	**97%**	**94%**	**95%**	**37%**	**100%**	**97%**

Table 14 RAS Underground Inventory

RAS Open Pit, RAS Underground and SRX Production Targets

Figure 15 below shows the sources of mill feed over the initial mine life from each mining area.

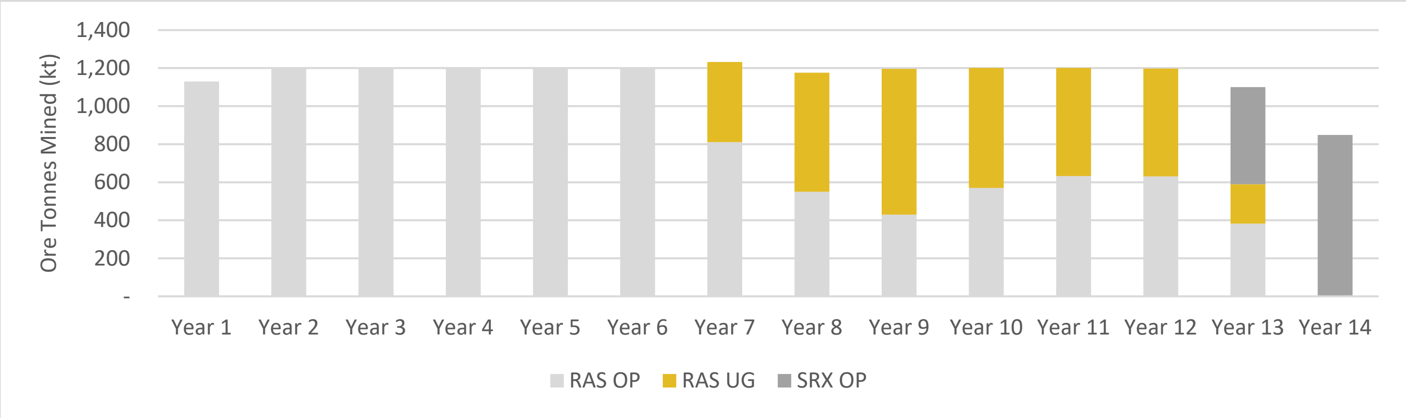


Figure 15 Open Pits vs Underground Mill Feed

Table 15 below shows production targets from the RAS open pit, RAS underground and SRX that underpin financial forecasts. Of the production forecast included in financial projections, 93% is in the Indicated Resource category. Reserve Estimates only include Indicated Resources.

	Unit	LoM	Yr0	Yr1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
Open pit mining RAS & SRX																	
Material moved	kt	218,821	18,471	32,592	31,648	32,898	33,239	22,961	12,787	11,628	7,854	1,270	1,407	2,868	1,325	4,443	3,430
Ore tonnes	kt	12,591	100	1,130	1,200	1,200	1,200	1,200	1,196	811	550	429	571	632	631	893	849
Grade	g/t	2.53	1.02	1.75	3.17	3.31	3.30	3.32	2.77	3.20	2.40	2.24	2.25	1.61	2.52	1.63	0.67
Contained gold	koz	1,024	3	64	122	127	127	128	106	83	42	31	41	33	51	47	18
Underground mining																	
Ore tonnes	kt	3,788								422	626	767	630	569	568	207	
Grade	g/t	2.6								2.29	2.46	3.00	2.74	2.73	2.40	1.86	
Contained gold	koz	316								31	50	74	55	50	44	12	
Processing after Stockpile Management																	
Tonnes	Kt	16,379		1,130	1,200	1,200	1,200	1,200	1,200	1,228	1,176	1,196	1,201	1,201	1,198	1,200	849
Grade	g/t	2.54		1.75	3.14	3.31	3.30	3.32	2.77	2.90	2.43	2.73	2.51	2.14	2.46	1.59	0.67
Contained gold	Koz	1,340		64	121	128	127	128	107	114	92	105	97	82	95	61	18
Recovery	%	93.1%		92.1%	93.8%	94.0%	94.2%	93.7%	93.2%	93.4%	93.2%	93.4%	93.2%	92.7%	93.2%	90.7%	81.8%
Gold production	**koz**	1,248		59	113	120	120	120	99	107	86	98	90	76	88	56	15

Table 15 Life of Mine Production Schedule (minor grade discrepancies due to stockpile management)

CIL Processing Plant – Design Update

The process plant for the RAS gold deposit is designed to deliver high gold recoveries with low operating costs through a conventional, industry-proven flowsheet. The updated plant design supports a nameplate throughput of 1.2Mtpa of primary ore, with a mechanical availability of 91.3%. Equipment selection prioritises reliability, ease of maintenance, and suitability for duty, while the compact plant layout minimises construction costs and facilitates operator access.

The plant includes a three-stage crushing circuit feeding a single ball mill, with classification via large-diameter cyclones. The gravity recovery circuit comprises dual centrifugal concentrators and an Intensive Leach Reactor for gold-rich concentrates, complemented by a carbon-in-leach (CIL) circuit with six adsorption tanks and a total residence time of 24 hours. Downstream processing includes an AARL elution circuit with parallel electrowinning cells, followed by sludge filtering and smelting to produce doré.

Environmental controls include an air/SO_2 cyanide destruction circuit and a ferric chloride precipitation process for arsenic removal. Tailings are pumped via a five-stage centrifugal pumping system to the TSF. The process plant is supported by sufficient automation to minimise operator intervention, while maintaining full manual override capability when required.

The plant flowsheet is shown in Figure 16.

Key Processing Design Criteria

Run of Mine (ROM) and Crushing Circuit

The ROM pad will contain up to 30 days of stockpiled ore to provide a buffer between the mine and the plant. The three-stage crushing circuit is designed to operate 24 hours per day at 70% utilisation, at a feed rate 131% above the mill feed rate. Ore will be stored in a fine ore bin to allow the downstream plant to continue operating during crushing circuit downtime.

A ROM grizzly with a 700mm aperture has been selected to minimise oversize material entering the ROM bin and causing downstream blockages. An apron feeder draws material from the ROM bin to the primary jaw crusher, selected due to moderate UCS values and ore abrasiveness. The single toggle crusher provides greater capacity than a similarly sized double toggle unit.

Crushed ore is conveyed to a double-deck sizing screen. Oversize from the top and bottom decks is directed to secondary and tertiary crushers respectively, while undersize is conveyed to the fine ore bin. Crushed product is recombined on the screen feed conveyor before final sizing. Ore is withdrawn from the fine ore bin by a variable speed belt feeder and delivered to the mill feed conveyor.

Milling

A single ball mill has been selected to reduce the crushed product to a circuit P80 of 106 µm.

Classification

Large-diameter (380 mm) cyclones are used for classification to minimise wear and reduce the potential for spigot blockages from coarse ball mill discharge material.

Gravity Concentration

Testwork indicates gravity gold recoveries of up to 32% are achievable. Two centrifugal concentrators process cyclone underflow slurry. Gravity concentrate is collected in a hopper in the gold room and treated in an Intensive Leach Reactor with dedicated electrowinning.

The reactor uses high-intensity cyanidation for efficient recovery of gold and silver from concentrate. Separate tanks and pumps are used to isolate the system for metallurgical accounting and prevent interference with the main elution circuit.

Trash Screens

A vibrating trash screen removes oversize material ahead of the leach and adsorption circuit. While minimal trash is expected from the ore, effective screening supports good carbon management.

Leach and Adsorption

Testwork indicates fast leach kinetics and minor preg-robbing. Approximately 85–90% of CIL gold extraction occurs within 4 to 8 hours. The CIL circuit includes six adsorption tanks, providing sufficient residence time and operational flexibility. Due to preg-robbing, a hybrid leach circuit is not viable, resulting in lower carbon loadings and increased elution frequency.

All tanks are the same size and arranged to allow maintenance on any tank without affecting overall circuit performance. Total residence time is 24 hours at 45% w/w density.

Elution

An AARL elution circuit has been selected due to the excellent supply and quality of raw water. This circuit separates elution from electrowinning, allowing greater flexibility and the ability to process additional batches if needed. A six-tonne carbon batch size has been nominated, based on calculated carbon movements required to accommodate the grade variability forecasted in the production plan. Under normal operating conditions, seven elution cycles are expected per week, with the gold room operating seven days a week.

Three parallel electrowinning cells, each with 12 cathodes, are proposed to achieve high pass efficiency (greater than 90%) and ensure low gold tenor in the spent electrolyte returning to the strip solution tank. This high efficiency results in a near-barren solution returning to the column, reducing both gold recirculation and the number of elution cycles required to reach target barren carbon grades. Each elution–electrowinning cycle is expected to take 8 to 12 hours, enabling additional cycles to be conducted during the week if necessary.

A sludging cell design has been adopted to simplify cathode handling. Gold sludge will be filtered using a vacuum pan filter, dried in an oven, and then smelted to produce doré.

Cyanide Destruct

An air/SO_2 circuit is used for cyanide destruction, offering lower operating costs and safer reagent handling compared to alternatives like Caro's acid. Testwork confirms its suitability, with WAD cyanide levels reduced to below 30ppm at the TSF discharge point.

Naturally Occurring Arsenic Removal

Arsenic that is naturally present in the ore mineralisation is removed via ferric chloride precipitation, forming a stable ferric arsenate species. This method was selected for its demonstrated amenability in testwork and its environmental stability.

TSF Pumping

Tailings slurry from the arsenic precipitation tank is pumped to the TSF using five centrifugal pumps in series.

Flow Sheet and Plant Designs

Figure 16 shows the flow sheet diagram of the process plant, while Figure 17 shows the general arrangement drawing of the plant infrastructure situated in Shepherds Creek. Figure 18 shows the plant location in relation to the project layout.

