Level 28, 88 Phillip St
Sydney NSW 2000
P: +61 2 8316 3997
F: +61 2 8316 3999
Info@silvermines.com.au
www.silvermines.com.au

21 July 2026

BOWDENS DEFINITIVE FEASIBILITY STUDY CONFIRMS ROBUST ECONOMICS AS RESERVES LIFT TO 93.5M OZ OF SILVER

Silver Mines Limited ("Silver Mines" or the "Company") is pleased to announce the results of its Definitive Feasibility Study ("DFS" or the "Study") for the Bowdens Silver Project ("BSP" or the "Project").

The Company is also pleased to announce the release of an updated Mineral Resource Estimate and Ore Reserve Estimate for the BSP.

HIGHLIGHTS

Definitive Feasibility Study and Life of Mine Plan

- **The DFS positions the BSP as one of the world's premier undeveloped silver projects. Silver Ore Reserves have been increased to 47.9Mt at 60.8 g/t Ag for 93.5Moz with 91% of the Project revenue expected to be derived from silver.**
- **The initial stage 1 of the proposed development will focus on a subset of the Life of Mine (LOM) plan containing 29.9Mt at 68 g/t Ag for 65.4Moz, over a 16-year period with a low strip ratio of 1.47:1.0 (waste:ore).**
- **A second stage supported by pre-feasibility study level works, subject to approvals, is expected to ultimately result in mining and processing of the remaining 18.3Mt for a total mine operational life of 26 years with a LOM strip ratio of 1.69:1.0.**
- **Initial capital costs are estimated at A$455m and All-In-Sustaining-Costs (AISC) estimated at A$32.91/oz (US$23/oz). LOM sustaining capex estimated at A$91m.**
- **Economic results for the LOM using a silver price of US$45/oz and A$:US$ exchange rate of $0.70 include the following financial metrics:**
    - **Life of mine operating margin of A$2.39bn (A$4.04bn at spot[1]).**
    - **Undiscounted, pre-tax operating surplus of A$1.94bn (A$3.59bn at spot[1]).**
    - **Pre-tax NPV$_{5\%}$ of A$1.04bn and IRR of 31.5% (A$2.01bn and 49% at spot[1]).**
    - **Profitability Index of 2.64x (4.17x at spot[1]).**
    - **Payback of 3.0-yrs from start of production (2.2-years at spot[1]).**
    - **NPV to Pre-production capital ratio of 2.3x (4.4x at spot[1]).**

[1]Spot Price is based on Silver Price of USD$61/oz and exchange rate of AUD/USD $0.69.

- **Key metrics over the first 5-years include:**
    - **Average production of 4.7Moz silver.**
    - **AISC of A$20.47/oz / US$14.33/oz.**
    - **Pre-tax operating cash flow of A$193m per annum (A$300m at spot).**
    - **95% of revenue from NSW Critical Minerals silver and zinc (silver at 94%).**
- **The LOM design is based on a silver price of US$35/oz.**
- **Over time, studies will be undertaken to investigate potential options for future production optimisation via ore-sorting and plant expansions.**

Project Approvals and Permits

- **The priority for the Bowdens team remains securing the Development Consent ("Consent") from the NSW state government. While it is not possible to provide a definitive timeline on the regulatory process, the Company remains confident that the Consent will be returned in due course.**
- **Once the Consent is obtained, the Company will be able to update timelines on finalising project approvals including the federal permit in accordance with the *Environmental Protection and Biodiversity Conservation Act 1999* (Cth) and the subsequent Mining License approval from the state government.**

Next Steps

- **With the DFS now finalised, work will commence on advanced engineering studies relating to the Front-End Engineering and Design (FEED) process.**
- **In addition, the Company is reviewing options for its biodiversity offsetting strategy in order to finalise commitments prior to commencement of any development activities.**

Ore Reserve Estimate

- **Ore Reserves total 47.9 million tonnes at 60.8 g/t silver, 0.36% zinc and 0.26% lead for an increase of 46% to the ore tonnes relative to the 2024 Ore Reserve[2] estimate with an A$30 per tonne NSR cut off.**
- **The contained silver in Ore Reserves has increased by 30% to 93.5 million ounces and silver is now forecast to contribute around 91% of the revenue in the mine plan, up from 85% in the 2024 Ore Reserves estimate.**

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[2] Silver Mines Limited (ASX:SVL) release: "Bowdens Silver Project Ore Reserves Increased to 71.7Moz Silver" dated 10 January 2025.

Mineral Resource Estimate

- **The 2026 Mineral Resource Estimate ("MRE") above 25g/t AgEq[3] cut-off grade reflects additional drilling (including a further 2,248 assays from 2,265m drilled), updated geological domains, improved metallurgy and changes to metal pricing assumptions.**

- **The updated MRE underpins the increase to the Reserves as detailed above.**

Silver Mines Managing Director, Jo Battershill commented: "The completion of the Bowdens DFS is a significant milestone for the Company and its shareholders. The Board of Silver Mines would like to extend its appreciation to both the shareholder base for its ongoing patience, and the site team for completing this extensive package of works – while facing significant distractions related to the approval process.

With the Project's consent now in the hands of the NSW government and Independent Planning Commission, it is important to focus on the economic benefits that the BSP can bring to the community and investors alike. With the regional coal industry under pressure from decarbonisation of the economy, the benefits of this project to the local community are increasingly important.

With a potential mine life now anticipated to be more than 25 years, the BSP is expected to deliver generational employment opportunities for more than 200 locals at steady state operations, hand-in-hand with strong financial returns to the state government from taxes and royalties, the local council through the voluntary planning agreement and investors alike.

We very much look forward to progressing the BSP through into the FEED process and subsequent development."

[3] Refer Notes on Resource Table 5 for AgEq calculations.

DFS Introduction

The recently completed DFS for the Bowdens Silver Project has resulted in significant improvements to the economic parameters of the Project.

The Bowdens Ore Reserve Estimate included in this announcement will require a staged development.

The DFS for the BSP specifically references a 29.9Mt ore mine design, which is less than the 2026 LOM design which contains 48.2Mt ore. The DFS mine design only includes Stage 1 and has been intentionally limited in scale to match the Stage 1 tailings landform which has been designed to hold 30Mt of filtered tailings. Stage 2 (which includes the additional 18.3Mt for the total 48.2Mt 2026 LOM Plan) requires a Stage 2 filtered tailings landform, all other mine infrastructure is the same as the DFS. The design of the Stage 2 filtered tailings landform is at pre-feasibility study level only. All other aspects of the 2026 LOM design are at DFS level. As a result, while the combined LOM financial metrics (Tables 1 and 3, "LOM Stage 1 and 2 Development" columns) reflect the full 26-year mine plan, investors should note that the Stage 2 component carries a higher level of uncertainty consistent with its pre-feasibility study status.

DFS Summary

The updated studies for the BSP have demonstrated strong economics and significant improvements to the Optimisation Study ("2024 Study") from December 2024.[4]

The key economics of the BSP from the DFS and 2026 LOM are set out in Table 1 below:

Table 1: Bowdens Silver Project Economic Parameters

Parameter	Unit	2024 Study (Dec 2024)	DFS Stage 1 Development (Jul 2026)	LOM Stage 1 and 2 Developments (Jul 2026)	Difference (DFS vs 2024 Study) Stage 1 Development	Difference (LOM vs 2024 Study) Stage 1 and 2 Developments
Processing Plant, Infrastructure and Sustaining Capital						
Process Plant (2.0Mtpa)	A$m	93.8	180.5	180.5	+92%	+92%
Non-Processing Infrastructure	A$m	122.1	113.3	113.3	-7%	-7%
Other	A$m	78.6	107.8	107.8	+37%	+37%
Indirect	A$m	37.1	52.9	52.9	+42%	+42%
Total (including contingency)	A$m	332	455	455	+37%	+37%
Sustaining Capital (LOM)	A$m	14.8	23.5	90.8	+52%	+513%
LOM Capex	A$m	346	478	545	+38%	+58%

[4] Silver Mines Limited (ASX:SVL) release: *"Optimisation Study Highlights Robust High-Margin Ag Project"* dated 20 December 2024.

Parameter	Unit	2024 Study (Dec 2024)	DFS Stage 1 Development (Jul 2026)	LOM Stage 1 and 2 Developments (Jul 2026)	Difference (DFS vs 2024 Study) Stage 1 Development	Difference (LOM vs 2024 Study) Stage 1 and 2 Developments
Production Summary						
Life of Mine (LOM)	Years	16.0	16.75	26.0	+0.75 years	+9.25 years
LOM Strip Ratio	W:O	1.49	1.47	1.69	-1%	+13%
Processing Rate	Mtpa	2.0	2.0	2.0	n/c	n/c
LOM Payable Silver Production	Moz	50.3	51.1	73.2	+2%	+46%
LOM Payable Zinc Production	Kt	31	27	46	-13%	+48%
LOM Payable Lead Production	kt	56	49	84	-13%	+50%
LOM Average Silver Production	Mozpa	3.7	3.8	3.2	+3%	-14%
LOM Average Silver Recovery	%	82.7	82.8	82.1	+1%	-1%
LOM Operating Costs						
Mining (including rehabilitation)	A$/t milled	17.99	19.85	20.37	+10%	+13%
Processing	A$/t milled	20.78	20.73	20.71	0%	0%
Administrative	A$/t milled	5.65	6.02	6.01	+7%	+6%
C1 Costs	A$/oz	23.28	26.91	29.75	+16%	+28%
C1 Costs	US$/oz	15.59	18.84	20.82	+21%	+34%
AISC	A$/oz	24.81	29.32	32.91	+18%	+33%
AISC	US$/oz	16.62	20.53	23.04	+24%	+39%
Project Economics						
LOM Pre-Tax Operating Margin	A$m	948	1,812	2,390	+91%	+152%
$NPV_{5\%}$ (Pre-Tax)	A$m	359	877	1,043	+144%	+191%
$NPV_{5\%}$ (Post-Tax)	A$m	253	626	736	+147%	+191%
IRR (Pre-Tax)	%	21.0	31.5	31.5	+50%	+50%
Payback (Post-Tax)	Years	3.9	3.0	3.0	-23%	-23%
Profitability Index	(x)	1.76	2.40	2.64	+36%	+50%
NPV (Pre-Tax) / Pre-Prod Capex	(x)	1.08	1.93	2.30	+78%	+113%

n/c = no change.

The DFS commenced in H1 2025 and has been running concurrently with ongoing permitting activities. A number of scoping changes relative to the 2024 Study were made during the DFS. The key changes include:

- Process flow sheet updated to minimise overgrinding of soft material. Changes include:
    - Addition of secondary crusher;
    - Inclusion of fine ore screening after the SAG mill to divert final size material directly to the flotation cells; and
    - Ability to by-pass fine ore screening when not needed.
- Tailings filtration methodology changed from a two-step process to a single step process. Optimisation plan included a desliming step with different dewatering methods for the fines and coarse product. DFS plan is to pressure filter the entire tails stream in a single step.

Revenue Assumptions

The commodity price and FX assumptions used in the DFS and Reserve financial models were derived from Consensus Economics and are set out in Table 2 below:

Table 2: Bowdens Silver Project FX Assumptions

Parameter	Unit	2024 Study (Dec 2024)	DFS Stage 1 Development (Jul 2026)	LOM Stage 1 and 2 Developments (Jul 2026)	Difference (DFS vs 2024 Study) Stage 1 Development	Difference (LOM vs 2024 Study) Stage 1 and 2 Developments
Commodity Prices						
Silver Price	US$/oz	29.00	45.00	45.00	+55%	+55%
Zinc Price	US$/lb	1.35	1.35	1.35	n/c	n/c
Lead Price	US$/lb	1.05	0.90	0.90	-14%	-14%
Exchange Rates						
AUD/USD		0.67	0.70	0.70	+4%	+4%

n/c = no change.

The LOM design (Reserve) has been based on a silver price of US$35/oz. Development studies indicate that a silver price assumption of US$50/oz to the Whittle pit optimisation model would increase the 'mining shell'[5] by more than 65% to ~80Mt for ~127Moz Ag. The US$50/oz shell was generated as a sensitivity case to demonstrate the extent of future pit cutbacks that

[5] A Whittle pit shell is an optimized theoretical mining outline, not a classified reserve or resource. It is an economic guide generated by mine planning software to show the maximum possible pit limits at a set commodity price. The Whittle pit shell should not be relied upon as an indication of the Company's declared reserves or resources.

may be required under higher silver price assumptions and to ensure mine and process infrastructure has been positioned appropriately.

Production Summary

The following charts provide a visual summary of the key production estimates.

Figure 1: Annual mined ore tonnes and Ag grade (2026 LOM versus 2024 Study)

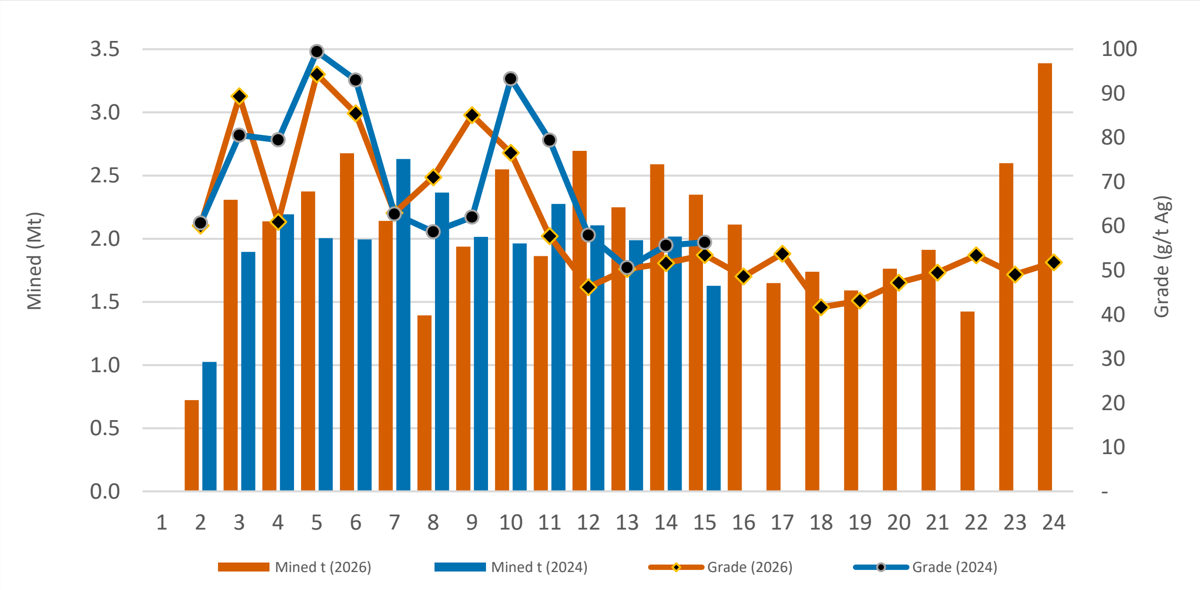


Figure 2: Annual processed tonnes and Ag grade (2026 LOM versus 2024 Study)

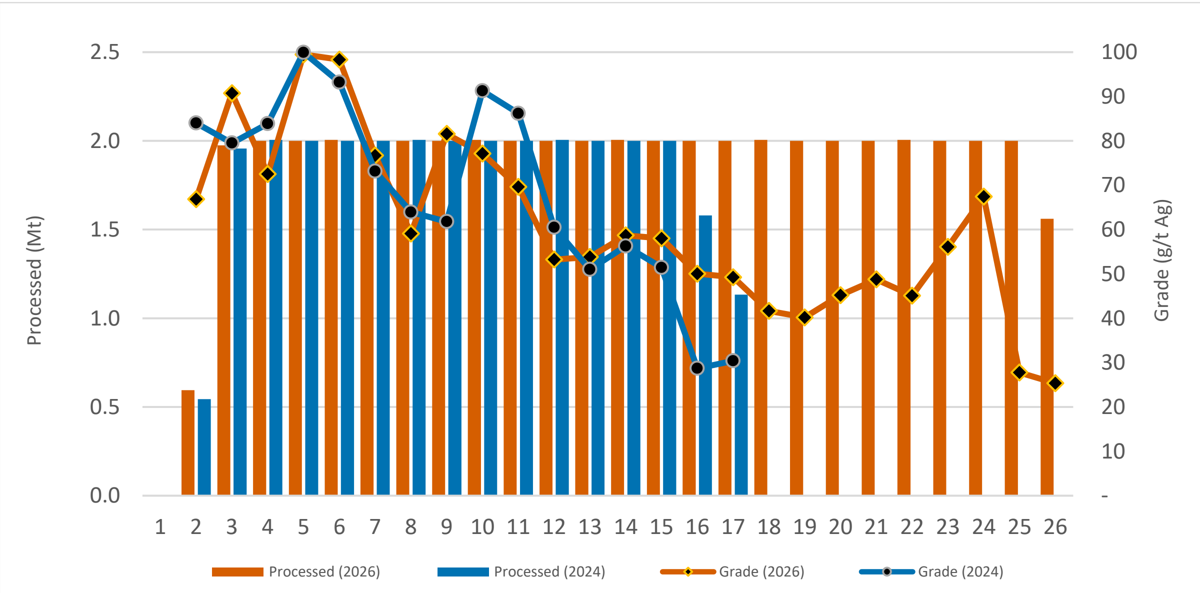


Figure 3: Annual Ag output and Cumulative LOM (2026 LOM versus 2024 Study)

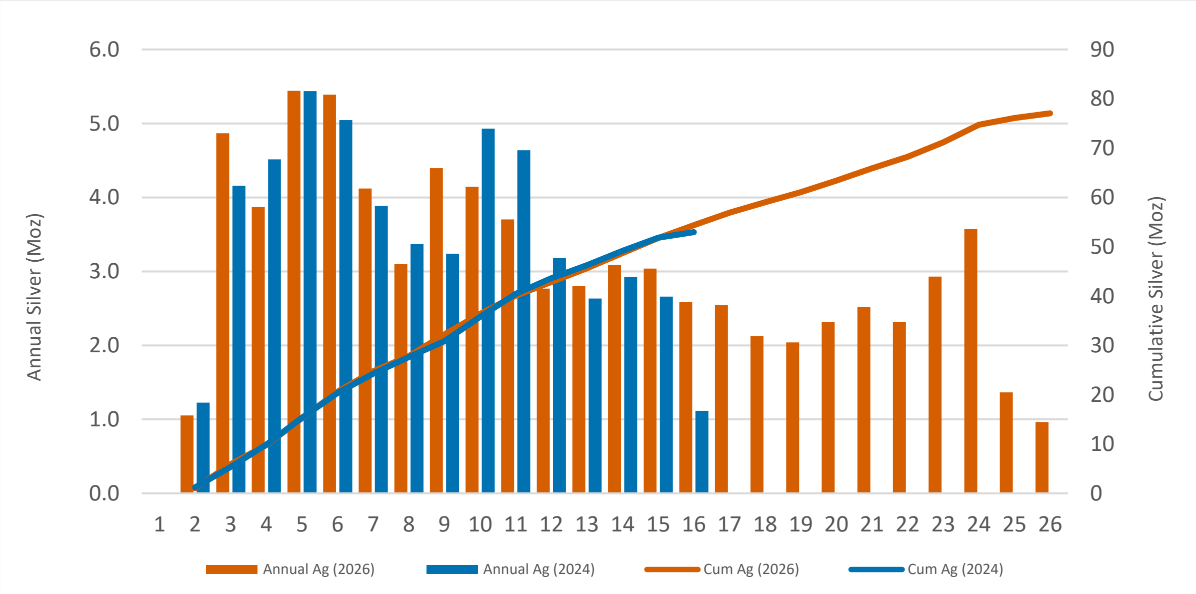
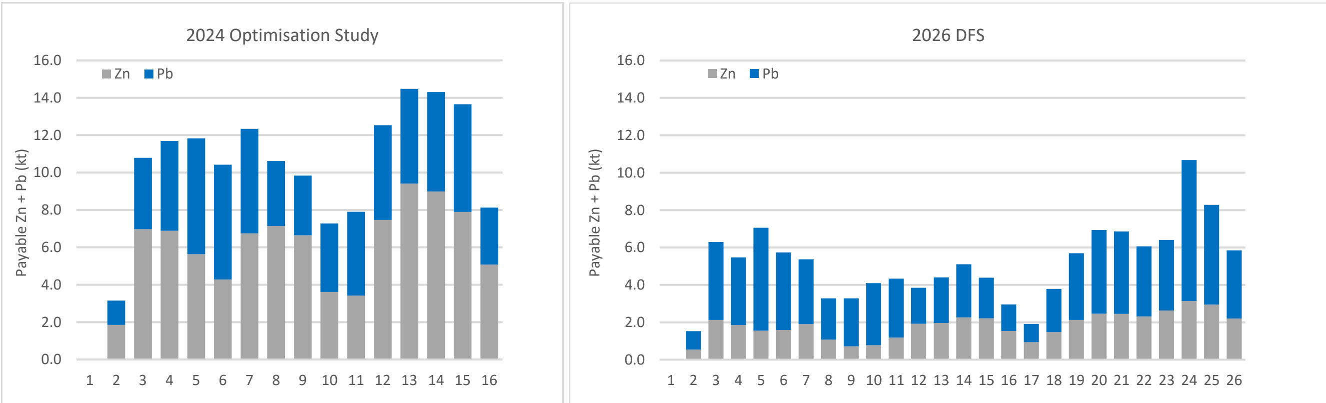


Figure 4: Annual Base Metal output (2026 LOM versus 2024 Study)



Capital Costs

The pre-production capital cost estimate of A$455 million (including mining pre-production) represents a 37% increase when compared to the 2024 Study capital cost estimate of A$331 million. Annual mining capex inflation has been running at between 10 – 15% since the 2024 Study – the cumulative impact of this inflation is responsible for a significant element of the higher capital estimate. The DFS highlighted a scope change from the 2024 Study requiring additional infrastructure associated with the filtered tailings.

From a sustaining capital perspective, the additional mine life derived from the LOM design leads to a requirement for expanded filtered tailings landform infrastructure beyond year 16. Subsequently, LOM sustaining capex has been increased from A$15m in the 2024 Study to A$91m over the potential 26-year life proposed by the LOM design. The design of the Stage 2 filtered tailings landform is at pre-feasibility level.

Operating Costs

Mining costs are based on a primarily owner operator cost model developed from first principles. The operating plan includes progressive rehabilitation, which is included in the mining operating costs.

Overall operating costs are also impacted by lower by-product credits, due to a lower lead price assumption and a higher AUD/USD FX forecast, and higher royalties relative to the 2024 Study due to the increased silver price.

Project Economics

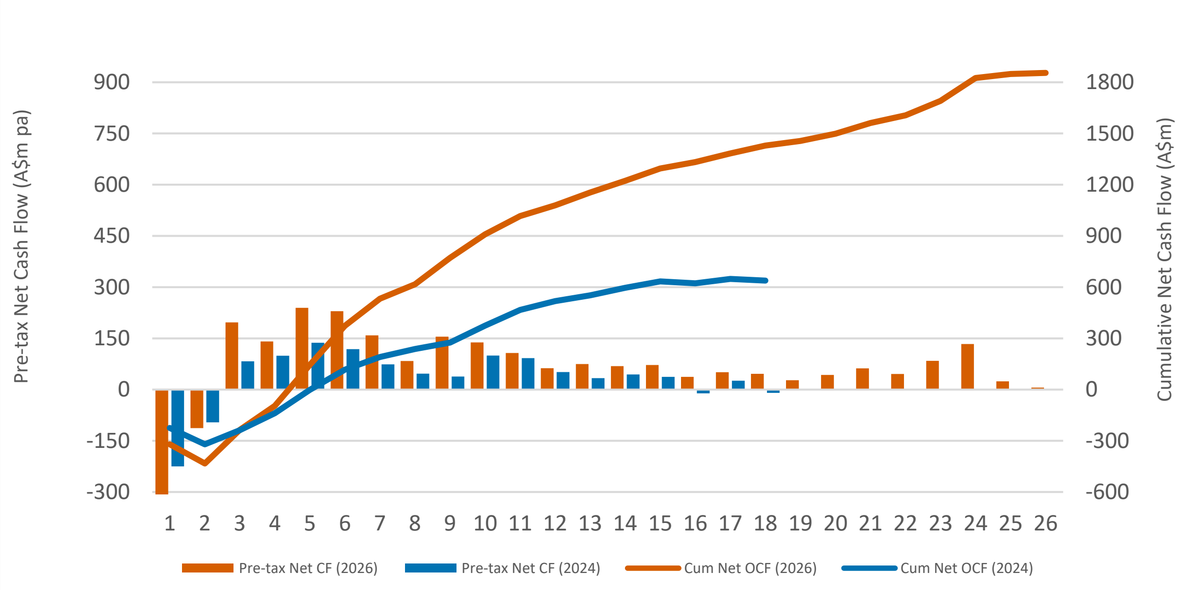
The financial model developed for the Project based on the 2026 LOM plan indicates a total operating margin of A$2.39 billion, with project payback 3.0 years from the commencement of production. Project Pre-Tax NPV_5 is estimated at A$1.04 billion, with an IRR of 31.5%. Project Post-Tax NPV_5 is estimated at $736 million, with an IRR of 26%.

Table 3: Bowdens Silver Project Economics

Project Economics						
Parameter	**Unit**	**2024 Study (Dec 2024)**	**DFS Stage 1 Development (Jul 2026)**	**LOM Stage 1 and 2 Developments (Jul 2026)**	**Difference (DFS vs 2024 Study) Stage 1 Development**	**Difference (LOM vs 2024 Study) Stage 1 and 2 Developments**
Revenue	A$m	2,510	3,538	5,175	+41%	+106%
Operating Expenses	A$m	1,562	1,726	2,785	+10%	+78%
Operating Margin	A$m	948	1,812	2,390	+91%	+152%
Undiscounted Cash Flow Pre-Tax	A$m	631	1,434	1,945	+127%	+208%

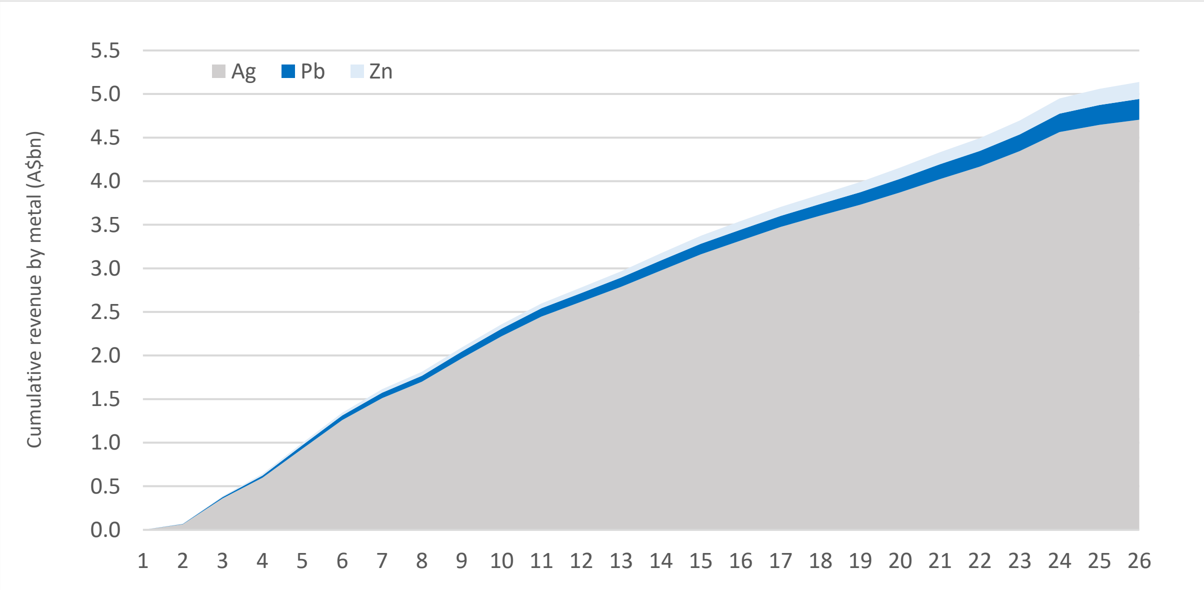
Project Economics						
Parameter	**Unit**	**2024 Study (Dec 2024)**	**DFS Stage 1 Development (Jul 2026)**	**LOM Stage 1 and 2 Developments (Jul 2026)**	**Difference (DFS vs 2024 Study) Stage 1 Development**	**Difference (LOM vs 2024 Study) Stage 1 and 2 Developments**
Undiscounted Cash Flow Post-Tax	A$m	469	1,056	1,412	+125%	+201%
$NPV_{5\%}$ (Pre-Tax)	A$m	359	877	1,043	+144%	+190%
$NPV_{5\%}$ (Post-Tax)	A$m	253	626	736	+147%	+191%
IRR (Pre-Tax)	%	21	31.5	31.5	+50%	+50%
IRR (Post-Tax)	%	18	26	26	+44%	+44%
Payback (Post-Tax)	Years	3.9	3.0	3.0	-23%	-23%
Profitability Index	(x)	1.76	2.40	2.64	+36%	+50%
NPV (Pre-Tax) / Pre-Prod Capex	(x)	1.08	1.93	2.30	+78%	+113%

Figure 5: Pre-tax Cash Flow – Annual and Cumulative (2026 LOM versus 2024 Study)



In terms of product value, under the 2026 LOM plan, the combination of silver and zinc, both of which are included on the NSW Critical Minerals and High-Tech Metals list, account for approximately 95% of revenue from the Bowdens Silver Project. Silver alone contributes more than 91% of revenue from the 2026 LOM plan, making the Bowdens Silver Project one of the most leveraged silver development projects in the world.

Figure 6: Cumulative Revenue by Metal



Next Steps

The short-term priority for the Bowdens team remains securing the Consent from the NSW state government. While it is not possible to provide a definitive timeline on the regulatory process, the Company remains confident that the Consent will be obtained in due course.

Once the Consent is obtained, the Company will be able to provide an updated timeline on finalising project approvals including the federal permit in accordance with the *Environmental Protection and Biodiversity Conservation Act 1999 (Cth)* and the subsequent Mining Lease approval from the state government.

With the DFS now finalised, work will commence on advanced engineering studies relating to the Front-End Engineering and Design (FEED) process. In addition, the Company is reviewing options for its biodiversity offsetting strategy with the aim of finalising potential commitments prior to commencement of any development activities. It is anticipated that advancing these work streams will enable the Company to continue discussions with potential funding partners.

DFS Cautionary Statement

Based on technical and economic studies, the Definitive Feasibility Study referred to in this DFS Summary Report examines the potential of developing the Bowdens Silver Project by constructing a single open pit Silver (Ag) mine also containing Zinc (Zn) and Lead (Pb) with an onsite processing plant and waste storage facilities to produce a high silver grade bulk concentrate for export and/or domestic refining and sale. The Definitive Feasibility Study outcomes, production targets and forecast financial information referred to in this document are based on industry standard technical and economic assessments. The Definitive Feasibility Study has been completed to a level of accuracy of -10/+20% in line with typical Definitive Feasibility level study accuracy. Stage 2 of the mine design (18.3Mt) incorporates a filtered tailings landform at pre-feasibility study level and the cost estimates for that component are more uncertain.

Silver Mines has reasonable grounds for disclosing production targets, since approximately 92.6% of the LOM Production Target is in the Measured Mineral Resource category, and 6.8% is in the Indicated Mineral Resource category. Just 0.6% of the LOM Production Target is in the Inferred Mineral Resource category. There is a low level of geological confidence associated with inferred mineral resources and there is no certainty that further exploration work will result in the determination of indicated mineral resources or that the production target itself will be realised.

The Definitive Feasibility Study also includes assumptions about the availability of funding. To achieve the range of outcomes indicated by the DFS, funding in the order of A$455 million will likely be required. Investors should note that there is no certainty that Silver Mines will be able to raise that amount of funding when needed. It is also possible that such funding may only be available on terms that may be dilutive to, or otherwise affect, the value of Silver Mines existing shares. The Company may also consider value-realisation strategies including a sale of the Project, a partial sale or the introduction of a joint venture partner. If the Company pursues any such strategy, this could materially reduce the Company's proportionate ownership of the Project and any financial benefits it derives from it. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the DFS.

While Silver Mines considers all the material assumptions in the Definitive Feasibility Study to be based on reasonable grounds, there is no certainty that they will prove to be correct or that the range of outcomes indicated will be achieved. The Mineral Resources underpinning the production targets in the Definitive Feasibility Study have been prepared by a competent person in accordance with the requirements in the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2012 Edition ("JORC Code"). The economic outcomes associated with the Definitive Feasibility Study are based on certain assumptions made for commodity prices, exchange rates and other economic variables, which are not within the Company's control and subject to change from time to time. Changes in such assumptions may have a material impact on economic outcomes. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the Definitive Feasibility Study.

Ore Reserve and Mineral Resource Estimates

Silver Mines is also pleased to announce the release of an updated MRE ("2026 MRE") and Ore Reserve Estimate ("2026 Reserve") for the BSP.

The updated 2026 MRE reflects additional drilling, more accurate geological domaining, additional metallurgical recovery test work and updated metal pricing assumptions. The MRE has been updated by H&S Consultants in conjunction with the Company and work with Datarock, helping to improve estimation domain consistency and accuracy. Refer to JORC Table 1 Sections 1-3 in Appendix 2 for detail.

Accordingly, an updated Ore Reserve for the BSP has also been completed. The Company, in conjunction with Resolve Mining Solutions, produced a 2026 Reserve for the BSP based on the M & I component of the 2026 MRE, refer to Table 4 and Figure 7. This represents a snapshot of the current economically extractable Resources for a given set of cost and pricing assumptions and modifying factors in accordance with the JORC Code. Refer to JORC Table 1 Section 4 in Appendix 2 for specific detail.

Ore Reserve

The Bowdens Silver Ore Reserve is estimated at 47.9 million tonnes at 60.8 g/t silver, 0.36% zinc and 0.26% lead for 93.5 million ounces of silver, 172 kilo-tonnes of zinc and 127 kilo-tonnes of lead in contained metal. The Ore Reserve Estimate is shown in Table 4.

Table 4: July 2026 Ore Reserve*

	Reserve Grades				Contained Metal		
Classification	**Tonnes (Mt)**	**Ag (g/t)**	**Zn (%)**	**Pb (%)**	**Ag (Moz)**	**Zn (kt)**	**Pb (kt)**
Proven	44.6	61.7	0.37	0.27	88.5	165	122
Probable	3.3	47.3	0.21	0.15	4.9	7	5
Total	47.9	60.8	0.36	0.26	93.5	172	127

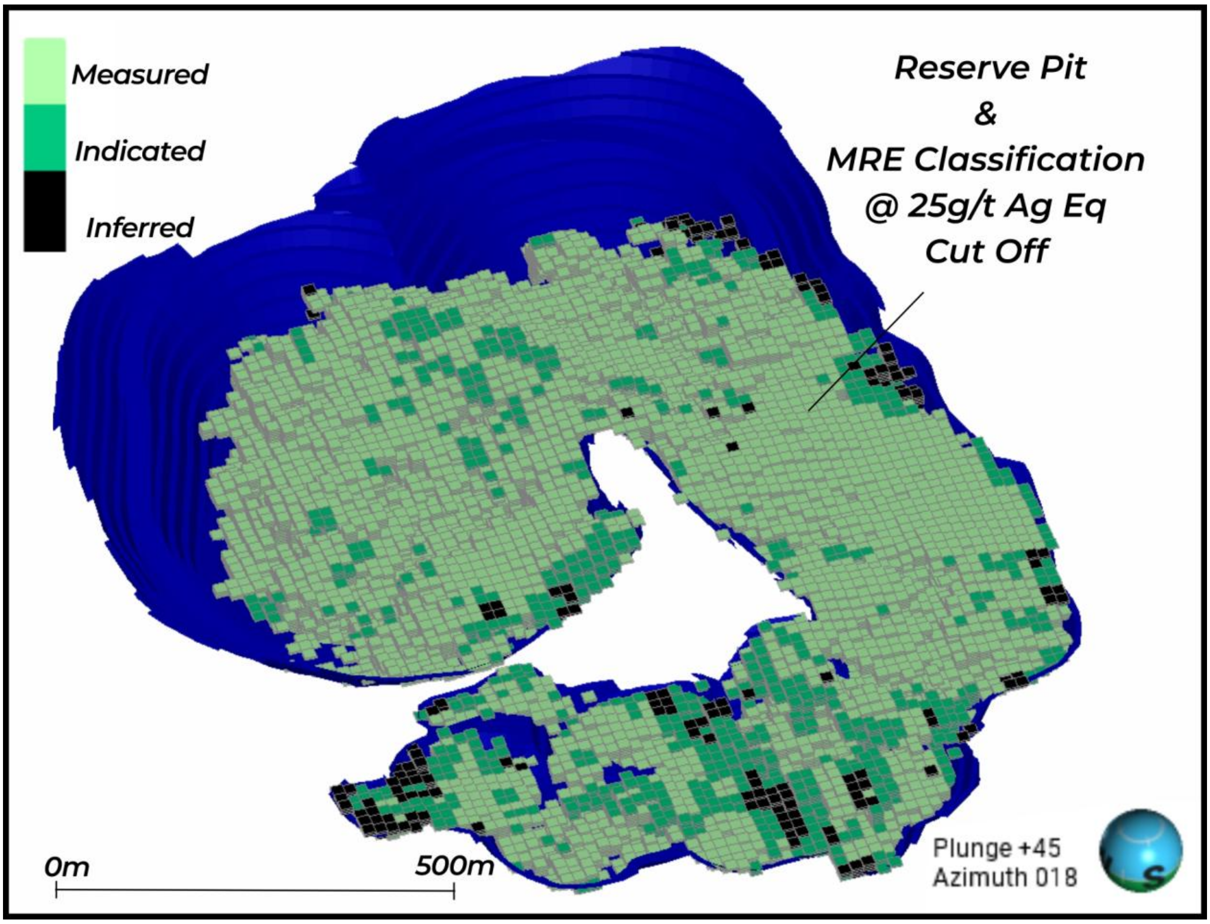
**Notes:*
1. *Ore Reserves are a subset of Mineral Resources.*
2. *Ore Reserves conform with and use the JORC Code definitions.*

3. *Ore Reserves are calculated using Silver, Lead and Zinc pricing of US$35/oz, US$0.95/lb and US$1.35/lb respectively.*
4. *Ore Reserves are calculated using a Net Smelter Return cut-off of A$30/t under these assumptions.*
5. *Tonnages are reported including mining dilution.*
6. *All figures are rounded to reflect appropriate levels of confidence which may result in apparent errors of summation.*

The 2026 Reserves were estimated from the 2026 MRE after consideration of the confidence levels of the resource categories and considering the relevant modifying factors.

No Mineral Resources classed as Inferred have been included in the 2026 Proven or Probable Reserves.

Figure 7: Schematic of Ore Reserve within Pit Designs by Resource Classification



Ore Reserve – Other Material Information Summary

A summary of other material information pursuant to ASX Listing Rules 5.9.1 is provided below. The Ore Reserve is based on the MRE. The Assessment and Reporting Criteria in accordance with the JORC Code and Guidelines are presented in Appendix 2 to this announcement.

Material Assumptions from the DFS

The 2026 Reserve update was completed by Resolve Mining Services Plus (“Resolve”).

The 2026 Reserve draws on updated data developed through the DFS, building on the 2024 Study. Financial modelling indicates that the 2026 Reserve is economically viable under consensus long-term metal prices. Revenue from Inferred Mineral Resources was not considered in the economic analysis.

Criteria Used for Classification

The 2026 Reserve, derived from Measured and Indicated Mineral Resources under the JORC Code, is classified as Proven and Probable. The Competent Person confirms that these classifications accurately reflect the technical and economic assessments.

Estimation Methodology

Mining operating costs are based on a comprehensive owner-mining model developed by Resolve from both first principle estimates, contractor quotes and detailed operating cost data provided by equipment manufacturers.

Key information provided by others includes:

- metal pricing, FX, marketing terms and administrative costs from Bowdens Silver;
- geotechnical study completed by Dempers and Seymour Pty Ltd;
- process plant design and operating cost estimate provided by Lycopodium; and
- metallurgical test work by KYSPYmet, ALS and Core Resources.

Net Smelter Return (“NSR”) calculations are applied to each MRE block based on cost factors, metallurgical response, and post marketing revenue.

Whittle software generated pit optimisation shells based on the Mineral Resource model are from NSR calculations and geotechnical inputs. A detailed pit design, based on the optimal pit shell, was developed which formed the basis of production scheduling and further economic evaluation.

Material Modifying Factors

The 2026 Reserves incorporate dilution, ore loss, and conversion from Measured and Indicated Resources after considering all technical, social, environmental, and financial factors. The 2026 Reserves’ economic viability is most sensitive to factors affecting revenue, including currency exchange rate, metallurgical recovery, and silver price.

The MIK recoverable model is based on a selective mining unit of 12.5m x 12.5m x 2.5m (half the parent block size in each direction) which accounts for internal dilution and some edge dilution, with no further mining dilution or ore loss applied during pit optimisation or subsequent mine planning.

Key project infrastructure items: process plant, filtered tailings thickening and filtration plant, filtered tailings landform, mine waste dumps, ROM pad and low-grade ore stockpile, offices, warehouse and laydown yards, process and mining workshops, road vehicle and mobile plant washdown facility, fuel storage and dispensing facility, internal haulage and access roads and relocated external public road.

The BSP is located in an established mining district in the Central Tablelands Region of New South Wales. Key nearby offsite infrastructure and facilities include:

- Access to Port Botany or Port of Newcastle via separate road routes;
- Major mining equipment service centres;
- Mining economy-based towns with broad range of mining, supply and maintenance services;
- Hospitals and other medical facilities; and
- Workplace training and education providers.

Refer to the DFS Report in Appendix 3 for full details regarding infrastructure.

Mining Method

The pit design incorporates drilling of 5m vertical height benches with 105mm production holes by tracked top hammer drill rigs, excavation by 200-ton class excavator in backhoe configuration, an 80-ton class excavator for backup production and batter management, and haulage by 100-ton trucks. Ore would be hauled directly to the Run of Mine pad for temporary stockpiling or direct tipped to the crushing circuit.

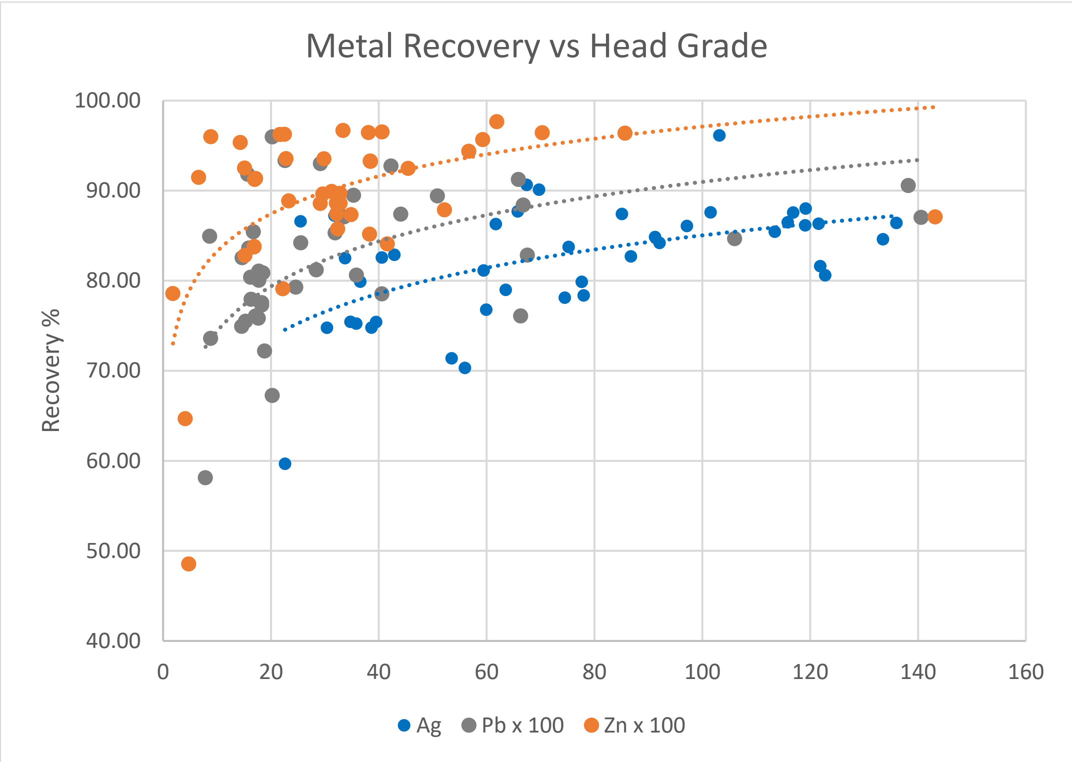
The operational design includes pre-determination of waste rock characteristics (topsoil, construction suitable, non-acid forming or potentially acid-forming) for direct haulage to specifically designed temporary or permanent waste dumps. Detailed Waste Rock classification has been undertaken.

A maintenance workshop along with ancillary facilities including fuel storage and supply, power supply, washdown, offices and material storage is required to support mining.

Processing Method

Numerous metallurgical studies conducted since the discovery of the deposit in 1989 have considered and tested various process methodologies focusing on different flotation sequences and conditioning producing different products. Under current market conditions (high demand for high silver content concentrates), single stage flotation producing a high-grade Ag concentrate with marketable Pb and Zn content is optimal. Figure 8 demonstrates the metal recovery versus head grade based on the most recent single stage flotation program using site water.

Figure 8: Recovery versus Head Grade



The process plant design, based on the DFS, is for a simplified flowsheet involving:

- Two-stage crushing;
- Two-stage SAG and ball milling with screens to remove fines prior to ball mill entry;
- rougher and scavenger flotation with concentrate regrind prior to cleaner flotation to produce a high-grade, bulk silver, lead and zinc concentrate; and
- concentrate dewatering utilising a thickener and filter to produce transportable concentrate.

An office complex, warehouse, maintenance workshops and power supply is required to support processing and administration.

Other Modifying Factors

Revenue factors, including head grade, long-term metal prices (Ag US$35/oz, Pb US$0.95/lb, Zn US$1.35/lb), exchange rates, and treatment charges, were based on detailed mine plans and corporate economic assumptions.

Silver is the principal commodity, with strong demand linked to renewable energy growth and tight supply. No offtake agreements currently exist. Marketing and smelting terms are based on information from relevant industry sources. The Company monitors market conditions and has engaged the services of Calgacus AG (Switzerland) to provide advice on concentrate marketing. It has been assumed that the concentrates will be shipped to China or Korea for smelting. Concentrates will be direct loaded from beneath the concentrate filter press into half-height sealed concentrate containers. The containers will then be road transported to Port Botany for shipment overseas. Whilst there are no formal contracts in place, indicative terms received from various parties were relatively consistent.

Economic modelling, using inputs from both internal and external specialists, indicates a positive NPV and confirms the project's economic viability. Sensitivity analysis shows the project NPV is most responsive to changes in silver price, with the AUD:USD exchange rate also a material driver. Sensitivity analyses indicate that project economics are not sensitive to market related costs. Refer to the DFS Report in Appendix 3 for the full sensitivity table.

The mine design minimises potential social impacts due to proximity to local communities.

Development consent from the NSW Independent Planning Commission is currently subject to redetermination, and a federal permit under the Environmental Protection and Biodiversity Conservation Act 1999 (Cth) and a mining Lease from the NSW state government are also required. All required permits and approvals are expected to be secured, and no material naturally occurring environmental risks have been identified to date, although ongoing environmental assessments are being finalised in connection with the approval process.

Cut-off

Ore classification is determined using an NSR cut-off that accounts for all costs, recoveries, and financial factors. The NSR cut-off for defining ore and waste in the production plan is A$30 per tonne.

<u>*Mineral Resource Estimate*</u>

The Bowdens Silver Deposit 2026 MRE update was completed by H & S Consultants ("H&S") using Multiple Indicator Kriging ("MIK") for Silver and Gold and Ordinary Kriging ("OK") for other metals. Refer to JORC Table 1 Section 3 in Appendix 2 for full details of sampling and estimation parameters. The 2026 MRE for the BSP as of July 2026 is shown in Table 5 and depicted in Figures 10 and 11.

Table 5: 2026 Mineral Resource Estimate (25 g/t AgEq cut-off)*

	Resource Grades						Contained Metal				
Resource Category	**Mass (Mt)**	**Ag (g/t)**	**Zn (%)**	**Pb (%)**	**Au (g/t)**	**AgEq (g/t)**	**Ag (Moz)**	**Zn (kt)**	**Pb (kt)**	**Au (koz)**	**AgEq (Moz)**
Measured	105	41	0.36	0.26	0.02	58	139	377	270	81	195
Indicated	38	23	0.43	0.29	0.08	44	29	163	110	103	54
M & I	**143**	**37**	**0.38**	**0.27**	**0.04**	**54**	**168**	**540**	**380**	**184**	**249**
Inferred	30	16	0.44	0.33	0.12	38	16	134	101	118	37
Total	**173**	**33**	**0.39**	**0.28**	**0.05**	**51**	**184**	**674**	**481**	**302**	**286**

**Notes*

1. *Bowdens Silver Mineral Resource Estimate reported to a 25g/t AgEq cut off extends from surface and is trimmed to above 300 metres RL, approximately 320 metres below surface, representing a potential target volume for future open-pit mining and expansion.*
2. *Bowdens' silver equivalent assumes prices of US$35.00/oz silver, US$2,976/t zinc and US$2,094/t lead with variable metallurgical recoveries based on individual grades, estimated from test work commissioned by Silver Mines Limited (see Table 15.2.2 in the DFS for further information). Ag recovery = (6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%, Pb recovery = (8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%, and Zn recovery = (10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%. Silver equivalent formula AgEq(g/t) = Ag(g/t) + Pb(%) x Pb_price/100 x Pb_recovery/Ag_price/31.103477 x Ag_price + Zn(%) x Zn_price/100 x Zn_recovery/Ag_price/31.103477 x Ag_price. Gold (Au) is reported as a separate attribute in Table 5 but is excluded from the AgEq formula for the 2026 MRE. This is because the 2026 DFS processing flowsheet is optimised for silver, lead and zinc flotation. Gold remains an attribute in the model for informational purposes and may be incorporated in future flowsheet assessments. It should be noted metal prices and recoveries used in the Ore Reserve are identical to those used for Mineral Resources.*
3. *In the Company's opinion, the silver, zinc and lead included in the metal equivalent calculations have a reasonable potential to be recovered and sold.*
4. *Stated Mineral Resources are partially inclusive of areas of the total Underground Mineral Resource Estimate at 150 g/t Silver Equivalent (AgEq) Cut-off Grade above 300mRL. See ASX announcement dated 5th September 2022.*
5. *Variability of summation may occur due to rounding.*
6. *Oxide and transitional material comprise 0.5% and 3.6% of the Resource tonnage, containing 1 Moz and 9 Moz AgEq respectively.*

Cut-off Grades and Comparison with Previous Estimates

The Resource estimation techniques and parameters remain largely unchanged from the 2024 Mineral Resource Estimate ("2024 MRE").[6] Estimates from 2024 to 2026 differ in the following respects and do not differ materially in outcome:

- the updates to data used in the estimates include minor updates to geological domaining and lagging density measurements;
- additional drilling and sampling including an additional 2,265m for 2,248 intervals of drilling assays from 16 diamond drill holes (refer to Appendix 1 for further details); and
- 2024 MRE domains were confirmed by diamond drill core for metallurgical sampling and updated as required.

[6]Silver Mines Limited (ASX:SVL) release: "Bowdens Silver Project Ore Reserves Increased to 71.7Moz Silver" dated 10 January 2025.

Updates to oxide and transitional surfaces from diamond drilling increased the proportion of transitional material but otherwise little material change between estimates occurred.

Figure 9: Location of additional drilling and metallurgical sampling programs for the 2026 MRE, 2026 DFS and Reserve pit designs

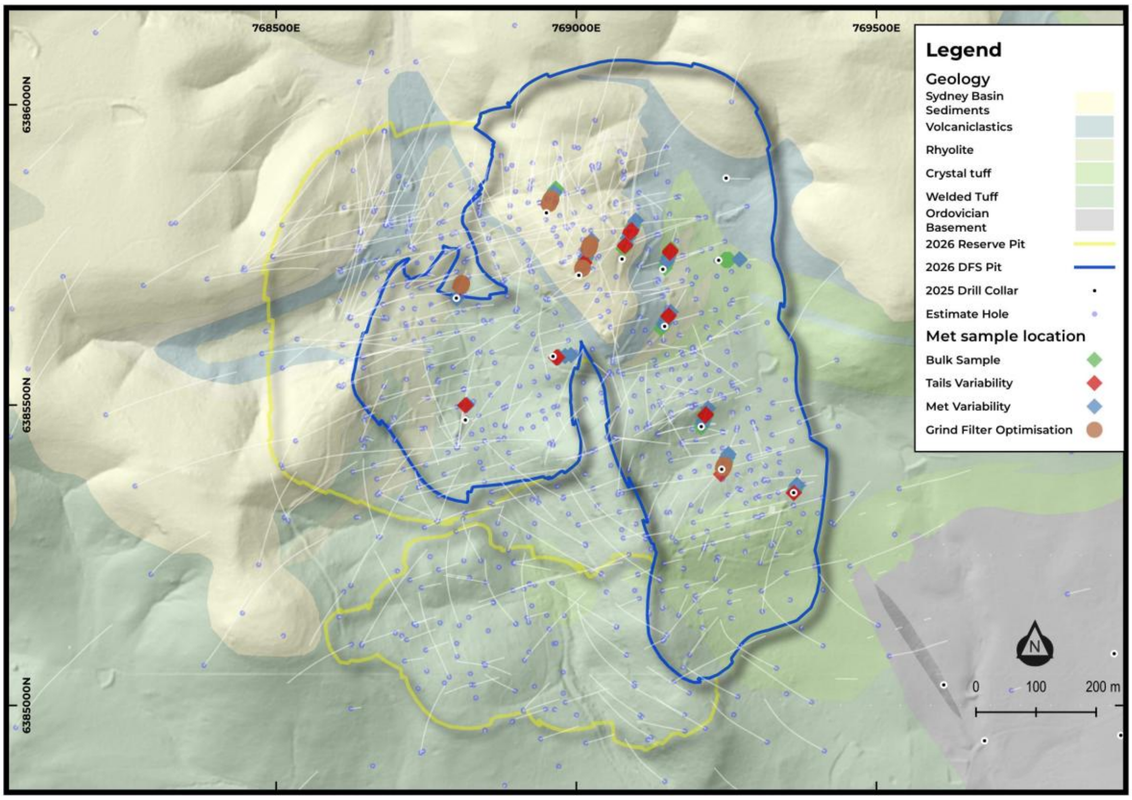


Figure 10: Cross Section of the 2026 MRE showing AgEq within 2026 Reserve Pit

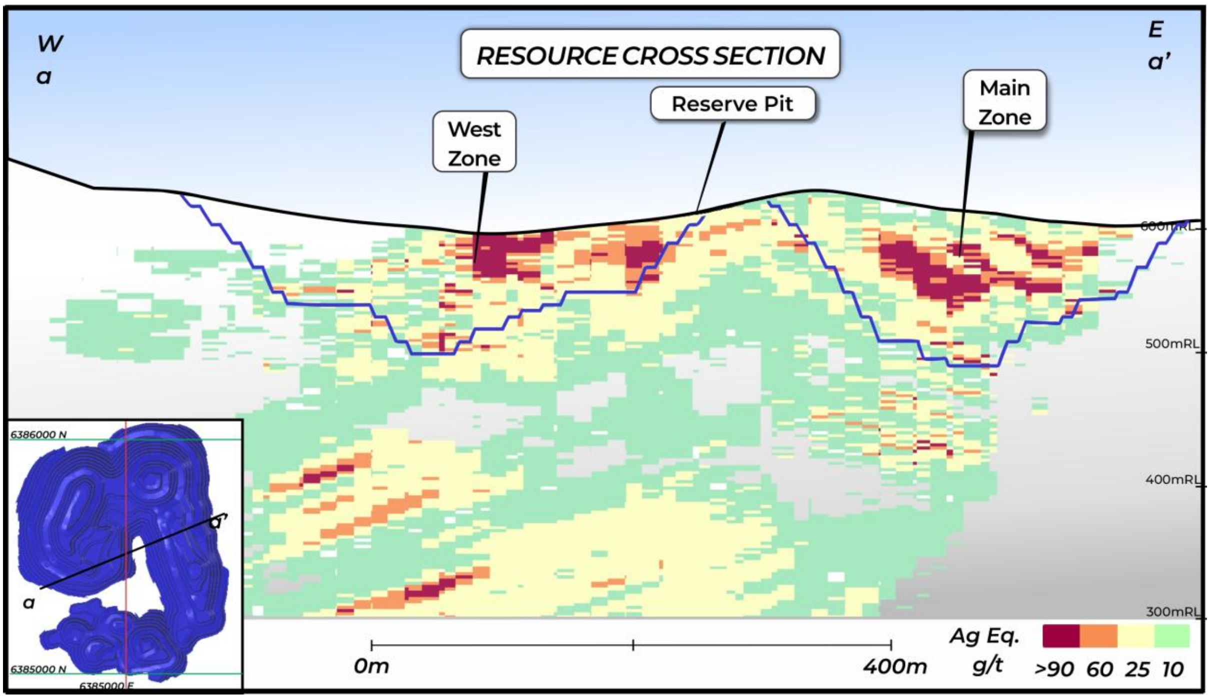


Figure 11: Long Section of the 2026 MRE showing AgEq within 2026 Reserve Pit

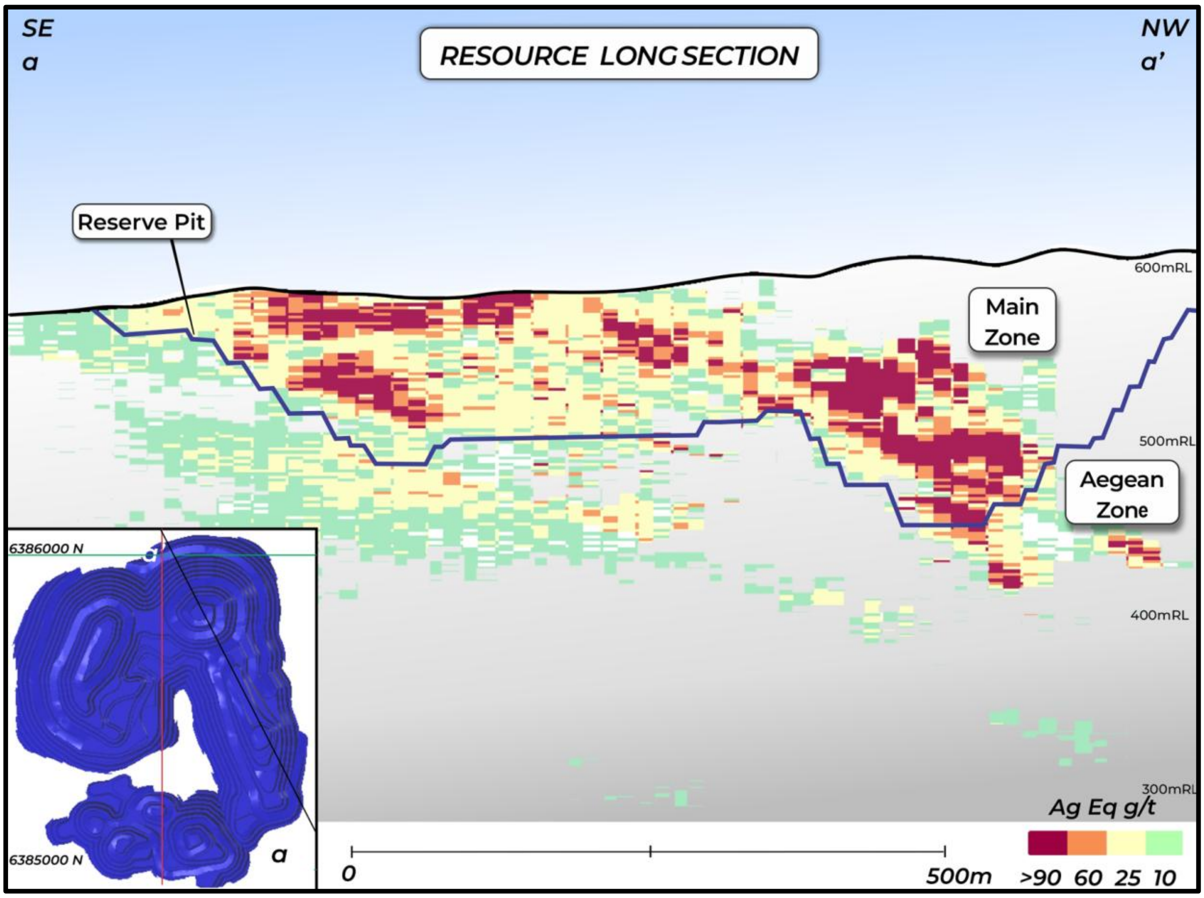


Figure 12: 2026 MRE Grade Tonnage Curve by AgEq

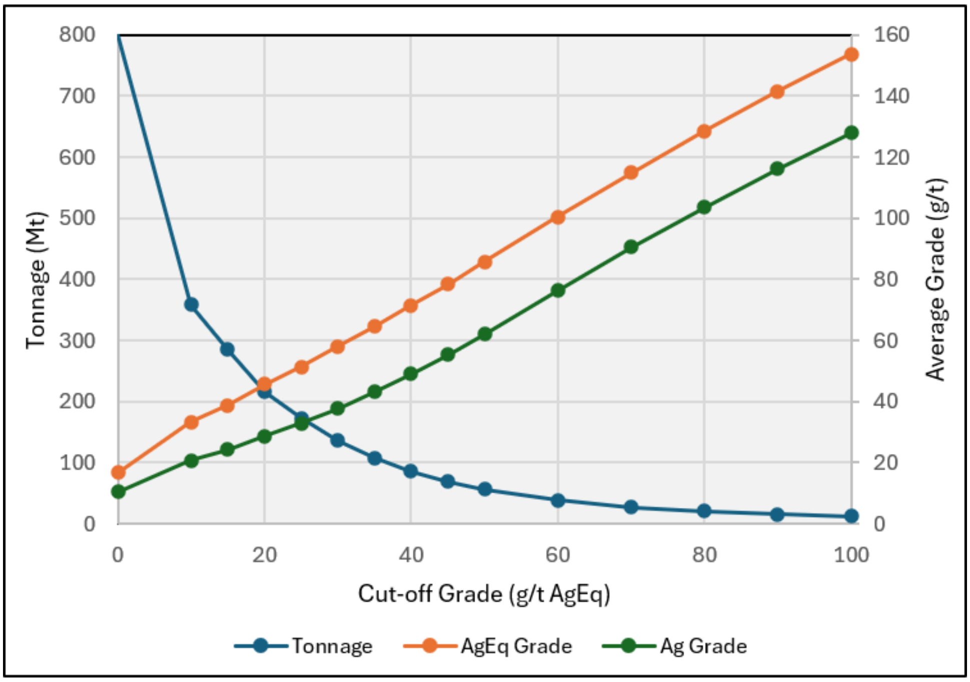


Table 6: 2026 MRE grade tonnage data by AgEq cut-off

		Resource Grades					Contained Metal				
Cut-off (AgEq)	**Tonnes (Mt)**	**AgEq g/t**	**Ag (g/t)**	**Zn (%)**	**Pb (t)**	**Au (g/t)**	**AgEq (Moz)**	**Ag (Moz)**	**Zn (kt)**	**Pb (kt)**	**Au (Koz)***
0	800	17	10	0.14	0.09	0.029	430	267	1,111	755	752
10	359	33	21	0.27	0.19	0.050	385	239	972	688	578
15	285	39	24	0.31	0.22	0.052	355	223	878	624	476
20	216	46	29	0.36	0.25	0.054	317	200	772	551	376
25	**173**	**51**	**33**	**0.39**	**0.28**	**0.054**	**286**	**184**	**674**	**481**	**302**
30	136	58	38	0.43	0.30	0.053	253	165	578	413	233
35	108	64	43	0.45	0.32	0.051	225	150	488	350	179
40	86	71	49	0.47	0.34	0.049	198	136	408	294	135
45	69	79	55	0.49	0.36	0.045	175	123	340	247	101
50	56	86	62	0.50	0.37	0.042	155	112	282	208	75
60	38	100	76	0.52	0.39	0.034	123	94	197	149	42
70	27	115	90	0.53	0.41	0.030	101	79	143	111	26
80	20	128	104	0.54	0.42	0.028	84	68	110	86	18
90	16	142	116	0.55	0.44	0.026	71	59	86	69	13
100	12	154	128	0.56	0.45	0.026	62	51	69	57	10

**Gold (Au) is reported as a separate attribute in Table 6 but is excluded from the AgEq formula for the 2026 MRE. This is because the 2026 DFS processing flowsheet is optimised for silver, lead and zinc flotation. Gold remains an attribute in the model for informational purposes and may be incorporated in future flowsheet assessments.*

In comparison to the 2024 MRE, changes to the 2026 MRE largely reflect updated assumptions to underlying pricing and recoveries for silver and base metals. The silver price used to calculate the silver equivalent cut-off is some 27% higher than the 2024 MRE. The effect of a higher denominator in the AgEq calculations results in the lower conversion of base metals into the AgEq estimate.

The new cut-off grade of 25 g/t AgEq takes into account the changes in metal prices and recoveries, and is broadly in line with the results of the DFS, allowing for some additional future upside.

The 2026 MRE reported above a 25g/t AgEq cut-off grade (2026) has comparable tonnages and grades to the 2024 MRE above a 30g/t AgEq cut-off grade (2024). The exception is gold, which shows a marked decrease in grade because this metal is no longer included in the silver equivalent formula. The current 2026 Reserve is based on a processing flow sheet that optimises payability for silver, lead and zinc.

Further Work

A portion of historical assaying did not include gold, and therefore the Company is assessing the potential for any future gold resource. Where available, historical samples will be assayed for gold.

The Company also intends drill targeting of near pit extensions.

MRE – Other Material Information Summary

A summary of other material information pursuant to ASX Listing Rules 5.8.1 is provided below for the updated 2026 MRE. The Assessment and Reporting Criteria is in accordance with the JORC Code and Guidelines.

Geology and Geological Interpretation

The BSP is situated on the north-eastern margin of the Lachlan Fold Belt. The deposit is on the southern edge of 7km wide volcanic caldera complex. The geology comprises flat-lying mid-Carboniferous Rylstone Volcanics intruding into and erupted over an Ordovician sequence of sediments called the Coomber Formation. The Rylstone Volcanics are partially overlain by a sequence of post-mineralisation marine sediments of the Sydney Basin (Shoalhaven Group). The Rylstone Volcanics range from 10 to more than 700 metres thick. Near the deposit thickness typically ranges from 50 to 250 metres. The silver mineralisation is associated with sulphides of silver, iron, lead and zinc and resides in fine fractures and veins within both Rylstone and Coomber formations. The zinc dominant mineralisation primarily occurs in the Bundarra Zone as stacked, flat-lying to moderately dipping zones of veins, breccias and fracture-fill sulphides associated with zinc, iron, lead, silver and gold within siltstones, shales and sandstones of the Coomber Formation.

The gold-dominant mineralisation is associated with an increase in zinc, lead, and sulphur particularly near the contact of the Rylstone Volcanics and Coomber Formation and near the Gully fault from which mineralisation at the contact was likely sourced. Gold occurs as irregular to partly formed grains about 5–50 µm in size. It fills fractures in the early-stage iron, lead and zinc sulphides and rims later-stage copper and lead sulphides. There are also small amounts of gold introduced late within quartz and carbonate veins.

Sampling and Sub-Sampling Techniques

Mineral Resources were estimated from reverse circulation (“RC”) and diamond core sampling by Silver Mines Limited (65%), Kingsgate Consolidated (10%), and Silver Standard, Golden Shamrock Mines and CRAE (25%). The resource database totals 549 generally vertical to inclined reverse circulation holes for a total of 67,424 metres and 321 inclined to vertical diamond core holes for a total 96,338 metres.

The majority of RC sampling was collected with either a riffle or cone splitter over 1 metre intervals. The majority of diamond core was sawn, either half or quarter cored. The minimum sample interval was 0.2 metres and the maximum sample interval was 5 metres, with over 90% of samples between 0.9 - 1.1m in length.

Drilling Techniques

The drilling used for the Mineral Resource estimation includes diamond drilling with diameter of NQ (47.6mm), HQ3 (61.1mm) and with PQ3 (83mm) for the upper sections of holes. Reverse circulation drilling used face sampling hammers of 5.5 inches diameter (139.7mm). Core orientations were completed using REFLEX ACT tools and Eastman Single Shot for pre 2007 holes.

Sample Analysis Method

For pre-Kingsgate Consolidated drilling, samples were pulverised to 85% passing 75 microns, split then analysed by acid digestion and AA or ICP determination. Since Kingsgate, samples have been analysed by a 4-acid digest with a multi-element ICP-AES determination. Gold determinations have been made via a combination of Neutron Activation Assay, photon assay and fire assay which show good replication between methods.

Estimation Methodology

Silver was initially estimated by recoverable Multiple Indicator Kriging ("MIK") into 25 x 25 x 5m panels. These estimates were then localised by discretising the metal distribution into sub-blocks with the dimensions of the selective mining unit (SMU) of 12.5 x 12.5 x 2.5m. The order of assigning the metal distribution to sub-blocks was based on an Ordinary Kriging (“OK”) estimate for silver into the sub-blocks.

Gold was estimated by MIK, using the e-type or average block grade at the scale of the panels; this coarser resolution reflects the substantial under-assaying of gold compared to silver in the Rylstone Volcanics.

All other attributes were estimated by OK, including Pb, Zn, Cu, S, As, Sb, Cd, Mn, Fe and dry bulk density. OK is considered appropriate because the coefficients of variation (CV=SD/mean) are generally low to moderate, and the grades are reasonably well structured spatially. Recoverable MIK was chosen for Ag primarily because it allows better mining selectivity than OK and is appropriate for estimation of mixed domains with higher variance.

A difference between the 2024 and 2026 models is the addition of two geometallurgical variables - Illite Crystallinity (a spectral measure of alteration) and Leeb Hardness (a measure of rock impact hardness) to assist in optimisation studies.

MIK estimates were generated using the GS3M software package, while OK estimates were generated in the Datamine software package.

Each of the major stratigraphic units (Rylstone, Coomber, Shoalhaven) were estimated separately, with each unit sub-divided into domains based on changes in mineralisation orientation.

Samples were composited to nominal 2.0m intervals within each unit for data analysis and resource estimation.

Classification Criteria

The classification scheme is based on the estimation search pass for silver. This scheme is considered to take appropriate account of all relevant factors, including the relative confidence in tonnage and grade estimates, confidence in the continuity of geology and metal values, and the quality, quantity and distribution of the data.

The classification appropriately reflects the Competent Person’s view of the deposit.

Specifically:

- Measured Resources are effectively based on a nominal drill hole spacing of 25x25m or less;
- Indicated Resources are based on a drill hole spacing of ~50x50m; and
- Inferred Resources are based on a drill hole spacing of ~100x100m.

These spacings are considered appropriate given the geological continuity observed at the Bowdens deposit.

Cut-off Grades

The cut-off grade is a silver equivalent (AgEq) value, based on grades and recoveries for silver, zinc and lead as shown below.

Metal	Unit	Price (USD)
Silver (Ag)	Ounce (oz)	$35.00
Zinc (Zn)	Tonne (t)	$2,976

Metal	Unit	Price (USD)
Lead (Pb)	Tonne (t)	$2,094

Variable metal recovery formulas were used, based on metallurgical testwork:

Ag recovery = (6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%,

Pb recovery = (8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%,

Zn recovery = (10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%.

Silver equivalent formula:

AgEq(g/t) = Ag(g/t)

+ Pb(%) x Pb_price/100 x Pb_recovery/Ag_price/31.103477 x Ag_price

+ Zn(%) x Zn_price/100 x Zn_recovery/ Ag_price/31.103477 x Ag_price

Metal prices and recoveries are consistent with those used for the Ore Reserves.

The adopted cut-off grade of 25 g/t AgEq is considered likely to be economic for the mining method and scale of operation envisioned for Bowdens Silver.

Mining and Metallurgical Methods, Parameters and other modifying factors considered to date

The Company has finalised a DFS as detailed in Appendix 3 which considers all modifying factors sufficiently for Resource classification.

The Bowdens Silver Mineral Resource is reported as a potential future open-pit mining scenario. The MRE has been reported extending from surface to 300mRL which is approximately 320 metres below surface, representing a potential volume for future open-pit optimisation models and expansion.

The recoverable MIK method implicitly incorporates internal mining dilution at the scale of the assumed SMU (selective mining unit). No specific assumptions were made about external mining dilution in the MRE.

Metallurgical test work to confirm the results of previous work has been conducted and optimisation studies on mining and processing completed. Other likely modifying factors pertaining to this Resource have been extensively assessed, including updating environmental, social community and other governance requirements in line with the project approval being redetermined by the Independent Planning Commission.

About the Bowdens Silver Project

The Bowdens Silver Project is in central New South Wales, approximately 26 kilometres east of Mudgee (Figure 13). The consolidated project area comprises 2,115 km^2 (521,000 acres) of titles covering approximately 80 kilometres of strike of the highly mineralised Rylstone Volcanics. Multiple target styles and mineral occurrences have potential throughout the district including analogues to Bowdens Silver, high-grade silver-lead-zinc epithermal and volcanogenic massive sulphide (VMS) systems and copper-gold targets.

Bowdens Silver is the largest undeveloped silver deposit in Australia with substantial resources and a considerable body of high-quality technical work completed. The project boasts outstanding logistics for mine development.

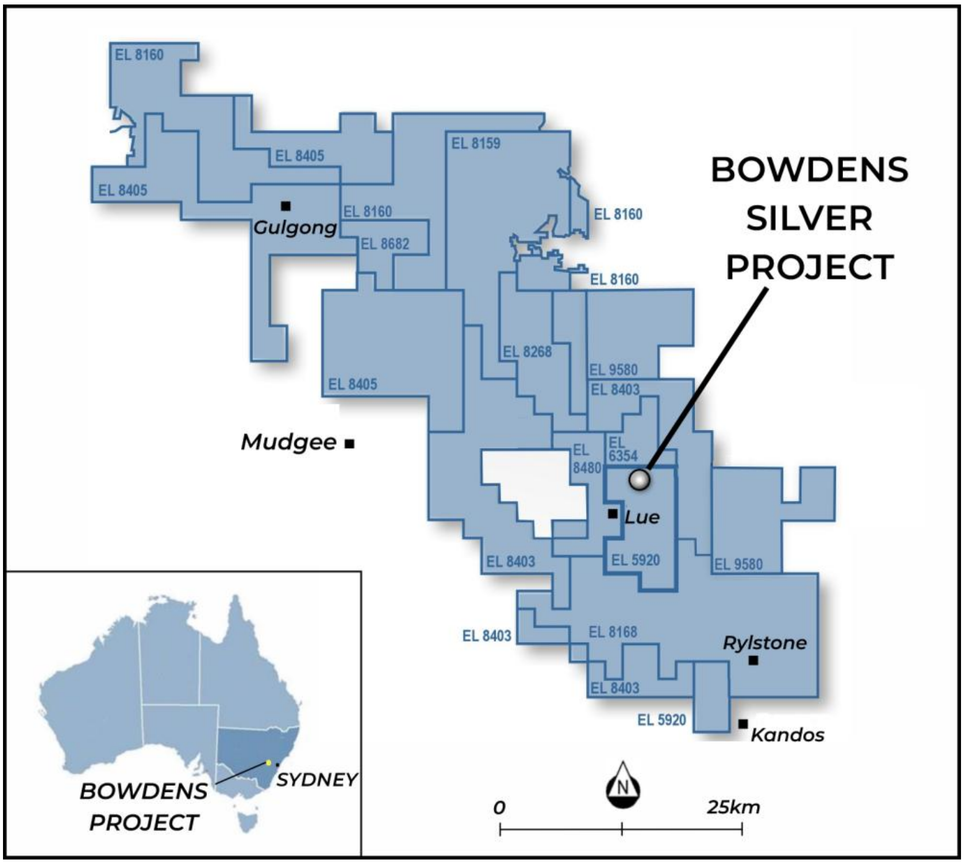


Figure 13: Silver Mines Limited tenement holdings in the Mudgee district.

This document has been authorised for release to the ASX by the Company's Board of Directors.

Further information:

Jo Battershill
Managing Director
Silver Mines Limited
+61 2 8316 3997

Christina Granger
Account Director
M+C Partners
+61 438 117 286

Competent Persons Statement

The information in this report that relates to Mineral Resources is based on and fairly represents information and supporting documentation compiled by Mr Arnold van der Heyden who is a Director of H & S Consultants Pty Ltd. Mr van der Heyden is a member and Chartered Professional (Geology) of the Australian Institute of Mining and Metallurgy and has sufficient experience that is 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 van der Heyden is not an employee of the Company and has no economic, financial or pecuniary interest in the Company other than in his capacity as a paid consultant engaged to prepare the Mineral Resource Estimate. There are no other relationships that could, in the reasonable opinion of the Competent Person, be regarded as giving rise to a conflict of interest. Mr van der Heyden consents to the inclusion in this report of the matters based on the information in the form and context in which it appears.

The information in this report that relates to Exploration Results that underpin the Mineral Resources and Ore Reserves for the Bowdens Silver Project is based on and fairly represents information and supporting information compiled by the Geology Department of Silver Mines Limited, which is managed by Dr Michael Fletcher, General Manager Group Geology of Silver Mines Limited. Database compilation and QAQC is reviewed by David Biggs who is an employee of the Company. Mr Biggs is a member of the Australasian Institute of Mining and Metallurgy (AusIMM) and has sufficient experience that is 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 Biggs is a full time employee of the Company and holds shares and options in the Company. There are no other relationships that could, in the reasonable opinion of the Competent Person, be regarded as giving rise to a conflict of interest. Mr Biggs consents to the inclusion in this report of the matters based on the information in the form and context in which it appears.

The information in this report that relates to Ore Reserves within the Bowdens Silver Project is based on and fairly represents information and supporting information compiled or reviewed by Mr Andrew Hutson, a Competent Person who is a Fellow of the Australian Institute of Mining and Metallurgy (AusIMM) and a full-time employee of Resolve Mining Services. Mr Hutson has sufficient experience which is relevant to the style of mineralisation and type of deposits under consideration and to the activity which he has undertaken to qualify as a Competent Person as defined in the JORC Code. Mr Hutson is not an employee of the Company and has no economic, financial or pecuniary interest in the Company other than in his capacity as a paid consultant engaged to prepare the Ore Reserve. There are no other relationships that could, in the reasonable opinion of the Competent Person, be regarded as giving rise to a conflict of interest. Mr Hutson consents to the inclusion in this report of the matters based on his information in the form and context in which it appears.

The Ore Reserves and Mineral Resources underpinning the production targets were prepared by a Competent Person in accordance with the JORC Code.

Previously reported Exploration Results, Mineral Resources and Ore Reserves that are referred to in this announcement were first announced by the Company on the dates specified in the body of this announcement. The Company confirms that it is not aware of any new information or data that materially affects the information included in the relevant market announcements and, in the case of estimates of Mineral Resources or Ore Reserves, that all material assumptions and technical parameters underpinning the estimates in the relevant market announcement continue to apply and have not materially changed.

Disclaimer

This announcement has been prepared by Silver Mines Limited based on information from its own and third party sources and is not a disclosure document. No party other than the Company has authorised or caused the issue, lodgement, submission, despatch or provision of this announcement, or takes any responsibility for, or makes or purports to make any statements, representations or undertakings in this announcement. Except for any liability that cannot be excluded by law, the Company and its related bodies corporate, directors, employees, servants, advisers and agents disclaim and accept no responsibility or liability for any expenses, losses, damages or costs incurred by you relating in any way to this announcement including, without limitation, the information contained in or provided in connection with it, any errors or omissions from it however caused, lack of accuracy, completeness, currency or reliability or you or any other person placing any reliance on this announcement, its accuracy, completeness, currency or reliability. This announcement is not a prospectus, disclosure document or other offering document under Australian law or under any other law. It is provided for information purposes and is not an invitation nor offer of shares or recommendation for subscription, purchase or sale in any jurisdiction. This announcement does not purport to contain all the information that a prospective investor may require in connection with any potential investment in the Company. Each recipient must make its own independent assessment of the Company before acquiring any shares in the Company.

Forward-Looking Statements

This announcement contains certain forward-looking statements, guidance, forecasts, estimates, prospects, projections or statements in relation to future matters that may involve risks or uncertainties and may involve significant items of subjective judgement and assumptions of future events that may or may not eventuate (“Forward-Looking Statements”). Forward-Looking Statements can generally be identified by the use of forward-looking words such as "anticipate", "estimates", "will", "should", "could", "may", "expects", "plans", "forecast", "target" or similar expressions and may include, without limitation, statements regarding plans, strategies and objectives of management, anticipated production and expected costs. Indications of, and guidance on future earnings, cash flows, costs, financial position and performance are also Forward Looking Statements.

Although the Forward Looking Statements contained in this announcement reflect management’s current beliefs to be reasonable assumptions, persons reading this announcement are cautioned that such statements are only predictions, and that actual future results or performance may be materially different. A number of factors could cause events and achievements to differ materially from the results expressed or implied in the Forward-

Looking Statements. These factors should be considered carefully, and prospective investors should not place undue reliance on the Forward-Looking Statements. Forward-Looking Statements necessarily involve significant known and unknown risks, assumptions and uncertainties that may cause the Company's actual results, events, prospects and opportunities to differ materially from those expressed or implied by such forward-looking statements. Although the Company has attempted to identify important risks and factors that could cause actual actions, events or results to differ materially from those described in Forward-Looking Statements, there may be other factors and risks that cause actions, events or results not to be anticipated, estimated or intended, including those risk factors discussed in the Company's public filings. There can be no assurance that the forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, prospective investors should not place undue reliance on forward-looking statements. Any Forward-Looking Statements are made as of the date of this announcement, and the Company assumes no obligation to update or revise them to reflect new events or circumstances, unless otherwise required by law.

This announcement may contain certain Forward-Looking Statements and projections regarding:

- Estimated Mineral Resources and Ore Reserves;
- Planned production and operating costs profiles;
- Planned capital requirements; and
- Planned strategies and corporate objectives.

The DFS referred to in this announcement is based on technical and economic assessments to support the estimation of Ore Reserves. Silver Mines believes it has reasonable grounds to support the results of the DFS, however there is no assurance that the intended development referred to will proceed as described. The production targets and forward-looking statements referred to are based on information available to the Company at the time of release and should not be solely relied upon by investors when making investment decisions. Material assumptions and other important information are contained in this release. Silver Mines cautions that mining and exploration are high risk, and subject to change based on new information or interpretation, commodity prices or foreign exchange rates. Actual results may differ materially from the results or production targets contained in this release. Further evaluation is required prior to a decision to conduct mining being made.

Non-IFRS Financial Measures

The Company uses certain non-IFRS financial measures and project evaluation metrics to assess the projected economic performance and financial outcomes of the BSP. These measures including, AISC, C1 costs, operating margin, operating cash flow, net present value (NPV), internal rate of return (IRR), Profitability Index and NPV-to-Capex (collectively referred to as "Non-IFRS Financial Measures") are not recognised under International Financial Reporting Standards ("IFRS").

The Company considers the Non-IFRS Financial Measures provide useful information about the estimated financial forecasts derived from the DFS, however, they should not be considered in isolation or as a substitute for measures of performance or cash flow prepared in accordance with IFRS.

Since the financial forecast and economic discussion in the announcement are not based on IFRS, they do not have standardised definitions and the way these measures have been derived may not be comparable to similarly titled measures used by other companies. Investors should therefore not place undue reliance on these Non-IFRS Financial Measures.

Risks

The Company considers that the following list, which is not exhaustive, represents some of the key risk factors relevant to the development of the Bowdens Project proposed by the DFS.

Further risk factors apply to the Company and its business (such as those previously announced to the ASX by the Company).

Commodity Price Volatility and Exchange Rate Risk

Of all variables, the financial outcome is most impacted by changes to revenue factors. Negative changes to the recovered silver or Australian dollar silver price, either by US dollar silver price variation or AUD:USD exchange rate fluctuations, would have a direct effect on revenue and derived cash flow.

Resource and Reserve Estimates

Resource and Reserve estimates are expressions of judgement based on knowledge, experience and industry practice, including compliance with the JORC Code. By their very nature, these estimates are imprecise and depend on interpretations that may prove to be inaccurate which means that the reconciliation and performance of the Reserve model is a risk that is inherent until production confirms the modelling. Major variances to contained metal in the Ore Reserve will have a negative impact on the revenue generated by the Project.

Approval Risks

The Company will be reliant on environmental and other regulatory approvals to enable it to proceed with the development of the Bowdens project. There is no guarantee that the required approvals will be granted, and delays in project permitting may delay the project from commencing production in the proposed timeframe.

Engagement with regulators to raise awareness of the project and the planned scope is ongoing.

Environmental, Health and Safety Risks

The Company expends significant financial and managerial resources to keep our stakeholders safe and to comply with a complex set of environmental, health and safety laws, regulations, guidelines and permitting requirements. The historical trend that the Company observes is toward stricter laws and the Company expects this trend to continue. The possibility of more stringent laws or more rigorous enforcement of existing laws exists in the areas of worker health and safety, the disposition of wastes, the decommissioning and reclamation of mining sites, restriction of areas where exploration, development and mining activities may take place, consumption and treatment of water, biodiversity requirements and other environmental matters, each of which could have a material adverse effect on the Company's exploration activities, operations and the cost or the viability of a particular project.

Personnel and Operating Costs

The skilled labour pool (management, technical and blue collar) in Australia and New South Wales is relatively inelastic. The cost of energy, labour, materials, services and other operating inputs are at historically high levels on a unit basis and inflationary pressures remain and may impact estimated costs in the DFS.

Supply Chain

The Project is dependent on an uninterrupted flow of materials, supplies and services. Any interruptions to the procurement of equipment, or the flow of materials, supplies and services could have an adverse impact on its future cash flows.

The Company will rely significantly on strategic relationships with material, equipment and service providers. The Company will also rely on third parties to provide essential contracting services. There can be no assurance that its existing relationships will continue to be maintained or that new ones will be successfully formed. The Project could be adversely affected by changes to such relationships or difficulties in forming new ones.

Development and Construction Risks

Construction costs and timelines can be impacted by a wide variety of factors, many of which are beyond the Company's control. These include, but are not limited to, weather and ground conditions, performance of the mining fleet and availability of appropriate materials required for construction, availability and performance of contractors and suppliers, delivery and installation of equipment, design changes, accuracy of estimates, global capital cost inflation, local in-country inflation and availability of accommodations for the workforce.

Development schedules are also dependent on obtaining the governmental approvals necessary for the operation of a project. The timeline to obtain these government approvals is often beyond the control of the Company. A delay in start-up or commercial production would increase capital costs and delay receipt of revenues. The ultimate and continued success of the project is dependent on a number of factors, including the construction of efficient development and production infrastructure within capital expenditure budgets and on schedule.

Operational Risks

The Company's operations may be delayed or prevented as a result of various factors, including weather conditions, mechanical difficulties or a shortage of technical expertise or equipment. There may be difficulties with obtaining government and/or third-party approvals; operational difficulties encountered with construction, extraction and production activities; unexpected shortages or increase in the price of consumables, plant and equipment; or cost overruns.

The Company's operations may be curtailed or disrupted by risks beyond its control, such as environmental hazards, industrial accidents and disputes, technical failures, unusual or unexpected geological conditions, adverse weather conditions, fires, explosions and other accidents.

The occurrence of any of these circumstances could result in the Company not realising its operational or development plans or in such plans costing more than expected or taking longer to realise than expected.

Any of these outcomes could have an adverse effect on the Company's financial and operational performance.

Concentrate Specification and Marketing Risk

The financial model assumes that the bulk silver-lead-zinc sulphide concentrate produced by the Project will meet buyer specifications and will be marketable on acceptable terms. There is a risk that the concentrate may not meet particular buyer specifications due to the presence of penalty elements (including arsenic, cadmium or fluorine) or grade variability, resulting in rejection, mandatory blending, reduced payabilities or sale on less favourable terms than assumed. No binding offtake agreement is in place at the date of this announcement and there is no guarantee that offtake will be secured on the assumed economic terms.

Tailings Storage and Infrastructure Risk

The Project relies on filtered tailings landform infrastructure for the management of process residues over its projected mine life. The Stage 2 filtered tailings landform has been assessed at pre-feasibility study level only. There is a risk that the design, construction or operation of tailings storage facilities may not perform as intended, or that regulatory requirements relating to tailings management become more stringent, resulting in additional capital costs, production disruption or, in an extreme case, closure of operations. Tailings storage failure, while considered low probability, could have material environmental and financial consequences.

Native Title, Land Access and Heritage Risk

The development and operation of the Project will require continued access to land that may be subject to native title claims or cultural heritage obligations. While the Company is not aware of any unresolved native title claims that would prevent development at the time of this announcement, future claims, heritage site identification or access negotiations could restrict or delay activities, increase compliance costs or require modifications to project design.

Key Personnel and Force Majeure Risk

The Project's success depends in part on retaining key technical and management personnel. The loss of one or more key personnel could adversely affect the Company's ability to progress development activities. The Project may also be adversely affected by events beyond the Company's reasonable control, including natural disasters, pandemic, civil disorder, industrial action, fire, flood or other force majeure events, which could disrupt the development timetable, increase costs or affect the Project's viability.

Capital Cost Contingency Risk

There is a risk that actual capital costs may exceed the DFS estimate, that the contingency allowance may prove insufficient to absorb cost overruns, or that capital cost inflation continues at elevated levels during the project construction period. Any material increase in capital costs above the DFS estimate would reduce project returns and may affect the ability to fund the Project on acceptable terms.

SILVER MINES LIMITED

APPENDIX 1: SIGNIFICANT INTERCEPT TABLES

Table 7: Additional Drill Collar Locations since 2024 MRE update used in estimation

Hole ID	GDA94 East	GDA94 North	RL (m)	Dip	Azimuth (grid)	Depth (m)	Drill Type	Program
BD25001	768809	6385498	601.8	-75	84	51.3	DDH	GeoMet
BD25002	769239	6385742	611.7	-60	90	126.7	DDH	GeoTech
BD25003	769141	6385710	624.9	-70	20	213.7	DDH	GeoMet
BD25004	769139	6385623	627.3	-70	30	222.8	DDH	GeoMet
BD25005	768945	6385812	657.5	-70	20	204.7	DDH	GeoMet
BD25006	769005	6385712	657.8	-70	20	207.7	DDH	GeoMet
BD25007	769072	6385743	646.5	-70	20	225.8	DDH	GeoMet
BD25008	769362	6385354	605.3	-70	20	113.8	DDH	GeoMet
BD25009	769208	6385462	617.7	-70	20	210	DDH	GeoMet
BD25010	769241	6385384	618.7	-70	20	210.7	DDH	GeoMet
BD25019	768800	6385677	606.5	-75	20	216.8	DDH	GeoMet
BD25020	768960	6385578	620.2	-75	84	141.8	DDH	GeoMet
BD25024	769252	6385877	615.5	-60	90	80	DDH	GeoTech

Table 8: Table of significant intercepts returned since 2024 MRE

Hole ID	Interval	From	Ag (g/t)	Pb (%)	Zn (%)	AgEq (g/t)	Au (g/t)	In Pit or Out
BD25001	24	21	53.6	0.22	0.44	67.4	-	In Pit[1]
incl.	1	29	246	0.67	2.05	309.7	-	In Pit[2]
incl.	2	39	114.5	0.75	0.89	149.9	-	In Pit[2]
BD25002	14.3	16.4	59.3	0.04	0.05	60.9	-	In Pit[1]
incl.	1.4	17.6	149.3	0.11	0.03	151.4	-	In Pit[2]
incl.	1.7	29	212.1	0.03	0.04	213.3	-	In Pit[2]
	1	65	91.2	0.06	0.08	93.6	-	In Pit[2]
	1	90	35.2	0.30	1.36	74.7	-	Out of Pit[3]
BD25003	44	109	39	0.10	0.18	44.2	-	In Pit[1]
incl.	2	113	112	0.11	0.08	115.1	-	In Pit[2]
incl.	1	122	96.7	0.04	0.05	98.3	-	In Pit[2]
incl.	1	137	580	1.13	3.88	699.2	-	In Pit[2]
incl.	1	146	166	0.12	0.08	169.3	-	In Pit[2]
incl.	1	151	117	0.42	0.02	124.1	-	In Pit[2]
BD25004	84	14	69.1	0.16	0.28	77.7	-	In Pit[1]
incl.	18	37	109	0.23	0.40	121.9	-	In Pit[2]
incl.	12	68	207.5	0.47	0.40	224.5	-	In Pit[2]
incl.	1	86	123	0.24	0.36	135.0	-	In Pit[2]
incl.	1	94	89.8	0.10	0.37	99.8	-	In Pit[3]
	11	112	19.3	0.09	0.30	27.4	-	Out of Pit[1]
incl.	1	113	87.5	0.18	0.49	102.0	-	Out of Pit[3]
incl.	1	122	30.5	0.27	1.13	63.6	-	Out of Pit[3]
BD25005	57	84	189.3	0.50	0.27	203.5	-	In Pit[1]
incl.	1	91	93.9	0.13	0.18	99.6	-	In Pit[2]

Hole ID	Interval	From	Ag (g/t)	Pb (%)	Zn (%)	AgEq (g/t)	Au (g/t)	In Pit or Out
incl.	34	103	298.8	0.78	0.33	319.6	-	In Pit[2]
BD25006	34	47	339	0.70	0.82	371.6	-	In Pit[1]
incl.	2	51	729.5	2.14	1.99	818.4	-	In Pit[2]
incl.	17	63	570.3	1.05	1.06	615.7	-	In Pit[2]
	18	95	27.2	2.06	0.53	76.5	-	In Pit[1]
	27	143	30.7	0.35	0.07	37.6	-	In Pit[1]
incl.	1	157	256	1.84	0.33	296.1	-	In Pit[2]
incl.	1	166	116	0.36	0.10	123.6	-	In Pit[2]
	17	182	70.3	0.15	0.17	76.1	-	In Pit - Out at 190m[1]
incl.	3	196	317.3	0.48	0.64	341.0	-	Out of Pit[2]
BD25007	9	53	42.7	0.16	0.20	49.4	-	In Pit[1]
incl.	1	61	178	0.64	0.19	192.7	-	In Pit[2]
	116	75	244.6	0.48	0.29	258.9	-	In Pit[1]
incl.	28	89	184.8	0.46	0.24	197.5	-	In Pit[2]
incl.	1	122	157	0.37	0.22	167.7	-	In Pit[2]
incl.	1	127	93.7	0.26	0.07	99.0	-	In Pit[2]
incl.	1	134	180	0.54	0.27	194.9	-	In Pit[2]
incl.	11	140	397.7	0.44	0.56	418.5	-	In Pit[2]
incl.	35	156	412.3	0.79	0.35	433.8	-	In Pit[2]
	5	211	309.5	0.18	0.01	312.3	-	In Pit[1]
incl.	1	214	1375	0.38	0.01	1381.2	-	In Pit[2]
BD25008	58	0	120.6	0.27	0.39	133.9	-	In Pit - Out at 30m[1]
incl.	39	1	162.8	0.37	0.53	181.5	-	In Pit - Out at 30m[2]
BD25009	71.9	1.1	81	0.17	0.27	89.6	-	In Pit[1]
incl.	15	10	142.3	0.34	0.60	162.4	-	In Pit[2]
incl.	13	30	110.3	0.21	0.33	121.0	-	In Pit[2]
incl.	3	53	123.9	0.18	0.23	131.6	-	In Pit[2]
incl.	2	63	376.5	0.57	0.97	410.7	-	In Pit[2]
incl.	1	72	245	0.19	0.29	254.4	-	In Pit[2]
	40	84	34.2	0.28	0.22	43.3	-	In Pit[1]
incl.	1	90	187	0.28	0.14	194.2	-	In Pit[2]
incl.	5	102	101.7	0.65	0.58	126.7	-	In Pit[2]
	26	161	20.2	0.04	0.13	23.4	-	Out of Pit[1]
incl.	1	161	92.1	0.09	0.06	94.5	-	Out of Pit[2]
BD25010	48	21	52.3	0.35	0.79	77.9	-	In Pit[1]
incl.	1	36	345	1.40	2.60	436.4	-	In Pit[2]
incl.	1	45	281	1.92	5.05	444.5	-	In Pit[2]
incl.	7	62	144.2	0.39	0.95	174.7	-	In Pit[2]
	5.9	87	37.9	0.60	1.63	89.6	-	In Pit[1]
incl.	1	87	99	0.71	1.27	143.5	-	In Pit[2]
	78	106	38.8	0.11	0.29	46.9	-	In Pit - Out at 115m[1]
incl.	2	130	139.9	0.43	0.94	170.9	-	Out of Pit[2]
incl.	1	138	155	0.44	1.23	193.6	-	Out of Pit[2]
incl.	1	177	165	0.04	0.06	166.7	-	Out of Pit[2]

Hole ID	Interval	From	Ag (g/t)	Pb (%)	Zn (%)	AgEq (g/t)	Au (g/t)	In Pit or Out
incl.	1	183	127	0.31	0.30	138.6	-	Out of Pit[2]
	1	195	436	0.25	0.69	457.1	0.02	Out of Pit[2]
BD25019	5.5	2.5	35.1	0.14	0.07	38.5	-	In Pit[1]
	67.7	36.3	49.9	0.84	0.56	77.9	-	In Pit[1]
incl.	1.5	52.5	383.7	4.95	4.01	574.1	0.14	In Pit[2]
incl.	2.4	60	539.8	9.55	3.01	785.8	0.10	In Pit[2]
incl.	1	75	116	1.05	0.27	140.3	0.06	In Pit[2]
incl.	1	83	135	0.88	0.20	154.3	0.01	In Pit[2]
	6	120	34.8	1.09	0.38	62.6	-	Out of Pit[1]
incl.	1	120	129	2.62	1.32	209.2	0.05	Out of Pit[2]
incl.	2	125	38.3	2.00	0.53	86.6	0.02	Out of Pit[3]
	1	164	61.4	0.73	0.30	80.5	0.02	Out of Pit[3]
	10.85	189.15	46.5	0.24	0.29	56.7	-	Out of Pit[1]
BD25020	47	0	50.5	0.17	0.29	59.6	-	In Pit[1]
incl.	1	12	674	1.21	0.96	719.8	-	In Pit[2]
incl.	2	17	130	0.45	1.03	163.6	-	In Pit[2]
incl.	1.4	32	215.6	0.58	1.56	265.1	0.01	In Pit[2]
	12	78	20.1	0.19	0.07	24.3	-	Out of Pit[1]
incl.	1	89	137	0.22	0.27	146.4	0.15	Out of Pit[2]
	7	128	70.1	0.02	0.08	72.0	-	Out of Pit[1]
incl.	2	128	211.5	0.06	0.23	217.4	0.38	Out of Pit[2]

1. Intercepts calculated using a 30g/t Ag cut-off, 10 metre internal dilution factor, and 5 metre minimum intercept length.
2. Intercepts calculated using a 90g/t Ag cut-off, 3 metre internal dilution factor, and 1 metre minimum intercept length.
3. Intercepts calculated using a 90g/t Ag Eq cut-off, 3 metre internal dilution factor, and 1 metre minimum intercept length.

Bowdens' silver equivalent assumes prices of US$35.00/oz silver, US$2,976/t zinc and US$2,094/t lead with variable metallurgical recoveries based on individual grades, estimated from test work commissioned by Silver Mines Limited (see Table 15.2.2 in the DFS for further information). Ag recovery = (6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%, Pb recovery = (8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%, and Zn recovery = (10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%. Silver equivalent formula AgEq(g/t) = Ag(g/t) + Pb(%) x Pb_price/100 x Pb_recovery/Ag_price/31.103477 x Ag_price + Zn(%) x Zn_price/100 x Zn_recovery/Ag_price/31.103477 x Ag_price. In the Company's opinion, the silver, zinc and lead included in the metal equivalent calculations have a reasonable potential to be recovered and sold. Gold (Au) is reported as a separate attribute in Table 8 but is excluded from the AgEq formula for the 2026 MRE. This is because the 2026 DFS processing flowsheet is optimised for silver, lead and zinc flotation. Gold remains an attribute in the model for informational purposes and may be incorporated in future flowsheet assessments.

Table 9: Metallurgical Sample Composition and unweighted average (location shown in Figure 9)

Metallurgical Composite ID & Holes	Depth from	Depth to	Sampled metres	Mean RL m	Mean GDA E	Mean GDA N
BS25_MetVar_01			5.5	601	768801	6385678
BD25019	2.5	8	5.5			
BS25_MetVar_02			6	578	768971	6385580
BD25020	40	47	6			
BS25_MetVar_03			3	554	769272	6385742
BD25002	65	68	3			
BS25_MetVar_04			3	521	769157	6385655
BD25004	110	114	3			
BS25_MetVar_05			3	520	769153	6385745
BD25003	109	115	3			

Metallurgical Composite ID & Holes	Depth from	Depth to	Sampled metres	Mean RL m	Mean GDA E	Mean GDA N
BS25_MetVar_06			6	471	769025	6385774
BD25006	195	201	6			
BS25_MetVar_07			5	535	768981	6385582
BD25020	85	90	5			
BS25_MetVar_08			3	520	769219	6385494
BD25009	102	105	3			
BS25_MetVar_09			6	566	769367	6385367
BD25008	39	45	6			
BS25_MetVar_10			3	495	768991	6385583
BD25020	128	131	3			
BS25_MetVar_11			6.9	535	769251	6385413
BD25010	85	92.9	6.9			
BS25_MetVar_12			6	445	769099	6385807
BD25007	210	216	6			
BS25_MetVar_13			4	490	769092	6385793
BD25007	163	167	4			
BS25_MetVar_14			6	516	769089	6385785
BD25007	134	141	6			
BS25_MetVar_15			5	531	768806	6385695
BD25019	75	80	5			
BS25_MetVar_16			6	555	769016	6385746
BD25006	106	112	6			
BS25_MetVar_17			5	536	769086	6385779
BD25007	114	119	5			
BS25_MetVar_18			5.1	539	769151	6385739
BD25003	88.9	94	5.1			
BS25_MetVar_19			5	571	769082	6385768
BD25007	78	83	5			
BS25_MetVar_20			5	567	768817	6385500
BD25001	33	38	5			
BS25_MetVar_21			6	520	769020	6385758
BD25006	143	149	6			
BS25_MetVar_22			4	561	769150	6385643
BD25004	68	72	4			
BS25_MetVar_23			4	587	769146	6385635
BD25004	41	45	4			
BS25_MetVar_24			4	529	768963	6385854
BD25005	134	138	4			
BS25_MetVar_25			2	581	769212	6385475
BD25009	38	40	2			
BS25_MetVar_26			6	519	769253	6385418
BD25010	103	109	6			

Metallurgical Composite ID & Holes	Depth from	Depth to	Sampled metres	Mean RL m	Mean GDA E	Mean GDA N
BS25_MetVar_27			1	603	769362	6385354
BD25008	2	3	1			
BS25_MetVar_28			2	598	769363	6385356
BD25008	7	9	2			
BS25_MetVar_29			2	599	769210	6385469
BD25009	19	21	2			
BS25_TailVar_01			12	503	769091	6385789
BD25007	142	159	12			
BS25_TailVar_02			3	577	768815	6385500
BD25001	21	29	3			
BS25_TailVar_03			8	568	769090	6385764
BD25003	142	143	1			
BD25007	68	75	7			
BS25_TailVar_04			8	491	769156	6385755
BD25003	136	150	8			
BS25_TailVar_05			11	554	769215	6385483
BD25009	62	73	11			
BS25_TailVar_06			2	611	769301	6385369
BD25008	0	1	1			
BD25010	1	2	1			
BS25_TailVar_07			11	591	768967	6385579
BD25020	25	36	11			
BS25_TailVar_08			9	521	768807	6385697
BD25019	83	94	9			
BS25_TailVar_09			7	543	769153	6385648
BD25004	86	93	7			
BS25_TailVar_10			6	575	769014	6385740
BD25006	75	142	6			
Gr25_MClin_Kspa			12	578	769246	6385398
BD25010	34	51	12			
Gr25_Siderite			22	505	768808	6385701
BD25019	94	116	22			
Gr25_Siliceous			24.6	503	769022	6385763
BD25006	150	179	24.6			
Gr25_WhiteMica			27	578	768964	6385823
BD25005	72	104	23			
BD25006	46	53	4			
BS25_BULK_HRD_680			80	548	769142	6385611
BD25005	124	132	8			
BD25006	80	101	8			
BD25007	108	190	21			
BD25009	31	85	30			

Metallurgical Composite ID & Holes	Depth from	Depth to	Sampled metres	Mean RL m	Mean GDA E	Mean GDA N
BD25010	26	47	13			
BS25_BULK_MED_500_680			76.9	562	769196	6385549
BD25003	118	123	5			
BD25004	53	80	21			
BD25005	95	144	11			
BD25008	18	37	16			
BD25009	1.1	5	3.9			
BD25010	50	70	20			
BS25_BULK_SFT_500			79.6	564	769090	6385775
BD25002	13	34	19.3			
BD25003	57	158	12.9			
BD25004	14	24	3.4			
BD25005	69	160	26			
BD25006	47	55	8			
BD25007	53	63	10			

Table 10: Drill Collar Locations for drill core referred to within DFS Report in Appendix 3

Hole ID	GDA94 East	GDA94 North	RL (m)	Dip	Azimuth (grid)	Depth (m)	Drill Type
BD17004	768610.4	6385600.2	626.6	-65.0	72.2	480.7	DDH
BD20012	768680.4	6385955.7	617.7	-70.0	190.0	351.7	DDH
BD21001	768825.0	6385900.0	626.0	-65.0	150.0	450.8	DDH
BD21003	768951.9	6385820.6	657.0	-75.0	21.0	405.0	DDH
BD21042	768800.5	6385428.4	604.6	-80.0	325.0	408.2	DDH
BD25004	769138.6	6385622.7	627.3	-70.0	30.0	222.8	DDH
BD25007	769072.3	6385743.1	646.5	-70.0	20.0	225.8	DDH
BD25019	768800.4	6385676.7	606.5	-75.0	20.0	216.8	DDH

Table 11: Table of significant intercepts for drill core referred to within DFS Report in Appendix 3

Hole ID	Interval	From	Ag (g/t)	Pb (%)	Zn (%)	Au (g/t)
BD25019	0.55	189.15	644	1.71	2.55	0.20
BD25007	4.00	161.00	2,247	2.37	1.03	<0.005
BD25004	1.00	47.00	180	0.24	0.21	<0.005
BD17004	1.00	373.35	60.8	0.48	14.20	1.38
BD21001	1.00	279.00	1,290	4.10	2.06	0.01
BD21003	1.00	312.00	628	0.03	0.01	<0.005
BD20012	1.00	231.20	211	4.77	7.47	0.02
BD21042	2.40	297.30	269	10.33	15.80	0.42

APPENDIX 2: JORC CODE 2012 Edition – ANNEXURE 1

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1m samples from which 3kg was pulverised to produce a 30g charge for fire assay.') In other cases, more explanation may be required such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information.*	• Resources were estimated from RC and diamond core sampling. • Results from exploratory RAB and Aircore drilling were not included in the resource dataset. • For pre-Kingsgate drilling, RC holes were generally sub-sampled by riffle splitting, or spear or grab sampling for rare wet samples and diamond core was halved with a diamond saw. Samples were analysed by several accredited commercial laboratories by either 3, 4 or aqua-regia acid digestion and AA or ICP determination. Quality control measures included use of standards, blanks, field duplicates and external laboratory checks by a variety of methods including neutron activation by GSM for any samples greater than 40ppm Silver. • CRAE also undertook a re-assay and umpire laboratory check program. • For Kingsgate and Silver Mines drilling, RC holes were sub-sampled by cyclone mounted cone splitters and diamond core was either halved or quartered with a diamond saw to provide representative assay sub-samples. The samples were analysed for a suite of elements including silver, lead and zinc by multi-acid digest with ICPAES determination. Measures taken to ensure the sample representivity included routine monitoring of sample recovery, RC field duplicates, and comparison of assay grades from closely spaced drill holes of different phases and types. Assay quality control measures included field duplicates, coarse blanks and reference standards. The available QAQC data demonstrate that the sampling and assaying are of appropriate quality for use in the current estimate. • For gold, master pulps <250g of historic samples sent to ALS Global in Orange and assayed for gold using fire assay technique (Au-AA23). • 400g sample taken from secondary split samples of historic RC holes (BRC17037, BRC17038, BRC17040, BRC17068, BRC17073, BRC17074, BRC17075 & BRC17076) and sent to ALS Global in Canningvale, Western Australia. These were assayed for gold through photon assay utilising a Chrysos Corporation machine.
Drilling techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• Diamond core diameters are nominally either PQ3, HQ3 or NQ. With PQ3 mainly used to pre-collar HQ3 • Selected diamond core prior to Kingsgate was orientated by Eastmans single shot conventional spear by CRAE in 1989. • Silver Mines and Kingsgate diamond core was oriented using Reflex ACT orientation tools.

Criteria	JORC Code explanation	Commentary
		• RC drilling was completed using face sampling hammers.
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.*	• Core recovery is estimated at greater than 95%. • Some zones (less than 10%) were broken core with occasional clay zones where some sample loss may have occurred. However, this is not considered to have materially affected the results. • RC samples are weighed for each metre and assessed for recovery, contamination and effect of water if present. • RC Recoveries were assessed via both recorded sample submission weights and spot checks of bulk sample mass averages for complete holes. • RC Densities were used from the block model to estimate RC sample recovery for each length of RC sampling combined with the drill bit diameter and bulk or split sample weights. • Bulk sample weights to assess RC recoveries were available for 23.8 % of drilling within the Resource bounds. • In 76.2 % of RC drill samples bulk weights were not available sample split weights drilling diameter with block mode densities were used to calculate recoveries. • RC sample recovery is estimated at ~75% based on bulk weights and 87.2% based on sample splits. The expected true sample recovery of sample splits is closer to 75%. • Duplicate samples of wet and later resampling of dry sampling media was undertaken to ensure that no significant sample bias was introduced due to varying sub sampling techniques when unable to be conventionally split due to sample condition. SOP to stop drilling when excessive water and wet sample occurred was noted in drilling logs. On average when recorded 97% of RC sampling is dry and the remaining 3% wet. • No significant relationship between sample recovery and grade exists in either diamond or RC drill samples is observed in scatter plots. Pearson Coefficient calculation of both bulk and sample weights. • Lower RC recoveries are noted to occur in highly porous areas near surface and extremely silicified areas.
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 diamond holes are logged using lithology, alteration, veining, mineralization and structure including geotechnical structure. • RC chip samples are logged using lithology, alteration, veining and mineralisation. • All core and chip trays are photographed using both wet and dry photography.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
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In all cases the entire hole is logged by a geologist.
Quantitative relogging of all Rylstone Volcanic diamond drill core was performed using a combination of geochemical features and depth registered core photography, allowing for quantitative determination of texture and chemistry to suitably distinguish geology and geotechnical features. These geological unites were also able to be interpreted in RC drilling using geo-chemistry and nearby diamond drilling.

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<tr><td>Sub-sampling techniques and sample preparation</td><td>

If core, whether cut or sawn and whether quarter, half or all core were 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 of the material being sampled.

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Minor selective sub-sampling based on geology to a maximum size of 1.3m and a minimum of 0.3m.
CRAE’s RC drilling did not record sample condition however this material was subject to extensive resample and re-assay to satisfy QAQC.
GSM RC Drilling Recorded 153 wet samples 1045 dry and 189 had no sample condition.
Silver Standard RC holes were sampled over one to two metre intervals with sub-samples generally collected by riffle splitting initially to 1/8 or 1/16 splits. Un-mineralised samples were composited over intervals of up to five metres for assaying. Diamond core was halved with a diamond saw with samples collected over intervals ranging from 0.2 to 5.0 metres and averaging 1.0 metre.
Kingsgate’s RC drilling was sampled over one metre intervals and sub-sampled by cyclone mounted cone splitters. 1.34% were recorded as wet, 16.33% were dry.
Kingsgate’s diamond core was sampled over lengths ranging from 0.3 to 2.2 with around 92% of samples representing one metre lengths. Core was either halved or more commonly quartered with a diamond saw to provide assay sub-samples.
RC Sample Conditions within Resource Estimate Boundary-

<table>
<tr><th>Operator</th><th>Dry %</th><th>Moist %</th><th>Wet %</th><th>Unrecorded * %</th><th>Total samples</th></tr>
<tr><td>CRA Exploration (CRAE)</td><td>0.0</td><td>0.0</td><td>0.0</td><td>100.0</td><td>1,662</td></tr>
<tr><td>Golden Shamrock Mines</td><td>79.5</td><td>2.1</td><td>11.6</td><td>6.8</td><td>1,314</td></tr>
<tr><td>Kingsgate Consolidated</td><td>11.8</td><td>13.1</td><td>1.7</td><td>73.4</td><td>9,783</td></tr>
<tr><td>Silver Mines / Bowdens Silver</td><td>79.8</td><td>0.1</td><td>0.1</td><td>20.0</td><td>11,785</td></tr>
<tr><td>Silver Standard</td><td>93.7</td><td>0.5</td><td>4.6</td><td>1.2</td><td>29,389</td></tr>
</table>
*With the exception of CRAE, missing records are observed to be due no record being made as drilling conditions were dry based on spot checks of drillers logs and water strike observations.

Silver Mines RC samples are collected from a cone splitter at a 6% split. The cyclone/splitter system is checked periodically throughout each hole and cleaned when necessary. To assess the representation of material sampled a duplicate 6% split sample is collected from

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</table>

Criteria	JORC Code explanation	Commentary
		a secondary -sample chute on the opposite side of the rotary cone splitter at the rate of 1/20, all intervals retained a secondary sample for check assay on site in drums. • Silver Mines core is cut using a Corewise core saw over lengths ranging from 0.3 to 1.3m with the majority of samples representing one metre lengths with core rotated 10 degrees to the orientation line to preserve the orientation for future reference. The half (NQ & HQ) or quarter (PQ) of the core without the orientation line is removed, bagged and sent to the laboratory for assay. Metallurgical test work samples are made from remaining cut core. • Sample sizes are considered appropriate for the rock type, style of mineralisation, the thickness and consistency of the intersections and assay ranges expected at Bowdens.
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, calibration factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	• Samples from all drilling phases were sent to commercial laboratories for preparation and analysis. No geophysical methods or hand-held XRF devices have been used for resource estimation. • Samples from pre Kingsgate drilling were analysed by several accredited commercial laboratories by either 3, 4 or aqua-regia acid digestion and AA or ICP determination. Quality control measures included use of standards, blanks, field duplicates and external laboratory checks by a variety of methods including neutron activation assaying and umpire laboratory check assaying. • Nominally operators (CRAE, GSM & Kingsgate's) samples were submitted for analysis by ALS in Orange, NSW. After oven drying, and jaw crushing for core samples, the samples were pulverised to nominally 85% passing 75 microns and 25 gram sub-samples digested by multi-acid digest and analysed by ICPAES for a suite of elements including silver, lead and zinc. Quality control measures included field duplicates, coarse blanks and reference standards. • Silver Mines samples were dispatched to ALS Global laboratories in Orange. At ALS the samples were pulverised to nominally 85% passing 75 microns with subsequent 4 acid digest and 33 multi-element analysis completed at ALS Brisbane using method ME-ICP61 and 4 acid digest and 38 multi-element analysis. • Assay results have been confirmed to be without significant bias via various methods including checks for contamination, accuracy and precision. Assays for certified reference materials inserted within the sampling stream are noted to nominally fall within +-2SD of the expected mean value, over various grade ranges for Ag, Pb, Zn and Au. Duplicate variance is noted to be high in diamond core consistent with the mineralisation style and sampling of diamond drill core. • A minority of batches were submitted to SGS West Wyallong for preparation as per ALS and SGS Townsville using method DIG41Q for multi element analysis. • Site Standards are inserted every 20 samples to check quality control and laboratory

Criteria	JORC Code explanation	Commentary
		standards and blanks every 25 samples to further check results. • Silver Mines Metallurgical samples were dominantly dispatched to representative flotation tests by KYSPYmet Adelaide and assay by BV Adelaide. • Grind & Filtration variability samples were dispatched to ALS Orange and transported to ALS Metallurgy Balcatta Perth for grind and filtration optimization, this included being subject to flotation testing under similar regimes established by KYSPYmet. • Pilot Plant Bulk flotation samples were dispatched directly to Core Resources Brisbane from composited crush materials returned from ALS Orange assay work and additional drill core as needed. Core Resources combining the individual samples, to form 3 bulk composites. • Most flotation work was by KYSPYmet with repeated flotation tests between laboratory's consistent with each other. I.e test between laboratories producing similar recoveries, for the representative zones sampled.
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 intersections calculated by site-geologists are procedural with calculations in Datashed and Leapfrog replicating calculated intersections. • Several independent authors reviewed pre-Silver Mines sampling data during preparation of previous resource estimates. • Both Silver Mines and Kingsgate's sampling, logging and survey data were electronically merged into a central database directly from original source files using Logchief field software and imported into an SQL database in accordance with database protocols and manuals. Data was viewed and interpreted using Leapfrog. • No Grade cutting was applied to the assay data for resource estimation as none of the grade estimations are strongly skewed. • Historical documents concerning the compilation of old reports have been cited and spot checks of compilations made extending back to CRAE's initial drill campaign.
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.*	• Accredited surveyors using high accuracy RTK (or similar) surveys accurately surveyed all resource drill hole collars. • Pre-Kingsgate holes were down-hole surveyed by single shot cameras. Kingsgate's drilling was surveyed by either Reflex EZ-shot or Eastman camera. Silver Mines drilling was surveyed by a Reflex EZ-shot electronic camera at 30m intervals down hole. • The terrain includes steep hills and ridges and with a LIDAR topographical model of 0.3 metre accuracy. • All collars recorded in MGA94 zone 55.

Criteria	JORC Code explanation	Commentary
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.*	• The drilling is designed as both infill and extensional to the overall mineral resource envelope. The nominal drill hole spacing is 50m (northing) by 50m (easting). • Hole spacing varies from around 50 by 50 m and locally closer parts of the higher grade ore zones to more than 100 by 100 m in peripheral areas. • The majority of holes were either orientated near vertically or northerly traversing mineralisation and easterly across regional structures. • The data spacing and distribution establishes geological and grade continuity adequately for the current resource estimates.
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.*	• Drill orientation was designed to intersect the projection of breccia zones and zones of veins within an overall mineralized envelope. • An interpretation of the mineralization has indicated that no sampling bias has been introduced.
Sample security	• *The measures taken to ensure sample security.*	• All samples bagged on site under the supervision of two senior geologists with polyweave bags tied with cable ties containing ~20kg of calico sample bags. Batches of secured polyweave bags with chain of custody documentation added being driven by site personnel to the independent laboratory. Counter signed chain of custody documentation for the submitted samples then being returned in addition to digital submission and acknowledgment by the Laboratory.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Pre-Kingsgate sampling techniques and data have been reviewed previously by credentialed external geological consultants and laboratories in the case of CRAE data, and most recently by Silver Mines geoscience staff. • Kingsgate sampling techniques and data have been reviewed by several external geological consultants including MPR and AMC. • Silver Mines sampling techniques and data have been independently reviewed by a number of external geological consultants including AMC, GeoSpy and H&S.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

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.*	• The Bowdens Resource is located wholly within Exploration Licence No EL5920, held wholly by Silver Mines Limited and is located approximately 26 kilometres east of Mudgee, New South Wales. • The tenement is in good standing. • The project has a 2.0% Net Smelter Royalty which reduces to 1.0% after the payment of US$5 million over 100% of the EL5920. • The project has a 0.85% Gross Royalty over 100% of EL5920.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• The Bowdens project was previously managed by Kingsgate Consolidated, Silver Standard Ltd, Golden Shamrock Mines and CRAE. The results also draw on work from the previous owners. Work carried out by these parties has been assessed and verified to be of a high standard with rigorous QAQC, and assay verification programs across multiple laboratories. Similarly spatial and accuracy of collars has been verified to be accurate and compiled in an orderly and verifiable database.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Bowdens Deposit is a low to intermediate sulphidation epithermal base-metal and silver system hosted in Carboniferous aged Volcanic rocks and Ordovician aged sediments. • Mineralisation includes veins, breccias and fracture fill veins within tuff and ignimbrite rocks, and semi massive veins, breccias and fracture fill in siltstone, shale and sandstone. • Mineralisation is overall shallowly dipping (~15 degrees to the north) with high-grade zones at contacts of intrusions, near faults and within brittle lithologies. There are several vein orientations within the broader mineralized zones including some areas of stock-work veins.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* o *easting and northing of the drill hole collar;* o *elevation or RL (Reduced Level elevation above sea level in metres) of the drill hole collar;* o *dip and azimuth of the hole;* o *down hole length and interception depth; and* o *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the*	• See tables in Appendix 1 for material drill hole information pertaining to the MRE update.

Criteria	JORC Code explanation	Commentary
	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 (e.g. 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.*	• Exploration results reported within the document are all length weighted averages aggregated using one of the following cut off and dilution criteria. 1.Intercepts calculated using a 30g/t Ag cut-off, 10 metre internal dilution factor, and 5 metre minimum intercept length. 2. Intercepts calculated using a 90g/t Ag cut-off, 3 metre internal dilution factor, and 1 metre minimum intercept length. 3. Intercepts calculated using a 90g/t Ag Eq cut-off, 3 metre internal dilution factor, and 1 metre minimum intercept length. Bowdens' silver equivalent assumes prices of US$35.00/oz silver, US$2,976/t zinc and US$2,094/t lead with variable metallurgical recoveries based on individual grades, estimated from test work commissioned by Silver Mines Limited. Ag recovery = (6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%, Pb recovery = (8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%, and Zn recovery = (10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%. Silver equivalent formula AgEq(g/t) = Ag(g/t) + Pb(%) x Pb_price/100 x Pb_recovery/Ag_price/31.103477 x Ag_price + Zn(%) x Zn_price/100 x Zn_recovery/Ag_price/31.103477 x Ag_price. In the Company's opinion, the silver, zinc, lead included in the metal equivalent calculations have a reasonable potential to be recovered and sold. Gold (Au) is reported as a separate attribute in Table 5, 6 and 8 but is excluded from the AgEq formula for the 2026 MRE. This is because the 2026 DFS processing flowsheet is optimised for silver, lead and zinc flotation. Gold remains an attribute in the model for informational purposes and may be incorporated in future flowsheet assessments. References to intercepts in Appendix 1 are on a length weighted average unless otherwise specified.
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 (e.g. 'down hole length, true width not known').*	• Mineralisation is both stratabound breccia and vein hosted. The stratigraphy dips moderately to the north in the Main Zone, Aegean and Northwest zones, while the majority of mineralised veins dip west. In Bundarra the mineralization is also stratabound and vein hosted dipping moderately to the Southwest. • Most holes have been drilled angled -60° to -80° to the north and east with occasional holes angled vertically.
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*	• Maps and cross-sections are provided in the body of this announcement and the attached DFS report.

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.*	• Comprehensive reporting is included in the body of the report covering low and high grade intercepts referenced here based on the different cut-offs. Cut offs include high grade Ag intercepts, low grade Ag intercepts and base metal intercepts based on AgEq. • Appendix 1 includes material drill hole information pertaining to the MRE update.
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 and potential deleterious or contaminating substances.*	• The Bowdens diamond holes were also utilised for bulk density measurements. Geotechnical logging has determined suitable ground conditions for mining. • Bulk sample sites have verified material estimates to be accurate. • Extensive metallurgical test work and flowsheet optimisation has been undertaken across all ore types and grade ranges. Results typically demonstrate excellent recoveries. • Forty six composites were made up of cut drill core (half or quarter PQ & HQ), drilled by SVL in 2025. Specifically, metallurgical zones representative of the Reserve production profile and geologies were composited. Refer to Figure 9 for mean composite locals and prior exploration results for collars in Appendix 1. • Four Composites were taken to optimise grind and filtration processes. • Twenty nine Composites were of SMU bench scale 3-6m across the deposit to represent the deposit variability with respect to grind and flotation. • Ten Composites were of SMU bench scale 3-6m across the deposit to represent the deposit variability with respect to tailings filtration. • Three Other bulk samples were based on the ore hardness for pilot plant flotation and bulk tailings generation of tails for filtration, • The bulk samples represent high, medium and low Ag grade samples from the three metallurgical zones (hard moderate and soft). • Checks for deleterious or other penalty elements (such as Cadmium of Mercury and Fluorine) have been assayed for in a routine manner and determined to be acceptable from metallurgical product results. Other penalty, elements including Arsenic have been appropriately estimated.
Further work	• *The nature and scale of planned further work (e.g. 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.*	• A portion of historical assaying did not include gold, and therefore the Company is assessing the potential for any future gold resource. Where available, historical samples will be assayed for gold. • The Company also intends drill targeting of near pit extensions.

SILVER MINES LIMITED

Criteria	JORC Code explanation	Commentary
		• Regionally other prospects are being advanced.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in the preceding sections 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.*	• All geological data is stored electronically with limited automatic validation prior to upload into the secure DataShed database, managed in the on-site office by Geological staff. The master drill hole database is located on an SQL server, which is backed up on a daily basis. • Basic checks were performed prior to this resource estimate to ensure data consistency, including checks for FROM - TO interval errors, missing or duplicate collar surveys, excessive down hole deviation, and extreme or unusual assay values. • All data errors/issues were reported to the Geological staff to be corrected or flagged as omitted but retained for audit in the primary DataShed database if repeat measurements could not be made.
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 Competent Person has visited the Bowdens project site on two occasions: for 2 days in late January 2022 and over a 2 week period in late July and early August 2017. During these visits, core samples and outcrops were examined and discussions were held with Silver Mines personnel about the geology and mineralisation of the deposit. No additional site visit was undertaken in connection with the 2026 MRE update. The Competent Person concludes that data collection and management were being performed in a professional manner.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• Silver Mines has developed a comprehensive geological interpretation of the Bowdens deposit based on geological logging, detailed petrography and chemical assays. In 2024 all diamond drill core photography within the Rylstone was analysed quantitatively separately and in conjunction with geochemical features. This work confirmed and further refined existing interpretations of unit contacts. Silver Mines personnel have a good understanding of the geology of the Bowdens deposit, and this is reflected in the wireframe models they prepared, which form a solid framework for Mineral Resource estimation. • Silver Mines had previously interpreted a series of thin higher-grade mineralised horizons or lenses in the Rylstone Volcanics and the underlying Coomber Formation, which have an average intersection length of 2.90m in the Rylstone and 6.25m in the Coomber. The Rylstone Mineralised Horizons (RMHs) are typically silver-rich, while the Bundarra lenses in the Coomber Formation are primarily base metal (lead-zinc) dominant. The seven RMHs are thought to represent paleo-boiling horizons and can be quite discontinuous with

Criteria	JORC Code explanation	Commentary
		numerous gaps and embayments. The six Bundarra lenses cut across stratigraphy and appear reasonably continuous spatially. The higher-grade lenses have variable orientation, with dominant directions of 12°>330° for the RMHs and 15°>180° for the Bundarra lenses. Additionally, SVL provided a mineralised fracture domain, which defines the boundary of the Ag, Pb and Zn propagation in the mineral system. This domain was derived in conjunction with DataRock Pty Ltd from core imagery and geochemistry, largely independent of silver assays. This domain was intended to be used as a hard mineralisation boundary within the Rylstone Volcanics for estimates of Ag, Pb, Zn, Sb and Cd. For other estimates of Au Mn, As, Cu, Sb, Fe and S it was treated as a soft boundary. • Thin higher-grade mineralised horizons or lenses were used to guide the overall orientation of the lower-grade mineralisation locally and divide the deposit into a number of different orientation domains. The Rylstone Volcanics are divided into five separate domains, while the Coomber Formation is split into three domains. The eastern edge of mineralisation is controlled but not constrained by the Eastern Fault, which forms a separate domain in each stratigraphic unit. • Surfaces for base of complete oxidation and top of fresh rock were also interpreted, based on geological logging and assays. Only a small proportion of mineralisation occurs within the relatively thin oxide zone, and there is no obvious evidence of depletion or enrichment of silver due to oxidation. • There is some scope for alternative geological interpretations of the deposit, principally in the correlation of intersections that comprise the different mineralised horizons or lenses. While this could affect estimates locally, it appears unlikely to have a significant impact on the global Mineral Resource estimate. • Geology guides and controls Mineral Resource estimation by using the local orientation of the higher-grade horizons or lenses to guide the overall orientation of the lower-grade mineralisation and divide the deposit into a number of different orientation domains. The eastern edge of mineralisation is effectively truncated by the Eastern Fault, which forms a separate domain in each stratigraphic unit. • The continuity of geology at Bowdens is controlled by stratigraphy and faulting. Continuity of grade has a weak stratigraphic control and is primarily controlled by local fracturing; faulting and permeability appear to act as broad control on localising mineralisation.
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.*	The open-pit Mineral Resources at Bowdens have an approximate extent of: • 900m east-west, • 1,2000m north-south, • From surface to 340m below surface.

Criteria	JORC Code explanation	Commentary
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters, 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 (e.g. 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.*	• Samples were composited to nominal 2.0m intervals within each unit for data analysis and resource estimation, reflecting the scale of open pit mining envisioned by Silver Mines. • The resource model uses a parent block size of 25x25x5m, while drill hole spacing is nominally 25x25m in the better drilled areas of the deposit. So, the parent block size is identical to the hole spacing, which is considered appropriate for MIK (multiple indicator kriging) estimation. Sub-blocks of 12.5 x 12.5 x 2.5m were used for ordinary kriging (OK) estimates, which is half the parent block dimensions in each direction and is considered appropriate. • The resource model uses the GDA94 (Geocentric Datum of Australia) grid, zone 55. • Silver was initially estimated by recoverable MIK into 25 x 25 x 5.0m panels. These estimates were then localised by discretising the metal distribution into sub-blocks with the dimensions of the selective mining unit (SMU) of 12.5 x 12.5 x 2.5m. The order of assigning the metal distribution to sub-blocks was based on an (OK) estimate for silver into the sub-blocks. • Gold was estimated by MIK, using the e-type or average block grade at the scale of the panels; this coarser resolution reflects the substantial under-assaying of gold compared to silver in the Rylstone Volcanics. • All other attributes were estimated by OK, including Pb, Zn, Cu, S, As, Sb, Cd, Mn, Fe and dry bulk density. OK is considered appropriate because the coefficients of variation (CV=SD/mean) are generally low to moderate, and the grades are reasonably well structured spatially. Recoverable MIK was chosen for Ag primarily because it allows better mining selectivity than OK. • MIK estimates were generated using GS3M software, while OK estimates were produced in Datamine software. Each of the major stratigraphic units (Rylstone, Coomber, Shoalhaven) were estimated separately, with each unit sub-divided into domains based on changes in mineralisation orientation. • A four pass search strategy was used for the OK grade estimates: 1. 35x35x12.5m search, 16-32 samples, minimum of 4 octants informed 2. 52.5x52.5x12.5m search, 16-32 samples, minimum of 4 octants informed 3. 105x105x25m search, 16-32 samples, minimum of 4 octants informed 4. 105x105x25m search, 8-32 samples, minimum of 2 octants informed

Criteria	JORC Code explanation	Commentary
		• An additional larger pass was used for some elements with fewer assays to ensure estimates in all blocks that had an estimated silver value. • The MIK estimates used 16-48 samples; search radii and octant constraints were identical to the OK estimates. • The oxide zone was estimated using a dynamic search parallel to topography. • The maximum extrapolation distance will be somewhat less than the maximum search radius due to the octants constraints requiring at least 2 drill holes. Maximum extrapolation distance is around 90m. • It is assumed that an Ag-Pb-Zn sulphide concentrate will be produced. All elements have been estimated independently for each domain. • No assumptions were made regarding the correlation of variables during estimation because each element was estimated independently. Some elements do show moderate to strong correlation in the drill hole samples, and the similarity in variogram models effectively guarantees that this correlation will be preserved in the estimates. • A number of potentially deleterious elements have been estimated, including As, Sb and S. • Dry bulk density was estimated directly into the model from the drill hole samples, using a similar methodology to the other elements; fewer samples were required, reflecting the wider distribution of density measurements. • The geological interpretation controls the Mineral Resource estimates through the use of the major stratigraphic boundaries, which were used as hard boundaries during estimation. The Eastern Fault also controls the Mineral Resource estimates locally, with mineralisation parallel to this structure. • No grade cutting was applied to any of the grade estimates because none of the grade distributions are strongly skewed. Sensitivity analysis on Ag estimates indicated that grade cutting has minimal impact on the grade estimates. • The new model was validated in a number of ways – visual comparison of block and drill hole grades, statistical analysis, examination of grade-tonnage data, and comparison with previous models. All the validation checks indicate that the grade estimates are reasonable when compared to the composite grades, allowing for data clustering. • The 2026 MRE reported above a 25g/t AgEq (2026) cut-off grade (2026) has comparable tonnages and grades to the 2024 MRE above a 30g/t AgEq (2024) cut-off grade (2024). The exception is gold, which shows a marked decrease in grade because this metal is no longer included in the silver equivalent formula. This indicates that the new Mineral Resource estimate takes appropriate account of this previous estimate.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>The deposit remains unmined so there is no reconciliation data.</td></tr>
<tr><td>Moisture</td><td>Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.</td><td>Tonnages are estimated on a dry weight basis. Moisture content has been determined for some of the density samples, by comparing sample weights before and after oven drying.</td></tr>
<tr><td>Cut-off parameters</td><td>The basis of the adopted cut-off grade(s) or quality parameters applied.</td><td>
The cut-off grade is a silver equivalent (AgEq) value, based on grades and recoveries for silver, zinc, and lead as shown below. Gold in no longer included in the AgEq formula because this metal was not included in the DFS study.
<table>
<tr><th>Metal</th><th>Unit</th><th>Price (USD)</th></tr>
<tr><td>Silver (Ag)</td><td>Ounce (oz)</td><td>$35.00</td></tr>
<tr><td>Zinc (Zn)</td><td>Tonne (t)</td><td>$2,976</td></tr>
<tr><td>Lead (Pb)</td><td>Tonne (t)</td><td>$2,094</td></tr>
</table>
Variable metal recovery formulas were used, based on metallurgical testwork:
Ag recovery = (6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%,
Pb recovery = (8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%,
Zn recovery = (10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%.
Silver equivalent formula:
AgEq(g/t) = Ag(g/t)
+ Pb(%) x Pb_price/100 x Pb_recovery/Ag_price/31.103477 x Ag_price
+ Zn(%) x Zn_price/100 x Zn_recovery/ Ag_price/31.103477 x Ag_price
It should be noted metal prices and recoveries used for Mineral Resources are identical to those used in the Ore Reserves.
The adopted cut-off grade of 25 g/t AgEq is considered likely to be economic for the mining method and scale of operation envisioned for Bowdens Silver.
</td></tr>
<tr><td>Mining factors or assumptions</td><td>Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It may not always be possible to make assumptions regarding</td><td>Surface mining by open pit method is currently planned for Bowdens.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	mining methods and parameters when estimating Mineral Resources. Where no assumptions have been made, this should be reported.	• The recoverable MIK method implicitly incorporates internal mining dilution at the scale of the assumed SMU. No specific assumptions were made about external mining dilution in the Mineral Resource estimates.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It may not always be possible to make assumptions regarding metallurgical treatment processes and parameters when reporting Mineral Resources. Where no assumptions have been made, this should be reported.*	• The recoveries for each metal are based on available metallurgical test work. It is assumed that sulphide ore will be treated by conventional froth flotation to produce a bulk Ag-Pb-Zn concentrate.
Environmental factors or assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	• It is proposed that all process residue and waste rock disposal will take place on site in purpose built and licensed facilities. • All waste rock and process residue disposal will be done in a responsible manner and in accordance with any mining license conditions.
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.*	• Dry bulk density is measured on-site using an immersion in water method (Archimedes principle) on selected core intervals for nominal 10cm samples. The Bowdens database contains 7,076 of these measurements in 254 drill holes. There are also a number of density measurements derived from weighing trays of core – this information confirms the immersion method results. • Samples are weighed before and after oven drying overnight at 110ºC to determine dry weight and moisture content.
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (i.e., relative confidence in tonnage/grade estimations, 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.*	• The classification scheme is based on the estimation search pass for Ag; Pass 1 = Measured, Pass 2 = Indicated and Pass 3 = Inferred. Pass 4 is not classified as part of the Mineral Resource Estimate but could be considered as a potential Exploration Target. • This scheme is considered to take appropriate account of all relevant factors, including the relative confidence in tonnage and grade estimates, confidence in the continuity of geology and metal values, and the quality, quantity and distribution of the data. • The classification appropriately reflects the Competent Person's view of the deposit.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• This Mineral Resource Estimate has been reviewed by Silver Mines and HSC personnel and no material issues were identified.

Criteria	JORC Code explanation	Commentary
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 level in the Mineral Resource estimates are considered to be in line with the generally accepted accuracy and confidence of the nominated JORC Mineral Resource categories. This has been determined on a qualitative, rather than quantitative, basis, and is based on the estimator's experience with a number of similar deposits elsewhere. The main factor that affects the relative accuracy and confidence of the Mineral Resource estimate is drill hole spacing, because there are no strong geological controls on the primary mineralisation. • The estimates are local, in the sense that they are localised to model blocks of a size considered appropriate for local grade estimation. The tonnages relevant to technical and economic analysis are those classified as Measured and Indicated Mineral Resources. • No production data is available because this deposit has not been previously mined.

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, 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 Mineral Resource estimate that this Reserve is based upon has been compiled by H&S Consultants Pty Ltd, using data supplied by Bowdens Silver. • The mine design is based on the 2026 Mineral Resource. • Silver and Gold were estimated by recoverable multiple indicator kriging (MIK) into 25 x 25 x 5.0m panels. These estimates were then localised by discretising the metal distribution into sub-blocks with the dimensions of the selective mining unit (SMU) of 12.5 x 12.5 x 2.5m. All other attributes were estimated by OK into sub-blocks, including Pb, Zn, S, Cu, As, Cd, Sb, Mn, Fe and dry bulk density. • The Mineral Resources reported are inclusive of the Ore Reserves.
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.*	• Andrew Hutson of Resolve Mining Services Plus (Competent Person) visited the site between 3^{rd} and 4^{th} July 2024. • The site visit included viewing all areas selected for development including the mining and processing areas plus the waste dump and tailings storage locations. Additionally, meetings were undertaken with technical staff and management.

Criteria	JORC Code explanation	Commentary
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 Ore Reserve estimate was based on the Definitive Feasibility Study (DFS) for the Project. • The DFS scope covers Stage 1 of the mine design (29.9Mt over approximately 16.75 years). Stage 2 of the mine design (18.3Mt, extending the mine life to approximately 26 years) incorporates a filtered tailings landform that has been assessed at pre-feasibility study level only. All other aspects of the Stage 2 mine design are at DFS level. The Ore Reserve and associated financial modelling have been prepared on this basis, and investors should assess Stage 1 (DFS-level) and Stage 2 (pre-feasibility-level) components accordingly. • Financial modelling completed to support this Ore Reserve estimate is based on the DFS and this modelling shows that the Ore Reserve is economically viable at metal prices supported by consensus long term price scenarios. • The LOM mine design associated with the Reserve and used in the economic model contains 0.63% Inferred material. However, no Mineral Resources classed as Inferred have been included in the 2026 Proven or Probable Reserves.
Cut-off parameters	• *The basis of the cut-off grade(s) or quality parameters applied.*	• Ore cut-off values are based on NSR values where the reporting NSR is defined as the net value A$ value per tonne of ore after consideration of all costs (mining, process, general and administration, product delivery), metallurgical recoveries, sustaining capital, concentrate metal payabilities and treatment charges, transport costs and royalties. • The NSR cut-off applied for the definition of ore and waste within the production plan is A$30/t. per tonne.
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.*	• The MIK modelling process accounts for any internal dilution within the orebody. No provision for edge dilution has been applied to the MIK modelling, and 100% mining recovery is applied. • The mined tonnage and diluted grade is based on the block model size, with excavator based open pit mining, and equipment size appropriate minimum mining widths applied. • Pit optimisations utilising the Lerchs-Grossmann algorithm with industry standard software were undertaken. This optimisation utilised the Mineral Resource model together with cost, revenue, and geotechnical inputs. • The resultant pit shells were used to develop detailed pit designs with due consideration of geotechnical, geometric, and access constraints. These pit designs were used as the basis for production scheduling and economic evaluation. • The mining method is conventional open pit whereby excavation, loading and hauling will be executed by 200t excavators matched to 100t trucks after drilling and blasting the majority of the ore and waste.

Criteria	JORC Code explanation	Commentary
	• *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.*	• Mine roads are designed to be suitable for the largest equipment travelling along routes typically 25m wide for dual lane traffic. • The geotechnical parameters have been applied based on geotechnical studies. • During the above process, Inferred Mineral Resources were excluded from mine schedules and economic valuations utilised to validate the economic viability of the Ore Reserves.

Metallurgical factors or assumptions	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.*
	• *Whether the metallurgical process is well-tested technology or novel in nature.*
	• *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.*
	• *Any assumptions or allowances made for deleterious elements.*
	• *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.*
	• *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*
		• Conventional crush and grind to p80 106µm followed by bulk flotation with regrind, cleaner and recleaner stages producing a single concentrate. Grinding and flotation testwork has been based on a variety of representative individual and composited cut drill core samples across the ore body and optimised to market conditions.
		• Process plant metal recovery curves have been developed from a combination of:
		  ○ Composited drill core representing physical pit domains
		  ○ Composited drill core representing geochemical domains
		  ○ Individual drill core variability samples
		• Deleterious elements have been estimated and factored into revenue calculations.
		• Two bulk samples sites have been blasted and 12 tons of material subject to pulverization and detailed chemical characterizations this work agrees with other assays from drilling.

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.*	• Detailed Waste Rock classification has been undertaken and verification programs are being prepared for regulatory approval.
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.*	• The Company has either title or lease agreements for the land on which all proposed infrastructure is situated. All other infrastructure and resources are in existence or have been appropriately planned for. • The Company owns the existing site offices, core yard facilities, mobile equipment, light vehicles and other housing and storage facilities associated with properties purchased. All infrastructure associated with the Project must be purchased. • Key project infrastructure items: process plant, filtered tailings thickening and filtrations plant, filtered tailings landform, mine waste dumps, ROM pad and low-grade ore stockpile, offices, warehouse and laydown yards, process and mining workshops, road vehicle and

Criteria	JORC Code explanation	Commentary
		mobile plant washdown facility, fuel storage and dispensing facility, internal haulage and access roads and relocated external public road. • The BSP is located in an established mining district in the Central Tablelands Region of New South Wales. Key nearby offsite infrastructure and facilities include: - Access to Port Botany or Port of Newcastle via separate road routes; - Major mining equipment service centres; - Mining economy based towns with broad range of mining, supply and maintenance services; - Hospitals and other medical facilities; and - Workplace training and education providers.
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.*	• Mining operating costs have been developed utilising an owner-mining model and generated from first-principles. Drill & blast costs were based on contractor estimates. Equipment maintenance & ownership costs were based on information supplied by the equipment manufacturer. • All costs and prices have been based in Australian dollars. Where a USD conversion is required, a factor of 0.70 has been applied. Exchange rates are based on the Company's assumptions. • Private royalties payable include a 2.0% Net Smelter Royalty (reducing to 1.0% after payment of US$5m) and a 0.85% Gross Royalty, both over EL5920. NSW state mining royalties have also been incorporated.
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.*	• Detailed feed grades were derived from the mine plan. Financial assumptions, including metal prices, exchange rates and NSR elements, treatment costs and transport, freight, and insurance costs were derived from Bowden corporate financial and economic assumptions. These economic assumptions are generally derived from relevant industry references such as analyst forecasts and industry commercial terms for similar products. • Assumed 100% ore mining recovery of the diluted Resource Model. • Ore Revenue Estimate metal price assumptions were: o Silver = US$35.00/oz o Lead = US$0.95/lb o Zinc = US$1.35/lb

Criteria	JORC Code explanation	Commentary
Market assessment	• *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *Price and volume forecasts and the basis for these forecasts.* • *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	• Silver is the primary commodity of the project and has experienced ongoing demand increases associated with the growth of renewable energy industries. Demand is currently exceeding supply and is expected to continue to do so for the near future. • No offtake agreements have been entered into. Marketing and smelting terms have been estimated from information provided by relevant entities.
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.*	• An overarching financial model of the Bowdens project, prepared by Silver Mines, using mining inputs prepared by Resolve, and other inputs consistent with the Ore Reserve estimate, indicates the project is economically viable with a positive NPV. • The financial model incorporates updated metal prices from those used in the Ore Reserve Estimate and are: o Silver = US$35.00/oz o Lead = US$0.95/lb o Zinc = US$1.35/lb • Sensitivity of the Bowdens Project to changes in the key drivers of silver price, mining cost, processing cost and geotechnical pit slope were carried out, and showed the project NPV to be most sensitive to significant changes in metal prices.
Social	• *The status of agreements with key stakeholders and matters leading to social licence to operate.*	• Silver Mines continues to negotiate a range of commitments with private landowners through the Land Access Agreement process and also through acquisition of freehold property.
Other	• *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* • *Any identified material naturally occurring risks.* • *The status of material legal agreements and marketing arrangements.* • *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. 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*	• There are no offtake agreements in place and no formal market studies have been undertaken. However, the company monitors market conditions and has engaged the services of Calgacus AG (Switzerland) to provide advice on concentrate marketing. • Potential delays to project commencement or completion due to external issues including: Delay in obtaining necessary government approvals, and delay in establishing grid power connection, due to reliance on external parties and associated permitting issues. • The Project is classified as State Significant Development (SSD) under the NSW Environmental Planning and Assessment (EP&A) Act 1979 and the Independent Planning Commission (IPC) of New South Wales is the consent authority. The project was previously granted development consent by the IPC in April 2023. Following a judicial review initiated

Criteria	JORC Code explanation	Commentary
	and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	by a local entity opposed to the project, the NSW Court of Appeals made a decision in August of 2024 to set aside the development consent previously granted by the IPC. The decision was based on a legal technicality and was not related to the merits of the project. The NSW government amended the EP&A Act in December 2024 to strengthen the intended legal interpretation of the Act. Refreshed environmental assessments have been provided to the NSW Department of Planning Housing and Infrastructure to assist their positive recommendation for redetermination of the project by the IPC. Following redetermination of the project, acquisition of a series of secondary approvals including the Mining Lease (ML) and the posting of rehabilitation bonds will need to occur prior to the commencement of mining. This DFS is based on further refinement of the 2024 Optimisation design with all components now at AACE International Class 3 or better engineering designs and cost estimates. Additional government approvals will be required. • It is expected all necessary approvals and licenses will be forthcoming when applied for progressively over the next phase of the project.
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).*	• Ore Reserves reported here are classified as both Proved and Probable as they are derived from Measured and Indicated Mineral Resources in accordance with the JORC Code (2012). • The Proved Ore Reserve is derived entirely from Measured Mineral Resources. [• The Probable Ore Reserve is derived from Indicated Mineral Resources. • No Probable Ore Reserves have been derived from Measured Mineral Resources. • The Competent Person is satisfied that the stated Ore Reserve classification reflects the outcome of technical and economic studies.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	• External audits of Ore Reserve Estimate have not been undertaken. • The Mineral Reserve estimate, mine design, scheduling, and mining cost model has been subject to internal peer review processes by Resolve Mining Solutions. No material flaws have been identified.
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.*	• Reporting of the project Ore Reserve considers; ○ the Mineral Resources compliant with the JORC Code 2012 Edition, ○ the conversion of these resources into Ore Reserves, and ○ the costed mining plan capable of delivering ore from a mine production schedule. • Dilution of the Mineral Resource model and an allowance for ore loss was included in the Ore Reserve estimate. All the Mineral Resources intersected by the open pit mine designs

Criteria	JORC Code explanation	Commentary
	• *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	classified as Measured and Indicated Resource have been converted to Proved and Probable Ore Reserves after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Project. • The mine planning and scheduling assumptions are based on current industry practice, which are seen as globally correct at this level of study. • The Project team has estimated the cost estimates and financial evaluation with specialist consultants and team members, which are considered sufficient to support this level of study. The accuracy of the cost estimate is -10/ + 20% in line with typical Definitive Feasibility level study accuracy. Stage 2 of the mine design (18.3Mt) incorporates a filtered tailings landform at pre-feasibility study level and the cost estimates for that component are more uncertain. • Ore Reserve is most sensitive to unfavourable changes in factors that influence revenue. These include mining dilution and ore loss, processing recovery, and silver price. Processing recovery has been based upon included feed grades and metallurgical testwork. Mining dilution and ore loss have been tested to within industry benchmarks for global values.

APPENDIX 3: DFS Report

Lycopodium

Bowdens Silver Pty Ltd

Bowdens Silver Project

Definitive Feasibility Study

BOWDENS SILVER

3319-GREP-001

July 2026

REV NO.	DATE	DESCRIPTION OF REVISION	BY	DESIGN APPROVED	PROJECT APPROVED	CLIENT APPROVED
B	20.07.26	ISSUED FOR STUDY				
A	17.07.26	ISSUED FOR REVIEW	SC	-	SC	-

Bowdens Silver Project

Definitive Feasibility Study

3319-GREP-001

Table of Contents

Page

Table of Contents

Table of Contents

Table of Contents

Table of Contents

Table of Contents

Page

Table of Contents

Table of Contents

Page

Table of Contents

Table of Contents

FIGURES

Table of Contents

Table of Contents

Table of Contents

Page

DISCLAIMER

This report has been prepared for Bowdens Silver Pty Ltd (Bowdens) by Lycopodium (APAC) Pty Ltd (Lycopodium) as an independent consultant and is based in part on information furnished by Bowdens and in part on information not within the control of either Bowdens or Lycopodium. While it is believed that the information, conclusions and recommendations will be reliable under the conditions and subject to the limitations set forward herein, Lycopodium does not guarantee their accuracy. The use of this report and the information contained herein shall be at the user's sole risk, regardless of any fault or negligence of Lycopodium.

1.0 SUMMARY

1.1 Introduction

1.1.1 Introduction

This Definitive Feasibility Study (DFS) report is for the Bowdens Silver Project (BSP or Project), an open cut silver, lead and zinc project near the village of Lue, in the state of New South Wales (NSW) in Australia. The Project is owned by Bowdens Silver Pty Ltd (Bowdens or the Company), a wholly owned subsidiary of Silver Mines Limited (Silver Mines). This report is styled in the format of a National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) for familiarity of international readers, but is not NI 43-101 compliant.

The Australian Joint Ore Reserves Committee Code (JORC) compliant Mineral Resource and Ore Reserve estimates are listed in Sections 1.13 and 1.14, respectively.

The DFS is for BSP and specifically references a 29.9 million tonnes (Mt) ore mine plan, which is a sub-set of the Life of Mine (LOM) plan which contains 48.2 Mt ore. Both the DFS and LOM plans and financial evaluations contain small amounts of Inferred Resources as demonstrated in Table 1.1.1.

Table 1.1.1 DFS and LOM Mine Plan Material Categories

Resource Category	Reserve Category	Unit	Mine Plan			
			DFS	%	LOM	%
Measured	Proven	t (dry)	28,572,550	95.5%	44,606,742	92.6%
Indicated	Probable	t (dry)	1,222,004	4.1%	3,251,400	6.8%
Meas & Ind	Proven & Probable	t (dry)	29,794,554	99.6%	47,858,141	99.4%
Inferred	N/A	t (dry)	113,130	0.4%	303,842	0.6%
Total Mine Plan		t (dry)	**29,907,684**	**100.0%**	**48,161,983**	**100.0%**

The DFS is based on AACE International Class 3 or better engineering designs and cost estimates for all site infrastructure and operations, based on significant field and laboratory test programs.

The DFS mine plan is considered Stage 1 and has been intentionally limited in scale to match the Stage 1 tailings landform which holds approximately 30 Mt of filtered tailings. The LOM plan, or Stage 2, includes the additional 18.3 Mt in the LOM plan and requires a Stage 2 filtered tailings landform, while all other mine infrastructure in the DFS design remains adequate. The design of the Stage 2 filtered tailings landform is at pre-feasibility level. All aspects (other than Stage 2 of the filtered tailings landform) of the LOM plan are at DFS (AACE International Class 3) level.

Unless otherwise noted, all physicals and financials provided in this report refer to the DFS mine plan, not the LOM plan.

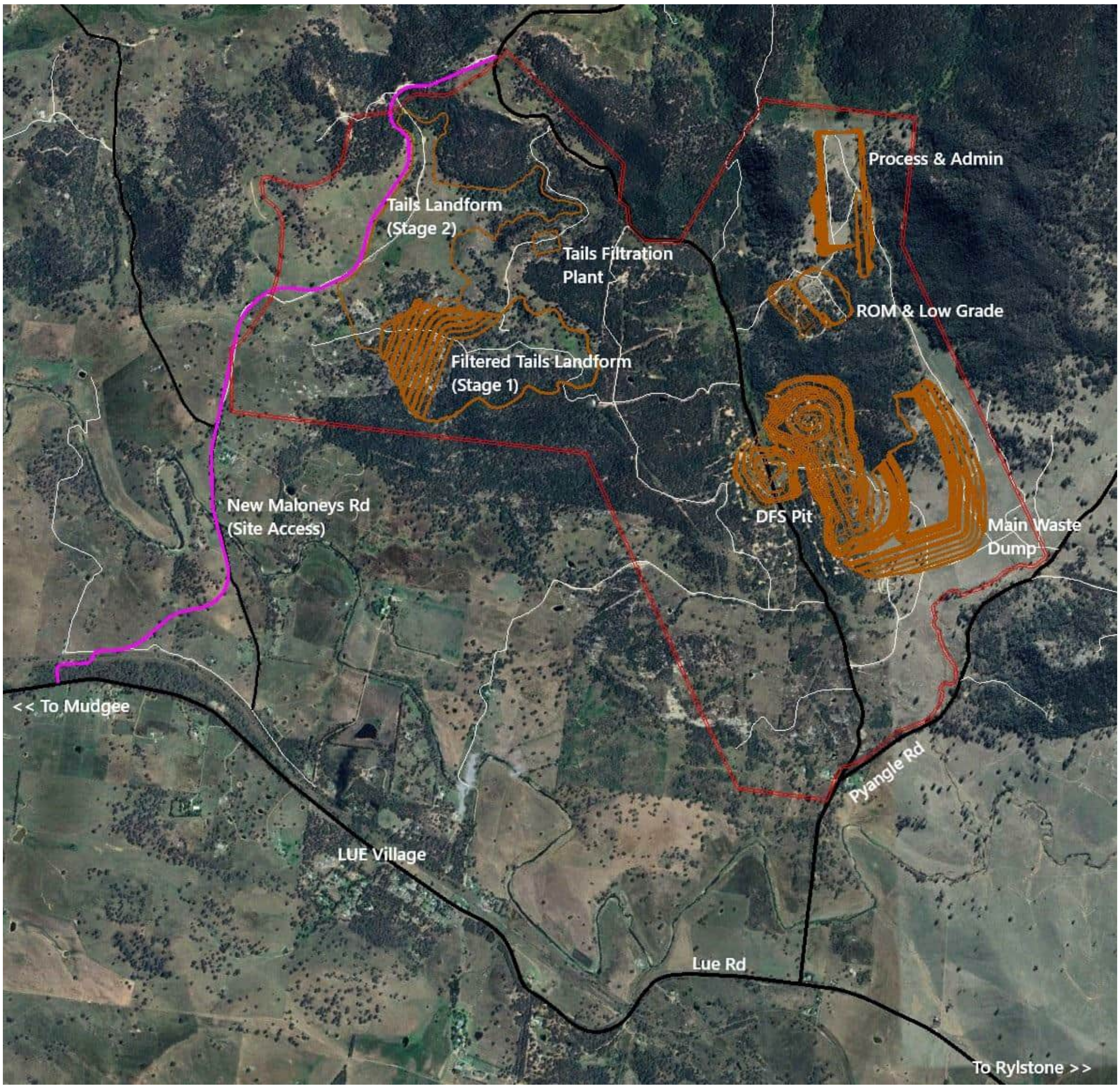
Lycopodium (APAC) Pty Ltd (Lycopodium), on behalf of Bowdens, is the lead author of the DFS with contributions from Bowdens and various other technical experts.

The purpose of the DFS was to comprehensively evaluate the Project, which has been determined to be technically feasible and economically robust. Revenue factors are highly conservative based on silver prices of US$35 /oz for the pit optimisations and US$45 /oz for the financial modelling.

1.1.2 General Site Layout

The Project general site layout with key infrastructure items is shown in Figure 1.1.1

Figure 1.1.1 General Site Layout



1.2 Reliance on Other Experts

The following parties are the key contributors to the DFS:

- Bowdens – Sections 1, 4, 5, 6, 7, 8, 9, 10, 11, 12, 18.1, 18.2, 18.11, 20, 22, 23, 25.2, 26.2.
- Lycopodium – Sections 2, 3, 13, 17, 18.3, 18.4, 18.9, 18.10, 21.3, 21.4, 21.5, 24, 25.1, 26.1.

- H & S Consulting Pty Ltd – Section 14.
- Resolve Mining Solutions – Sections 15, 16, 21.1, 21.2.
- SRK Consulting (Australasia) Pty Ltd (SRK) – Sections 18.5, 18.6, 18.7, 18.8.
- Calgacus AG (Switzerland) – Section 19.

1.3 Property Description and Location

The Project is located near the village of Lue, approximately 26 kilometres (km) from the regional town of Mudgee, in the Mid-Western Regional Council (MWRC) Local Government Area (LGA) of NSW.

The Project site is located approximately 2.5 km northeast of the village of Lue. A large topographical ridge lies between the Project and village centre which provides a natural visual shield and noise attenuation.

Nearby the village of Lue, land is predominantly zoned for primary production and is used mainly for larger grazing and cropping enterprises with interspersed small hobby farms and lifestyle properties. Other land uses nearby Lue include viticulture, olive production, small accommodation venues, and a motocross track.

Approximately 27 km to the north of the site are three well established and operating coal mines owned by Glencore, Yancoal, and Peabody.

Silver Mines is an Australian based resource company that is publicly listed on the Australian Securities Exchange (ASX:SVL). Its subsidiary company, Bowdens,) owns a number of Exploration Licences (ELs) issued by the NSW Government in the wider region that allow for the exploration of Group 1 Metallic Minerals. The Bowdens deposit is located wholly within EL5920, and the Mining Lease Application (MLA) area covers areas within EL5920 and adjoining EL6354.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Project lies between the regional towns of Mudgee and Rylstone, in the Central Tablelands Region of NSW. BSP is located in an established mining district with outstanding logistics, local suppliers and operators to support future mine development.

The main access to the Project is via Lue Road, which provides a link between Ulan Road, near Mudgee in the west, and Bylong Valley Way near Rylstone in the east. Lue Road is an approved B-double route for general mass limit (GML) and B-double vehicles up to 25 metres (m) long between Ulan Road and Bylong Valley Way.

Site access is via Lue Road, which is the main road between Mudgee and Rylstone, and smaller, unsealed sections of Pyangle Road and Maloneys Road. As the existing Maloneys Road bisects the site, a section of Maloneys Road will be realigned to the west of Lue.

The site is situated on the outer western flanks of the Great Dividing Range which is dominated by elevated rocky ridges separated by either broad and flat or partially confined alluvial valley settings. The Project site is located within a rural area principally used for cattle and sheep grazing, with some cropping and small scale production of olives and grapes.

The topography generally ranges in elevation from approximately 770 m Australian Height Datum (AHD) within peaks and ridges associated with the Great Dividing Range in the northeast, to elevations below 550 m AHD within the alluvial valley of Lawsons Creek to the southeast of the Project.

1.5 History

The original exploration licences covering the Bowden property and a number of surrounding properties were granted in 1989 and 1990 to CRA Exploration Pty Limited (CRAE). In July 1989, follow-up exploration of an anomalous stream sediment sample and mineralised float, led to the discovery of outcropping sulphide-bearing rocks which assayed 860 grams per tonne (g/t) silver, 1.0 percent (%) zinc, and 0.5% lead. Between 1989 and 1992, CRAE carried out exploration activities which resulted in the discovery of the Bowdens Gift Zone of outcropping mineralisation approximately 500 m east of the discovery outcrops.

The Project had various owners from 1994 to 2011, at which time it was purchased by Kingsgate Consolidated Limited (Kingsgate). Kingsgate utilised the existing drilling database to complete a Joint Ore Reserves Committee (JORC) code-compliant resource estimate in late 2011 and again later in November 2012. Kingsgate also commenced a feasibility study (FS) and environmental impact study that were halted in 2015.

In March 2016, Silver Mines announced it had entered into a Sale and Purchase agreement with Kingsgate whereby it would acquire 85% of BSP. The purchase of the Project by Silver Mines was completed on 29 June 2016. The acquisition of the remaining 15% of the Project was announced on 29 December 2016.

Since owning BSP, Silver Mines and Bowdens have progressed through various Resource and Ore Reserve upgrades, produced an Environmental Impact Statement (EIS) covering multiple environmental, social and economic assessments, and continued mineral exploration activities on site and within the wider region.

1.6 Geological Setting and Mineralisation

The Project sits on the eastern margin of the Lachlan Orogen and the western edge of the Sydney Basin, where Late Carboniferous felsic volcanic rocks of the Rylstone Volcanics unconformably overlie the older Coomber Formation basement. The deposit is interpreted as an intermediate-sulphidation epithermal silver-lead-zinc (Ag–Pb–Zn) (± gold (Au)) system, developed on the southern margin of a caldera-related volcanic complex, with mineralisation controlled by extensional faults, permeable volcanic units, and the emplacement of dacitic intrusive rocks.

Ore occurs in several styles (stratiform fracture-fill veins, stockworks, hydrothermal breccias, and deeper replacement-style sulphides) with strong vertical and lateral zonation from cooler, silver-rich distal zones to hotter, deeper base-metal and gold-bearing zones. The assemblage is dominated by silver sulphosalts, galena, sphalerite, pyrite, and carbonate gangue; reflecting a long-lived, polyphase hydrothermal system driven by structurally focused fluid flow and evolving thermal gradients.

1.7 Deposit Types

In terms of deposit style, BSP shows the closest affinity to Ag-dominant intermediate-sulphidation epithermal systems such as Creede, Colorado, while also sharing certain shallow-level and structural characteristics with low-sulphidation systems such as El Peñón and regionally relevant similarities with the Drake–Mount Carrington district. These systems typically exhibit high Ag:Au ratios, commonly Ag head grades ranging from the 100s to locally 1,000s g/t, and combined Pb and Zn grades generally in the order of 1% to 5%, although substantial variation occurs between deposits and within individual ore zones.

1.8 Exploration

Since its discovery by CRAE in 1989, the deposit has been explored through successive campaigns of surface sampling, drilling and geophysics by Golden Shamrock Mining (GSM), Silver Standard Australia Pty Limited (Silver Standard), Kingsgate and Silver Mines, including metallurgical and geotechnical programs. Surface geochemistry comprises more than 2,400 samples (rock chip, soil, stream sediment and bulk rock) within the deposit area, confirming a partially outcropping silver orebody.

Geological and structural mapping has progressively defined the deposit's detailed features, and more recently the integration of geophysics and geochemistry has improved the inference of geological boundaries. Geophysics has contributed at several scales. Heliborne VTEM (2004) gave a subtle response and airborne magnetics and radiometrics (2016) defined surficial alteration. Induced polarisation (2017), ground gravity (2020), and 2D seismic (2022 - 2023) helped define the core of the system, its structure and geology. Collectively, these surveys have delineated the intra-caldera setting, mapped the low-density fractured Main Zone via gravity linked IP responses to higher-temperature pyrite–base-metal associations, and identified conjugate fault sets over a deep intrusive body tapped by major regional structures.

1.9 Drilling

Drilling at BSP has been conducted in successive campaigns between 1989 and 2025 by CRAE, GSM, Silver Standard, Kingsgate, and Silver Mines. The database holds 1,138 holes for 175,269 m within 1 km of the 2025 Resource block model consisting of 332 diamond core (DC) holes (100,505 m) and 585 reverse-circulation holes (70,068 m), with the balance made up of water-bore, percussion, auger and aircore holes drilled for non-resource purposes. DC was drilled at PQ3, HQ3 or NQ3 diameters and oriented by conventional spear in the early campaigns and Reflex ACT tools under Kingsgate and Silver Mines.

Most holes were drilled at -60 degrees (°) to -80° towards the north and east to intersect breccia zones and vein projections, with no significant sampling bias identified, and collars have been resurveyed to sub-10 cm accuracy in MGA94 Zone 55. Core recovery is excellent. A length-weighted mean of 98.6% (median 100%) across 66,773 intervals, with a mean Rock Quality Designation (RQD) of 88.8% (median 97%). The nominal 50 m by 25 m spacing over higher-grade zones, widening peripherally, is considered adequate to establish geological and grade continuity for resource estimation.

1.10 Sampling Preparation, Analyses and Security

Diamond drill core is prepared by halving (NQ and HQ) or quartering (PQ) to provide representative 1 m assay intervals. RC drilling used face-sampling hammers, with sub-sampling by riffle splitting (CRAE, GSM, Silver Standard) or cyclone-mounted rotary cone splitters (Kingsgate, Silver Mines). Sample weights are monitored per metre, with primary RC samples typically 2 - 3 kilograms (kg) and retained archive duplicates of equivalent mass. Security measures include tamper-evident polyweave bags with zip ties, sequentially numbered sample identifications (IDs), digital and paper chain-of-custody documentation, and direct transport from site to the laboratory (ALS Orange (ALS)) by operator staff.

The analytical history spans multiple laboratories (Amdel, Analabs, Becquerel, ALS, SGS), with modern assays using fire assay for Au and four-acid digest ICP-MS for Ag–Pb–Zn and multi-element analysis; laboratory bias checks show generally good agreement between them. The Quality Assurance / Quality Control (QA/QC) programme comprises certified reference materials (CRM) (15 types, 11,977 insertions), field blanks, and field and pulp duplicates, with control charts monitoring performance against certified expected values and action limits. Field duplicate repeatability is better for RC (90.6% of Ag pairs within 30% relative difference) than for DC (74.7%), reflecting the heterogeneous, vein-style mineralisation rather than any analytical problem; a conclusion supported by repeat assays showing that grade variability, not assaying, drives the duplicate variation.

1.11 Data Verification

This report is styled in the format of a NI 43-101 Technical Report for familiarity of international readers, but is not NI 43-101 compliant. Data verification is as presented in each section of the report and has not been formally verified by Qualified Persons (QPs), other than the Mineral Resource Estimate (MRE) and Ore Reserve Estimate (ORE). However, all key assessments and engineering designs have been completed by professional entities and the verification by QPs can be arranged if Bowdens so elects.

1.12 Minerals Processing and Metallurgical Testing

Metallurgical testwork for BSP has been undertaken over more than two decades by multiple operators and laboratories, generating an extensive dataset covering mineralogy, comminution, flotation, and dewatering characteristics to support DFS process design. Early programs (1989 - 2013) established that the mineralisation is amenable to conventional sulphide flotation and defined the relationship between grind size, liberation, and metallurgical performance, with silver predominantly associated with galena and sulphosalt minerals. Subsequent programs (2017 - 2026), progressively refined comminution parameters, flotation performance, and variability understanding, and evaluated alternative flowsheet configurations.

DFS testwork demonstrated that differential flotation, while technically viable, is sensitive to operating conditions and variability, and subject to metal and impurity deportment challenges, particularly with tetrahedrite-group minerals. In response, a simplified bulk flotation flowsheet was developed and adopted, comprising bulk sulphide rougher flotation, regrind of bulk concentrate to approximately 80% product passing size (in microns) (P_{80}) 20 micrometres (μm), and two-stage cleaning. This approach delivers high and consistent recoveries of Ag, Pb, and Zn, reduces sensitivity to ore variability, and improves operational robustness relative to differential circuits. High silver grade bulk concentrates are currently highly marketable.

Comminution testwork indicates that the ore exhibits low to moderate hardness and is suitable for conventional Semi-Autogenous Grinding (SAG) –ball milling, with a primary grind size of approximately P_{80} 106 μm providing an appropriate balance between throughput, energy efficiency, and downstream metallurgical performance. Flotation variability testwork confirms consistent bulk sulphide recovery across a wide range of feed compositions, with performance primarily controlled by mineralogy, liberation, and grind size. Bulk flotation testwork on representative composites demonstrates that high recoveries of silver and base metals can be achieved at low concentrate mass pull, producing a single bulk Ag–Pb–Zn concentrate suitable for downstream upgrading and handling.

Dewatering testwork confirms that concentrates are amenable to conventional thickening and filtration, and that pressure filtration of tailings is technically feasible, achieving filter cake moisture contents typically in the range of approximately 15 - 20 percent by weight (wt%) H_2O and producing material suitable for landform (dry-stack) disposal. Overall, the metallurgical testwork demonstrates that the selected comminution, bulk flotation with regrind and cleaning, and dewatering processes are well understood, technically robust, and appropriate for BSP, providing a sound basis for 2026 DFS process design and performance assumptions.

1.13 Mineral Resource Estimates

The MRE was undertaken by H & S Consulting Pty Ltd (H&SC) using GS3 and Datamine software.

The MRE is shown in Table 1.13.1 as documented in *HSC_Bowdens_Update_20260528*.

Table 1.13.1 Updated Mineral Resource Estimate at 25 g/t AgEq Cut-off

Unit	Class	Mt	AgEq g/t	Ag g/t	Pb %	Zn %	Au g/t	Moz AgEq
Coomber	Measured	8.2	45	14	0.39	0.65	0.13	12
Coomber	Indicated	19.2	43	13	0.39	0.61	0.14	26
Coomber	Meas. & Ind.	27.4	43	13	0.39	0.63	0.14	38
Coomber	Inferred	23.9	38	12	0.38	0.51	0.14	29
Rylstone	Measured	96.4	59	44	0.25	0.34	0.01	183
Rylstone	Indicated	19.1	45	34	0.18	0.24	0.03	28
Rylstone	Meas. & Ind.	115	57	42	0.24	0.32	0.02	211
Rylstone	Inferred	6.5	41	32	0.15	0.20	0.04	9
Total	**Measured**	**105**	**58**	**41**	**0.26**	**0.36**	**0.02**	**195**
Total	**Indicated**	**38.3**	**44**	**23**	**0.29**	**0.43**	**0.08**	**54**
Total	**Meas. & Ind.**	**143**	**54**	**37**	**0.27**	**0.38**	**0.04**	**249**
Total	**Inferred**	**30.4**	**38**	**16**	**0.33**	**0.44**	**0.12**	**37**
Total	**Total**	**173**	**51**	**33**	**0.28**	**0.39**	**0.05**	**286**

The MRE is compliant with the Australian JORC. While this report as a whole is not compliant with or being issued as an NI 43-101 Technical Report, the MRE complies with all disclosure requirements set out in the NI 43-101 *Standards of Disclosure for Mineral Project* (2023) and adheres to the 2019 CIM *Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (2019 CIM Guidelines).

1.14 Ore Reserve Estimate

The ORE was undertaken by Resolve Mining Solutions using Dassault Systèmes' Geovia suite of software.

The ORE is shown in Table 1.14.1 and is based on the 2026 MRE discussed in Section 1.13 and Section 15, and documented in *RMS SVM003 Bowden Silver Mine 2026 DFS and Ore Reserve (Final) 29052026*.

Table 1.14.1 Ore Reserve Estimate

Classification	Reserve Grade				Contained Metal		
	Tonnes Mt	Ag g/t	Zn %	Pb %	Ag Moz	Zn kt	Pb kt
Proved	44.6	61.7	0.37	0.27	88.5	165	122
Probable	3.3	47.3	0.21	0.15	4.9	7	5
Total	**47.9**	**60.8**	**0.36**	**0.26**	**93.5**	**172**	**127**

It is important to note that the net smelter return (NSR) calculation used for the original Whittle optimisation runs was conservatively based on an Ag price of US$35 /oz.