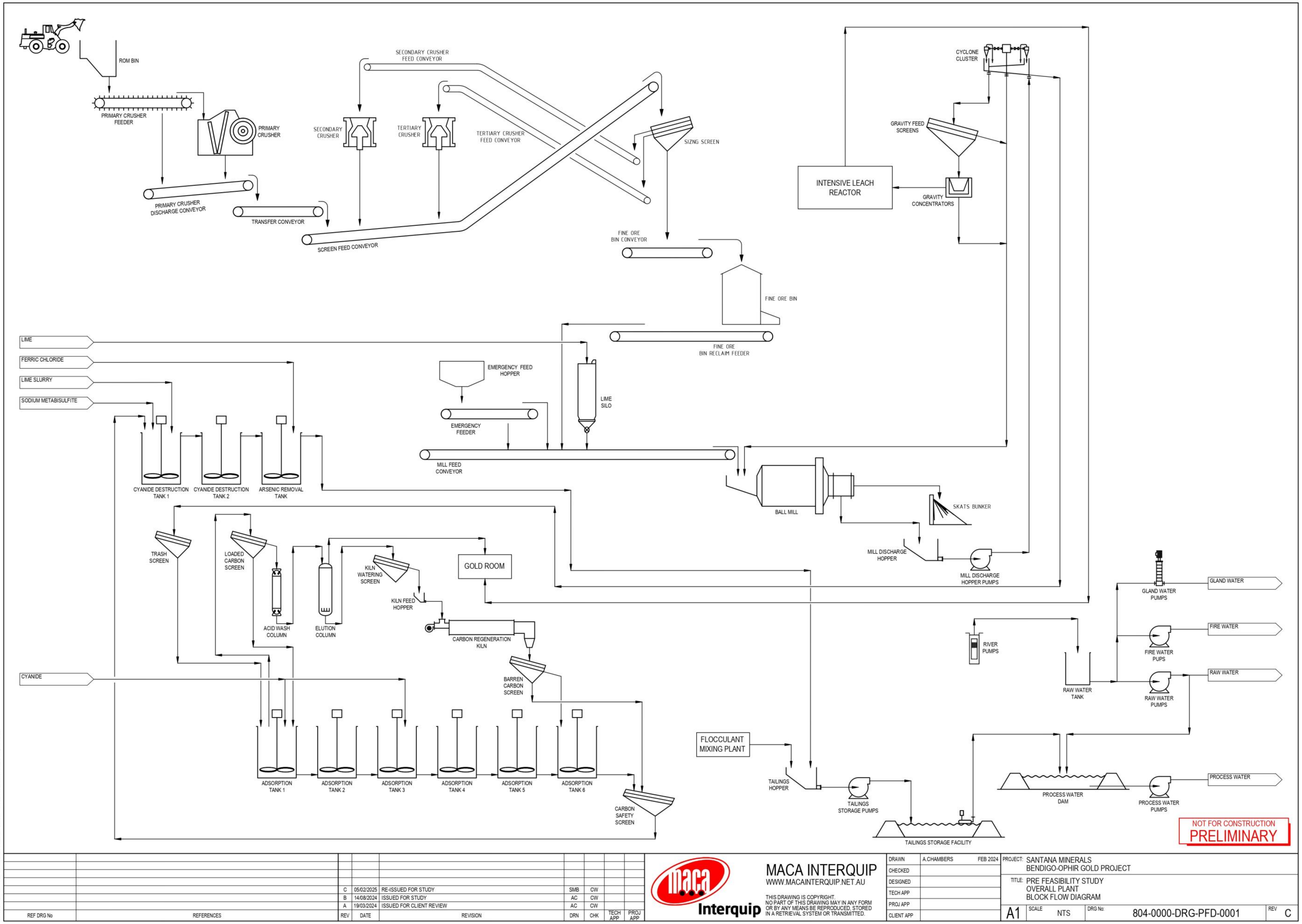


Figure 16 Process Plant Flow Sheet

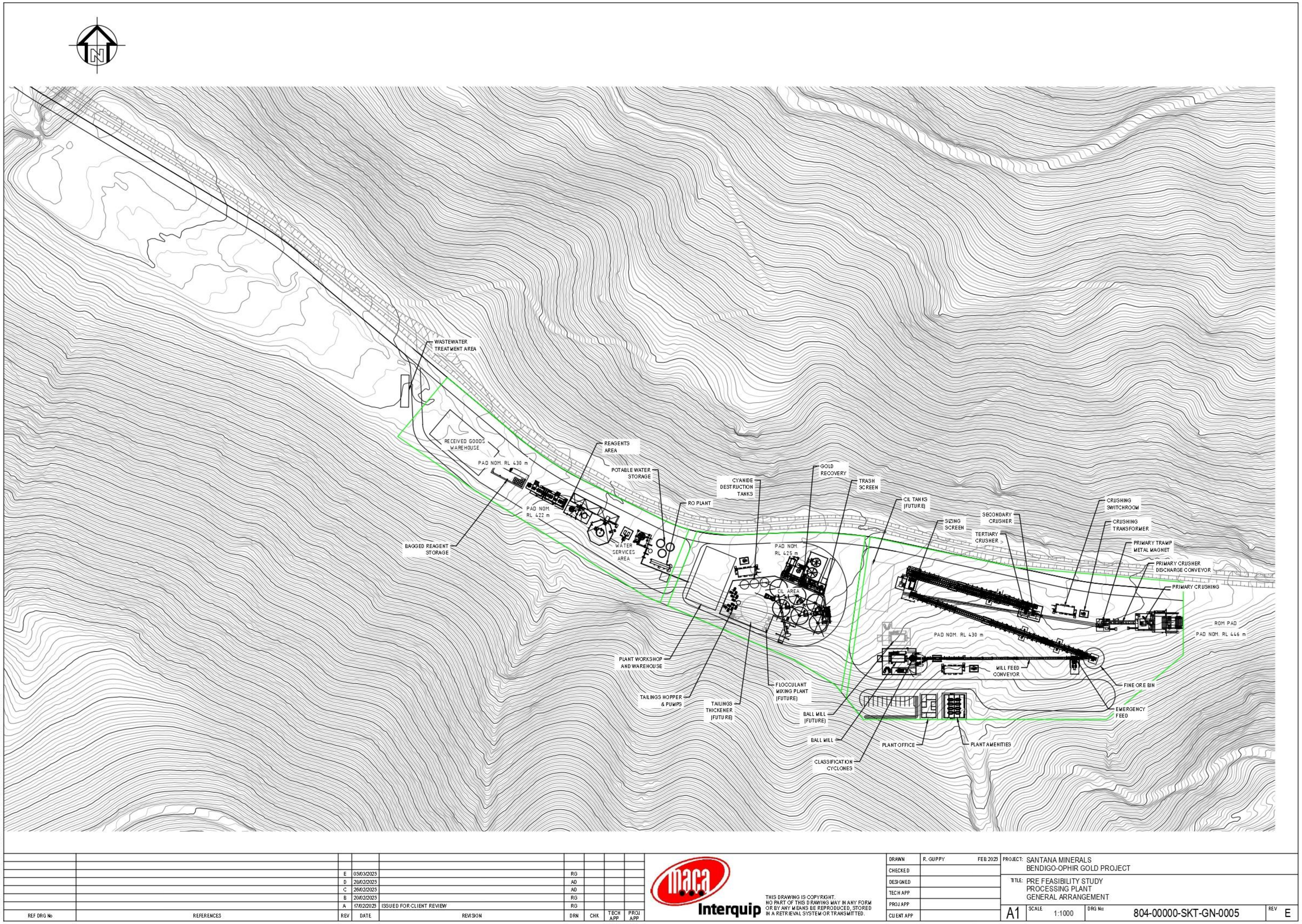


Figure 17 General Arrangement Drawing of BOGP Processing Plant

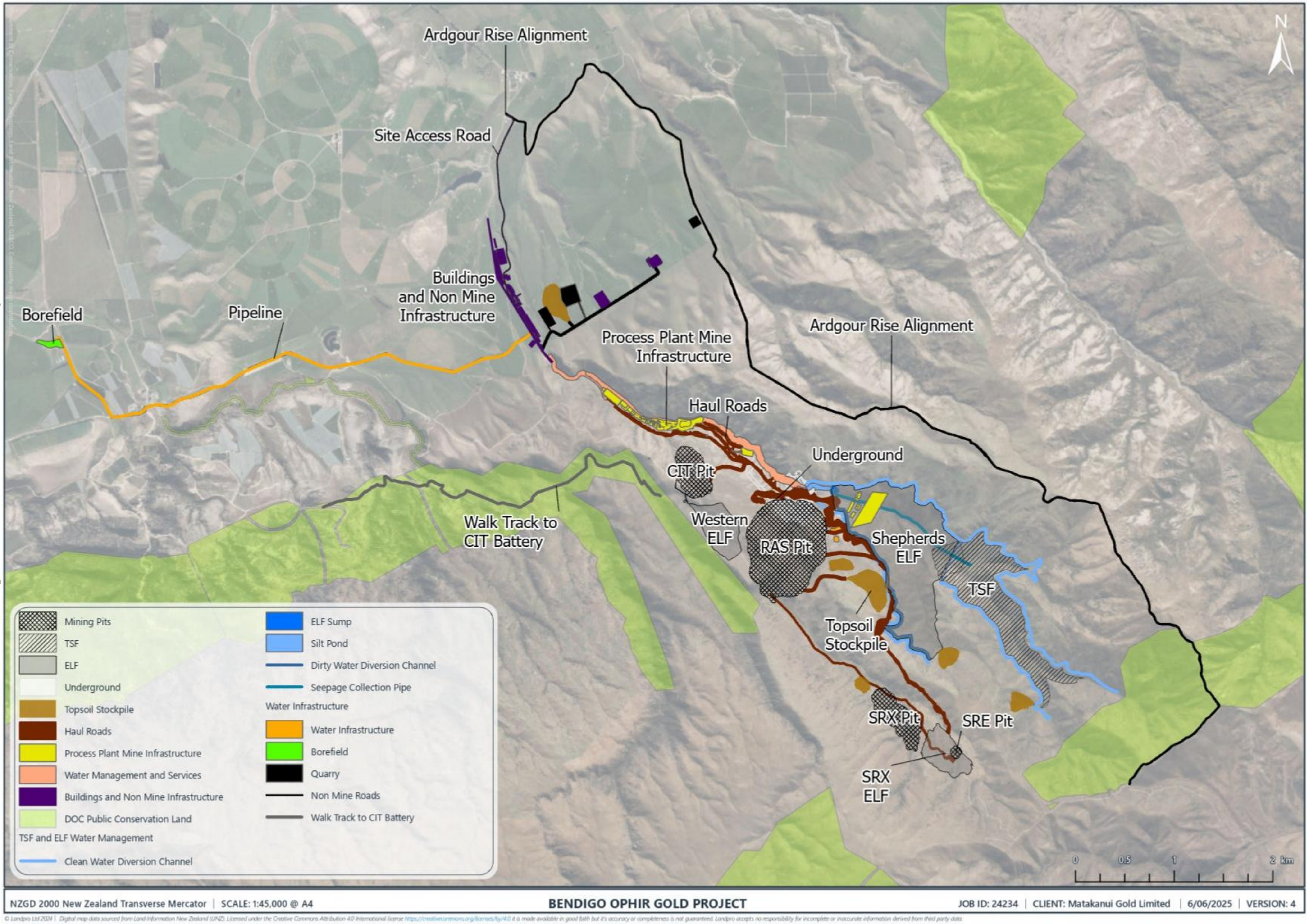


Figure 18 Project Layout

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RAS Gold Recovery Calculations[2]

Gold recovery estimation in this Updated PFS is based on a regression analysis of metallurgical testwork, replacing the prior flat recovery assumption of 92.4%. The analysis correlates gold head grades with measured residue grades, resulting in the regression formula:

Residue (ppm) = 0.0411 × Head Grade (ppm) + 0.0671

This relationship was used to dynamically estimate recoveries on a period-by-period basis from the scheduled head grades. The model shows a strong fit with an R^2 value of 0.87, providing confidence in the reliability of predicted recoveries across the mine life. The regression method offers a more accurate, grade-sensitive approach compared to using a constant recovery factor. It also better captures metallurgical variability over time and ensures recovery assumptions remain aligned with the evolving grade profile. For comparative purposes, the average gold recovery based on this analysis is 93%.

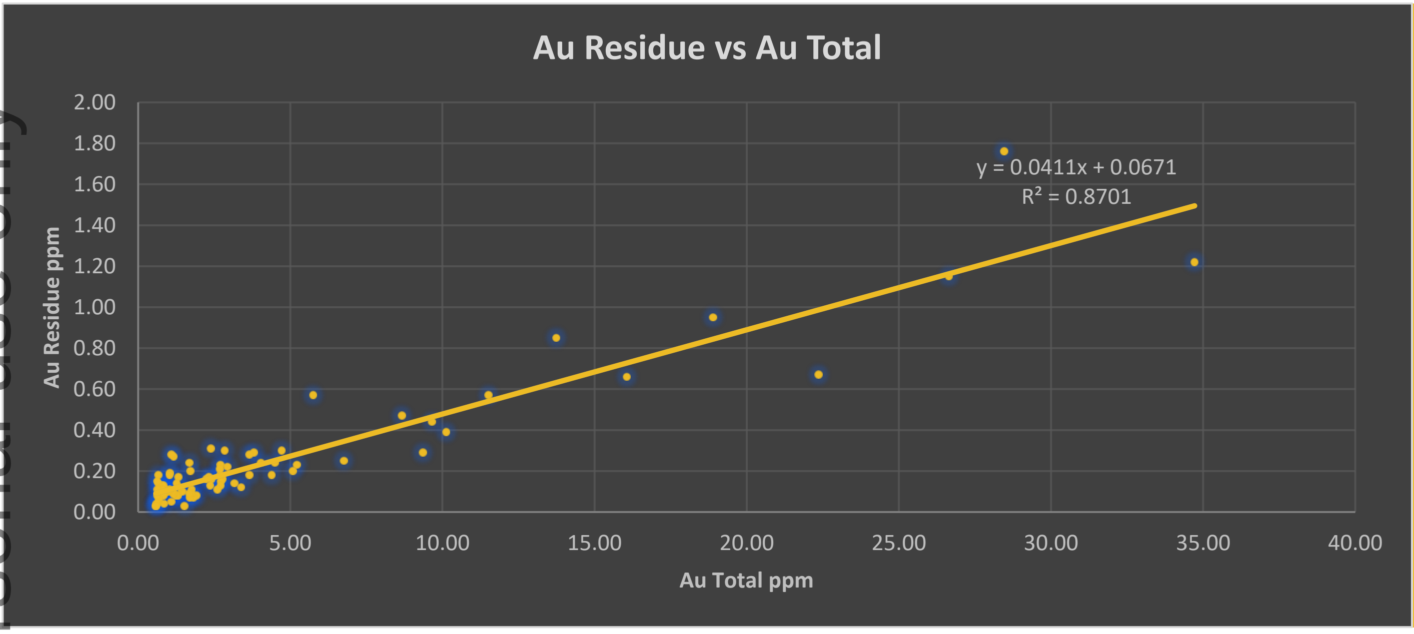

Figure 19 Gold Recovery Analysis, Regression Line of Residues

SRX Gold Recovery Calculations[3]

Additional SRX metallurgical testwork was carried out subsequent to the November 2024 PFS. A master composite and eight additional variability samples were analysed at IMO laboratories in Perth, with leach tests performed at a P80 106 micron grind size (the same grind size as proposed for RAS mill feed). Recent results show improvements to overall recoveries which have been factored into the Updated PFS for SRX ore, now at an average of 81.8%.

		MC	VC#1	VC#2	VC#3	VC#4	VC#5	VC#6	VC#7	VC#8
Calculated Ore Head Grade		1.10	0.76	1.24	1.91	0.98	0.86	1.05	1.35	1.51
Assay Ore Head Grade		0.66	0.51	0.46	1.55	1.74	0.47	1.32	0.90	0.89
Gravity Recovery		22.9%	36.7%	51.5%	44.6%	22.6%	30.5%	35.5%	36.0%	54.2%
24 Hour Overall Recovery		68.3%	90.9%	83.3%	75.0%	75.3%	86.7%	86.1%	86.4%	84.6%
Leach Residue Grade		0.35	0.07	0.21	0.48	0.24	0.11	0.15	0.18	0.23

[3]ASX announcement 26 June 2025 - Improved Metallurgical Outcomes from RAS and SRX (see Appendix 1)

Operating Costs

Open Pit Mining Costs

Mining costs were estimated from first principles based on an owner operator mining model and cross checked with contract mining rates from reputable service providers.

The revised fleet consists of:

- 3 x 200t excavators on waste.
- 1 x 100t excavator on ore.
- Average of 20 x 90t rigid dump trucks.
- 4 x 60t articulated dump trucks.
- 3 x drill and blast drill rigs.
- 2 x water carts.
- 1 x ROM loader.
- 2 x graders.

The averaged mining unit cost over the life-of-mine (LoM) is approximately NZ$4.61/t and includes the following activities:

- Clearing and grubbing.
- Ore and waste drill and blast.
- Ore and waste load and haul.
- Crusher feed.
- Ancillary equipment for supporting mining activities.
- Mine management and technical services costs.
- Progressive rehabilitation.
- 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
- Haul road maintenance.

The breakdown in mining unit costs for the revised fleet are:

	Operating Costs	NZD $M	Unit cost NZD $/t
1	Loading	141.7	0.65
2	Hauling	386.2	1.77
3	Drilling	36.7	0.17
4	Blasting	46.0	0.21
5	Grade control sampling	1.3	0.01
6	Crusher feed	29.4	0.13
7	Ancillary	170.6	0.78
8	Mining operations overheads	194.4	0.88
9	Pit clear, grub and rehab	1.7	0.01
	Total Mining Costs	**1,008**	**4.61**

Table 16 RAS Open Pit Mining Costs

Underground Mining Costs

Underground mining operational cost inputs have not materially changed, however changes in quantities and labour inputs have resulted in the following operational cost inputs:

Item	NZD $M
Fleet and establishment capex	78.3
Capital development (sustaining)	26.3
Total capex	**104.6**
Direct operating cost	28.5
Mine fleet running costs	45.7
Infill drill costs	3.6
Power	27.4
Labour	113.2
Paste plant operations	47.7
Miscellaneous	3.1
Total opex	**269.4**
Ore tonnes	3.78
Total cost per tonne	**98.7**

Processing Costs

Processing costs, summarised in Table 18, have been estimated by MACA Interquip Mintrex based on the defined metallurgical characteristics of the ore, updated input costs, and the revised process flow sheet, assuming a 1.2Mtpa nameplate capacity. Key technical considerations when calculating processing costs have not changed since the November PFS, they include:

Parameter	Value
Power cost	$0.13/kWh
Crushing work index	6.1 kWh/t
Bond rod mill work index	19.0 kWh/t
Bond ball mill work index	20.1 kWh/t
Abrasion index	0.31
Ore specific gravity	2.72

Table 17 Key Processing Assumptions

Processing costs of $24.0/t have been calculated for the 1.2Mtpa plant, with variable costs for: power ($4.60/t), consumables ($8.06/t) and maintenance ($0.97/t) calculated at $13.63/t, with the balance in fixed costs largely associated with process plant labour featured in Table 19.

Table 18 below also shows an estimate for General and Administration (G and A) per tonne of mill feed, related to administration costs, and labour costs pertaining to positions detailed in Table 20.

Processing Costs	NZD$/t
Processing cost	24.0
General and administration cost	4.4
Processing cost	**28.4**
Laboratory[4]	3.8

[4] Laboratory costs include full-service sampling for plant, grade control and resource drilling and are included in total processing cost cash flows.

Table 18 Processing Costs

Position	Number	Position	Number
Manager Processing	1	Lab Technicians	4
Senior Metallurgist	1	Maintenance Superintendent	1
Plant Metallurgist	2	Maintenance Planner	1
Shift Supervisor	4	Mechanical Supervisor	1
Crushing Operators	4	Boilermakers	2
Milling Operators	4	Fitters	4
CIL Operators	4	Trades Assistant	4
Reagents/Water Operators	4	Electrical Supervisor	1
Goldroom Operators	2	Electricians	3
Loader Operator	4	Water intake Operators	4
Senior Chemist/RSO	1	Sick & Holiday/Relief	8
Chemist/RSO	1	**Total**	**65**

Table 19 Process Plant Labour

General and Administration Costs

The G and A costs shown in Table 18 are broken into two broad categories: G and A (labour), with positions shown in Table 20 below, and G and A (admin) for worker transport, consultants, light vehicles, office expenses, fees, recruitment and auditing.

Total G and A costs disclosed in the cash flow statement also include a provision for site specific items including land leases, water supply (power), ecology costs, water treatment and closure bonding adding another $5.30/t of ore processed.

Site Administration Roles	Steady State
General Manager	1
Geology Manager	1
Administrator	2
H&S Manager	1
H&S Officer	2
HR Manager	1
HR Officer	1
Payroll	1
Community Relations Manager	1
Environmental Manager	1
Environmental Officer	1
Financial Controller	1
Accountant	1
Accounts Clerk	1
Warehouse Supervisor	1
Warehouse & Delivery	3
Purchasing Officer	2
Cleaners	2
Bus Drivers	2
Security/Paramedic/Emergency Response	6
Sub Total	**32**

Table 20 General and Administration Labour

Capital Costs

Establishment Costs

Table 21 outlines the current estimate of establishment costs, reflecting detailed assessments of bulk earthworks, site infrastructure, and camp requirements.

Area	NZD $M
Owners construction team	1.8
Land (pre-payments and leasing)	7.2
Infrastructure (access and buildings)	53.2
Earthworks	14.2
TSF wall	11.1
Equipment	8.2
Total	**95.7**

Table 21 Site Establishment Costs

The earthworks packages, including: haul road construction, dam and infrastructure pad preparation, pioneering activities, and stripping works at RAS for deployment of the major mining fleet, has now been fully scheduled and costed. Figure 20, Figure 21, and Figure 22 below illustrate the infrastructure locations and access routes from the pit to key haulage destinations: the Tailings Storage Facility (TSF), the ROM pad, the Engineered Landform (ELF), and the Western Engineered Landform (WELF).