The ORE is compliant with the Australian JORC guidelines. While this report as a whole is not compliant with or being issued as an NI 43-101, the Reserve complies with all disclosure requirements set out in the NI 43-101 *Standards of Disclosure for Mineral Project* (2023) and adheres to the 2019 CIM *Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (2019 CIM Guidelines).

1.15 Mining Operations

As previously discussed, the DFS mine plan is a sub-set of the LOM plan. Unless otherwise noted, all physicals and financials provided in this report refer to the DFS mine plan, not the LOM plan. The DFS component satisfies a Class 3 engineering design while the LOM only needs to demonstrate being economically mineable under reasonable financial assumptions.

The DFS mining inventory is shown in Table 1.15.1.

Table 1.15.1 DFS Mine Inventory

Description	Unit	Qty
Mining		
Ore	Mt	29.91
Waste	Mt	43.65
Total	Mt	73.56
SR		1.46
Grade		
Ag	g/t	67.52
Pb	%	0.25
Zn	%	0.34

The Project mine optimisation, design and scheduling was undertaken by Resolve Mining Solutions using Dassault Systèmes' Geovia suite of software. Based on the geometry and physical properties of the deposit and the surrounding terrain and the proximity of the deposits to surface, the most appropriate mining method is open pit mining using rigid frame trucks and backhoe configured excavator.

To facilitate a consistent mine production schedule and to allow the early targeting of the higher grade ores, the DFS mine design and schedule incorporates five pit stages. Most stages have a northern ramp for ore and a southern ramp for waste.

1.15.1 Equipment and Labour

The mining plan is based on in-house (Owner operator) mining with the exceptions of a rock-on-ground drill and blast contractor and an original equipment manufacturer (OEM) maintenance agreement. Mining is planned on a continuous dayshift operation with a two-panel roster.

For the purposes of the DFS, an all Caterpillar mining fleet has been used for sizing selections and costing, but this is not to be interpreted as a final selection. Direct mining equipment (including ROM operations) includes:

- 6020B excavator.
- 777 trucks.
- D9T bulldozers.
- 844 wheel dozer.
- 14M grader.
- 745 ADT watercarts.
- 988 wheel loaders.

Additional to direct mining and ROM operations, there is a requirement to rehandle the tailings from the filter plant to the filtered tailings landform (FTL), with the following dedicated fleet:

- 988 wheel loader.
- 745 ADT ejector trucks.
- 825 soil compactor.

The operating cost estimate is based on a first principles owner mining (except maintenance, and drill and blast) cost model.

1.16 Ore Processing Recovery Methods

The process plant has been designed to achieve the following key objectives:

- Prevention of workplace hazards.
- Maximise Ag, Pb and Zn recovery.
- Maximise Ag payability.
- Minimise overgrinding of highly altered soft materials (when present) in ore to maximise filtration rate of tailings.
- Minimise process operability and metal recovery impacts of variable geo-metallurgical domains in ore feed.
- Minimise capex and opex in balance with the above items.

The key process design criteria are:

- Throughput of 2.0 Mtpa of ore with a primary grind size of 80% passing (P_{80}) 106 µm.
- Crushing plant utilisation of 60.0% (5,256 hours per year (h/y)).
- Grinding and flotation plant utilization of 91.3% (8,000 h/y) supported by crushed ore storage and standby equipment in critical areas.

The treatment facility incorporates the following unit process operations:

- Two-stage crushing comprising primary jaw crushing followed by secondary crushing to produce a crushed product with a particle size of 80% passing (P_{80}) 35 mm.
- Primary and ball mill grinding circuit operating in SAB mode (SA mill and ball mill in series), with classification via fine screening and hydrocyclones.

- Bulk rougher flotation of sulphide ore comprising a train of rougher flotation cells with reagent addition at the conditioning stage and selected cells.
- Rougher concentrate regrind using a stirred mill operating in open circuit.
- Two-stage cleaner flotation comprising Cleaner 1 tank cells followed by Cleaner 2 flotation utilising a Jameson Cell to produce a final concentrate.
- Bulk concentrate thickening, filtration, storage, and load-out, including pressure filtration and storage in a covered stockpile facility prior to transport.
- Tailings thickening followed by filtration and disposal to a filtered tailings landform, with return of filtrate to the process water system.
- Reagent preparation and distribution systems supporting flotation and dewatering operations.
- Process plant and tailings treatment utilities including water services and air services.

1.17 Project Infrastructure

Key project infrastructure items have been located and designed with the following objectives:

- Minimise overall disturbance footprint and impacts to biodiversity.
- Minimise haulage / travel distances of materials and personnel.
- Minimise noise, dust and light emissions.
- Minimise raw water usage and release of potentially impacted water.
- Minimise external visibility.

Project infrastructure items include:

- Process plant.
- Filtered tailings thickening and filtrations plant.
- Filtered tailings landform.
- Mine waste dumps.
- ROM pad and low-grade ore stockpile.
- Offices.
- Warehouse and laydown yards.
- Process and mining workshops.
- Road vehicle and mobile plant washdown facility.
- Fuel storage and dispensing facility.
- Internal haulage and access roads.
- Relocated external public road (Maloneys Road).

1.18 Market Studies and Contracts

The final product is a high precious metals (HPM) concentrate containing Ag, Pb, Zn, and Au, which will be transported to a smelter for further processing. While the Company is monitoring market conditions and the high demand for HPM concentrates, there are no offtake agreements in place and no formal market studies have been undertaken.

For DFS purposes, costs are based on overland transport of the HPM concentrate in sealed half-height containers from site to Port Botany plus ocean freight to China or Korea. Details of transport and shipping costs (A$183 /wet metric tonne (wmt)) are provided in Table 1.21.2.

Treatment Charges (TCs) (US$60 /dry metric tonne (dmt)) and Refining Charges (RCs) (US$0.45 /oz Ag) are conservatively estimated at roughly the average of 2024 and 2025 pricing, although current market conditions are more favourable.

A pilot scale flotation program was conducted to produce sufficient concentrates to complete the necessary shipping tests, generate a safety data sheet (SDS) and detailed assays. Classifications of the HPM concentrate are provided in Table 1.18.1.

Table 1.18.1 Dangerous Goods Classifications

Transport Code / Regulation	Material Covered	Classification
Dangerous Goods (ADG and IMDG)	Packages goods by road, rail and sea	Class 9 (UN 3077)
IMSBC Code	Solid bulk cargoes	Group A[1] and Group B (MHB(TX))
MARPOL Annex V	Residues of solid bulk cargoes	Harmful to the Marine Environment (HME)

[1]If shipping the material as a solid bulk cargo, the concentrate is classified as a Group A cargo (may liquefy). For Group A cargoes, the shipper shall be responsible for ensuring that a test to determine the TML is conducted within six months of the date of loading of the cargo.

1.19 Environmental Studies, Permitting and Social or Community Impact

1.19.1 Environmental Studies

An EIS was completed by R.W. Corkery & Co in May 2020 and amended with updates July 2021 and March 2022. The EIS included rigorous assessments of water resources, visual amenity, human health, traffic, society, biodiversity and other environmental aspects, as well as the benefits of the project to the local and regional economy; which provides an in depth understanding of the Project's potential impacts and benefits. The EIS concluded that the benefits of the Project outweigh the potential impacts.

The DFS incorporates design improvements to major infrastructure items and operational changes that further reduce any potential impacts to the environment and community.

1.19.2 Permitting

The project is classified as State Significant Development (SSD) under the NSW Environmental Planning and Assessment (EP&A) Act 1979 and the Independent Planning Commission (IPC) of NSW is the consent authority. The project was previously granted development consent by the IPC in April 2023.

Following a judicial review initiated by a local entity opposed to the project, the NSW Court of Appeals made a decision in August of 2024 to set aside the development consent previously granted by the IPC. The decision was based on a legal technicality and was not related to the merits of the project. The NSW government amended the EP&A Act in December 2024 to strengthen the intended legal interpretation of the Act.

Refreshed environmental assessments have been provided to the NSW Department of Planning Housing and Infrastructure (DPHI) to assist the redetermination of the project by the IPC. Following redetermination of the project, acquisition of a series of secondary approvals including the Mining Lease (ML) and the posting of rehabilitation bonds will need to occur prior to the commencement of mining.

This DFS is based on further refinement of the 2024 optimisation design with all components now at AACE International Class 3 or better engineering designs and cost estimates. Additional government approvals will be required after redetermination of the previously approved project.

1.19.3 Community Impact

The Project will deliver substantial positive economic benefits on a local and statewide scale through employment, business opportunities and state revenue. There will also be localised perceived negative impacts mainly through amenity impacts and competition with agricultural lifestyles.

1.20 Capital and Operating Costs

1.20.1 General

The key capital and operating cost inputs have been estimated from a variety of sources as listed below:

- Resource – H&SC.
- Mining – Resolve Mining Solutions, SRK, Caterpillar, Roc-Drill Pty Ltd, Goodyear Australia Pty Limited, Bowdens.
- Processing – Lycopodium, Bowdens, equipment suppliers.
- G&A – Bowdens, Schneider Electric (Australia) Pty Ltd.
- TC & RC – Calgacus, Bowdens.
- Shipping and Transport – Qube Holdings Ltd.

1.20.2 Capital Costs

The initial capital cost for the DFS is estimated at A$454.5M (US$318.2M), including a contingency allowance of A$30.0M (US$21.0M). Sustaining capital is estimated at A$23.5M (US$16.5M). Details are summarised in Table 1.20.1 in Australian dollars (A$) and United States dollars (US$).

Table 1.20.1 DFS Capital Costs

Item / Category	2026 DFS A$	2026 DFS US$
Process and Tails Plant	180,492,517	126,344,762
Site Infrastructure	67,372,771	47,160,940
Offsite Infrastructure	45,924,101	32,146,871
Waste Management	71,041,683	49,729,178
Mobile Equipment	5,898,790	4,129,153
Capitalised Pre-Production	30,878,385	21,614,869
Indirects	52,911,944	37,038,361
Initial Capital	***454,520,191***	***318,164,134***
Infrastructure and Equipment	13,097,362	9,168,153
Waste Management	10,452,564	7,316,795
Sustaining Capital	***23,549,926***	***16,484,948***
Total Capital	**478,070,118**	**334,649,082**

1.20.3 Operating Costs

Total and unit operating costs are provided in Table 1.20.2.

Table 1.20.2 DFS Total and Unit Operating Costs

Operating Costs (Real exc. Stockpile Adj.)	A$...M	A$ /t Milled	US$...M	US$ /t Milled
Mining	573.02	19.16	401.11	13.41
Rehabilitation	20.57	0.69	14.40	0.48
Processing	620.13	20.73	434.09	14.51
G&A	180.10	6.02	126.07	4.22
TC / RCs	68.84	2.30	48.19	1.61
Shipping Costs	74.94	2.51	52.46	1.75
Royalties	88.90	2.97	62.23	2.08
NSW State Royalty	99.40	3.32	69.58	2.33
Total Operating Costs	**1,725.89**	**57.71**	**1,208.12**	**40.40**
Mining ($ /t Mined)		**7.76**		**5.43**

Brook Hunt cash cost summaries in Ag terms are provided in Table 1.20.3.

Table 1.20.3 DFS C1 Operating Costs Summary

C1 Operating Costs	A$...M	A$ /oz	US$...M	US$ /oz
C1 Costs				
Mining (inc. Grade Control and Stockpile Adj.)	575.75	11.27	403.03	7.89
Rehabilitation	20.57	0.40	14.40	0.28
Processing	620.13	12.14	434.09	8.50
G&A	180.10	3.53	126.07	2.47
TC / RCs	68.84	1.35	48.19	0.94
Shipping Costs	74.94	1.47	52.46	1.03
Royalties	88.90	1.74	62.23	1.22
Zinc Revenue	(113.13)	(2.21)	(79.19)	(1.55)
Lead Revenue	(139.75)	(2.74)	(97.82)	(1.92)
Gold Revenue	(1.59)	(0.03)	(1.09)	(0.02)
C1 Costs	***1,374.79***	***26.91***	***962.35***	***18.84***
C2 Costs				
C1 Costs	1,374.79	26.91	962.35	18.84
Initial Capital Depreciation	451.79	8.84	316.25	6.19
Sustaining Capital Depreciation	23.55	0.46	16.48	0.32
C2 Costs	***1,850.13***	***36.22***	***1,295.09***	***25.35***
C3 Costs				
C2 Costs	1,850.13	36.22	1,295.09	25.35
NSW State Royalty	99.40	1.95	69.58	1.36
C3 Costs	***1,949.52***	***38.17***	***1,364.67***	***26.72***
All-in-Sustaining Cost (AISC)				
C3 Costs	1,949.52	38.17	1,364.67	26.72
Initial Capital Depreciation	-451.79	-8.84	-316.25	-6.19
All-in-Sustaining Cost (AISC)	**1,497.74**	**29.32**	**1,048.42**	**20.53**

1.21 Economic Analysis

Economic analyses have been carried out for four separate options; two mining inventories and two metal price decks for each mining inventory. The two mining inventories are the DFS mine plan and the LOM plan as discussed in Sections 1.14 and 1.15. The two metal price decks are the base case (at less than consensus Ag prices) and a spot case.

The four options or cases are:

- DFS Base (this DFS).
- DFS Spot.
- LOM Base.
- LOM Spot.

The economic analyses have been carried out for the project using a financial model developed by Northshore Capital Advisors. The model has been developed using quarterly periods taking into account revenues, processed tonnages and grades, process recoveries, metal prices, operating costs, refining charges, royalties, tax and capital expenditures (both initial and sustaining). The economic analyses have been conducted on a 100% equity (no debt financing) basis and in constant dollar terms. Sunk costs (for tax purposes) and future sale of land assets were included in the model.

The Base Case economic analysis is conservatively based on a Ag price of US$45 /oz, less that the current consensus price. Results at spot prices are also provided to demonstrate the upside potential of the project. As discussed in Section 1.14, it is important to note that the original NSR calculation and resulting Whittle optimisation were based on a very conservative Ag price of US$35 /oz. Therefore, the financial model is extremely conservative and demonstrates the robustness of the project.

Exchange rates and metal prices incorporated in the financial models are provided in Table 1.21.1.

Table 1.21.1 FX and Price Deck

Parameter	Unit	Base Case		Spot Price	
		A$	US$	A$	US$
Exchange Rate	A$:US$	0.70		0.69	
Silver Price	$ /oz	64.29	45.00	87.96	61.00
Zinc Price	$ /lb	1.93	1.35	2.34	1.62
Lead Price	$ /lb	1.29	0.90	1.24	0.86
Gold Price	$ /oz	4,929	3,450	5,977	4,145

Transport and shipping costs incorporated in the financial model are provided in Table 1.21.2.

Table 1.21.2 Transport and Shipping Costs

Transport and Shipping Costs	A$ /wmt	US$ /wmt
Container Hire and Loading	16.98	11.89
Inland Freight to Port of Departure	84.29	59.00
Freight Insurance	2.03	1.42
Sea Freight Cost	78.57	55.00
Sampling and Assaying	1.42	0.99
Transport, Shipping, TC & RC Costs	**183.29**	**128.31**

Key physicals of the DFS and LOM plans are and presented in Table 1.21.3.

Table 1.21.3 Key Physicals

Project Physicals	Unit	2026 DFS	2026 LOM
Project Life (inc. Construction)	Years	16.75	26.00
Operating Life	Years	15.25	24.50
Ore Mined	kt	29,908	48,162
Waste Mined	kt	43,935	81,372
Strip Ration	x	1.47	1.69
Ag Mined Grade	g/t	67.52	60.63
Zn Mined Grade	%	0.34%	0.36%
Pb Mined Grade	%	0.25%	0.26%
Au Mined Grade	g/t	0.01	0.02
Ag Recovery	%	82.8%	82.1%
Zn Recovery	%	90.8%	91.2%
Pb Recovery	%	80.9%	81.4%
Ag Recovered in Concentrate	koz	53,768	77,056
Zn Recovered in Concentrate	kt	91.02	156.96
Pb Recovered in Concentrate	kt	61.47	103.22
Au Recovered in Concentrate	koz	7.45	22.48
Ag Grade in Concentrate	g/dmt	4,494	3,692
Zn Grade in Concentrate	%	24.5%	24.2%
Pb Grade in Concentrate	%	16.2%	15.9%
Concentrate Produced	k dmt	372	649
Transport Weight	k dmt	413	721
Payable Ag	koz	51,080	73,203
Payable Zn	kt	26.6	45.8
Payable Pb	kt	49.3	83.7
Payable Au	koz	0.32	7.50

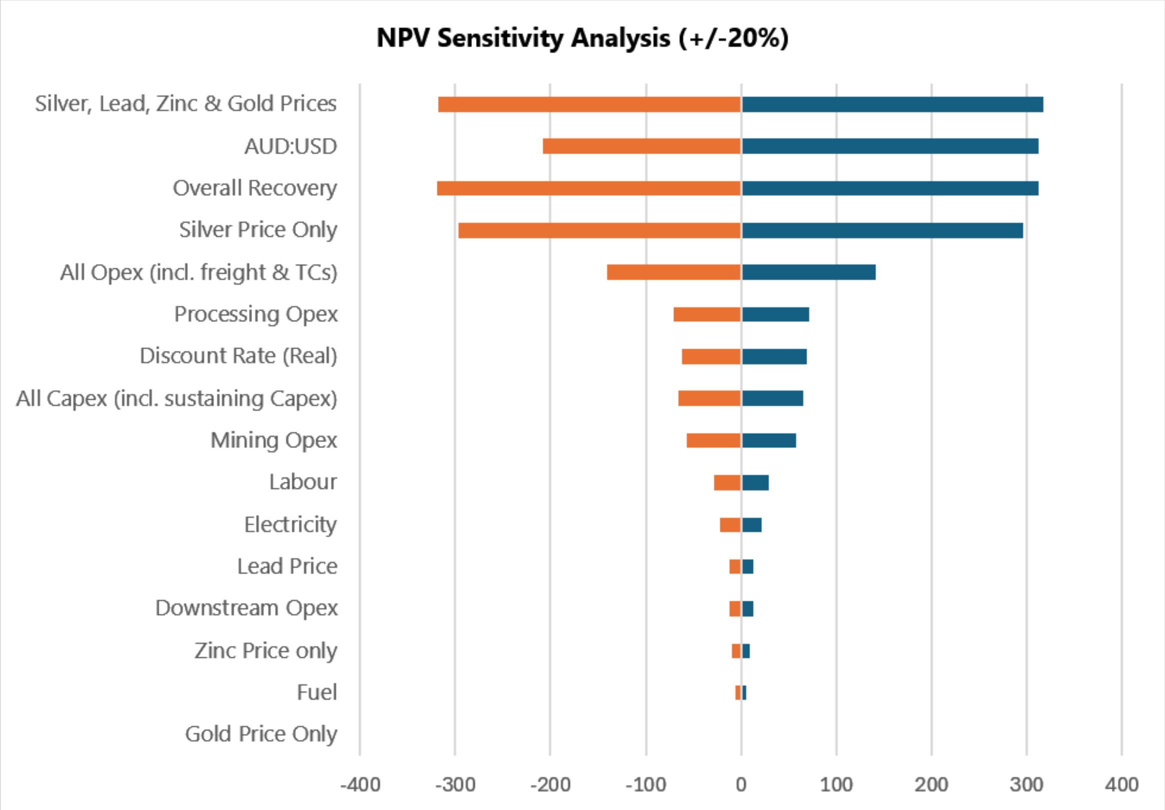
Key Base Case financial metrics of the DFS and LOM are presented in Table 1.21.4.

Table 1.21.4 Base Case Key Financial Metrics

Project Financials (Ungeared): Real Unless Stated					
Description	Unit	2026 DFS		2026 LOM	
		A$	US$	A$	US$
Revenue from Payable Metals	$...M	3,538.1	2,476.7	5,175.1	3,622.5
Operating Expenses	$...M	(1,725.9)	(1,208.1)	(2,784.7)	(1,949.3)
Operating Margin	$...M	1,812.2	1,268.6	2,390.4	1,673.3
Post Closure Sale of Land and Buildings	$...M	100.0	70.0	100.0	70.0
Initial Capex	$...M	(423.6)	(296.5)	(423.6)	(296.5)
Pre-Production Expenses	$...M	(30.9)	(21.6)	(30.9)	(21.6)
Initial Capex and Pre-Production Expenses	$...M	(454.5)	(318.2)	(454.5)	(318.2)
Sustaining Capex	$...M	(23.5)	(16.5)	(90.8)	(63.6)
Total Capital	$...M	(478.1)	(334.6)	(545.4)	(381.8)
Undiscounted Cashflow Pre-Tax	$...M	1,434.2	1,003.9	1,945.0	1,361.5
Tax Payable	$...M	(378.5)	(265.0)	(532.5)	(372.8)
Undiscounted Cashflow After Tax	$...M	1,055.6	738.9	1,412.5	988.7
C1 Cost (Ag basis)	$ /oz	26.91	18.84	29.75	20.82
C2 Cost (Ag basis)	$ /oz	36.22	25.35	37.16	26.01
C3 Cost (Ag basis)	$ /oz	37.17	26.72	39.08	27.36
All-in-Sustaining Cost (AISC): Ag Basis	$ /oz	29.32	20.53	32.91	23.04
C1 Margin (Ag Basis)	%	58.1%		53.7%	
C2 Margin (Ag Basis)	%	43.7%		42.2%	
C3 Margin (Ag Basis)	%	40.6%		39.2%	
AISC Margin (Ag Basis)	%	54.4%		48.8%	
Project NPV (Pre-Tax)	$...M	877.4	614.2	1,043.3	730.3
Project NPV (Post Tax)	$...M	625.7	438.0	735.8	515.0
Project IRR (Pre-Tax)	%	31.5%		31.5%	
Project IRR (Post Tax)	%	26.2%		26.2%	
Project Payback Period from Construction Start	Years	4.6		4.6	
Project Payback Period from Production Start	Years	3.0		3.0	
Maximum Project Drawdown (Nominal)	$...M	455.2	318.6	455.2	318.6
Breakeven Silver Price	$ /oz	37.54	26.28	37.53	26.27
Silver Price to achieve 30% After Tax IRR	$ /oz	70.31	49.22	70.31	49.22
Profitability Index	x	2.40 x		2.64 x	
Ratio of NPV to Pre-Production Capital	x	1.93		2.30	

Figure 1.21.1 shows the key project post-tax NPV sensitivities for the Base Case.

Figure 1.21.1 Base Case - Sensitivity Analysis



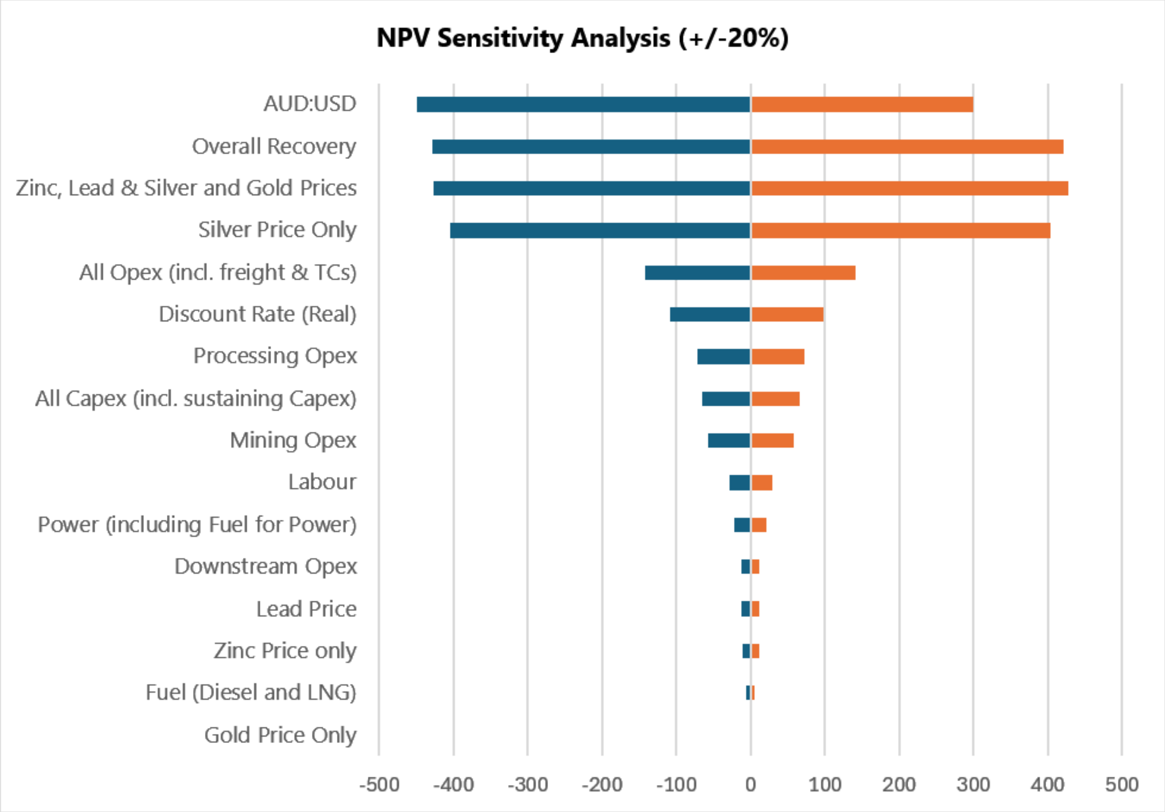
Key Spot Case financial metrics of the DFS and full LOM are presented in Table 1.21.5.

Table 1.21.5 Spot Case Key Financial Metrics

Project Financials (Ungeared): Real Unless Stated					
Description	Unit	2026 DFS		2026 LOM	
		A$	US$	A$	US$
Revenue from Payable Metals	$...M	4,766.6	3,305.7	6,948.7	4,818.9
Operating Expenses	$...M	(1,812.2)	(1,256.8)	(2,911.2)	(2,018.9)
Operating Margin	$...M	2,954.4	2,048.9	4,037.4	2,800.0
Post Closure Sale of Land and Buildings	$...M	100.0	69.4	100.0	69.4
Initial Capex	$...M	(425.3)	(294.9)	(425.3)	(294.9)
Pre-Production Expenses	$...M	(30.9)	(21.4)	(30.9)	(21.4)
Initial Capex and Pre-Production Expenses	$...M	(456.2)	(316.4)	(456.2)	(316.4)
Sustaining Capex	$...M	(23.6)	(16.4)	(90.9)	(63.1)
Total Capital	$...M	(479.8)	(332.7)	(547.2)	(379.5)
Undiscounted Cashflow Pre-Tax	$...M	2,574.6	1,785.5	3,590.3	2,489.9
Tax Payable	$...M	(720.7)	(499.8)	(1,026.1)	(711.6)
Undiscounted Cashflow After Tax	$...M	1,853.9	1,285.7	2,564.2	1,778.3
C1 Cost (Ag basis)	$ /oz	27.22	18.88	29.92	20.75
C2 Cost (Ag basis)	$ /oz	36.56	25.35	37.35	25.90
C3 Cost (Ag basis)	$ /oz	39.51	27.40	40.28	27.93
All-in-Sustaining Cost (AISC): Ag Basis	$ /oz	30.64	21.25	34.09	23.64
C1 Margin (Ag Basis)	%	69.1%		66.0%	
C2 Margin (Ag Basis)	%	58.4%		57.5%	
C3 Margin (Ag Basis)	%	55.1%		54.2%	
AISC Margin (Ag Basis)	%	65.2%		61.2%	
Project NPV (Pre-Tax)	$...M	1,660.7	1,151.7	2,012.0	1,395.3
Project NPV (Post Tax)	$...M	1,174.6	814.6	1,414.6	981.0
Project IRR (Pre-Tax)	%	49.0%		49.0%	
Project IRR (Post Tax)	%	40.3%		40.3%	
Project Payback Period from Production Start	Years	2.2		2.2	
Maximum Project Drawdown (Nominal)	$...M	456.2	316.4	456.2	316.4
Breakeven Silver Price	$ /oz	37.08	25.95	37.98	26.59
Silver Price to Achieve 30% After Tax IRR	$ /oz	69.90	48.93	69.87	48.91
Profitability Index	x	3.63 x		4.17 x	
Ratio of NPV to Pre-Production Capital	x	3.64		4.41	

Figure 1.21.2 shows the key project post-tax NPV sensitivities for the Spot Case.

Figure 1.21.2 Spot NPV Sensitivity Analysis



1.22 Adjacent Properties

Silver Mines subsidiary companies, Bowdens and Bowdens Agriculture Pty Ltd, own the majority of land that falls within the mine site boundary as well as those that border it. One privately owned parcel of land falls within the Project area. The Company has an agreement in place with the landowner that allows for mining activities to occur under a commercial agreement.

A registered Native Title Claim (Warrabinga Wiradjuri #7) covers a vast tract of land of more than 14,000 square kilometre (km^2) in multiple LGAs. This claim covers the BSP mine site area. A portion of NSW Crown Land exists within MLA601 where Native Title has not been extinguished. The Company has completed an agreement in relation to this parcel with the registered Native Title Claimants. In addition, the related Section 31 Deed has subsequently been executed by the NSW Minister for Natural Resources on behalf of the State of NSW. This completed the 'Right to Negotiate' process in accordance with Section 31 of the Australian Native Title Act 1993.

Bowdens currently holds 11 exploration licences: EL5920, EL6354, EL8159, EL8160, EL8168, EL8268, EL8403, EL8405, EL8480, EL8682, and EL9580. These ELs are all within the MWRC LGA. The ELs are all 100% owned by the Company and comprise 2,115 km^2 (521,000 acres) of titles covering approximately 80 km of strike along the highly prospective Carboniferous Rylstone Volcanics. Within and immediately adjacent to these tenements are other ELs held by a range of other companies and holders.

1.23 Other Relevant Data and Information

Various risks and opportunities have been identified for the project and are presented in Section 24 of this report. The Company believes all of these opportunities and risks can be satisfactorily addressed in the final design phases of the project.

1.24 Interpretation and Conclusions

The Ore Resource Estimate is compliant with the Australian JORC, and suitable to support the DFS.

The DFS satisfies an AACE International Class 3 engineering design as is required for a DFS.

The metallurgical testwork completed to date provides a robust and internally consistent basis for the process design adopted in the 2026 DFS. Any residual gaps that should be addressed during final detailed engineering are detailed in Section 13.

The process plant design proposed for the Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet consists of unit operations that are well proven in the industry and are based on results from testwork conducted on samples collected from the Bowdens resource.

Waste rock and tailings management infrastructure satisfy Class 3 engineering designs incorporating geochemical and geophysical test results. The design concepts ensure stable landforms with minimal potential for downstream impacts.

The DFS has comprehensively evaluated the Project, which has been determined to be technically feasible and economically robust. The DFS incorporates Class 3 or better engineering designs and cost estimates for all site infrastructure, based on significant field and laboratory test programs.

2.0 INTRODUCTION

This report is styled in the format of a NI 43-101 for familiarity of international readers but is not intended or claimed to be NI 43-101 compliant. Therefore, the typical contents of Section 2 - Introduction are not relevant to this report.

3.0 RELIANCE ON OTHER EXPERTS

The following parties have contributed to the DFS:

- Bowdens – Sections 1, 4, 5, 6, 7, 8, 9, 10, 11, 12, 18.1, 18.2, 18.11, 20, 22, 23, 25.2, 26.2.
- Lycopodium – Sections 2, 3, 13, 17, 18.3, 18.4, 18.9, 18.10, 21.3, 21.4, 21.5, 24, 25.1, 26.1.
- H&SC – Section 14.
- Resolve Mining Solutions – Sections 15, 16, 21.1, 21.2.
- SRK – Sections 18.5, 18.6, 18.7, 18.8.
- Calgacus – Section 19.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Project is located near the village of Lue, approximately 26 km southeast from the regional town of Mudgee, in the MWRC LGA of NSW.

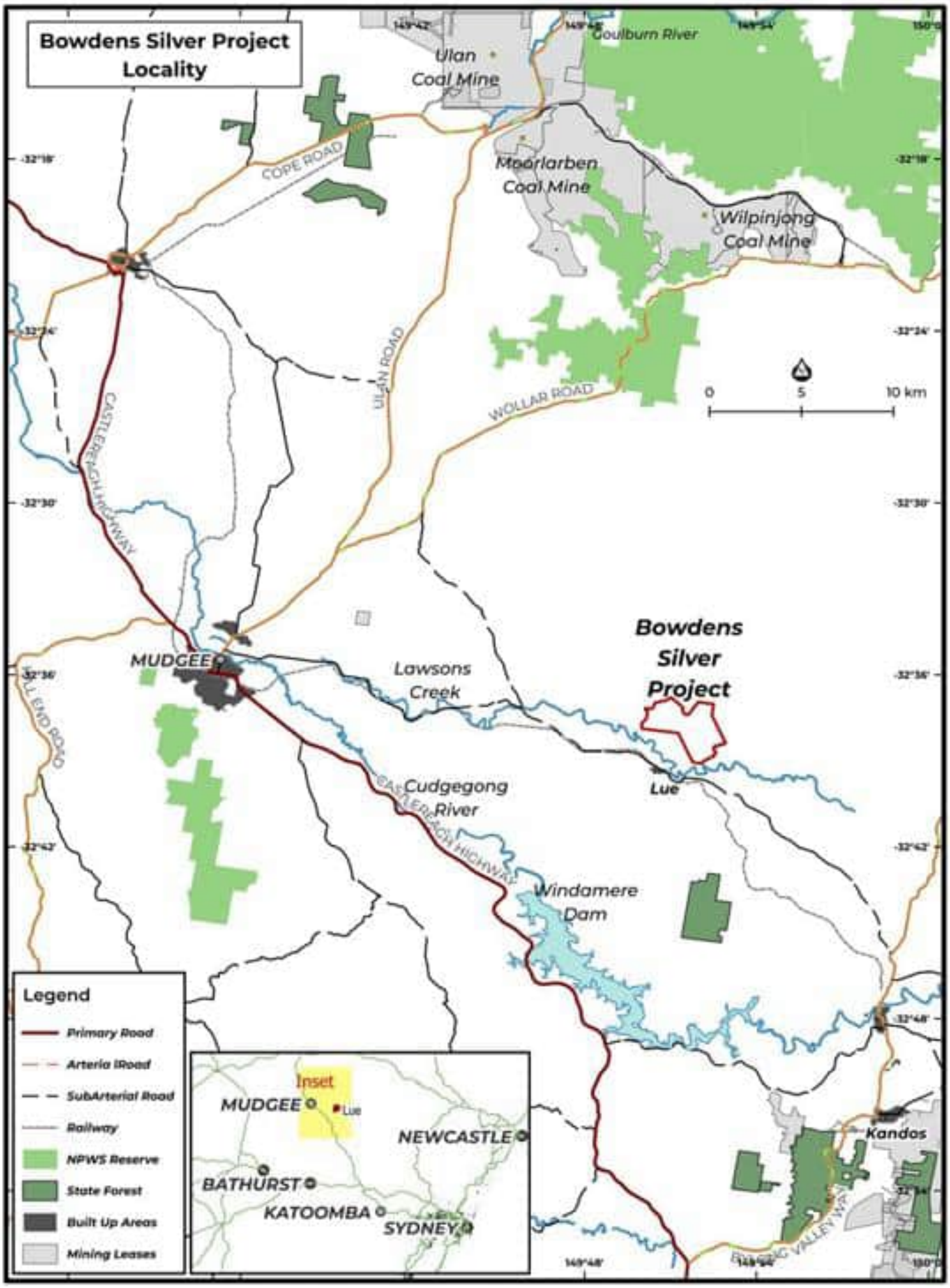
The mine site is located approximately 2.5 km northeast from the village of Lue. A large topographical ridge (locally known as Bingman's Hill) lies between the site and village and provides the benefit of shielding the village from visual aspects, as well as relief from potential noise impacts.

At the last Census (2021), Lue had a population of 216 residents, the majority of whom reside within the Lue village. The Census also identified 105 private dwellings throughout the Lue suburb boundary. The village includes a public school, community facilities such as a Rural Fire Service shed and park / recreation area, a licenced hotel, as well as a limited number of small businesses and small lot residential land parcels.

Nearby the village of Lue, land is predominantly zoned for primary production and is used mainly for larger grazing and cropping enterprises, with interspersed small hobby farms / lifestyle properties. Other land uses within the region include viticulture and olive production, small accommodation venue providers and a motocross track.

Regionally, the Project is located approximately 40 km south of a major coal producing district that hosts the Wilpinjong, Moolarben, and Ulan Coal Mines.

Figure 4.1.1 Bowdens Silver Project Location

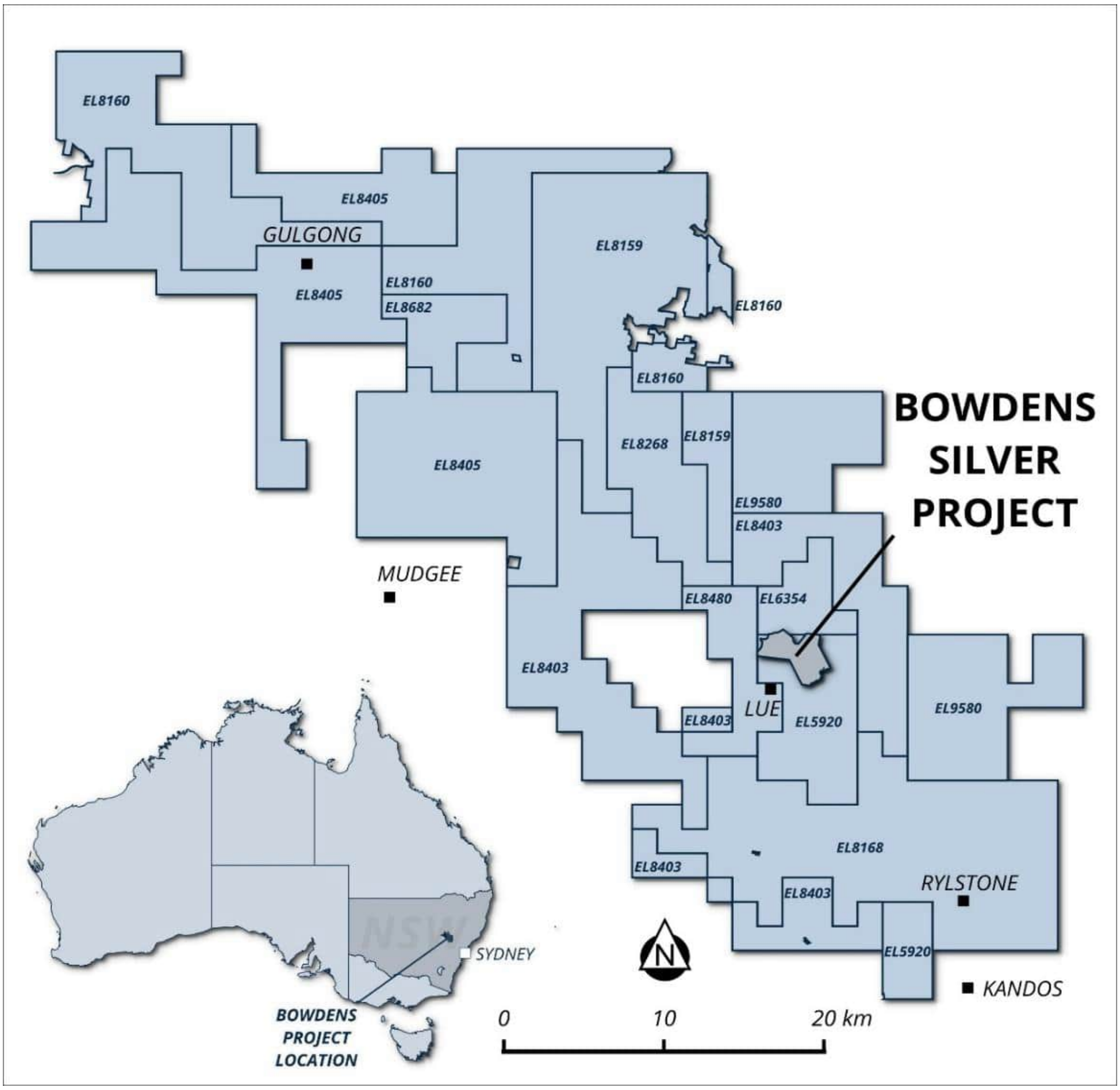


4.2 Ownership

Silver Mines is an Australian based resource company that is publicly listed on the Australian Securities Exchange (ASX:SVL). Its subsidiary company, Bowdens , is the owner of the BSP and currently holds 11 ELs, as shown in Figure 4.2.1, issued by the NSW Government in the wider region that allow for the exploration of Group 1 Metallic Minerals. These are all 100% owned by the Company and comprise 2,115 km^2 (521,000 acres) of titles covering approximately 80 km of strike of the highly prospective Carboniferous Rylstone Volcanics. The deposit sits within Exploration Licence 5920 (EL5920).

In March 2021, the Company submitted its MLA (MLA601) with the then Department of Regional NSW – Mining, Exploration and Geoscience (now NSW Resources) for Group 1 Minerals silver, zinc and lead. MLA601 covers approximately 998.85 hectares (ha) and falls within both EL5920 and EL6354.

Figure 4.2.1 Bowdens Silver Project Leases



4.3 Royalties

The Company is subject to the following royalty obligations:

- A 2.0% Net Smelter Royalty which reduces to 1.0% after the payment of US$5M over tenement EL5920.
- A 1.00% Gross Revenue Royalty over tenement EL5920 (commencing after 20 million ounces (Moz) of silver is produced).
- A 1.00% Gross Revenue Royalty over tenements EL6354, EL8159, EL8160, EL8168, EL8268, EL8480, EL8682, EL8526, EL8973, EL8974, EL8975, EL8403 and EL8405.
- A 1.0% Gross Revenue Royalty over an 85% interest in tenements EL8160, EL8159, EL8168, EL8268, EL5920 and EL6354.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

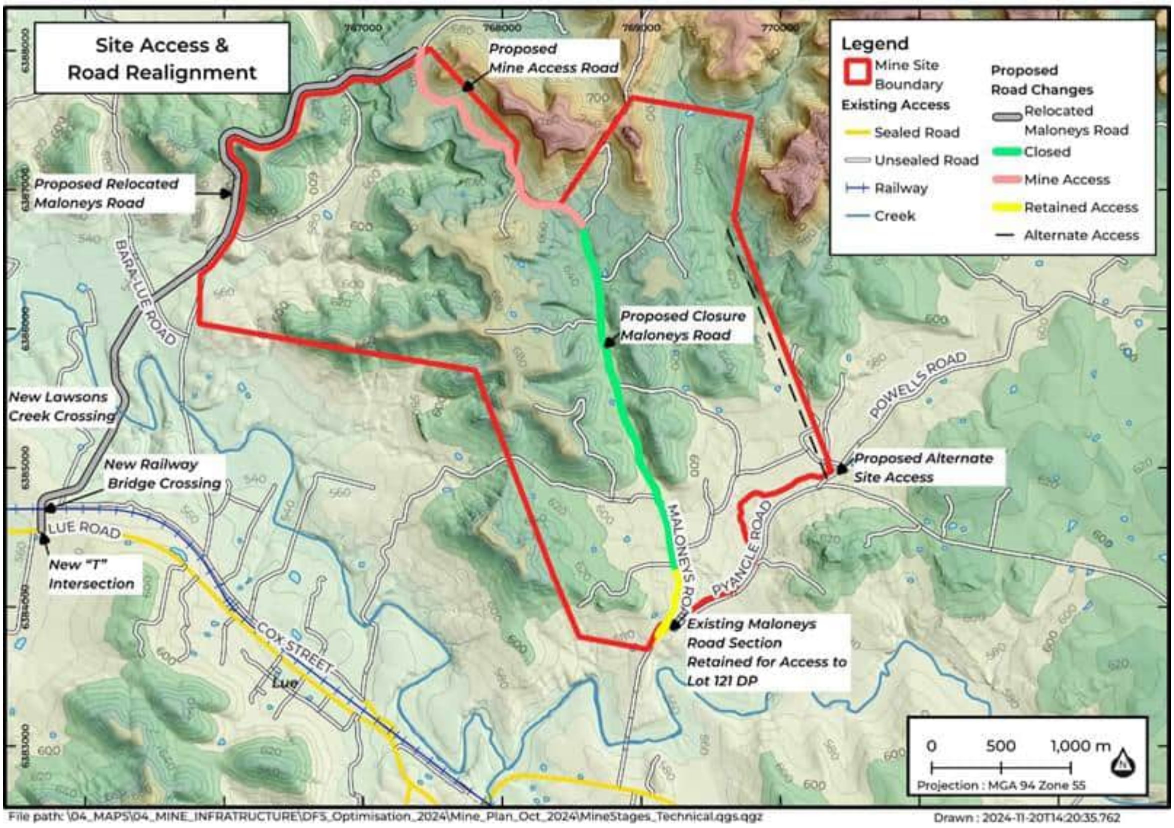
5.1 Accessibility

The main access to the mine site is via Lue Road, which provides a link between Ulan Road, near Mudgee in the west, and Bylong Valley Way near Rylstone in the east. Lue Road is an approved B-double route for GML vehicles up to 25 m long between Ulan Road and Bylong Valley Way.

Site access is currently via Lue Road, which is the main road between Mudgee and Rylstone, and smaller, unsealed sections of Pyangle Road and Maloneys Road. As the existing Maloneys Road bisects the site, a section of Maloneys Road will be realigned to the west of Lue. Approximately 4.5 km of the existing Maloneys Road will be closed and replaced with a re-aligned section approximately 5.3 km long.

The majority of goods transported to and from site will be transported via the state road network through Mudgee. By connecting the new Maloneys Road directly to Lue Road north of Lue, regular mine related heavy traffic through Lue will be eliminated. By Transport for NSW (TfNSW) rules for Lue Road, B-double deliveries and concentrate haulage on Lue Road are restricted to non-school bus hours.

Figure 5.1.1 Site Access



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5.2 Climate and Physiography

Temperature and humidity data sourced from the Mudgee Airport Automatic Weather Station (AWS) shows that January is the warmest month with a mean maximum temperature of 31.0 degrees Celsius (°C) and mean minimum temperature of 16.1°C. July is the coldest month, with a mean maximum temperature of 14.4°C and a mean minimum temperature of 1.1°C.

The lowest average relative humidity generally occurs in the summer months, with January and December sharing the lowest relatively humidity values throughout the year. The highest average relative humidity occurs in June.

Rainfall data were sourced both from Mudgee Airport AWS and the two onsite weather stations. Average annual rainfall at Mudgee Airport AWS is 663.2 millimetres (mm).

Notable differences between the wind speeds and directions can be attributed to the differing topographical features of the site, as winds experienced at the mine site are affected in the following manner:

- A general absence of southeasterly flow at the mine site due to blocking by the elevated terrain immediately to the southeast of the station. A component of the flow is likely channelled into the dominant northeasterly flow currently experienced.
- A general absence of northeasterly flow and a dominance of southeasterly flow at Lue due to winds from the northeastern direction being blocked by elevated terrain immediately to the northeast of the station.

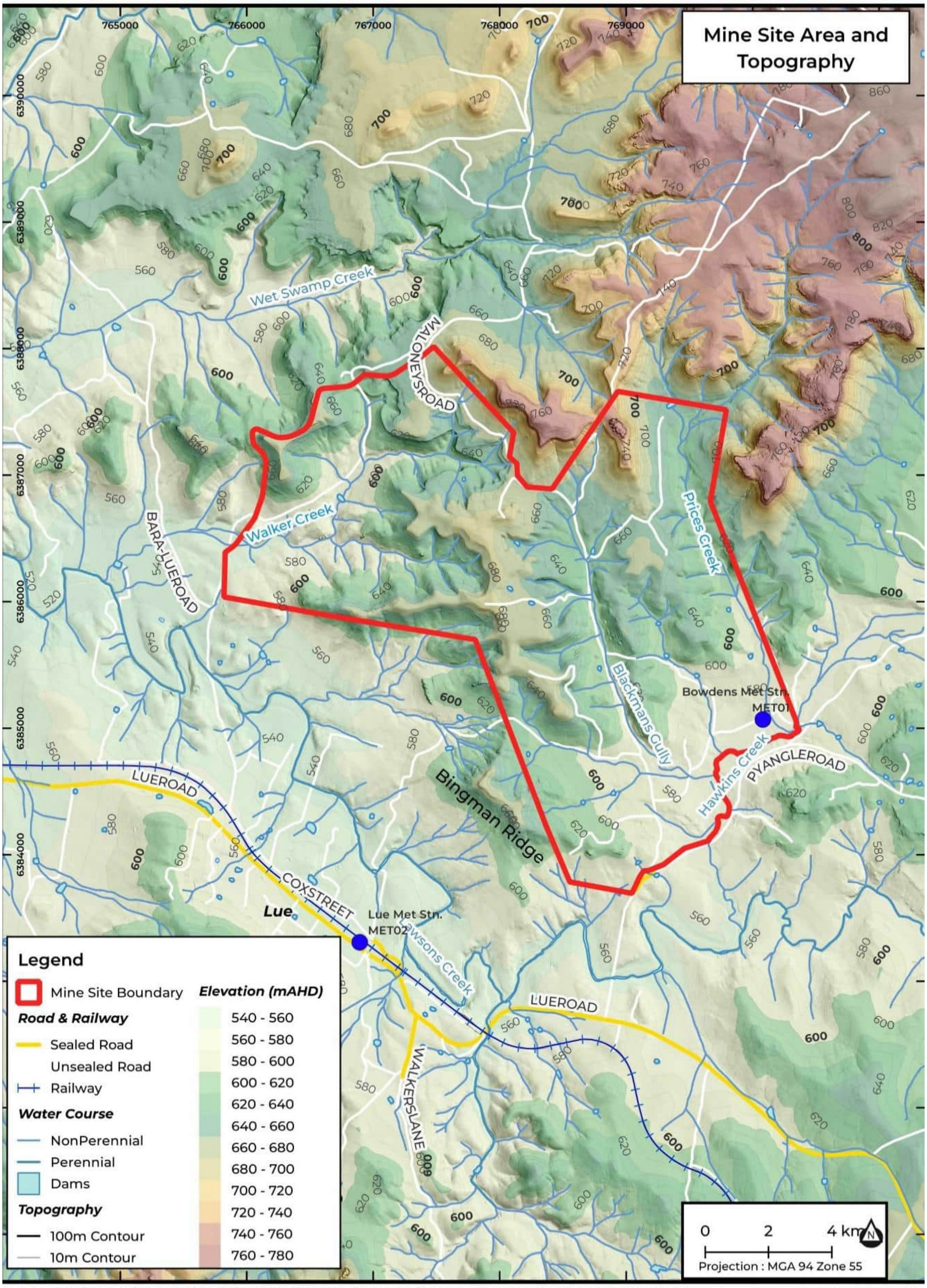
As a consequence, wind blowing from the northeast of the mine site is generally diverted to the northwest before it reaches Lue. As shown by the data, the mine site is located in a valley orientated northeast to southwest and is hence associated with topographical interference. The wind regime at the mine site reflects the topography of the area with the predominant winds blowing from the north, northeast, and southwest directions.

The mine site is situated on the outer western flanks of the Great Dividing Range, which is dominated by elevated rocky ridges separated by either broad and flat or partially confined alluvial valley settings. The location of the mine site is located within a rural area principally used for cattle and sheep grazing, with some cropping and small production of olives and grapes.

The topography generally ranges in elevation from approximately 770 m AHD within peaks and ridges associated with the Great Dividing Range in the northeast, to elevations below 550 m AHD within the alluvial valley of Lawsons Creek to the southeast of the mine site.

Figure 5.2.1 displays the topography within and immediately adjacent to the mine site, with the key ridges and watercourses within the mine site identified. The topography of the eastern and central sections of the mine site and its immediate surrounds is primarily influenced by three north / south orientated ridges with small intermediate valleys, whilst three generally northeast / southwest orientated ridges influence the topography in the western section of the mine site. These ridges contain (from east to west) the confined valleys of the ephemeral Price Creek, an unnamed watercourse, Blackmans Gully and Walkers Creek, plus valleys of minor drainage features.

Figure 5.2.1 Mine Site Area and Topography



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5.3 Local Resources and Infrastructure

BSP is located in an established mining district in the Central Tablelands Region of NSW.

Key nearby offsite infrastructure and facilities include:

- Access to Port Botany or Port of Newcastle via separate road routes.
- Major mining equipment service centres.
- Mining economy based towns with broad range of mining, supply and maintenance services.
- Hospitals and other medical facilities.
- Workplace training and education providers.

5.4 Power and Water

Power supply, distribution and water are covered in Sections 18.2, 18.3 and 18.5, respectively.

5.5 Employment and Accommodation

The peak construction workforce is estimated to be 320 personnel, although some of these personnel will be engaged in offsite activities such as procurement, engineering, drafting, and administration. The instantaneous workforce numbers (both on and offsite) will vary depending on the actual work being performed at any point.

The site establishment and construction workforce will predominately comprise of persons engaged under one the following employment arrangements:

- Employed by the lead contractor engaged to construct the processing plant and primary site infrastructure components engaged from local, state and interstate locations.
- Employed by other contractors or service providers undertaking specific tasks such as clearing, bulk earthworks, waste rock and tailings foundation preparations, road construction, and electrical reticulation engaged primarily from local and state locations.
- Employed directly by Bowdens engaged primarily from local areas such as Kandos, Rylstone, and Mudgee.

Non-local personnel may be accommodated in a variety of facilities including Company owned housing, temporary commercial facilities, and rental agreements.

The operational workforce is estimated to be 220 personnel. While there will be a few specialist contractors required, such as the drill and blast contract, the majority of all operational and technical personnel are planned to be directly employed by Bowdens.

Bowdens is committed to prioritising employment of local personnel, especially those living within daily driving distance of the site. Non-local permanent staff and temporary contractors (projects, shutdown maintenance, etc.) may be accommodated through a number of options, including Company owned housing, long-term commercial accommodation agreements, and short-term hire accommodation. Allocation of Company owned housing will be prioritised for permanent staff with school aged children.

6.0 HISTORY

6.1 Previous Ownership

The original exploration licences covering the 'Bowden' property and a number of surrounding properties were granted in 1989 and 1990 to CRAE. In July 1989, follow-up exploration of an anomalous stream sediment sample, led to the discovery of outcropping sulphide-bearing rocks which assayed 860 parts per million (ppm) silver, 1.0% zinc and 0.5% lead. Between 1989 and 1992, CRAE carried out exploration activities which resulted in the discovery of the Bowdens Gift Zone of outcropping mineralisation approximately 500 m east of the discovery outcrops.

In 1994, CRAE sold the project to GSM Exploration Pty Limited (GSME), a subsidiary of GSM. The original prospect was re-named Main Zone South, and the mineralisation extending north-northwest was named Main Zone North. The Bundarra North and South Zones were later discovered 400 m west of the Main Zone. GSME undertook a scoping study (SS) of 18.8 Mt of resource at 99 g/t Ag including metallurgical testing and preliminary environmental studies.

In 1997, GSM merged with Ashanti Goldfields Limited, and GSME was acquired by Silver Standard Resources Inc. GSME was renamed Silver Standard Australia Pty Limited (Silver Standard). Silver Standard continued exploration activities, with a FS commenced in mid-1998. However, work on the FS was suspended in favour of additional drilling to expand and delineate the mineral resources and ore reserves. The drilling and feasibility work by Silver Standard led to the definition of a Canadian NI 43-101 compliant resource in 2004, completion of a SS and additional metallurgical work. The project was then put on care and maintenance in 2006 while Silver Standard advanced other projects.

In October 2011, Kingsgate completed the purchase of the Project from Silver Standard and created Kingsgate Bowdens Pty Limited. Kingsgate used the existing drilling database to complete a JORC code-compliant resource estimate in late 2011 and again later in November 2012 at the conclusion of the 2012 exploration drilling program. Kingsgate also commenced a FS and environmental impact study that were halted in early 2015.

6.2 Project Purchase

In March 2016, Silver Mines announced it had entered into a Sale and Purchase agreement with Kingsgate whereby it would acquire 85% of BSP. The purchase of BSP by Silver Mines was completed on 29 June 2016. The acquisition of the remaining 15% of the Project was announced on 29 December 2016.

Since owning BSP, Silver Mines and Bowdens have progressed through various Resource and Ore Reserve upgrades, produced an EIS covering multiple environmental, social and economic assessments and continued mineral exploration activities on site and within the wider region.

7.0 GEOLOGICAL SETTING AND MINERALISATION

7.1 Introduction

The Project lies on the eastern margin of the Palaeozoic Lachlan Orogen and the western edge of the Permo-Triassic Sydney Basin, in the Central Tablelands of NSW. The deposit is an intermediate-sulphidation Ag–Pb–Zn (±Au) epithermal system hosted within the Late Carboniferous Rylstone Volcanics and the underlying Late Ordovician Coomber Formation, on the southern margin of an approximate 7 km diameter caldera complex.

This section describes the regional and local geological setting, the lithostratigraphy of the deposit, controlling structures and alteration, and the styles and zonation of mineralisation. It draws on regional mapping (Pemberton et al., 1994; Perry, 1998), deposit-scale studies (Leech 2003, Cranney et al., 2014; Keothammavong, 2018; Lay, 2019; Biggs et al., 2022; Carter, 2023) and the Bowdens internal geological model maintained by the project geological team.

Two references frame the descriptive order that follows: the regional geological setting (Section 7.2) provides the tectonic and stratigraphic context, while the deposit-scale description (Section 7.3 onward) details the host lithologies, structure, alteration and mineralised areas Main Zone, South East, Western, and South West.

7.2 Regional Geology

The Bowdens deposit is in the Central Tablelands of NSW, Australia, on the eastern margin of the Palaeozoic Lachlan Orogen and the western margin of the Permo-Triassic Sydney Basin (Figure 7.2.1). These two geological provinces form the regional framework for the Property and are summarised below.

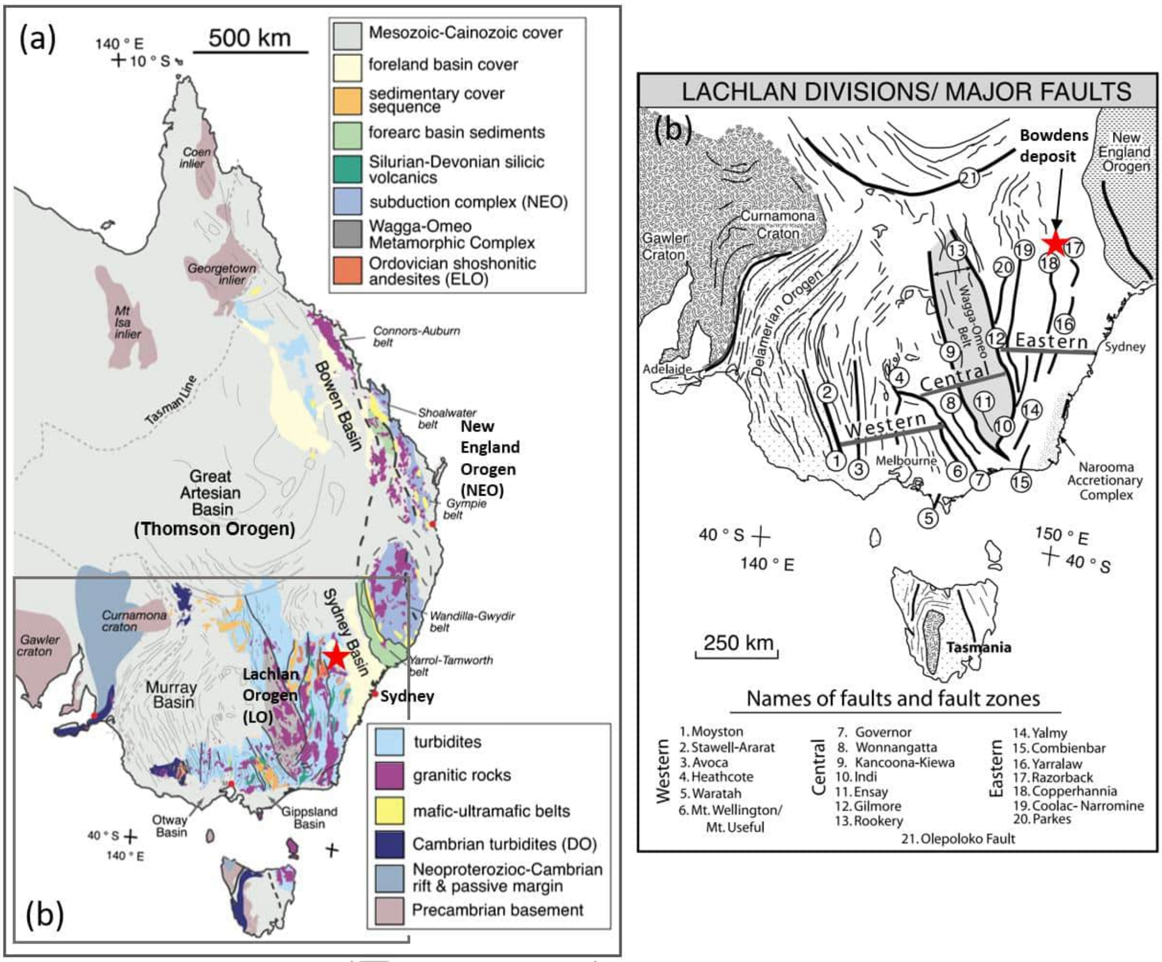
7.2.1 Lachlan Orogen

The Lachlan Orogen, containing the Bowdens Volcanic Complex (shown in Figure 7.2.1), is an approximately 700 km wide, north-south trending belt of deformed Palaeozoic rocks that forms part of the Tasmanides of eastern Australia, a composite Neoproterozoic to Mesozoic accretionary system that comprises roughly one third of the Australian continent (Coney et al., 1990; Gray and Foster, 1997; Foster and Gray, 2000). The Lachlan Orogen shown in Figure 7.2.1 is bounded by:

- The Thomson Orogen to the north.
- The New England Orogen to the northeast.
- The Delamerian Orogen to the west.
- The Sydney–Bowen Basin to the east (Scheibner, 1987).

The orogen is dominated by deformed deep and shallow marine sedimentary sequences, chert, and mafic volcanic rocks. It is interpreted to have developed through stepwise accretion of oceanic sequences, accompanied by Late Ordovician to Devonian shortening and structural thickening, producing crust 35 - 40 km thick (Foster and Gray, 2000).

Figure 7.2.1 Regional Geology



After Gray & Foster 2004.

The Lachlan Orogen is conventionally divided into western, central, and eastern sub-provinces, each distinguished by differences in lithology, metamorphic grade, structural history, and tectonic evolution (Gray, 1997; Gray and Foster, 1997, 2004; Foster and Gray, 2000). The Bowdens deposit lies within the Eastern Sub-province, which is characterised by mafic volcanic and volcaniclastic rocks, carbonates, turbidites, and extensive black shales.

7.2.2 Capertee High

Within the Eastern Sub-province, the deposit sits in the northern part of the Capertee High, an uplifted block within the Capertee Anticlinorium (Carr et al., 2003). The Capertee High preserves a succession of shallow marine to sub-aerial platform sediments of Late Silurian to early Middle Devonian age, together with (Pemberton et al., 1994):

- Late Ordovician basement of the Coomber Formation.
- Late Devonian fluvial to shallow-marine sedimentary rocks of the Lambie Group.
- Middle Carboniferous granitic intrusions.
- Late Carboniferous silicic pyroclastics, lavas, and epiclastics of the Rylstone Volcanics.

The dominant structural grain trends northwest, defined by a series of anticlinal folds of the Capertee Anticlinorium that run parallel to the Hill End Trough (Pringle and Elliot, 1998). To the east of the northern Capertee High, the Palaeozoic stratigraphy is concealed beneath the younger Sydney Basin succession (Pemberton et al., 1994).

The Bowdens deposit is mainly hosted within the Rylstone Volcanics, but mineralisation may extend into the underlying Coomber Formation. The Sydney Basin sediments overlie the volcanic stratigraphy. The stratigraphic column for local geology is presented in Table 7.2.1 and regional geology is shown in Figure 7.2.2, which also displays the large package of fertile Rylstone Volcanics sequences within our lease package. The Bowdens deposit sits on the southern edge of the Bowdens Volcanic Complex (Figure 7.2.3), which also contains several other calderas that offer strong exploration targets.

7.2.3 Sydney Basin

The Sydney Basin forms the southern end of the 1,600 km long Sydney–Gunnedah–Bowen Basin system, a north–south trending rift to foreland basin that developed during the Late Carboniferous to Early Permian in front of the deformed New England Orogen (Glen, 2005; Danis et al., 2011). The basin comprises a generally flat-lying sequence 2 - 4 km thick, approximately 250 km long and 100 km wide, covering approximately 37,000 km^2 onshore and approximately 15,000 km^2 offshore. It developed as a series of north-trending grabens and half-grabens in response to limited east–west crustal extension (Danis et al., 2011).

The basin is divided into western, northern, central, and southern domains. In the area of the Bowdens deposit, the basal stratigraphy belongs to the Shoalhaven Group, dominated by sandstone, siltstone, shale, polymictic conglomerate, claystone, minor tuff, carbonate, and evaporite (Bembrick et al., 1980). Locally, the group is represented by the Snapper Point Formation and the overlying Berry Siltstone (Colquhoun et al., 1997; Perry, 1998; Pringle and Elliot, 1998). The Snapper Point Formation unconformably overlies the Bowdens deposit and forms the immediate cover sequence.

Figure 7.2.2 Regional Geology at Bowdens

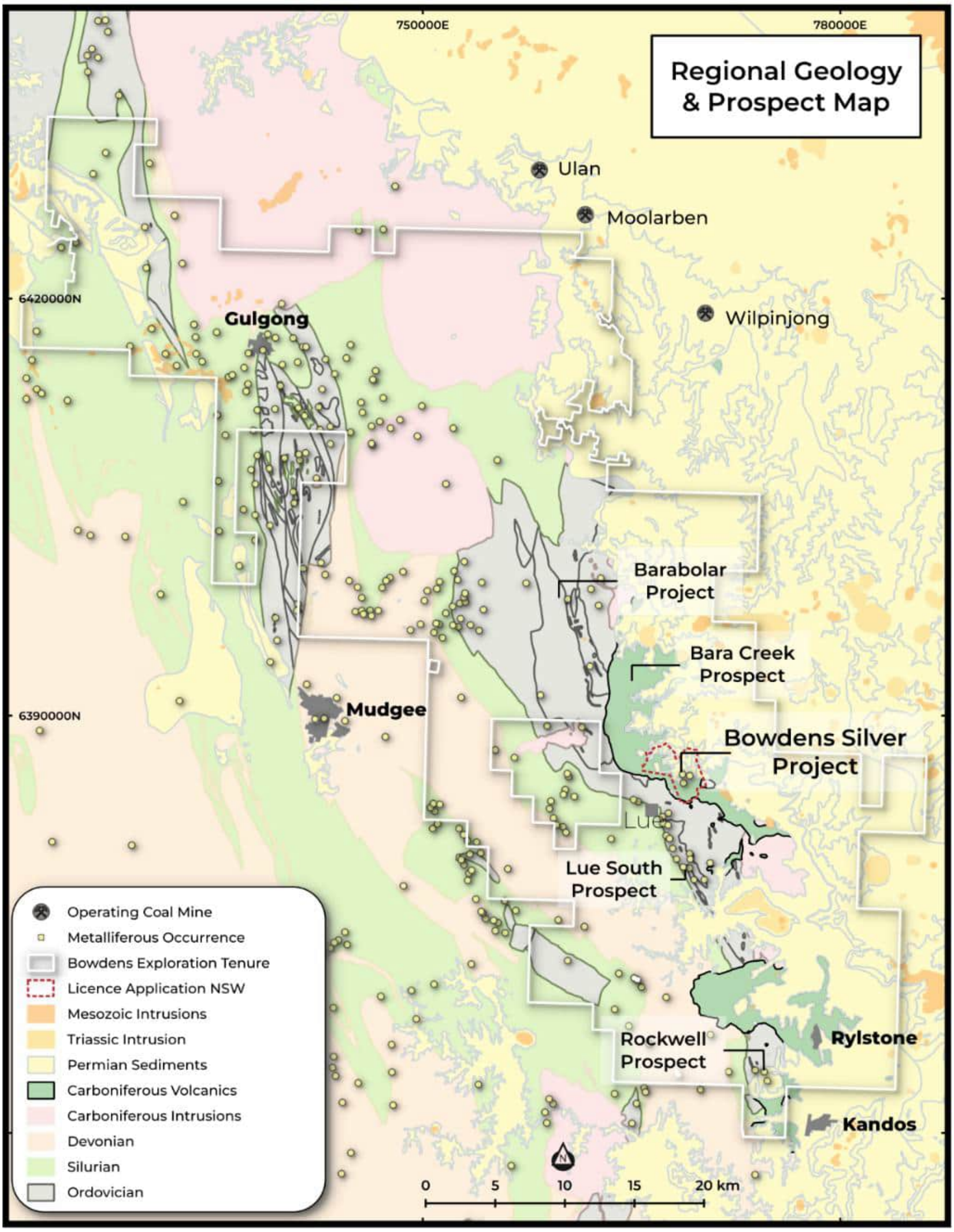


Figure 7.2.3 Bowdens Volcanic Complex

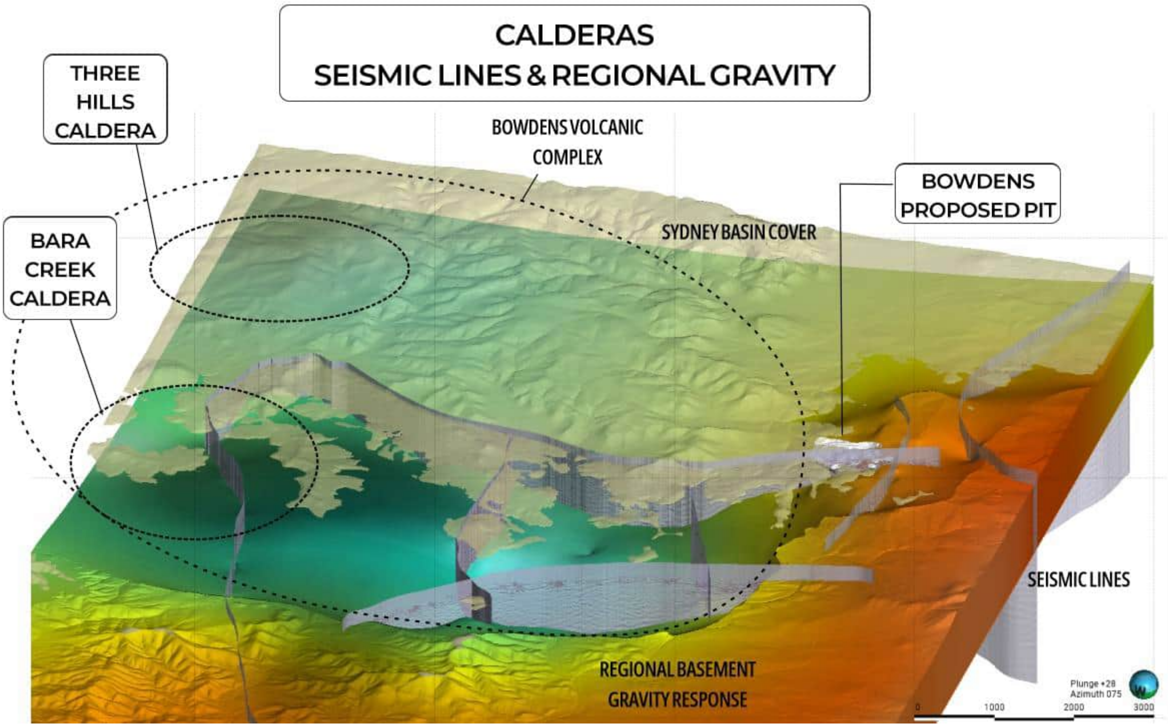
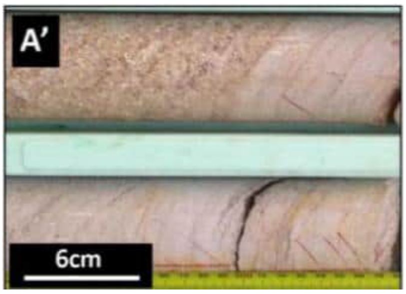
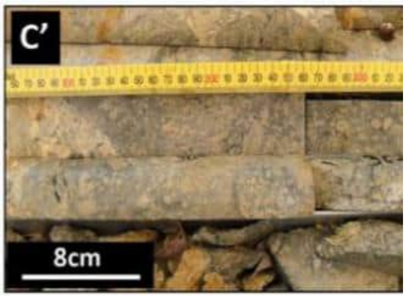
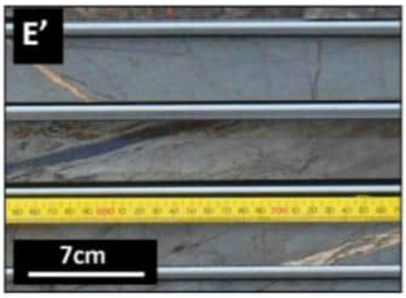


Table 7.2.1 Regional Lithology

Formation	Age	Description	Drill Core Photo
Shoalhaven Group (Snapper Point Formation)	Early Permian to Triassic	Quartz-rich, medium-grained sandstone of the Sydney Basin, unconformably overlying the Rylstone Volcanics. Mineralised clasts have been observed locally within basal conglomerate channels in the northwest of the deposit, demonstrating erosion of the underlying mineralised system prior to basin deposition.	
Rylstone Volcanics	Late Carboniferous 305-309 Ma (Geoscience Australia / CSIRO	A suite of rhyolitic to dacitic pyroclastic, intrusive and epiclastic volcanic succession unconformably overlying the Coomber Formation; host to the Bowdens Ag–Pb–Zn-Au mineralisation. (Langworthy, 1986; Perry, 1998; Pringle and Elliot, 1998).	 
Coomber Formation	Late Ordovician to Early Silurian (Fergusson and Colquhoun, 1996; Meakin and Morgan, 1999)	Fine-grained volcanogenic metasedimentary rocks and intermediate metavolcanic and associated voncalinclastic rocks and minor metadiorite intrusions (Pringle and Elliot, 1998; Keothammavong, 2018). Actinolite–tremolite in the mafic rocks indicates greenschist-facies metamorphism.	

Cautionary Statement: In relation to the disclosure of visual mineralisation in these figures, the Company cautions that visual estimates of mineral abundance should never be considered a proxy or substitute for laboratory analysis where concentrations or grades are the factor of principal economic interest. Visual estimates also potentially provide no information regarding impurities or deleterious physical properties relevant to valuations. The Company notes that these figures are not intended to illustrate potential mineralisation of silver or other metals. No silver or other base metals is visible in these figures. Instead, the purpose of these images is to illustrate the lithology and physical characteristics of the formations including the Shoalhaven Group, Rylstone Volcanics and Coomber Formation.

7.3 Deposit Geology

7.3.1 Deposit Scale Stratigraphy

The Bowdens stratigraphic column comprises of three major formations, tabulated youngest to oldest below with a representative drill core photo in Table 7.2.1.

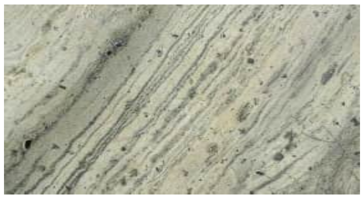
The stratigraphy of the Rylstone Volcanics at Bowdens is interpreted as a proximal intra-caldera silicic volcanic sequence produced during explosive volcanism and subsequent caldera collapse (Perry, 1998; Biggs et al., 2022). Thickness of the volcanic pile increases from tens of metres in the south to over 500 m approximately 4 km north of the deposit. Geophysical evidence, including gravity and aerial magnetics, supports a large caldera complex approximately 7 km in diameter centred north of the deposit, pictured in Figure 7.2.3.

The basal Rylstone succession is dominated by pyroclastic rocks (welded tuffs / ignimbrites, crystal-rich tuffs, autobrecciated and flow-banded rhyolite lavas), overlain by interbedded epiclastic units of low-energy volcaniclastic ash siltstones and higher-energy volcaniclastic and lapilli conglomerates (Langworthy, 1986; Perry, 1998).

Rylstone Volcanics Deposit Scale Sub-Units

Defining lithology consistently in a volcaniclastic environment is difficult because textural features can be the only discriminant and boundaries are commonly gradational; most deposit-scale sub-unit boundaries are therefore supported by immobile-element geochemistry. Rylstone Volcanics sub-units noted and used in geological modelling are in Table 7.3.1, and surface geology presented in Figure 7.3.3.

Table 7.3.1 Rylstone Volcanics Sub-Units

Sub-Unit	Description	Figure of Texture
Pyroclastic Breccias and Tuffs	A coarser lapilli tuff to tuff breccia. A thin (few meters) of silicified ash tuff unit locally occurs in the upper portions of the pyroclastic sequence. Multiple breccia generations are present, ranging from magmatic to hydrothermal, with juvenile clasts indicating syn-depositional brecciation and violent hydraulic fracturing associated with ore-stage events (Carter, 2023).	
Silicified Ash Tuff	Fine-grained ash tuff with intense silicification and chemical affinity to the underlying dacite. Forms a cap to mineralisation in places. Alteration includes pervasive silica flooding of the groundmass, secondary quartz overprinting primary feldspars, and intense argillic alteration with kaolinite replacement of feldspar (Carter, 2023).	
Dacite Breccia	Brecciated dacite pipe and flow, erupted through the underlying crystal tuff dome and onto the palaeosurface. Forms high-permeability zones enriched in ferro-manganoan carbonate, defines depressions in the oxidation profile, and is locally capped by silicified ash tuffs.	

Sub-Unit	Description	Figure of Texture
Dacite Intrusive	Pre-mineralisation dacitic intrusive, central to development of the hydrothermal system. Geochemical signature (Carter, 2023) Zr / TiO_2 consistent with dacitic composition; identified by immobile element ranges Zr 180–210 ppm, Y 20–30 ppm; subtle elevated Cr (>18 ppm); and associated Mn-carbonate alteration halos.	
Welded Tuff (Ignimbrite)	Welded ignimbrite units of varying welding intensity, recording multiple flow events that onlap from the west in the south of the deposit. This overlies the crystal tuff. The crystal fragments are the same as those in the crystal tuff, and glass fragments are more common, in places forming fiamme. These fragments are set in a vitroclastic, locally vesicular, groundmass of volcanic glass that exhibits varying degrees of elongation and flattening. The welded nature of the groundmass in the ignimbrite has resulted in a horizon that has much less primary permeability than the crystal tuff and tuff breccia units.	
Crystal-rich Tuff	The crystal tuff is generally well sorted and comprises abundant feldspar, minor quartz and muscovite, and trace primary iron oxide crystal fragments. Minor lithic clasts (up to 2-4mm in diameter) are most commonly altered glass fragments and rare glass shards. The crystal and minor lithic fragments are set in a very fine altered vitric ash groundmass. The Crystal-rich tuff forms a dome structure likely welding overlying units on its emplacement and is cut by ascending faults. This unit is characterised by a ground mass of feldspar crystals.	

Cautionary Statement: In relation to the disclosure of visual mineralisation in these figures, the Company cautions that visual estimates of mineral abundance should never be considered a proxy or substitute for laboratory analysis where concentrations or grades are the factor of principal economic interest. Visual estimates also potentially provide no information regarding impurities or deleterious physical properties relevant to valuations. The Company notes that these figures are not intended to illustrate potential mineralisation of silver or other metals. No Silver or other base metals is visible in these figures. Instead, the purpose of these images is to illustrate the lithology and physical characteristics of the formations including the Shoalhaven Group, Rylstone Volcanics and Coomber Formation.

7.3.2 Deposit Scale Lithology and Major Structures

Figures 7.3.1 to 7.3.3 present a map and cross sections of the deposit. Mineralisation dominantly sits within crystal and welded tuffs of the Rylstone volcanics and occasionally within the upper pyroclastics. This sequence is uncomfortably overlain by the unmineralised Shoalhaven Group. Several faults cross the deposit, as shown in Figures 7.3.1 to 7.3.3. It has been suggested that the main controls on mineralisation are the Eastern and Bundarra Faults, although debate continues over whether mineralisation is more diffuse and unpredictable on the local scale.

Figure 7.3.2 shows the likely geometry of crystal tuff dome upwelling beneath the (green) welded tuff, emplaced synthetic to and in regionally active faults in the Coomber formation. The Eastern Fault appears to be a strong control on emplacement of the dacites. Evidently the melt evolved from rhyo-dacitic to dacitic in multiple pulses.

Figure 7.3.1 Deposit Lithology and Major Structures

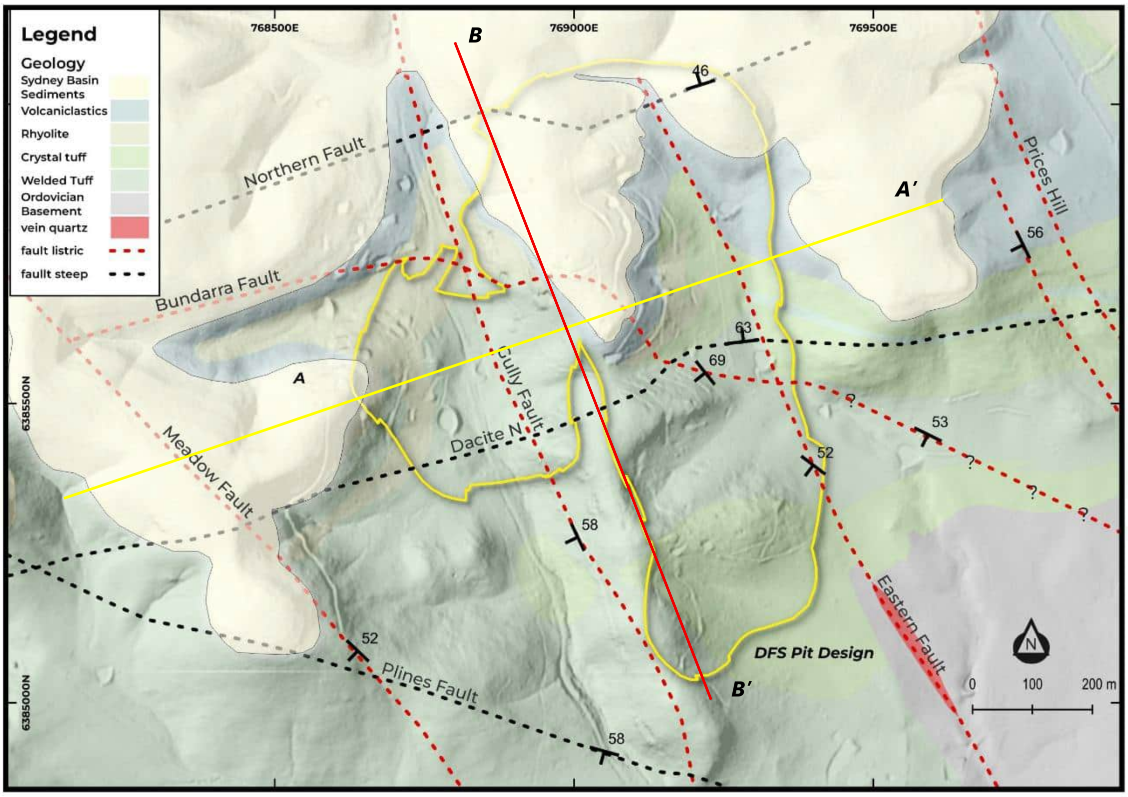


Figure 7.3.2 Bowdens East-West Section A-A′ Looking North

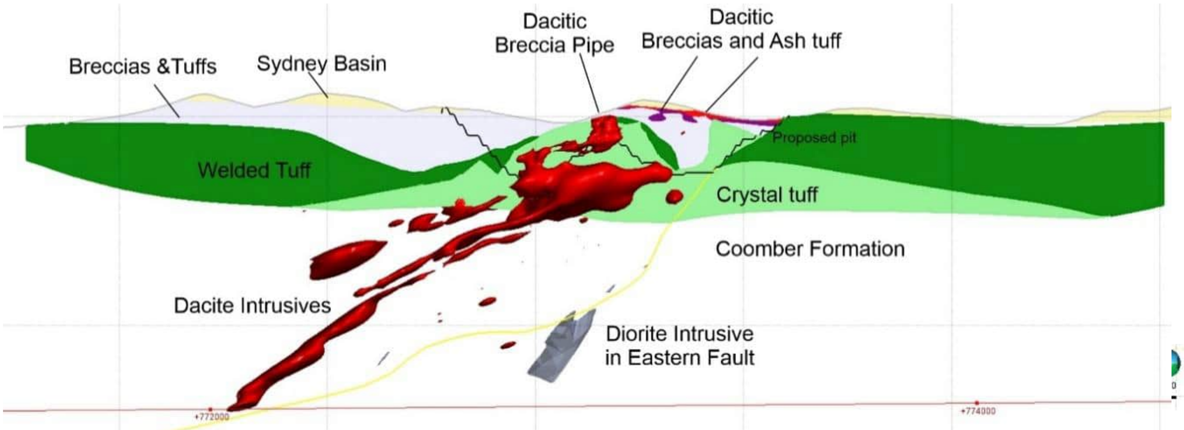
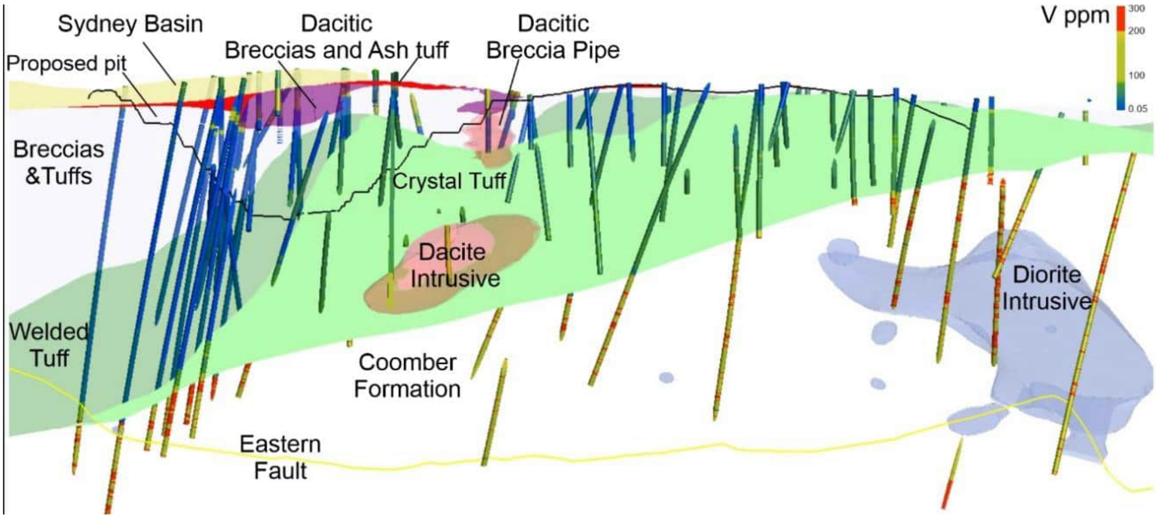


Figure 7.3.3 shows a long section of pit and general stratigraphy highlighting the use of vanadium clearly showing the distinction between the crystal tuff in green and the underlying Coomber Formation and the overlying volcanic sequences.