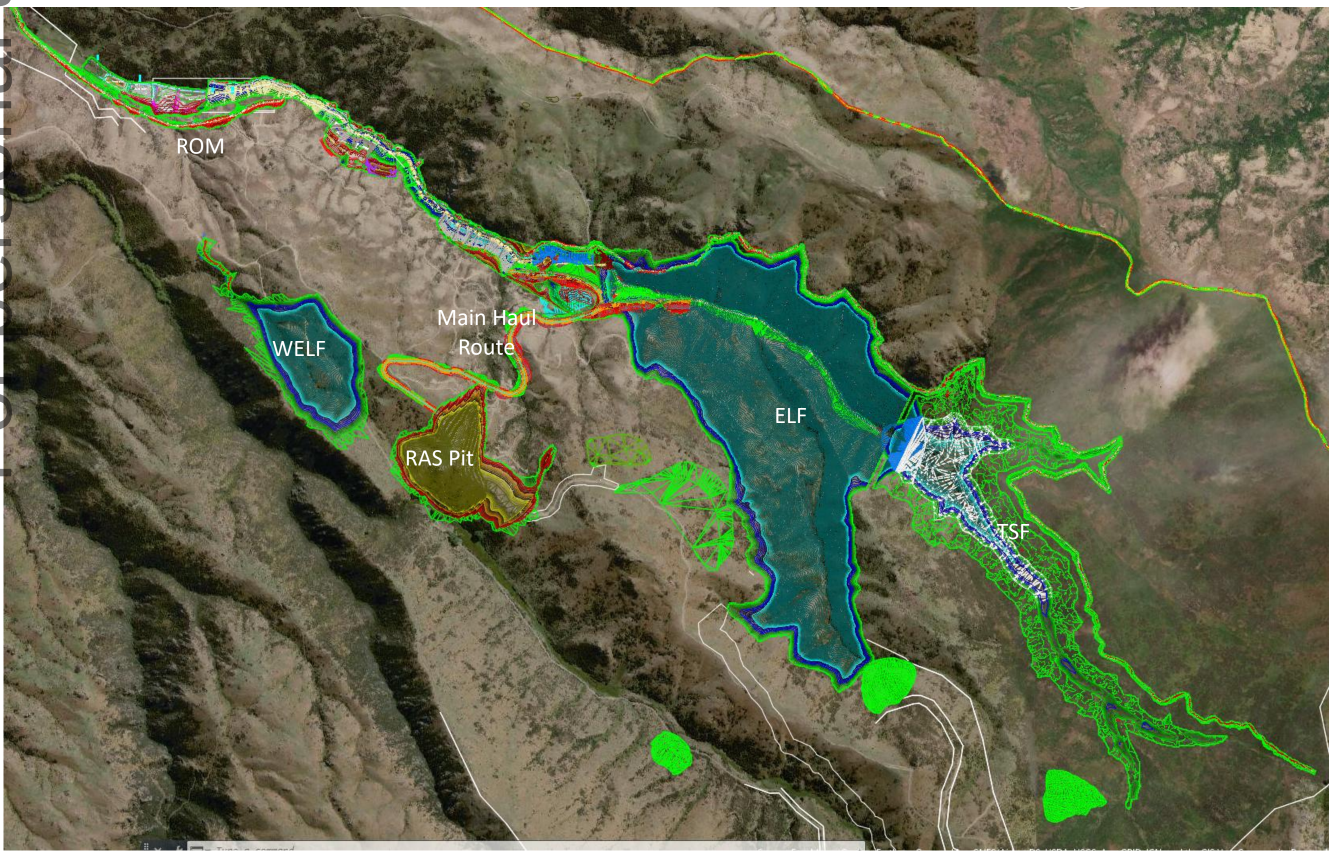

Figure 20 Plan View of RAS Starter Pit with Access to ELFs, TSF and ROM

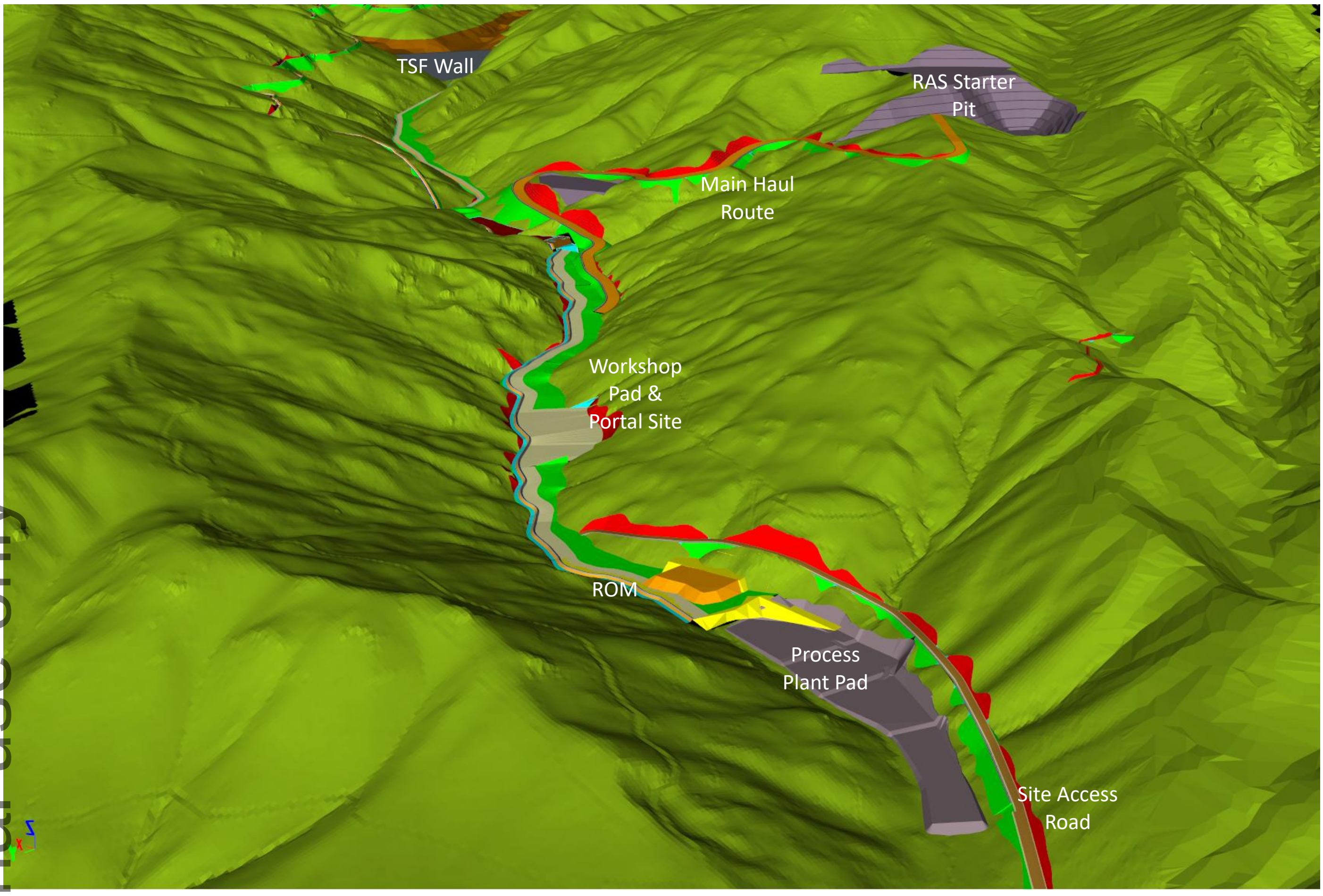

Figure 21 Shepherds Creek Infrastructure Looking East

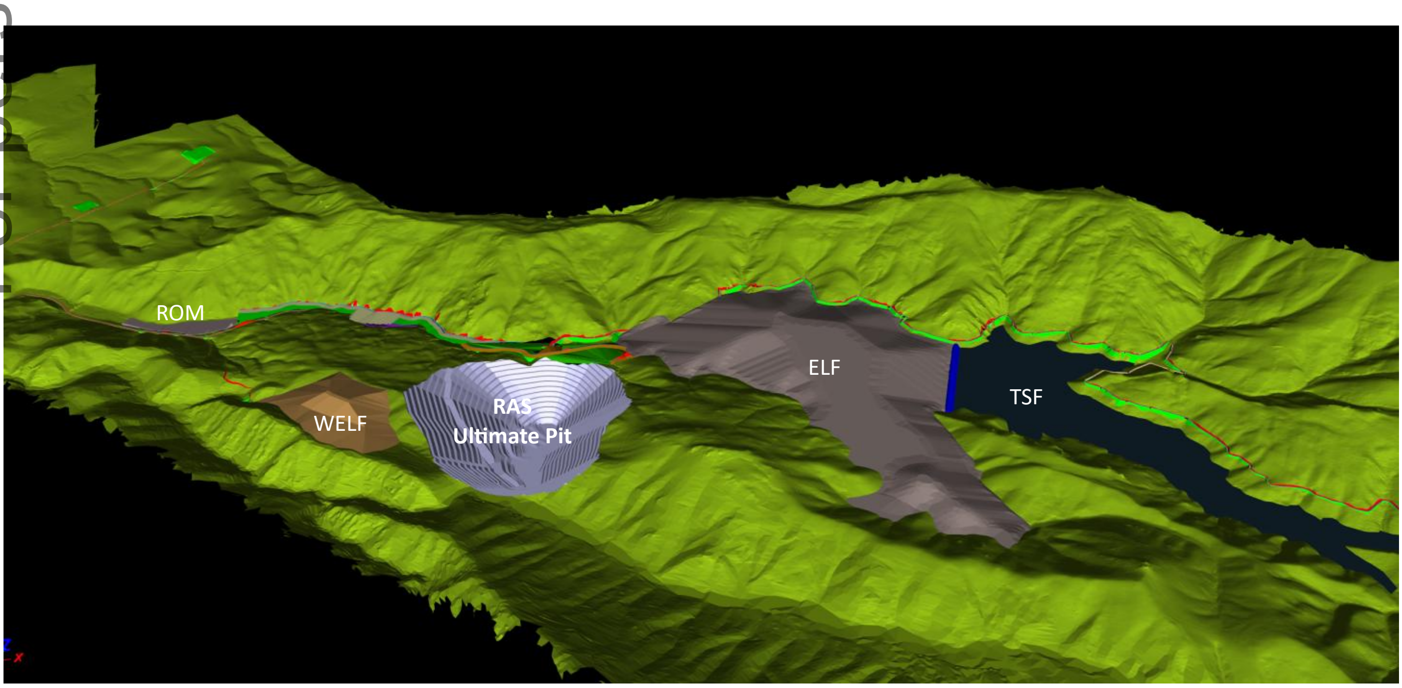

Figure 22 RAS Ultimate Pit Shell Looking North

Process Plant Capex

Table 22 below shows the capital cost estimate for the process plant is NZD$107.7 million, exclusive of contingency and laboratory costs. This figure reflects the updated 1.2Mtpa plant configuration, incorporating three-stage crushing and a single ball mill, and includes all associated mechanical, electrical, structural, and civil components.

While no contingency is applied at the plant level, a separate 10% contingency allowance has been incorporated into the overall project capital estimate to account for uncertainty and potential scope variation. The plant cost forms a major component of the total development capital and has been prepared with input from MACA Interquip Mintrex based on recent vendor quotes and engineering inputs. Santana will provide an integrated owners construction team to manage the build under an EPCM style contract, with positions outlined in Table 23.

Description	NZD$M
Contractor indirects	4.1
P & G	0.3
First fills	1.0
Equipment spares	1.5
EPCM	21.3
Commissioning	0.9
Crushing	17.8
Milling	14.6
Leach and adsorption	9.7
Gold recovery	4.7
Reagents	1.6
Water services	0.5
Piping	10.3
E&I	16.8
Construction overheads	2.6
Total	**107.7**

Table 22 Process Plant CAPEX Breakdown

The Owners Construction team shown below will be supported by administration personnel accounted for in General and Administration labour.