Figure 7.3.3 Bowdens North-South Section B-B′ Looking East



7.3.3 Structures

North–south and east–west faults are the main conduits for intrusive and hydrothermal activity at Bowdens. The Northern Fault bounds the deposit to the north and dips north, consistent with Caldera collapse to the north of the deposit, whereas the Bundarra Fault is one of a later, south-dipping fault set. Secondary structural control on vein-network development is provided by numerous west-dipping listric faults. The Gully Fault lies within the deposit, while the Eastern Fault often bounds mineralisation to the east. Comparable listric structures include the Meadow Fault, 600 m west of the deposit, which hosts base-metal mineralisation, and Prices Gully Fault, 1 km to the east, which has localised silver mineralisation at depth. See Figure 7.3.4.

The Eastern Fault and Prices Gully Fault are characterised by strong deformation, sericite–clay alteration and silica flooding, whereas the Gully and Bundarra faults are marked by fault gouge and clay-rich milling. Minor-offset east–west faults also cross the deposit area and include shallow splays linked to the major listric structures. The structural setting is considered analogous to the Round Mountain deposit in Nevada, where mineralisation formed in an upward-fanning network of conjugate low- to moderate-angle normal faults and veinlets, similar to features observed on seismic lines across the Bowdens deposit.

Figure 7.3.4 Deposit East-West Cross Section A-A′ with Faults in Rylstone

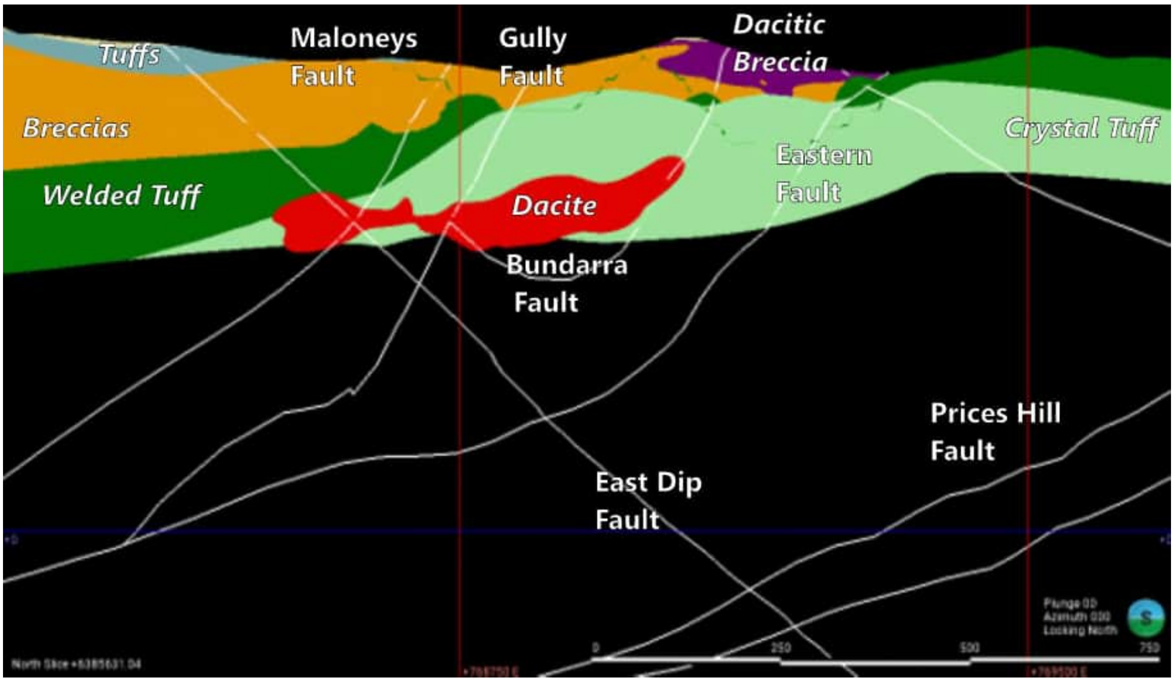


Figure 7.3.4 shows felsic volcanics, and west dipping listric faults, SW dipping Bundarra Fault below the dacite in red and with east dipping conjugate faults acting to localise mineralisation, offsets to the Eastern Fault are estimated approximately 20 m in the proximity of the pit designs.

This interpretation conforms to a predicted pattern of an extensional fault-fracture network within the Rylstone volcanics above reactivated ring fracture margins from multiple late-stage caldera collapse (Rhys et al., 2020).

7.3.4 Deposit Alteration

The Bowdens deposit is an epithermal carbonate–base-metal Ag–Pb–Zn (± Au) system. Initially classed by Leach (2003) as low sulphidation. This deposit style and system-scale paragenesis and alteration zonation was made from drill core petrography and x-ray diffraction (XRD) analysis. Subsequently, Carter (2023) among others, mapped the host lithologies, alteration facies and mineral paragenesis of the high-grade Northwest and Aegean zones using core logging, XRF / µ-XRF, quantitative XRD, Scanning Electron Microscope (SEM), Electron Probe Microanalysis (EPMA) and sulphur-isotope analyses, confirming Leaches early work.

Leach (2003) defined an overall paragenetic sequence at BSP that is initiated by an early phase of fluidised brecciation, followed by a polyphasal hydrothermal event grading from early wall rock replacement and open-space fill of:

quartz (± adularia) → sulphides → carbonate → clay

This sequence is comparable to that described by Corbett and Leach (1998) for low-sulphidation carbonate–base metal–gold systems of the Southwest Pacific and other magmatic arc environments. The sequence is locally repeated in individual samples, indicating a periodically pulsating hydrothermal system and supported from textural observations (Rhys, Lewis and Rowland, 2020) in drill core.

The hotter mineral assemblage reflects a 200 – 250°C upflow regime, and the peripheral assemblage a 100 – 200°C steam-heated, CO_2-rich environment. The intensity of adularia alteration correlates strongly with high-grade Ag intersections. Mn-rich carbonates (rhodochrosite) and Mn-bearing alteration halos are diagnostic of ore-stage mineralisation and accompany silicification of higher permeability horizons; Fe-dolomite, ankerite and siderite dominate the post-ore assemblage. The dacite intrusion records pervasive illite–carbonate alteration with disseminated pyrite, consistent with the broader deposit alteration footprint and indicating that it was emplaced syn- to pre-mineralisation (Graham, 2018).

Alteration Paragenesis

The alteration paragenesis from petrological observations and XRD are presented in Table 7.3.2.

Table 7.3.2 Alteration Paragenesis

Type	Mineral Assemblage	Description
Fluidised / Magmatic Brecciation	quartz + illitic clay + pyrite	Local brecciation of the host volcanics, accompanied by transport, milling and sealing of clasts in a comminuted matrix typically replaced by fine-grained quartz + illitic clay + pyrite. These breccias are most common in BGD 029, where rare basement shale clasts at 90.7 m record significant vertical transport. The fluidised breccias were interpreted to record the exsolution of magmatic gases during the early stages of hydrothermal system and may relate to the pre-mineralisation dacite emplacement.
Quartz-Dominated Events	Quartz + illitic clay ± pyrite, quartz ± illitic clay ± pyrite ± arsenopyrite ± trace sphalerite	Quartz is the dominant mineral in the early hydrothermal events that post-date the fluidised breccias. Early-stage quartz is very fine-grained, locally banded, and intergrown with very fine illitic clay ± pyrite within fractures and crackle to mosaic breccias. Later stages comprise clear quartz ± illitic clay ± pyrite ± arsenopyrite ± trace sphalerite in sheeted to anastomosing vein and veinlet networks. Primary plagioclase and K-feldspar in the wallrock have been initially replaced by secondary K-feldspar that microscopically displays the characteristics of adularia. Trace adularia also locally lines fractures and cavities and, rarely, overgrows depositional quartz.

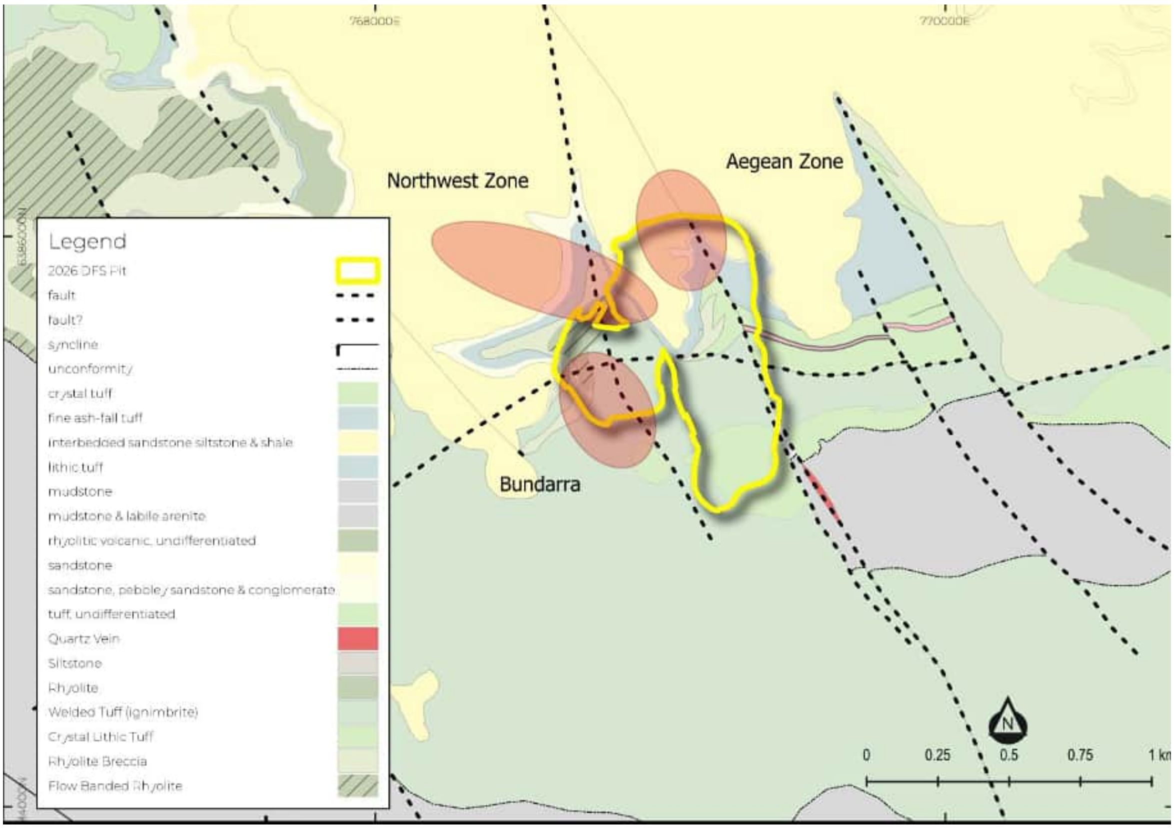
Type	Mineral Assemblage	Description
Sulphide-Dominated Events	pyrite → arsenopyrite → sphalerite → galena → tennantite (± chalcopyrite) → silver minerals	Sulphide abundance increases progressively in the later quartz-dominated events. The overall depositional sequence described initially by Leach followed: pyrite → arsenopyrite → sphalerite → galena → tennantite (± chalcopyrite) → silver minerals, with considerable overlap. Quartz abundance decreases and carbonate increases through the sulphide event. Sulphides occur both as open-space fill (vesicles, fractures, breccia matrix) and as wallrock replacement (most pervasive in volcanic breccia matrix; least developed in welded tuff, where mineralisation is dominantly fracture-controlled). Sphalerite is generally the most abundant base-metal sulphide and commonly displays compositional zonation, grading from Fe-rich (red/orange in thin section) cores to Fe-poor (pale yellow) rims and from earlier to later depositional events. These zonations record progressive cooling of the hydrothermal fluid and are locally polyphasal or reversed, again indicating multiple pulses of hydrothermal activity. Chalcopyrite is rare, both as 'chalcopyrite disease' blebs in sphalerite and as discrete grains, reflecting a copper-poor fluid in the proposed pit areas.
Carbonate-Dominated Events	rhodochrosite ± kutnahorite ± ankerite, mangano-siderite ± siderite	Both quartz- and sulphide-dominated events are intergrown with, and commonly overgrown by, carbonate minerals. Early carbonates are typically manganiferous (rhodochrosite ± kutnahorite ± ankerite) and grade locally to more iron-rich species (mangano-siderite, siderite) in later depositional events (e.g.134.2 m in BGD 001). Galena and tennantite mineralisation continues into the carbonate event. Carbonate becomes progressively finer-grained with time and locally develops fine colloform banding.
Clay-Dominated Events	Illite / sericite, illite-smectite to low-temperature smectite, kaolinite after illite, marcasite + carbonate + clay	Clay minerals are the dominant wallrock-replacement species and are also deposited in trace to rare amounts in fractures and breccias throughout the quartz-, sulphide- and carbonate-dominated events. They commonly fill open space and overgrow all other phases. XRD analyses indicate a range from moderately high-temperature, well-crystalline illite / sericite through poorly crystalline interlayered illite–smectite to low-temperature smectite-rich mixed-layer clays. These have been mapped at scale across the system. Kaolinite overgrows and partially replaces illitic clay in samples from the north-eastern drillholes, where the final stages of galena and silver mineralisation locally extend into late kaolinite. Marcasite is the principal sulphide intergrown with the late carbonate–clay assemblage.

Deeper Alteration Zonation in the Northwest and Aegean Zones

Quantitative XRD work by Carter (2023) defined a clear proximal-to-distal zonation based on proximity to up flow zones. With core temperatures ranging from 200 - 250°C, hosting polyphasal vein breccias transitioning to LS style mineralisation ≈ 100 - 200°C formed in the presence of steam-heated CO_2-rich waters. With evolving alteration assemblages of:

adularia + quartz–Illite → illite–muscovite–carbonate → illite–kaolinite–carbonate ± smectite

Figure 7.3.5 Deposit with the Aegean, Northwest and Bundarra Zones



Distribution and Zonation of Clay Development

These petrological observations are refined, confirmed and extended using Hyperspectral scanning of drill core and drilling chips. Mapping of clay species, carbonate species, Fe-sulphides and sphalerite Fe-content along section lines (Leach, 2003) showed a coherent zonation.

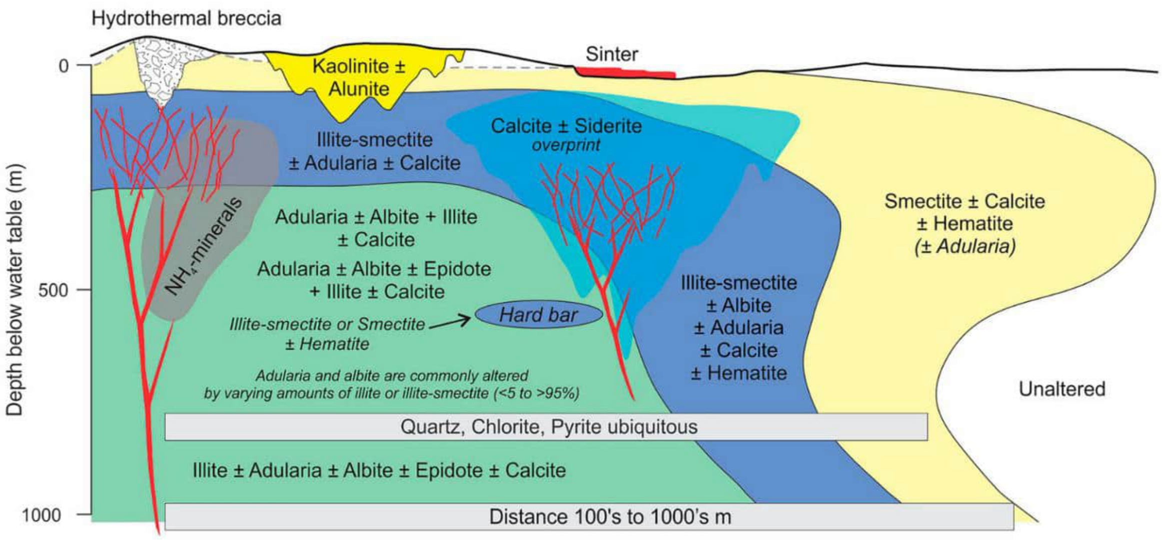
Table 7.3.3 Zonation of Clay Development

Zone	Description
High-temperature Core	crystalline illite / sericite, quartz– adularia and arsenopyrite-bearing assemblages define a roughly N–S high- temperature, broadening to the south and representing the upflow / outflow channel of the system.
Western Margin	Rapid lateral grading to low-temperature smectite–Illite over short distances (e.g. towards BGR 154), indicating a steep thermal gradient and proximity to the cool margin of the system.
Eastern Margin	More gradual grading from interlayered illite–smectite to smectite–illite over >500 m, consistent with an easterly-directed outflow.
Northern and eastern peripheries	Fe-rich carbonate (siderite, mangano-siderite), kaolinite and marcasite, formed from cool, low-pH, CO_2-rich steam-heated waters condensing above and around the margins of the boiling system. Mangano-siderite at deeper levels in the north-west records incursion of these acidic waters beneath a northerly outflow zone.

Observations show Illite / illite–smectite at shallow levels in the Main Zone passing downward into cooler illite–smectite at depth, characteristic of peripheral outflow zones in active geothermal systems. The mixing of deep, near-neutral, high-temperature quartz–adularia–illite / sericite fluids with shallow, cool, low-pH CO_2-rich steam-heated waters drove deposition of iron and base-metal sulphides, a decrease in sulphur activity and pH of the mineralising fluid. This also delivered the subsequent silver mineralisation.

This zonation is directly analogous to LS–IS epithermal systems and modern geothermal analogues (Davatzes & Hickman, 2010; Heap et al., 2020; Rhys et al., 2020) in agreement with Leaches work now more rigorously defined by (Simpson and Christie, 2019). As depicted by the blue areas in Figure 7.3.6, the Bowdens deposit preserves various components such sinter at surface in the west. Beneath the sinter, siderite zones outward to calcite at the north-western periphery, while kaolinite–alunite alteration develops near surface in the southwest. In the main zone, adularia albite and hematite are noted, accompanied by pervasive muscovite-illite alteration and lesser smectite development.

Figure 7.3.6 Alteration in Epithermal System



Likely after Simpson and Christie, 2019.

7.3.5 Mineralisation

Mineralisation styles across the deposit, is controlled by host-rock competence and inferred fluid temperature. Temperature of emplacement falls away from the Bundarra fault and Eastern faults. The emplacement of the dacite and crystal tuff units were close enough to mineralisation that they played a role in elevating local thermal gradients and welding of other units. The brittle, strongly welded tuffs sustained discrete polyphasal vein breccias, while less competent crystal tuffs and basement siltstones favoured matrix-supported veining and replacement in fine micro fractures, often giving the appearance of 'disseminations. Evidence of fluid over pressures and crack seal mechanisms typical of epithermal systems are noted. This includes flat cataclasite horizons and flat lying faults with slicken fibres in fault gouge, often underlying mineralised horizons. Steeply anastomosing veins, fractures and vein networks propagate orthogonally from those mineralised horizons.

The current resource Estimate (Hellman & Schofield, 2026) suggest four possible Resource areas, from northeast to southwest of the system: the **Main Zone** (NE pit), the **South East Zone** (SE pit), the **Western Zone** (NW pit) and the **Bundarra Zone** (beneath the NW and SW pits, within the Coomber basement). The cool distal **Aegean Zone** extends below and beyond the NE pit (Figure 7.3.5). Five mineralisation styles have been identified across the deposit. There is also a minor supergene halide mineralisation zone that has formed at the top of the Rylstone Volcanics where they are not buried beneath Sydney Basin cover. A summary of these styles and typical associations are presented in Table 7.3.4, and the mineralisation styles associated with each zone are presented in Table 7.3.5. Photos of each style are shown in Figures 7.3.7 to 7.3.10.

Table 7.3.4 Economic Mineral Distribution

Metal	Mode of Mineralisation	Description
Silver	Ag-Sulphosalts	The dominant mode in vein- and breccia-hosted ore. The Western Zone is dominated by acanthite (Ag_2S) and polybasite / pearceite ($(Ag,Cu)_{16}(As,Sb)_2S_{11}$). The Aegean Zone is characterised by tetrahedrite-group Ag-sulphosalts (incl. freibergite), pyrargyrite (Ag_3SbS_3), polybasite / pearceite and stephanite (Ag_5SbS_4) that pervasively rim and replace galena (Carter, 2023).
	Ag in galena (lattice substitution)	Coupled substitution of Ag^+ + Sb^{3+} for 2 Pb^{2+} allows up to ~7,800 ppm Ag in individual galena grains; LA-ICP-MS analyses returned typical averages of 2,000–3,300 ppm Ag with 843–2,274 ppm Sb (Cranney et al., 2014). This is a significant contributor to Ag tonnage in high-Pb, low-sulphosalt domains of the Main and South East Zones.
	Ag bearing micro-veinlets and sulphide grains	(5 – 150 μm) intergrown with, or independent of, base-metal sulphides (Cranney et al., 2014).
	Ag halides	Supergene cerargyrite (AgCl) with subordinate embolite ((Cl,Br)Ag) developed in the shallow oxide profile, reflecting Cl-rich, I-poor weathering fluids (Cranney et al., 2014).
Lead	Galena	Two principal generations: an early, coarse, colloform-banded galena intergrown with sphalerite and pyrite, and a later, finer-grained galena pervasively rimmed and replaced by Ag-sulphosalts (especially in the Aegean Zone). The later galena is the dominant Ag carrier where sulphosalts are subordinate. Most abundant in the Main, South East and Western zones. In the Coomber-hosted Bundarra Zone, galena is more Bi-rich than in the Rylstone-hosted zones, consistent with a higher-temperature, deeper-source signature (Lay, 2019).
Zinc	Honey coloured low-Fe sphalerite	Dominant form in the Rylstone hosted zones, intergrown with galena and pyrite in colloform-banded ore and frequently brecciated and infilled by later galena, pyrite, chalcopyrite and Ag-sulphosalts. A late, uncommon sphalerite generation carries chalcopyrite disease and Ag-sulphosalt infill.
	High-Fe Sphalerite	Coomber hosted sphalerite in the Bundarra zone. Accompanied by abundant chalcopyrite, recording higher-temperature, and deeper roots of the system (Lay, 2019). Massive to sub-massive replacement sphalerite–pyrite–galena–chalcopyrite forms the upper Bundarra lenses synthetic with the dacite intrusion in the hanging wall of the Gully Fault and Bundarra Faults.

Metal	Mode of Mineralisation	Description
Gold	Discrete electrum grains	Present in the Western Zone. Anhedral to subhedral electrum (5 – 50 µm) infilling brecciated early sulphides and rimming later chalcopyrite, Ag-sulphosalts and galena. Electrum is the final ore-stage phase, restricted to the lower half of the Western Zone. The recognition of free-milling electrum has implications for metallurgical recovery beyond what is typically expected of an Ag–Pb–Zn epithermal system.
	Au associated with chalcopyrite quartz-sulfide veins	Present at depth in the Bundarra zone defining the hot roots of the system beneath the SW pit. Au abundance correlates positively with chalcopyrite development, with both increasing toward the west and southwest. Background Cu is 300 ppm in the Coomber Formation against 5 ppm in the Rylstone Volcanics
Other Accessory Metals	Chalcopyrite	Uncommon in the Rylstone-hosted zones but abundant in the Bundarra Zone and at depth, recording the hot, deep roots of the hydrothermal system. Cu shows a positive depth and southwest trend.
	Arsenopyrite and marcasite	Late- and post-ore phases respectively, locally associated with Ag-sulphosalts.
	Bi-bearing phases	Rare matildite-related Ag–Bi sulphosalts at depth (Lay, 2019).
	Manganese	Rhodochrosite and Mn-rich Fe-dolomite are diagnostic ore-stage gangue, spatially associated with the highest-grade Ag horizons.

Table 7.3.5 Bowdens Mineralisation Styles

Mineralisation Styles	Figure	Host Rock	Inferred T °C	Sulphide / Ag Assemblage	Found in Zones
Carbonate stringer + fracture-fill	7.5.2, 7.5.5 b & c	Welded / crystal tuff, HW Eastern Fault	~150	Mn/Fe-carbonate + polybasite/pearceite + tetrahedrite-group; Ag-only	Main, Aegean
Polyphasal colloform breccia veins	7.5.1, 7.5.5 d	Competent welded tuff with adularia–quartz–illite alteration	200 – 250	Acanthite + polybasite + pyrargyrite ± freibergite, sphalerite + galena ± chalcopyrite, late electrum	Western, Main feeders
Quartz–sulphide–carbonate–clay banded veins	7.5.5 e & f	Permeable crystal tuff	150 – 200	Galena (Ag-rich by lattice substitution) + sphalerite + pyrite ± rhodochrosite	Main, South East
Quartz–chalcedony–sulphide crackle / mosaic breccia	7.5.5 a & b	Strongly welded ignimbrite and Breccias	~200 high-flux	Galena + sphalerite + arsenopyrite + marcasite + pyrite	Main, upper Western, SE
Matrix-supported veining, massive and semi-massive sulphides, and manto replacement	7.5.4	Volcanic breccia matrix, lower Rylstone and Coomber siltstones and carbonaceous units, against dacite	250 – 300	Fe-rich sphalerite + Bi-galena + chalcopyrite + minor Au + Ag–Bi sulphosalts	SW Bundarra
Supergene halide		Top (<20 m) of outcropping Rylstone Volcanics	~50	cerargyrite ± embolite	Top (<20 m) of outcropping Rylstone Volcanics

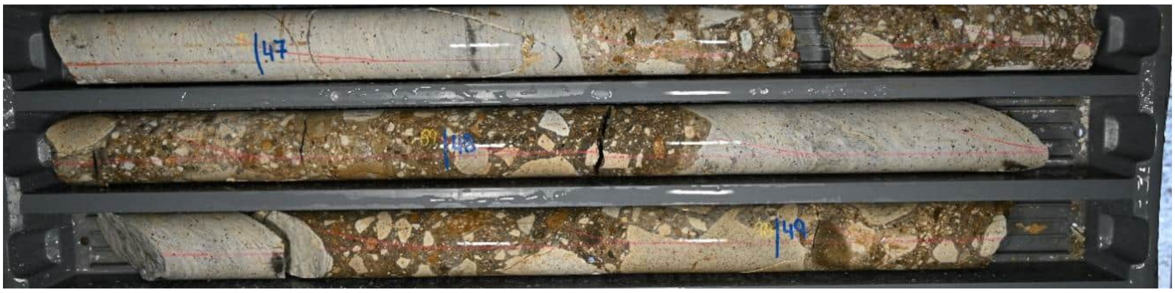
Figure 7.3.7 NQ Core with Steep Polyphase Vein

Polyphase Carbonate vein; hole BD25019; 189.15 m - 189.7 m; 0.55 m @ 644 g/t Ag, 1.70% Pb, 2.55% Zn and 0.2 g/t Au.

Figure 7.3.8 Fracture Fill Vein and Stockwork

Fracture fill vein and stockwork; hole BD25007; 161.0 m - 165.0 m; 4 m @ 2,247 g/t Ag, 2.37% Pb and 1.03% Zn.

Figure 7.3.9 Hydrothermal Breccia BD25004 47 – 48 m



Hydrothermal Breccia; hole BD25004; 47.0 m - 48.0 m; 1 m @ 180 g/t Ag, 0.25% Pb and 0.21% Zn.

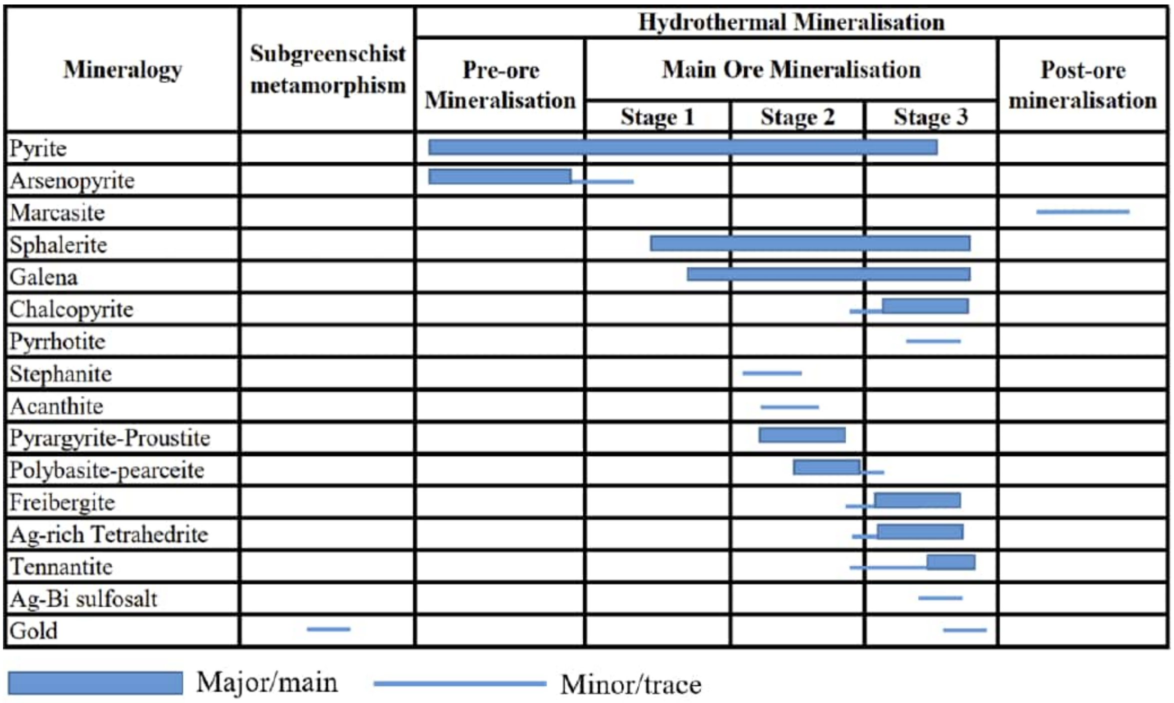
Figure 7.3.10 Bundarra Zone Drill Core

Bundarra Zone; hole BD17004; 373.35 m-374.35 m; 1 m @ 60.8 g/t Ag, 14.20% Zn, 0.48% Pb and 1.4 g/t Au.

Mineralisation Paragenesis

Mineralisation at the Bowdens deposit consists of a pre-ore stage, three main ore mineralisation stages, and minor post ore mineralisation. The distribution of ore minerals through these stages is summarised in Figure 7.3.11.

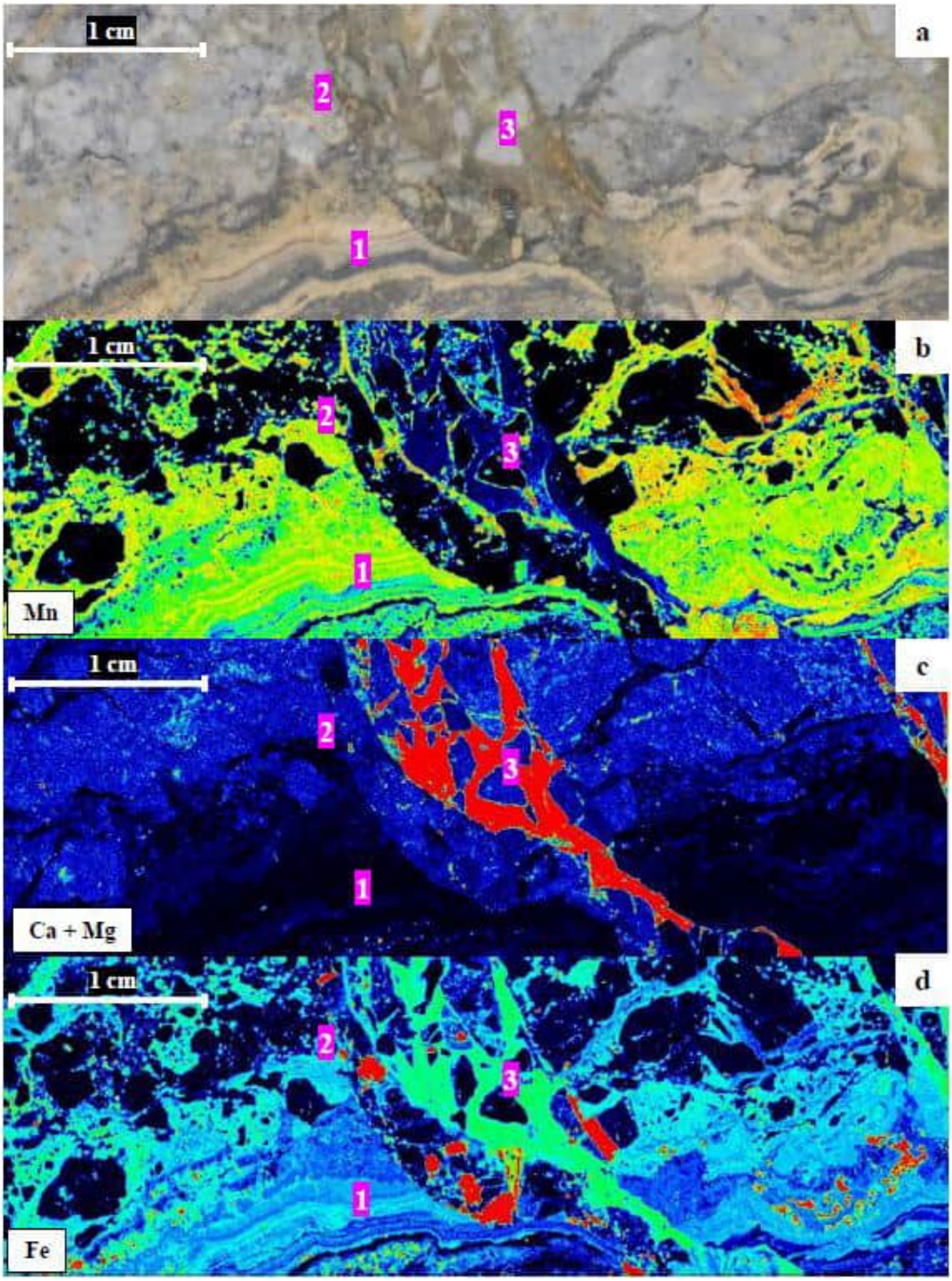
Figure 7.3.11 Mineral Paragenesis (Lay, 2019)



The four stages in the paragenesis are well illustrated in Figure 7.3.12 showing:

- Slide A - the sample photograph.
- Slide B - Mn µ-XRF (Micro -XRF) intensity map highlighting the abundance of Mn-rich carbonates in the early colloform banding (1) and subsequent hydrothermal breccia infill (2), while waning to background levels in the post-ore breccia channel (3).
- Slide C – combined Ca and Mg µ-XRF intensity map, outlining the post-ore breccia channel (3) cutting colloform banding (1) and first hydrothermal brecciation stage (2).
- Slide D – Fe µ-XRF intensity map, Fe is a minor component of the early colloform banded carbonates (1), increasing marginally in concentration within the infilling mineralisation of the first hydrothermal brecciation (2) and being most abundant in carbonates of the post-ore breccia channel (3). (Carter, 2023).

Figure 7.3.12 Micro XRF of Polyphase Vein Texture



Aegean Zone; hole BD21001; 279.0 m - 280.0 m; 1 m @ 1,290 g/t Ag, 4.10% Pb and 2.06% Zn.

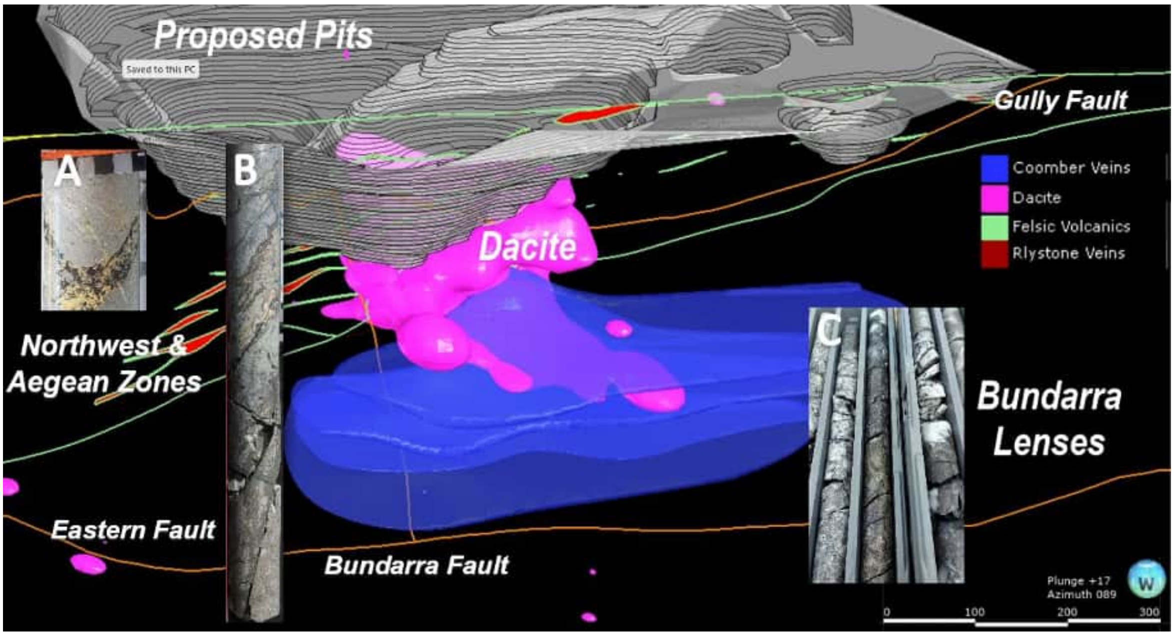
Controls on Mineralisation at BSP

- **Structural** - listric extensional faults and Caldera collapse structures (Eastern, Gully, Bundarra) acted as principal fluid conduits, with high-grade mineralisation localised in their hanging walls and at intersections with rheologically competent host units. Mineralisation typically terminates against the Eastern fault but extends beyond in cross faults and shallow splays to the Eastern fault.
- **Lithological / rheological** - strongly welded crystal and vitric tuffs of the Rylstone Volcanics, particularly where pre-altered to a competent quartz–adularia–illite assemblage, sustained brittle failure under repeated hydraulic fracturing events, generating the discrete polyphasal vein breccias that host the highest grades. Weakly welded crystal tuffs at the base of the Western Zone host more diffuse anastomosing micro-vein networks (Carter, 2023). Additionally, smectite development pre mineralisation may have developed retard porosity capping fluid flows from below.
- **Magmatic / thermal** - the dacite intrusion trapped upwelling hydrothermal fluids and locally focused replacement mineralisation along its hangingwall contacts (Bundarra). Trace-element and chondrite-normalised REE signatures of the dacite are indistinguishable from the host Rylstone Volcanics, indicating co-magmatic emplacement (Graham, 2018).
- **Hydrothermal sealing** - early silica–pyrite and adularia dominant alteration reduced primary permeability and enabled the build-up of fluid overpressure required for subsequent hydraulic fracturing and ore-stage emplacement (Carter, 2023). A pattern characteristic of LS–IS epithermal upflow zones (Simmons & Browne, 2000; John et al., 2018; Heap et al., 2020).
- **Temperature** - the Rylstone-hosted zones reflect a cooler, distal expression (Aegean ≈150°C; Western 200 – 250°C in up flow conduits), whereas the Coomber-hosted Bundarra Zone records the higher-temperature roots, marked by tremolite–actinolite, abundant adularia and chalcopyrite-rich quartz–sulphide veining.

Mineralised Zones Outside Pit

Figures 7.3.5 and 7.3.13 depict the three mineralised zones below the proposed pit in plan view (Figure 7.3.5) and as a 3D model (Figure 7.3.13). Inset A shows BD21003 Aegean Zone acanthite-carbonate vein at 312 m in welded tuff where declining temperatures favour silver dominant mineralisation. The Inset B shows BD20012 Northwest Zone polyphasal hydrothermal brecciation and colloform banded veining. Along strike of the Gully fault at depth from the Northwest zones, the Bundarra Lenses shown in blue show comprise manto style and stratigraphically controlled horizons of mineralisation, developed in carbonate and andesitic tuff stratigraphic horizons within the Coomber formation in close association with the Dacite intrusive and Bundarra fault.

Figure 7.3.13 Mineralised Zones Beyond Pits 3D Model



A) BD21003; 312.0 m - 313.0 m; 1m @ 628 g/t Ag, 0.026% Pb and 0.009% Zn.

B) BD20012; 231.2 m - 232.2 m; 1m @ 211 g/t Ag, 4.77% Pb and 7.40% Zn.

C) BD21042; 297 m – 299 m; 2m @ 269 g/t Ag, 15.80% Zn, 10.33% Pb, 0.78% Cu and 0.42 g/t Au.

7.4 Current Interpretation

The combined alteration and mineralisation patterns indicate that the historically defined Bowdensdeposit (Main Zone) was formed on the northern margin of a larger hydrothermal system (Leach, 2003) and depicted in Figure 7.4.1. High-temperature fluids out-flowed from south of the present prospect area, north along a major structure (Gully fault) on the western side of BSP and northeast along dilatant structures and permeable volcanic units. Cool, low-pH, steam-heated waters permeated the volcanic pile around the northern and eastern margins. Mixing of these two fluid types in dilatant, fracture-permeable hosts (welded tuff, NNW-trending dilational structures between bounding N–S faults; Corbett, 1998) controlled the location and grade of silver–base-metal mineralisation.

The deeper Northwest and Aegean zones (Carter, 2023) show evidence of being a high-grade core up flow zone (Northwest) within rheologically competent, intensely welded tuff at the base of the Rylstone Volcanics. The Northwest zone high-grade lodes cluster around the Gully fault and Bundarra faults, while Aegean Zone mineralisation is bound against the Eastern fault.

The assemblage in the Northwest and Aegean zones (chalcopyrite, abundant tetrahedrite-group minerals, electrum) more closely mirrors the underlying Coomber Formation than the overlying Main Zone (Lay, 2019; Carter, 2023), indicating that these zones represent the vertical and lateral transition between the IS-character Coomber Formation at depth and the LS-character Main Zone above. Combined with similar $\delta^{34}S$ signatures across all zones (Keothammavong, 2018; Carter, 2023). This supports a single, structurally controlled, long-lived hydrothermal system with a deeper, hotter, more metal-proximal source under the Coomber Formation and progressively cooler, LS-character expression upward into the Rylstone Volcanics away from the dacite. The system shows close analogy to the 300 – 700 m sub-surface levels of the active Broadlands geothermal system, New Zealand (Simmons et al., 2000), and to up flow–distal pairings such as Waihi (Coromandel Peninsula, NZ; Hughes & Barker, 2017; Barker et al., 2019).

Figure 7.4.1 Hyperspectral Alteration Temperature Intensity Model

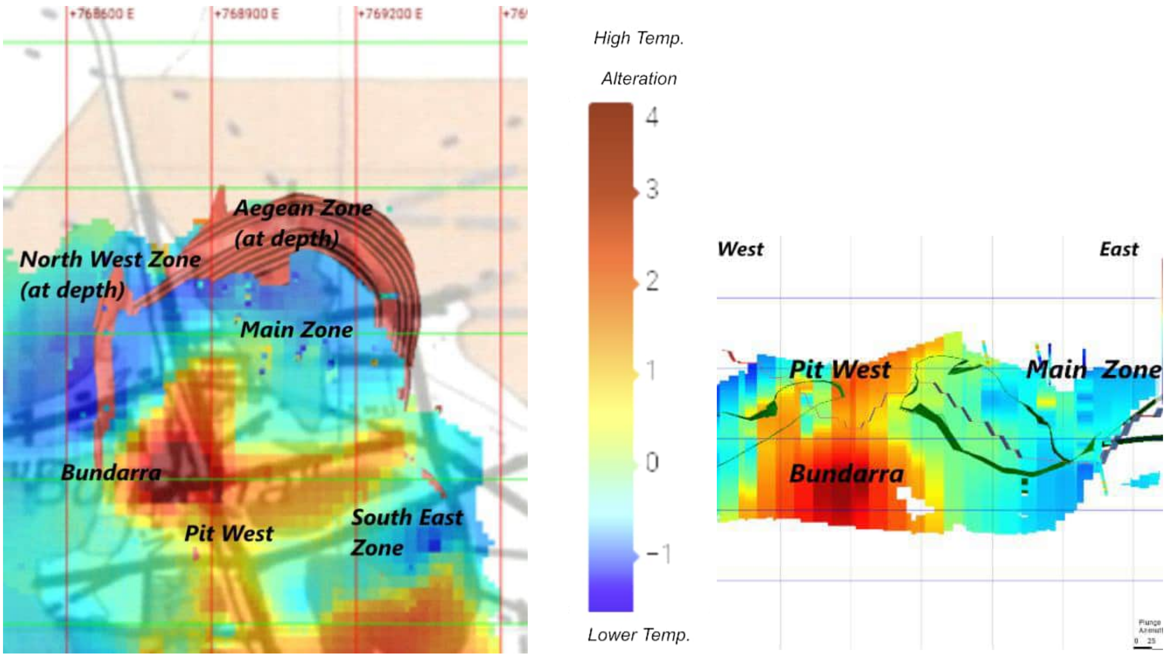


Figure 7.4.1 shows a plan and section of hyperspectral modelling, confirming observed petrological alteration trends surrounding the dacite along mapped faults in felsic volcanics. The east-west section in Figure 7.4.1 shows the mineralised areas with 'alteration intensity' or an approximate map of the temperatures at which minerals formed. The plan shows structural mapping, and the section shows the pit and welded tuff contacts overlying hotter areas.

8.0 DEPOSIT TYPES

8.1 Introduction

The geological setting, host stratigraphy, structural framework, alteration and mineralisation styles that underpin the model adopted at BSP are described in detail in Section 7; this section summarises the deposit-type classification and comparable deposit types.

8.2 Bowdens Deposit Classification

The Bowdens deposit is classified as a low- to intermediate-sulphidation, carbonate–base-metal Ag–Pb–Zn (± Au) epithermal system by multiple authors. It is hosted on the southern margin of an approximate 7 km diameter Late Carboniferous (305 – 309 Ma) caldera complex within the Rylstone Volcanics, with deeper feeder mineralisation extending into the underlying Late Ordovician Coomber Formation basement (Section 7.3).

Diagnostic features supporting this classification (Section 7.4, Section 7.5) include:

Host setting - proximal intra-caldera silicic pyroclastic, intrusive and epiclastic facies (welded tuff, crystal-rich tuff, dacite breccia and a syn- to pre-mineralisation dacite intrusion) over a metasedimentary basement, on the southern margin of a caldera-bounded structural corridor.

Structural control - N–S and E–W listric extensional faults (Gully, Eastern, Bundarra, Northern faults) and conjugate fracture networks acting as fluid conduits, with mineralisation localised at fault intersections and in dilatant flat-lying fracture-permeable horizons (Cox, 2020; Rhys *et al.*, 2020; Rhys, Lewis and Rowland, 2020; Morales-Leal *et al.*, 2022).

Alteration - a quartz (± adularia) → sulphide → carbonate → clay paragenetic sequence, with adularia–quartz–illite/sericite assemblages in the high-temperature core (200 – 250°C), illite–smectite ± kaolinite ± Fe-carbonate ± marcasite assemblages on the cooler steam-heated margins (100 – 200°C), and Mn-rich carbonate (rhodochrosite, kutnahorite, Mn-Fe-dolomite) diagnostic of ore stage.

Mineralisation - polyphasal colloform–crustiform veins, hydrothermal breccias, sheeted/stockwork fracture-fill networks and disseminated to semi-massive replacement; mineral assemblage of pyrite → arsenopyrite → sphalerite → galena → tennantite-tetrahedrite (± chalcopyrite) → Ag-sulphosalts (acanthite, polybasite, freibergite) ± electrum; locally a thin (<20 m) supergene halide zone (cerargyrite ± embolite) where Permian cover is absent.

Metal zonation - vertical and lateral zonation from a cool (approximately 150°C), Ag-sulphosalt-dominated, base-metal-poor distal expression in the Aegean Zone, through Ag–Pb–Zn-dominant Main and South East Zones, to hotter, deeper, Au-Cu-bearing chalcopyrite–galena–sphalerite assemblages in the Western and Bundarra Zones near the dacite and the Coomber Formation contact.

8.2.1 Genetic Model

The deposit is interpreted as the preserved upper to intermediate level of a long-lived, structurally controlled hydrothermal system associated with caldera-margin extensional faulting and a syn- to pre-mineralisation dacite intrusion that drove the thermal anomaly (Leach, 2003; Cranney et al., 2014; Carter, 2023; Section 7.4). Near-neutral, deep, high-temperature quartz–adularia–illite / sericite fluids ascended along listric and ring-fracture-related conduits and mixed in dilatant, fracture-permeable hosts (welded and crystal-rich tuffs) with shallow, cool, low-pH, CO_2-rich steam-heated waters. Mixing and boiling drove deposition of base-metal sulphides followed by Ag-sulphosalt and electrum mineralisation, with progressive cooling recorded by Fe-zoning in sphalerite and the late carbonate–clay assemblages.

The system has been noted to be similar to, the 300 – 700 m sub-surface levels of the active Broadlands–Ohaaki geothermal system, New Zealand (Simmons et al., 2000); the upflow–distal pairing at Waihi, Coromandel Peninsula, New Zealand (Hughes and Barker, 2017; Barker et al., 2019); and they have an upward-fanning conjugate normal-fault and veinlet network architecture of the Round Mountain Au–Ag deposit, Nevada (Section 7.4).

8.2.2 Implications for Exploration

The deposit-type model directly informs the exploration and resource-definition programs (Sections 9 and 10):

Targeting - caldera-margin listric and ring-fracture intersections, dilatational jogs between N–S bounding faults, and the contact between competent welded tuff and underlying basement are the principal target architectures, both within EL 5920 (e.g. Rockwell Prospect and Bara Creek being similar analogues) and along strike in the Rylstone Volcanics in similar volcanic settings.

Vectoring - adularia intensity, Mn-carbonate development, illite crystallinity (hyperspectral), Fe-content of sphalerite, and the spatial distribution of Ag-sulphosalt versus chalcopyrite–electrum assemblages are used as proximity and temperature vectors toward high-grade upflow zones (Northwest, Bundarra) and to discriminate distal cool zones (Aegean).

Depth potential - analogy with carbonate-base-metal (CBM) and CBM-style systems (e.g. Porgera, Kelian) and the recognition of free-milling electrum and Au-Cu-bearing assemblages at depth in the Bundarra Zone support continued testing of the system below current resource limits, particularly beneath the pit and along the Gully and Bundarra fault corridors.

Geometallurgy - the recognised mineralogical zonation (supergene halides, primary Ag-sulphosalts, base-metal sulphides, deeper electrum–chalcopyrite) underpins ore-type domaining for metallurgical and grade-control programs.

8.3 Comparison with Type Low and Intermediate-Sulfidation Deposits

The descriptive framework for low-sulfidation (LS) and intermediate-sulfidation (IS) epithermal deposits adopted here follows the USGS Descriptive Models for Epithermal Gold–Silver Deposits (Vikre et al., 2015, Chapter F, '*Linkages between Volcanotectonic Settings, Ore-Fluid Compositions, and Epithermal Precious Metal Deposits*', 2005). BSP is positioned as an Ag-dominant, IS, carbonate-base-metal–style deposit, transitional toward LS character at shallow levels and toward CBM character at depth.

Table 8.3.1 summarises the form, host architecture, alteration, ore mineralogy and historical metal endowment of BSP against representative LS and IS type-deposits and locally relevant Australian analogues.

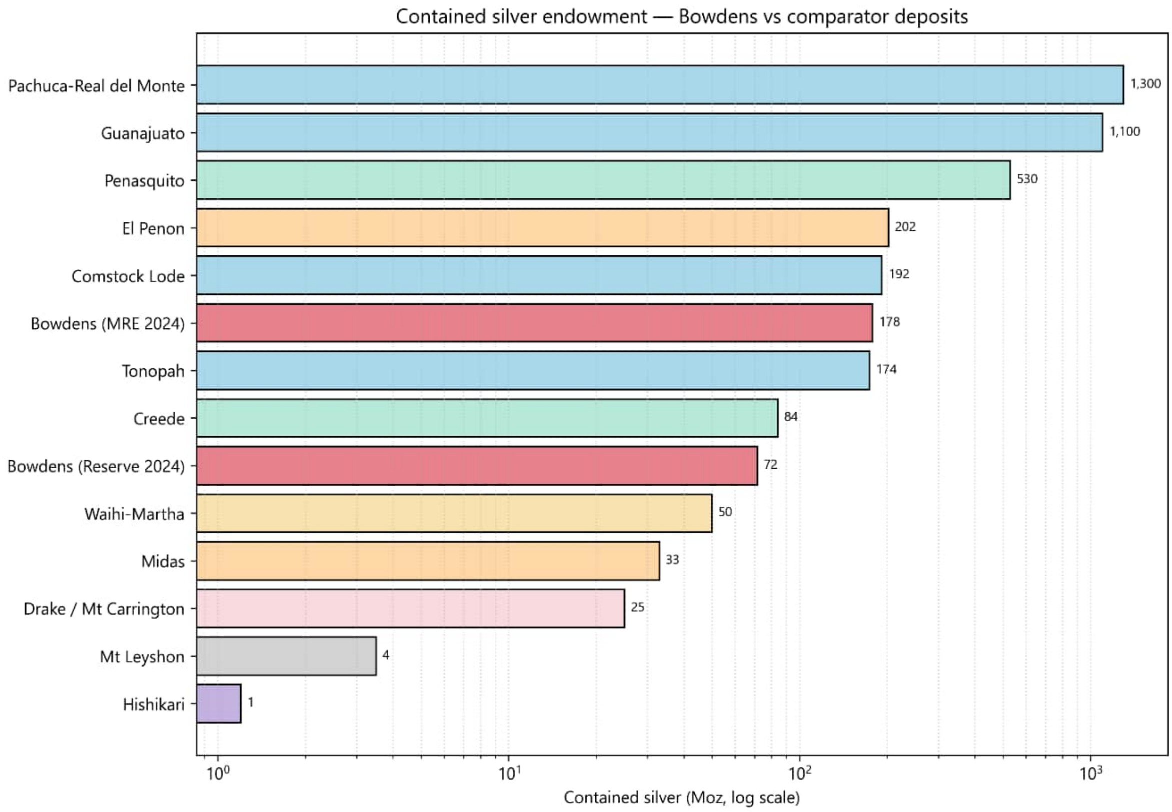
Table 8.3.1 Comparison of Bowdens with LS and IS Type Epithermal Deposits

Attribute	LS Type	IS Type	Bowdens	El Peñón	Creede	Drake / Mt Carrington
Class	LS Au(–Ag)	IS Ag(–Au)	IS Ag–Pb–Zn	LS Au–Ag	IS Ag–Pb–Zn	IS Ag–Au–Pb–Zn
Setting and Age	Extensional arc, caldera margins; Mesozoic–Recent	Continental-arc extensional; Palaeozoic–Recent	Rylstone caldera margin, Lachlan Orogen; ~305 – 309 Ma	Paleocene belt, N Chile; fault-bounded basin; ~52 – 53 Ma	San Juan caldera complex; ~25 Ma	New England Orogen; Late Permian
Form and Host	Sheeted / stockwork veins, sinter; silicic volcaniclastics	Multistage veins, breccias, replacement; andesite–rhyolite	Colloform–crustiform feeder veins, flat-lying stockwork, breccia, disseminated replacement; welded / crystal tuff + metased. basement	Six banded quartz–adularia(–carbonate) veins; rhyolitic dome complex, Paleocene–Eocene volcanics	Sheeted vein-breccias; dacitic ignimbrite	Veins, breccias, stockworks; dacite–rhyolite
Structural Control	Normal / ring faults, dilatant jogs	Extensional / transtensional faults, intersections	N–S & E–W listric normal faults; conjugate fractures; flat permeable horizons	N–S normal + NE/NW transtensional faults	Graben-bounding normal faults	Extensional faults
Alteration and Gangue	Quartz–adularia–illite ± calcite; chalcedony, sinter	Quartz–illite ± adularia; Mn-carbonate ore-stage	Quartz(±adularia) → sulphide → Mn-carbonate (rhodochrosite, kutnahorite) → illite ± kaolinite; Mn-carbonate diagnostic	Quartz–adularia–illite (adularia–sericite); chlorite, calcite distal	Quartz–illite–chlorite; Mn-carbonate ore-stage	Quartz–sericite–illite + carbonate
Ore minerals	Electrum, acanthite, low-Fe sphalerite, minor galena	Electrum, acanthite, sulphosalts, sphalerite, galena ± chalcopyrite	Acanthite, polybasite, freibergite, galena, zoned sphalerite, pyrite, arsenopyrite, marcasite; deep electrum	Electrum, native Au–Ag, acanthite; minor base metals	Acanthite, native Ag, sphalerite, galena, chalcopyrite	Acanthite, sulphosalts, galena, sphalerite
Ag / Au and Metals	Ag / Au <100; Au-dominant	Ag / Au 50 – 10,000; Ag-dominant, Pb–Zn ± Cu	Ag / Au ~thousands; Ag-dominant; Pb+Zn ~0.6 – 1.4%	Ag / Au ~28; Au-rich, base-metal-poor	Ag / Au ~1,200; Pb+Zn 4.3%	Ag / Au ~100 – 300; Pb–Zn appreciable
Grade	5–>10 g/t Au; 10s – 100s g/t Ag	1 – 15 g/t Au; 100s – 1,000s g/t Ag; 1 – 5% Pb+Zn	31 g/t Ag (MRE) / 68 g/t Ag (Reserve); ~0.3% Pb, ~0.4% Zn; intercepts >2,000 g/t Ag	~8 g/t Au, ~125 g/t Ag (reserve avg); bonanza veins higher	1.3 g/t Au, 1,600 g/t Ag, 4.3% Pb+Zn	1 – 3 g/t Au, 100 – 500 g/t Ag, 1 – 3% Pb+Zn
Endowment	Hishikari ~11 Moz Au; Midas 2.4 Moz Au, 33 Moz Ag	Comstock 8.3 Moz Au, 192 Moz Ag; Peñasquito 9.7 Moz Au, 530 Moz Ag	MRE 178 Moz Ag (179 Mt); Reserve 71.7 Moz Ag (32.8 Mt)	7.2 Moz Au + 202 Moz Ag (total endowment)	0.15 Moz Au + 84 Moz Ag + 150 kt Pb–Zn	~25 Moz Ag + 0.3 Moz Au

Sources: Vikre et al. (2015); Sillitoe & Hedenquist (2003); Corbett & Leach (1998); Warren et al. (2008); Arancibia et al. (2006); SVL (2024); Hellman & Schofield (2024); Carter (2023). Full bibliographic detail in Section 26.

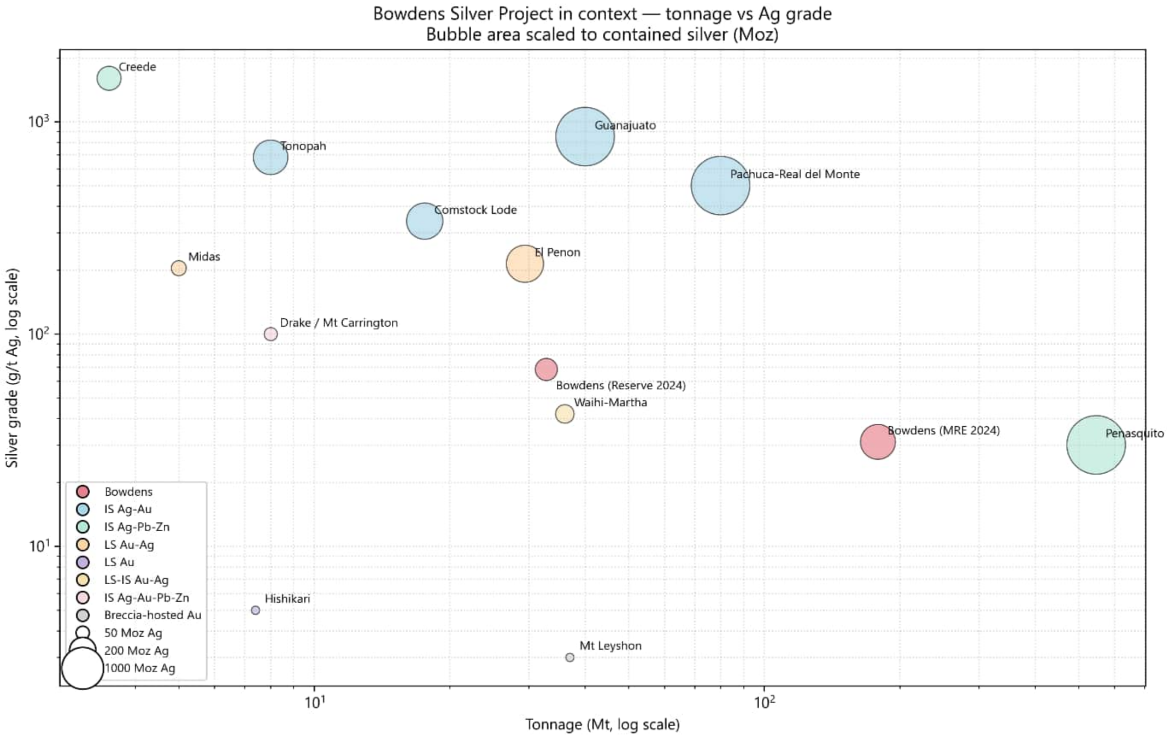
On a contained-silver basis BSP sits among the larger IS Ag-dominant systems globally (cf. Comstock ~192 Moz Ag, Tonopah ~174 Moz Ag, Creede ~84 Moz Ag, Fresnillo / Peñasquito districts; Vikre et al., 2015). The Bowdens 2024 Resource and 2024 Reserve are both shown in Figure 8.3.1 for comparison.

Figure 8.3.1 Deposit Type Comparison



The Resource base of BSP is largely reflecting the scale of the system, with the Reserve being a smaller sub-set accommodating current modifying factors. A comparable grade system of Penasquito, containing similar metal ratios of Ag, Zn and Pb and in production suggests the Bowdens Resource base to be reasonable as similar scale and grade systems can be economically exploited. Various deposits are presented in Figure 8.3.2 showing tonnages versus the silver grade for similar type deposits for comparison.

Figure 8.3.2 Project Scale Global Context

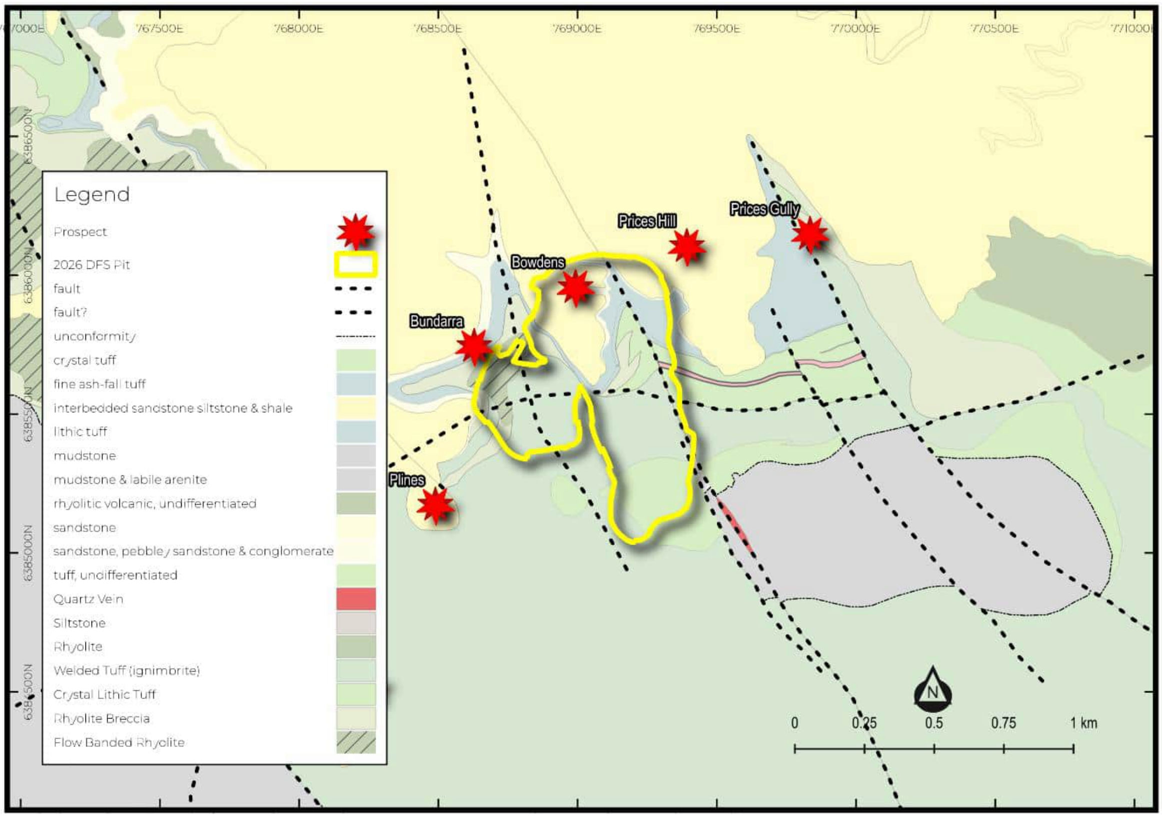


9.0 EXPLORATION

9.1 Introduction

Exploration over the deposit area (excluding drilling, see Section 10), includes desktop studies, field mapping and geochemical surveys (including soil, rock chip, stream sediment, and bulk rock sampling) and geophysical surveys. Exploration targets generated from these datasets include Plines, Bundarra, Prices Hill, and Prices Gully, depicted in Figure 9.1.1.

Figure 9.1.1 Bowdens Deposit with Surface Geology and Nearby Exploration Targets



9.2 Surface Geochemistry

There are 2,437 samples within a 1,000 m radius of the Bowdens deposit centroid (768840 E, 6385228 N, MGA Zone 55). The deposit area has been subject to significant surface geochemical sampling various operators using varying methods.

Tables 9.2.1 to 9.2.3 provide summary of the sampling by the different operators over time.

Table 9.2.1 Rock Chip Samples (Pit Area in ppm)

Operator	N	Ag Median	Ag P_{95}	Ag Max.	Pb Median	Pb P_{95}	Zn Median	Zn P_{95}
CRAE	46	17.0	390.0	1,940	430	3,788	128	3,712
Silver Standard	854	4.7	126.6	980	170	2,694	37	395
Kingsgate Consol.	29	0.2	0.9	1.0	15	340	186	825
Silver Mines	12	0.0	100.0	100	11	2,415	85	969

Table 9.2.2 Soil Samples (Pit Area in ppm)

Operator	N	Ag Median	Ag P_{95}	Ag Max.	Pb Median	Pb P_{95}	Zn Median	Zn P_{95}
CRAE	959	1.0	16.0	164.0	50	1,250	60	472
Silver Standard	204	8.3	32.6	63.3	352	1,481	241	764
Kingsgate Consol.	76	0.3	19.6	49.8	32	949	30	276
Silver Mines	115	0.0	7.5	22.2	17	232	27	158

Table 9.2.3 Surface Au ppm Results (Pit Area in ppm)

Operator	Type	N	Au Median	Au P_{90}	Au P_{95}	Au Max.
CRAE	Rockchip	45	0.0100	0.020	0.030	0.100
CRAE	Soil	954	0.0100	0.030	0.040	0.100
CRAE	Stream	42	0.0050	0.040	0.049	3.650
Silver Standard	Rockhip	495	0.0050	0.010	0.010	0.139
Silver Standard	Stream	29	0.0005	0.001	0.002	0.010
Kingsgate Consol.	Insectmound	14	0.1000	0.200	0.235	0.300
Kingsgate Consol	Rockchip	29	0.0050	0.005	0.005	0.130
Kingsgate Consol	Soil	64	0.0009	0.100	0.100	0.100
Silver Mines	Soil	114	BDL	BDL	BDL	BDL

9.3 Mapping

Surface mapping near the Bowdens deposit has been ongoing since 1990 and refined through several mapping programs and geochemical sampling. Silver Mines has since evolved the interpreted surface geological model via integration of geochemistry and geophysical response at surface to infer geological boundaries at surface, as shown in Figure 9.3.1.

9.4 Geophysical Surveys

The deposit area has been subjected to numerous geophysical survey methods described in detail in Table 9.4.1. The mineralisation response with respect to the geophysical survey methods has been examined in numerous ways. As the economic silver mineralisation is not highly correlated with sulphide content or minerals which provide a definitive and clear geophysical response, no definitive geophysical signature is immediately apparent. In part this is due to low iron and copper content of the system and increasing economic value correlating with a decline in total sulphide content.

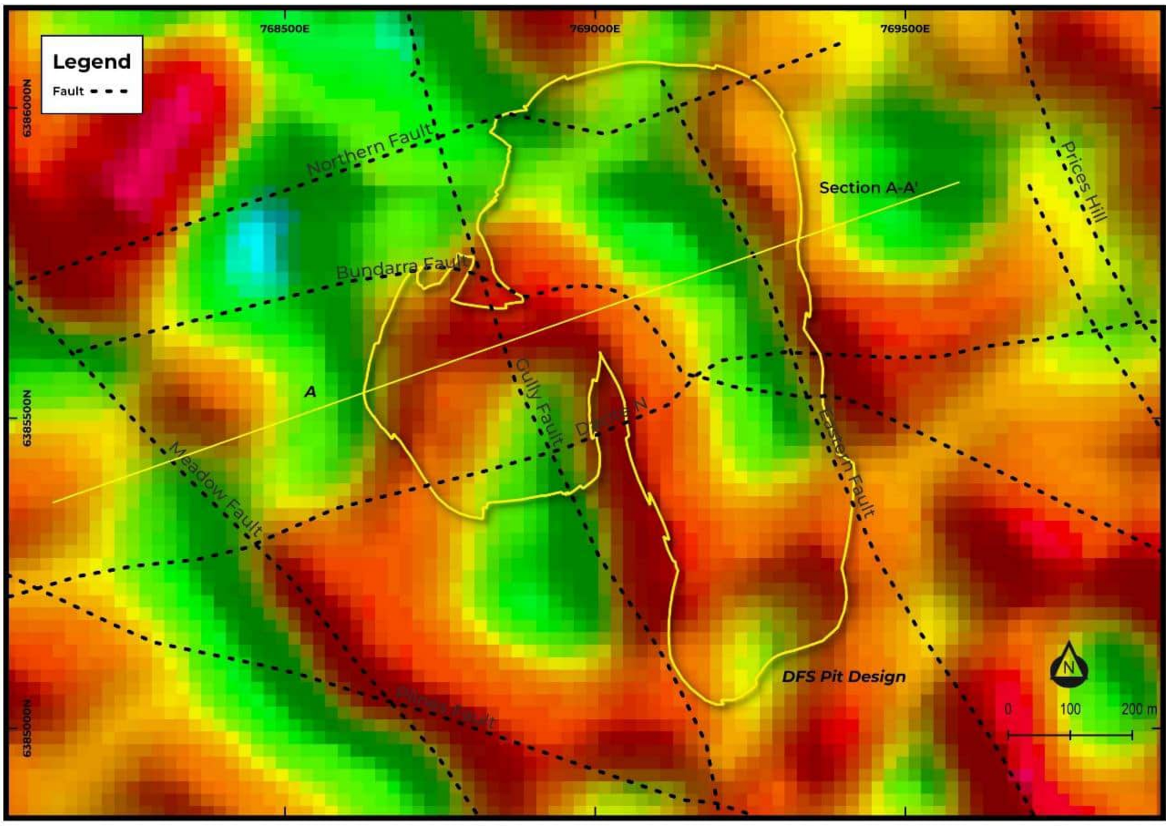
Table 9.4.1 Geophysical Surveys Over the Bowdens Deposit

Activity	Contractor	Survey Spacing	Year	Outputs 2D / 3D
Heliborne Airborne VTEM	GeoTech Ltd	150 m EW lines	2004	2D inversions
Fixed wing Airborne - Magnetics	Fathom Geophysics, Aerosystems	100 m spaced lines flown 090 - 270° with 1,000 m spaced tie lines flown 000 - 180°	2016	Both
Fixed wing Airborne - Radiometrics	Fathom Geophysics, Aerosystems	100 m spaced lines flown 090 - 270° with 1,000 m spaced tie lines flown 000 - 180°	2016	2D
Ground-Gravity	Atlas Geophysics / GeoDiscovery	80 m x 320 m	2020	2D 3D inversions
Ground-Induced Polarization (IP)	Fender Geophysics / GeoDiscovery	200 m spaced Lines using a 2D Array	2017	Both
Seismic Surveying	VelSeis	5 m along line	2022	2D PSDM
Seismic Surveying	Oceania Geo / VelSeis	5 m along line	2023	2D PSDM

9.4.1 Gravity Response

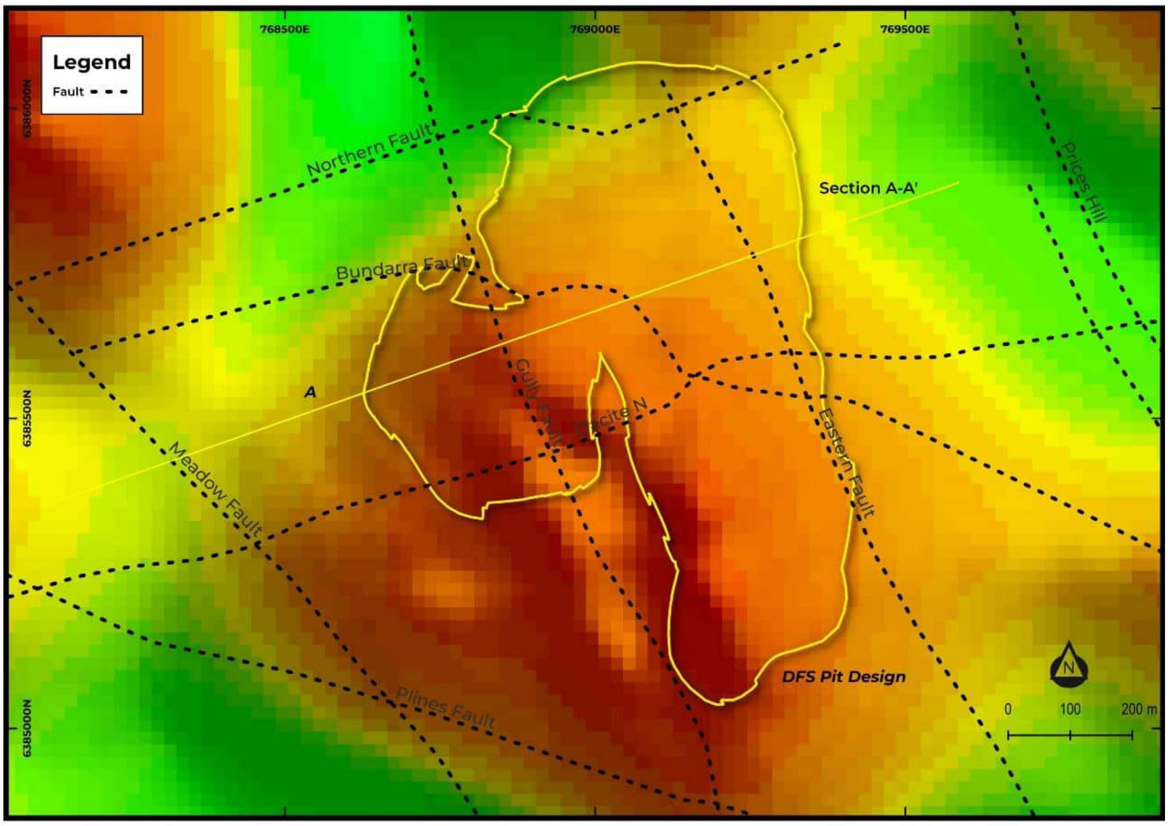
The shallow gravity response of the deposit can be seen in Figure 9.4.1 where low-density permeable areas define the Main Zone (green) and higher density (red / pink) areas highlight less permeable areas of intense silicification or well-formed crystalline intrusive bodies (such as dacite). The pit covers the NW segment of a circular gravity low linked to intense fracturing and alteration.

Figure 9.4.1 Shallow Gravity Response and Structures



A lower pass gravity filter highlighting broader and deeper features is shown in Figure 9.4.2. This highlights a central dense feature with offsets correlated to major faults. Notably it also defines the plan extent of silver greater than 50 g/t intercepted in drilling.

Figure 9.4.2 Deposit Deep Gravity Response and Structures



9.4.2 Induced Polarisation Response

The induced polarisation (IP) response of the deposit was found to correspond to increased levels of pyrite and base metals within the Coomber Formation and Bundarra Zones shown in Figure 9.4.3. IP surveys confirmed structural interpretations of the silicified eastern fault acting as a resistor and broad trends in pyrite content depicted in cross section by Figure 9.4.4. Pits and resource outlines pertain to those proposed from 2017.

Figure 9.4.3 IP Depth Slice Through Bundarra (2017)

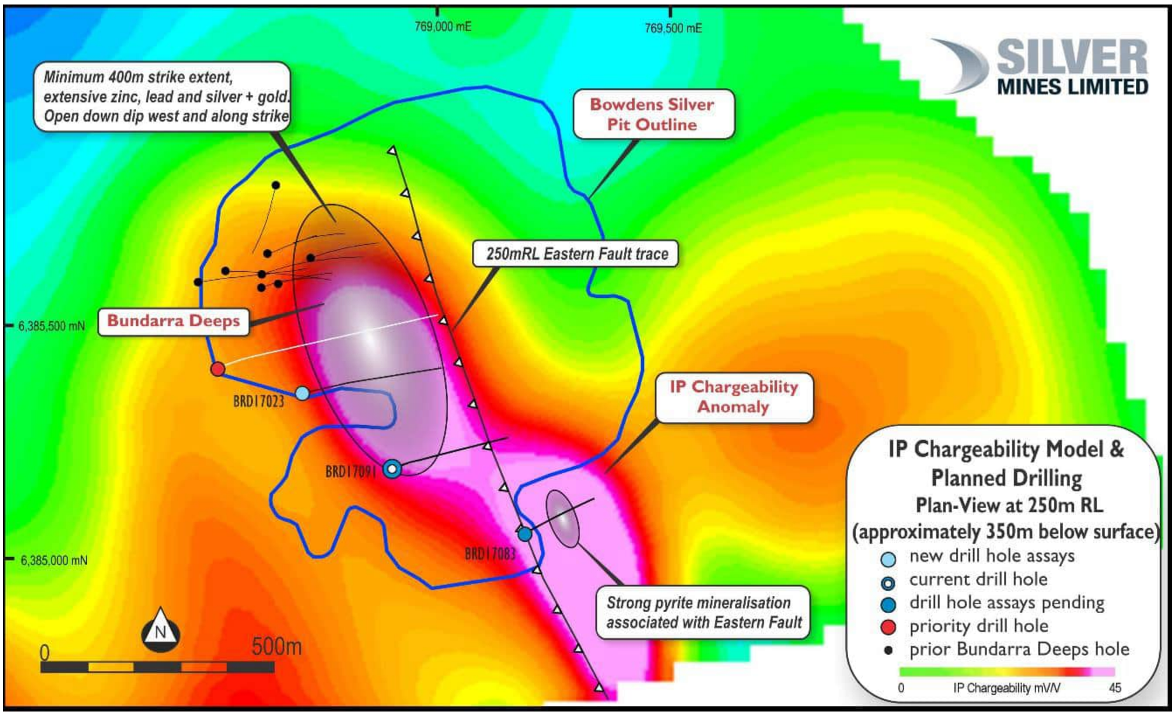
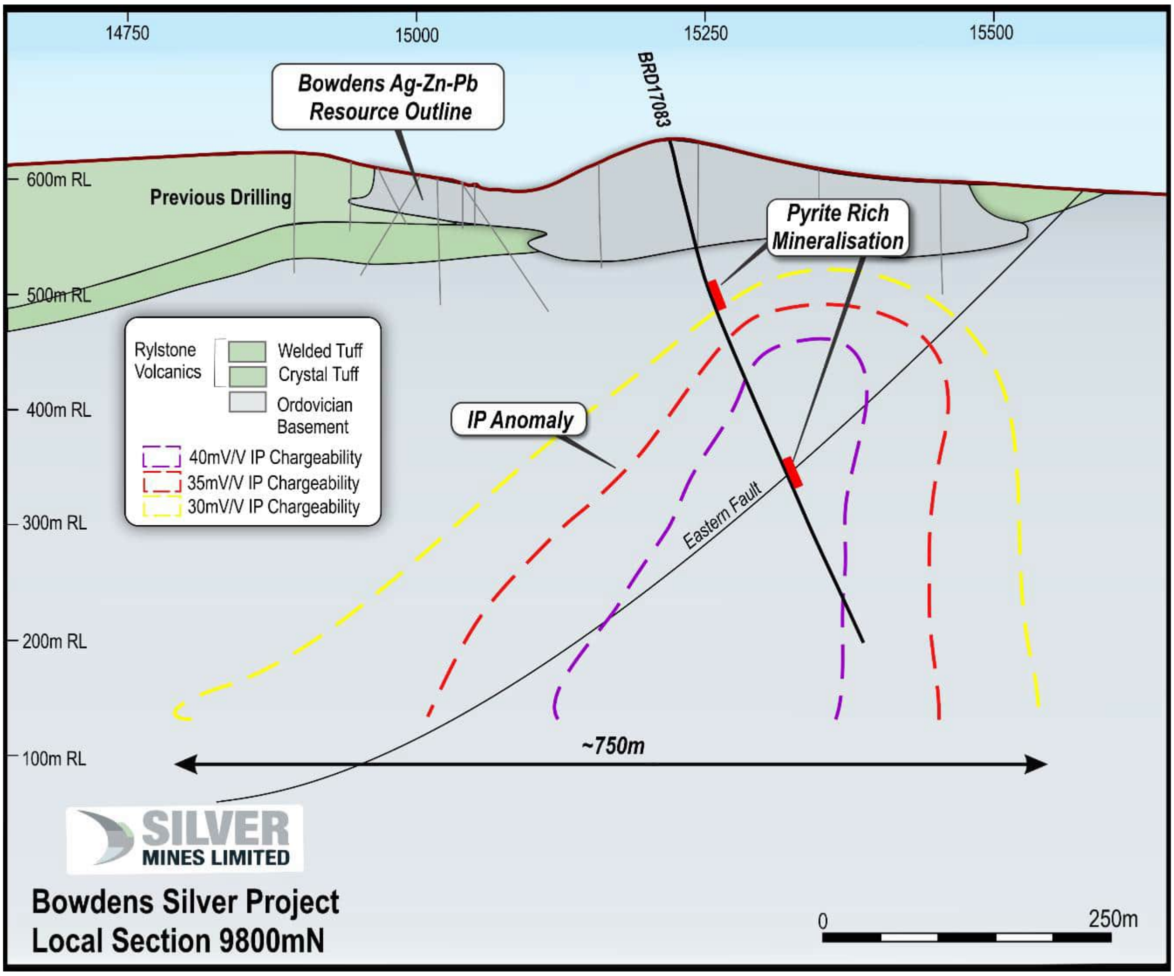


Figure 9.4.4 Cross Section of IP Response 2017

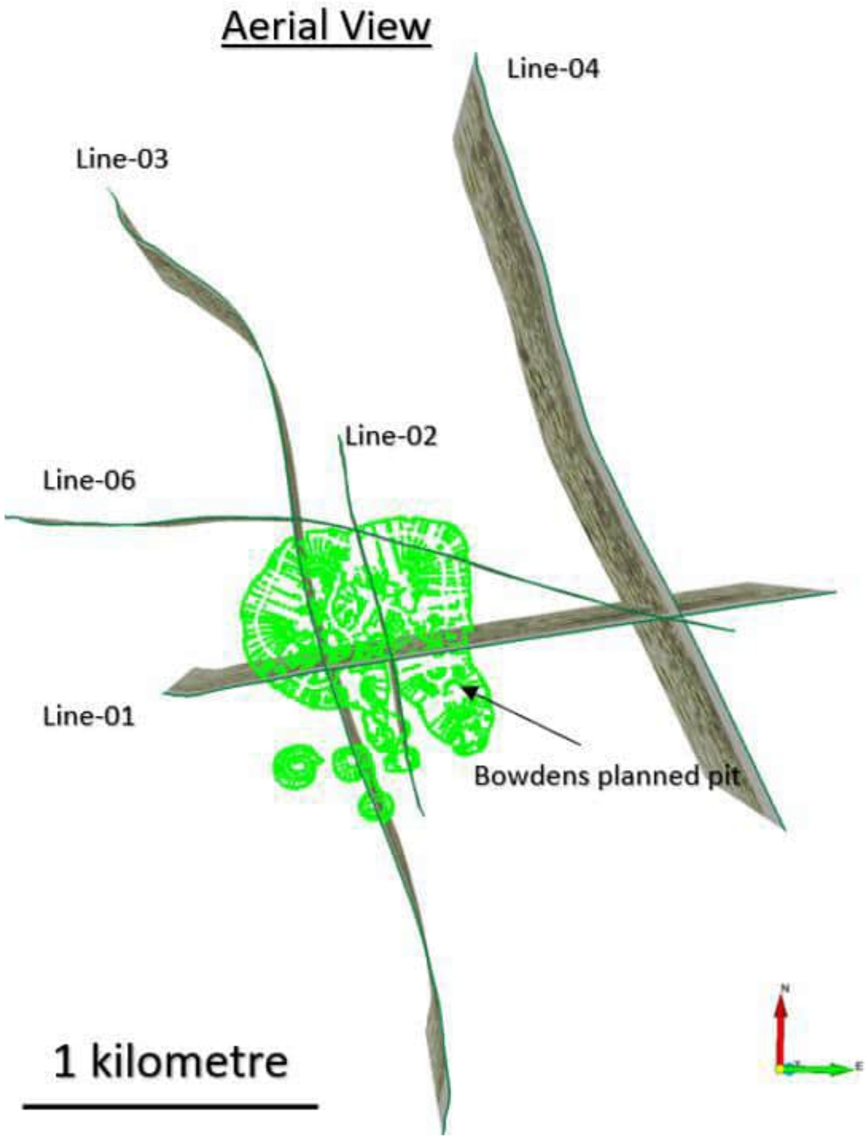


9.4.3 Seismic Response

Two seismic surveys over the deposit were completed after an assessment of drill core velocities was completed. The initial surveys demonstrate that sufficient velocity contrast existed between units of interest and scale of geological features.

The first seismic survey (2022), shown in Figure 9.4.5, was then extended regionally to 100 km across the Rylstone volcanics. Agreement between lines was good and major lithology features could be picked based on established drillhole ties.

Figure 9.4.5 Deposit Seismic Coverage 2022

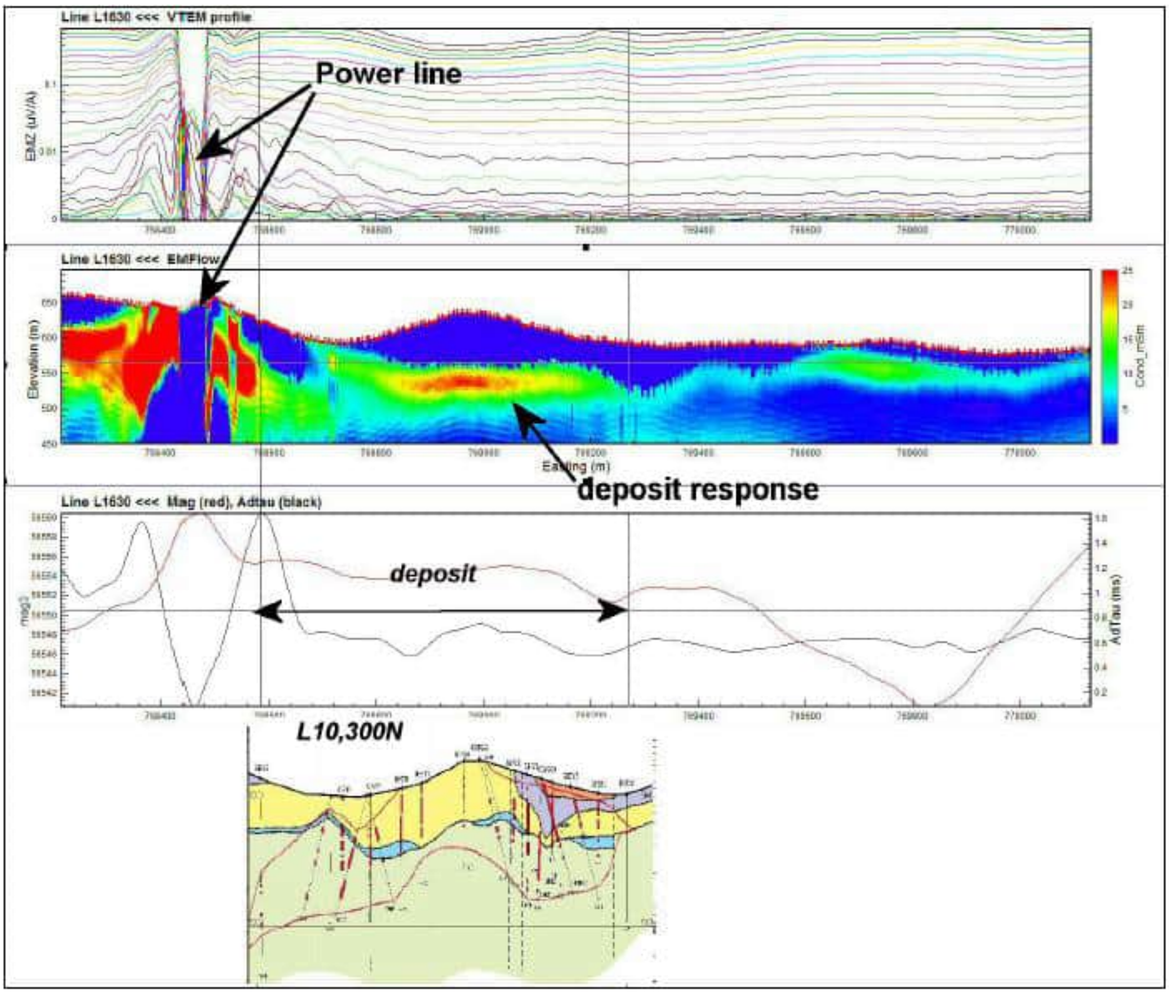


9.4.4 Versatile Time-Domain Electro Magnetics (VTEM) and Inversions

During early exploration various airborne and ground techniques were applied, primarily magnetic and IP surveying as well as some borehole electrical logging. The electrical work indicated that the contrasts between the host and mineralized rock were subtle and not of the tenor that the EM technique would normally be considered as a direct targeting tool (Condor Consulting, 2005). Figure 9.4.6 which shows a section across the deposit also highlights the impact that the 500Kv powerline has on electromagnetic techniques.

Regionally 80 flight lines were flown with a helicopter at a nominal height of 50m and inverted laterally and spatially, aiding in volcanic facies, fault interpretation and drill targeting regionally. Subsequently, this was also inverted naively without decimation or undue filtering at high resolution, to distinguish resistors and conductors in the sub surface over and surrounding the deposit. Several responses like the Bowdens deposit are noted within this dataset regionally.

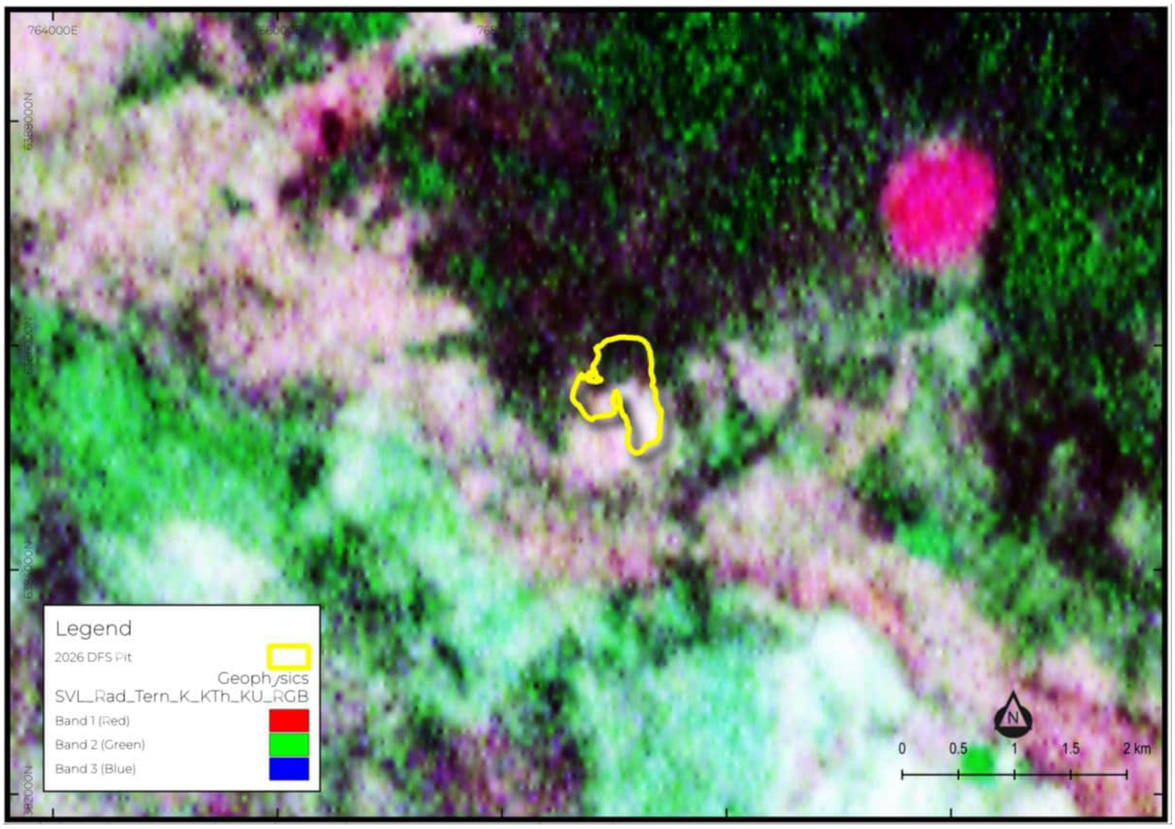
Figure 9.4.6 Deposit VTEM Response



9.4.5 RadioMetrics and Magnetics

A detail aeromagnetic and radiometric survey was flown in 2016 at high resolution across the entire tenement package surrounding the deposit. Post processing by Fathom Geophysics rendered numerous layers available for geological interpretation. The magnetic response of the deposit itself is unsurprisingly subdued (given the lack of iron within the Rylstone). The radiometric signature displayed in Figure 9.4.7 can be well associated with outcropping areas of higher adularia alteration in crystal tuffs (high K) (Figure 9.1.1).

Figure 9.4.7 Bowdens Radiometric Response (KThU – RGB)



10.0 DRILLING

10.1 Introduction

Drilling at the Bowdens deposit has been completed in successive campaigns between 1989 and 2025 by a succession of explorers and the current owner. The drilling database has evolved through successive exploration campaigns conducted by CRA Exploration (CRAE), Golden Shamrock Mines (GSM), Silver Standard (SS), Kingsgate Consolidated (Kingsgate) and, since 2016, Silver Mines Limited.

10.1.1 Extent of drilling

The drilling considered in this report is constrained to collars located within 1 km of the 2025 resource block-model bounds. The 1 km buffer therefore spans 767,100 – 770,600 mE and 6,383,700 – 6,387,300 mN, depicted in Figure 10.1.1.

10.1.2 Drilling Type and Operator

The Bowdens deposit has been drilled by five successive tenement holders. In chronological order these were CRAE, GSM, SS, Kingsgate and, since 2016, Silver Mines (operating as Bowdens).

The drillhole database details are presented in Table 10.1.1, and the distribution of these drilling campaigns are depicted in Figure 10.1.1 and 10.1.2. The database comprises 1,138 holes for 175,269 m located within 1 km of the 2026 resource block model. Of these, 332 are diamond drill (DDH) holes and 570 are reverse circulation (RC) holes, with the balance comprising reverse circulation collars with diamond drill tails (RC DDT), water bore (WB), percussion (PERC), auger (Aug) and aircore (AC) holes drilled for hydrogeological, geotechnical and reconnaissance purposes. The Resource drilling metres were completed by Silver Mines (diamond) and Silver Standard (RC) and relied on for estimation. The database highlights that 30% of the holes were DDH and 50% were RC holes.

Table 10.1.1 Bowdens Deposit Drilling Overview

Operator	Years	Drill type[1]	Holes	Metres	Procedures	Survey Method[5]	Survey Records	% Recovery Logged[2]	% Recovery[3]	% of Holes Total	% of Metres Total
CRA Exploration	1989	RC	33	3318	GS19980156 & PR89065	NR	34	0		2.9	1.9
		DDH	5	962	GS19980156 & PR89065	Eastman	27	0		0.4	0.5
Golden Shamrock Mines	1994 – 1995	RC	19	2327	GSM-SOP-1996 Annual Page 16	Eastman	60	0		1.7	1.3
		DDH	3	751	GSM-SOP-1996 Annual Page 16	Eastman	24	80	100	0.3	0.4
Silver Standard	1997 – 2007	RC	295	31226	2002-2003 RC splitting Procedures	Eastman	938	0		25.9	17.8
		DDH	45	7051	2007 Documented SOP	Eastman	277	90	92	4.0	4.0
		PERC	14	1304	2002-2003 RC splitting Procedures	Assumed Vertical	14		n/a	1.2	0.7
		RC DDT	3	638	2002-2003 RC splitting procedures	Eastman	21	40	98	0.3	0.4
		AC	7	146	-	-				0.6	0.1
Kingsgate Consolidated	2012 – 2014	RC	116	12742	2012 Bowdens Drilling SOP	Reflex EZ-Shot	244	38	77.3	10.2	7.3
		DDH	33	5173	2012 Bowdens Drilling SOP	Reflex EZ-Shot	196	99	97	2.9	3.0
		WB	17	906	2012 Bowdens Drilling SOP	Reflex EZ-Shot	15			1.5	0.5
		RC DDT	3	779	2012 Bowdens Drilling SOP	Reflex EZ-Shot	11	50	100	0.3	0.4
		Aug	110	338	2012 Bowdens Drilling SOP	COLL	110			9.7	0.2

Operator	Years	Drill type[1]	Holes	Metres	Procedures	Survey Method[5]	Survey Records	% Recovery Logged[2]	% Recovery[3]	% of Holes Total	% of Metres Total
Silver Mines / Bowdens Silver	2016 – 2025	DDH	246	86568	GEO-SOP-545	Reflex EZ-Shot	3117	96	99	21.6	49.4
		RC	107	14571	GEO-SOP-540	Reflex EZ-Shot	546	64	74.7	9.4	8.3
		RC DDT	9	4468	GEO-SOP-540 & GEO-SOP-545	Reflex EZ-Shot	136	48	99	0.8	2.5
		WB	6	1800	GEO-SOP-540	COLL	6			0.5	1.0
		PERC	67	201	GEO-SOP-540	COLL	67			5.9	0.1
Total_RC			570	64184			1822	21.2	75.4	50.1	36.6
Total_DDH			332	100505			3641	94.4	98.6	29.2	57.3
Total_PERC			81	1505			81			7.1	0.9
Total_RC_DDT			15	5885			168	47.3	99.1	1.3	3.4
Total_WB			23	2706			21			2.0	1.5
Total_Aug			110	338			110			9.7	0.2
Total_AC			7	146						0.6	0.1
Total_Combined			1138	175269			5843	92	84.5	100.0	100.0

Figure 10.1.1 Drilling by Company Name

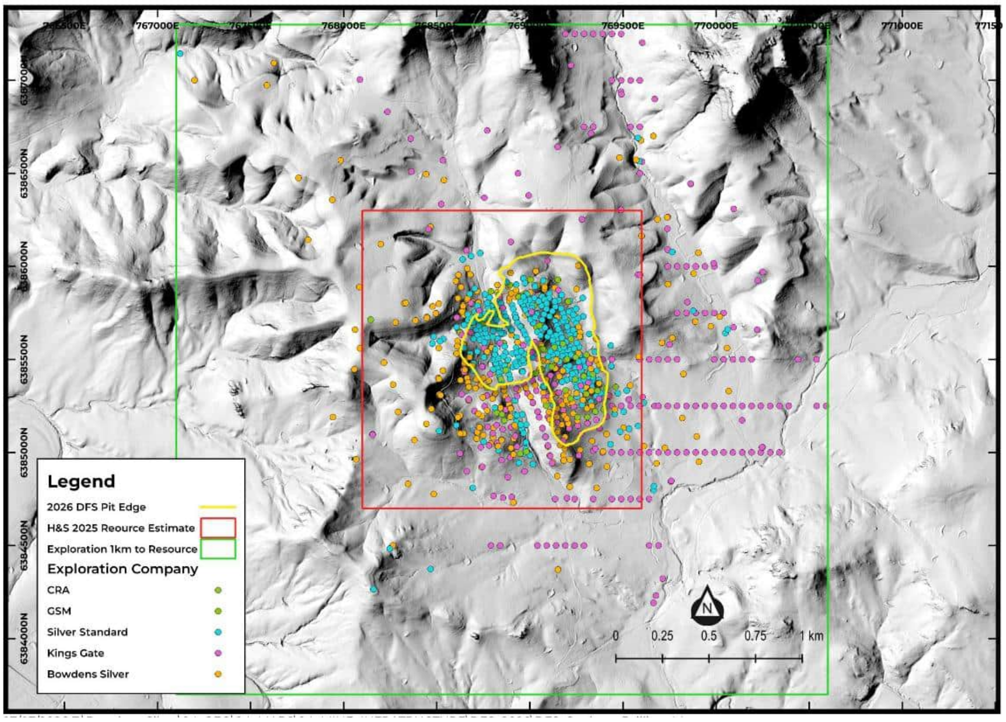


Figure 10.1.2 Drilling by Hole Type

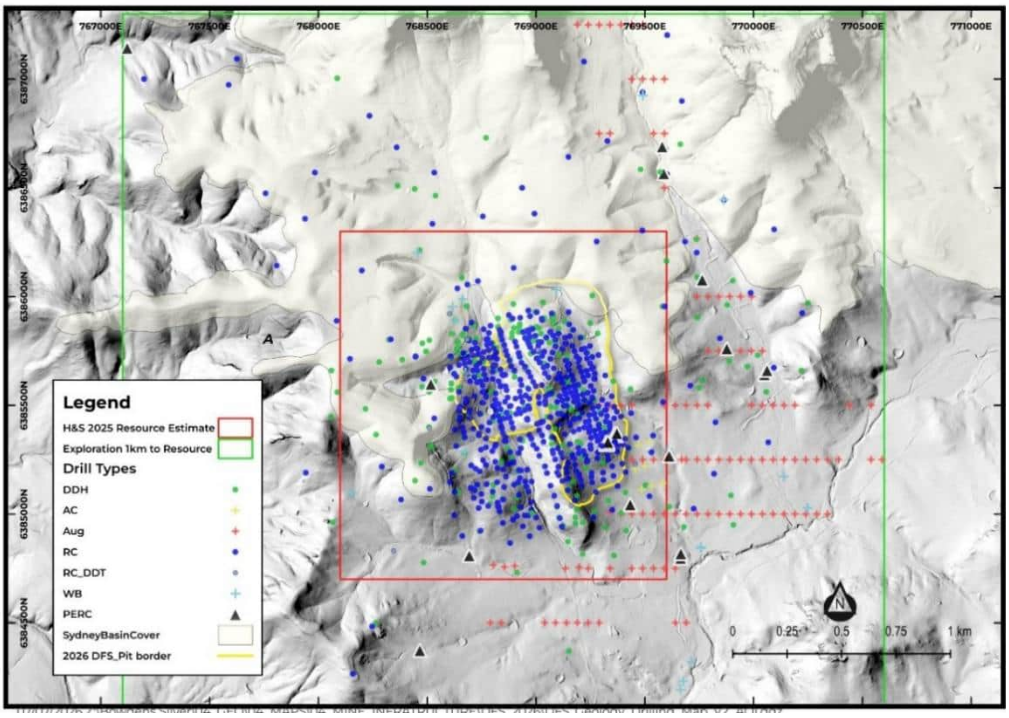
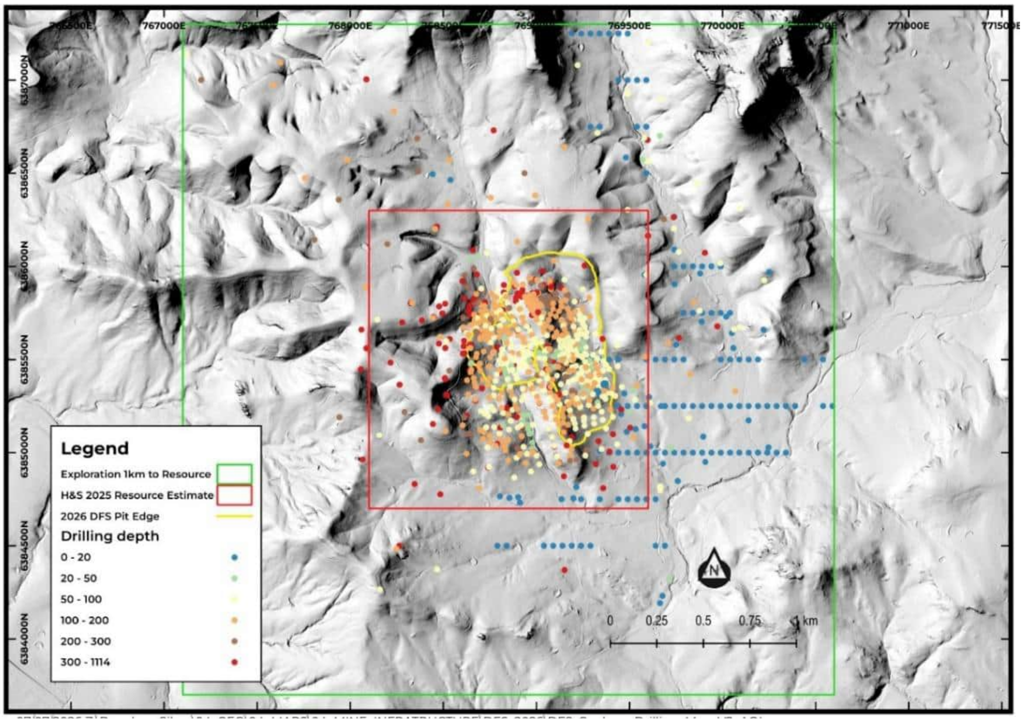


Figure 10.1.3 Drilling by Depth



Note all holes <20 m are Auger holes, with deeper drilling dominantly over the pit and to the west, targeting beneath the pit.

10.1.3 Drilling Procedures

Drill Records

Documentation of drilling operator core handling and sub sampling processes exist for all drilling, except air core and some percussion, neither of which are used for estimation purposes. Earlier drilling by GSM and Silver Standard and RC sub-sampling, including issues and rectifications well documented by a 2008 Silver Standard internal data-collection report (Elliot, 2007) and by original statutory reporting of drilling campaigns and Resources estimates between 1989 - 1997.

Land Surface, Datum and Collar Surveys

The location of nearly all drill collars is considered accurate down to 1 m in all dimensions, with reduced levels (RLs) in the database measuring the top of the drill collar casing at the time of survey. The drilling database collar coordinates used over the lifetime of the project range from a local Bowdens Grid, AMG66 and MGA94, with all three grids sharing the same AHD.

The historic land surface is available from a 1999 Geospectrum survey for audit of historical collar elevations. The current surface has been derived from a LiDAR survey tied to ground control points collected in 2017 via drone. State government LiDAR surveys flown in 2016 are in good agreement, with differences in RL attributable to differences in capture, filtering and interpolation methods of lidar capture.

The original local Bowdens Grid was converted to MGA94. All surveys of drill collars were converted to MGA94, and check surveys of historic drill collars were performed at various stages, with original coordinates preserved in the database. Collar locations for use in Resource estimation, have been surveyed in local grid (1989 – 2002) and resurveyed by O'Ryan Geospatial progressively (2017 – 2025) in MGA94 Zone 55. A historical pick-up program was performed to ensure that, where possible, collar coordinates could be confirmed and located to sub-10 cm accuracy.

Downhole surveying

The adjustment of the magnetic surveys to the MGA grid was checked and a correction of 11.6° applied universally to drilling surveys to align with the MGA grid rather than true north. This correction is confirmed by the database with the median difference between grid (MGA) and magnetic azimuths across 5,414 paired survey records being 11.6°.

Downhole surveys were acquired by single-shot cameras (pre-Kingsgate), Reflex EZ-Shot or Eastman cameras (Kingsgate) and Reflex EZ-Shot electronic cameras at 30 m intervals (Silver Mines), supplemented by north-seeking gyroscope surveys. The dominant survey methods recorded in the database are summarised in Table 10.1.1.

Drilling

DC diameters are nominally either PQ3, HQ3 or NQ3. Selected DC was oriented by conventional spear; Kingsgate and Silver Mines DC was oriented using Reflex ACT orientation tools. DC was halved with a diamond saw (NQ and HQ) or quartered (PQ) to provide representative assay of 1 m intervals and sub-samples.

RC drilling was completed using face-sampling hammers. For Silver Standard, GSM and CRAE drilling, RC holes were generally sub-sampled by riffle splitting, or by spear or grab sampling for rare wet samples. Kingsgate and Silver Mines RC holes were sub-sampled by cyclone-mounted cone splitters. RC samples are weighed for each metre and assessed for recovery, contamination and the effect of water if present.

A number of WB, percussion (PERC), auger (Aug) and air core (AC) holes were drilled for blast holes, hydrogeological, geotechnical, environmental and reconnaissance purposes. Results from These drill types are not included in the resource sample dataset. These holes are summarised in Table 10.1.1.

10.2 Summary

In total, 1,138 drillholes for 175,269 m lie within 1 km of the 2025 resource block model. DC (332 holes; 100,505 m) and RC (585 holes; 70,068 m) drilling, completed predominantly by Silver Mines and Silver Standard, account for most resource samples. Drilling spans 1989 - 2025, with the largest annual campaigns in 2017 and 2021 – 2023. The nominal drill spacing of 50 m by 25 m over the higher-grade zones, increasing to 100 m by 100 m peripherally, is considered adequate to establish geological and grade continuity for resource estimation.

10.3 Interpretation of All Relevant Results

10.3.1 Drilling Commentary

Most holes have been drilled angled at -60° to -80° towards the north and east (traversing mineralisation and crossing regional structures respectively), with occasional vertical holes. Drill orientation was designed to intersect the projection of breccia zones and zones of veins within an overall mineralised envelope, and interpretation of the mineralisation intercepts indicates that no significant sampling bias has been introduced.

10.3.2 Other Material Factors

Detail of logging recovery across core drilling has been regularly recorded. Geotechnical logging of 66,773 core intervals across 325 diamond holes returns a length-weighted mean core recovery of 98.6% (median 100%), with only 2% of intervals recording recovery below 90%. These lower-recovery zones are typically broken core or occasional clay zones and are not considered to have materially affected the results, and no significant relationship between sample recovery and grade is evident. Rock quality is correspondingly good, with a length-weighted mean RQD of 88.8% (median 97%).

RC sample recovery, contamination and water effects were monitored on a per-metre basis for the bulk of drilling.

Sample length is generally less than true thickness (dominantly 1 m) because most holes are drilled at an angle to the moderately north-dipping, vein-hosted and stratabound mineralisation.

11.0 SAMPLING PREPARATION, ANALYSES AND SECURITY

11.1 Introduction

Drilling over the Project has been sampled by multiple operators since the early 1990s using RC and DC drilling for the purposes of Resource estimation. Diamond drilling sample preparation has remained largely constant during the time with logging QA/QC procedures evolving from changes in standard materials, digital and assay technology as opposed to the sample media itself. The most material difference in sampling media pertains to RC drilling where manual riffle splitting (1989 – 2003) was changed to rotary cone splitting from 2008 onwards. Procedures and re-assay for both splitting types evolving from assays returned and issues identified. Kingsgate sampling process formalised in 2012, evolved to the current controlled standard operating procedures (GEO-SOP-540 for RC; GEO-SOP-545 for logging / sampling).

This section describes sample types and collection, on-site and laboratory sample preparation, security measures, analytical laboratories and methods, and the quality-control programmes.

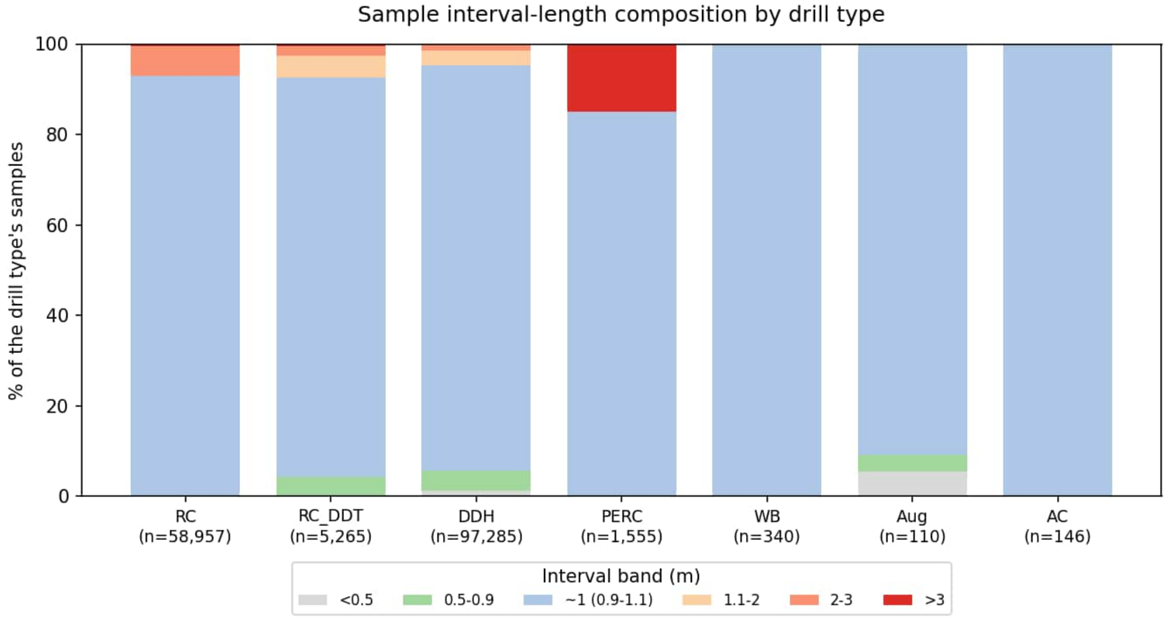
11.2 Sample Preparation and Security

11.2.1 Typical Sample Types and Collection

Samples relied upon in Resource estimation comprise RC chips and diamond drill core (HQ, PQ and NQ) sampled on nominal1 m intervals. In some cases, composites were made by Silver Standard and re-assays made following positive results. Vertical percussion holes often distal to the ore body were analysed on 3 m basis and not material to the Estimates results are presented in Figure 11.2.1. DC is cut into half (NQ / HQ) or quarters (PQ, or HQ in the case of duplicates) for sampling. Sampling is typically by 1 m interval, except where lithological contacts require adjustments with all composites for drilling shown in Table 11.2.1.

Sampling of RC chips are also by 1 m interval. The condition (dry / moist / wet) of each sample is recorded and split at the rig to a representative primary lab sample with a retained archive / duplicate.

Figure 11.2.1 Drilling Sample Length by Drill Type



RC sample weight's assays and sampling conditions are presented here in Figures 11.2.2 to 11.2.4 for each operator and are in accordance with the practices of the operators stated standard operating procedures (SOP) or reported practice. The impact of wet RC sampling conditions is assessed in two ways.

The sum of drilling by operator and sample types is shown in Figures 11.2.4 and 11.2.5, for RC and diamond respectively. QA/QC processes and sample material comprise certified reference materials, duplicates and field blanks inserted into the sample stream prior to lab submission. Diamond drilling was often collared in PQ with HQ tails. It is clear from these figures that most of the diamond drilling was undertaken by Silver Mines, while most of the RC drilling was undertaken by three separate companies.

Figure 11.2.2 RC Drilling Samples by Operator

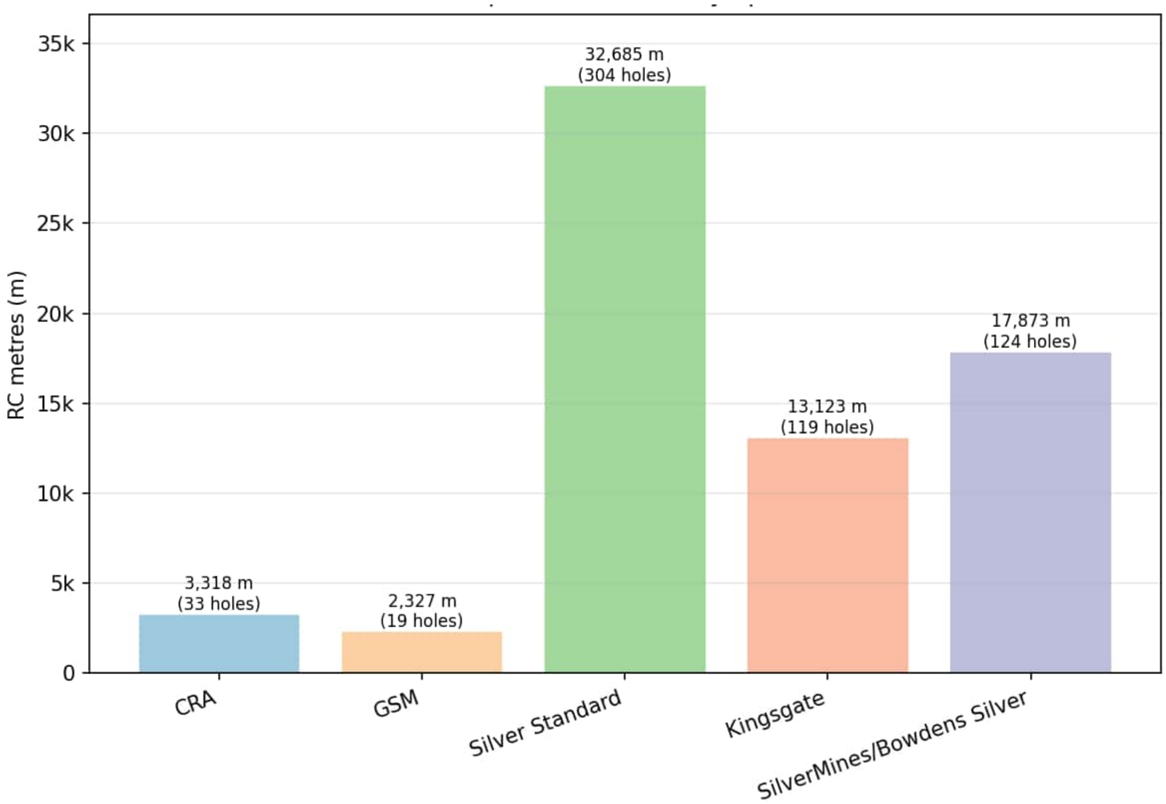
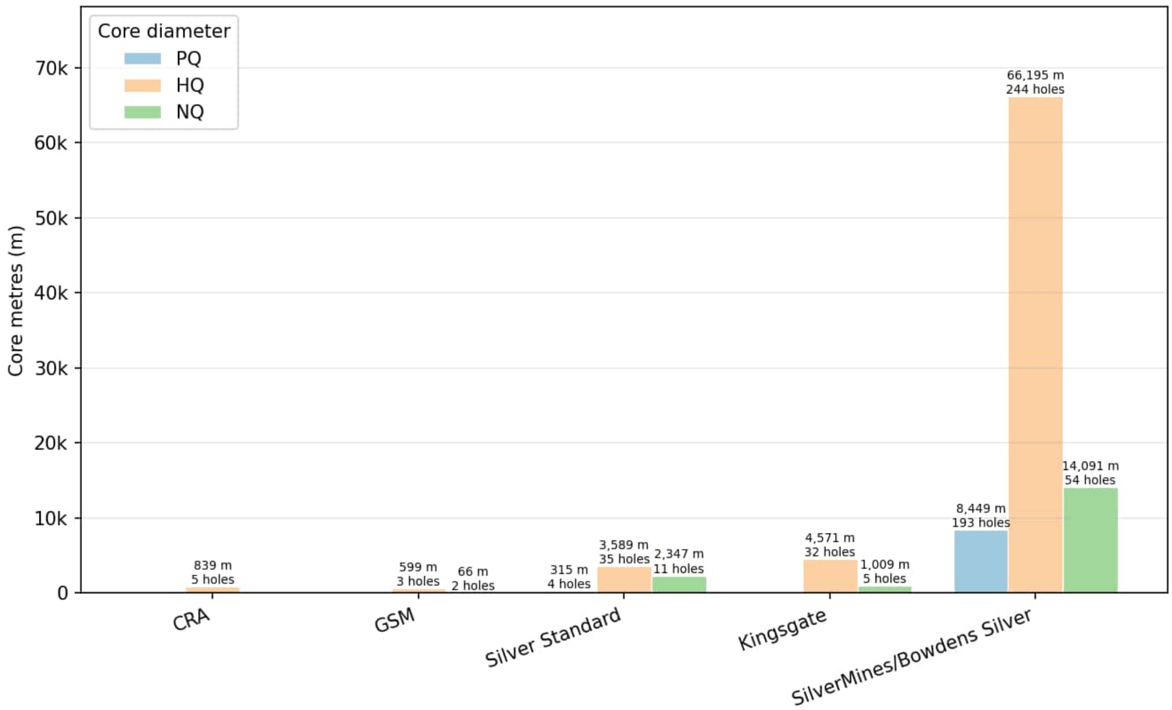


Figure 11.2.3 Diamond Drilling by Operator



11.2.2 Prior Operator Reverse Circulation Sampling

All procedures for drilling and QA/QC were documented by the various operators over time in statutory reporting half yearly and yearly. SOPs from all companies are available. The QA/QC practices and sample splitting procedures covering RC sampling of each operator are borne out by summary statistics of compiled QC data RC drilling in Figure 11.3.1 and 11.2.2. RC sample splits were nominally under 3 kg and ranged from 1/8 up to 1/16 splits of bulk samples.

11.2.3 Diamond Drill Core Sampling (All Operators)

Core mark-up and handling is conducted in accordance with Bowdens SOPs or similar earlier procedures from CRAE to Silver Mines. Drilling DC is laid out sequentially in core trays and marked for orientation and depth prior to geological and geotechnical logging. No geotech logging was performed on drillholes from 1989 but was for all diamond drilling thereafter.

Following logging, core is photographed (dry then wet) before cutting. The digital photographs are loaded to DataRock for depth registration and post processing of geological features.

Nominal sample length is 1 m unless varied to accommodate lithological contacts. Core is sawn in half with one side as the primary sample and the other (with orientation line) retained in the tray. The primary half is quarter-cored to produce a primary and duplicate samples which are placed in pre-numbered calico bags for dispatch to the laboratory.

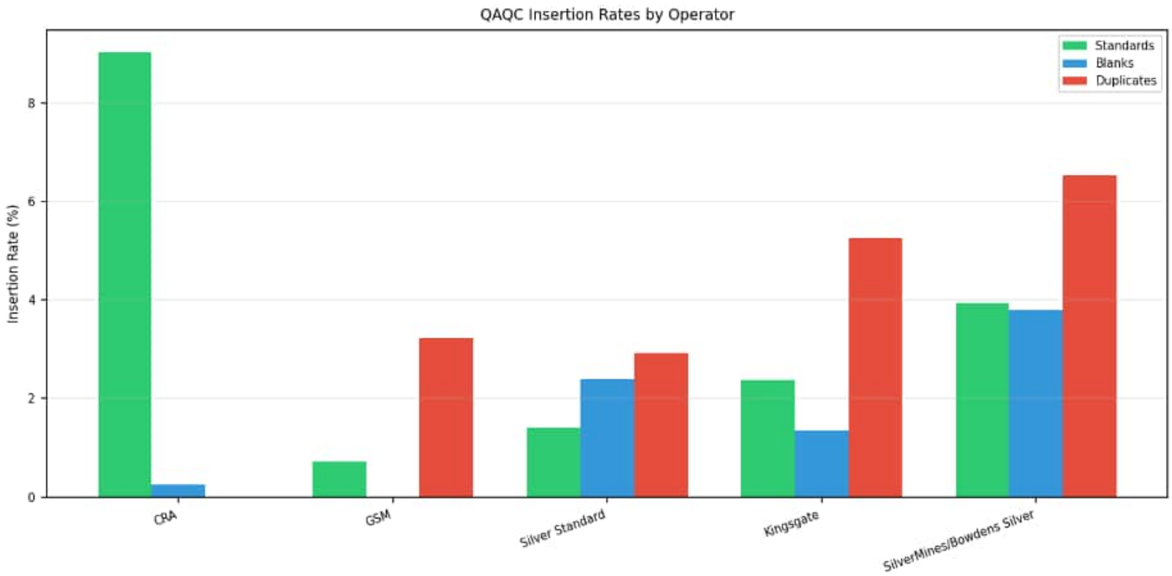
Sample dispatch and assaying follows sample collection with sample submission and chain of custody forms completed prior to dispatch to the laboratory. Physical chain-of-custody is maintained through bar-coded or sequentially numbered sample IDs and scanned PDFs for each dispatch. Samples are receipted at the National Association of Testing Authorities (NATA) accredited laboratory and documentation returned.

The analytical packages nominally used by Bowdens are LOG-22 (logging), CRU-21 (crush) and PUL-23 (pulverise) feeding ME-ICP61 and ME-MS61 for multi-element and base metals analysis with over range techniques used when necessary.

11.3 Quality Assurance and Quality Control Procedures

The QA/QC programmes for Bowdens include CRMs, blank standards and field / pulp duplicates. Historical insertion rates are demonstrated by Figure 11.3.1. for RC and diamond drilling.

Figure 11.3.1 QA/QC Insertion Rates by Operator



Note that Bowdens, Kingsgate and Silver Standard completed most of the resource drilling.

Repeatability for wet duplicates is good, but it is unclear whether any increase or decrease in grade occurred due to the wet sampling process. Note that a relatively few number of wet samples were logged, so this is unlikely to be a material factor. Approximately 20% of samples were not logged for dryness as shown in Figure 11.3.2.

Duplicate samples of diamond core don't have a good correlation. This may be due to non-homogeneous vein distribution and/or nuggety silver occurrences between core halves.

Duplicates of RC drilling tends to show better repeatability. This trend is consistent across elements (Ag, Pb and Zn), as demonstrated in Figures 11.3.5, 11.3.6 and 11.3.7.

Figure 11.3.2 RC Sample Condition Assay and Weights

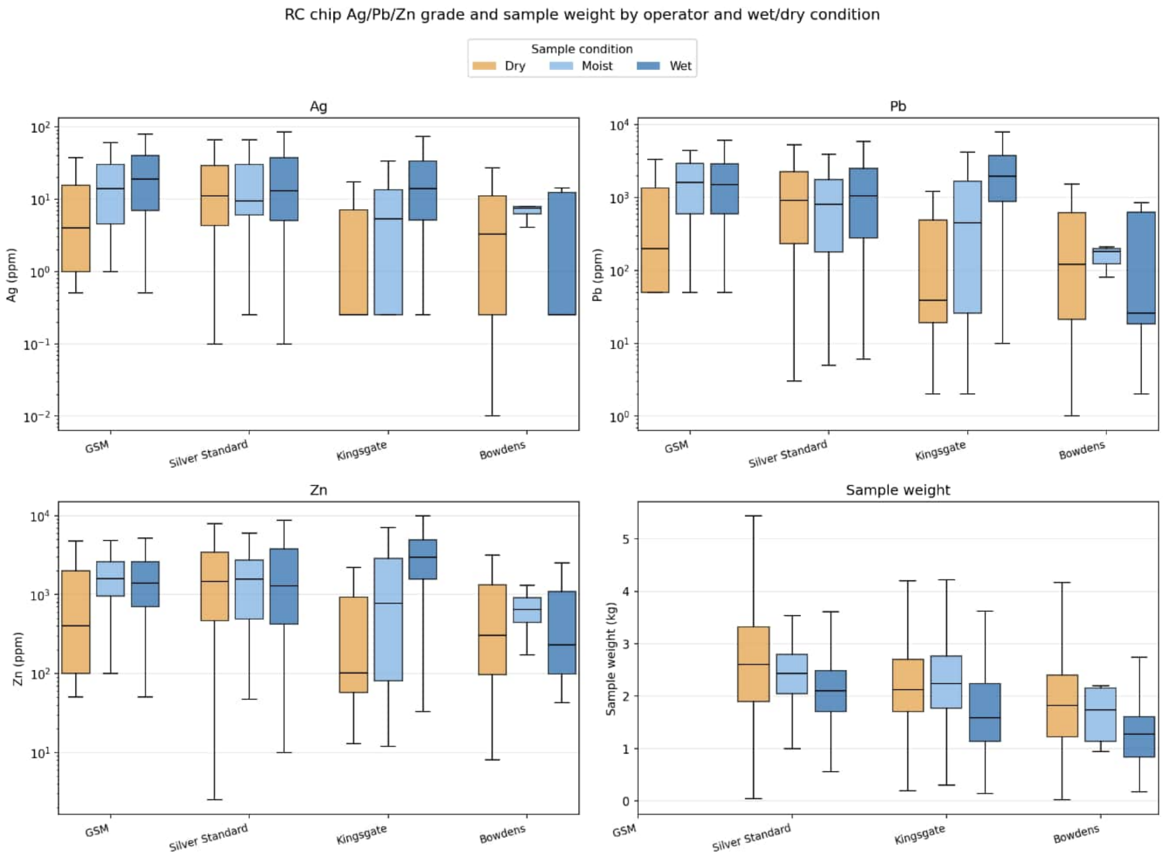
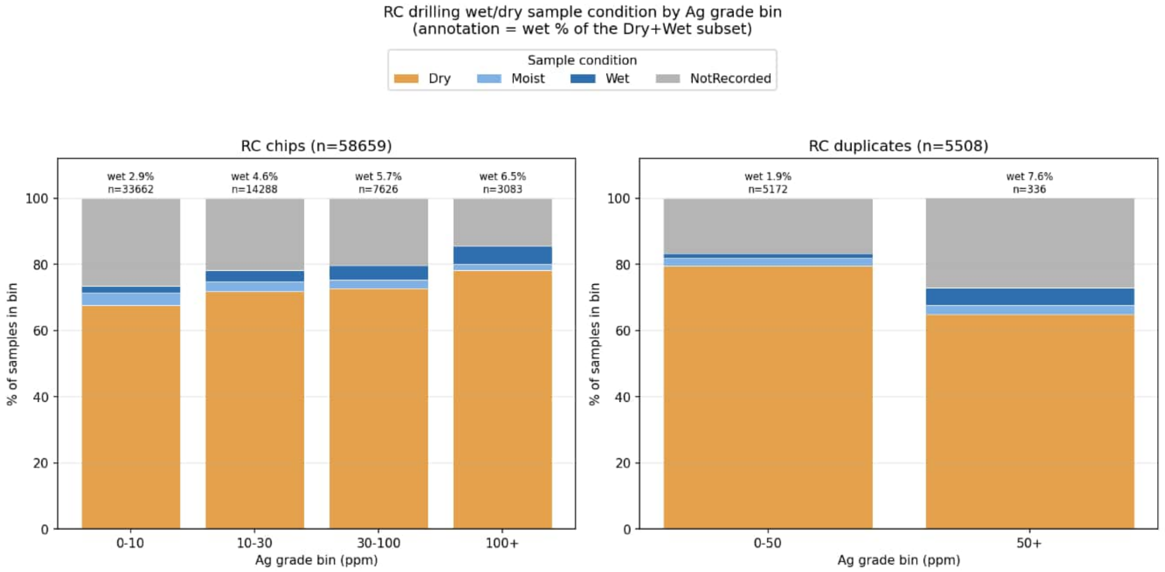


Figure 11.3.3 RC Drilling by Condition and Grade Bin



Note that majority of RC samples are dry, and there is a slight increase in number of wet samples as grade increases.

Figure 11.3.4 RC Duplicate Samples by Sample Condition

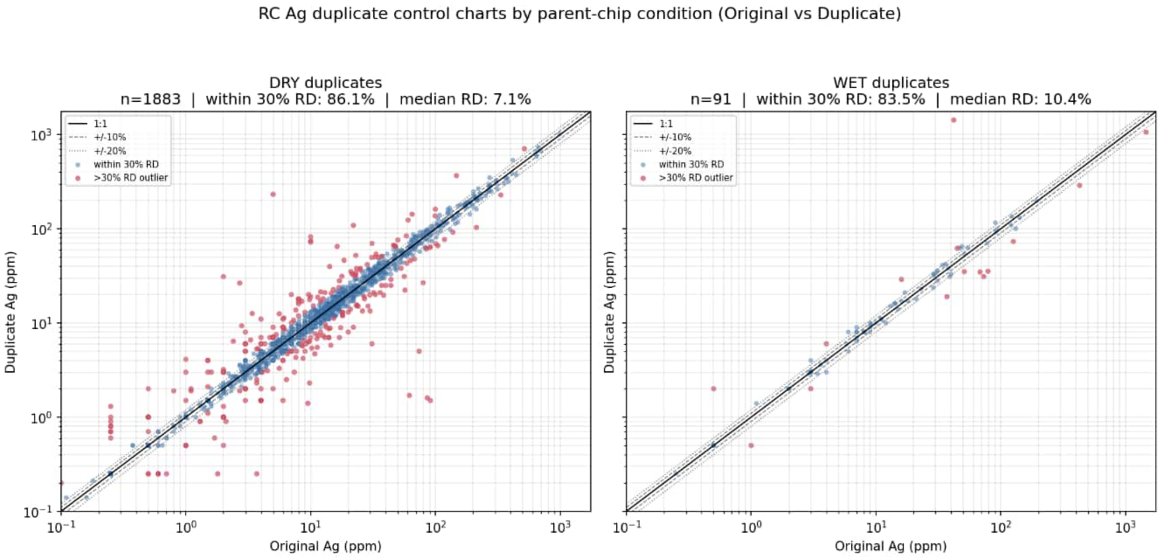


Figure 11.3.5 Ag Duplicate by Grade and Sample Type

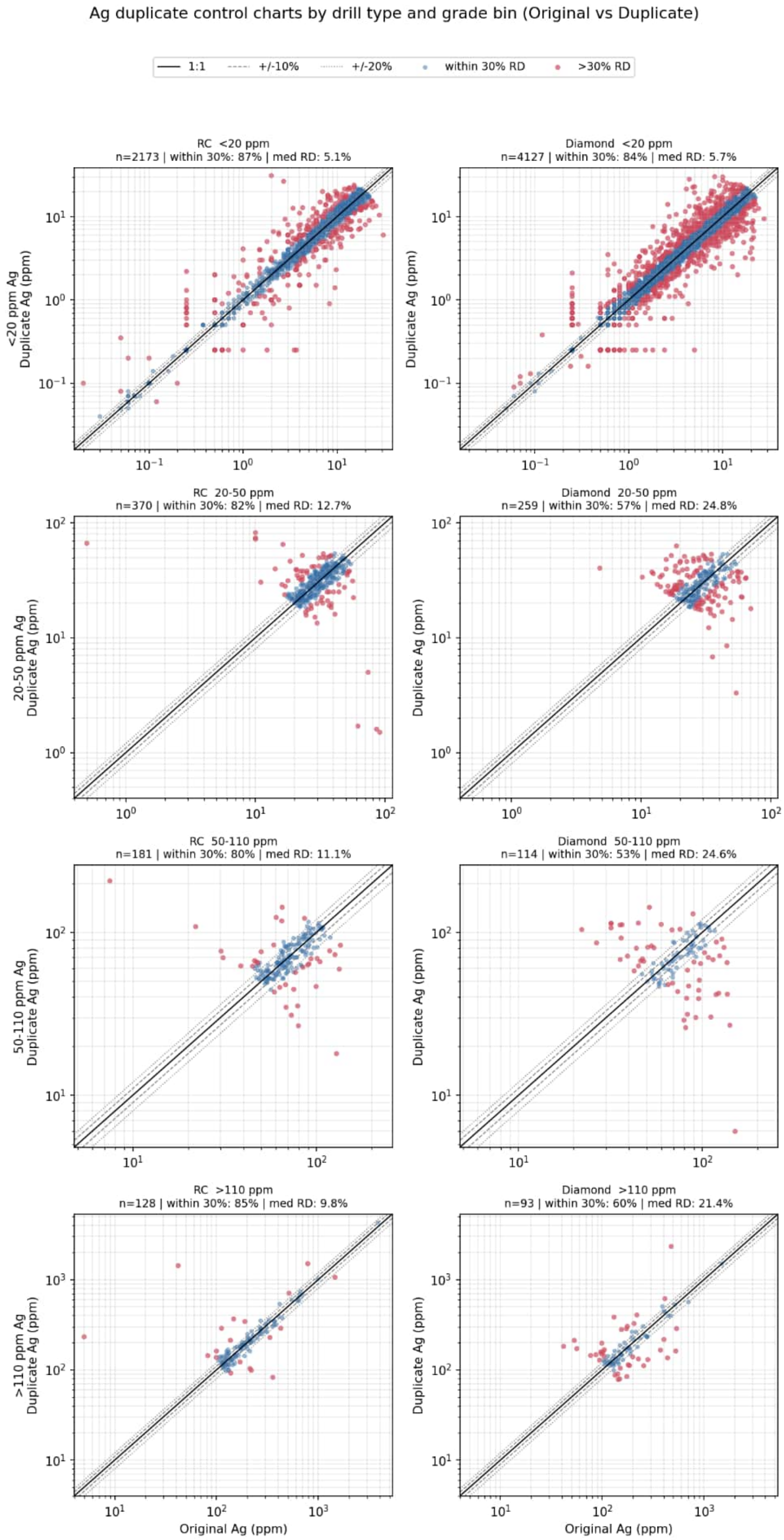


Figure 11.3.6 RC Field Duplicates, Pb

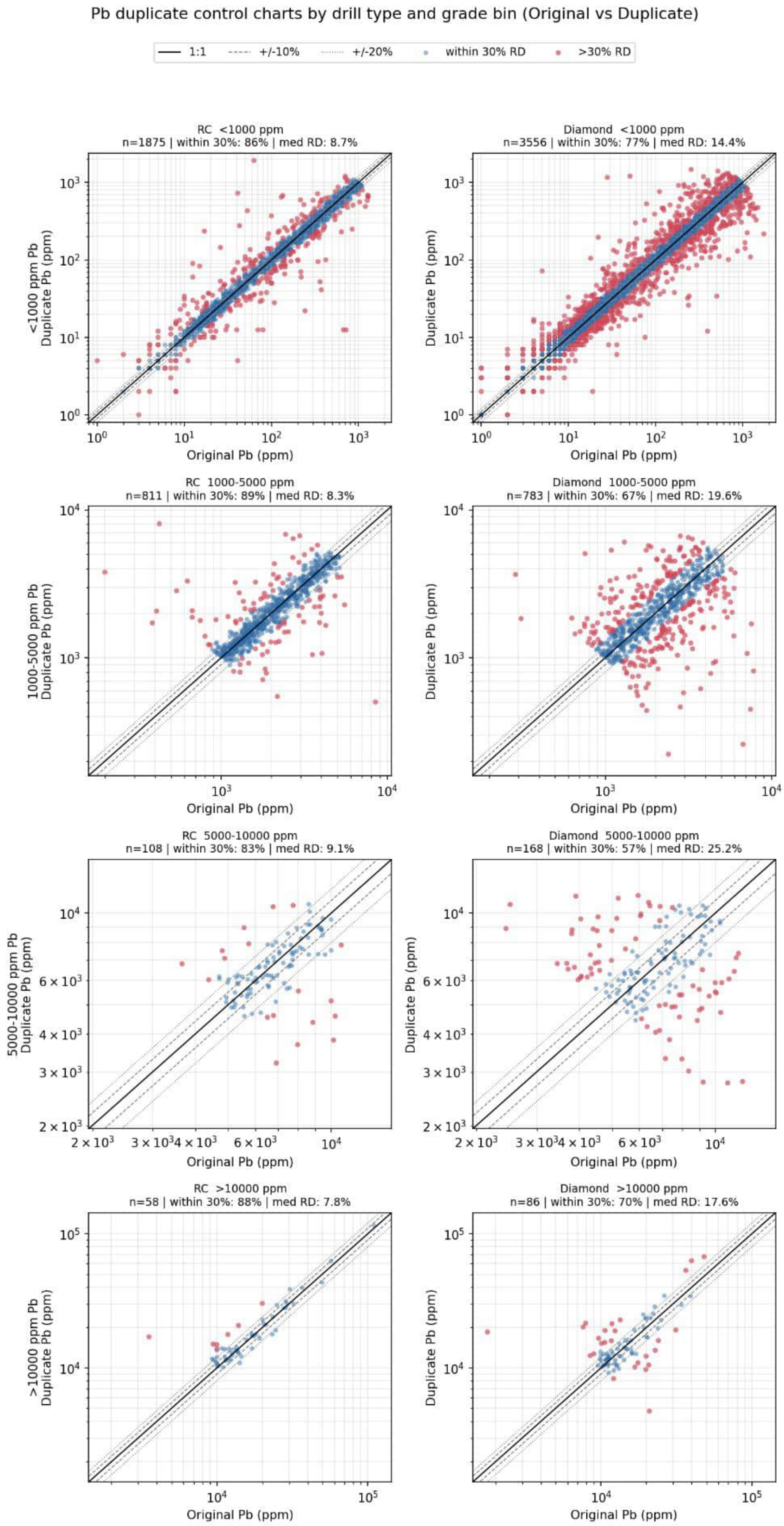
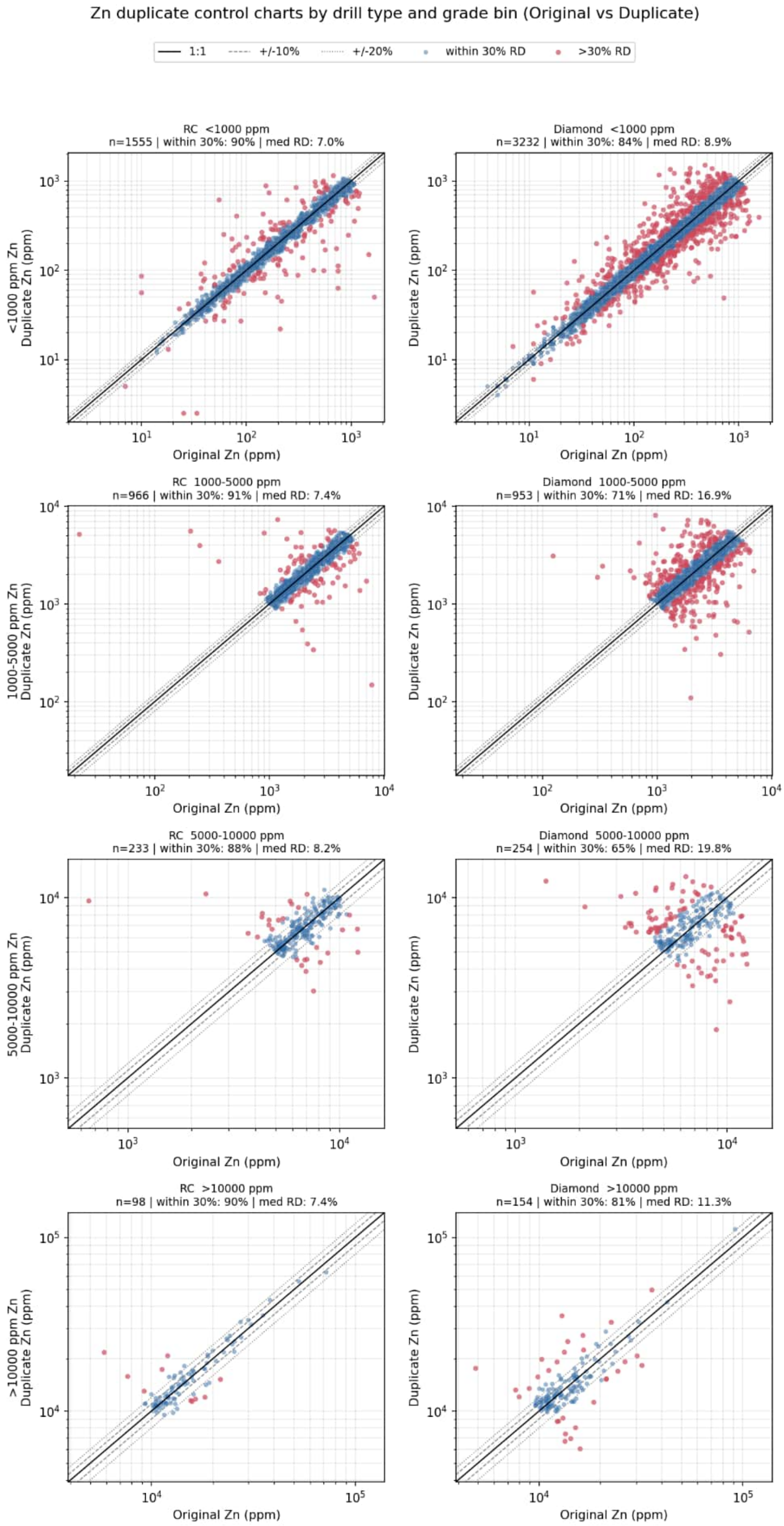
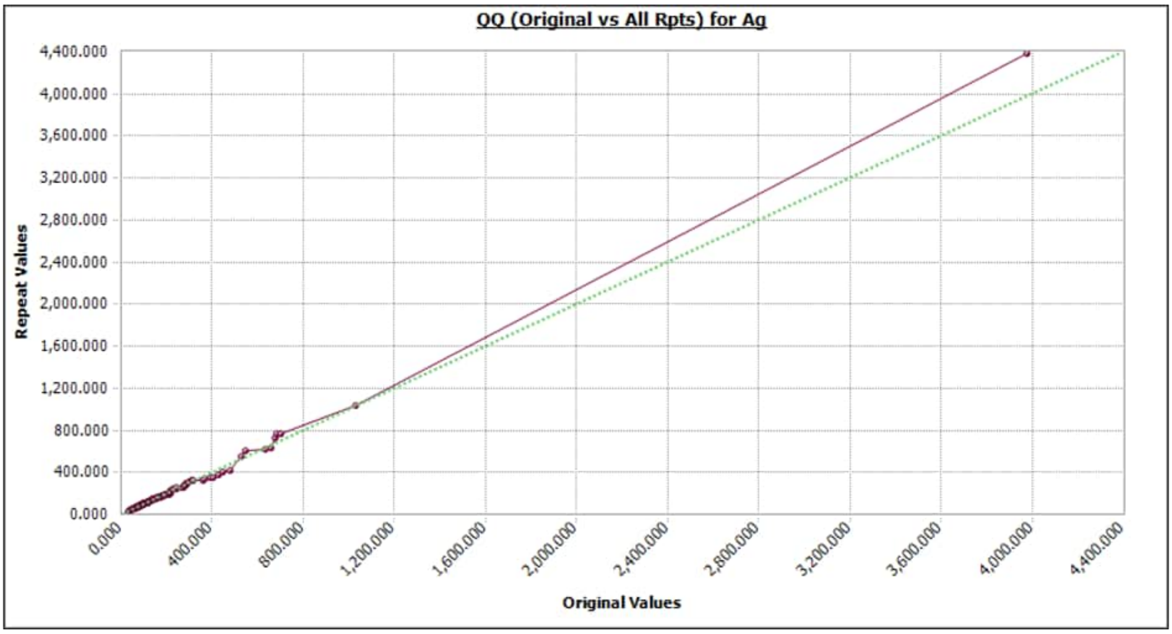


Figure 11.3.7 RC Field Duplicates, Zn



Repeat assays have good correlation confirming that the variability of duplicate assays is geology related rather than assay, as demonstrated in Figure 11.3.8.

Figure 11.3.8 Repeats



12.0 DATA VERIFICATION

As previously discussed, this report is styled in the format of a NI 43-101 Technical Report for familiarity of international readers, but is not NI 43-101 compliant. Data verification is as presented in each section of the report and has not been formally verified by QPs. However, all key assessments and engineering designs have been completed by professional entities and the verification by QPs can be arranged if Bowdens so elects.

13.0 MINERALS PROCESSING AND METALLURGICAL INVESTIGATIONS

13.1 Introduction

This section summarises the mineral processing and metallurgical investigations completed to support the DFS. Metallurgical testwork has been undertaken over a period exceeding 25 years by multiple project owners, consultants and independent laboratories, including Metcon Laboratories (Metcon), CSIRO, G&T Metallurgical Services (G&T), ALS Metallurgy, JKTech, KYSPYmet and Core Resources. The cumulative body of work provides a comprehensive metallurgical library encompassing mineralogical characterisation, comminution, flotation, concentrate quality, rheology, dewatering and tailings characterisation.

The metallurgical understanding of the deposit has evolved progressively through successive stages of project development. Early testwork undertaken during the 1990s and early 2000s focused on establishing whether the BSP mineralisation was amenable to conventional flotation processing and identifying the principal controls on silver recovery. These programs demonstrated strong flotation performance, confirmed that silver is predominantly associated with galena and sulphosalt minerals, and established the fundamental relationship between mineral liberation, grind size and metallurgical recovery. The work also demonstrated the ability to produce separate Pb-Ag and Zn concentrates and highlighted the benefits of concentrate regrinding for improved sulphide separation and concentrate upgrading.

The 2012 ALSM program represented the first integrated study designed to provide engineering design data for project development. This extensive program expanded the understanding of ore variability through lithology-based sampling and composite development, while generating comprehensive comminution, flotation, rheology and dewatering data. Detailed mineralogical investigations confirmed favourable galena and sphalerite liberation characteristics, while flotation optimisation studies established key design parameters, including grind size, flotation density and reagent regime. The program also evaluated alternative processing options, including bulk sulphide flotation and pyrite recovery, and provided the technical basis for the differential Pb-Ag and Zn flotation flowsheet considered during earlier project studies.

Subsequent work undertaken by JKTech in 2017 focused on validating and refining the conclusions of the ALSM program while broadening the understanding of ore variability and operational risk factors. The program incorporated additional comminution testing, flotation variability assessments, rheology investigations, process water studies and evaluation of alternatives to cyanide-based reagent schemes. The results confirmed the suitability of conventional SAG-ball mill grinding and differential flotation while identifying the significance of clay content, slurry rheology, water chemistry and mineralogical variability as key process performance drivers. The study also reinforced the relationship between silver recovery and lead recovery and demonstrated the continued importance of effective sphalerite depression for concentrate quality control.

With advancement of the DFS, metallurgical testwork shifted from confirmation of process amenability to optimisation of recovery, concentrate quality, process robustness and overall project economics. Extensive programs undertaken during 2023 investigated a range of underground and open pit ore composites representing the principal mining domains. These studies evaluated complex differential flotation flowsheets incorporating separate Ag, Pb and Zn recovery stages, together with multiple grinding strategies, cleaner and recleaner configurations, regrinding approaches and reagent schemes. The work significantly improved understanding of mineralogical variability, silver deportment, arsenic behaviour and concentrate quality. While the differential flowsheets demonstrated that separate saleable concentrates could be produced, the testing also highlighted increasing circuit complexity, sensitivity to ore variability and challenges associated with impurity management.

Further investigations completed on Main Zone (MZ), Pit West (PW) and Pit South (PS)composite samples refined the understanding of recovery mechanisms, flotation kinetics and concentrate upgrading requirements. These programs demonstrated the strong influence of Ag head grade on overall metallurgical recovery and confirmed that arsenic deportment, Zn rejection and mineral liberation remained critical considerations for flowsheet selection. Additional cleaner, recleaner and selective regrinding studies provided valuable insights into the trade-off between concentrate grade, recovery and process complexity.

The most recent phase of metallurgical development focused on simplification of the processing route and optimisation of project economics. Bulk flotation (production of a single silver-lead-zinc concentrate) investigations completed during 2024 and subsequent optimisation studies demonstrated that a simplified bulk sulphide flotation flowsheet could achieve robust Ag, Pb and Zn recoveries while reducing circuit complexity, reagent consumption, operating risk and capital intensity. The work showed that, although differential flotation could generate higher selectivity under some conditions, the bulk flotation approach provided a more practical and reliable solution across the range of ore types expected to be processed throughout the life of mine. These studies ultimately formed the basis of the 2026 DFS process design.

In parallel with flotation development, significant advances were made in the understanding of tailings management and dewatering performance. Historical thickening and filtration investigations were supplemented by recent rheology, filtration and filtered tailings placement studies, providing the information required for selection of the tailings dewatering strategy and integration of water recovery into the overall process design. The combined dataset provides confidence that the selected processing and tailings management approach is technically achievable across the expected range of ore and operating conditions.

The cumulative results of these metallurgical programs demonstrate a progression from initial amenability testing through process definition, variability assessment, optimisation and 2026 DFS design support. The results confirm that the Bowdens mineralisation is amenable to conventional crushing, grinding and sulphide flotation processing, with metallurgical performance primarily controlled by mineral liberation, grind size, mineralogical variability, reagent regime and impurity deportment. The outcomes of the testwork have been used to establish design criteria, recovery assumptions, equipment sizing parameters and operating philosophies adopted for the 2026 DFS process plant and tailings dewatering facilities.

13.2 Historical Metallurgical Testing

13.2.1 1990s to 2000s Early Testwork

Early metallurgical programs demonstrated that the Bowdens ore is amenable to conventional flotation processing, with strong recovery of sulphide minerals and the ability to produce separate Pb–Ag and Zn concentrates. Testwork across multiple ore types indicated a consistently favourable flotation response, reflecting relatively simple sulphide mineral associations at coarse grind sizes. Mineralogical studies undertaken during this period confirmed that silver is predominantly associated with galena and, to a lesser extent, complex sulphosalt minerals, establishing that effective lead flotation is a key driver for overall silver recovery.

Detailed testwork identified that, while acceptable recoveries could be achieved at relatively coarse primary grind sizes (typically 100 to 150 µm), improved selectivity between sulphide minerals and gangue required additional liberation. This led to the recognition that regrinding of rougher or intermediate concentrates was necessary to improve mineral separation efficiency and concentrate quality, particularly for fine-grained silver-bearing phases. Early programs therefore established the fundamental relationship between grind size, liberation, and flotation performance that continues to underpin the current process design.

A range of flowsheet configurations were evaluated, including bulk sulphide flotation followed by downstream separation, as well as sequential (differential) flotation circuits targeting Ag / Pb recovery prior to Zn flotation. Sequential flowsheets generally produced higher-grade concentrates with improved selectivity, as the staged recovery allowed for selective reagent application and controlled separation of sulphide species. However, these advantages were offset by increasing circuit complexity, including multiple reagent addition points, tighter process control requirements, and increased sensitivity to ore variability and mineralogical changes.

In addition, early observations highlighted that reagent schemes required for effective differential flotation – particularly the use of depressants to control sphalerite and pyrite, were highly sensitive to operating conditions such as pH, oxidation state, and grind size. This resulted in variable performance across different ore types and test conditions. The combined effects of flowsheet complexity, reagent sensitivity, and variable metallurgical response were recognised early as key challenges, ultimately informing the later transition toward simpler and more robust bulk flotation processing strategies.

Key historical testwork and reports include:

- Metcon Laboratories report 95206 – Preliminary Metallurgical Testing of Ag-Pb-Zn Ore from the Bowden's Gift Deposit, April 1996.
- Metcon Laboratories report 97391 – Development of Flotation Conditions with Bowdens Ag-Pb-Zn Ore, May 1998.
- AMDEL Limited report N078CO98 – Comminution Test, August 1998.
- AMDEL Limited report N115CO98 – Comminution Test, November 1998.
- CSIRO Mineral report DMR-956 – Mineralogical Investigation of Nine Metcon Ore Samples Using QEM SEM, December 1998.

- Metcon Laboratories report 98589 – Metallurgical Testwork with Bowdens Silver Ore from the Norther, Southern, and Bundara Ore Zones, July 1999.
- Metcon Laboratories report 00854 – Metallurgical Testwork aimed at Maximising Silver Recoveries from Bowdens Silver Ore Main Zone South Composite, January 2001.
- Metco Laboratories report M0744 – Standard Metallurgical Response of Six New Bowdens Silver Ore Composites from Year 2003 Drilling Program, April 2004.
- G&T Metallurgical Services report KM1711 – The Metallurgical Response of Four Composites, December 2005.
- G&T Metallurgical Services report KM1825 – The Metallurgical Responses of Bowdens Ores, January, 2007.

13.2.2 2012 ALS Metallurgy Program

A comprehensive metallurgical test program was undertaken by Bowdens in 2012 to establish and validate key process design parameters for the Bowdens deposit. The program expanded on earlier work by G&T Metallurgical Services and Metcon, with testwork completed by ALS Metallurgy (Sydney) and comminution work by ALS Ammtec (Perth).

The primary objective was to define processing characteristics across the orebody and confirm design inputs. The test program covered:

- Ore characterisation such as mineralogy, lithology-based behaviour.
- Geotechnical and comminution properties including UCS, crushing work index, Bond rod / ball mill indices, abrasion index, SMC / JK parameters.
- Processing performance such as flotation response (batch and locked cycle), thickening (dynamic / static), and slurry rheology.
- Chemical aspects including cyanide dosing strategy and tailings speciation.

Additionally, the program assessed the potential inclusion of a pyrite flotation and leach circuit, which was later excluded on economic grounds.

Mineralogy

The ore comprises a low-volume polymetallic sulphide assemblage dominated by sphalerite (Zn) and galena (Pb), with minor silver present as sulphosalts, sulphides, and native silver. Pyrite is the principal sulphide gangue mineral. The non-sulphide gangue is largely composed of silicate minerals, including feldspar, quartz, and mica. Mineralogy photos provided in Figures 13.2.1, 13.2.2, 13.2.3, and 13.2.4.

Figure 13.2.1 Mineralogy Photo 1

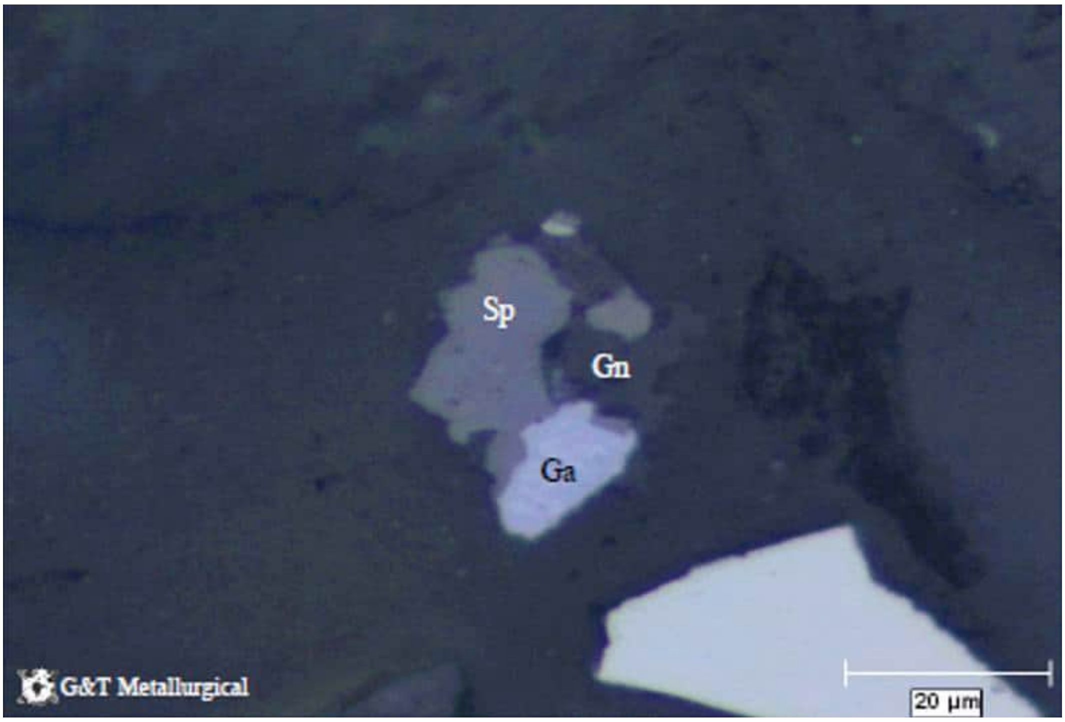


Figure 13.2.2 Mineralogy Photo 2

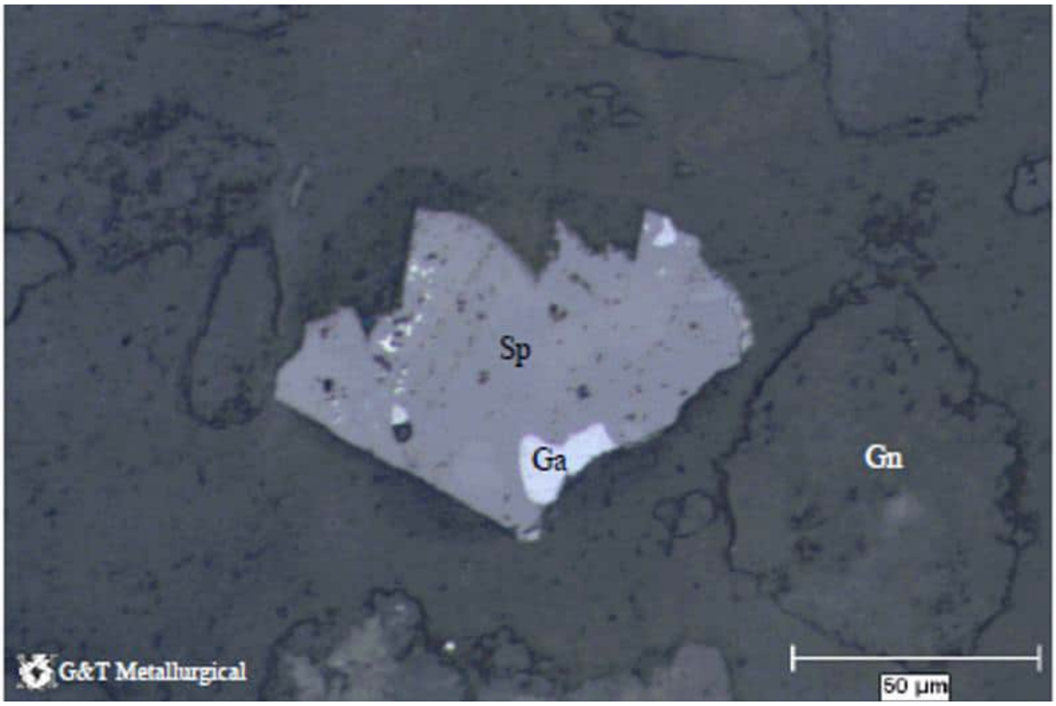


Figure 13.2.3 Mineralogy Photo 3

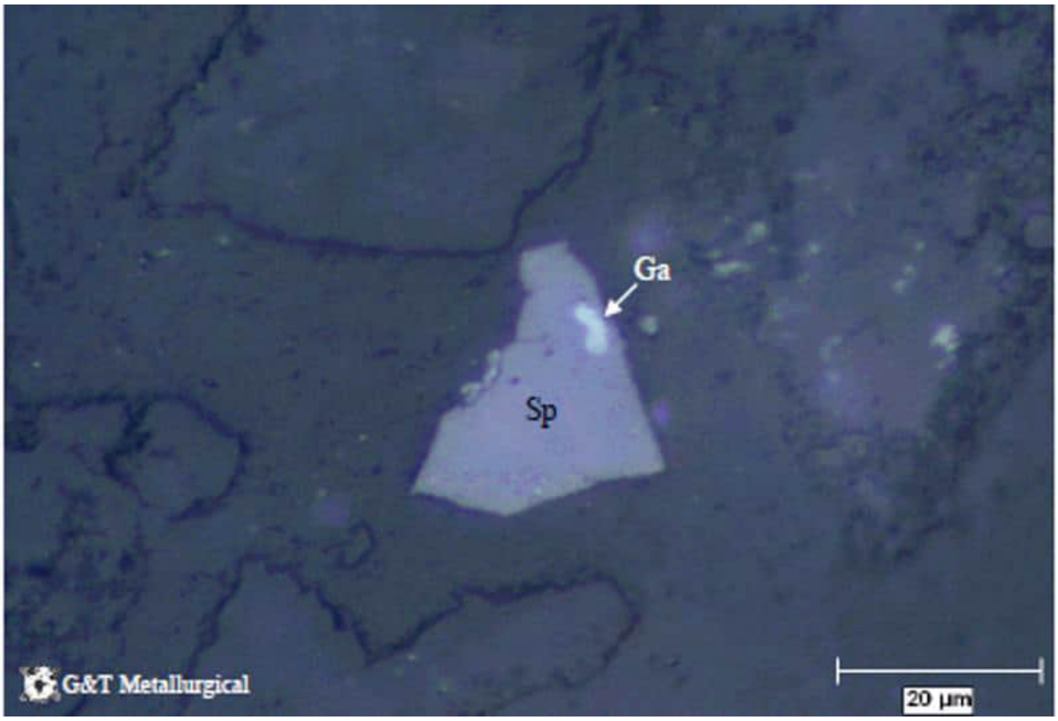
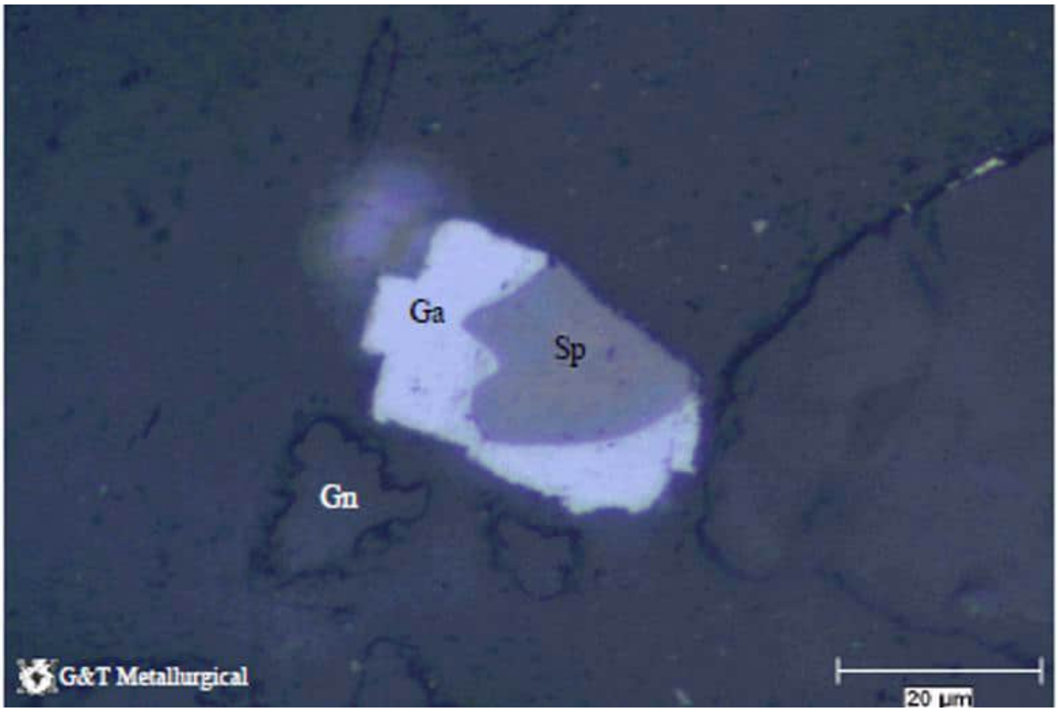


Figure 13.2.4 Mineralogy Photo 4



Whole Ore QEMSEM analysis of rougher concentrates from crystal tuff (B016) at a 150 µm grind showed clear deportment characteristics for Ag, Pb, and Zn.

Silver occurs mainly as acanthite / argentite (Ag_2S) and stephanite, and is strongly enriched in the Pb concentrate, where it is partly liberated but also associated with galena. A proportion of fine silver (<20 µm) remains locked, particularly below ~15 µm. In the Zn concentrate, Ag is less well recovered, with a higher proportion of locked particles associated with sphalerite.

Lead occurs entirely as galena and is generally well liberated, with minor locking in sphalerite- or pyrite-rich particles. Some galena reports to the Zn concentrate, mostly as locked or middling particles.

Zinc occurs as sphalerite, with a portion reporting to the Pb concentrate. While much of this sphalerite is liberated, some remains associated with galena or pyrite, indicating potential for improved separation through flotation optimisation.

Overall, the results indicate good liberation at the test grind size, but fine particle locking and sulphide associations limit complete separation, highlighting opportunities for regrind and flotation circuit optimisation.

Comminution Testwork

Comminution testwork indicated that the ore exhibits low to moderate hardness with distinct variability by lithology,

- UCS results range from approximately 14 megapascal (MPa) to 136 MPa, with ignimbrite representing the hardest material, breccia the softest, and crystal tuff showing intermediate strength.
- Bond comminution indices further support moderate competency, with crushing work index values of approximately 7.6 to 12.1 kWh/t and Bond Ball Mill Work Index (BBWI) values ranging from 10.6 to 24.1 kWh/t. Bond abrasion indices of 0.05 to 0.33 indicate low to moderate abrasivity. The rod-to-ball mill work index relationship suggests a low propensity for pebble generation in a SAG milling circuit.
- SMC test results confirm suitability for SAG milling, with DWi values ranging from 2.48 to 7.74 kWh/m^3 and Mia values from 9.9 to 23.5 kWh/t. Of particular importance are the JKMRC determined ore impact parameter (A×b) values, which range from 32 to 97. The A×b parameter is a measure of ore competency and resistance to impact breakage in a SAG mill, with higher values indicating ore that breaks more readily and is generally associated with higher SAG throughput and lower energy consumption. All measured A×b values exceed the typical SAG suitability threshold of 25, indicating that the ore should respond well to SAG grinding. The lower A×b values recorded for some ignimbrite samples (35 to 39) reflect their greater competency, whereas breccia and some crystal tuff samples exhibit significantly higher A×b values (up to 97), suggesting easier breakage and improved grindability.
- The ta parameter, which ranged from 0.33 to 1.04, provides an indication of ore toughness and its resistance to abrasion and attrition. The observed ta values indicate that the ore is not excessively abrasive and is unlikely to cause abnormal liner or grinding media wear.

Pulp Viscosity

Pulp viscosity testing was conducted at grind sizes close to the selected plant grind of P_{80} ~106 µm and solids contents ranging from 25 to 65% weight by weight (w/w). Results showed that slurry viscosity increased progressively with solids concentration, with a sharp rise occurring above approximately 55 to 60% solids. At 65% solids, viscosities ranged from 7,200 to 13,000 cps, compared with only 10 to 35 cps at 25% solids, indicating a significant deterioration in slurry flow characteristics at high densities.