Position	Number
Project Manager	1
Assistant Project Manager	1
Contracts Manager	1
Contract Administrator	1
Civil Superintendent	1
Buildings Superintendent	1
Schedule Control/cost/QS	2
Planner	1
Total	**9**

Table 23 Owners Construction Team

Capitalised Pre-strip

Approximately 6.5 million bcm (17.5 million tonnes) of waste rock will be removed prior to commercial and sustainable gold production. This material has been estimated at the mining cost of $4.61/t, and is included in the CAPEX budget at $73.8 million.

Financial Modelling Assumptions

The financial evaluation within this Updated PFS is based on recent average market pricing to provide a balanced representation of prevailing economic conditions. The following assumptions have been adopted:

- Base-case gold price: US$2,220/oz benchmarked against peer Australian gold producers' recent studies.
- Current gold price: US$3,138/oz, representing a conservative average price over the three-month period to 15 June 2025.
- Exchange Rate: NZD:USD: 0.580 representing a six-month average to 15 June 2025.
- Exchange Rate: NZD:AUD: 0.915 representing a six-month average to 15 June 2025.
- Diesel pricing: NZD$1.35/ltr.
- Contingency added to all pre-production capital: 10%.
- Contingency added to mining costs (including pre-strip capex): 10%.

These pricing assumptions are considered appropriate for a PFS level assessment and reflect a moderate position relative to recent market volatility. Sensitivities to gold prices have also been assessed and are presented in Table 4.

Further assumptions made in the financial model include:

- A real discount rate of 6.5% was applied. This was based on: an internal weighted average cost of capital calculation, a refined risk adjustment for project fundamentals, interest rate climate, and a peer analysis of comparative projects.
- All estimated costs are nominal without adjustments for inflation.
- A corporate tax rate of 28% has been applied, with allowances for New Zealand tax losses of (NZD$39 million).
- 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.

Table 24 below shows the cash flows over the initial LoM.

Financial highlights (@ AUD$4,950/oz)		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	LOM
Gold production	oz			59	113	120	120	120	99	107	86	98	90	76	88	56	15			
Cash flow																				
Cash flow (pre-tax)	AUD ‘mil	(43)	(233)	94	339	360	353	349	296	334	266	341	315	252	312	178	9	1	(5)	3,517
Cash flow (post-tax)	AUD ‘mil	(43)	(233)	94	273	267	239	258	215	245	193	254	243	180	245	101	6	1	(5)	2,534
Gold revenue ($’000)	AUD 'mil	-	-	290	562	594	594	594	493	529	424	485	447	378	437	276	74	-	-	6,177
Initial life of mine operating costs																				
Total open pit mine operating costs	AUD 'mil	-	-	55	127	137	141	109	58	52	34	6	6	15	7	17	13	-	-	777
Total underground operating costs	AUD 'mil	-	-	-	-	-	-	-	-	30	43	48	40	36	35	15	-	-	-	246
Total ore processing operating costs	AUD 'mil	-	-	29	30	30	30	30	30	31	30	30	30	30	30	30	23	-	-	416
Total general and admin costs[2]	AUD 'mil	-	-	8	10	11	13	13	12	12	11	10	10	10	10	9	9	6	5	158
Royalties	AUD 'mil	-	-	25	51	54	53	57	51	51	38	48	44	35	43	25	3	-	-	576
Total cash operating cost	AUD 'mil	-	-	117	218	232	238	209	152	176	156	142	130	125	125	96	48	6	5	2,173
Total cash operating surplus *(EBITDA)*	AUD 'mil	-	-	173	344	362	356	385	341	353	268	343	317	254	313	179	26	(6)	(5)	4,004
Non-cash costs																				
Life of mine depreciation and amortisation	AUD 'mil	-	-	21	25	26	26	26	26	39	41	41	42	42	42	35	46	-	-	480
Total cost of sales	AUD 'mil	-	-	138	243	257	264	236	178	215	197	183	172	167	167	131	94	6	5	2,652
Historical PPE	AUD 'mil			2	3	3	3	3	3	3	3	3	3	3	3	2	2	-	-	36
Net profit before tax (NPBT)	AUD 'mil	-	-	149	316	334	328	356	312	311	225	299	272	209	268	142	(22)	(6)	(5)	3,489
Corporate tax payable (28.0%)	AUD 'mil	-	-	(21)	(89)	(94)	(93)	(100)	(88)	(85)	(65)	(86)	(79)	(61)	(78)	(42)	(2)	-	-	(983)
Estimated net profit after tax	AUD 'mil	-	-	128	227	240	235	255	224	226	159	214	194	148	191	100	(24)	(6)	(5)	2,506

Table 24 LoM Cash Flows (totals may not sum due to decimal place rounding)

Environmental & Mine Closure

A conceptual Mine Closure Plan has been developed to support the upcoming FTA consent application, with the overarching objective of achieving a final landform and site that is safe, stable, non-polluting, and aligned with the expectations of both regulators and local stakeholders (see Figure 23). The closure strategy incorporates a multidisciplinary approach, drawing on input from experts in geochemistry, geotechnical design, mine scheduling, ecology, rehabilitation, and visual impact assessment.

As part of the Updated PFS, the Company has also made significant financial provisions for long-term ecological outcomes (included in site specific costs discussed above in 'General and Administration Costs').

Key elements addressed include:

- Legal and consent obligations.
- Closure risks and landform stability.
- Post-mining land uses.
- Measurable completion criteria.
- Community consultation.

Collectively, these measures ensure that mine closure is not only technically and environmentally sound but also contributes lasting value to the community.

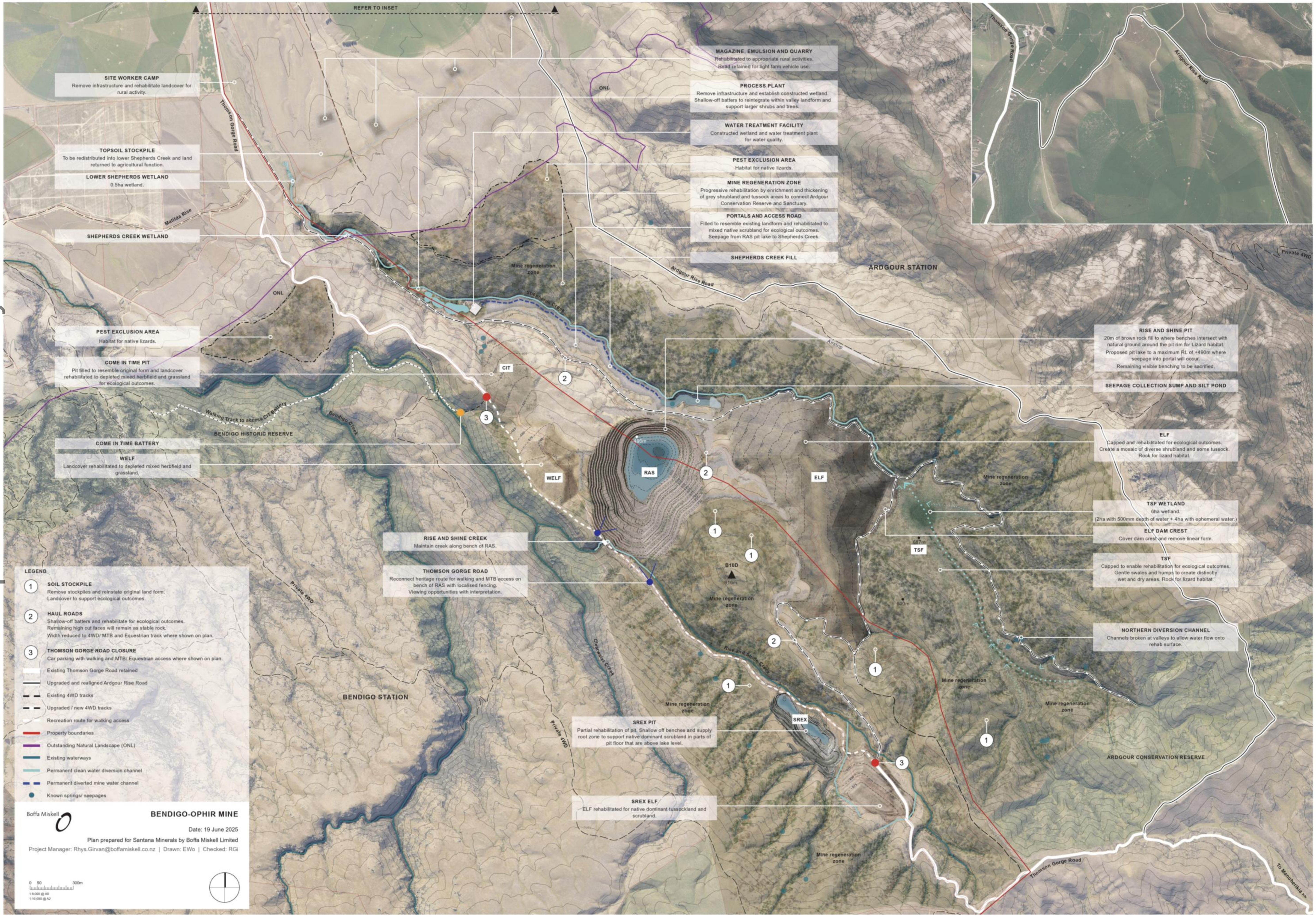

Figure 23 Rehab Plan Showing Closure Landscape

Project Implementation

Project implementation remains contingent on two parallel approvals: financing and resource consent. The Company expects to lodge its Fast-track Approval application in the next quarter, supported by a full suite of technical assessments, including a comprehensive Assessment of Environmental Effects. While the legislation is designed to enable streamlined approvals, timelines remain subject to interpretation and emerging process precedent. For planning purposes, the Company has adopted a working assumption of a six-month approval timeframe from the date of submission of the FTA.

Following approval and financial close, the implementation timeline is expected to commence 1 January 2026 and includes approximately one month for site mobilisation and early civil works, followed by three months of pioneering activity to establish: initial benches at the southern end of the RAS deposit, internal haul roads, pads and dams, and ancillary earthworks in advance of the heavy mining fleet being deployed.

In parallel, detailed engineering under an EPCM-style delivery model is already underway, with equipment selection, procurement planning, and contractor engagement in progress. The Company is positioned to move into approved early works ahead of consenting, with major construction immediately following resource consents and financing milestones. Please see Figure 24 below for the Company's scheduled indicative timeline.