Flotation Testwork

Key flotation outcomes from the testwork are:

- Reagent optimisation demonstrated that flotation selectivity is strongly dependent on hydrogen ion exponent (measure of acidity of alkalinity) (pH) control and zinc depression. Tests showed that the use of sodium metabisulphite (SMBS) alone at natural pH lowered slurry pH to approximately 4.5 to 5.0, resulting in poor selectivity and up to 92% of Zn reporting to the lead rougher concentrate. The combination of 60 g/t NaCN and 200 g/t $ZnSO_4$, together with lime addition to maintain pH 8 to 9, provided effective zinc and pyrite depression while maintaining high lead recovery. Soda ash was also found to improve silver flotation kinetics, reduce slurry viscosity and decrease mass pull compared with lime, although lime remained more effective for pyrite suppression.
- Flotation density testing indicated that high pulp densities promoted slime entrainment and reduced concentrate quality. Rougher flotation at 35% solids produced significantly higher mass pull and dilution, whereas densities around 25% to 30% solids achieved a superior balance between recovery and concentrate grade. This behaviour is consistent with the viscosity work, which identified increasing rheological problems at elevated solids concentrations.
- Grind size optimisation confirmed that a primary grind of P_{80} 106 µm was the most suitable operating point. Coarser grinds of 150 µm and 212 µm generally resulted in lower recoveries and reduced selectivity.
- Locked cycle testwork demonstrated optimum metallurgical performance at the selected 106 µm grind size.
- Analysis of both locked cycle and variability test results demonstrated a strong correlation between silver recovery and lead recovery, confirming that silver is predominantly associated with galena and is recovered through the lead flotation circuit. This relationship became increasingly evident as silver and lead head grades increased.
- Weathered and aged materials performed significantly worse than fresh ore. Although some weathered samples achieved silver recoveries above 60%, concentrate grades and base metal recoveries were generally poor and highly variable.
- Concentrate quality was generally considered marketable. Concentrate impurity levels, including arsenic, antimony, and iron, were generally within acceptable limits.

Dewatering Testwork

Key results from the thickening and filtration testwork include:

- Magnafloc M10 identified as the preferred tailings flocculant at approximately 40 g/t dosage.

- Tailings thickener underflow densities of 53 to 56% solids (w/w) achieved at 0.3 to 1.0 t/m²h flux rates.
- Static settling tests achieved settled densities of 63% solids for lead concentrate and 52.5% solids for zinc concentrate.
- Vacuum filtration rates on lead and zinc concentrates ranged from 340 to 570 kg/m²h.
- Concentrate moisture contents ranged from 14% to 16% for Pb and 20% to 22% for Zn, exceeding transportable moisture limits and highlighting the need for further dewatering optimisation.

Bulk Sulphide Flotation Testwork

ALS investigated an alternative bulk sulphide flotation flowsheet in which all sulphide minerals were recovered into a single bulk concentrate prior to regrinding and sequential flotation to produce separate Pb-Ag, Zn and pyrite concentrates. Initial testwork demonstrated that a bulk sulphide concentrate could be produced with very high recoveries, achieving 94% Ag, 99% Pb, 98% Zn and 85% sulphur (S) recovery into a concentrate representing approximately 15% mass pull. However, while sulphide recovery was excellent, subsequent separation of the bulk concentrate proved less selective than direct sequential flotation.

13.2.3 2017 JK Tech Program

The 2017 JK Tech metallurgical program was undertaken to validate previous ALS testwork findings and further evaluate ore variability across the Bowdens deposit. Nineteen samples representing crystal tuff, breccia / rhyolite and welded tuff / ignimbrite lithologies were subjected to comminution, flotation, process water, rheology and cyanide residue investigations. The results broadly confirmed the suitability of the proposed SAG milling and differential Pb-Ag / Zn flotation flowsheet whilst highlighting the importance of ore variability, slurry rheology, reagent selection and process water quality on metallurgical performance.

Cyanide Alternative Assessment

One objective of the JKTech program was to evaluate alternatives to cyanide for sphalerite depression. SMBS, sodium sulphite and sodium sulphide were assessed as potential replacements.

Sodium metabisulphite performed poorly and failed to achieve adequate Pb-Zn selectivity. Significant entrainment occurred in the Pb rougher concentrate, resulting in excessive Zn misreporting to the Pb circuit and lower overall Zn recovery. Sodium sulphite produced variable results and generally achieved lower concentrate grades than the standard reagent suite. In contrast, sodium sulphide generated Pb and Zn grade-recovery performance broadly comparable to the conventional flowsheet and was identified as the most technically viable cyanide substitute.

Cyanide Residues in Tailings

Analysis of rougher flotation tailings filtrate returned:

- Free cyanide: 1.9 – 3.0 milligrams per litre (mg/L) (average 2.51 mg/L).
- WAD cyanide: 4.22 – 8.36 mg/L (average 6.49 mg/L).
- Total cyanide: 4.3 – 8.7 mg/L (average 6.72 mg/L).

- Total alkalinity: 166 – 569 mg/L as $CaCO_3$ (average 258 mg/L).

Key Conclusions

The JKTech 2017 program significantly increased confidence in the Bowdens process flowsheet by:

- Confirming the suitability of SAG-ball mill grinding across all lithologies.
- Demonstrating BBWI of 8.1 to 24.1 kWh/t, DWi values of 2.14 to 8.00 kWh/m^3, and A×b values of 31.5 to 108.5.
- Confirming production of lead concentrates typically exceeding 50% Pb and zinc concentrates of approximately 50% to 60% Zn.
- Demonstrating significant metallurgical variability within and between lithologies.
- Identifying elevated slurry viscosity and clay-related rheology as key operational risks.
- Confirming that soda ash may partially mitigate rheological challenges.
- Demonstrating that cyanide remains the most effective depressant for maximising Ag recovery.
- Identifying sodium sulphide as the preferred non-cyanide alternative, albeit with lower Ag recovery.
- Demonstrating that process water quality can significantly affect flotation selectivity and concentrate quality.
- Confirming that residual cyanide concentrations in flotation tailings are relatively low and manageable.

13.2.4 2022 Underground and Open Pit Composite Test Program

The metallurgical testwork program initially investigated a three-stage differential flotation flowsheet comprising sequential recovery of Ag, Pb, and Zn minerals. The circuit was designed to include an initial silver flotation stage targeting sulphosalt and copper-associated Ag minerals, followed by conventional Pb flotation and a downstream Zn flotation circuit. This approach was intended to maximise overall Ag recovery by selectively targeting Ag hosted in copper-bearing minerals, such as tetrahedrite–tennantite, which had been identified in earlier mineralogical studies as significant carriers of Ag.

While the differential flowsheet was technically feasible and successfully demonstrated the ability to produce separate concentrates, testwork results highlighted several operational and metallurgical challenges. The circuit required complex reagent regimes, including multiple collectors, depressants, and modifiers applied at different stages of the flowsheet, resulting in difficult process control and high sensitivity to operating conditions such as pH, oxidation state, and grind size. This complexity was further compounded by the need to carefully balance competing flotation responses across stages, particularly where minerals suppressed in one stage were required to be recovered in subsequent stages.

In addition, performance across different ore composites was inconsistent, reflecting variability in mineralogical composition, particularly with respect to the abundance and deportment of silver-bearing sulphosalt and copper minerals. The selective Ag flotation stage often exhibited variable recovery and grade performance, with some composites demonstrating limited response due to differences in mineral liberation or surface chemistry.

A critical issue identified during testing was the deportment of arsenic and other penalty elements (e.g. antimony) associated with tetrahedrite-group minerals. These elements preferentially reported to early flotation concentrates, resulting in elevated impurity levels that would negatively impact concentrate marketability and potentially incur smelter penalties. The combination of arsenic deportment issues, along with variability in metallurgical response, reduced confidence in the robustness of producing multiple saleable concentrates.

Overall, while the differential flotation approach demonstrated metallurgical potential, it was determined to be operationally complex, sensitive to variability, and less robust in terms of consistent performance and concentrate quality. These limitations ultimately prompted further investigation into alternative flowsheet configurations, leading to the development of the simplified bulk flotation strategy adopted in the DFS.

Underground Composite - Rougher Flotation Testwork

Three new composites were created for inclusion in an underground mine expansion SS. Initial underground flotation testwork evaluated a sequential Ag and Pb rougher flotation flowsheet using three representative underground ore composites: Aegean, Bundarra, and North West. The program focused on assessing the impact of primary grind size, reagent regime, collector selection and sulphide depressants on Ag, Pb and arsenic recovery. Overall, the testwork demonstrated that flotation performance is highly ore type dependent, with silver-rich Aegean ores responding favourably to fine grinding, Bundarra ores exhibiting strong Pb flotation characteristics, and North West ores showing consistently high Pb recoveries but elevated Zn and arsenic entrainment.

The rougher flotation program demonstrated that:

- Aegean ore is Ag-dominant and requires fine grinding (38 µm to 45 µm) to maximise liberation and Ag recovery.
- Bundarra ore shows poor Ag flotation response but excellent Pb flotation characteristics, producing high-grade lead concentrates with recoveries exceeding 90%.
- North West ore exhibits consistent lead recovery and moderate Ag recovery, with flotation performance only weakly dependent on grind size.
- Collector 3418A did not provide a consistent metallurgical benefit over SIPX across any of the underground composites.
- NaCN / $ZnSO_4$ was effective in reducing Zn and iron contamination in Pb concentrates and forms the basis for subsequent cleaner-stage optimisation.
- A primary grind size of approximately 45 µm P_{80} was selected as the preferred operating condition for continued testing and flowsheet development.

Underground Composite - Cleaner Flotation Testwork

Cleaner flotation testwork was undertaken to evaluate concentrate upgrading performance following the rougher flotation program. Testing focused on Ag and Pb cleaner stages using a primary grind size of P_{80} 45 µm, with ongoing reagent optimisation aimed at improving concentrate grade and reducing Zn, iron and arsenic contamination. The results confirmed that significant upgrading can be achieved without regrinding; however, impurity rejection, particularly arsenic and Zn, remained a challenge for certain ore types.

The cleaner flotation program confirmed that substantial concentrate upgrading can be achieved for all underground ore types at a primary grind size of 45 µm P_{80}, without requiring immediate regrinding. The principal outcomes were:

- Aegean delivered strong silver upgrading with cleaner upgrade ratios of 3.6 to 4.4, although increased SMBS addition produced little additional benefit.
- Bundarra produced an excellent lead concentrate (>54% Pb) but remained a poor Ag flotation feed.
- North West achieved the best overall cleaner flotation performance, producing exceptionally high silver grades and lead recoveries exceeding 89%.
- Zinc rejection improved through NaCN / $ZnSO_4$ addition, although elevated Zn levels persisted in some concentrates.
- Arsenic remained the most significant concentrate quality risk, particularly for the North West silver concentrates.
- The results justified progression to recleaner-stage testing to further improve concentrate quality and investigate impurity control.

Underground Composite - Recleaner Flotation Testwork

Following completion of the cleaner flotation program, a Pb recleaner test series was conducted on all three underground composites to assess the potential for further concentrate upgrading and impurity rejection. Due to the low mass pull generated in the Ag cleaner concentrates, additional Ag recleaning stages were impractical and testing was limited to the Pb flotation circuit.

Key outcomes of the underground composite recleaner testing were:

- Pb recleaning provided additional concentrate upgrading without requiring regrinding.
- North West achieved the strongest combined performance with Pb recoveries exceeding 90% and Ag recoveries in the mid-80% range.
- Aegean produced high Ag recovery but lower Pb grades due to its low Pb head grade.
- Bundarra remained an excellent Pb flotation feed but generated limited Ag concentrate.
- Duplicate testing confirmed good reproducibility of the selected operating conditions.

Underground Composite - Zinc Flotation

Zinc flotation investigations were conducted only on the Bundarra and North West composites. The Aegean composite was excluded owing to its very low Zn head grade (<0.1% Zn) and the significant proportion of Zn already recovered during the Ag and Pb flotation stages. Under these conditions, insufficient Zn remained available to justify a dedicated Zn flotation circuit.

The Zn flotation programs further refined the underground flowsheet and highlighted important differences between ore sources. Key findings include:

- Zn flotation was highly successful for Bundarra, producing final concentrates exceeding 48.8% Zn at recoveries above 90%.
- North West Zn mineralisation responded poorly to flotation, with substantial Zn losses to tailings and low final concentrate grades.

- Additional mineralogical investigation of North West zinc occurrence is required before further process optimisation is undertaken.

Open Pit Composite – Rougher Flotation Testwork

Initial open pit flotation testwork evaluated a sequential Ag and Pb rougher flotation flowsheet for three open pit ore domains: Main Ag Zone, Central West, and Southern Zone. The program investigated the effects of primary grind size, collector dosage, depressant addition and collector type on Ag and Pb recovery, concentrate quality and impurity rejection. Results showed that the three domains responded differently to flotation, reflecting significant mineralogical variation, particularly with respect to arsenic and Zn deportment.

The open pit rougher flotation program highlighted significant differences in metallurgical response between the Main Ag Zone, Central West and Southern Zone ore domains. Overall, the results demonstrated that finer grinding generally improved flotation performance, particularly for Ag-rich ores, although the magnitude of the benefit varied between ore types. A primary grind size of approximately 45 µm P_{80} was identified as a practical compromise between metallurgical performance and grinding energy requirements. The alternative collector 3418A provided only limited benefits compared with SIPX and did not consistently improve recoveries. The use of NaCN and $ZnSO_4$ proved effective in improving concentrate selectivity within the lead flotation circuit by reducing the recovery of Zn- and iron-bearing sulphides. Across all open pit composites, arsenic and Zn remained the dominant concentrate quality challenges.

Open Pit Composite – Cleaner Flotation Testwork

Cleaner flotation testwork was undertaken using a staged approach to evaluate the upgrading potential of both Ag and Pb rougher concentrates from the open pit composites. Tests assessed the performance of single-stage Ag and Pb cleaners under varying reagent regimes, with ongoing optimisation of collector and depressant additions. Based on the outcomes of the rougher test program, a primary grind size of P_{80} 45 µm was adopted for the Main Ag Zone and Central West composites, while the Southern Zone was retained at a coarser P_{80} 106 µm due to its previously demonstrated flotation response.

The open pit cleaner flotation program demonstrated that significant concentrate upgrading can be achieved for all three composite types without the use of concentrate regrinding. The Main Ag Zone produced the highest Ag concentrate grades, including a Ag cleaner concentrate exceeding 72,000 g/t Ag, while Pb cleaner concentrates consistently exceeded 63% Pb. Central West also delivered strong Ag upgrading; however, exceptionally high arsenic levels remained the major metallurgical challenge despite meaningful arsenic rejection during cleaning. The Southern Zone responded favourably to cleaning at a relatively coarse grind size, with sodium silicate additions improving concentrate quality and supporting higher Ag and Au grades.

The results also highlighted that Zn and arsenic continue to be the principal concentrate quality issues across the open pit ore types. Increased SMBS additions improved Zn rejection in the Main Ag Zone and Central West composites, but complete rejection was not achieved. Similarly, although cleaning reduced arsenic recoveries, concentrate arsenic grades remained elevated where arsenic head grades were high.

Open Pit Composite – Primary Grind Review

During the flotation program, the selected primary grind size was re-evaluated to assess whether a coarser primary grind, combined with subsequent regrinding ahead of the cleaner stages, could provide comparable metallurgical performance while reducing grinding energy requirements.

For both the Main Ag Zone and Central West composites, silver recovery to the silver rougher concentrate showed a clear improvement with decreasing grind size. In the Main Ag Zone, silver rougher recovery increased from approximately 49.3% at P_{80} 150 μm to 79.8% at P_{80} 38 μm, demonstrating that silver recovery is strongly dependent on mineral liberation. A similar trend was observed for the Central West composite, where silver rougher recovery increased from 40.1% at P_{80} 150 μm to approximately 71% to 74% at grind sizes of 38 μm to 53 μm. These results indicate that finer grinding promotes the liberation and recovery of silver-bearing sulphides into the primary silver concentrate.

The Pb content recovered in the silver rougher concentrate also generally increased with reducing grind size, further supporting the conclusion that additional liberation benefits are achieved at finer primary grinds. As silver recovery to the silver rougher concentrate increased at finer grind sizes, the remaining silver available for recovery in the lead rougher stage progressively declined. Consequently, silver recovery to the lead rougher concentrate was generally higher at coarser primary grind sizes. Lead recovery in the lead rougher circuit remained relatively stable across the tested grind sizes.

When the Ag and Pb rougher concentrates were combined, overall silver recovery remained relatively constant at primary grind sizes finer than approximately 106 μm, with a noticeable decline only occurring at the coarsest grind of 150 μm. Combined lead recovery also showed little sensitivity to grind size, varying only marginally across the test range, with approximately a one percent reduction observed at the coarsest grind size. These results suggest that the loss in metallurgical performance associated with a moderately coarser grind may be relatively small when considering the combined rougher circuit performance.

The findings indicate that there is potential to operate at a coarser primary grind, particularly around 75 μm to 106 μm P_{80}, provided that a targeted regrind stage is incorporated within the downstream flowsheet. Overall, the grind review demonstrated that although the highest Ag recoveries were consistently achieved at fine primary grind sizes of 38 μm to 45 μm P_{80}, the combined rougher circuit performance remained relatively robust at grind sizes up to 106 μm P_{80}.

Open Pit Composites – Lead Recleaner Test

A lead recleaner test program was undertaken on the Main Ag Zone and Central West open pit composites to assess the potential for further upgrading of the Pb concentrate and improving overall silver recovery. Due to the low mass pull generated in the silver cleaner concentrates, additional silver recleaning stages were not practical and testing remained focused on the lead flotation circuit. In all cases, silver cleaner tails were combined with the lead rougher feed to maximise recovery of unrecovered precious metals and lead values.

The initial recleaner tests were conducted using the selected primary grind size of P_{80} 45 µm without any intermediate regrinding stage. Subsequent tests investigated the effect of increasing the primary grind size to P_{80} 106 µm and P_{80} 150 µm while maintaining the same circuit configuration. No regrind stage was included after the rougher flotation stage, allowing assessment of the direct effect of primary grind size on cleaner circuit performance.

Results demonstrated that the finer primary grind produced the highest silver recoveries to the combined cleaner concentrates. The reduction in silver recovery at coarser primary grind sizes is consistent with earlier rougher flotation observations, where liberation of the silver-bearing sulphide minerals was shown to be dependent on particle size.

Overall, the lead recleaner testwork confirmed that high combined silver recoveries can be achieved without concentrate regrinding when operating at a fine primary grind size.

Open Pit Composites – Regrind Lead Recleaner Tests

A series of regrind cleaner tests was undertaken on the Main Ag Zone and Central West composites to investigate the potential benefits of incorporating a concentrate regrind stage prior to cleaner flotation. The objective of this program was to assess whether a coarser primary grind could be adopted while maintaining concentrate grade and metal recovery through selective regrinding of the combined rougher concentrates.

Overall, the results support the concept of operating with a coarser primary grind followed by targeted concentrate regrinding, allowing improved liberation and upgrading while potentially reducing overall grinding energy requirements. The Main Ag Zone produced particularly strong outcomes, achieving high Ag recovery and Pb concentrate grades approaching 60% Pb at approximately 90% recovery. Although arsenic remained a significant concentrate quality issue, especially within the Main Ag Zone, the regrind stage improved arsenic rejection and represents an important optimisation opportunity for future flowsheet development and locked-cycle testing.

Open Pit Composites – Zinc Circuit Recleaner Tests

Several zinc recleaner flotation tests incorporating a regrind stage were completed on Central West and MZ composite samples. The testwork followed a flowsheet comprising combined silver-lead rougher flotation, regrinding, cleaner and recleaner flotation, followed by zinc rougher flotation, regrinding and zinc cleaning stages to a final zinc recleaner concentrate.

The testwork demonstrated that while the lead-silver circuit produced strong concentrate grades and recoveries, the zinc recleaner circuit was unable to consistently generate a marketable zinc concentrate from either composite, primarily due to low concentrate grades, poor zinc recoveries and persistent arsenic contamination.

13.2.5 2023 Main, Pit West and Pit South Composite Test Program

A set of nine open pit composites were created targeting a set of silver head grades (100, 60 and 30 g/t) for the MZ, PW and PS zones. The metallurgical testwork program initially investigated a three-stage differential flotation flowsheet comprising sequential recovery of Ag, Pb, and Zn minerals. The circuit was designed to include an initial silver flotation stage targeting sulphosalt and copper-associated silver minerals, followed by conventional lead flotation and a downstream zinc flotation circuit. This approach was intended to maximise overall silver recovery by selectively targeting silver hosted in copper-bearing minerals, such as tetrahedrite–tennantite, which had been identified in earlier mineralogical studies as significant carriers of silver.

Rougher Flotation Testwork

A series of nine sighter flotation tests was using reagent conditions developed during the underground test programs. The flowsheet incorporated SMBS as a sphalerite depressant in the silver rougher stage, sodium carbonate (Na_2CO_3) for pH control, and sodium cyanide (NaCN) together with zinc sulphate ($ZnSO_4$) as depressants in the lead rougher stage. A3894 was used as the silver collector, 3418A as the lead collector and methyl iso butyl carbinol (MIBC) was used as the frother throughout both flotation circuits.

Overall, the sighter test program demonstrated that the adopted reagent suite and flotation conditions were capable of achieving combined silver recoveries exceeding 80% for the higher-grade composites. The results further confirmed the strong influence of silver head grade on overall recovery performance and provided a sound basis for the optimisation testwork that followed.

Recleaner Flotation Testwork

A series of recleaner flotation tests was conducted using the reagent scheme developed during the previous test program. The flotation conditions included SMBS as a sphalerite depressant in the silver rougher stage, Na_2CO_3 for pH control, and NaCN together with $ZnSO_4$ as depressants in the lead flotation stage. Collectors comprised A3894 in the silver circuit and 3418A in the lead circuit, with MIBC used as the frother throughout. During the cleaner and recleaner stages, both A3894 and 3418A collectors were added in conjunction with NaCN and $ZnSO_4$ depressants.

The testwork demonstrated generally strong lead flotation performance across all composite types. The relationship between lead grade and recovery was favourable across most samples, with the majority of final recleaner concentrates producing grades exceeding 35% Pb.

The silver grade-recovery relationship indicates that high silver recoveries can be maintained while upgrading the concentrate through successive cleaning stages. The results confirm the effectiveness of the Ag-Pb flotation strategy while also highlighting opportunities for further optimisation through improved liberation.

Despite the positive Pb and Ag results, Zn rejection remained problematic across all composite types at the coarse grind size tested. Appreciable amounts of zinc continued to report to the Ag-Pb concentrates, indicating that sphalerite depression was not fully effective under the tested grinding conditions. The persistence of zinc in the concentrate products suggests that additional liberation through regrinding may be necessary to improve gangue rejection and concentrate quality.

Overall, the recleaner test program demonstrated the ability to produce upgraded silver-lead concentrates with high lead recoveries (>75%) and excellent silver recovery in the higher-grade composites, while identifying coarse grind liberation as the principal constraint affecting Ag recovery, Zn rejection and final concentrate quality.

Recleaner Flotation Testwork – Reagent Optimisation without Regrind

A series of nine recleaner flotation tests was undertaken at a primary grind size of P_{80} 106 µm to assess the impact of reagent optimisation on concentrate grade, recovery and selectivity. The test program retained the existing rougher-cleaner-recleaner flowsheet but did not incorporate a regrind stage. Modified reagent conditions included increased SMBS addition in the silver rougher, increased NaCN and $ZnSO_4$ addition in the lead rougher and cleaning stages, a longer lead rougher flotation time, and exclusion of the second lead rougher concentrate from the combined cleaner feed in selected tests.

Overall, the reagent optimisation program demonstrated that increasing depressant dosages and removing the second lead rougher concentrate from the cleaner feed did not produce a consistent improvement in flotation performance across the composite suite. While some increases in lead and silver concentrate grades were achieved, these were generally accompanied by reductions in metal recovery. The most pronounced negative effect was observed in the PS composites, while the MZ and PW composites showed only limited benefits. The results indicate that the original reagent regime provided a better balance between concentrate grade and recovery and that reagent optimisation alone was insufficient to significantly improve metallurgical performance in the absence of a regrind stage.

Conclusions

While the differential flowsheet was technically feasible and successfully demonstrated the ability to produce separate concentrates, testwork results highlighted several operational and metallurgical challenges. The circuit required complex reagent regimes, including multiple collectors, depressants, and modifiers applied at different stages of the flowsheet, resulting in difficult process control and high sensitivity to operating conditions such as pH, oxidation state, and grind size. This complexity was further compounded by the need to carefully balance competing flotation responses across stages, particularly where minerals suppressed in one stage were required to be recovered in subsequent stages.

In addition, performance across different ore composites was inconsistent, reflecting variability in mineralogical composition, particularly with respect to the abundance and deportment of silver-bearing sulphosalt and copper minerals. The selective silver flotation stage often exhibited variable recovery and grade performance, with some composites demonstrating limited response due to differences in mineral liberation or surface chemistry.

An issue identified during testing was the deportment of arsenic and other penalty elements (e.g. antimony) associated with tetrahedrite-group minerals. These elements preferentially reported to early flotation concentrates, resulting in elevated impurity levels that would negatively impact concentrate marketability and potentially incur smelter penalties.

JK 2017 Reagent Scheme Flotation Tests (T52 to T58)

A series of flotation tests was completed on the 2023 composites using reagent conditions developed during the 2017 JK flotation test program. The objective was to evaluate a simplified flotation flowsheet capable of producing separate lead and zinc concentrates without a dedicated silver flotation stage. This approach revisited the historical reagent scheme and assessed its suitability for the current MZ, PW and PScomposite samples.

The tests were conducted at a primary grind size of P_{80} 106 µm. Lime was used as the pH modifier, added during primary grinding at a fixed dosage, resulting in flotation pH values ranging from approximately 6.7 to 9.2 across the test series. In the lead flotation circuit, NaCN and $ZnSO_4$ were used to depress sphalerite and pyrite, while 3418A served as the collector and MIBC as the frother. Lead rougher flotation was conducted for 7 min, followed by a lead cleaner stage of 9 min.

Zinc flotation was undertaken on the lead circuit tailings using a fixed addition of 200 g/t $CuSO_4$ as the sphalerite activator, regardless of feed grade. Cytec A3477 was used as the zinc collector and MIBC as the frother. Lime addition was restricted to the zinc regrind stage, increasing pulp pH from approximately 7.8 to greater than 11.8 before zinc cleaning. No pH adjustment was made between the Pb and Zn rougher stages.

Overall, the JK 2017 reagent scheme successfully produced highly upgraded silver-bearing lead concentrates while simplifying the flowsheet by eliminating the separate silver flotation stage. The results confirm the strong silver upgrading potential of the lead circuit; however, they also highlight variable silver deportment, with some silver reporting to the zinc concentrate where silver is associated with sphalerite or arsenic-bearing sulphides. Arsenic rejection remained limited, while zinc depression was generally effective under the conditions tested.

Additional Zinc Cleaner Testwork

A series of zinc cleaner tests was completed to evaluate the production of a final zinc concentrate using a flowsheet comprising silver rougher flotation, lead rougher flotation, combined Ag / Pb cleaning without regrinding, and a dedicated zinc recovery circuit. The zinc flotation feed consisted of the lead rougher tailings, Ag / Pb cleaner tailings and Ag / Pb recleaner tailings, while the zinc rougher concentrate was reground prior to zinc cleaning.

The results demonstrate that a marketable zinc concentrate could be produced from most composite samples following regrinding and two-stage cleaning.

Although the zinc flotation circuit successfully recovered zinc into a separate concentrate, it also recovered a proportion of the contained silver. Approximately 10% to 15% of the total silver reported to the final zinc concentrate, indicating that some silver minerals remained associated with sphalerite or other zinc-bearing sulphides rejected from the Ag-Pb circuit.

13.2.6 2024 Bulk Flotation Flowsheet Development

Following completion of the sequential Ag-Pb and Zn flotation investigations, a comprehensive bulk flotation test program evaluated the potential for producing a single bulk Ag / Pb / Zn concentrate using the same nine composites used in the 2023 program. The revised approach was driven by a commercial assessment which indicated that a high-silver grade bulk concentrate would be economically attractive, even with significant zinc-related smelter penalties. The bulk flotation strategy offered the potential to simplify the process flowsheet by eliminating the need for separate Ag-Pb and Zn concentrate products while maintaining strong recoveries of valuable metals.

The test program was undertaken in three stages comprising batch rougher and cleaner flotation development, bulk recleaner optimisation and locked-cycle verification testing. Initial work focused on reagent selection, flotation kinetics, pH control and cleaning strategies using a MZ Master composite, before expanding to variability testing on MZ, PW and PS composites spanning a range of silver head grades. Subsequent optimisation included finer regrinding, additional cleaning stages and reagent modifications aimed at improving concentrate grade, recovery and selectivity. The final phase of testing involved locked-cycle operation to simulate continuous plant performance using site water and to establish design criteria for the proposed processing facility.

Collectively, the testwork demonstrated that a bulk Ag / Pb / Zn flotation flowsheet is capable of producing a low-mass, high-value concentrate at consistently high Ag, Pb and Zn recoveries across a range of ore types. While arsenic deportment will likely result in some penalties, the testwork confirmed the technical viability of the bulk concentrate strategy and established the basis for the preferred flotation flowsheet adopted for process plant design. Results from the bulk rougher and cleaner tests, bulk recleaner optimisation program and locked-cycle testwork are discussed in the following sections.

Bulk Rougher and Cleaner Flotation Testwork

Following review of the combined Ag / Pb-Zn flotation programme, the flotation strategy was modified to investigate the production of a bulk Ag / Pb / Zn concentrate for the open pit ores. This change was driven by an assessment undertaken by Bowdens which indicated that a high silver-arsenic bulk concentrate could remain commercially viable, even with a penalty equivalent to approximately 50% of the contained zinc value under anticipated smelter terms.

Consequently, a new MZ Master composite was blended from all three MZ composites, all three PW composites and the PS 100 composite. PS 60 and PS 30 composites were not included as they represent mine zones that are not within the 2024 optimisation mining inventory. The MZ Master composite was utilised for bulk flotation development and optimisation. Initial bulk flotation Tests T59 and T60 investigated the concept, while Tests T61 to T65 focused on reagent and circuit optimisation. Kinetic flotation investigations were subsequently completed in Tests T66 to T69. Test conditions are summarised in Table 13.2.1, with metallurgical results presented in Table 13.2.2.

Table 13.2.1 Bulk Rougher Flotation Test Conditions

Test	Sample	Conditions
T59	MZ Master	Bulk Ag / Pb / Zn rougher at pH 8.5 with $CuSO_4$ / SIBX / MIBC
T60	MZ Master	Bulk Ag / Pb / Zn rougher at pH 8.5 with $CuSO_4$ / SIBX / MIBC
T61	MZ Master	As per Test 46 but with Zn rougher at natural pH 8 - 9 and higher $CuSO_4$ addition rate plus Zn cleaner / recleaner at pH 11 adjusted with lime
T62	MZ Master	As per Test 61 but with A4037 instead of SIBX in all Zn flotation stages
T63	MZ Master	As per Test 62 but with additional regrind of Zn rougher conc 1+2 prior Zn cleaner / recleaner and lower collector addition rate in Zn roughers
T64	MZ Master	As per Test 63 but with Na_2CO_3 as pH adjustment in Zn flotation
T65	MZ Master	As per Test 64 but with collector SIBX instead of A4037 in Zn flotation stages
T66	MZ Master	As per Test 59 but with collector A4037 instead of SIBX and one additional rougher stage
T67	MZ Master	As per Test 59 but with collector A4037 instead of SIBX and one additional rougher stage
T68	MZ Master	As per Test 66 but with Na_2CO_3 instead of lime and at pH 10 instead of 8.5 in rougher 1
T69	MZ Master	As per Test 66 with addition of Na_2SiO_3 and higher pH regulated with lime and additional 100 g/t $CuSO_4$ addition in rougher 3

Table 13.2.2 Bulk Rougher Flotation Test Results

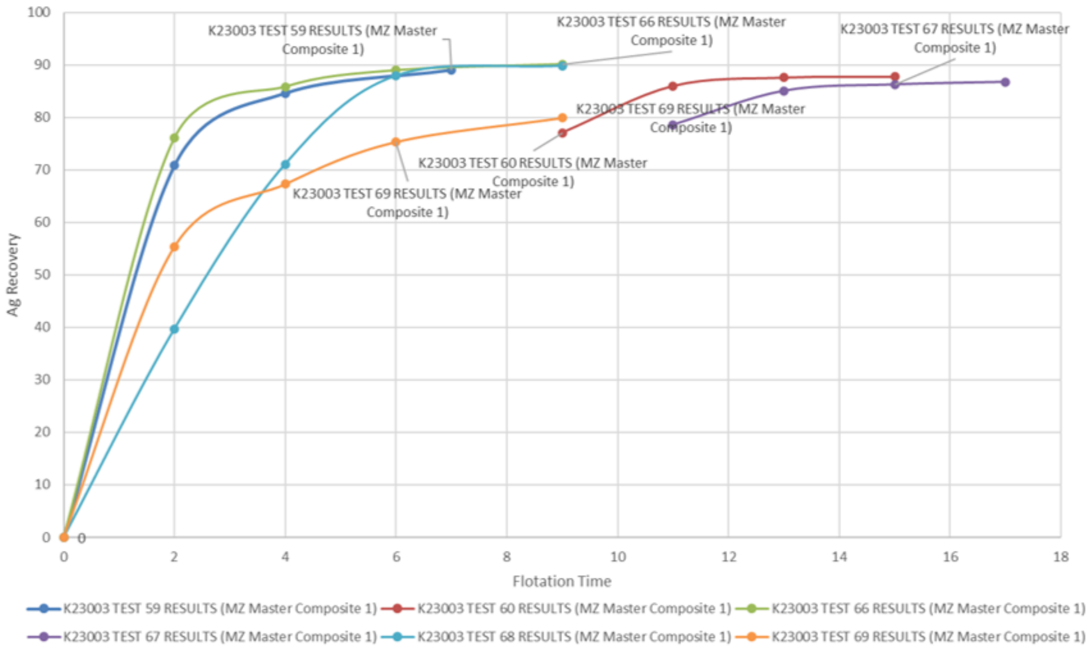
		Stream	Size µm	%wt	Ag g/t	Ag Rec	As g/t	As Rec	Pb %	Pb Rec	Zn %	Zn Rec
59	K23003 TEST 59 RESULTS (MZ Master Composite 1)	Ag / Pb / Zn Rougher Conc 1	106	2.86	2,970	70.8	1,740	37.0	9.92	76.5	9.7	76.3
		(Ag / Pb / Zn Rougher Conc 1-2)		4.41	2,298	84.6	1,550	50.8	7.39	87.9	7.1	86.7
		(Ag / Pb / Zn Rougher Conc 1-3)		5.69	1,875	89.0	1,368	57.9	5.94	91.1	5.7	89.6
		(Calculated Head)		100.00	120	100.0	134	100.0	0.37	100.0	0.4	100.0
60	K23003 TEST 60 RESULTS (MZ Master Composite 1)	Ag / Pb / Zn Cleaner Conc 1	106	1.97	4,630	77.1	2,640	62.2	14.50	79.6	12.2	80.0
		(Ag / Pb / Zn Cleaner Conc 1-2)		2.52	4,048	86.0	2,445	73.5	12.51	87.6	10.6	88.4
		(Ag / Pb / Zn Cleaner Conc 1-3)		2.75	3,777	87.6	2,315	75.9	11.59	88.6	9.8	89.5
		(Ag / Pb / Zn Cleaner Conc 1-4)		2.82	3,690	87.8	2,268	76.3	11.32	88.7	9.6	89.6
		(Ag / Pb / Zn Rougher Conc 1-3)		4.97	2,113	88.8	1,302	77.3	6.56	90.7	5.5	90.5
		(Calculated Head)		100.00	118	100.0	84	100.0	0.36	100.0	0.3	100.0
61	K23003 TEST 61 RESULTS (MZ Master Composite 1)	Ag / Pb Re-Recleaner Conc 1-2	106 / 45	0.61	13,340	71.1	3,200	23.8	46.20	82.7	9.5	18.6
		(Ag / Pb Recleaner Conc 1-2)		0.63	12,876	71.6	3,123	24.3	44.64	83.4	9.3	19.1
		Zn Recleaner Conc 1-2		1.34	1,348	15.9	2,880	47.4	1.22	4.8	16.0	69.1
		(Zn Cleaner Conc 1-2)		1.58	1,157	16.1	2,502	48.5	1.06	4.9	13.6	69.3
		(Zn Rougher Conc 1-2)		3.23	585	16.6	1,317	52.2	0.56	5.3	6.7	70.0
		(Calculated Head)		100.00	114	100.0	82	100.0	0.34	100.0	0.3	100.0
62	K23003 TEST 62 RESULTS (MZ Master Composite 1)	Ag / Pb Re-Recleaner Conc 1-2	106 / 45	0.64	13,920	71.0	3,400	38.5	48.80	78.6	9.9	20.2
		(Ag / Pb Recleaner Conc 1-2)		0.68	13,186	71.9	3,287	39.8	46.36	79.8	9.7	21.0
		Zn Recleaner Conc 1-2		0.59	2,860	13.4	1,800	18.7	2.60	3.8	34.6	64.5
		(Zn Cleaner Conc 1-2)		0.87	1,961	13.7	1,364	21.1	1.80	4.0	23.3	64.7
		(Zn Rougher Conc 1-2)		2.58	691	14.2	568	26.0	0.67	4.3	7.9	65.1
		(Calculated Head)		100.00	125	100.0	57	100.0	0.40	100.0	0.3	100.0

		Stream	Size µm	%wt	Ag g/t	Ag Rec	As g/t	As Rec	Pb %	Pb Rec	Zn %	Zn Rec
63	K23003 TEST 63 RESULTS (MZ Master Composite 1)	Ag / Pb Re-Recleaner Conc 1-2	106 / 45	0.64	15,620	74.0	4,000	29.8	48.60	83.6	9.6	16.9
		(Ag / Pb Recleaner Conc 1-2)		0.68	14,891	75.1	3,869	30.6	46.48	85.1	9.3	17.5
		Zn Recleaner Conc 1-2		0.52	2,880	11.2	2,920	17.8	2.46	3.5	42.6	61.5
		(Zn Cleaner Conc 1-2)		0.90	1,734	11.7	2,335	24.7	1.51	3.7	24.6	61.7
		(Zn Rougher Conc 1-2)		3.00	600	13.4	1,331	46.8	0.55	4.4	7.5	62.4
		(Calculated Head)		100.00	134	100.0	85	100.0	0.37	100.0	0.4	100.0
64	K23003 TEST 64 RESULTS (MZ Master Composite 1)	Ag / Pb Re-Recleaner Conc 1-2	106 / 45	0.60	14,020	75.6	2,960	23.1	50.40	86.5	9.8	20.0
		(Ag / Pb Recleaner Conc 1-2)		0.63	13,450	76.0	2,888	23.7	48.41	87.1	9.6	20.6
		Zn Recleaner Conc 1-2		0.48	2,880	12.3	2,520	15.6	2.84	3.9	40.4	65.2
		(Zn Cleaner Conc 1-2)		0.63	13,450	12.5	2,106	17.3	2.22	4.0	30.5	65.5
		(Zn Rougher Conc 1-2)		1.83	815	13.4	1,065	25.5	0.86	4.5	10.6	66.2
		(Calculated Head)		100.00	111	100.0	77	100.0	0.35	100.0	0.3	100.0
65	K23003 TEST 65 RESULTS (MZ Master Composite 1)	Ag / Pb Re-Recleaner Conc 1-2	106 / 45	0.59	14,360	73.6	2,960	25.1	51.60	83.6	9.4	17.8
		(Ag / Pb Recleaner Conc 1-2)		0.63	13,710	74.0	2,873	25.6	49.28	84.1	9.2	18.3
		Zn Recleaner Conc 1-2		1.28	1,550	17.1	3,060	55.7	1.78	6.2	17.3	70.3
		(Zn Cleaner Conc 1-2)		1.43	1,394	17.2	2,778	56.6	1.61	6.3	15.5	70.4
		(Zn Rougher Conc 1-2)		2.71	757	17.7	1,569	60.6	0.91	6.7	8.2	70.9
		(Calculated Head)		100.00	116	100.0	70	100.0	0.37	100.0	0.3	100.0
66	K23003 TEST 66 RESULTS (MZ Master Composite 1)	Ag / Pb / Zn Rougher Conc 1	106	2.26	3,890	76.1	1,580	20.5	11.60	74.8	11.1	80.3
		(Ag / Pb / Zn Rougher Conc 1-2)		3.24	3,060	85.8	1,507	28.1	9.14	84.5	8.5	87.9
		(Ag / Pb / Zn Rougher Conc 1-3)		4.14	2,481	89.0	1,362	32.4	7.43	87.8	6.8	89.9
		(Ag / Pb / Zn Rougher Conc 1-4)		4.94	2,107	90.1	1,213	34.4	6.32	89.1	5.7	90.9
		(Calculated Head)		100.00	116	100.0	174	100.0	0.35	100.0	0.3	100.0

		Stream	Size µm	%wt	Ag g/t	Ag Rec	As g/t	As Rec	Pb %	Pb Rec	Zn %	Zn Rec
67	K23003 TEST 67 RESULTS (MZ Master Composite 1)	Ag / Pb / Zn Cleaner Conc 1	106 / 45	1.42	6,560	78.6	2,560	18.7	19.40	79.0	18.9	83.2
		(Ag / Pb / Zn Cleaner Conc 1-2)		1.78	5,680	85.1	2,592	23.7	16.30	83.0	15.7	86.5
		(Ag / Pb / Zn Cleaner Conc 1-3)		1.96	5,231	86.3	2,547	25.6	14.91	83.6	14.4	86.9
		(Ag / Pb / Zn Cleaner Conc 1-4)		2.07	4,974	86.7	2,501	26.6	14.13	83.8	13.6	87.1
		(Ag / Pb / Zn Rougher Conc 1-3)		4.49	2,395	90.4	1,360	31.3	6.94	89.1	6.4	88.2
		(Calculated Head)		100.00	119	100.0	195	100.0	0.35	100.0	0.3	100.0
68	K23003 TEST 68 RESULTS (MZ Master Composite 1)	Ag / Pb / Zn Rougher Conc 1	106	0.82	5,550	39.7	1,300	5.5	18.50	43.9	21.3	54.9
		(Ag / Pb / Zn Rougher Conc 1-2)		1.87	4,345	71.1	1,615	15.6	13.66	74.2	13.6	80.0
		(Ag / Pb / Zn Rougher Conc 1-3)		3.10	3,238	88.0	1,728	27.7	9.61	86.6	9.1	89.6
		(Ag / Pb / Zn Rougher Conc 1-4)		3.75	2,733	89.9	1,557	30.2	8.14	88.8	7.7	90.9
		(Calculated Head)		100.00	114	100.0	193	100.0	0.34	100.0	0.3	100.0
69	K23003 TEST 69 RESULTS (MZ Master Composite 1)	Ag / Pb / Zn Rougher Conc 1	106	1.59	3,980	55.4	1,560	14.2	5.51	25.9	6.1	30.0
		(Ag / Pb / Zn Rougher Conc 1-2)		2.61	2,950	67.3	1,443	21.6	5.05	38.9	5.1	41.4
		(Ag / Pb / Zn Rougher Conc 1-3)		3.60	2,392	75.4	1,420	29.3	4.61	49.0	5.8	65.0
		(Ag / Pb / Zn Rougher Conc 1-4)		4.60	1,986	79.9	1,303	34.3	4.14	56.3	5.2	74.7
		(Calculated Head)		100.00	114	100.0	174	100.0	0.34	100.0	0.3	100.0

The flotation kinetics (Test 59 – 69) are illustrated in Figure 13.2.5, which demonstrates the recovery response of the bulk flotation circuit through successive rougher and cleaner stages. The results show that high silver recoveries were maintained throughout the optimisation programme, with cleaner-stage silver recoveries consistently ranging between approximately 85% and 87%. Test T67, which utilised a total flotation residence time of 17 min, demonstrated that high silver recoveries could be achieved without significant losses in the cleaner circuit, indicating that the bulk flotation approach was capable of maintaining strong silver recovery while simplifying the overall flowsheet.

Figure 13.2.5 Bulk Rougher / Cleaner Flotation Kinetics



Overall, the bulk flotation test programme confirmed the viability of recovering Ag, Pb, and Zn into a single bulk concentrate product, substantially simplifying the process flowsheet compared with the previously investigated sequential Ag / Pb and Zn flotation circuits. The approach consistently achieved silver recoveries in excess of 85%, while supporting the commercial objective of producing a marketable bulk concentrate despite elevated arsenic content and partial zinc payment penalties.

Bulk Recleaner Flotation Testwork

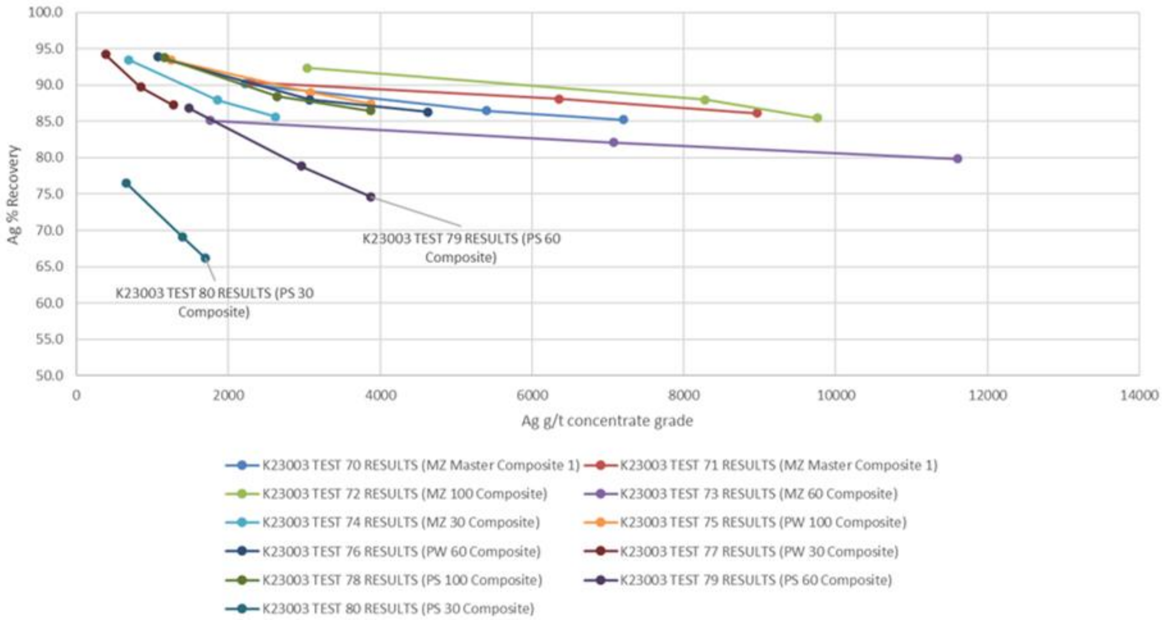
A series of bulk Ag / Pb / Zn flotation tests incorporating cleaner and recleaner stages was completed on the MZ Master composite and the MZ, PW and PS composite suites across a range of Ag head grades (100, 60 and 30 composites). The program aimed to optimise the bulk flotation flowsheet through an increased regrind residence time, with the regrind stage extended from 4 min to 8 min. Based on lock-cycle grinding measurements, the longer regrind period is expected to have produced a product size in the order of P_{80} 20 to 30 µm, improving mineral liberation prior to cleaning. Metallurgical results are reported in Table 13.2.3 and Figure 13.2.6. Note that while tests were conducted on PS 60 and PS 30 composites, they are not in the mining inventory.

Table 13.2.3 Bulk Recleaner Flotation Test Results

		Stream	Size µm	%wt	Ag g/t	Ag Rec	As g/t	As Rec	Pb %	Pb Rec	Zn %	Zn Rec
70	K23003 TEST 70 RESULTS (MZ Master Composite 1)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	1.38	7,203	85.2	3,155	23.5	22.4	87.6	21.0	88.2
		(Ag / Pb / Zn Cleaner Conc 1-4)		1.86	5,400	86.5	2,760	27.8	16.79	88.7	15.6	88.7
		(Ag / Pb / Zn Rougher Conc 1-4)		4.73	2,217	90.2	1,487	38.1	6.85	91.9	6.2	89.8
		(Calculated Head)		100.00	116	100.0	185	100.0	0.35	100.0	0.3	100.0
71	K23003 TEST 71 RESULTS (MZ Master Composite 1)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	1.14	8,961	86.2	2,855	23.5	26.37	85.3	25.0	89.7
		(Ag / Pb / Zn Cleaner Con 1-4)		1.65	6,355	88.1	2,611	31.1	18.64	87.0	17.5	90.3
		(Ag / Pb / Zn Rougher Con 1-4)		4.69	2,293	90.1	1,333	45.1	6.73	89.2	6.2	91.0
		(Calculated Head)		100.00	119	100.0	139	100.0	0.35	100.0	0.3	100.0
72	K23003 TEST 72 RESULTS (MZ 100 Composite)	(Ag / Pb / Zn Recleaner Con 1-4)	106 / 45	0.99	9,755	85.4	1,293	17.5	29.11	80.6	28.1	93.5
		(Ag / Pb / Zn Cleaner Con 1-4)		1.21	8,279	87.9	1,460	24.0	25.07	84.2	23.4	94.3
		(Ag / Pb / Zn Rougher Conc 1-4)		3.45	3,037	92.3	1,004	47.3	9.00	86.5	8.2	95.2
		(Calculated Head)		100.00	113	100.0	73	100.0	0.36	100.0	0.3	100.0
73	K23003 TEST 73 RESULTS (MZ 60 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	0.53	11,610	79.9	1,892	12.5	24.46	80.4	26.2	92.5
		(Ag / Pb / Zn Cleaner Conc 1-4)		0.90	7,078	82.1	2,074	23.2	14.88	82.5	15.7	93.2
		(Ag / Pb / Zn Rougher Conc 1-4)		3.76	1,756	85.1	1,120	52.3	3.68	85.2	3.8	93.6
		(Calculated Head)		100.00	78	100.0	81	100.0	0.16	100.0	0.2	100.0
74	K23003 TEST 74 RESULTS (MZ 30 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	0.96	2,622	85.6	1,578	21.1	15.15	81.3	27.6	95.3
		(Ag / Pb / Zn Cleaner Conc 1-4)		1.38	1,861	87.9	1,492	28.8	10.63	82.5	19.1	95.7
		(Ag / Pb / Zn Rougher Conc 1-4)		4.00	685	93.4	830	46.4	3.86	86.5	6.7	96.5
		(Calculated Head)		100.00	29	100.0	72	100.0	0.18	100.0	0.3	100.0
75	K23003 TEST 75 RESULTS (PW 100 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	1.92	3,870	87.4	12,863	26.1	29.12	82.8	29.5	95.6
		(Ag / Pb / Zn Cleaner Conc 1-4)		2.46	3,076	89.0	15,700	40.8	23.08	84.1	23.1	96.0
		(Ag / Pb / Zn Rougher Conc 1-4)		6.37	1,247	93.4	12,201	82.2	9.28	87.5	9.0	96.8
		(Calculated Head)		100.00	85	100.0	946	100.0	0.68	100.0	0.6	100.0

		Stream	Size µm	%wt	Ag g/t	Ag Rec	As g/t	As Rec	Pb %	Pb Rec	Zn %	Zn Rec
76	K23003 TEST 76 RESULTS (PW 60 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	1.15	4,625	86.3	5,471	22.7	11.84	67.2	24.1	85.7
		(Ag / Pb / Zn Cleaner Conc 1-4)		1.77	3,070	88.0	4,881	31.1	7.89	68.8	15.8	86.3
		(Ag / Pb / Zn Rougher Conc 1-4)		5.41	1,070	93.9	2,671	52.2	2.78	74.3	5.3	88.3
		(Calculated Head)		100.00	62	100.0	277	100.0	0.20	100.0	0.3	100.0
77	K23003 TEST 77 RESULTS (PW 30 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	2.18	1,275	87.2	10,833	26.9	18.03	92.7	31.2	96.4
		(Ag / Pb / Zn Cleaner Conc 1-4)		3.38	843	89.7	12,107	46.7	11.74	93.9	20.2	96.9
		(Ag / Pb / Zn Rougher Conc 1-4)		7.82	383	94.2	8,154	72.7	5.17	95.6	8.8	97.4
		(Calculated Head)		100.00	32	100.0	878	100.0	0.42	100.0	0.7	100.0
78	K23003 TEST 78 RESULTS (PS 100 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	2.59	3,781	86.5	3,806	19.1	47.27	87.0	16.2	92.4
		(Ag / Pb / Zn Cleaner Conc 1-4)		3.88	2,642	88.4	3,644	27.4	32.13	88.6	10.9	93.1
		(Ag / Pb / Zn Rougher Conc 1-4)		9.38	1,158	93.7	3,372	61.3	13.54	90.3	4.6	94.0
		(Calculated Head)		100.00	116	100.0	516	100.0	1.41	100.0	0.5	100.0
79	K23003 TEST 79 RESULTS (PS 60 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	1.13	3,871	74.6	32,854	32.5	7.25	57.5	6.9	75.0
		(Ag / Pb / Zn Cleaner Conc 1-4)		1.56	2,958	78.8	30,159	41.3	5.53	60.5	5.2	78.5
		(Ag / Pb / Zn Rougher Conc 1-4)		3.41	1,484	86.8	19,733	59.2	2.75	66.0	22.5	81.3
		(Calculated Head)		100.00	58	100.0	1,137	100.0	0.14	100.0	0.1	100.0
80	K23003 TEST 80 RESULTS (PS 30 Composite)	(Ag / Pb / Zn Recleaner Conc 1-4)	106 / 45	1.27	1,699	66.2	2,112	4.8	17.40	75.6	38.3	93.9
		(Ag / Pb / Zn Cleaner Conc 1-4)		1.62	1,393	69.1	3,598	10.4	14.16	78.4	30.4	95.0
		(Ag / Pb / Zn Rougher Conc 1-4)		3.81	657	76.5	3,588	24.4	6.57	85.3	13.1	96.3
		(Calculated Head)		100.00	33	100.0	560	100.0	0.29	100.0	0.5	100.0

Figure 13.2.6 Bulk Rougher / Cleaner / Recleaner – Ag Grade-Recovery Curve



Locked-Cycle Testwork

Locked-cycle testing was undertaken on the MZ Master composite 1 using the established flotation conditions, which had previously demonstrated strong bulk concentrate performance. The flowsheet comprised four stages of bulk Ag / Pb / Zn rougher flotation followed by regrinding of the combined rougher concentrate and subsequent Ag / Pb / Zn cleaner and recleaner flotation stages. The locked-cycle test was conducted over six cycles, with the recleaner tailings recycled to the subsequent cycle to simulate steady-state plant operation. Rougher tailings and cleaner tailings were discarded as final tailings and were not recycled. The locked-cycle flowsheet configuration is illustrated in Figure 13.2.7, while results are reported in Table 13.2.4.

The locked-cycle program was conducted using site water rather than Adelaide tap water, which had been used for much of the preceding batch testwork. Comparison with earlier results suggested that the site water may have had a slight detrimental effect on both concentrate grade and recovery.

Figure 13.2.7 Locked-Cycle Configuration

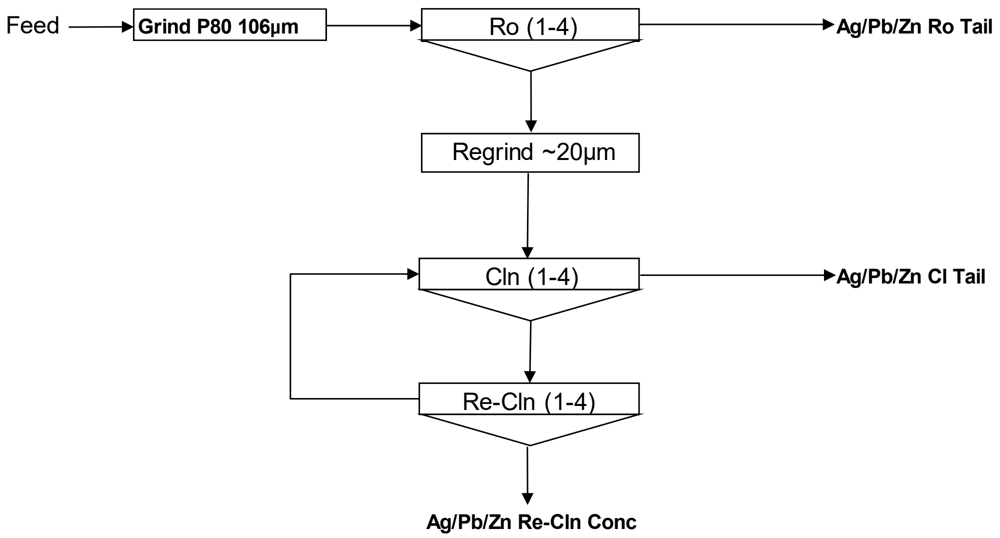


Table 13.2.4 Locked-Cyle Flotation Test Results

Lock Cycle Composite Final Recleaner Concentrate (Cyc 1-6)	Weight %	Assay Ag ppm	As ppm	Cu %	Fe %	Pb %	Sb ppm	Total S %	Zn %
K23003 Test 101 Results (PS 30 Composite)	1.36	1,344	2,091	0.21	8.9	20.4	376	31.5	44.5
K23003 Test 100 Results (PS 60 Composite)	1.39	2,948	28,569	0.16	37.7	5.6	609	42.9	5.3
K23003 Test 99 Results (PS 100 Composite)	2.81	3,519	3,647	0.67	6.7	44.5	1,524	21.3	16.3
K23003 Test 98 Results A (PW 30 Composite)	2.30	**1,170**	11,379	0.12	9.8	16.7	502	27.2	35.9
K23003 Test 97 A Results (PW 60 Composite)	1.20	**4,193**	5,151	0.19	17.8	11.8	1,094	31.8	27.3
K23003 Test 96 Results A (PW 100 Composite)	2.14	**3,346**	13,561	0.15	9.4	27.5	1,131	25.8	28.2
K23003 Test 95 A Results (MZ 30 Composite)	1.20	4,193	5,151	0.19	17.8	11.8	1,094	31.8	27.3
K23003 Test 94 Results B (MZ 60 Composite)	0.69	**8,382**	1,938	0.17	13.6	18.8	926	23.2	20.4
K23003 Test 93 Results A (MZ 100 Composite)	1.16	**10,097**	1,262	0.26	8.5	27.4	987	23.2	26.2
K23003 Test 92 Results A (MZ Master Composite 1)	1.21	**8,642**	2,852	0.20	11.6	24.1	736	24.9	23.2

Lock Cycle Composite Final Recleaner Concentrate (Cyc 1-6)	Distribution, % Ag	As	Cu	Fe	Pb	Sb	Total S	Zn
K23003 Test 101 Results (PS 30 Composite)	60.5	4.5	30.8	8.9	81.5	33.5	34.6	96.6
K23003 Test 100 Results (PS 60 Composite)	71.7	31.6	21.7	30.1	56.4	40.9	40.4	71.1
K23003 Test 99 Results (PS 100 Composite)	80.6	19.1	65.7	8.0	90.6	76.2	21.6	87.9
K23003 Test 98 Results A (PW 30 Composite)	**75.2**	27.6	31.4	11.2	87.4	55.2	37.3	96.4
K23003 Test 97 A Results (PW 60 Composite)	**79.0**	21.5	24.8	10.5	77.3	39.6	35.8	85.2
K23003 Test 96 Results A (PW 100 Composite)	**82.7**	27.0	34.3	10.0	88.4	69.7	31.2	97.6
K23003 Test 95 A Results (MZ 30 Composite)	79.0	21.5	242.8	10.5	77.3	39.6	35.8	85.2
K23003 Test 94 Results B (MZ 60 Composite)	**78.1**	19.6	12.7	11.7	76.1	57.4	30.0	83.8
K23003 Test 93 Results A (MZ 100 Composite)	**86.4**	21.0	18.9	13.5	78.5	55.1	43.6	87.3
K23003 Test 92 Results A (MZ Master Composite 1)	**88.0**	43.8	25.5	16.7	87.1	70.9	39.1	89.9

The relationship between silver head grade and silver recovery is presented in Figure 13.2.8. The data indicates a positive correlation that can be represented by a power function, with higher Ag feed grades generally producing improved silver recoveries. However, the correlation coefficient was relatively weak due to the limited number of data points available.

Final concentrate grade-recovery relationships for Ag is shown in Figure 13.2.9. Overall, the lock cycle programme confirmed the effectiveness and robustness of the bulk Ag / Pb / Zn flotation flowsheet, supporting its selection as the preferred basis for plant design. High recoveries of Ag, Pb and Zn were consistently achieved, with the MZ composites producing the most favourable concentrate quality. While arsenic remains a significant challenge, particularly for the PW and PS ore domains, blending strategies appear capable of mitigating its impact on final concentrate specifications. The lock cycle results therefore provide strong support for adoption of the bulk concentrate flowsheet as the preferred processing route for the open-pit resource.

Figure 13.2.8 Locked-Cycle Ag Head Grade vs. Recovery

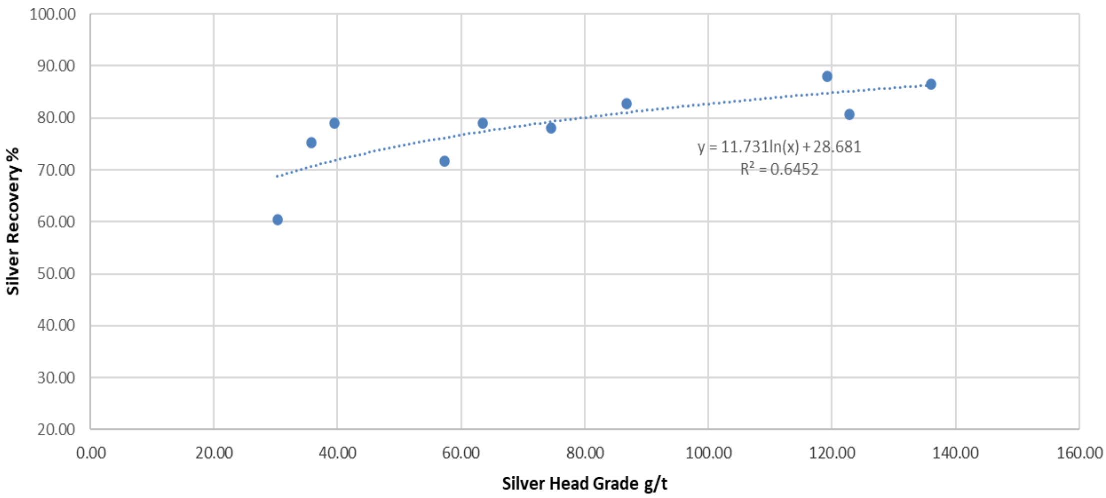
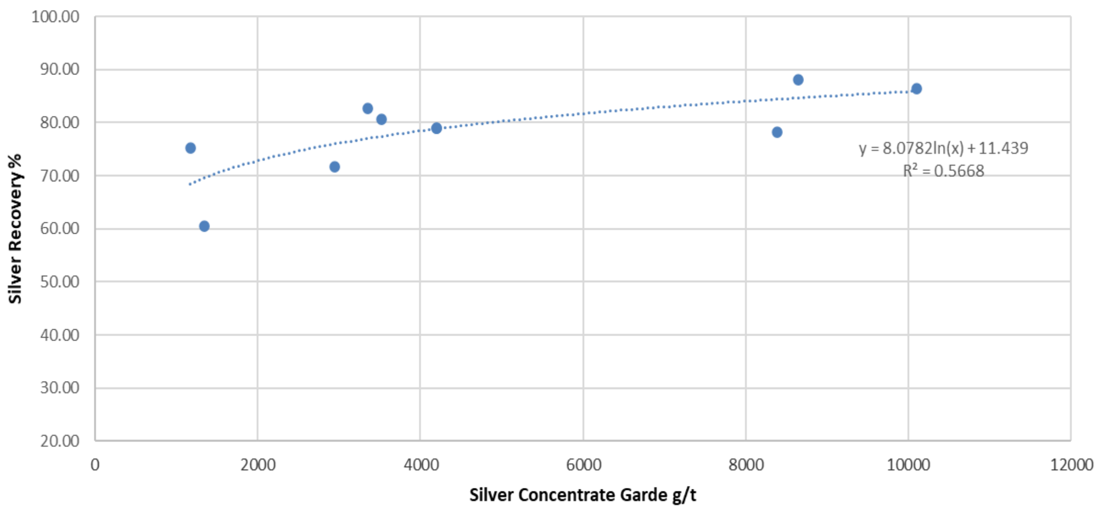


Figure 13.2.9 Locked-Cycle Ag Concentrate Grade vs. Recovery



13.3 2025 to 2026 DFS Metallurgical Testing

A comprehensive metallurgical testing program was undertaken during 2025 and 2026 to provide DFS-level confidence in the selected Bowdens process flowsheet and to address residual technical risks identified during earlier study phases. The program comprised crushing and grinding studies, flotation variability testing, geochemical assessments, comminution modelling, and tailings variability filtration testing by various specialist laboratories. The primary objectives were to quantify metallurgical variability across a representative range of ore types, validate recovery assumptions, assess concentrate quality and impurity deportment, and generate additional design data for process plant engineering and filtered tailings evaluation. Emphasis was placed on confirming the robustness of the simplified bulk flotation flowsheet adopted for the DFS and understanding the effects of ore mineralogy, sulphide content, grade and hardness on grinding, flotation and filtration performance. The results of these programs provide a substantial expansion of the project metallurgical database and form the basis for the process design criteria, metallurgical recovery models and operating philosophy adopted for the 2026 DFS.

13.3.1 2026 KYSPYmet Flotation Variability Program

The 2026 KYSPYmet Metallurgical Variability (MetVar) program was undertaken to evaluate metallurgical variability across a broad range of Bowdens mineralisation styles and grades and to provide additional confidence in the DFS bulk flotation flowsheet. Twenty-nine MetVar unique samples, representing the expected range of Ag, Pb, Zn, S and deleterious element distributions within the deposit, were subjected to head assay characterisation, grind establishment testing and staged bulk flotation assessment. The program was designed to quantify the effects of ore variability on grindability, concentrate mass pull, metal recovery, concentrate quality and impurity deportment, while generating a statistically robust dataset suitable for process design validation and geometallurgical interpretation.

Head Assay

Table 13.3.1 summarises the head assay results for the 29 MetVar samples prepared for the BSP metallurgical testwork program. The MetVar composites were selected to represent the anticipated range of mineralogical and geochemical variability across BSP, including variation in silver grade, sulphide abundance, base metal content, and concentrations of elements that may influence flotation performance and concentrate quality. The sample suite therefore provides a robust basis for assessing metallurgical response across the range of ore types expected to be encountered during mining and processing.

Table 13.3.1 KYSPYmet Variability Samples Head Assay

Samples	Ag ppm	As ppm	Bi ppm	Cd ppm	Cu ppm	Fe pct	K pct	Pb ppm	S pct	Sb ppm	Zn ppm
BS25_MetVar_01	35	85	0.19	5.3	14	1	5.9	1419	0.22	16	707
BS25_MetVar_02	35	104	0.45	7.8	17	2.9	5.9	820	1.15	8	2056
BS25_MetVar_03	52	48	0	1.4	14	2.2	5	632	0.18	17	632
BS25_MetVar_04	47	561	0.02	13	11	2.3	5.3	1295	1.96	31	3215
BS25_MetVar_05	62	11	0.25	0.9	13	1	4.8	2960	0.15	10	968
BS25_MetVar_06	163	733	0.9	15.9	12	2.1	5.7	2507	1.39	48	3515
BS25_MetVar_07	35	321	0.15	4.4	103	4.2	5.6	2848	0.7	14	1196
BS25_MetVar_08	74	339	0.02	14.4	14	1.5	4.8	6200	0.97	20	3408
BS25_MetVar_09	59	172	0.06	7.9	15	1.2	6.1	664	0.57	16	456
BS25_MetVar_10	145	2383	1.06	3.8	113	8.5	4.4	443	6.07	30	1547
BS25_MetVar_11	37	3521	0.04	58.8	45	2.3	5.9	5708	2.62	27	15307
BS25_MetVar_12	260	38	0.02	0.3	33	0.7	6.1	1604	0.54	33	142
BS25_MetVar_13	986	115	0.03	2.7	144	1.1	8.2	10458	0.65	61	720
BS25_MetVar_14	103	28	0.05	7.3	30	1.1	6.8	2676	0.31	15	1870
BS25_MetVar_15	50	203	0.43	10.5	88	3.7	6.7	6803	1.93	36	2248
BS25_MetVar_16	42	86	0.07	33	51	0.7	5.1	29675	1.42	38	8673
BS25_MetVar_17	315	41	0.02	6.2	23	1	5.2	6149	0.44	17	1956
BS25_MetVar_18	28	18	0.09	0.9	15	0.7	3	842	0.22	14	507
BS25_MetVar_19	302	66	0.2	19.9	54	0.9	5.5	4252	0.57	57	3892
BS25_MetVar_20	57	755	0.16	21.3	25	2.7	5.1	1766	2.42	27	4546
BS25_MetVar_21	30	50	0.02	4.8	23	1.2	3.5	6808	0.88	16	1282
BS25_MetVar_22	170	234	0.17	19.3	40	0.9	6.9	4091	0.65	28	4760
BS25_MetVar_23	67	169	0.27	6.1	51	1.1	4.4	1615	0.72	28	1610
BS25_MetVar_24	808	146	0.24	30.3	104	0.6	5.9	14735	0.82	87	5561
BS25_MetVar_25	50	98	0.15	7.1	15	2	4.8	926	1.62	8	1831
BS25_MetVar_26	45	722	0.05	20.9	29	2.8	5.5	2719	2.21	11	5505
BS25_MetVar_27	200	322	0.01	0.3	10	2	5.7	2300	0.18	31	132
BS25_MetVar_28	184	1353	0.04	43.6	29	1.8	6.4	5435	2.17	19	7990
BS25_MetVar_29	147	263	0.16	22.5	74	2.1	4.4	2955	1.52	27	5245

Representative Samples

While the full suite of tests programs were run on all 29 MetVar samples, only 18 samples were used in the final process design criteria and economic evaluation. MetVar samples 1, 24, 27 and 29 were oxide samples. MetVar samples 3, 5, 13, 15, 17, 19, 22 and 28 represented identified abnormal geological features such as faults and intrusives that contained grade but not representative of the main ore body, or were excessively high grade.

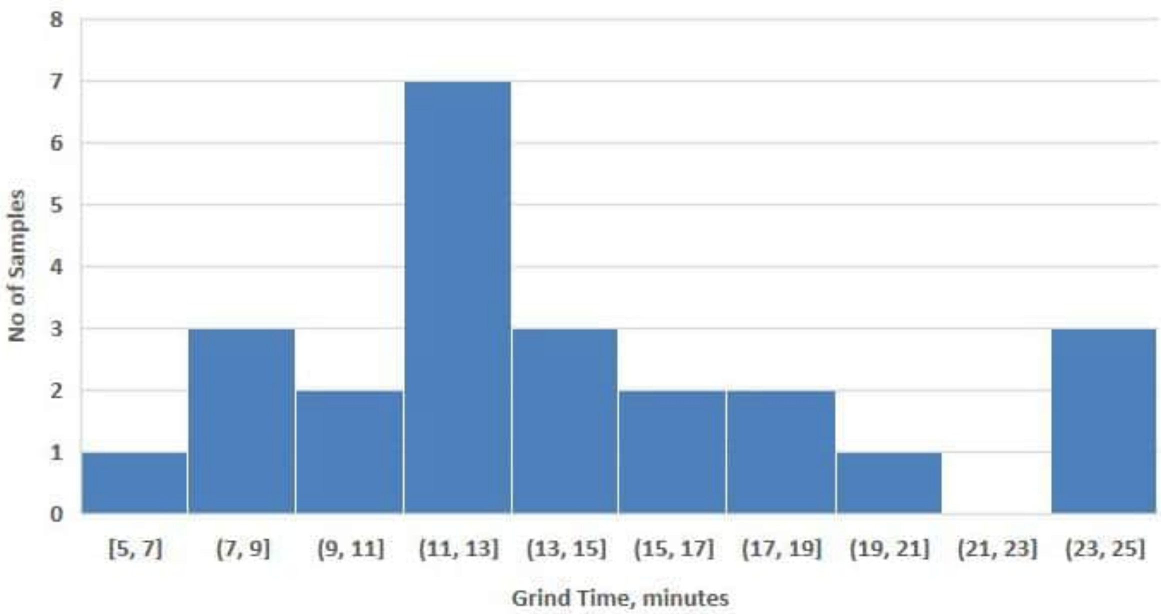
Excluding high grade samples demonstrates a relatively conservative approach to assessing metallurgical recovery, given the high recoveries achieved from those samples.

Grind Establishment

Grind establishment testing was conducted on the MetVar samples to determine the grinding time required to achieve the target flotation feed size of P_{80} 106 µm. The testwork was completed using a laboratory rod mill, with target grind times established by interpolation between successive grinding intervals. The resulting grind times provide an indication of the relative hardness and grinding characteristics of the ore types represented within the MetVar program.

The distribution of grind establishment results is shown in Figure 13.3.1. The histogram indicates that the majority of samples are clustered between 11 and 15 min, with the largest population occurring between 11 and 13 min. This suggests that most Bowdens deposit ore types exhibit moderate grinding resistance and comminution behaviour close to the overall average. A smaller number of samples required considerably longer grind times of greater than 19 min, representing the hardest materials tested, while several composites achieved the target grind size in less than 8 min, indicating relatively soft and readily ground ore types.

Figure 13.3.1 Variability Sample – Sulphide Grind Time Histogram



The variability program grind establishment tests demonstrate that the Bowdens deposit orebody exhibits a broad but manageable range of comminution characteristics, particularly with the selected comminution flowsheet. The results confirm that the variability composites capture the expected range of ore hardness and provide a sound basis for evaluating the relationship between grinding performance, mineral liberation and flotation response. Together with the flotation variability results, the grind establishment data supports the development of process design criteria capable of accommodating the anticipated range of feed variability throughout the LOM.

Variability Testwork Evaluation

The metallurgical variability flotation program provides a comprehensive assessment of flotation performance across a broad range of ore types representative of the Bowdens deposit. The results presented in Tables 13.3.2, 13.3.3 and 13.3.4 demonstrate the expected progression of concentrate upgrading through the flotation circuit, with successive cleaning stages reducing concentrate mass while increasing concentrate grade and selectively rejecting gangue and non-target minerals.

Table 13.3.2 MetVar Rougher Concentrate

Samples	Mass	Concentrate Grade					Concentrate Recovery				
	Wt pct	Ag ppm	As ppm	Pb pct	Zn pct	S pct	Ag	As	Pb	Zn	S
BS25_MetVar_01	*4.40*	*659*	*1,353*	*1.59*	*3.60*	*21.00*	*83.50*	*55.49*	*79.42*	*92.38*	*81.75*
BS25_MetVar_02	1.16	2,824	281	3.25	2.27	12.12	55.09	4.97	51.29	36.48	65.51
BS25_MetVar_03	*6.03*	*426*	*4,122*	*1.29*	*3.57*	*21.27*	*84.53*	*64.91*	*90.01*	*94.44*	*76.73*
BS25_MetVar_04	1.72	1,784	363	12.51	1.63	5.23	62.14	16.57	73.91	41.59	58.07
BS25_MetVar_05	*4.68*	*2,163*	*7,899*	*3.19*	*4.66*	*22.55*	*98.15*	*74.90*	*95.15*	*96.95*	*83.18*
BS25_MetVar_06	2.70	873	5,867	10.45	3.19	17.75	92.37	83.57	96.63	97.26	72.91
BS25_MetVar_07	4.55	2,065	2,934	14.25	7.17	14.82	92.48	57.82	97.68	97.66	70.11
BS25_MetVar_08	8.53	759	11,474	2.31	0.72	32.10	95.93	85.94	97.15	92.80	89.80
BS25_MetVar_09	8.29	468	26,873	7.58	15.51	27.34	95.48	83.35	95.28	89.75	87.72
BS25_MetVar_10	3.12	3,522	1,045	4.25	0.48	15.65	90.44	51.28	81.26	82.55	72.84
BS25_MetVar_11	4.08	8,876	2,053	21.31	1.51	12.32	95.21	73.17	95.84	96.25	80.48
BS25_MetVar_12	2.39	4,595	652	9.71	6.61	9.12	94.15	33.31	91.02	93.05	61.00
BS25_MetVar_13	*7.76*	*3,523*	*2,098*	*8.85*	*2.62*	*22.33*	*98.67*	*84.65*	*97.51*	*98.04*	*86.87*
BS25_MetVar_14	7.61	577	910	35.19	10.70	17.63	94.07	70.10	90.71	98.93	88.43
BS25_MetVar_15	*1.74*	*847*	*572*	*2.93*	*1.41*	*9.78*	*65.24*	*24.06*	*64.89*	*51.48*	*74.73*
BS25_MetVar_16	4.75	6,895	1,169	9.48	8.39	12.67	98.85	64.58	94.32	97.10	87.44
BS25_MetVar_17	*7.12*	*622*	*7,216*	*1.82*	*5.21*	*25.77*	*82.65*	*68.94*	*88.18*	*97.20*	*80.08*
BS25_MetVar_18	5.37	1,189	917	19.02	3.99	19.87	97.12	61.94	96.43	98.82	82.89
BS25_MetVar_19	*3.73*	*3,640*	*2,299*	*8.07*	*10.52*	*13.50*	*42.46*	*53.94*	*85.29*	*95.84*	*77.59*
BS25_MetVar_20	2.90	2,301	2,914	5.24	4.77	17.81	95.82	73.13	90.73	96.54	74.00
BS25_MetVar_21	5.19	2,146	2,214	2.36	4.05	29.51	91.44	79.12	80.11	90.20	78.11
BS25_MetVar_22	*6.65*	*617*	*7,409*	*3.30*	*5.95*	*31.61*	*95.65*	*79.99*	*96.99*	*97.40*	*91.06*
BS25_MetVar_23	7.15	2,358	8,920	4.00	10.47	27.71	95.78	57.71	82.23	98.98	90.34
BS25_MetVar_24	*7.78*	*1,597*	*2,507*	*3.15*	*6.96*	*17.25*	*93.09*	*86.86*	*86.32*	*95.51*	*84.25*
BS25_MetVar_25	2.71	1,850	3,928	1.37	1.04	23.21	83.77	77.39	43.49	68.41	84.69
BS25_MetVar_26	2.87	7,921	841	18.03	5.57	11.35	97.10	43.72	97.46	96.37	82.65
BS25_MetVar_27	*5.18*	*7,783*	*2,242*	*22.10*	*7.45*	*11.40*	*96.16*	*79.30*	*98.60*	*96.68*	*85.58*
BS25_MetVar_28	*1.30*	*1,447*	*923*	*2.15*	*3.63*	*4.30*	*44.28*	*27.55*	*16.34*	*55.08*	*53.33*
BS25_MetVar_29	2.31	4,788	4,807	2.46	5.30	20.58	79.57	26.29	21.77	93.22	71.35

Samples	Mass	Concentrate Grade					Concentrate Recovery				
	Wt pct	Ag ppm	As ppm	Pb pct	Zn pct	S pct	Ag	As	Pb	Zn	S
Minimum	1.16	426	281	1.29	0.48	4.30	42.46	4.97	16.34	36.48	53.33
Maximum	8.53	8,876	26,873	35.19	15.51	32.10	98.85	86.86	98.60	98.98	91.06
Standard Deviation	2.27	2,427	5,271	8.15	3.53	7.49	16.20	23.03	22.09	18.08	9.83
Mean	4.61	2,728	4,028	8.32	5.14	18.19	85.90	60.16	81.93	87.48	78.40
Coefficient of Variation	0.49	0.89	1.31	0.98	0.69	0.41	0.19	0.38	0.27	0.21	0.13

The cleaner flotation results presented in Table 13.3.3 demonstrate the effectiveness of regrinding and staged cleaning in upgrading the rougher concentrate. Average concentrate mass decreased from 4.61% in the rougher stage to 1.90% in the cleaner stage, while average silver grade increased from 2,728 ppm to 6,415 ppm. Similarly, average lead and zinc grades increased from 8.32% and 5.14% to 18.4% and 13.0%, respectively. These improvements were accompanied by moderate reductions in recovery, which is consistent with the rejection of gangue minerals and less-liberated sulphide particles during cleaning.

Table 13.3.3 MetVar Cleaner Concentrate

Samples	Mass	Concentrate Grade					Concentrate Recovery				
	Wt pct	Ag ppm	As ppm	Pb pct	Zn pct	S pct	Ag	As	Pb	Zn	S
BS25_MetVar_01	*1.52*	*1,801*	*2,693*	*4.4*	*10.4*	*35.7*	*78.5*	*38.0*	*75.4*	*91.8*	*47.8*
BS25_MetVar_02	0.25	11,657	523	13.3	10.0	18.2	49.8	2.0	45.9	35.3	21.6
BS25_MetVar_03	*2.04*	*1,170*	*8,862*	*3.7*	*10.5*	*37.8*	*78.8*	*47.3*	*87.0*	*94.0*	*46.2*
BS25_MetVar_04	0.62	4,587	646	32.9	4.2	10.6	57.7	10.6	70.1	38.5	42.3
BS25_MetVar_05	*1.84*	*5,442*	*15,092*	*7.9*	*11.8*	*36.6*	*96.8*	*56.1*	*92.7*	*96.5*	*52.9*
BS25_MetVar_06	0.90	2,518	12,487	30.7	9.5	28.7	88.4	59.0	94.0	96.5	39.2
BS25_MetVar_07	1.70	5,370	4,302	37.7	19.1	23.7	89.7	31.6	96.4	97.2	41.8
BS25_MetVar_08	3.84	1,630	21,156	5.1	1.6	47.7	92.7	71.3	96.3	92.1	60.1
BS25_MetVar_09	3.57	972	15,344	17.1	35.3	30.8	85.5	20.5	92.9	88.1	42.6
BS25_MetVar_10	1.03	10,414	2,520	12.6	1.4	34.6	88.1	40.7	79.1	80.7	53.0
BS25_MetVar_11	2.78	12,751	2,927	31.0	2.2	17.1	93.1	71.0	94.8	95.8	76.1
BS25_MetVar_12	0.83	12,589	1,455	27.2	18.8	22.8	89.8	25.9	88.8	92.3	53.2
BS25_MetVar_13	*2.57*	*9,409*	*3,406*	*26.3*	*7.9*	*31.1*	*87.3*	*45.5*	*95.8*	*97.7*	*40.0*
BS25_MetVar_14	5.33	792	918	48.6	15.0	21.3	90.3	49.5	87.8	97.2	74.9
BS25_MetVar_15	*0.59*	*2,380*	*1,274*	*8.2*	*4.0*	*22.3*	*62.3*	*18.2*	*61.9*	*49.9*	*58.0*
BS25_MetVar_16	1.94	16,707	2,103	22.8	20.4	24.2	98.0	47.5	92.6	96.7	68.4
BS25_MetVar_17	*2.78*	*1,445*	*11,735*	*4.5*	*13.3*	*38.2*	*75.0*	*43.8*	*84.3*	*96.8*	*46.4*
BS25_MetVar_18	2.30	2,661	837	43.7	9.3	20.6	92.9	24.2	94.7	98.2	36.8
BS25_MetVar_19	*1.59*	*8,370*	*4,118*	*18.5*	*24.5*	*25.9*	*41.7*	*41.2*	*83.5*	*95.3*	*63.6*

Samples	**Mass**	**Concentrate Grade**					**Concentrate Recovery**				
	Wt pct	**Ag ppm**	**As ppm**	**Pb pct**	**Zn pct**	**S pct**	**Ag**	**As**	**Pb**	**Zn**	**S**
BS25_MetVar_20	0.92	7,044	9,182	16.0	15.0	31.9	92.6	72.8	87.8	95.9	41.9
BS25_MetVar_21	1.86	5,570	3,677	6.3	11.2	39.1	84.9	47.0	76.9	89.4	37.0
BS25_MetVar_22	*2.06*	*1,805*	*8,877*	*10.4*	*19.1*	*34.5*	*86.8*	*29.7*	*95.0*	*96.9*	*30.8*
BS25_MetVar_23	2.55	5,977	10,307	10.5	29.2	34.4	86.7	23.8	77.2	98.5	40.0
BS25_MetVar_24	*2.52*	*4,653*	*4,026*	*9.4*	*21.4*	*23.6*	*87.8*	*45.1*	*83.4*	*94.9*	*37.3*
BS25_MetVar_25	1.29	3,717	6,688	2.6	2.1	38.2	79.8	62.5	39.4	66.2	66.1
BS25_MetVar_26	1.29	17,201	1,518	39.4	12.3	21.6	94.7	35.4	95.5	95.6	70.5
BS25_MetVar_27	*3.33*	*11,939*	*3,293*	*33.9*	*11.5*	*17.2*	*94.9*	*74.9*	*97.4*	*96.3*	*82.8*
BS25_MetVar_28	*0.32*	*5,592*	*3,269*	*5.4*	*13.9*	*16.3*	*41.6*	*23.7*	*10.1*	*51.3*	*49.0*
BS25_MetVar_29	1.07	9,887	8,255	4.8	11.3	37.2	76.4	21.0	19.6	92.3	59.9
Minimum	0.25	792	523	2.6	1.4	10.6	41.6	2.0	10.1	35.3	21.6
Maximum	5.33	17,201	21,156	48.6	35.3	47.7	98.0	74.9	97.4	98.5	82.8
Standard Deviation	1.16	4,787	5,292	13.8	8.2	8.9	15.9	19.2	23.0	18.7	14.8
Mean	1.90	6,415	5,913	18.4	13.0	28.3	81.5	40.7	79.2	86.5	51.0
Coefficient of Variation	**0.61**	**0.75**	**0.89**	**0.75**	**0.63**	**0.31**	**0.19**	**0.47**	**0.29**	**0.22**	**0.29**

Further upgrading was achieved during recleaner flotation. As shown in Table 13.3.4, the average concentrate mass was reduced to 1.40%, while average silver grade increased to 8,544 ppm. Average lead and zinc grades increased further to 23.7% and 18.0%, respectively, reflecting the effectiveness of final-stage cleaning in producing a higher-quality concentrate. Although incremental recovery losses occurred during upgrading, average silver, lead and zinc recoveries remained relatively high at 78.6%, 76.2% and 85.5%, respectively.