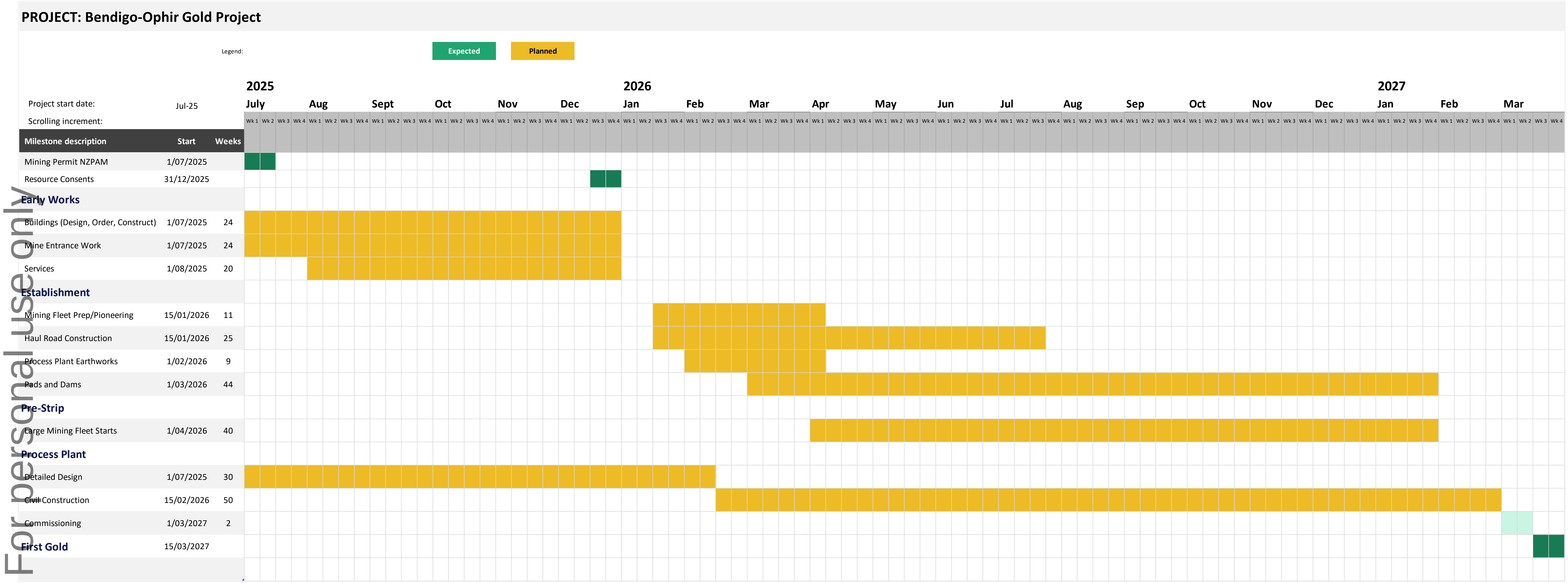

Figure 24 Construction Timeline

Permitting - Mining Permit (NZPAM – Crown Minerals Act 1991)

The Bendigo-Ophir Gold Project is located within Minerals Exploration Permit (MEP) 60311. Following the release of the November 2024 PFS, Santana lodged its application for a Minerals Mining Permit (MMP) with New Zealand Petroleum and Minerals (NZPAM), covering the Rise and Shine (RAS) deposit and associated infrastructure areas. The application, submitted under the Crown Minerals Act 1991 (CMA), is supported by the March 2025 Mineral Resource Estimate, which delineates over 1.5Moz in the Indicated category, and by the development metrics outlined in the November 2024 PFS.

Under Section 23 of the CMA, a mining permit may be granted where the Minister is satisfied that:

- An Indicated mineable mineral resource or exploitable deposit has been identified;
- The permit area is appropriate; and
- The proposed mine plan economically depletes the resource in accordance with good industry practice.

Fast-track Consenting (Resource Management Reform)

In December 2024, the Fast-track Approvals Act (FTA) was passed into law, establishing a new statutory framework for streamlining development approvals across multiple regulatory regimes. The FTA replaces elements of the Resource Management Act 1991 (RMA) for eligible projects and introduces a unified process overseen by an independent Fast-track Advisory Panel and Ministers of the Crown.

The Company's Bendigo-Ophir Gold Project was confirmed in 2024 as a project eligible to access the Fast-track pathway. The Company intends to submit its FTA application in the next quarter, with all supporting technical assessments, including a comprehensive Assessment of Environmental Effects which is now completed.

This consolidated "one-stop-shop" consenting framework is expected to materially reduce the time, cost, and complexity associated with project approvals and is viewed as a key enabler of the Company's planned development timeline.

Risks and Opportunities

Opportunities are presented by:

- Potential for resource extension down-dip at RAS to convert additional ounces into the production schedule.
- Further upside from ongoing exploration and infill drilling, particularly at CIT and RAS South.
- Additional ounces could be added through resource conversion, particularly if gold prices remain elevated.
- Accommodation availability in Central Otago presents a cost-saving opportunity for construction phase logistics.
- Integration of known Inferred resources into the underground development creates a longer-term production pipeline and improved project optionality.

A structured risk assessment was undertaken across all areas of the November 2024 PFS study, identifying 108 discrete risks categorised by project phase (design, consent, construction, operations) and ownership (engineering, implementation, operations, environment, corporate).

Key risks identified include:

- Timing uncertainties related to the newly enacted FTA legislation.
- Mobilisation of the mining fleet in alignment with consent approval and construction readiness.
- Timely connection of high-voltage power infrastructure to the site.
- Managing the pre-strip ramp-up to secure first ore and cash flow without delay.
- Execution of project financing on acceptable terms.

The Company has put in place mitigation strategies across each of these areas and continues to manage risk actively as part of its implementation readiness planning.

Funding

The Board believes there is a reasonable basis to expect that development funding for the Bendigo-Ophir Gold Project will be secured, based on the strength of the Updated PFS and the Company's project attributes:

- The PFS outlines robust economics, with a significant improvement in capital efficiency, early access to high-grade ore, and strong free cash flow generation, even at a base case gold price of US$2,220/oz.
- The total pre-production capital requirement is estimated at A$277 million, a reduction from earlier studies, reflecting staged pit development and a downsized process plant more closely aligned with the selectively mined ore.
- The Project has a 13.8-year life with strong operating margins and a value-accretive NPV relative to current market capitalisation.
- The Company is in early-stage discussions with potential financiers, and has also successfully conducted equity raisings in the past.
- Santana Minerals has a tight capital structure and owns 100% of the Project, offering clear financing flexibility.
- The Board and management team have extensive experience in financing and delivering mining projects, providing confidence to potential lenders and investors.
- Current gold market conditions remain favourable, with recent precedent transactions demonstrating robust appetite for financing well-advanced, technically sound gold developments.

While discussions with potential financiers are ongoing, there is no guarantee that funding will be secured on favourable terms or within expected timeframes. A typical funding structure is likely to involve a combination of debt and equity on typical 60-40 debt to equity split and may be subject to terms that could dilute or otherwise impact the value of existing shareholders.

The Company may also consider alternative financing strategies, including the potential sale or joint venture of part of its interest in the Project or the monetisation of future revenue streams.

Bendigo-Ophir Gold Project Mineral Resource Estimate

The Project contains a Mineral Resource Estimate (MRE) calculated at a cutoff grade of 0.5 g/t Au with top cuts applied, as at March 2025:

Deposit	Category	tonnes (Mt)	Au grade (g/t)	Contained Gold (koz)
RAS	Indicated	18.9	2.5	1,538
	Inferred	7.6	2.2	542
RAS Total	Indicated and Inferred	26.5	2.4	2,080
CIT	Inferred	1.2	1.5	59
SRX	Indicated	2.2	0.8	54.7
SRX	Inferred	2.9	1.0	90.5
SRX Total	Indicated and Inferred	5	0.9	145
SRE	Indicated	0.4	0.8	10.3
SRE	Inferred	1.1	1.2	42
SRE Total	Indicated and Inferred	1.5	1.1	52
BOGP Total	Indicated	21.5	2.3	1,603
	Inferred	12.7	1.8	734
BOGP Total	**Indicated and Inferred**	**34.3**	**2.1**	**2,337**

Table 25 Bendigo-Ophir Gold Project Mineral Resource March 2025

Previous Disclosure - 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 634k Oz" dated 28 September 2021 (as to the CIT deposit)
- ASX announcement titled "Bendigo-Ophir Gold Project - Pre-Feasibility Study" dated 15 November 2024 (as to the MRE for the SRX and SRE deposits)
- ASX announcement titled "RAS Mineral Resource Estimate Review" dated 4 March 2025 (as to the MRE for the RAS deposit)
- ASX announcement titled "Improved Metallurgical Outcomes from RAS and SRX" dated 26 June 2025 (as to the metallurgy results for RAS and SRX)

Copies of those 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 and, in the case of the estimates of Mineral Resources, all material assumptions and technical parameters underpinning those 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 Ore Reserves that underpin the production target and forecast financial information set out in this report were prepared in accordance with the 2012 Edition of the JORC Code and are based on, and fairly represent, information and supporting documentation compiled under the supervision of Mr Damian Spring, a Competent Person who is a Fellow of The Australasian Institute of Mining and Metallurgy. Mr Spring is a full-time employee of Santana Minerals Ltd and has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the JORC Code. Mr Spring consents to the inclusion in this report of the matters based on his supervision in the form and context in which they appear. Mr Spring is eligible to participate in STI and LTI schemes in place as performance incentives for key personnel.

The information in this report that relates to Exploration Results (including metallurgical results) is based on information compiled by Mr Alex Nichol who is a Member of the Australian Institute of Geoscientists. Mr Nichol is a full time employee and has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as Competent Persons as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves.' Mr Nichol consents to the inclusion in this report of the matters based on their information in the form and context in which it appears. The Company confirms that the form and context in which the Competent Person's findings are presented have not been materially modified. Mr Nichol is eligible to participate in STI and LTI schemes in place as performance incentives for key personnel.

Appendix 1

Updated metallurgical test results for recoveries related to SRX and RAS low grade came from hole locations shown in Figure 25, Figure 26, Figure 27 with corresponding results discussed in ASX Announcement "Improved Metallurgical Outcomes from RAS and SRX" dated 26 June 2025.