Table 13.3.4 MetVar Recleaner Concentrate (Final Concentrate)

Samples	Mass	Concentrate Grade					Concentrate Recovery				
	Wt pct	Ag ppm	As ppm	Pb pct	Zn pct	S pct	Ag	As	Pb	Zn	S
BS25_MetVar_01	*0.98*	*2,669*	*3,155*	*6.6*	*16.0*	*38.7*	*75.4*	*28.9*	*73.6*	*91.3*	*33.6*
BS25_MetVar_02	0.16	16,850	565	19.0	15.4	16.6	45.7	1.4	41.7	34.4	12.5
BS25_MetVar_03	*1.15*	*1,979*	*10,865*	*6.4*	*18.6*	*40.7*	*74.8*	*32.6*	*84.9*	*93.5*	*27.9*
BS25_MetVar_04	0.48	5,586	606	40.9	5.2	11.2	54.6	7.8	67.8	37.2	34.9
BS25_MetVar_05	*1.29*	*7,673*	*17,190*	*11.2*	*16.8*	*37.6*	*96.1*	*45.0*	*91.8*	*96.2*	*38.3*
BS25_MetVar_06	0.63	3,530	14,200	43.4	13.6	24.7	86.6	46.9	93.0	96.0	23.5
BS25_MetVar_07	1.29	6,908	3,616	39.2	25.1	24.3	87.6	20.2	76.1	96.7	32.6
BS25_MetVar_08	2.95	2,073	24,854	6.6	2.0	50.8	90.6	64.4	96.0	91.5	49.2
BS25_MetVar_09	3.00	1,115	9,089	20.0	41.5	30.8	82.6	10.2	91.2	87.1	35.8
BS25_MetVar_10	0.80	13,166	2,909	16.0	1.8	38.1	86.3	36.4	77.9	78.5	45.3
BS25_MetVar_11	2.04	16,805	3,785	41.5	3.0	20.9	90.2	67.5	93.4	95.1	68.4
BS25_MetVar_12	0.66	15,455	1,412	32.5	23.4	25.0	87.6	20.0	84.2	91.2	46.2
BS25_MetVar_13	*1.60*	*14,362*	*3,660*	*41.8*	*12.6*	*26.5*	*82.8*	*30.4*	*94.9*	*97.3*	*21.2*
BS25_MetVar_14	4.70	864	766	53.1	16.7	21.3	86.8	36.4	84.6	95.2	65.8
BS25_MetVar_15	*0.42*	*3,184*	*1,522*	*10.8*	*5.5*	*27.7*	*59.7*	*15.6*	*58.1*	*48.5*	*51.5*
BS25_MetVar_16	1.47	21,599	2,121	29.3	26.9	27.2	95.8	36.2	90.0	96.2	58.0
BS25_MetVar_17	*1.84*	*2,078*	*12,813*	*6.6*	*20.0*	*40.3*	*71.4*	*31.6*	*82.5*	*96.4*	*32.4*
BS25_MetVar_18	1.59	3,635	560	56.6	13.2	20.5	87.7	11.2	84.6	96.2	25.3
BS25_MetVar_19	*1.25*	*10,423*	*3,890*	*22.9*	*30.9*	*27.8*	*40.8*	*30.7*	*81.1*	*94.5*	*53.7*
BS25_MetVar_20	0.60	10,445	10,692	23.8	22.7	28.4	90.1	55.6	85.4	95.3	24.5
BS25_MetVar_21	1.19	8,357	4,257	9.7	17.4	38.3	81.6	34.9	75.5	88.8	23.2
BS25_MetVar_22	*1.31*	*2,711*	*6,745*	*16.1*	*29.9*	*32.3*	*82.9*	*14.4*	*93.3*	*96.5*	*18.3*
BS25_MetVar_23	1.98	7,491	8,705	13.3	37.4	33.9	84.5	15.6	75.7	98.2	30.7
BS25_MetVar_24	*1.42*	*7,958*	*4,805*	*16.3*	*37.7*	*29.5*	*84.6*	*30.4*	*81.2*	*94.3*	*26.3*
BS25_MetVar_25	0.93	4,930	7,360	3.4	2.9	41.3	76.8	49.9	37.0	64.7	51.8
BS25_MetVar_26	1.08	19,964	1,547	45.9	14.6	23.0	91.9	30.2	93.1	94.7	62.8
BS25_MetVar_27	*2.60*	*14,900*	*3,688*	*42.2*	*14.5*	*20.2*	*92.2*	*65.4*	*94.4*	*93.9*	*76.0*
BS25_MetVar_28	*0.18*	*9,200*	*4,860*	*7.9*	*23.3*	*23.9*	*38.8*	*20.0*	*8.3*	*48.8*	*40.9*
BS25_MetVar_29	0.87	11,851	9,011	5.6	13.8	41.5	74.2	18.6	18.7	91.5	54.2
Minimum	0.16	864	560	3.4	1.8	11.2	38.8	1.4	8.3	34.4	12.5
Maximum	4.70	21,599	24,854	56.6	41.5	50.8	96.1	67.5	96.0	98.2	76.0
Standard Deviation	0.97	6,018	5,696	16.3	10.8	9.0	15.9	17.6	22.7	19.0	16.5
Mean	1.40	8,544	6,181	23.7	18.0	29.8	78.6	31.3	76.2	85.5	40.2
Coefficient of Variation	**0.69**	**0.70**	**0.92**	**0.69**	**0.60**	**0.30**	**0.20**	**0.56**	**0.30**	**0.22**	**0.41**

The MetVar testwork demonstrates that the Bowdens flotation circuit is capable of delivering consistent metallurgical performance across a wide range of feed compositions. Although feed variability influences concentrate mass pull, recovery, concentrate grade, and impurity deportment, the circuit maintained strong silver and base metal recoveries throughout the program. The results further demonstrate that staged regrinding and cleaning effectively increase concentrate quality and provide an important mechanism for controlling arsenic deportment. Collectively, the variability dataset provides a sound technical basis for predicting metallurgical performance, defining concentrate quality ranges, assessing impurity risks, and supporting downstream process design and dewatering evaluations.

Notably, the MetVar program achieved results that, on average, were very similar to the preceding developmental and optimisation bulk flotation tests.

13.3.2 2026 Core Resources Bulk Flotation Program

A pilot scale bulk flotation testwork program was undertaken by Core Resources, with the main objectives being to generate representative concentrate for shipping toxicity tests and tailings products for filtration tests by prospective vendors. Three master composites were prepared to represent the principal comminution domains identified within the deposit, comprising a soft (SFT), medium (MED), and hard (HRD) ore type. The program incorporated rougher flotation, concentrate regrinding, cleaner flotation and secondary cleaner flotation stages to generate representative bulk sulphide concentrates for product characterisation and environmental assessment studies.

Head Assay

The head assays for the three bulk flotation composites are presented in Table 13.3.5. The composites displayed significant variability in Ag, Pb, Zn and S grades, reflecting the range of mineralisation styles and sulphide contents encountered within the Bowdens deposit.

Table 13.3.5 Bulk Flotation Samples Head Assay

Samples	Ag ppm	As ppm	Cu pct	Fe pct	Pb pct	Sb ppm	Zn pct	S pct
SFT Bulk	53	0.01	0.003	1.1	0.17	12.5	0.25	0.54
MED Bulk	115	0.04	0.003	1.1	0.25	12.5	0.38	0.81
HRD Bulk	92	0.04	0.02	1.3	0.37	12.5	0.39	1.13

Grind Establishment

Prior to flotation testing, grind establishment tests were undertaken to determine the milling time required to achieve the target flotation feed size of P_{80} 106 µm. The grind establishment times, which are generally proportional to ore hardness were 11.5, 27.7 and 23 min for the SFT, MED and HRD composites, respectively. This result is indicative of the predictability of softer minerals but less discernability between the harder minerals in the ore in any given sample.

Following rougher flotation, the combined rougher concentrates were reground to improve mineral liberation prior to cleaner flotation. Measured regrind product sizes ranged from approximately 19 to 20 µm across the three composites.

Bulk Flotation Evaluation

The flotation performance achieved for the three bulk composite samples is summarised in Table 13.3.6. All composites produced low-mass, high-grade bulk sulphide concentrates.

Despite differences in feed grade and sulphide content, the final concentrate mass pull remained relatively consistent at approximately 1.1 wt.% to 1.2 wt.% of feed.

Table 13.3.6 Summary of Bulk Flotation Performance (SFT, MED, HRD)

SFT									
Recoveries (%)	**Mass (%)**	**Ag (%)**	**As (%)**	**Cu (%)**	**Fe (%)**	**Pb (%)**	**Sb (%)**	**Zn (%)**	**S (%)**
Recleaner	1.13	70.3	35.4	23.2	15.4	74.9	27.5	79.1	59.4
Cleaner	1.88	70.8	42.1	24.7	17.3	75.9	28.3	79.4	61.8
Rougher	6.63	81.8	64.9	33.2	28.4	86.1	37.0	88.1	80.7
Grades		**Ag (%)**	**As (%)**	**Cu (%)**	**Fe (%)**	**Pb (%)**	**Sb (ppm)**	**Zn (%)**	**S (%)**
Recleaner		3,480	0.26	0.11	15.0	9.68	450	15.5	25.4
Cleaner		2,107	0.19	0.07	10.1	5.90	279	9.4	15.9
Rougher		691	0.08	0.03	4.7	1.90	103	3.0	5.9
Cl. Tail		130	0.04	0.01	2.6	0.31	34	0.4	1.9
Feed		56	0.01	0.01	1.1	0.15	19	0.2	0.5
MED									
Recoveries (%)	**Mass (%)**	**Ag (%)**	**As (%)**	**Cu (%)**	**Fe (%)**	**Pb (%)**	**Sb (%)**	**Zn (%)**	**S (%)**
Recleaner	1.07	84.2	22.8	22.2	7.8	79.3	39.8	93.2	41.1
Cleaner	1.36	85.7	35.9	23.3	13.2	80.1	41.9	93.4	51.3
Rougher	4.55	89.4	60.9	26.9	30.8	84.1	48.5	94.3	81.3
Grades		**Ag (%)**	**As (%)**	**Cu (%)**	**Fe (%)**	**Pb (%)**	**Sb (ppm)**	**Zn (%)**	**S (%)**
Recleaner		7,264	0.92	0.28	9.7	18.30	864	33.6	30.2
Cleaner		5,802	1.14	0.23	12.9	14.51	715	26.5	29.5
Rougher		1,810	0.58	0.08	9.0	4.55	248	8.0	14.0
Cl. Tail		108	0.34	0.02	7.3	0.30	48	0.1	7.4
Feed		92	0.04	0.01	1.3	0.25	23	0.4	0.8
HRD									
Recoveries (%)	**Mass (%)**	**Ag (%)**	**As (%)**	**Cu (%)**	**Fe (%)**	**Pb (%)**	**Sb (%)**	**Zn (%)**	**S (%)**
Recleaner	1.24	86.1	16.5	27.6	6.7	89.5	32.6	93.3	32.4
Cleaner	1.91	88.0	34.5	28.8	16.6	90.7	35.5	93.7	47.6
Rougher	4.43	90.5	60.7	31.0	32.8	93.5	39.8	94.3	70.7
Grades		**Ag (%)**	**As (%)**	**Cu (%)**	**Fe (%)**	**Pb (%)**	**Sb (ppm)**	**Zn (%)**	**S (%)**
Recleaner		6,727	0.41	0.40	7.4	25.45	521	28.8	26.7
Cleaner		4,470	0.56	0.27	12.0	16.77	368	18.8	25.6
Rougher		1,986	0.42	0.13	10.2	7.47	178	8.2	16.4
Cl. Tail		93	0.32	0.02	8.9	0.39	34	0.1	9.4
Feed		97	0.03	0.02	1.4	0.35	20	0.4	1.0

Note: Recleaner = Final Concentrate.

The results demonstrate that the rougher / regrind / cleaner / recleaner flowsheet produced a high-grade bulk sulphide concentrate from all three ore domains while achieving strong Ag, Pb and Zn recoveries. The consistency of concentrate quality and concentrate mass pull across the SFT, MED and HRD composites provides confidence that the proposed process flowsheet can accommodate the expected range of ore variability within the Bowdens deposit.

It was also notable that despite the good results, there were significant recovery losses to the cleaner tail, particularly in the case of the SFT sample. Subsequent analyses showed that that the losses were somewhat concentrated in the coarser fractions, indicating some realistically achievable recovery upside with improved classification of the re-ground rougher concentrate.

Table 13.3.7 Size-by Size Assays of Cleaner Tail (SFT, MED, HRD)

Screen Size	SFT Assays			MED Assays			HRD Assays		
Fraction (µm)	Ag (g/t)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)
150	410	0.92	1.62	353	0.60	0.37	93	0.25	0.14
125	441	1.09	0.83	337	0.62	0.34	152	0.44	0.17
106	380	0.92	0.63	411	0.67	0.31	164	0.57	0.19
75	234	0.42	0.44	354	0.64	0.25	262	0.74	0.22
53	158	0.27	0.47	232	0.28	0.16	135	0.43	0.14
38	184	0.35	0.62	240	0.25	0.13	95	0.23	0.09
20	186	0.40	0.74	203	0.20	0.08	83	0.15	0.06
-20	66	0.27	0.28	84.4	0.29	0.06	95	0.41	0.10

13.4 Flotation Process design Criteria and Metal Recoveries

The measurement of flotation performance from the relevant testwork was used as the basis for the DFS process design criteria (PDC) and the project economic evaluation. The contributing testwork programs were:

- KYSPYmet testwork development and optimisation program (2024 - 2025).
- Metallurgical Variability Tests (2026).
- Bulk Flotation (Core Resources, 2026).

In total, 43 individual tests were used to the develop the flotation PDC and corresponding metal recoveries.

At the mine grades used in the DFS, the average metal recoveries to final concentrate are:

- Silver, 82.8 %.
- Lead , 80.9 %
- Zinc, 90.8 %.

13.5 Comminution Circuit Design

This section summarises the comminution testwork, trade-off studies (TOSs) and circuit optimisation investigations completed by Orway Mineral Consultants (OMC) in support of the DFS process plant design for BSP. The objective of the work was to characterise ore competency, evaluate the influence of ore variability on grinding performance, optimise circuit configuration, minimise unnecessary fines generation, and establish a robust and flexible comminution circuit suitable for operation over the life of mine. The final comminution design is supported by historical databases, additional 2025 grindability testing and comprehensive JKSimMet modelling studies.

The DFS comminution design is supported by:

- Historical comminution investigations undertaken by ALS Ammtec, ALS Metallurgy, JKTech and OMC.
- Additional 2025 grindability testing on representative ore composites.
- JKSimMet circuit modelling and simulation studies.
- Evaluation of seven alternative comminution configurations.
- Engineering trade-off assessments considering throughput, energy efficiency, water balance, operability, maintainability, product quality and capital cost.
- Selection of a flowsheet customised for Bowdens deposit ore, that despite being unique, only uses commercially proven technologies and unit processes.

Particular emphasis was placed on balancing grinding efficiency, operational consistency with sufficient flexibility, water balance requirements, flotation feed quality and downstream dewatering performance. Previous metallurgical investigations identified that excessive fines generation has the potential to adversely affect flotation selectivity, concentrate filtration and tailings dewatering. Consequently, the DFS assessed not only throughput and energy consumption but also the production of ultrafine material and the ability of alternative circuit configurations to consistently deliver a stable flotation feed size distribution.

13.5.1 Comminution Optimisation Program

The optimisation study was undertaken to:

- Confirm design comminution parameters.
- Assess ore variability and blend performance.
- Minimise unnecessary fines generation.
- Improve grinding circuit efficiency.
- Establish a robust and flexible circuit configuration.
- Support DFS equipment sizing and process design.

Particular attention was given to maintaining operational flexibility and ensuring compatibility with downstream flotation and filtration requirements. Previous testwork identified rheology, fines generation, variable ore hardness and water balance as key risks requiring consideration during circuit selection.

Sample Selection and Ore Characterisation

Representative composite samples covering the principal ore types anticipated during mining were selected for testing, including MCLIN-KSPA, Siderite, Siliceous and White Mica composites.

Testing confirmed significant but manageable variability across the orebody as shown in Table 13.5.1.

Table 13.5.1 Key Comminution Parameters

Parameter	Range
Bond Ball Mill Work Index	1.9 - 18.4 kWh/t
A×b	3.8 - 65.3
Specific Gravity	2.36 - 2.67
UCS	19.4 - 106.8 MPa
Abrasion Index	0.101 - 0.339 g

The measured A×b values significantly exceed the generally accepted SAG milling threshold of approximately 25, confirming favourable impact breakage behaviour and indicating that all tested ore types should perform satisfactorily within a SAG milling environment. The relatively high A×b values reduce the likelihood of critical-size build-up and suggest that SAG throughput should remain robust despite variations in ore competency.

The 2025 grindability results generally fall within the ranges established by the historical OMC database and therefore provide additional confidence that the test composites are representative of future plant feed. Comparison against historical datasets indicates that no previously unidentified hardness domains were identified that would materially affect the selected grinding circuit design.

White mica-rich material demonstrated a greater tendency to generate fines and exhibited product size distribution slopes lower than average OMC database values. This behaviour is important because excessive generation of ultrafines can adversely affect flotation selectivity through slime coating and entrainment, increase reagent consumption and reduce concentrate and tailings filtration performance. Consequently, fines management became a major consideration during circuit selection.

The variability data indicated that the selected circuit must accommodate both competent and softer ore types without compromising throughput or grind size control. Adoption of variable speed drives on both the SAG and ball mills provides operational flexibility to manage changing ore competency throughout the mine life.

To further enhance grinding circuit control and reduced slimes generation, particularly in light of variable ore hardness, secondary crushing was selected in preference to a higher aspect ratio SAG mill and pebble crusher. This change will make the primary (SAG) mill more consistent to operate, reduce slimes generation and improve power efficiency.

Comminution Circuit Design Basis

The principal design basis adopted for the optimisation study are summarised below in Table 13.5.2. The selected grind size is consistent with historical ALS, JKTech and metallurgical testwork programs, which demonstrated that flotation performance is optimised at approximately P_{80} 106 µm.

Table 13.5.2 Comminution Circuit Design Basis

Parameter	Design Basis
Plant Throughput	2.0 Mt/a
Product Grind Size	P_{80} 106 µm
SAG Feed Size	F_{80} 40 mm
Primary Crusher Feed Size	Up to 900 mm
Crushing Circuit Utilisation	60%

Circuit Evaluation, Optimisation and Selection

OMC completed a comprehensive comminution review and JKSimMet simulation program to evaluate alternative grinding circuit configurations and assess the impact of ore variability on circuit performance. Seven comminution circuit configurations were investigated, including HPGR–Ball Mill, conventional SABC circuits, SAB circuits with cyclone classification, SAB circuits with fine screening, and hybrid screening and cyclone classification arrangements. The evaluation considered throughput, specific energy consumption, product size distribution, circulating load, water balance, product density, equipment sizing, operational flexibility and capital cost. Particular emphasis was placed on managing fines generation and its potential impact on downstream flotation, concentrate filtration and tailings dewatering performance.

The simulation program demonstrated that HPGR-based circuits achieved the lowest specific energy consumption of approximately 17.6 kWh/t. However, these benefits were offset by increased capital expenditure, greater operational complexity and reduced flexibility. Conventional SABC circuits achieved acceptable grinding performance but generated higher circulating loads and provided less operational flexibility for the management of ore variability. Several SAB-based alternatives failed to satisfy minimum product density and water balance requirements and were subsequently eliminated from further consideration.

A key outcome of the study was that circuit configuration, according to the model, had relatively little influence on fines generation compared with the inherent breakage characteristics of the ore. Product size distribution slopes ranged from approximately 0.40 to 0.52 and were generally consistent with the historical OMC database average of approximately 0.41. White mica-rich ore types exhibited a greater tendency to generate fines; however, overall product size distributions remained within the historical range established by OMC. These findings indicate that ore mineralogy and ore blending strategies exert greater control over fines generation than grinding circuit configuration alone.

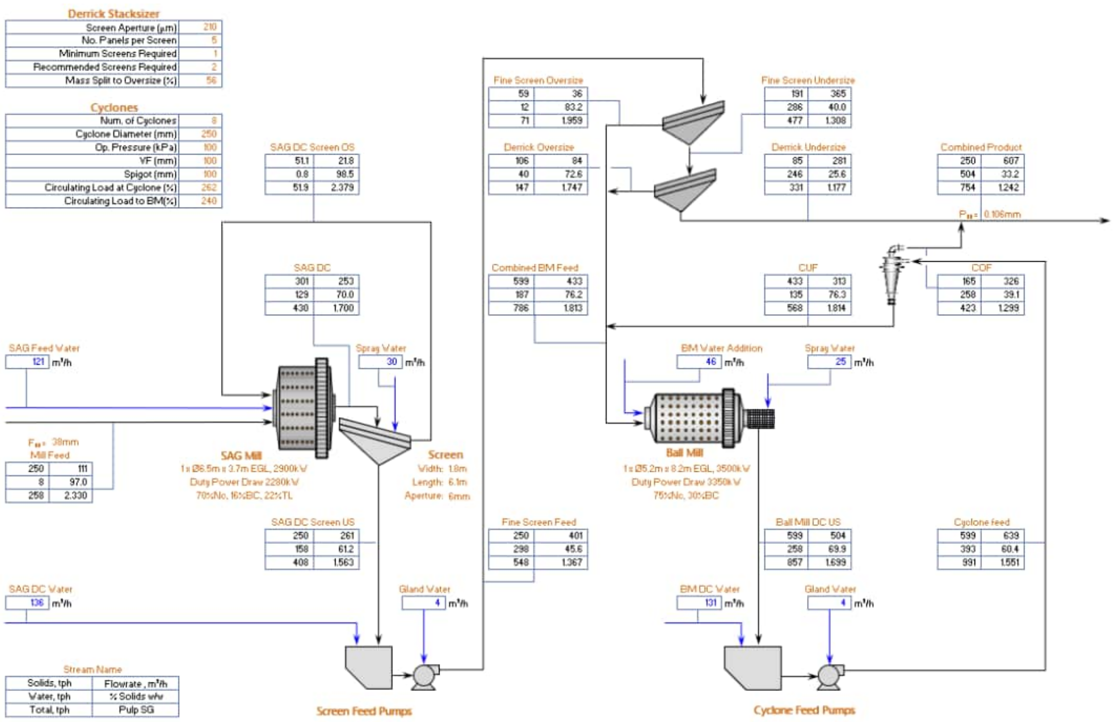
Management of ultrafine material was a primary driver of the optimisation study. Excessive fines generation has the potential to adversely affect flotation selectivity through slime coating and entrainment mechanisms, increase reagent consumption and reduce filtration efficiency in both concentrate and tailings dewatering circuits. Consequently, the preferred circuit was selected not only on the basis of energy efficiency and throughput but also on its ability to minimise overgrinding and provide a stable flotation feed.

Water balance was another critical differentiator between the alternative configurations. Several technically viable circuits were ultimately rejected because they could not realistically achieve the minimum slurry density required to support downstream flotation and filtration performance. The preferred configuration consistently achieved a final grinding product density of approximately 33% solids while maintaining acceptable rheological properties. This outcome is particularly important given previous metallurgical investigations that identified slurry density and viscosity as significant factors affecting flotation kinetics, gangue entrainment and water recovery efficiency.

Following completion of the trade-off assessment, Simulation 5 (Figure 13.5.1) was selected as the preferred comminution configuration. The circuit comprises:

- Primary crushing.
- Secondary crushing.
- SAG milling.
- Fine screening.
- Derrick ultra-fine screening.
- Ball milling.
- Cyclone classification.

Figure 13.5.1 Orway Simulation (Sim 5) Selected



The selected configuration was considered the most balanced solution because it:

- Meets the required 2.0 Mtpa plant throughput target.
- Achieves the target flotation feed grind size of P_{80} 106 µm.
- Produces approximately 33% solids in the final grinding product.
- Provides operational flexibility across varying ore types.
- Removes ultra-fine and final product-sized material generated in the SAG mill prior to it entering the ball milling circuit and the cyclones.
- Satisfies water balance requirements.
- Supports future plant expansion opportunities.
- Is energy efficient.
- Minimises technical and operational risk.

A significant advantage of the selected flowsheet is the incorporation of ultra-fine screening. Unlike conventional cyclone-only classification systems, the screening arrangement removes correctly sized particles before overgrinding occurs. This improves classification efficiency, reduces circulating load, lowers specific energy consumption and limits the generation of ultrafine particles. The screening-based approach also improves grind size control and provides greater flexibility when processing ores exhibiting variable competency, clay content or fines-generation characteristics.

OMC modelling concluded that the selected secondary crushing + SAG milling + fine screening + Derrick screening + ball milling + cyclone configuration achieved metallurgical performance broadly comparable to more complex SABC alternatives while providing superior water balance performance, improved slurry density control and greater operational flexibility. Importantly, the arrangement allows early removal of correctly sized particles from the grinding environment, thereby reducing opportunities for overgrinding and supporting the project objective of minimising fines generation across the full range of anticipated ore types. These findings strongly influenced the final DFS comminution design and form the basis for the selected process flowsheet.

13.6 Tailings Dewatering and Filtration Testwork

Tailings dewatering and filtration testwork was completed between January and March 2026 at the Metso Perth Technology Centre as part of the Bowdens Silver Tailings Variability (TailVar) program. The program evaluated the suitability of pressure filtration technologies for producing a filtered tailings product suitable for dry stacking and supporting the DFS tailings management strategy. Testing was undertaken on ten tailings variability samples (BS25_TailVar_01 to BS25_TailVar_10) using Metso Larox® Fast-Acting Filter Press (FFP), Pressure Filter (PF) and Filter Press (FP) technologies. The samples were generated by ALS Metallurgy, and involved primary grinding to P_{80} 106 µm, rougher flotation and concentrate re-grind to ensure that the fines components were correctly represented.

The primary objectives of the filtration program were to:

- Assess filtration performance across the expected range of tailings variability.
- Determine achievable filter cake moisture contents and filtration capacities.
- Evaluate the effect of tailings variability on filtration performance.
- Assess the effectiveness of filter aids and chemical conditioning.
- Generate design criteria for filtration equipment selection and sizing.

Feed Characteristics and Tailings Variability

The feed characteristics of the ten variability samples are summarised in Table 13.6.1. Feed solids concentrations ranged from 50 to 56%wt, solids specific gravity ranged from 2.53 to 3.07, and slurry densities ranged from 1.46 to 1.55 kg/L. Particle size distributions were relatively consistent, with P_{80} values ranging from approximately 98 to 104 µm; however, the proportion of ultrafine material (<10 µm) varied considerably between samples, ranging from approximately 19% to 34%.

Table 13.6.1 Properties of Received Samples

	BS25_TailVar_01	**BS25_TailVar_02**	**BS25_TailVar_03**	**BS25_TailVar_04**	**BS25_TailVar_05**
Feed					
Solids content (% w/w)	55	50	55	55	56
Solids SG	2.61	2.77	2.80	2.73	2.70
Slurry density (kg/l)	1.50	1.47	1.54	1.52	1.53
Liquid density (kg/l)	1.0	1.0	1.0	1.0	1.0
pH	6.5	6.9	6.8	6.6	7.4
Temperature (°C)	Ambient	Ambient	Ambient	Ambient	Ambient
Composition of solids	Mine Tails	Mine Tails	Mine Tails	Mine Tails	Mine Tails
Composition of liquid	Site Water	Site Water	Site Water	Site Water	Site Water
PSD P10 (µm)*	2.2	2.5	2.3	2.3	2.7
PSD P50 (µm)*	40.3	38.7	29.0	27.7	49.5
PSD P80 (µm)*	100.1	103.2	99.7	99.1	104.4
PSD P90 (µm)*	132.1	131.5	136.3	140.0	129.3
PSD <10µm (%)*	30.1	29.8	33.6	34.4	24.5
Corrosive	**N**	**N**	**N**	**N**	**N**

	BS25_TailVar_06	**BS25_TailVar_07**	**BS25_TailVar_08**	**BS25_TailVar_09**	**BS25_TailVar_10**
Feed					
Solids content (% w/w)	56	50	55	50	55
Solids SG	2.55	2.68	2.84	3.07	2.53
Slurry density (kg/l)	1.50	1.46	1.55	1.51	1.50
Liquid density (kg/l)	1.0	1.0	1.0	1.0	1.0
pH	7.3	7.0	7.2	7.2	7.4
Temperature (°C)	Ambient	Ambient	Ambient	Ambient	Ambient
Composition of solids	Mine Tails	Mine Tails	Mine Tails	Mine Tails	Mine Tails
Composition of liquid	Site Water	Site Water	Site Water	Site Water	Site Water
PSD P10 (µm)*	3.5	2.4	3.1	2.0	2.8
PSD P50 (µm)*	50.5	36.6	39.3	34.1	47.5
PSD P80 (µm)*	102.4	98.4	97.7	97.8	102.9
PSD P90 (µm)*	125.0	126.4	123.1	126.7	125.5
PSD <10µm (%)*	19.0	29.7	24.9	31.0	24.4
Corrosive	**N**	**N**	**N**	**N**	**N**

Filtration Performance

Filtration performance results are summarised in Tables 13.6.2 and 13.6.3. Across the ten variability samples tested using Metso Larox® FFP technology, filtration capacities ranged from approximately 48 to 194 kg DS/m²·h, while final filter cake moisture contents ranged from approximately 14.8 to 22.3%wt H_2O. Total cycle times varied from approximately 10 to 31 min, reflecting the influence of feed solids concentration, particle size distribution, fines content, slurry rheology and test operating conditions. Overall, the results demonstrate that pressure filtration is technically viable across the full range of BSP tailings variability.

Table 13.6.2 Filtration Performance Results - Range

Sample	Feed Solids % w/w	Filtration Capacity Range kg DS/m²·h	Cake Moisture Range % w/w	Cycle Time Range min
BS25_TailVar_01	55	73 – 112	18.7 – 20.9	14.5 – 23
BS25_TailVar_02	50	48 – 57	17.7 – 19.7	23 – 31.3
BS25_TailVar_03	55	64 – 79	19.0 – 20.6	19.5 – 24.5
BS25_TailVar_04	55*	51 – 125	18.9 – 22.3	12.5 – 31.5
BS25_TailVar_05	56	105 – 157	14.8 – 16.8	13 – 16.5
BS25_TailVar_06	56	111 – 159	16.4 – 17.7	12 – 15
BS25_TailVar_07	50	62 – 86	18.3 – 20.2	18.5 – 25
BS25_TailVar_08	55	143 – 194	17.5 – 18.5	10 – 12.5
BS25_TailVar_09	50	57 – 73	18.3 – 20.0	21.5 – 27
BS25_TailVar_10	55	103 – 156	15.8 – 16.5	12.5 – 16.3

***BS25_TailVar_04 was subsequently tested at reduced solids concentration owing to high slurry viscosity.**

Table 13.6.3 Filtration Performance Results - Optimum

Sample	Feed Solids % w/w	Optimum Capacity kg DS/m²·h	Optimum Cake Moisture % w/w	Optimum Cycle Time min	Cake Thickness mm	Wet Cake Density kg/dm³
BS25_TailVar_01	55	105	19.4	15	31	2.1
BS25_TailVar_02	50	50	17.9	29.5	27	2.22
BS25_TailVar_03	55	75	19.7	20	29	2.14
BS25_TailVar_04	55*	114	18.9	14.5	33	2.03
BS25_TailVar_05	56	137	15.8	13	36	1.96
BS25_TailVar_06	56	137	17.3	12	36	1.84
BS25_TailVar_07	50	85	19.1	18.5	30	2.15
BS25_TailVar_08	55	171	18.2	10	33	2.11
BS25_TailVar_09	50	68	18.8	22	27	2.16
BS25_TailVar_10	55	130	16.5	12.8	36	1.83

Cake Moisture and Handling Characteristics

Filter cake moisture decreased progressively with increasing drying time; however, the rate of moisture reduction diminished at longer drying times. This behaviour indicates that extending the drying cycle beyond the optimum operating point provides only marginal reductions in residual cake moisture while reducing overall filtration capacity. The testwork was therefore used to define practical operating conditions that balanced cake moisture and throughput performance.

Visual assessment of the dewatered filter cakes demonstrated favourable handling characteristics across the full range of variability samples. Under optimum operating conditions, the majority of samples produced competent and mechanically stable cakes with moisture contents generally between approximately 16%wt H_2O and 18%wt H_2O. The filter cakes maintained their structural integrity during discharge testing while readily breaking apart under vibratory forces, indicating suitability for transport, handling and dry-stack tailings disposal.

Even the more difficult filtering samples produced mechanically stable cakes with acceptable handling characteristics, although these materials generally retained slightly higher residual moisture contents and required longer filtration cycle times. Overall, all samples generated cakes considered suitable for downstream tailings handling and placement.

Effect of Filter Aids

The effect of filter aids was assessed primarily using tailings sample BS25_TailVar_04. Testwork included the addition of flocculants, coagulants, lime, surfactants, calcium chloride and Celite body-feed material under a range of operating conditions. The results demonstrated that none of the tested reagents produced a consistent or significant improvement in either filtration capacity or final cake moisture relative to the baseline test conditions. Calcium chloride, another known filter aid, was also tested on BS25_TailVar-01 sample, also with no noticeable effect.

Based on these findings, Metso concluded that filtration performance is governed principally by intrinsic tailings properties, including particle size distribution, fines content, slurry viscosity and solids concentration. Consequently, the remainder of the test program was conducted without the use of filter aids, and no filter aid addition is currently considered necessary or useful for the Bowdens tailings filtration circuit.

Filtrate Quality and Filter Cloth Performance

Filtrate quality was consistently acceptable across all variability samples. Filtrate was typically turbid during the initial stages of cake formation but clarified rapidly as the filter cake developed. Measured suspended solids concentrations generally ranged between approximately 900 and 2,000 mg/L, which is considered suitable for return to the process water circuit.

Filter cloth performance was also favourable throughout the program. Minimal solids retention was observed following cake discharge, and any residual solids were readily removed using conventional water sprays. These observations indicate a low propensity for filter cloth blinding and support reliable long-term filtration operation with normal maintenance practices.

Conclusions and Relevance to Process Design

In summary, the filtration testwork provides a robust basis for filtration equipment selection, preliminary filter sizing and integration of pressure filtration into the DFS process flowsheet. The results support the filtered tailings and dry-stack disposal strategy adopted for the 2026 DFS and demonstrate that BSP tailings can be dewatered to a stable, transportable and stackable product across the expected range of metallurgical variability.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following section describes the MRE for BSP, completed by H&SC. Completion of the MRE involved the assessment of a validated drill hole database, which included all the data completed through to 14 November 2025. Completion of the MRE also included the assessment of updated 3D mineral resource models (mineral resource domains), 3D topographic surface model and 3D oxidation surface models.

While Ag is the main commodity of interest at BSP, Pb and Zn also potentially contribute value to the Project. Consequently, a silver equivalent (AgEq) formula is used to incorporate the value of all three metals, based on multiple indicator kriging (MIK) grade estimates for Ag and ordinary kriging (OK) grade estimates for Pb and Zn.

The MRE is reported in Table 14.8.1 above a 25 g/t AgEq cut-off grade and takes into consideration that the deposit will be mined by the open pit method.

While this DFS report as a whole is not NI43-101 compliant, the MRE complies with all disclosure requirements for Mineral Resources set out in the NI43-101 *Standards of Disclosure for Mineral Projects* (2023). The classification of the MRE is consistent with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definitions) and adheres to the 2019 CIM *Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (2019 CIM Guidelines).

14.2 Drillhole Database

The database export for the new open-pit MRE was performed on 14 November 2025 (EstimationExport20251114.mdb).

Drilling since the 2024 MRE has added 16 new holes totalling 2,265 m along with 2,248 new assays to the Bowdens database. The new holes are either geometallurgical or geotechnical holes and do not have significant impact on the overall resource outline.

Once the less reliable hole types (e.g. RAB, AC, blastholes, etc.) were removed and the data was restricted to holes that intersect the Bowdens model volume, a total of 870 holes totalling 163,763 m remain. Table 14.2.1 presents a summary of data available for the 2026 MRE update.

Table 14.2.1 Bowdens Resource Database Summary

Item	2024		2026		New		% New	
	Holes	Records	Holes	Records	Holes	Records	Holes	Records
Collar	854	161,498	870	163,763	16	2,265	1.84%	1.38%
Survey	850	5,212	866	5,300	16	88	1.85%	1.66%
Assay	843	151,306	858	153,554	15	2,248	1.75%	1.46%
Geology	811	54,165	826	54,533	15	368	1.82%	0.67%
Ag	837	147,936	852	150,184	15	2,248	1.76%	1.50%
Au	607	66,549	622	68,797	15	2,248	2.41%	3.27%
Pb	837	147,932	852	150,180	15	2,248	1.76%	1.50%
Zn	837	147,930	852	150,178	15	2,248	1.76%	1.50%
S	485	109,324	500	111,572	15	2,248	3.00%	2.01%
Lith	811	54,123	826	54,491	15	368	1.82%	0.68%
WEATH	803	52,399	814	52,661	11	262	1.35%	0.50%
OXID	786	50,734	797	50,996	11	262	1.38%	0.51%
SG	254	7,076	285	7,519	31	443	10.88%	5.89%

The greatest increase is in the number of density measurements because data for 2024 holes became available in 2026.

Some basis checks were performed on the data to ensure internal consistency and data validity, and no new issues were identified. The issue of some holes missing assays from surface does not appear to have been addressed.

While H&SC did not perform detailed database reviews, as validation and responsibility for data quality rests with Bowdens personnel, H&SC considers that the Bowdens database is suitable for resource estimation. H&SC has not reviewed the 2026 QA/QC or sample recovery data for Bowdens.

14.3 Geological Interpretation

Bowdens personnel have a sound understanding of the geology of the Bowdens deposit and this is reflected in the wireframe models they prepared which form a solid framework for resource estimation.

The Rylstone and the Coomber formations were divided into several sub-domains to reflect the local orientation of mineralisation. While the higher-grade veins in the Rylstone Volcanics are broadly conformable with stratigraphy, the Bundarra veins in the Coomber Formation are disconformable to stratigraphy. In 2024 Bowdens staff stated that a revised Coomber formation interpretation was likely to be warranted using detailed facies analysis and seismic datasets that are now available.

Bowdens personnel provided updated wireframe models for the following features:

- Major stratigraphic units – Coomber Formation, Rylstone Volcanics and Shoalhaven Group.
- Internal stratigraphy for Coomber Formation and Rylstone Volcanics.

- Dacite and dolerite intrusions.
- Eastern and Gully Fault surfaces.
- Top of fresh rock and base of complete oxidation surfaces.
- Soil horizon.
- Mineralised domain boundary.

Current topography and the higher-grade Rylstone Mineralised Horizons and Bundarra veins were not updated, so the domaining and orientation of mineralisation in these units was essentially unchanged from the 2024 MRE.

14.4 Data Analysis

Only limited data analysis was performed as part of this exercise because this estimate represents a small incremental change to the previous MRE.

A nominal composite length of 2.0 m, with a minimum length of 0.99 m, was retained for data analysis and resource estimation. This length represents the longer of the more common sample intervals and is compatible with previous resource estimates. It is also considered appropriate for the chosen model block size and estimation search radii.

A comparison of summary composite statistics outside (0) and inside (1) the Rylstone mineralised domain for the major elements is presented in Table 14.4.1. The mineralised domain generally represents 77% of total composites in the Rylstone and has substantially higher average grades than samples outside this domain.

Table 14.4.1 Composite Statistics Out / Inside Rylstone Mineralised Domain

Element	Domain	Samples	Min	Max	Mean	SD	CV
Ag ppm	0	13,169	0.015	538	2.1	10.1	4.91
Ag ppm	1	43,611	0.031	3,740	31.8	89.3	2.81
Pb %	0	13,169	0.0001	2.41	0.02	0.07	4.41
Pb %	1	43,613	0.00035	14.58	0.18	0.36	1.98
Zn %	0	13,169	0.00025	2.80	0.03	0.07	2.49
Zn %	1	43,613	0.00025	16.30	0.26	0.41	1.62
Au ppm	0	3,525	0.0005	5.60	0.02	0.13	5.56
Au ppm	1	15,272	0.0025	6.84	0.03	0.16	4.76
S %	0	10,508	0.005	8.19	0.48	0.73	1.54
S %	1	25,846	0.0075	12.61	1.11	1.07	0.96

Indicator statistics for silver and gold used the same sets of cumulative proportions as implemented for the 2023 and 2024 MREs.

Density data was treated in the same manner as the 2023 and 2024 MREs, with water displacement samples retained for estimation, the same sample exclusions applied and values outside ±3 standard deviations of the 2023 data trimmed out. Average values by unit have not materially changed despite the additional measurements.

Variography was not updated because the new data only comprises a small proportion of the total data set and is not expected to have a substantial impact on local grade continuity. This assumption may warrant checking with the next update.

14.5 Geology Model

The dimensions of the final 2026 BSP open-pit block model were expanded to accommodate a potentially larger pit and details are provided in Table 14.5.1 in GDA94 zone 55 coordinates. The block size shown represents sub-blocks within larger 25 x 25 x 5 m parent blocks; most grades were estimated into sub-blocks.

Table 14.5.1 2026 Block Model Dimensions

Parameter	X	Y	Z
Origin	768,100	6,384,700	100
Maximum	769,600	6,386,300	700
Sub-Lock Size	12.5	12.5	2.5
Number of Blocks	120	128	240
Length	1,500	1,600	600

A geological model was constructed by flagging blocks below topography with the unit, sub-unit and oxidation wireframes using the codes defined in 2023 - 2024. Sub-blocks were used for topography and the oxidation boundaries, while no block splitting was used for stratigraphy. The 2023 - 2024 sub-domain codes were assigned using the orientation sub-domain wireframes with no block splitting. This geological block model formed the framework for estimation of grades and other attributes.

14.6 Grade Estimation

Grades were generally estimated using the same parameters as the 2023 - 24 MREs, with the mineralised domain used as a hard boundary for some elements (Ag, Pb, Zn, Sb and Cd) in the Rylstone Volcanics.

Silver was initially estimated by recoverable multiple indicator kriging (MIK) into 25 x 25 x 5.0 m panels. These estimates were then localised by discretising the metal distribution into sub-blocks with the dimensions of the selective mining unit (SMU) of 12.5 x 12.5 x 2.5 m. The order of assigning the metal distribution to sub-blocks was based on an OK estimate for silver into the sub-blocks.

Gold was estimated by MIK, using the e-type or average block grade at the scale of the panels; this coarser resolution reflects the substantial under-assaying of gold compared to silver, particularly in the Rylstone Volcanics. There are some holes with only occasional gold grades in the Rylstone Volcanics; these values were retained and the remaining unassayed intervals in these holes were assigned low default values (0.005 g/t Au). Holes without any gold assays in the Rylstone Volcanics were removed from the composite file used for grade estimation. The Coomber Formation was estimated using all available gold assays and no default values were inserted. These procedures may require review in future estimates. Gold grade estimates have lower confidence than silver, particularly in the Rylstone Volcanics, due to under-assaying of gold compared to silver.

All other attributes were estimated by OK into sub-blocks, including Pb, Zn, S, Cu, As, Cd, Sb, Mn, Fe and dry bulk density. OK is considered appropriate for these attributes because the coefficients of variation (CV=SD/mean) are generally low to moderate, and the grades are reasonably well structured spatially. Recoverable MIK was chosen for Ag primarily because it allows better mining selectivity than OK. Bowdens also requested OK estimates for silver and gold, which were generated using uncut grades into sub-blocks.

All grades were estimated using nominal 2.0 m composites. The final MIK search parameters for Ag and Au are given in Table 14.6.1; the OK search parameters for the major elements (Ag, Pb, Zn, S) were similar, as shown in Table 14.6.2. Search ellipsoid orientations were identical to those used for each sub-domain in the 2023 - 24 MREs.

Table 14.6.1 MIK Search Parameters

Pass	Radii			Samples		Octants
	X	Y	Z	Min	Max	Min
1	35	35	12.5	16	48	4
2	52.5	52.5	12.5	16	48	4
3	105	105	25	16	48	4
4	105	105	25	8	48	2

Table 14.6.2 OK Search Parameters

Pass	Radii			Samples		Octants
	X	Y	Z	Min	Max	Min
1	35	35	12.5	12	32	4
2	52.5	52.5	12.5	12	32	4
3	105	105	25	8	32	4
4	105	105	25	8	32	0

The initial MIK estimates were produced using GS3 software, while the local MIK and OK estimates were generated in Datamine software. The final model was compiled and evaluated in Datamine. Most of the new OK estimates were generated using the new and faster Datamine COKRIG process.

No top cutting was applied to any of the grade estimates because none of the grade distributions are strongly skewed. The mean grade was applied to all indicator classes, including the top indicator class.

Both the Rylstone Volcanics and Coomber Formation were treated as single units for grade estimation, with multiple separate orientation sub-domains using internal soft boundaries. The Shoalhaven Group was treated as a single flat domain. The contacts between the major units (Coomber, Rylstone and Shoalhaven) were treated as hard boundaries.

Elements affected by oxidation (Zn, S, Cd and Mn) were estimated separately within the oxide zone using oxide samples only. A dynamic search was used with the Z direction of the search ellipse and variogram model perpendicular to a smoothed topographic surface.

Blocks with Ag estimates but missing other economic elements were assigned low default values based on examination of peripheral model grades, grouped by UNIT code.

While Ag is the main commodity of interest at Bowdens, Pb, Zn and Au also potentially contribute value to the project. Consequently, a silver equivalent (AgEq) formula is used to incorporate the value of three metals, based on the MIK estimates for Ag and OK estimates for Pb and Zn. This formula is based on assumed metal prices and metallurgical recoveries derived from available test work. Note that Gold is not included in the AgEq formula because it currently represents less than 1% of revenue in the DFS study.

Table 14.6.3 provides the price and recovery assumptions used to calculate the current 2026 formula, which are significantly different from the 2024 version where the other metals had higher weighting compared to silver:

- 2024: AgEq = Ag + Pb x 26.12 + Zn x 35.69 + Au x 74.25.

Table 14.6.3 2026 Silver Equivalent Assumptions

Metal	Price	Recovery*
Ag	US$35 /oz	75.6%
Pb	US$2,094 /t	81.0%
Zn	US$2,976 /t	86.4%

***Recovery values are averages calculated above 25 g/t AgEq cut-off grade.**

Variable metal recovery formulas were used, based on metallurgical testwork:

- Ag recovery = (6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%.
- Pb recovery = (8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%.
- Zn recovery = (10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%.

2026 silver equivalent formula:

- AgEq(g/t) = Ag(g/t) + Pb(%) x Pb_price/100 x Pb_recovery/Ag_price/31.103477 x Ag_price + Zn(%) x Zn_price/100 x Zn_recovery/ Ag_price/31.103477 x Ag_price.

Simplified 2026 version using average recovery values calculated above 25 g/t AgEq cut-off grade:

- AgEq(g/t) = Ag(g/t) + Pb(%) x 20.0 + Zn(%) x 30.4.

  (This is not the formula implemented for the evaluation of the MRE and is only provided as an approximation for the purposes of comparison with the 2024 formula).

Density was estimated directly by OK from the validated water displacement measurements, which were treated as point samples. Density was estimated for the Rylstone Volcanics and Coomber Formation separately using the same orientation sub-domains as the other elements for the search ellipse and variogram model, while the Shoalhaven Group was estimated separately using a flat search and variogram model. A dynamic search parallel to topography was used for the oxide zone in the Rylstone Volcanics.

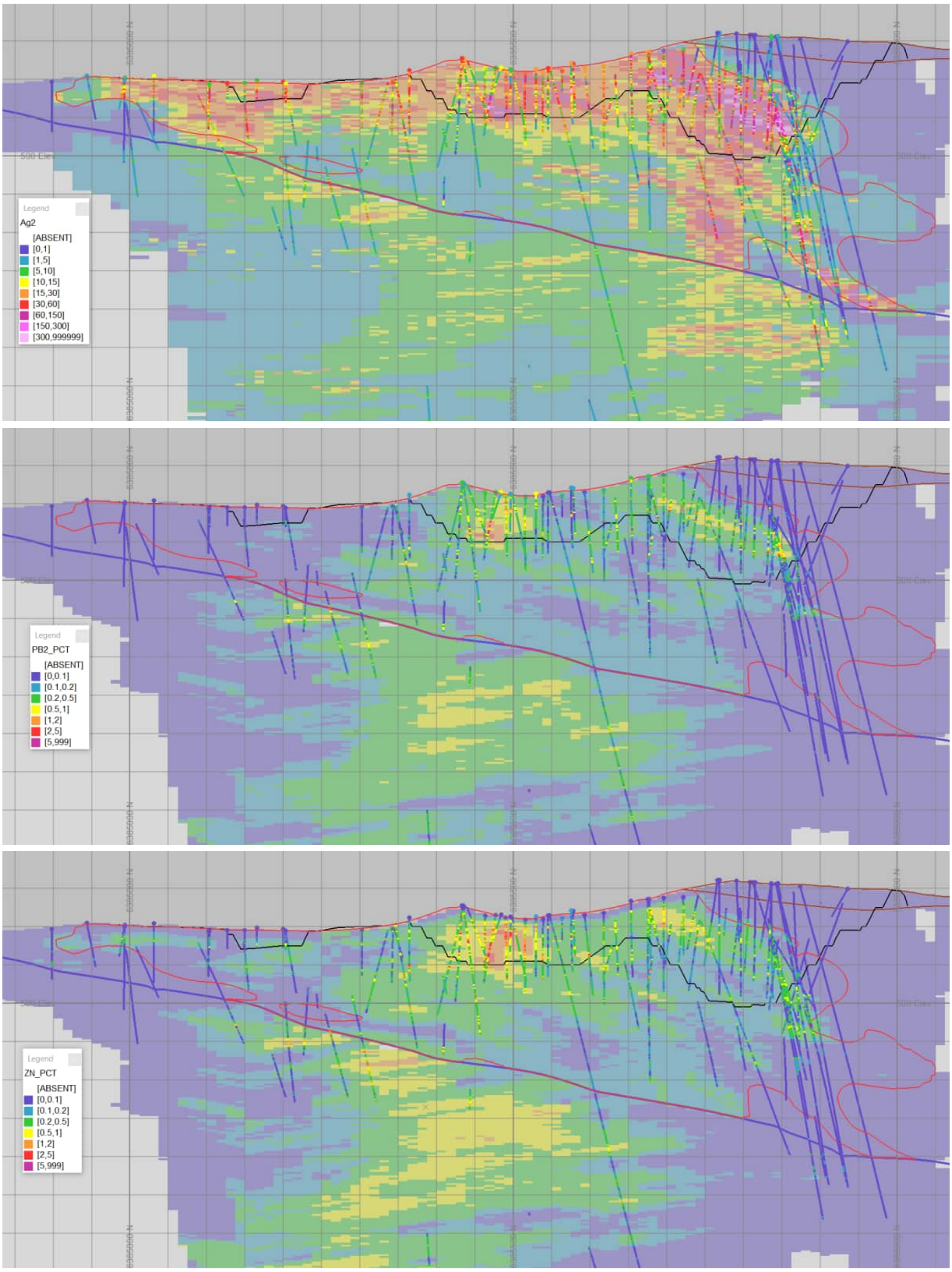
Due to the low number of density samples, the initial search was the same as Pass 2 in Table 14.6.2 with a minimum of 12 samples, while the second search equates to Pass 3 with a minimum of eight samples; the third search doubled the Pass 3 radii with a minimum of eight samples. Un-estimated blocks were assigned default values based on average values by sub-unit, while un-estimated blocks in the oxide zone used average sub-unit values discounted by 2%.

14.7 Model Validation

The new model was validated in a number of ways – visual comparison of block and drill hole grades, statistical analysis (summary statistics and swath plots), examination of grade-tonnage data, and comparison with the previous July 2024 model.

Visual comparison of block and drill hole grades shows good agreement in all areas examined and no obvious evidence of excessive smearing of high-grade assays, as shown in Figure 14.7.1 and Figure 14.7.2. The dynamic search in the oxide zone for elements depleted by oxidation appears to give reasonable results, for example zinc in Figure 14.7.1 and sulphur in Figure 14.7.2.

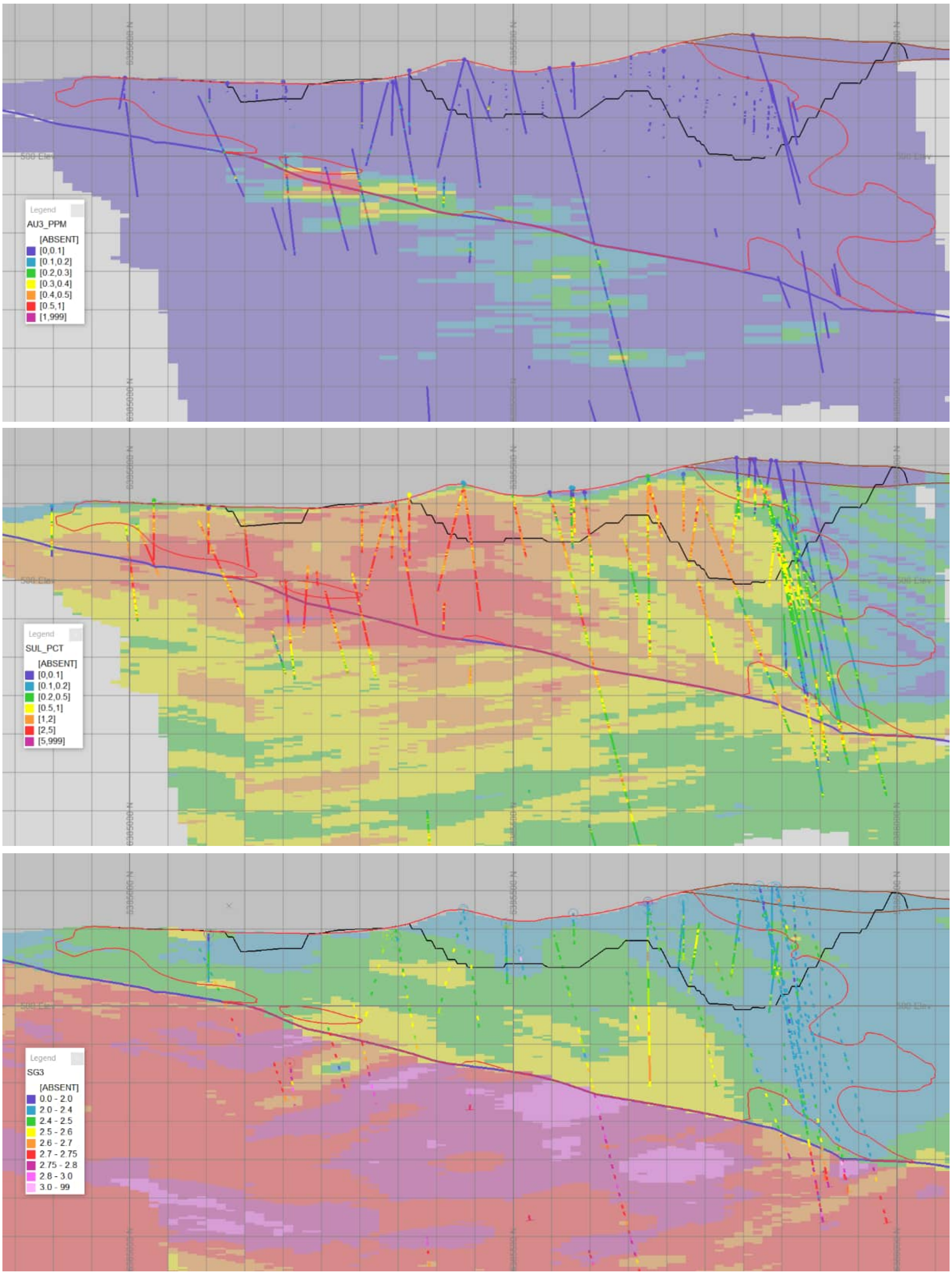
Figure 14.7.1 Long Section 768,950 mE – Model and Drillholes for Ag, Pb and Zn



Top=Ag, Middle=Pb, Bottom=Zn, All ± 25 m section window.

Lines: Blue=Base of Rylstone, Brown=Shoalhaven, Black=base of Pit, Red=Mineralised Domain.

Figure 14.7.2 Long Section 768,950 mE – Model and Drillholes for Au, S and SG



Top=Au, Middle=S, Bottom=SG, All ± 25 m section window.

Lines: Blue=Base of Rylstone, Brown=Shoalhaven, Black=base of Pit, Red=Mineralised Domain.

Global comparisons of average sample composite and block model grades for the Coomber Formation and Rylstone Volcanics (units 1 and 2 respectively) are presented in Table 14.7.1. The composite statistics are length weighted and not de-clustered, while the block grades are volume weighted. Average block grades for each attribute are generally lower than the composites for the economic elements due to the clustering of drilling in the higher-grade parts of the deposit, particularly in the Rylstone. These results are considered acceptable when data clustering is taken into account. Data in the Coomber Formation is less clustered than the Rylstone Volcanics as shown by the closer average block and sample grades.

Table 14.7.1 Average Composite and Model Grades by Unit

Element	Unit	Samples	Mean	Blocks	Mean	Blk / Sam
Ag ppm	1	18,386	5.083	673,837	4.305	84.7%
Pb %	1	18,386	0.112	789,488	0.084	75.0%
Zn %	1	18,386	0.210	789,488	0.153	72.9%
Au ppm	1	15,885	0.044	775,379	0.031	70.4%
S %	1	17,487	0.829	786,035	0.700	84.4%
Ag ppm	2	57,776	24.51	703,664	10.25	41.8%
Pb %	2	57,778	0.141	703,664	0.066	47.0%
Zn %	2	57,778	0.200	713,520	0.098	49.0%
Au ppm	2	19,329	0.031	779,731	0.012	40.2%
S %	2	37,314	0.911	718,405	0.665	73.0%

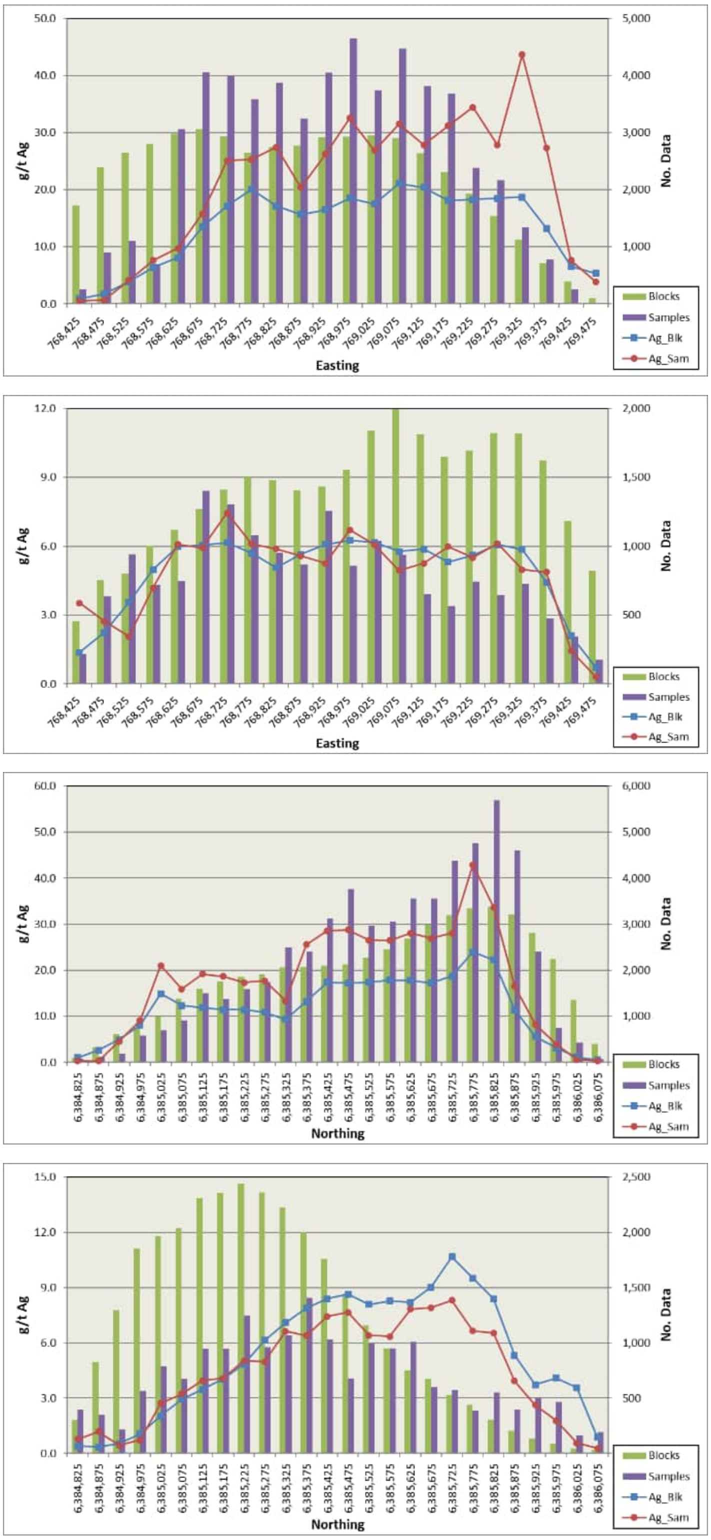
Table 14.7.2 compares average silver composite grades to average block grades by estimation pass in the Rylstone Volcanics. Pass 1 blocks have similar average grade to the composites, but block and composite grades diverge at higher pass numbers. This demonstrates the impact of data clustering and explains some of the results in the previous tables.

Table 14.7.2 Average Silver Grades by Estimation Pass

Item	Pass	Samples	Mean	Blk / Comp
Composites		57,776	24.51	100%
Blocks	1	216,951	24.28	99%
Blocks	2	106,661	10.55	43%
Blocks	3	181,967	3.55	14%
Blocks	4	198,085	0.87	4%

Figure 14.7.3 shows swath plots of average silver grades for samples and blocks in the Rylstone Volcanics and Coomber formation by Easting and Northing. Blocks are restricted to elevation above 300 metres reduced level (mRL) and Pass 1 - 3 for silver; block numbers are divided by 10 for scaling. Block grades follow a smoothed version of the trends in the sample grades, as would be expected, allowing for data clustering in the drill holes samples. This confirms that the model estimates are reasonable locally.

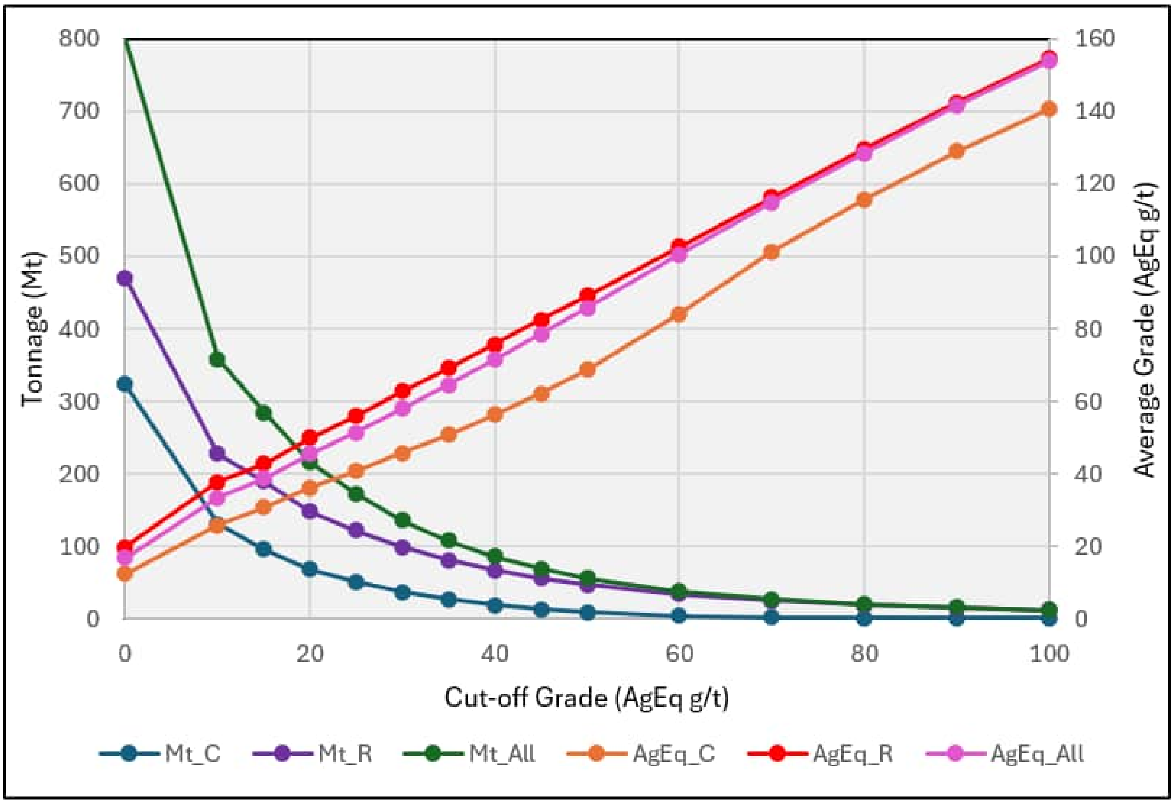
Figure 14.3 Swath Plots for Silver



Left = Rylstone, Right = Coomber.

A grade-tonnage curve for the new Bowdens block model is presented in Figure 14.7.4 at a range of 2026 silver equivalent (AgEq) cut-off grades by unit. Blocks are restricted to above 300 mRL and Pass 1 - 3 for silver. It shows a smooth transition between cut-off grades and no obvious kinks, peaks or other features suggestive of over-constraining or conditional bias in the grade estimates.

Figure 14.7.4 MRE Grade-Tonnage Curve for Bowdens by AgEq



R=Rylstone, C=Coomber.

Grade-tonnage data in Table 14.7.3 shows that all elements generally increase consistently with rising AgEq cut-off grades. Blocks are restricted to above 300 mRL and Pass 1 - 3 for silver.

Table 14.7.3 MRE Grade-Tonnage Data by AgEq Cut-Off Grade

Cut-Off	Mt	AgEq g/t	Ag g/t	Pb %	Zn %	Au g/t	S %
0	800	17	10	0.09	0.14	0.029	0.87
10	359	33	21	0.19	0.27	0.050	1.22
15	285	39	24	0.22	0.31	0.052	1.27
20	216	46	29	0.25	0.36	0.054	1.32
25	173	51	33	0.28	0.39	0.054	1.34
30	136	58	38	0.30	0.43	0.053	1.36
35	108	64	43	0.32	0.45	0.051	1.36
40	86.2	71	49	0.34	0.47	0.049	1.37
45	69.3	79	55	0.36	0.49	0.045	1.37
50	56.2	86	62	0.37	0.50	0.042	1.36
60	38.2	100	76	0.39	0.52	0.034	1.33
70	27.2	115	90	0.41	0.53	0.030	1.30
80	20.4	128	104	0.42	0.54	0.028	1.28
90	15.7	142	116	0.44	0.55	0.026	1.26
100	12.5	154	128	0.45	0.56	0.026	1.25

The updated MRE was compared to the previous 2024 open-pit model above a 30 g/t AgEq cut-off grade including gold. A comparison of the old and new models using the same 2024 AgEq formula is shown in Table 14.7.4, which reveals a slight increase in total tonnage and equivalent metal, a small increase in silver grade and a small decrease in other grades.

Table 14.7.4 Comparison of 2026 and 2024 Models

Model	Unit	Mt	g/t AgEq	g/t Ag	% Pb	% Zn	g/t Au	MozAgEq
2024	Coomber	62.7	50.4	11.1	0.358	0.521	0.153	101.6
2026	Coomber	63.5	50.0	11.4	0.353	0.512	0.149	102.1
% Diff	Coomber	1.2%	-0.8%	2.9%	-1.4%	-1.7%	-2.3%	0.5%
2024	Rylstone	115.9	62.2	42.3	0.239	0.319	0.032	231.8
2026	Rylstone	115.6	62.4	42.5	0.239	0.319	0.031	231.9
% Diff	Rylstone	-0.3%	0.3%	0.5%	-0.1%	-0.2%	-4.1%	0.1%
2024	Total	178.6	58.1	31.3	0.281	0.390	0.074	333.4
2026	Total	179.1	58.0	31.5	0.279	0.387	0.073	334.0
% Diff	Total	0.3%	-0.1%	0.5%	-0.6%	-0.7%	-2.3%	0.2%

Above 30 g/t AgEq 2024 cut-off grade.

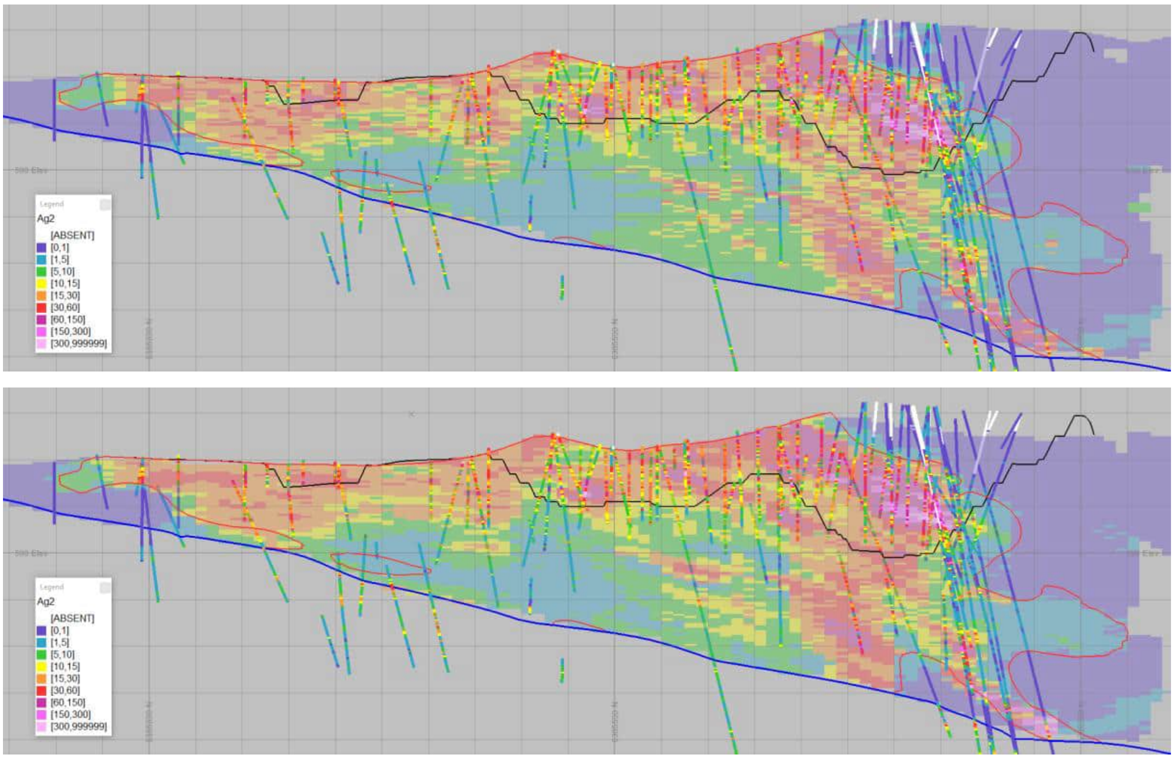
A comparison of the MIK and OK estimates for silver and gold within the Rylstone Volcanics was made by recalculating the 2024 silver equivalent formula using the OK grades for Ag and Au. The OK estimate has higher tonnage and generally lower grades than the MIK estimate above a 30 g/t AgEq cut-off grade. Visually, the OK estimates for Ag appear smoother than those for MIK, as shown in Figure 14.7.5, although gold estimates show the opposite because of the larger block size used for the gold MIK model.

Table 14.7.5 Rylstone Estimate Comparison – MIK vs. OK

Model	Mt	AgEq g/t	Ag g/t	Au g/t	Pb %	Zn %	MozAgEq
MIK	115.6	62.4	42.5	0.031	0.239	0.319	232
OK	119.2	60.4	40.5	0.033	0.237	0.317	231
% Diff	3.1%	-3.2%	-4.9%	6.8%	-0.6%	-0.6%	-0.3%

Above 30 g/t AgEq 2024 cut-off grade.

Figure 14.7.5 Long Section 768,950 mE



MIK Ag above, OK Ag below.

14.8 Resource Classification

The resource classification for the updated model is identical to the scheme used in the 2023 and 2024 MREs, based on the estimation search pass for silver. The search passes are effectively an index of proximity to sufficient data to produce reasonable estimates. Pass 1 uses data at a nominal spacing of 25 x 25 m, Pass 2 uses 50 x 50 m, and Pass 3 uses 100 x 100 m. The 2026 search radii are unchanged from 2023 - 2024, so the resource classification scheme for the new model is identical to the previous estimate.

The final Class field was derived directly from the MIK estimation pass for silver, without modification, with Class 1 material classified as Measured, Class 2 as Indicated and Class 3 as Inferred. Pass / Class 4 material could be considered to be an exploration target.

The concept of 'eventual economic extraction' was applied to the new resource estimates in the same way as in 2023 and 2024: estimates were restricted to above a nominal elevation of 300 mRL, which is effectively an average depth of around 320 m below surface. This is considered to be a reasonable depth for a potential open pit mining operation and Mineral Resources are presented on this basis in Table 14.8.1 at a 25 g/t AgEq (2026) cut-off grade. These resources effectively include all of the estimated mineralisation in the Rylstone Volcanics and around half of the estimated mineralisation in the Coomber Formation (excluding Class 4).

Table 14.8.1 Updated Mineral Resource Estimate at 25 g/t AgEq (2026) Cut-Off

Unit	Class	Mt	AgEq g/t	Ag g/t	Pb %	Zn %	Au g/t	MozAgEq
Coomber	Measured	8.2	45	14	0.39	0.65	0.13	12
Coomber	Indicated	19.2	43	13	0.39	0.61	0.14	26
Coomber	**Meas & Ind**	**27.4**	**43**	**13**	**0.39**	**0.63**	**0.14**	**38**
Coomber	Inferred	23.9	38	12	0.38	0.51	0.14	29
Rylstone	Measured	96.4	59	44	0.25	0.34	0.01	183
Rylstone	Indicated	19.1	45	34	0.18	0.24	0.03	28
Rylstone	**Meas & Ind**	**115**	**57**	**42**	**0.24**	**0.32**	**0.02**	**211**
Rylstone	Inferred	6.5	41	32	0.15	0.20	0.04	9
Total	Measured	105	58	41	0.26	0.36	0.02	195
Total	Indicated	38.3	44	23	0.29	0.43	0.08	54
Total	**Meas & Ind**	**143**	**54**	**37**	**0.27**	**0.38**	**0.04**	**249**
Total	Inferred	30.4	38	16	0.33	0.44	0.12	37
Total	**Total**	**173**	**51**	**33**	**0.28**	**0.39**	**0.05**	**286**

Notes:

1. **The effective date of the BSP MRE is July 2026.**
2. **The MRE was estimated by Arnold van der Heyden, MAusIMM and CPGeo of H&SC, who is an independent QP as defined by JORC.**
3. **The classification of the current Mineral Resource Estimate into Measured, Indicated and Inferred is consistent with current 2014 CIM *Definition Standards - For Mineral Resources and Ore Reserves*.**
4. **All figures are rounded to reflect the relative accuracy of the estimate.**
5. **All Resources are considered to have reasonable prospects for eventual economic extraction.**
6. **Mineral Resources that are not Ore Reserves do not have demonstrated economic viability.**
7. **Open pit Mineral Resources are reported at a cut-off grade of 25 g/t AgEq above a nominal elevation of 300 mRL, which is effectively an average depth of around 320 m below surface. Cut-off grades are based on a metal prices and recoveries detailed in Table 14.6.3.**
8. **The results of imposing a depth limit on mineralisation are used solely for the purpose of testing the 'reasonable prospects for eventual economic extraction' by open-pit mining and do not represent an attempt to estimate ore reserves.**
9. **There are Ore Reserves on the Property and Mineral Resources are reported inclusive of Ore Reserves.**
10. **There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading.**

14.9 Conclusions

The Bowdens database now incorporates additional drilling since the 2024 MRE was completed, with an extra 16 holes drilled totalling 2,265 m and 2,248 new silver assays. The new holes are either geometallurgical or geotechnical holes and do not have significant impact on the overall resource outline.

A significant number of holes were identified with missing assays from surface, particularly from the 2021 drilling program. Bowdens indicated they had withheld the material for potential metallurgical sampling after these gaps were identified. While some of these may be covered by nearby holes, a number occur in areas where extra assays would be very useful.

While H&SC did not perform detailed database validation or QA/QC analysis and responsibility for data quality rests with Bowdens personnel, H&SC considers that the Bowdens database is suitable for mineral resource estimation.

Bowdens personnel have a sound understanding of the geology of the Bowdens deposit and this is reflected in the wireframe models they prepared, which form a solid framework for resource estimation.

The Rylstone and Coomber formations were divided into several sub-domains to reflect the local orientation of mineralisation.

While the higher-grade veins in the Rylstone Volcanics are broadly conformable with stratigraphy, the Bundarra veins in the Coomber Formation are disconformable with stratigraphy.

Grades were generally estimated using the same parameters as the 2024 MRE, including the implementation of a mineralised domain as a hard boundary for some elements (Ag, Pb, Zn, Sb and Cd) in the Rylstone Volcanics.

Silver grades were initially estimated by recoverable MIK into larger panels and then localised by discretising the estimated metal distribution into sub-blocks.

Gold was estimated by MIK, using the e-type or average block grade estimate at the scale of the panels; this coarser resolution reflects the lower confidence in gold grade estimates due substantial under-assaying of gold compared to silver, particularly in the Rylstone Volcanics.

All other attributes were estimated by OK into sub-blocks, including Pb, Zn, S, Cu, As, Cd, Sb, Mn, Fe and dry bulk density. Dry bulk density was estimated directly by OK using available immersion method data. Bowdens also requested OK estimates for silver and gold, which were generated using uncut grades into sub-blocks.

All grades were estimated using nominal 2.0 m composites. All attributes were estimated independently and separately within each stratigraphic unit. Elements affected by oxidation (Zn, S, Cd and Mn) were estimated with search parameters oriented parallel to topography in the oxide zone. No top cutting was applied to any of the grade estimates because none of the grade distributions are strongly skewed.

The updated model was validated in a number of ways, including visual and statistical comparison of block and drillhole grades, examination of grade-tonnage data, and comparison with the previous estimate. The model was also reviewed by Bowdens personnel.

The resource classification for the updated model is based on the estimation search pass for silver, using a scheme identical to the 2023 and 2024 MREs. The MRE has been restricted to above a nominal elevation considered reasonable for a potential open pit mining operation and has been classified according to the JORC 2012 guidelines. The classification of the MRE is consistent with the 2014 CIM definitions and adheres to the 2019 CIM guidelines.

The new MRE represents a slight increase in total tonnage and equivalent metal, with a small increase in silver grade and a small decrease in other economic metal grades compared to the previous 2024 open-pit MRE.

15.0 MINERAL RESERVE ESTIMATES

Resolve Mining Solutions were engaged by Bowdens to prepare the 2026 Ore Reserve and associated mining component of the DFS.

15.1 Statement of Reserves

The definition of a mineral reserve, as defined by the JORC Code, is:

- A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include the application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.
- Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

The mineral reserve classification is also based on the degree of certainty that can be attached to the estimate. This classification is broken into two categories: Probable and Proven. These are defined by the JORC as:

- A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probably Mineral Reserve is lower than that applying to a Proven Mineral Reserve.
- A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

The estimated Mineral Reserves are shown in Table 15.1.1.

Table 15.1.1 Summary of Mineral Reserves for Bowden Silver Deposit

Classification	Reserve Grade				Contained Metal		
	Tonnes Mt	Ag g/t	Zn %	Pb %	Ag Moz	Zn kt	Pb kt
Proved	44.6	61.7	0.37	0.27	88.5	165	122
Probable	3.3	47.3	0.21	0.15	4.9	7	5
Total	**47.9**	**60.8**	**0.36**	**0.26**	**93.5**	**172**	**127**

Notes:

Ore Reserves are a subset of Mineral Resources.

Ore Reserves conform with and use the JORC Code 2012 definitions.

Ore Reserves are calculated using Silver, Lead and Zinc pricing of US$35 /oz, US$0.90 /lb and US$1.35 /lb respectively.

Ore Reserves are calculated using a Net Smelter Return cut-off grade of 30 A$ /t.

Tonnages are reported including mining dilution and recovery factors.

All figures are rounded to reflect appropriate levels of confidence which may result in apparent errors of summation.

15.2 Basis of Estimate

15.2.1 Resource Model

The pit optimisations and reserves are based on the 2026 resource model, as discussed in Section 14.0. The resource model was provided by Bowdens and was classified by Measured, Indicated and Inferred Resources. Only the Measured and Indicated Resources were used in the mining study and resultant reserve estimates.

The 2026 geological block models were provided as a MIK model (or partial percentage model). It contained resource classification, oxidation zone, and density estimates for regular blocks with dimensions of 12.5 m x 12.5 m x 2.5 m.

Topographic and geological surfaces were provided by Bowdens and reflected the latest information available.

15.2.2 Dilution and Ore Recovery

The MIK recoverable model was designed to a selective mining unit (SMU) of half the parent block size in each direction, which is down to 12.5 m x 12.5 m x 2.5 m which accounts for internal dilution and some edge dilution. As such no further mining dilution or ore loss was applied during pit optimisation or subsequent mine planning.

15.2.3 Geotechnical Considerations

Dempers & Seymour Geotechnical and Mining Consultants produced an updated pit slope design criteria in their report dated 6 August 2025 based around the 2024 Optimisation Study pit design. The report recommended the pit slopes as shown in Figure 15.2.1 and Table 15.2.1, both reproduced from the geotechnical report.

Figure 15.2.1 Geotechnical Domains

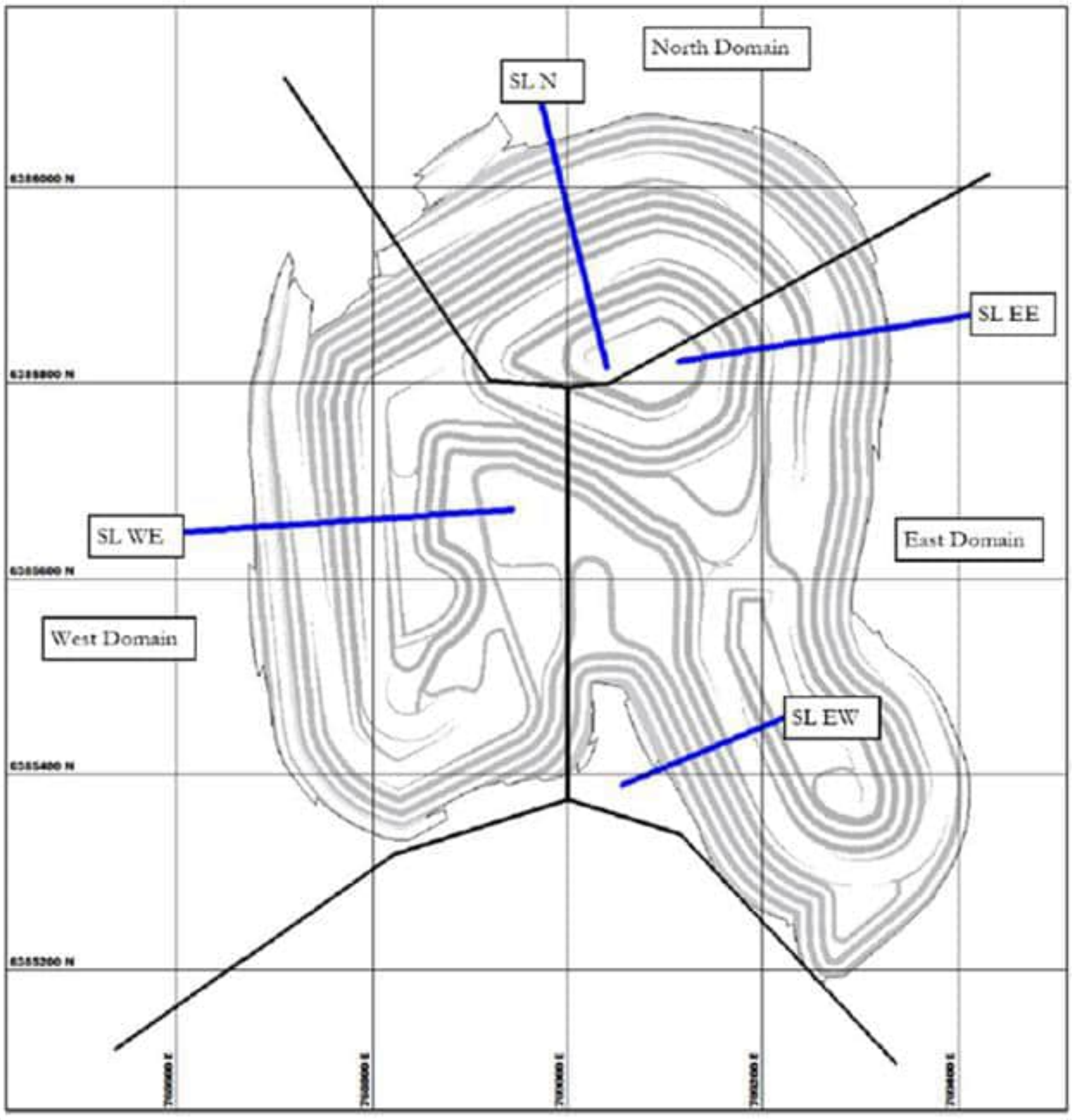


Table 15.2.1 Pit Slope Design Configuration

Domain	From	To	Bench Height m	Berm Width m	Batter Angle °	Slope Angle °
West Domain – West Wall	Surface	630	15	6	55	
	630	510	15	6	55	
	510	470	20	7	70	
	470	460	10	Pit Floor	70	51
West Domain – East Wall	Surface	585	15	6	55	
	585	570	15	6	65	
	570	470	20	7	65	
	470	460	10	Pit Floor	70	54
North Domain	Surface	625	15	6	55	
	625	565	15	6	60	
	565	505	20	7	65	
	505	440	20	7	70	50
East Domain – West Wall	Surface	630	15	6	55	
	630	570	15	6	65	
	570	450	20	7	70	52
East Domain – East Wall	Surface	630	15	6	55	
	630	570	15	6	65	
	570	450	20	7	70	52

15.2.4 Net Smelter Return

The NSR calculations utilise the metallurgical testwork results and market responses for a single bulk concentrate, while the estimated revenue prices were supplied by Bowdens from their ongoing market studies.