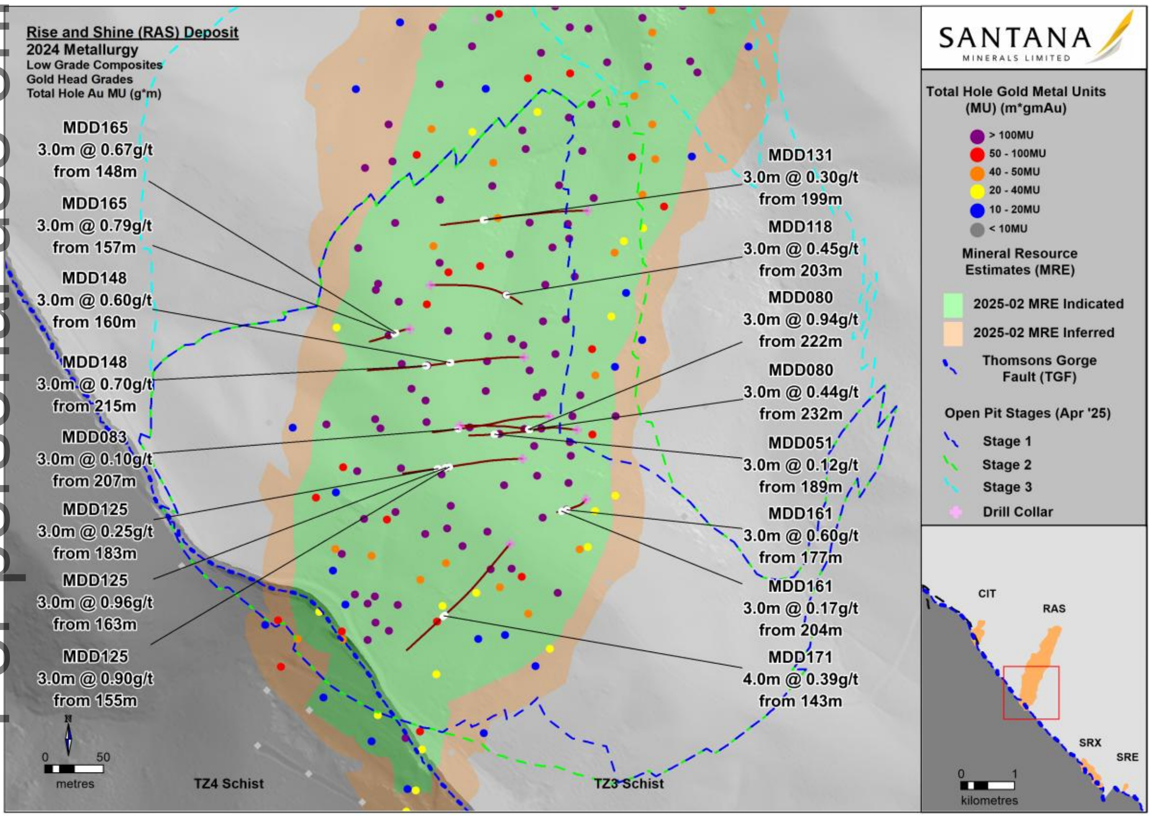

Figure 25 RAS Low Grade Metallurgical Results Hole Locations

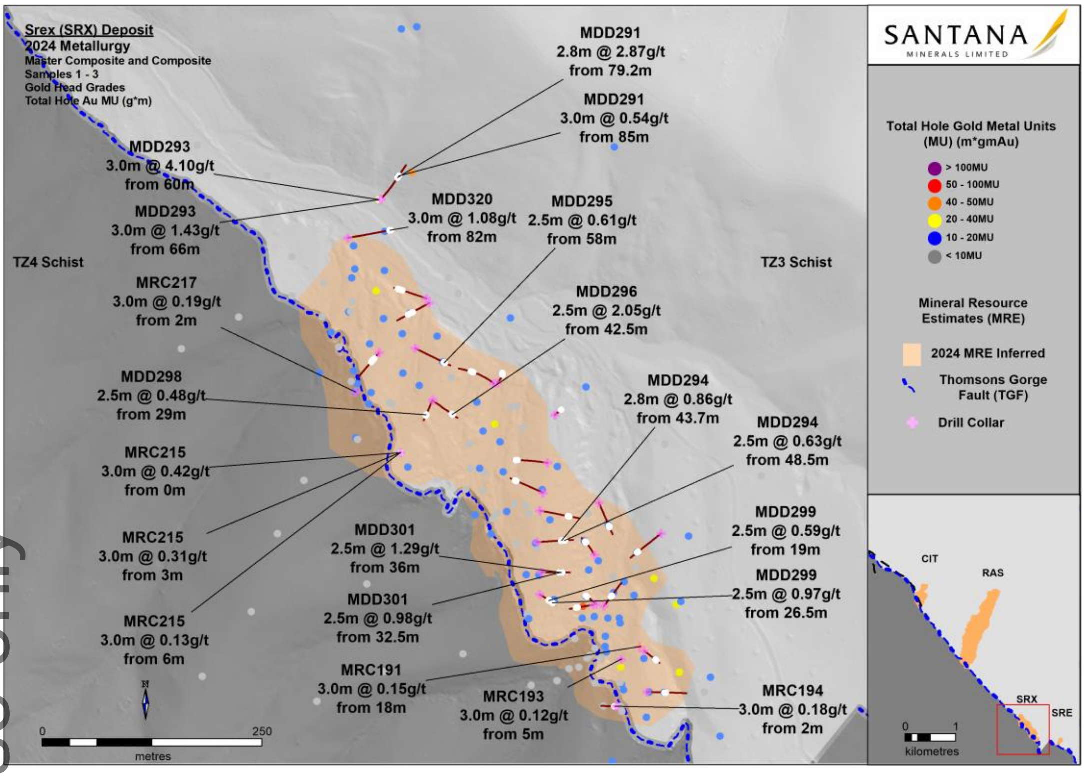

Figure 26 SRX Metallurgical Results Hole Locations Part 1

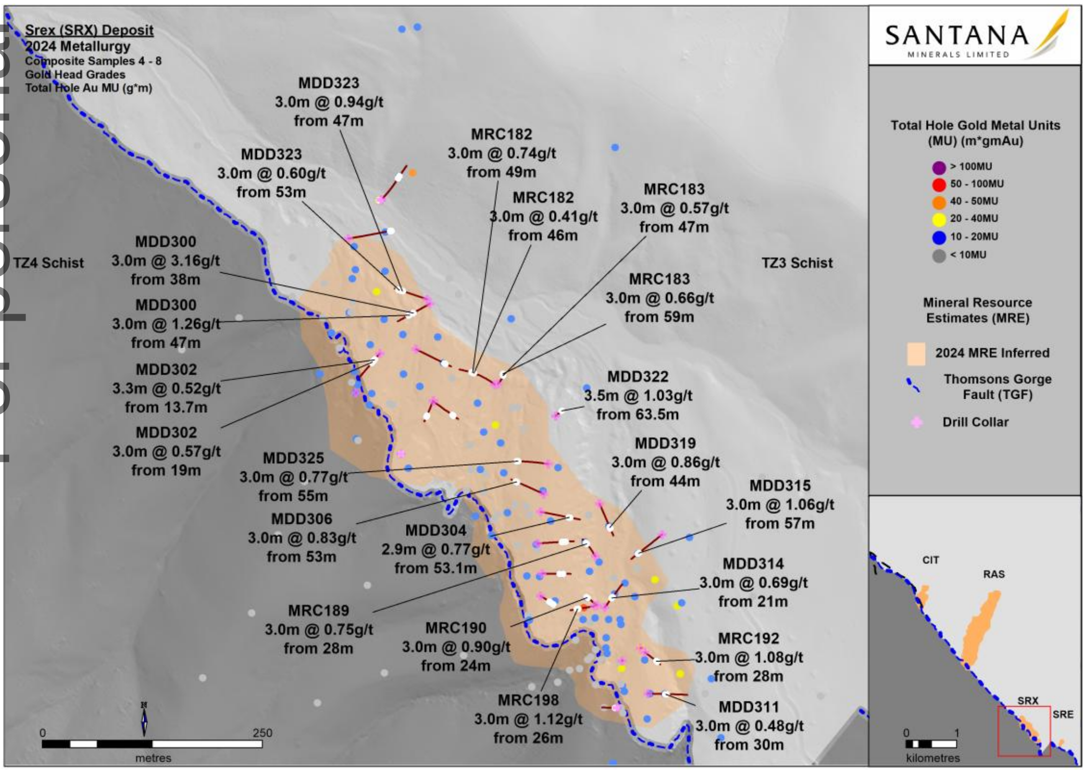

Figure 27 SRX Metallurgical Results Hole Locations Part 2

SANTANA MINERALS LIMITED

JORC Code, 2012 Edition – Table 1

Section 1: Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	*Nature and quality of sampling (eg 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 (eg '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 (eg 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 mineralization being reported.

Criteria	JORC Code explanation	Commentary
Drilling techniques	*Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg 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 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

Criteria	JORC Code explanation	Commentary
		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 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*	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 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.

Criteria	JORC Code explanation	Commentary
	of the material being sampled.	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 (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie 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. No geophysical tools have been used in this MRE.

SANTANA MINERALS LIMITED

Criteria	JORC Code explanation	Commentary
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 (PHA > BLEG > SFA > FAA) 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 system used.* *Quality and adequacy of topographic control.*	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. 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.

Criteria	JORC Code explanation	Commentary
		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 alongy 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 reported if material.*	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.
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

Criteria	JORC Code explanation	Commentary
		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

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		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 breccisa (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:* ○ *easting and northing of the drill hole collar* ○ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* ○ *dip and azimuth of the hole* ○ *down hole length and interception depth* ○ *hole length.*	Not applicable as no exploration results are being reported.

Criteria	JORC Code explanation	Commentary
	• *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does 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*	Not applicable as no exploration results are being reported.

Criteria	JORC Code explanation	Commentary
	hole collar locations and appropriate sectional views.	
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.*	Metallurgical test results reported in body of text. Testing protocols are as follows: RAS LG All test work carried out by ALS Perth Comminution test work consisted of Bond Rod Mill index assessment of one sample. Leach Recovery assessment was conducted via the following test protocol: (1) The sample was ground to the target grind size (nominally P80 106 µm) with the ground sample being transferred into a suitably sized plastic bottle, together with a sufficient quantity of Perth tap water to establish 45% solids (w/w). (2) Sufficient hydrated lime was added to the slurry to establish a pH of 10.0, and the slurry was thoroughly agitated for 5 minutes. (3) The pH of the slurry sample was measured again, and if necessary, more lime was added to achieve a pH of 10.0. (4) An addition of solid sodium cyanide was made to the slurry sample to establish an initial nominal cyanide solution strength of 0.05% (w/v). (5) Activated carbon was added to the resulting leach slurry at 20 g/L. (6) The leach slurry was sparged with oxygen gas at each checkpoint. (7) Intermediate 30 mL solution aliquots were removed after times of 2, 4, 8, 12 and 24 hours had elapsed. These were taken to provide solution samples for solution cyanide strength determination via titration with silver nitrate, and also for gold, silver, and copper analysis. Additionally, sub-samples of carbon were taken from the solution, dried, weighed and

Criteria	JORC Code explanation	Commentary
		submitted for gold, silver and copper analysis. At each sampling interval, the pH and dissolved oxygen levels of the pulp were also measured, and if necessary, more lime and cyanide were added to maintain a pH of higher than 9.5 (up to 10.0) and to maintain a cyanide solution strength of 0.03% (w/v). (8) At the termination of the test (24 hours), the terminal pH, oxygen, and cyanide levels were determined, and a solution sample was taken for gold, silver, and copper analysis. (9) The residual slurry sample was passed through a screen to recover the carbon, with the screen undersize being filtered, washed, and dried to provide leach residue solids. A sub-sample from the leach residue was submitted for gold, silver, and copper analysis. The captured carbon was washed, low-temperature dried and weighed, with a sub-sample being submitted for gold, silver, and copper analysis. SRX All test work carried out by IMO Perth Comminution test work consisted of Crusher Work Index, SAG Circuit Specific Energy and Bond Ball Mill Index assessments. Gravity Concentration. Gravity concentration recovery was conducted via the following test protocol: Bulk grind to P80 300µm Single pass through 3” Knelson Concentrator Intensive leach (Acacia conditions) of concentrate Combination of Leach and Knelson tails Leach and Tail assays Calculation of recovery Leach Recovery Grind Optimisation Various grind sizes were tested (P80 75, 106 and 150 µm) while leach variables were maintained at the following:

Criteria	JORC Code explanation	Commentary
		• 500 ppm NaCN initial, maintained at 300 ppm; • Dissolved oxygen maintained between 8-10 mg/L by air addition; • 40% solids in Perth tap water; • pH maintained at 9.5-10.0 with lime; • Monitoring at 2, 4, 8, 24 and 48 hours Reagent Optimisation The addition of carbon, lead nitrate and increased dissolved oxygen were assessed to gauge the effects on gold recovery. Only carbon was found to have a positive effect on recovery. Variability Leach Assessment Each variability gravity tailings sample was assessed via a single cyanide leach test with the following optimised parameters: • 500 ppm NaCN initial, maintained at 300 ppm; • Grind size of P80 106 µm; • 40% solids in Perth tap water; • pH maintained at 9.5-10.0 with lime; • Dissolved oxygen maintained at 8-10 ppm with air; • Carbon addition of 20 g/L; • 24-hour leach duration
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.* • *Data validation procedures used.*	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. 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.*	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

Criteria	JORC Code explanation	Commentary
	• *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.*	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 2023.2.0) using the vein interpretation function. Intervals were tagged as one of eleven domains based on gold grade, logged vein type. Ten low grade domains (LG1 to LG9 and the steep western domain) were created. A single high grade domain was interpreted in the core of the LG1 domain to enclose a continuous zone of mineralisation above about 1g/t. The high grade domain was created to prevent the high grade data having excessive influence outside the zone of high grade mineralisation. Due to the nuggety nature of the mineralisation some intervals below these cut off grades were included in the domains. Conversely, sporadic high grade samples also exist within the low grade domains, but these do not form continuous zones that may be confidently interpreted at the scale of the drill spacing. Not all mineralisation was included in the geological interpretation. Scattered, discontinuous assays were excluded from the wireframe model. Areas of consistent waste within the mineralised wireframes were modelled and removed from the mineralised volume. The edges of the mineralised wireframes were controlled with a combination of boundary strings and HW/FW control points. Wireframes were terminated less than 50% of the hole spacing distance beyond the last drill hole intersection. In the HW of the deposit the Thompsons Gorge Fault (TGF) truncated the mineralised wireframes. The geometry of the main zone immediately below the TGF is well defined with alternative interpretations unlikely. Alternative interpretations of the gold mineralization geometry deeper (more than about 40 m) below the TGF and in the steep western 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. 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 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

Criteria	JORC Code explanation	Commentary
		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. 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,*	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.

Criteria	JORC Code explanation	Commentary
	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.* • *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.*	Domains LG2 to LG9 were combined for statistical and geostatistical analysis. The coefficient of variation (CV) of the composites at RAS was 3.5 in the high grade domain, 2.8 in LG1 and 5.5 in domains LG2 to LG9 combined. Outlier grade limits were determined from log histograms, cumulative probability plots, assessment of the reduction in CV versus metal lost and then checked visually for spatial continuity. The outlier grades were then used to cut extreme grades prior to use in grade interpolation. The top cuts applied were 60 g/t Au in the high grade domain and 20 g/t Au in the low grade domains. After top cutting the CV composites reduced to 1.5 in the high grade domain, 2.0 in LG1 and 2.0 in domains LG2 to LG9 combined. Variogram models were determined from experimental correlograms of composites below the outlier limit grade for the high grade, LG1 and LG2-9 combined domains. There are insufficient data in the steep western domain to create robust experimental variograms, therefore the LG2-9 domain variogram model was appropriately rotated to reflect the geometry of the steep domain. The variogram model had a relative nugget effects of 55% to 75%. The major axes typically plunged 0 to 10 degrees towards 000 to 010 and were parallel to the intersection of the TGF and splay shears. The semi-major axes plunged 15 to 25 degrees towards 080. The minor axes were orthogonal to the major and semi-major axes. Together, the major and semimajor axes approximate the orientation of the splay shears. The total ranges were 50 m to 75 m for the major axes, 30 m to 40 m for the semi-major axis and 10 m to 15 m in the minor axis directions. 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 50 % 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. The blocks were interpolated by ordinary kriging of the top cut composites in two passes. The first pass used a minimum of 4 and a maximum of 15 composites from within a 100m by 100 m by 20 m ellipsoid oriented parallel to the variogram model. A maximum of 3 composites were used per hole. Gold domain boundaries were treated as hard boundaries. A small proportion of the blocks were not interpolated by pass 1, mostly in the margins of the LG1 domain at the northern (deepest) end of the mineralisation. A second interpolation pass using the same parameters as pass one except the search ellipsoid was expanded to 150 m by 150 m by 30 m and the maximum per hole restriction was removed. Check estimates were completed on the RAS MRE as follows: combining the LG1 and HG domains; outlier restriction at 12.5 m; and nearest neighbour interpolation. 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, February 2024 and July 2024. There has been no production from the BOGP to allow reconciliation of the

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Criteria	JORC Code explanation	Commentary
		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. The MRE is geologically controlled by the use of domains interpreted with reference to the geological model. The block model was validated against drilling grades visually in section and in plan, using swath plots, and by comparison of the block model volumes to domain wireframe volumes. No reconciliation data is available as mining has not commenced. 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 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.

Criteria	JORC Code explanation	Commentary
		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. 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 (0.50 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 A$4,390/oz escalated by 30% to allow for the reasonable prospects test.

SANTANA MINERALS LIMITED

Criteria	JORC Code explanation	Commentary
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. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	No allowance has been made for mining dilution or mining recovery.
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 92%. Further work is ongoing 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*	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

SANTANA MINERALS LIMITED

Criteria	JORC Code explanation	Commentary
	reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	Other studies have commenced as mine studies advance including noise, traffic, lighting and visual.
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 2,653 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*	Earlier iterations of the RAS MRE were reviewed by AMC Consultants in 2023 and RSC Consultants in 2024. AMC 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. RSC concluded that extreme grades were not adequately restricted. This issue has been addressed by the application of top cuts (previously outlier restriction was used) and the use of a high grade domain.
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	Category	tonnes (Mt)	Au grade (g/t)	Contained Gold (koz)
RAS	Indicated	18.9	2.5	1,538
	Inferred	7.6	2.2	542
RAS Total	Indicated and Inferred	26.5	2.4	2,080
CIT	Inferred	1.2	1.5	59
SRX	Indicated	2.2	0.8	54.7

Criteria	JORC Code explanation	Commentary

SRX	Inferred	2.9	1.0	90.5
SRX Total	Indicated and Inferred	5	0.9	145
SRE	Indicated	0.4	0.8	10.3
SRE	Inferred	1.1	1.2	42
SRE Total	Indicated and Inferred	1.5	1.1	52
BOGP Total	Indicated	21.5	2.3	1,603
	Inferred	12.7	1.8	734
BOGP Total	**Indicated and Inferred**	**34.3**	**2.1**	**2,337**

- The Mineral Resources are reported inclusive of Ore Reserves

Criteria	JORC Code explanation	Commentary
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 under supervision by Damian Spring who visits the site frequently, working and residing nearby the project.
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 – Updated 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:

Criteria	JORC Code explanation	Commentary
		○ RAS Open pit: 0.5g/t ○ RAS Underground 1.75g/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 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.*	• The resource block models as received were re-blocked for open pit mining assessment to simulate the assessed minimum mining unit (7.5m x 7.5m x 2.5m). • The Lerch Grossman algorithm (LG) was used to check manual pit designs using sets of possible open pit mining shells for RAS. This process was performed on all Indicated resources as well as a high grade – high confidence core wireframe at RAS. • 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 utilized. • 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 100 tonne haul trucks loaded by 250 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)
Southwest	350 to 065	All	36.8	50	7.5	15
West	065 to 160	All	34.8	50	9	15
Northeast	160 to 235	All	45.7	60	6	15
East	235 to 350	All	42.5	60	7.5	15

(1) Slope aspect measured as the direction the wall dips towards.
(2) Inter-ramp angle.

Criteria	JORC Code explanation	Commentary
		(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 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
- 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.

Criteria	JORC Code explanation	Commentary
		• 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 92% Indicated ○ RAS UG scheduled is 87% Indicated ○ SRX scheduled is 99% 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, refueling 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 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*	• The final selected flowsheet is appropriate to the style on mineralization and involves: ○ a three stage crush (121mm); ○ single stage ball mill (p80 106 micron), with the addition of another 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 RAS of 100kg plus variability samples (10).

Criteria	JORC Code explanation	Commentary
	appropriate mineralogy to meet the specifications?	The 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 RAS master and variability testwork is complete. The variability samples supported the Master composite recoveries at 106 micron grind of 93.4% 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

SANTANA MINERALS LIMITED

Criteria	JORC Code explanation	Commentary
		>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. • 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 1.2Mtpa processing facility ○ 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

Criteria	JORC Code explanation	Commentary
		○ A 100l/sec borefield and pipeline to site ○ Administration, mining and processing offices ○ A warehouse ○ Mobile fleet workshops for Open pit and Underground ○ Refueling and plant washdown facilities ○ Metallurgical and assay laboratory • 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 May 2025. ○ TSF estimate by Engineering Geology Limited, October 2024, updated May 2025. ○ 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: ○ Sustaining capital projects

Criteria	JORC Code explanation	Commentary
		○ 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 Valoreum 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 Valoreum 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: • Au Price: US$2,000/oz ○ US$:NZ$ exchange: 0.58 Metal prices assumed for Base Case of the Pre-feasibility are: • Au Price: US$2,000/oz ○ US$:NZ$ exchange: 0.58 • Metal Price and exchange rate assumptions have been benchmarked against industry peers (for Au)
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 gold doré sales, there is a well-established and transparent market.

SANTANA MINERALS LIMITED

Criteria	JORC Code explanation	Commentary
	• *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	

Economic	• *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.*
	• *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*
		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;
		• Gold prices assumed for Base Case of the Pre-feasibility Study (no escalation has been applied to selling prices);
		• The base case NPV used
		  ○ Au Price: US$2,220/oz
		    ○ US$:NZ$ exchange: 0.58
		• A discount rate of 6.5% has been applied to calculate NPV
		• The base case post tax NPV is $AUD780M
		• Sensitivities have been assessed at various selling prices for Au

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.*	• 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

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	• *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	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 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

Criteria	JORC Code explanation	Commentary
		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.* • *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).*	• Material classified as Indicated Mineral Resources has been converted to Probable Ore Reserve • 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*	• The design, schedule and financial model for the BOGP has been completed to a Pre-feasibility standard with a +/-15% 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.

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	Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	