The NSR calculation was applied to all blocks within the resource model that had an Ag, Pb or Zn grade that produced a net profit, or loss, for that block. Any block with a positive NSR would be classified as potential mill feed and therefore included for the open pit optimisations.

Generally open pit optimisations use the block grade as the main input for their calculations. To use the NSR as the main input, the calculated net profit was divided by the block tonnage to create a grade-like element for the optimisations.

The main parameters for the NSR calculations are summarised in Table 15.2.2.

Table 15.2.2 Net Smelter Return Parameters

Parameter	Unit	Value	
Silver Recovery	%	(6.375*Ln(Ag) + 55.199)100, Max=90.0% & Min=70.0%	
Lead Recovery	%	(8.1465*Ln(Pb) + 92.677)100, Max=95.0% & Min=75.0%	
Zinc Recovery	%	(10.298*Ln(Zn) + 98.57)100, Max=97.0% & Min=75.0%	
Arsenic Recovery	%	(3.8199*Ln(As) – 10.658)100, Max=35.0% & Min=10.0%	
Gold Recovery	%	80.0%	
Concentrate Moisture Content	%	10.0%	
Concentrate Mass	% of feed	(0.5797*(Pb+Zn+S))+0.2446	
Silver Price	US$ /oz	35.00	
Lead Price	US$ /lb	0.95	
Zinc Price	US$ /lb	1.35	
Gold Price	US$ /oz	2,200	
A$: US$	-	0.69	
Silver Deduction	g/dmt	50.0	
Silver Payability (after deduction)	%	95.0%	
Lead Deduction	%	3.0%	
Lead Payability (after deduction)	%	95.0%	
Gold Deduction	g/dmt	1.0	
Gold Payability (after deduction)	%	90.0%	
Zinc Payability Matrix	%	Zn Conc. Grade	Payability
	%	0.0% - 10.0%	10.0%
	%	10.0% - 15.0%	20.0%
	%	15.0% - 20.0%	25.0%
	%	>20.0%	30.0%
Processing Cost	A$ /ore	23.62	
G&A Costs	A$ /ore	5.44	
Base Metal Refining Cost	US$ /dmt	50.00	
Silver Refining Cost	US$ /oz	0.30	
Gold Refining Cost	US$ /oz	15.00	
Container Charges	A$ /dmt	25.00	
Sampling & Assay	A$ /dmt	1.42	
Inland Freight (incl insurance)	A$ /wmt	87.03	
Sea Freight	US$ /dmt	55.00	
NSW Royalty (after deductions)	%	4.0%	
Fitzroy River Royalties	% NSR	1.0%	
Other Royalties	% Revenue	1.35%	

15.2.5 Mining Costs

The mining costs utilised for the open pit optimisations remain unchanged from the 2024 optimisation which were built up from an owner-mining scenario with cost inputs from Westrac NSW, except for the addition of 15% to allow for inflation from the estimation date.

The owner-mining cost model used a number of key parameters that are constant for each piece of equipment and for all options, these are summarised in Table 15.2.3. The main parameters to determine the mining costs are the equipment purchase and operating cost which were supplied by the manufacturer except for the labour costs. The operating costs shown in the table include the financing cost of the capital as well as operator labour, fuel, maintenance and ground engaging tools.

Drilling and blasting was excluded from the first-principles cost model, as it was agreed that the most likely scenario was than a specialist drill and blast contractor would be engaged to undertake this activity. As such, Ausdrill and Roc-Drill were approached to supply a proposal to supply a 'rock-on-ground' price and supplied the unit rates shown in Table 15.2.3. The table also shows the calculated average unit rate from the production plan supplied with the request.

Table 15.2.3 Mining Cost Parameters

Parameter	Unit	Value
Global Parameters		
Swell Factor	%	35%
Moisture Content	%	4%
Finance Rate	%	7%
Finance Duration	yrs	5
Diesel Price	A$ /L	$1.15
Equipment Purchase Price		
Excavator	A$...M	2.65
Truck	A$...M	2.25
Bulldozer	A$...M	2.00
Wheel Loader	A$...M	1.80
Wheel Dozer	A$...M	3.42
Grader	A$...M	1.27
Water Truck	A$...M	1.08
Equipment Operating Cost		
Excavator	A$ /h	456
Truck	A$ /h	332
Bulldozer	A$ /h	429
Wheel Loader	A$ /h	404
Wheel Dozer	A$ /h	485
Grader	A$ /h	323
Water Truck	A$ /h	321

Parameter	Unit	Value
Drill and Blast Unit Costs		
Establishment	A$	$168,000
Fixed Charges	A$ /month	$299,326
Diesel Charge	L/bcm	0.076
Drill and Blast		
Drilling 89 mm	A$ /m	$5.40
Drilling 102 mm	A$ /m	$5.79
Drilling 115 mm	A$ /m	$6.09
Drilling 127 mm	A$ /m	$6.39
Emulsion	A$ /t	$1,532
Downhole Delays	A$ ea	$7.77
Surface Delay	A$ ea	$8.70
Booster	A$ ea	$12.39

Bowdens supplied all the salary and on-cost parameters into the study, which were derived in-house from suitable publicly available surveys.

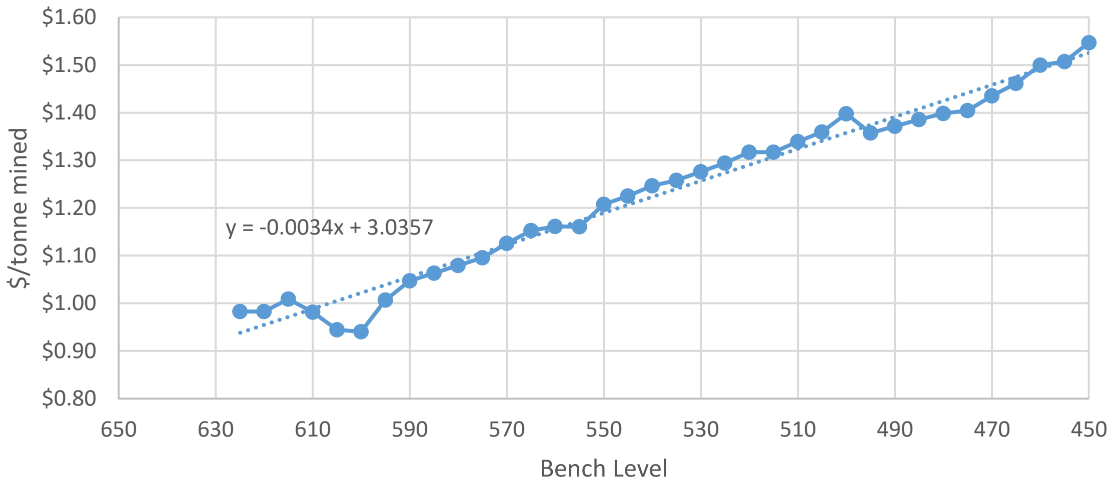
All the mining cost parameters were applied to the open pit optimisations. Generally, this is a fixed unit cost per tonne mined and then an incremental cost for depth increases.

The basis of design cost model excluded the capital component of the mining fleet and accumulated all the costs except haulage into the fixed component. The truck modelling undertaken allowed the truck hours and tonnage be calculated for each bench, thus estimating the incremental increase for pit depth. The final costs estimated for the open pit optimisations are shown in Table 15.2.4 and Figure 15.2.2.

Table 15.2.4 Mining Cost Basis of Optimisation Parameters

Parameter	Unit	Value
Equipment Ownership	A$ /t	$0.63
Labour	A$ /t	$2.84
Diesel	A$ /t	$0.81
Maintenance Parts	A$ /t	$0.92
GET / Tyres	A$ /t	$0.32
Mining Fixed Cost	A$ /t	$5.52
Mining Variable Cost	A$ /t/m	-0.0034(x bench) + 3.0357
Drill and Blast	A$ /t	$2.02

Figure 15.2.2 Incremental Haulage Cost by Bench Level



15.2.6 Material Categories

Table 15.2.5 summarises the material categories for the Reserves.

Table 15.2.5 Material Categories

Parameter	Value
Ore	
HG	> 70 NSR
MG	50 – 70 NSR
LG	30 – 50 NSR
Waste	
PAF	> 0.3% S
NAF (NMD)	0.1% – 0.3% S
NAF	< 0.1% S

15.3 Pit Optimisation Results

Pit optimisations were undertaken in Whittle software using the Pseudoflow algorithm. The Pseudoflow algorithm follows the same general principle as the Lerchs Grossman algorithm and determines a series of incremental pit shells representing the break-even points based on the given block model, slope, cost, and recovery data at varying revenue factors.

15.3.1 Optimisation Results

The initial optimisations focused on the Measured and Indicated mineralization which will allow the development of a potential ORE as the study progresses. All optimisations were undertaken to produce a pre-capital discounted cash flow (DCF) from a 2.0 Mtpa processing rate and a peak mining rate of 6.0 Mtpa.

Figure 15.3.1, Figure 15.3.2 and Table 15.3.1 show the results of Run 1. The optimisation outlined 48.3 Mt within the RF=1.0 pit shell at a strip ratio of 1.5:1. The results show that there is limited project value at revenue factors less than 0.34, however above this the value increases steadily as the revenue factor increases with no significant step changes within the range of revenue factors ran. The tonnage graph does indicate some large increases in the waste tonnage at RF=0.62 and RF=0.82, which generally correspond to the opening up of different mining areas such as the western and southern regions.

Figure 15.3.1 Optimisation Tonnage Results, Measured and Indicated Only

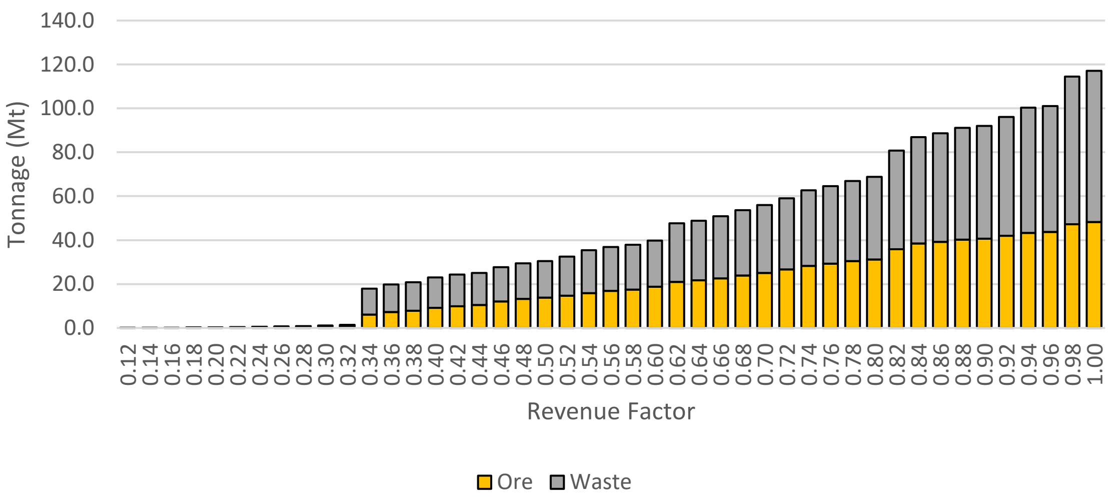
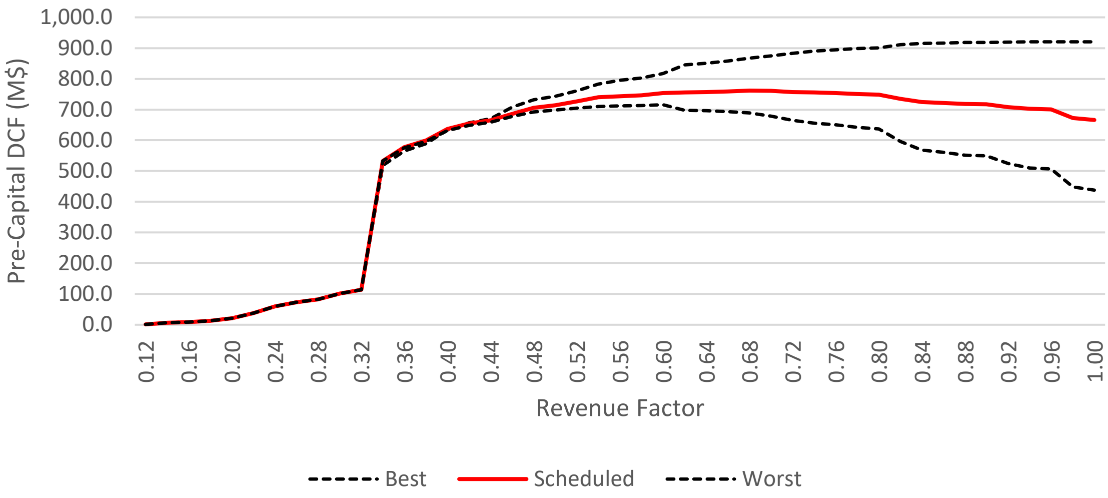


Table 15.3.1 Optimisation Results, Measured and Indicated Only

RF	Ore Mt	Ag g/t	Au g/t	Pb %	Zn %	Waste Mt	Total Mt	W:O Ratio	Conc t	DCF A$...M
0.44	10.42	100.7	0.008	0.30	0.33	14.62	25.04	1.7	119,258	$532.3
0.46	12.03	95.6	0.008	0.29	0.34	15.60	27.63	1.5	141,212	$577.1
0.48	13.19	92.8	0.008	0.28	0.33	16.29	29.47	1.4	152,164	$599.6
0.50	13.78	91.6	0.008	0.28	0.33	16.71	30.50	1.4	158,260	$636.8
0.52	14.72	89.9	0.008	0.28	0.32	17.74	32.45	1.4	168,370	$654.2
0.54	15.89	88.3	0.008	0.27	0.32	19.47	35.36	1.4	180,422	$664.8
0.56	16.87	86.2	0.008	0.27	0.32	19.93	36.81	1.3	192,917	$686.4
0.58	17.49	85.1	0.008	0.27	0.32	20.35	37.85	1.3	200,746	$706.0
0.60	18.80	82.8	0.008	0.27	0.31	21.01	39.81	1.2	217,571	$714.4
0.62	21.00	81.4	0.013	0.26	0.30	26.73	47.74	1.4	240,513	$726.1
0.64	21.68	80.4	0.013	0.25	0.30	27.19	48.87	1.3	249,255	$739.7
0.66	22.55	79.4	0.012	0.25	0.30	28.33	50.88	1.3	260,248	$743.7
0.68	23.88	77.9	0.012	0.25	0.30	29.78	53.65	1.3	278,096	$746.8
0.70	25.03	76.6	0.014	0.25	0.30	30.91	55.94	1.3	294,965	$753.6
0.72	26.59	74.7	0.014	0.25	0.31	32.40	58.99	1.3	319,281	$755.7
0.74	28.31	73.0	0.015	0.25	0.31	34.35	62.66	1.3	343,765	$757.2
0.76	29.25	72.1	0.015	0.25	0.32	35.37	64.61	1.3	359,603	$759.3
0.78	30.53	70.8	0.015	0.25	0.32	36.35	66.88	1.3	379,384	$761.7
0.80	31.21	70.3	0.015	0.25	0.32	37.55	68.76	1.3	387,750	$760.7
0.82	35.90	66.7	0.014	0.27	0.35	44.90	80.79	1.3	461,898	$756.4
0.84	38.51	65.1	0.014	0.27	0.36	48.44	86.95	1.3	505,567	$755.4
0.86	39.23	64.7	0.014	0.27	0.36	49.45	88.68	1.3	516,165	$753.5
0.88	40.24	64.1	0.014	0.27	0.36	50.90	91.14	1.3	532,560	$750.7
0.90	40.69	63.8	0.014	0.27	0.36	51.28	91.97	1.3	538,895	$748.1
0.92	42.02	63.3	0.014	0.27	0.36	54.08	96.09	1.3	553,903	$734.4
0.94	43.32	62.7	0.014	0.27	0.36	56.97	100.29	1.4	572,856	$724.2
0.96	43.66	62.5	0.014	0.27	0.36	57.41	101.08	1.4	577,200	$721.6
0.98	47.17	61.4	0.014	0.27	0.36	67.38	114.55	1.5	631,110	$718.1
1.00	48.26	60.7	0.018	0.26	0.36	68.92	117.18	1.5	649,344	$716.8

Figure 15.3.2 Optimisation DCF Results, Measured and Indicated Only



While there are inconsequential quantities of inferred in the upper regions of the resource model it is important to understand how the deeper Inferred mineralisation could affect the final pit limits should it be expanded over time. For example, increased depth to the open pit would push the pit limits out and this could impinge on the processing area or waste dumps, resulting in either additional costs to move these facilities or sterilisation of valuable mineralisation.

Therefore, an optimisation including Inferred should always be undertaken before finalising infrastructure locations, preferably the optimisation used to locate infrastructure should also have a high revenue factor to allow for increasing metal prices over time.

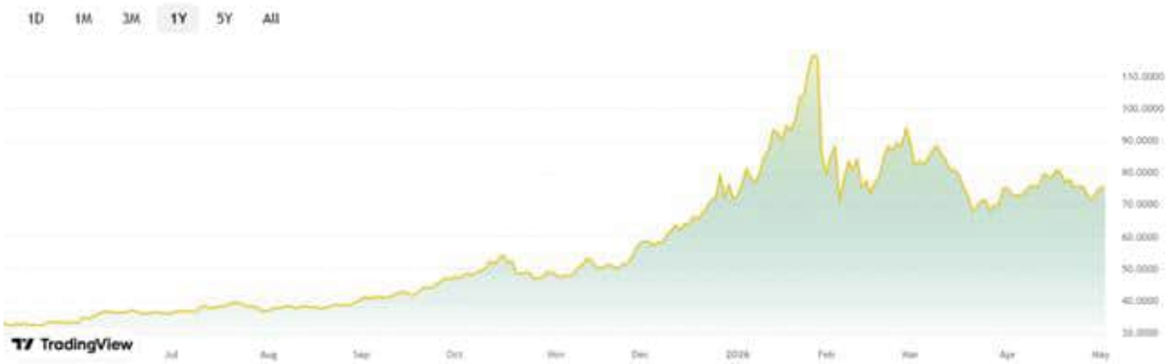
The inclusion of the Inferred mineralisation increased the mill feed to 51.0 Mt and the overall pit size to 118.3 Mt, which is not material. Table 15.3.2 summarises this run and shows that the increase in tonnage has a small negative effect on the DCF due to the additional operating time, however this is also affected by the schedule used to create the DCF. Overall, the inclusion of Inferred at the base case factors does not materially in affect the overall pit shell size or inventory.

Table 15.3.2 Optimisation Results, Including Inferred

RF 1.0	Ore Mt	Ag g/t	Au g/t	Pb %	Zn %	Waste Mt	Total Mt	W:O Ratio	Conc t	DCF A$...M
Excl	48.26	60.7	0.018	0.26	0.36	68.92	117.18	1.5	649,344	$717
Incl	51.02	59.4	0.020	0.26	0.35	67.29	118.31	1.4	678,206	$662

At the time of Reserve estimation, the silver price had been on a rising path over the last year after being relatively constant around US$25 - US$30 /oz for the previous 2 - 3 years. The price peaked at over US$110 /oz in January 2026 but has retreated and steadied around US$60 - US$70 from March 2026. Figure 15.3.3 shows the silver price for the last 12 months, highlighting the January peak and the more consistent price after this.

Figure 15.3.3 Silver Price

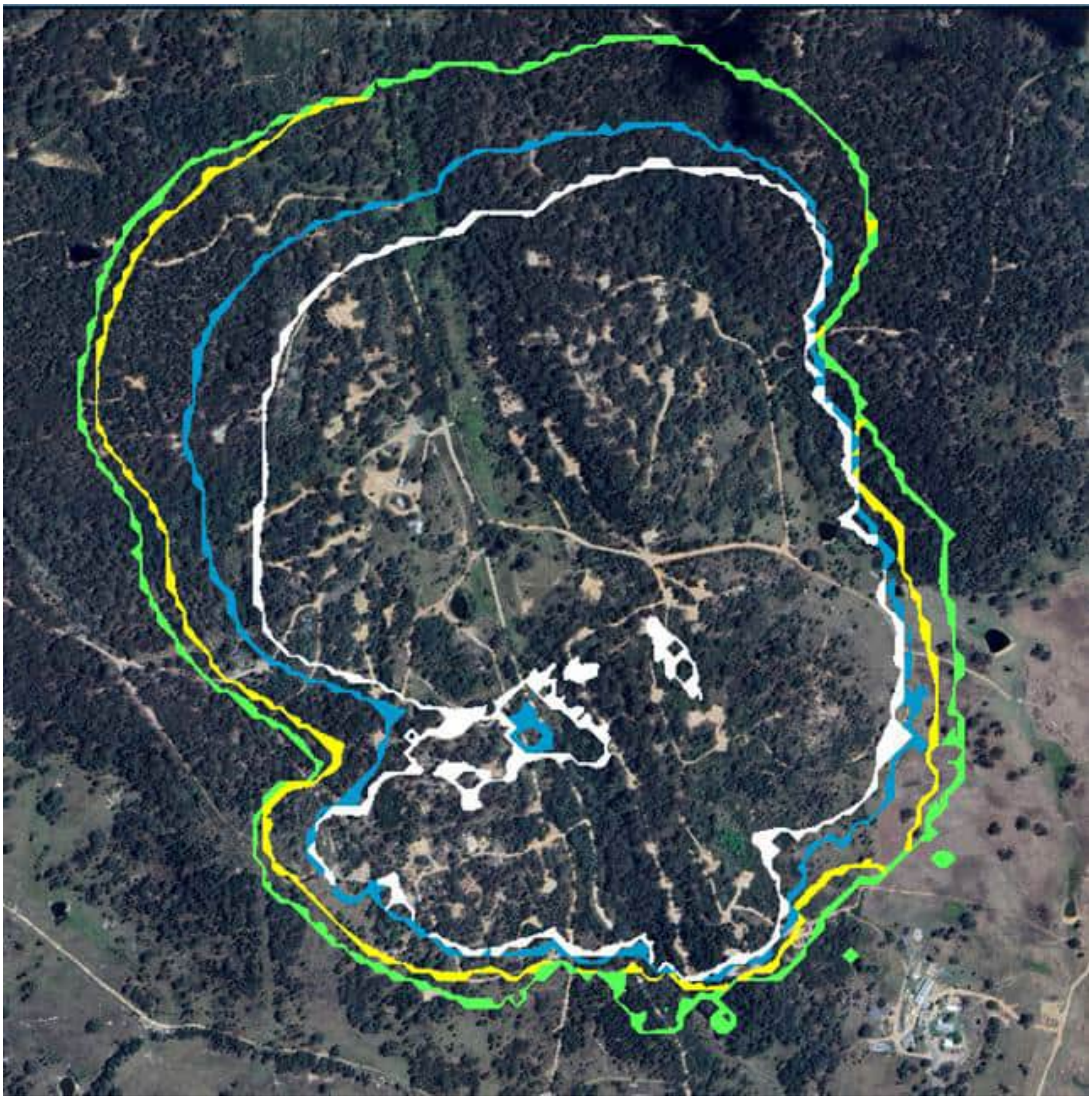


Source: www.goldbullionaustralia.com.au

The physical constraints on BSP in relation to location and size of infrastructure, in particular the main waste dump, determined that higher silver price optimisations were necessary to assess potential future impacts. Optimisations were run at US$50 /oz and US$100 /oz, both including and excluding Inferred material.

Figure 15.3.4 shows the four main results (Whittle Shells), US$35 (48.3 Mt @ 60.7 g/t Ag), US$50 (80.2 Mt @ 49.2 g/t Ag) and US$100 /oz (167.8 Mt @ 33.3 g/t Ag) cases, and then the US$100 /oz including Inferred. The shells tend to expand to the north and northwest with the eastern wall relatively similar between the three excluding Inferred shells, and the southern limits also being relatively consistent.

Figure 15.3.4 Silver Price Pit Shells - Plan View



15.3.2 Pit Shell Selection

Selection of the shells for the staged design is driven by both quantitative and qualitative factors. The incremental increase in the Whittle shells is driven by value without consideration of operational constraints such as maintaining access considering the terrain, deferring the creek and powerline diversions, and opening up pit backfilling options.

The quantitative factors are the incremental tonnages and value between shells, which allows the bringing forward of higher grades, and higher value by mining a specific shell sequence.

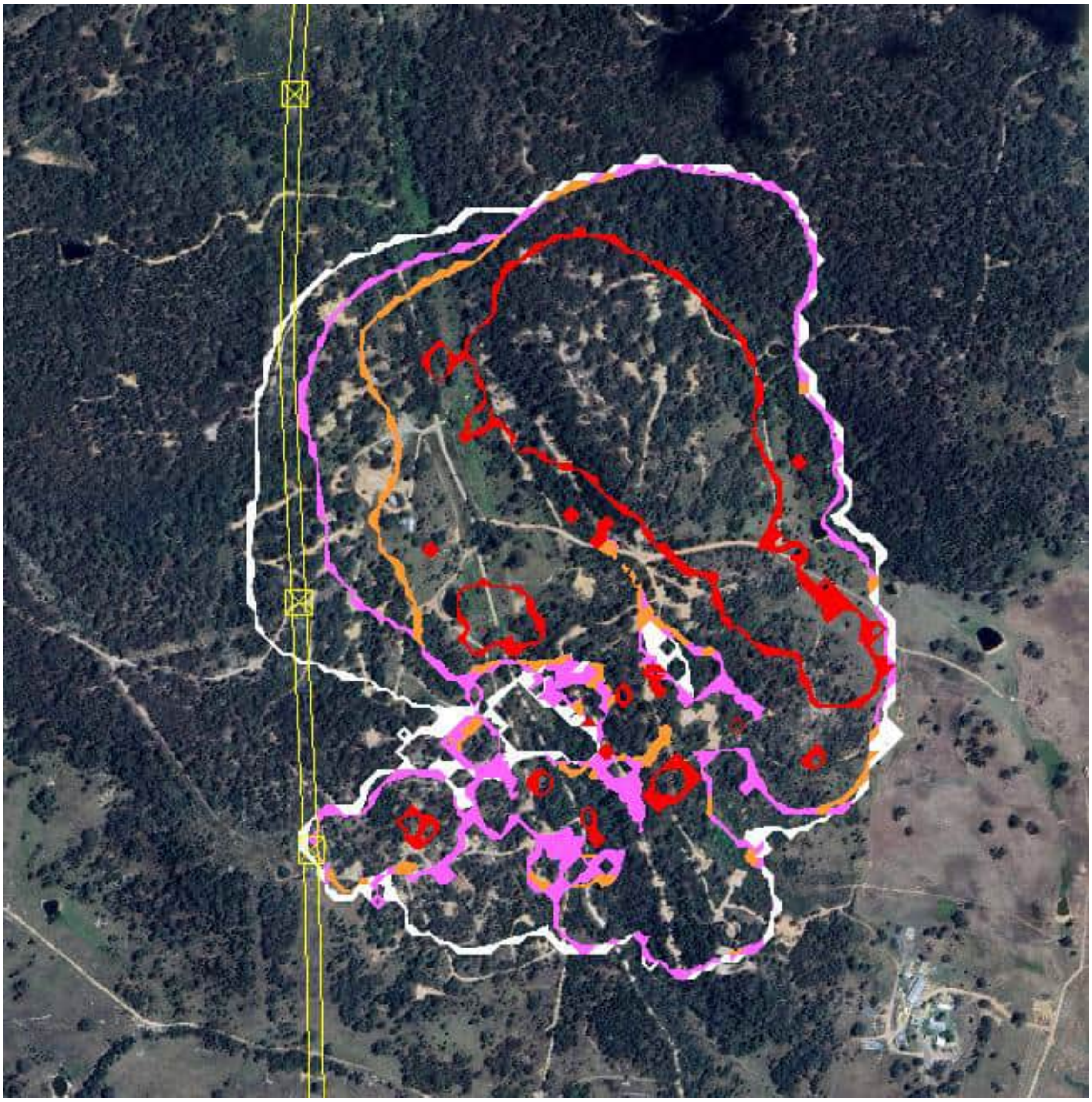
Table 15.3.1 indicates that the RF=0.34 (Shell 12) makes a suitable starter pit as it is the first one with significant ore. It contains over 6.0 Mt of mill feed so would sustain the plant for over three years and has relatively high grade.

Beyond this shell, the incremental increases are evident at RF=0.62 (Shell 26) and RF=0.82 (Shell 36) before one last step at RF=0.98 (Shell 44).

The key shells used to guide the mine design are shown in Figure 15.3.5 and include:

- Shell 12 (red) which was selected as the starter pit for both the northeast and southeast regions. The scattered pits shown are not used for the starter designs.
- Shell 35 (orange) which guides the DFS final pit design. This shell was selected over Shell 36 as it has the necessary step off from the powerline to ensure that relocation can be deferred as long as possible and the contained ore tonnage matches the DFS (Stage 1) mine design constraints.
- Shell 45 (white) being the RF=1.0 shell which guides the Ore Reserve pit design.

Figure 15.3.5 Incremental Pit Shells - Plan View



15.4 Mine Design

References to specific makes or models of equipment are not final selections but represent an appropriate size range and were used in the design, scheduling, and calculation of equipment requirements.

15.4.1 Pit Design Criteria

The pit slope constraints for the mine design are as per geotechnical investigations summarised in Section 15.2.3.

The haul road design parameters used in the pit designs are summarised in Table 15.4.1, and are based upon the manufacture's standards and best practice operating procedures. The equipment selection process nominated 100 t class trucks, with an operating width of 6.5 m, as the preferred unit and this was used for all the productivity and haulage calculation.

Table 15.4.1 Haul Road Parameters

Parameter	Unit	Value
Ramp Gradient	%	10%
Gradient Style	-	Constant
Ramp Width (double lane)	m	25

It is assumed that the safety windrow will be at least half of the height of the largest tyre in use on the road, and that road shoulders and a highwall drain have also been included in the road width calculations.

15.4.2 Waste Rock Dump Design Criteria

The waste rock dumps and stockpiles have been designed with sufficient capacity to hold all the material mined from the open pits. The design parameters are summarised in Table 15.4.2.

Table 15.4.2 Waste Dump and Stockpile Parameters

Parameter	Unit	Value
Lift Height	m	5
Lift Angle	deg	35
Final Slope Angle	Ratio	1:2.5

BSP mine waste has been geochemically classified into three categories. Type 1 material is NAF with a sulphur content of <0.1%. Type 2 material is also NAF but with the potential for Neutral Mine Drainage (NMD) of metals and has a sulphur content between 0.1% and 0.3%. Type 3 material is PAF and has a sulphur content >0.3%.

The mine design incorporates the encapsulation of Type 2 and Type 3 material inside Type 1 material. The geological model includes sulphur grades for both the ore and waste blocks. The mine design and schedule incorporates the consideration of both volume and timing of waste generation to maximise the direct placement of Type 1 waste in its final location and minimise rehandle requirements.

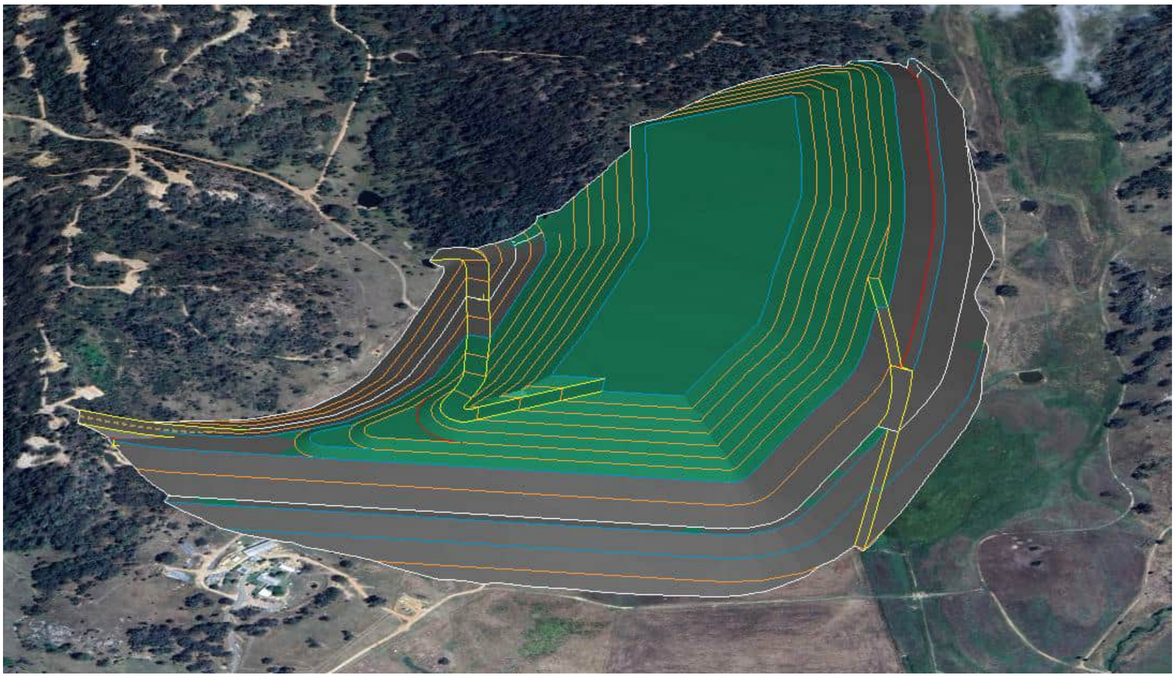
Main Waste Dump

The technical design for the main waste dump was completed by SRK as a fully encapsulated dump where the Type 3 waste is encapsulated by Type 2, which in turn is encapsulated with Type 1 waste.

Figure 15.4.1 shows the dump midway through construction including the Type 1 (grey) outer layers being designed around the inner Type 3 core. The figure shows the different slopes being used with the inner Type 3 being an 'as-dumped' steeper slope, but the outer shell being placed and shaped to the final rehabilitation slope during construction. Figure 15.4.2 shows the addition of the final encapsulation layer of Type 1 waste.

Figure 15.4.1 Main Dump, Type 1 Outer Layer and Type 3 PAF Inner Dump

Figure 15.4.2 Main Dump, Type 1 Encapsulation



Pit Backfilling

When the main waste dump is completed in Year 10, all additional mine waste is scheduled to be placed into the open pit as backfill. The Type 2 and 3 wastes must still be encapsulated and require either covering layers of Type 1 waste or ensuring that any exposed PAF dump faces are below the returning water table at closure. Hydrological studies have indicated that the water table will return to the 565 m level, therefore this is the maximum level PAF can be dumped without further encapsulation with Type 1 material.

To accommodate all the waste in the production schedule, the backfill in the eastern pit must be raised above the returning water table, and therefore will require encapsulation with Type 1 waste. Another consideration is that the southern pits mine through the Blackmans Gully Creek, and rehabilitation planning should be undertaken to replace the creek line at closure. This can be achieved by backfilling the southern pits with waste from the northwest stages.

Figure 15.4.3 depicts the post closure Reserve pit with the eastern and southern pits backfilled, the returned water level in the western pit and the diversion of Blackmans Gully Creek around the western pit and over the backfilled southern pits.

Figure 15.4.3 Type 3 PAF Backfill Layout

15.5 Pit Stage Designs

To facilitate an even mine plan and to allow the early targeting of high grade ore, seven pit stages were designed including the final Reserve pit design.

Figure 15.5.1 shows the layout of the seven stages as:

- Stage 1 - dark green.
- Stage 2 - red.
- Stage 3 – blue.
- Stage 4 - pink.
- Stage 5 – orange.
- Stage 6 - green.
- Stage 7 – grey.

A description of each stage is provided below and the individual inventories are listed in Table 15.5.1 and described below. The final inventory and design are shown in Table 15.5.2 and Figure 15.5.2.

Stage 1 - Northeast Starter Pit

The northeast starter pit targets the near surface high grade ores towards the north of the deposit. This pit needs to be commenced from the top of the ridge and worked down to the creek level, and therefore is required to have multiple ramp accesses, including one south towards the waste dump.

Within this design there is sufficient room to advance a smaller high grade cut on the West-southwest face, which will access the high grade earlier, enhancing payback time frames.

The wall on the southwest face of the pit is designed to pit limits which will allow a geotechnical review of the bater and berm configuration for long term stability.

Stage 2 - Southeast Starter Pit

The southeast starter pit is a southern extension to the Stage 1 design and is suitable for concurrent mining as is low strip ratio while remaining relatively high grade. To minimise land disturbance and manage haulage distances, the ore from this stage should be hauled through Stage 1 to the crushers.

Stage 3 - Northeast Cutback

Stage 3 is a final wall cutback to the northeast of the starter pit, but does extend sufficiently south to allow a deepening of the southeast starter pit. This stage also connects to Stage 2 to allow the use of the ramps for waste haulage to the main dumping area.

Stage 4 - Northwest Starter Pit

The northwest starter pit is size limited to ensure Stages 1 - 5 do not exceed 30 Mt of ore and does not require the relocation of the 500 kV powerline (DFS mine design).

Stage 4 includes a temporary creek diversion and haul road on the western wall and connects to the main pit on the east upper benches. Ore haulage is planned to exit the pit to the north, while waste will exit to the south for access to the main waste dump. Prior to completion of this Stage 4, in-pit backfilling of Stage 3 will be available.

Stage 5 - Southeast Cutback

The southeast cutback is the final stage of the DFS mine design. For the Reserve mine design and schedule, Stage 5 is mined before the Stage 4 to allow backfilling of the eastern pits to extend south. The stage aligns all the walls, except the northwest corner to final designs.

Stage 5 ore haulage will be through the Stage 3 ramp to exit north of the pit while waste will exit via a southern ramp for access to the main waste dump.

Stage 6 - Southern Pits

The southern pits are four small pits that merge into one at the upper levels, and are standalone pits that only connect to the eastern areas in the upper benches. These pits are lower grade and contain less lead. They do, however, have elevated gold grades compared to the rest of the deposit. All haulage will exit to the north, using the creek diversion around Stage 4 as a haul road, with waste being deposited as backfill into the eastern pits.

Stage 7 - Northwest Cutback

The northwest cutback is the final stage of the Reserve design and requires the relocation of the 500 kV powerline and the installation of the final creek diversion.

The stage is mined in two parts - one for the main ore production requirements of the schedule, and the other being to ensure the final creek diversion is installed to manage surface water.

Figure 15.5.1 Pit Stage Layout

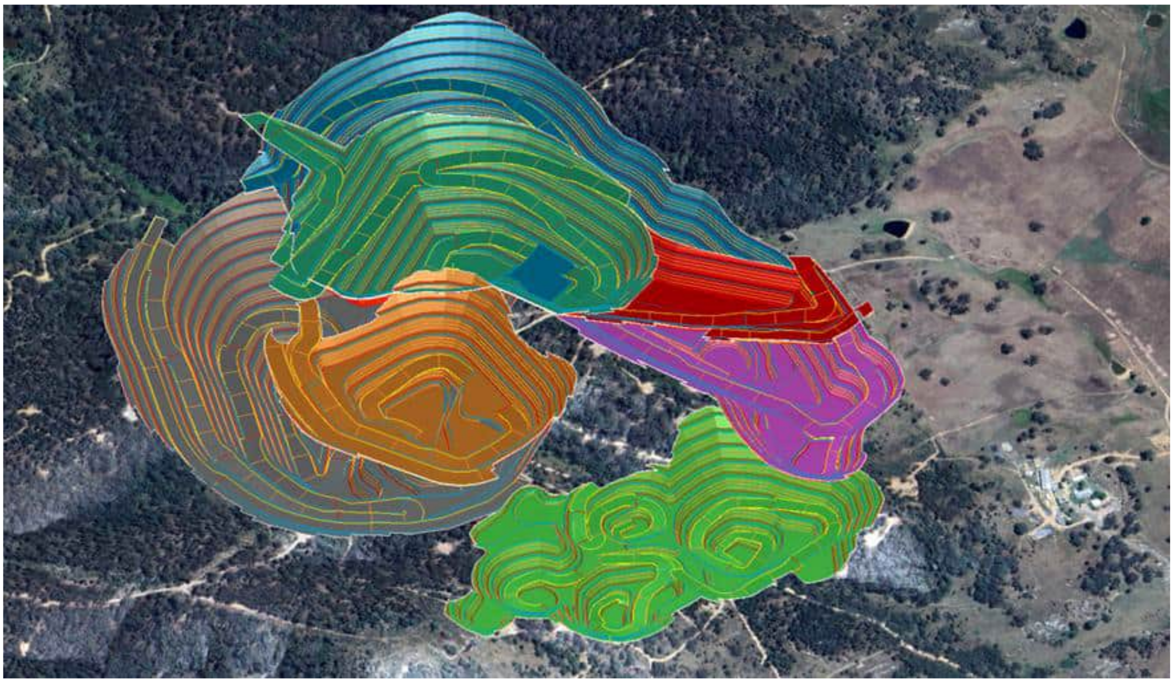


Table 15.5.1 Pit Stage Inventory

Stage	Material	Mt	Ag g/t	Pb %	Zn %
1	HG	4.77	100	0.35	0.30
	LG	1.98	30	0.26	0.26
	ORE	***6.75***			
	PAF	4.33			
	NAF	6.36			
	Total	**17.44**			
2	HG	1.98	86	0.24	0.45
	LG	0.86	30	0.21	0.36
	ORE	***2.84***			
	PAF	0.98			
	NAF	0.03			
	Total	**3.85**			
3	HG	4.75	98	0.23	0.18
	LG	2.29	33	0.10	0.14
	ORE	***7.04***			
	PAF	4.45			
	NAF	13.00			
	Total	**24.49**			

Stage	Material	Mt	Ag g/t	Pb %	Zn %
4	HG	2.22	68	0.31	0.57
	LG	1.98	29	0.24	0.42
	ORE	***4.20***			
	PAF	3.32			
	NAF	0.93			
	Total	**8.45**			
5	HG	3.37	67	0.22	0.41
	LG	3.70	31	0.20	0.37
	ORE	***7.07***			
	PAF	9.02			
	NAF	0.03			
	Total	**16.12**			
6	HG	2.45	67	0.08	0.18
	LG	2.24	31	0.06	0.12
	ORE	***4.69***			
	PAF	5.26			
	NAF	2.15			
	Total	**12.10**			
7	HG	7.20	68	0.42	0.51
	LG	6.36	28	0.29	0.44
	ORE	***13.56***			
	PAF	16.19			
	NAF	12.54			
	Total	**42.29**			

Figure 15.5.2 Final Pit Design

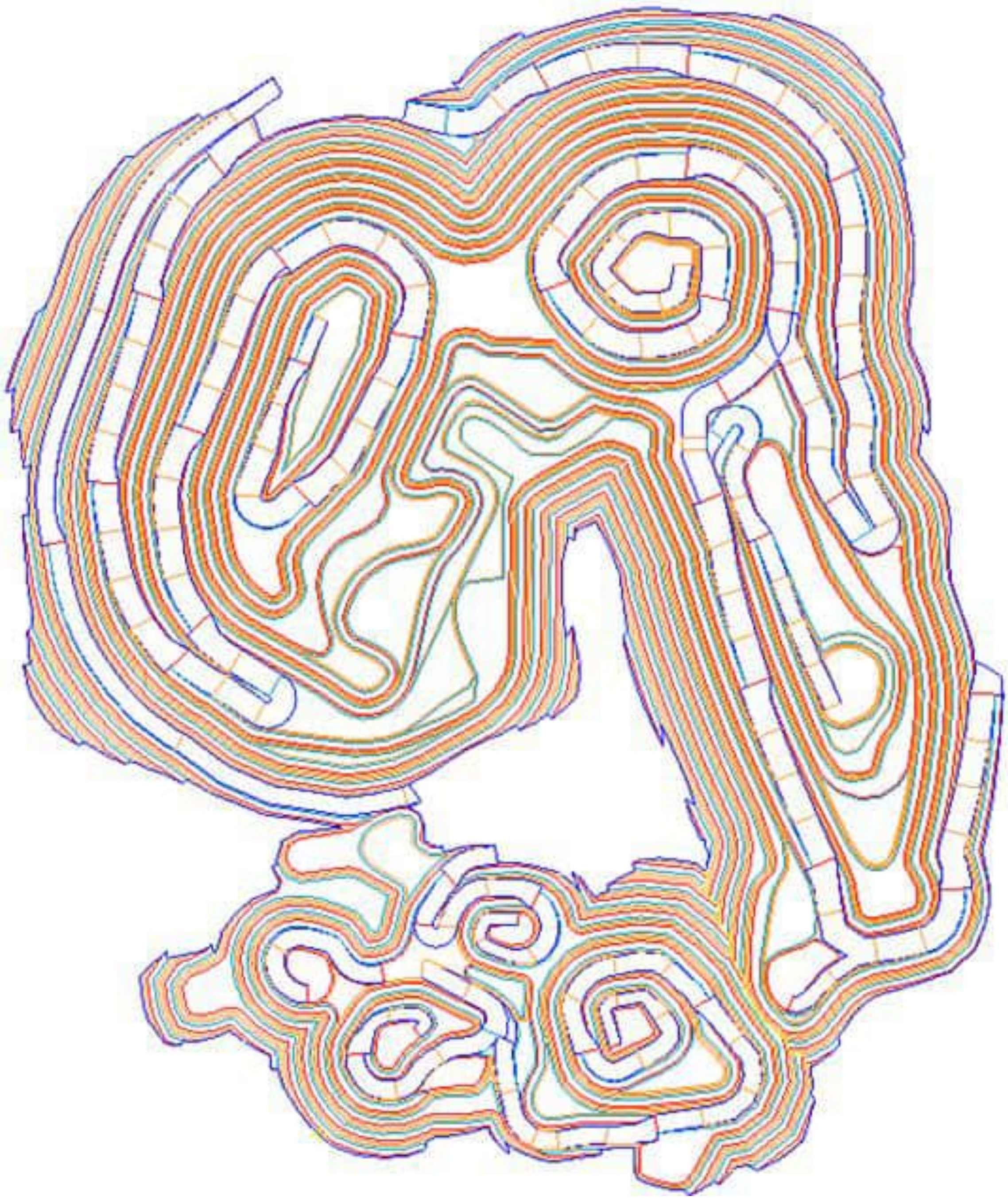


Table 15.5.2 Final Pit Inventory

Stage	Material	Mt	Ag g/t	Pb %	Zn %
All	HG	28.34	82.0	0.30	0.38
	LG	19.82	30.1	0.22	0.33
	ORE	48.16	60.6	0.26	0.36
	PAF	54.71			
	NAF	26.13			
	Total	**128.00**			

15.6 Risk Assessment

A simple risk assessment, Table 15.6.1, for the mine development stage of the Project identified risks related to the investigations, data quality, estimations, planning, and decision making.

Table 15.6.1 Mine Development Risk Assessment

Risk	Likelihood of Occurrence	Potential Severity of Impact	Risk Ranking
Over-estimation of mineral resources	Possible	Minor	Medium
Under-estimation of mineral resources	Possible	Minor	Medium
Under-estimation of mining costs for reserve estimate	Possible	Minor	Medium
Under-estimation of processing and administration costs	Possible	Minor	Medium
Over-estimation of metallurgical performance for reserve estimate	Unlikely	Minor	Low
Over-estimation of silver price for reserve estimate	Unlikely	Moderate	Medium
Over-estimate of ore reserves	Unlikely	Moderate	Medium
Pit slopes are too steep	Possible	Minor	Medium
Feasibility design is not fully achievable	Unlikely	Minor	Low
Unsuitable mining method	Unlikely	Minor	Low
Production schedule is not achievable	Unlikely	Minor	Low

The pit limits are not significantly sensitive to change in study parameters; however, the design has allowed for a buffer between the pit limits and any permanent infrastructure or waste rock dumps.

The silver price used for the mineral reserve calculations is conservative given current pricing and long term forecast, and the optimisation parameters are robust enough to withstand a decrease in silver price from 2026 levels while remaining economic.

16.0 MINING METHODS

16.1 Mining Method

16.1.1 DFS Versus LOM Mine Plans

The DFS specifically references a 29.9 Mt ore mine plan, which is a sub-set of the LOM mine plan which contains 48.2 Mt ore. Both the DFS and LOM mine plans contain small amounts of Inferred Resources as demonstrated in Table 16.1.1.

Table 16.1.1 DFS and LOM Mine Plan Material Categories

Resource Category	Reserve Category	Unit	Mine Plan			
			DFS	%	LOM	%
Measured	Proven	t (dry)	28,572,550	95.5%	44,606,742	92.6%
Indicated	Probable	t (dry)	1,222,004	4.1%	3,251,400	6.8%
Meas & Ind	Proven & Probable	t (dry)	29,794,554	99.6%	47,858,141	99.4%
Inferred	N/A	t (dry)	113,130	0.4%	303,842	0.6%
Total Mine Plan		t (dry)	29,907,684	100.0%	48,161,983	100.0%

The DFS is based on AACE International Class 3 or better engineering designs and cost estimates for all site infrastructure and operations, based on significant field and laboratory test programs.

The DFS mine plan is considered Stage 1 and has been intentionally limited in scale to match the Stage 1 filtered tailings landform (FTL) which holds approximately 30 Mt of filtered tailings. The FTL (Section 18.7) is located in the Walkers Creek valley which has southern, northern and main valley arms. The southern arm is the most practical starting point for the FTL and based on the design, will hold 16.7 Mm^3 at 1.8 t/m^3 or 30 Mt of compacted filtered tailings.

The larger LOM plan (Stages 1 and 2 combined) includes the additional 18.3 Mt in the LOM plan and requires a Stage 2 filtered tailings landform, while all other mine infrastructure in the DFS design remains adequate. The design of the Stage 2 filtered tailings landform is at pre-feasibility level. All aspects (other than Stage 2 of the filtered tailings landform) of the LOM plan are at DFS (AACE International Class 3) level.

The mining of the DFS and LOM mine plans are discussed together in this section, but it should be noted that the level of detail and design for the DFS component (Stage 1, first 14 years) is higher than for the remaining years. The DFS component satisfies an AACE International Class 3 engineering design while the LOM component only needs to demonstrate being economically mineable under reasonable financial assumptions.

16.1.2 General Approach

Based on the geometry and physical properties of the deposit and the surrounding terrain and the proximity of the deposits to surface, the most appropriate mining method is conventional open pit mining. These properties also allow for a smooth mine production schedule and equipment requirement profile. Therefore, a majority owner mining operation is planned. Contractors are planned for drill and blast operations, and equipment maintenance (with selected equipment supplier).

The mining strategy incorporates mining at a relatively constant 5.5 Mtpa rate with pit staging primarily targeting highest grade first but also considering operational constraints. In the early years, ore with an NSR >A$80 /t is process while ore with an NSR A$80 /t is stockpiled. Note that mining rates are limited to be the same as the original approval with ore production capped at 2.58 Mtpa.

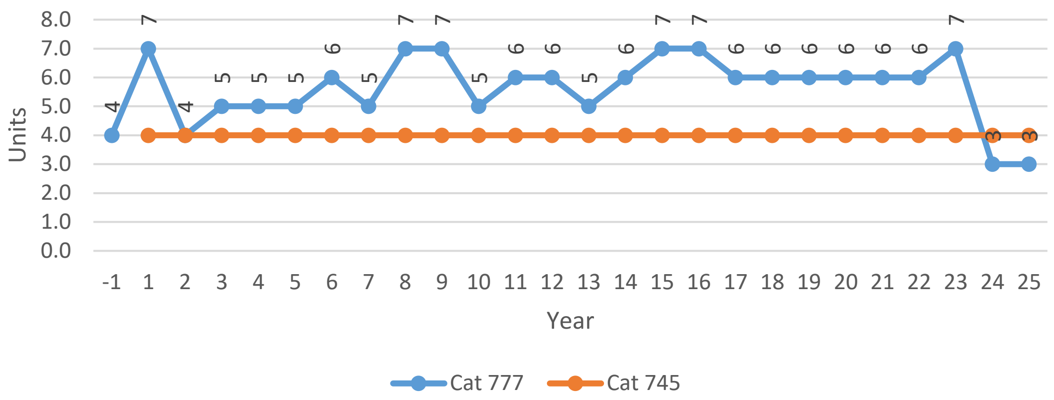
16.1.3 Mining Equipment

The DFS mine schedule and costs are based on a Caterpillar fleet with an OEM maintenance contract. This does not represent a final equipment selection in terms of make or model; rather it allowed for simple and accurate scheduling and cost estimation through information provided by Westrac (Caterpillar agent).

Caterpillar 777 (90 t) trucks were selected based upon the density of the fresh rock material matched to a tray volume that also allowed for a full load of oxide and transition material. The Caterpillar 6020 excavator was matched to the haul trucks for the purposes of DFS mine scheduling. A single 6020 excavator handles the full DFS production schedule. For the LOM schedule, the capacity of a single 6020 excavator is marginally exceeded during Year 15 when the permanent creek diversion is installed. The additional limited duration loading capacity will be supplied by the back-up / batter puller excavator.

The truck fleet fluctuates between five and seven trucks for most of the DFS and LOM schedules as demonstrated in Figure 16.1.1. Further detailed scheduling prior to operations will attempt to smooth the truck fleet requirement out, especially the need for a seventh truck in Year 1.

Figure 16.1.1 Haul Truck Fleet Requirements



Mining operations will also include the loading, haulage, placement and compaction of tailings on the FTL. Four Caterpillar 745 ADT ejector trucks are required for this haulage.

The Caterpillar equipment models used for the DFS primary mining and ROM operations fleet are:

- 6020 excavator.
- 777 trucks.
- D9T bulldozers.
- 844 wheel dozer.
- 14M grader.
- 745 ADT watercarts.
- 988 wheel loaders.

Additional fleet for FTL operations is:

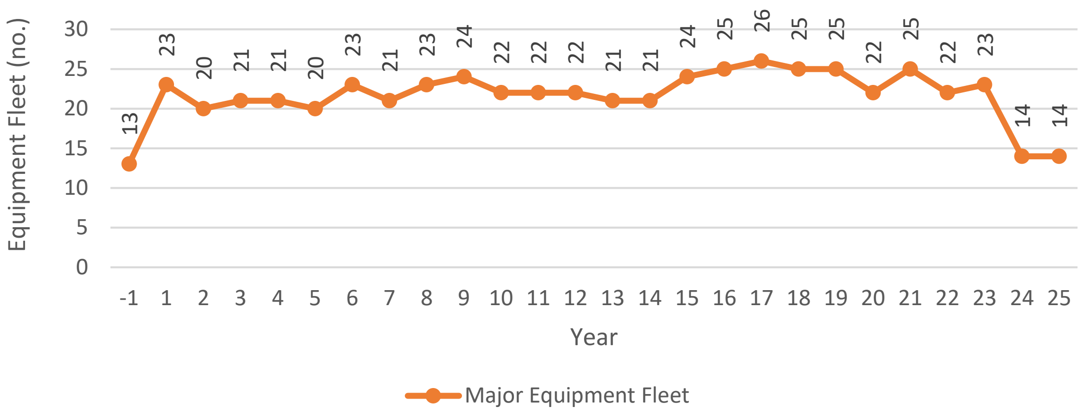
- 988 wheel loader.
- 745 ADT ejector trucks.
- 745 ADT watercart.
- 825 soil compactor.
- D9T bulldozer.

The DFS cost model also includes a fleet of support equipment such as light vehicles, lighting plants, fuel truck, rock breaker and mobile cranes. Note that mobile maintenance vehicles are included in the OEM maintenance service agreement.

The drill fleet is based contractor submissions which includes planned penetration rates for the various material types and expected utilisations.

The total mining mobile equipment fleet over the LOM plan is shown in Figure 16.1.2, which indicates a peak requirement of 26 units of heavy plant.

Figure 16.1.2 Total Fleet Requirements



16.1.4 Personnel Levels

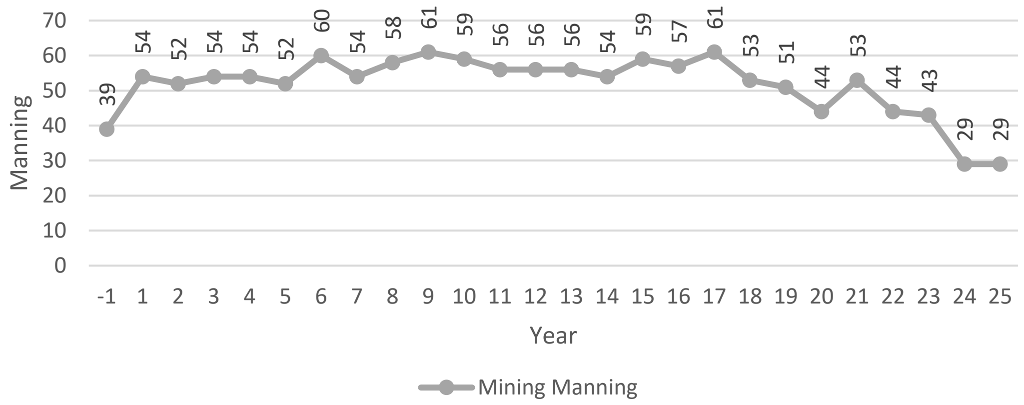
Personnel levels, and associated costs, are estimated from the equipment list and the production schedule, based on:

- Day shift only primary mining and FTL operations on a two-panel roster.
- 24/7 ore processing.

The operator numbers required were estimated directly from the operating machinery with additional personnel for labourers and ROM loading operations. Service and maintenance personnel are not included in the manning costs structure as these are included under the OEM maintenance service agreement, however a Maintenance Superintendent was retained for the company. The drill and blast supervision, operators and maintenance personnel are included in the contractor numbers.

Total operational personnel numbers are shown in Figure 16.1.3, excluding management, administration and technical services.

Figure 16.1.3 Mining Operators and Supervision Levels



16.2 Mine Production Schedule

The production schedule activities include primary production, drill and blast, ROM operations, FTL operations and stockpile rehandle to provide the optimal blend of ore to the processing plant throughout the LOM.

16.2.1 DFS Mining Schedule

Table 16.2.1 and Figures 16.2.1 to 16.2.2 show the mining and processing schedule for the 14 year DFS mine schedule, with 18 months of low grade processing after mining is completed.

Figure 16.2.1 DFS Annual Mining by Pit Stage

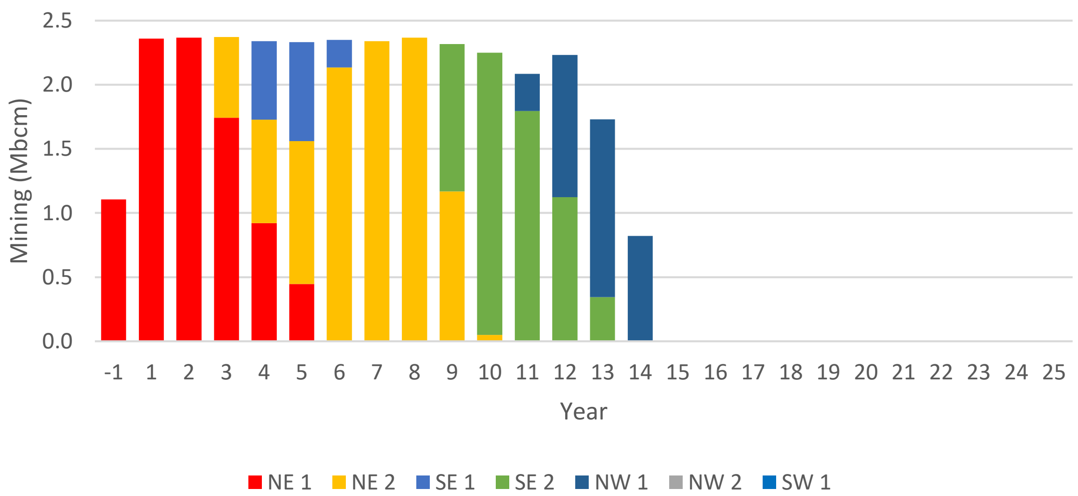


Figure 16.2.2 DFS Annual Mining by Material Type

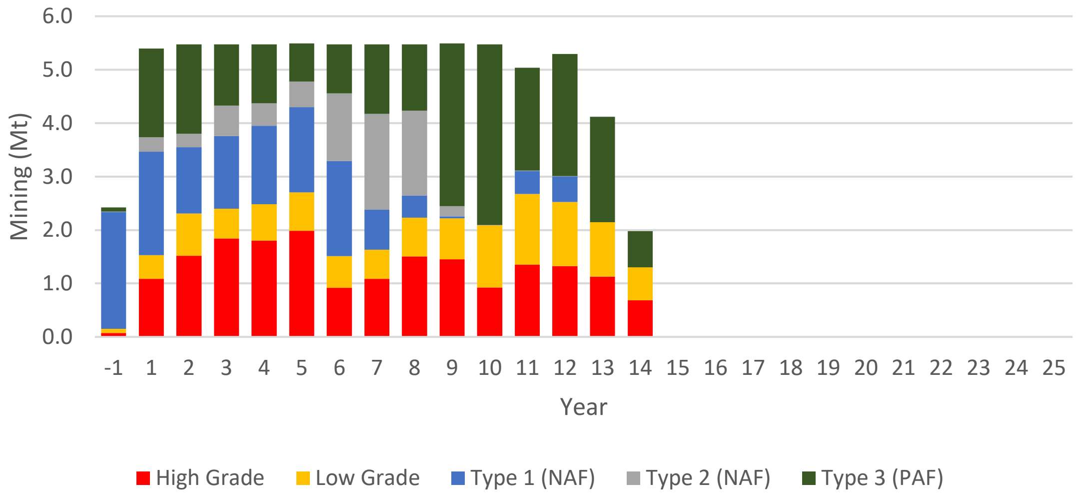


Table 16.2.1 DFS Production Schedule

Description	Unit	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	21	23	24	25	TOTAL
Ore Processing																												
Total Mill Feed	**Mt**	**0.00**	**1.57**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**0.32**										**29.91**
	Ag	0.0	77.4	81.6	89.7	98.8	96.5	57.9	72.9	76.1	77.3	56.9	56.8	59.1	50.6	42.4	27.3	27.3										**67.5**
	Pb	0.00	0.26	0.28	0.35	0.33	0.30	0.18	0.19	0.21	0.24	0.22	0.20	0.20	0.25	0.27	0.27	0.27										**0.25**
	Zn	0.00	0.38	0.36	0.32	0.28	0.36	0.24	0.20	0.16	0.21	0.33	0.40	0.34	0.49	0.51	0.43	0.43										**0.34**
Indicated (or better)	Mt	0.00	1.57	2.00	2.00	1.99	1.99	1.96	1.99	2.00	2.00	1.99	2.00	1.99	2.00	2.00	1.99	0.31										**29.79**
Inferred	Mt	0.00	0.00	0.00	0.00	0.01	0.01	0.04	0.01	0.00	0.00	0.01	0.00	0.01	0.01	0.00	0.01	0.00										**0.11**
	%	0%	0%	0%	0%	0%	1%	2%	1%	0%	0%	0%	0%	0%	0%	0%	0%	0%										**0%**
Mining																												
HG	Mt	0.08	1.08	1.52	1.84	1.80	1.99	0.92	1.09	1.51	1.45	0.92	1.35	1.33	1.13	0.68	0.00	0.00										**18.69**
LG	Mt	0.07	0.45	0.79	0.56	0.68	0.72	0.59	0.55	0.72	0.77	1.16	1.33	1.20	1.02	0.62	0.00	0.00										**11.22**
Waste (Type 1)	Mt	2.19	1.94	1.24	1.36	1.47	1.59	1.78	0.74	0.42	0.03	0.00	0.43	0.48	0.00	0.00	0.00	0.00										**13.68**
Waste (Type 2 & 3)	Mt	0.09	1.93	1.93	1.72	1.52	1.19	2.18	3.10	2.83	3.24	3.39	1.93	2.29	1.97	0.68	0.00	0.00										**29.97**
Total Mined	**Mt**	**2.43**	**5.40**	**5.47**	**5.47**	**5.47**	**5.49**	**5.47**	**5.47**	**5.47**	**5.49**	**5.47**	**5.03**	**5.29**	**4.12**	**1.98**	**0.00**	**0.00**										**73.56**
Topsoil	Mt	0.03	0.05	0.01	0.02	0.05	0.05	0.02	0.00	0.01	0.01	0.01	0.01	0.01	0.00	0.00	0.00	0.00										**0.29**
Ore Rehandle	Mt	0.00	0.21	0.34	0.11	0.02	0.37	0.93	0.43	0.08	0.15	0.50	0.18	0.18	0.25	0.98	2.00	0.32										**7.06**
Type 1 Rehandle	Mt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.76	0.76	0.76	0.45	0.00	0.00	0.00	0.00	0.00										**2.73**
Total Movement	**Mt**	**2.46**	**5.66**	**5.83**	**5.60**	**5.55**	**5.92**	**6.43**	**5.91**	**6.33**	**6.41**	**6.74**	**5.68**	**5.49**	**4.37**	**2.97**	**2.00**	**0.32**										**83.64**

16.2.2 LOM Mining Schedule

Figures 16.2.3 and 16.2.4, and Table 16.2.2 show the mining and processing schedule for the 23 year LOM schedule, with two years of low grade processing after mining is completed.

Figure 16.2.3 Annual Mining by Pit Stage

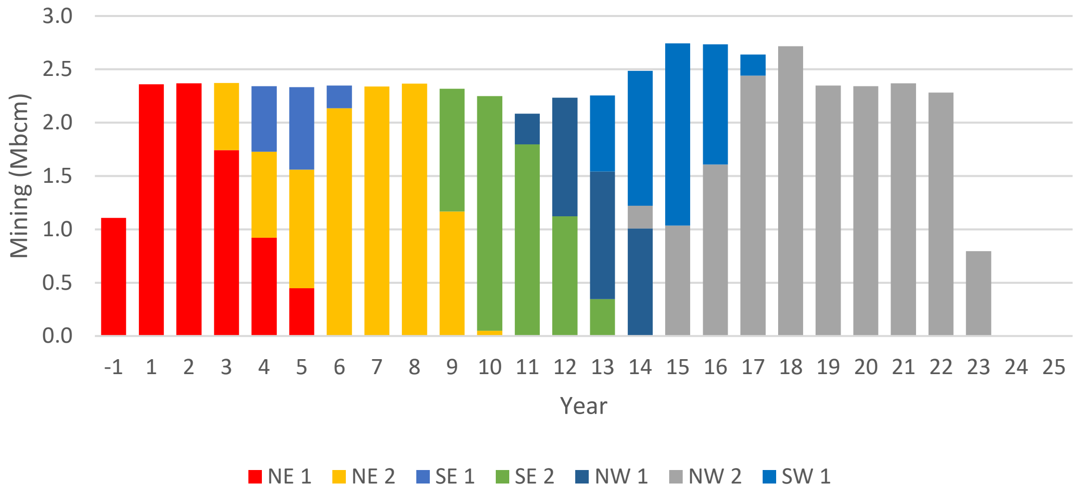
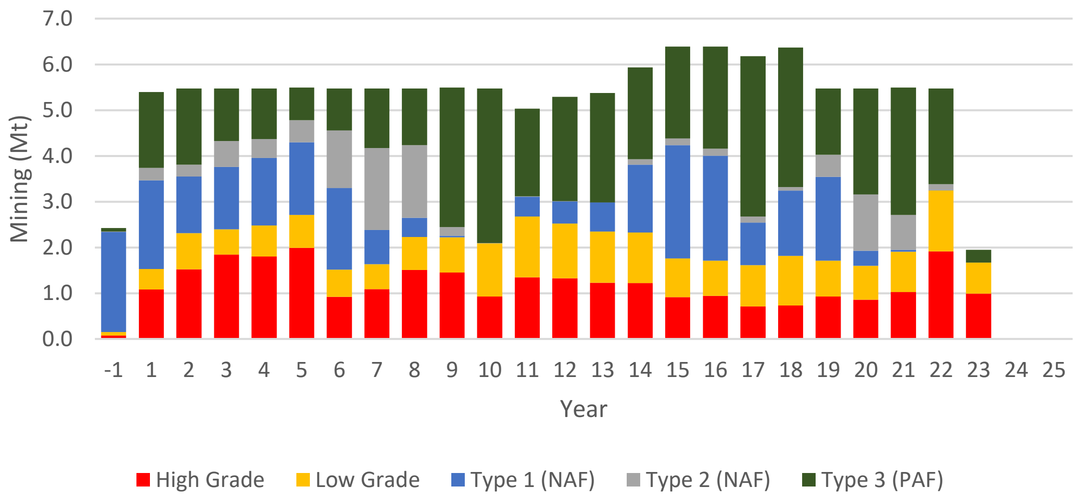


Figure 16.2.4 Annual Mining by Material Type



The additional tonnes in the LOM plan results in additional waste rock and tailings. The waste rock is primarily placed as in-pit backfill in the completed early stages of the open pit. An additional FTL will be required, located in the adjacent valley to the north of the DFS FTL.

Table 16.2.2 LOM Production Schedule

Description	Unit	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	21	23	24	25	TOTAL
Ore Processing																												
Total Mill Feed	**Mt**	**0.00**	**1.57**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**2.01**	**2.00**	**2.00**	**2.00**	**0.56**	**48.16**
	Ag	0.0	77.4	81.6	89.7	98.8	96.5	57.9	72.9	76.1	77.3	56.9	56.8	59.1	53.3	55.3	48.8	48.2	40.4	41.0	46.4	45.4	51.7	63.5	47.9	25.6	25.6	60.6
	Pb	0.00	0.26	0.28	0.35	0.33	0.30	0.18	0.19	0.21	0.24	0.22	0.20	0.20	0.21	0.20	0.10	0.15	0.24	0.30	0.36	0.29	0.25	0.40	0.49	0.34	0.34	0.26
	Zn	0.00	0.38	0.36	0.32	0.28	0.36	0.24	0.20	0.16	0.21	0.33	0.40	0.34	0.39	0.41	0.22	0.26	0.34	0.43	0.45	0.43	0.41	0.59	0.51	0.50	0.50	0.36
Indicated (or better)	Mt	0.00	1.57	2.00	2.00	1.99	1.99	1.96	1.99	2.00	2.00	1.99	2.00	1.99	1.95	1.98	1.92	1.98	2.00	1.99	2.00	2.00	2.00	1.99	2.00	2.00	0.56	47.86
Inferred	Mt	0.00	0.00	0.00	0.00	0.01	0.01	0.04	0.01	0.00	0.00	0.01	0.00	0.01	0.05	0.02	0.08	0.02	0.01	0.01	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.30
	%	0%	0%	0%	0%	0%	1%	2%	1%	0%	0%	0%	0%	0%	3%	1%	4%	1%	0%	0%	0%	0%	0%	0%	0%	0%	0%	1%
Mining																												
HG	Mt	0.08	1.08	1.52	1.84	1.80	1.99	0.92	1.09	1.51	1.45	0.92	1.35	1.33	1.23	1.22	0.92	0.94	0.71	0.73	0.93	0.86	1.03	1.91	0.99	0.00	0.00	28.34
LG	Mt	0.07	0.45	0.79	0.56	0.68	0.72	0.59	0.55	0.72	0.77	1.16	1.33	1.20	1.12	1.10	0.85	0.77	0.90	1.09	0.78	0.74	0.88	1.34	0.68	0.00	0.00	19.82
Type 1	Mt	2.19	1.94	1.24	1.36	1.47	1.59	1.78	0.74	0.42	0.03	0.00	0.43	0.48	0.64	1.48	2.47	2.29	0.93	1.42	1.83	0.33	0.04	0.01	0.00	0.00	0.00	25.13
Type 2 & 3	Mt	0.09	1.93	1.93	1.72	1.52	1.19	2.18	3.10	2.83	3.24	3.39	1.93	2.29	2.39	2.13	2.15	2.39	3.64	3.13	1.94	3.55	3.54	2.22	0.28	0.00	0.00	54.66
Total Mined	**Mt**	**2.43**	**5.40**	**5.47**	**5.47**	**5.47**	**5.49**	**5.47**	**5.47**	**5.47**	**5.49**	**5.47**	**5.03**	**5.29**	**5.38**	**5.93**	**6.39**	**6.39**	**6.18**	**6.37**	**5.47**	**5.47**	**5.49**	**5.47**	**1.95**	**0.00**	**0.00**	**127.95**
Topsoil	Mt	0.03	0.05	0.01	0.02	0.05	0.05	0.02	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.03	0.05	0.04	0.01	0.01	0.06	0.05	0.00	0.00	0.00	0.00	0.00	0.53
Ore Rehandle	Mt	0.00	0.21	0.34	0.11	0.02	0.37	0.93	0.43	0.08	0.15	0.50	0.18	0.18	0.13	0.19	0.30	0.53	0.88	0.67	0.56	0.55	0.58	0.19	1.23	2.00	0.56	11.89
Type 1 Rehandle	Mt	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.76	0.76	0.76	0.45	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.46	1.93	2.05	1.13	0.63	0.04	8.98
Total Movement	**Mt**	**2.46**	**5.66**	**5.83**	**5.60**	**5.55**	**5.92**	**6.43**	**5.91**	**6.33**	**6.41**	**6.74**	**5.68**	**5.49**	**5.51**	**6.15**	**6.74**	**6.96**	**7.07**	**7.05**	**6.10**	**6.54**	**8.00**	**7.71**	**4.31**	**2.63**	**0.60**	**149.35**

16.2.3 Mine Development Sequence

Figure 16.2.5 to Figure 16.2.12 show the key development phases of the production schedule:

- Year 0: The end of the pre-strip operations showing the Northeast Starter Pit and the basal layer of the main waste dump.
- Year 3: The commencement of the Northeast Cutback and Southeast Starter Pit, with the development of the main waste dump.
- Year 8: The commencement of the Southeast Cutback along with the construction of the first Type 1 Stockpile and the near completion of the main waste dump.
- Year 10: The commencement of the Northwest Starter Pit and backfilling of the Northeast Pit.
- Year 14: At the end of the DFS mine plan.
- Year 15: The mining of the Southern Pits and commencement of the Northwest Cutback.
- Year 18: The installation of the permanent creek diversion, the backfilling of the Southern Pits and construction of in-pit Type 1 stockpile.
- Year 23: The completion of mining.

Figure 16.2.5 Mine Development Year 0

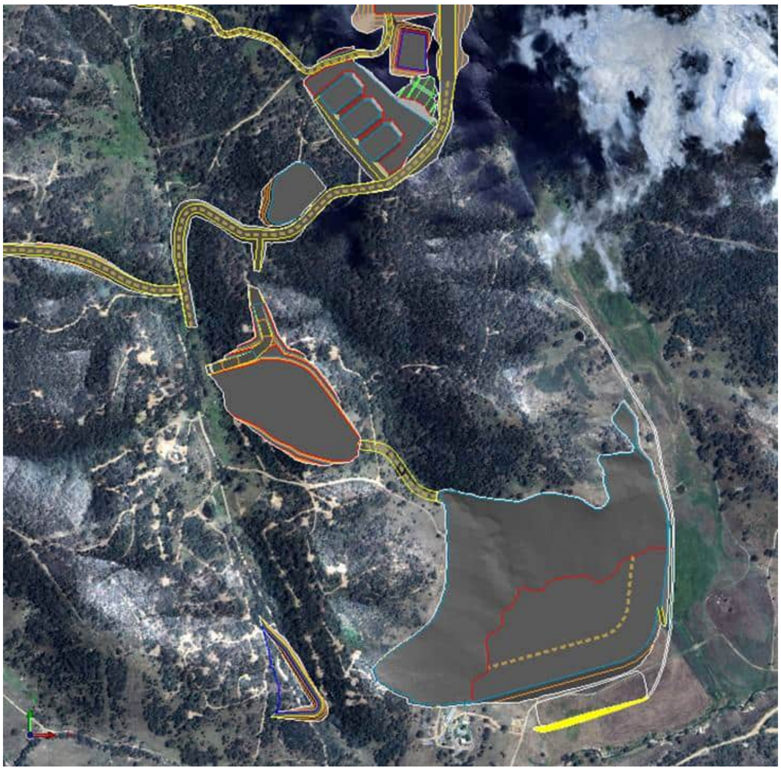


Figure 16.2.6 Mine Development Year 3

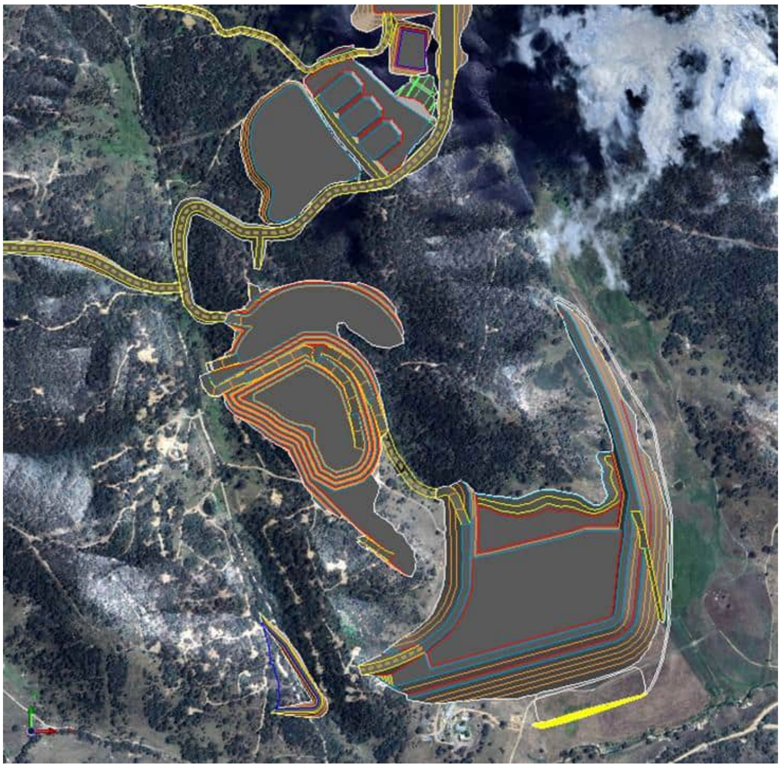


Figure 16.2.7 Mine Development Year 8

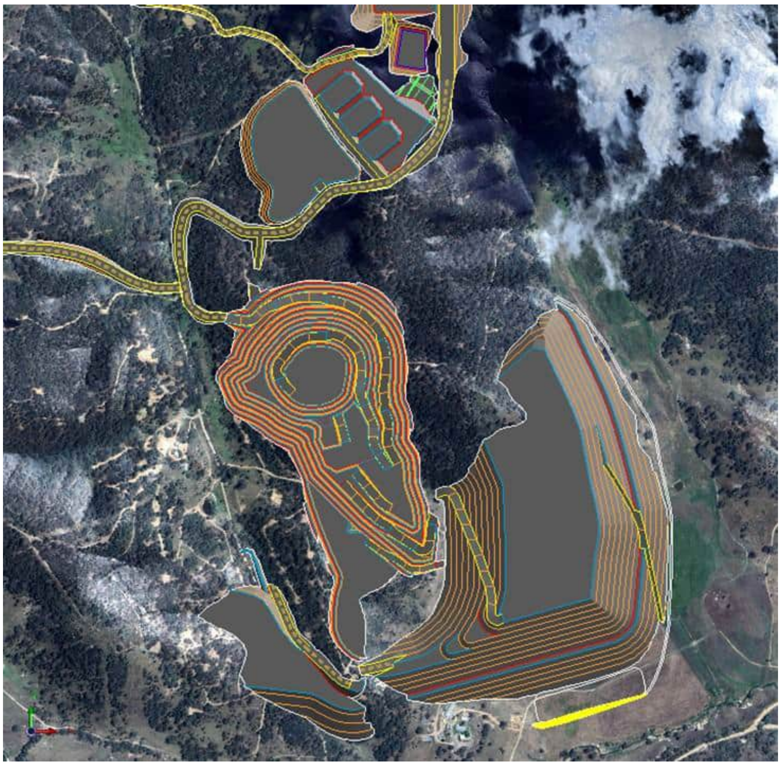


Figure 16.2.8 Mine Development Year 10

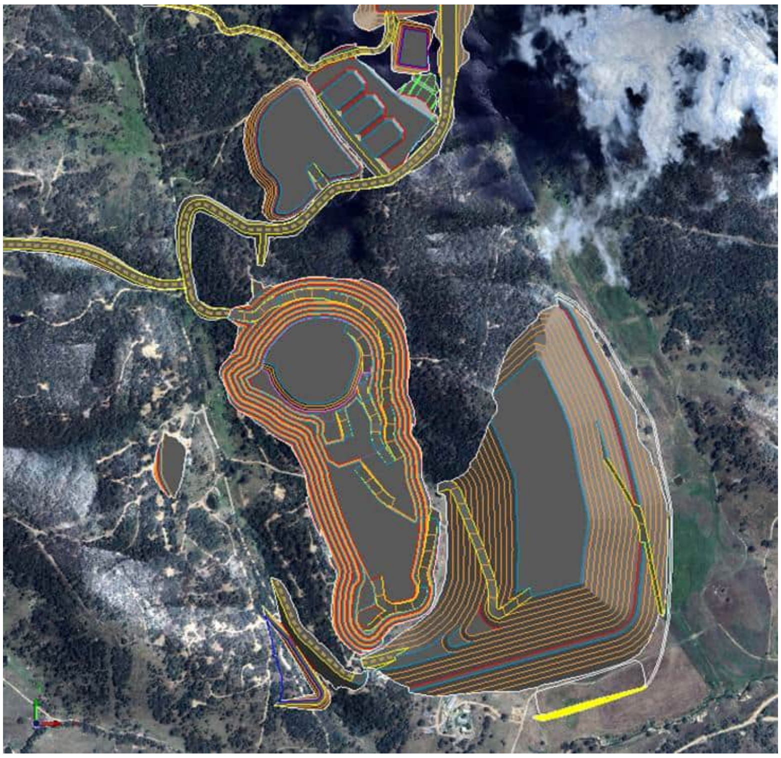


Figure 16.2.9 Mine Development Year 14

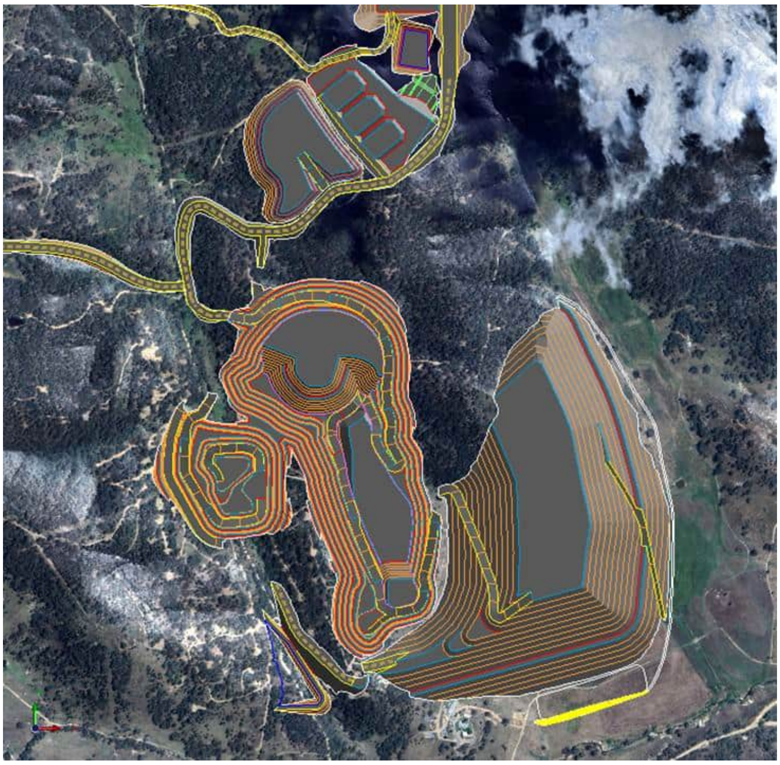


Figure 16.2.10 Mine Development Year 15

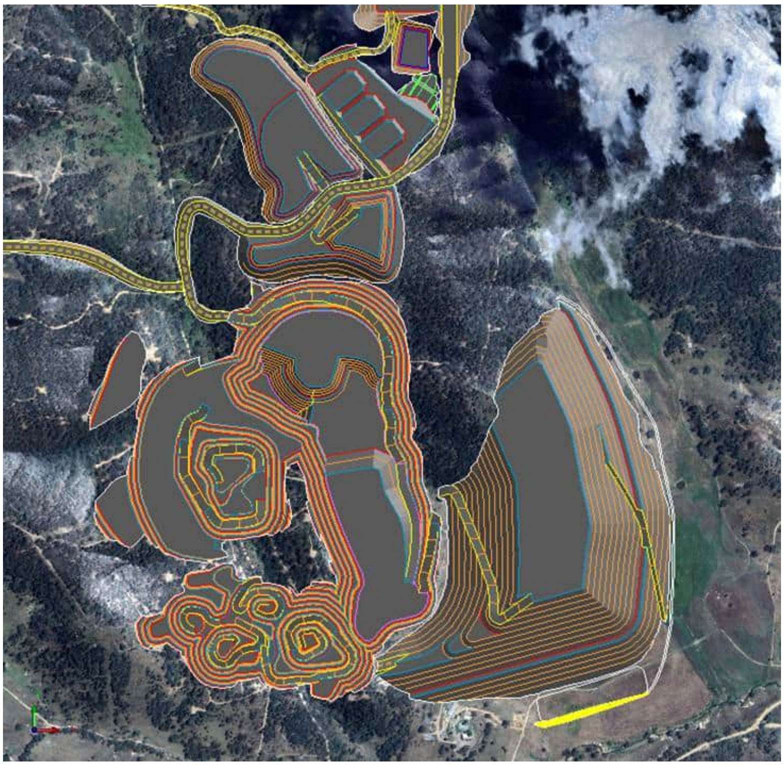


Figure 16.2.11 Mine Development Year 18

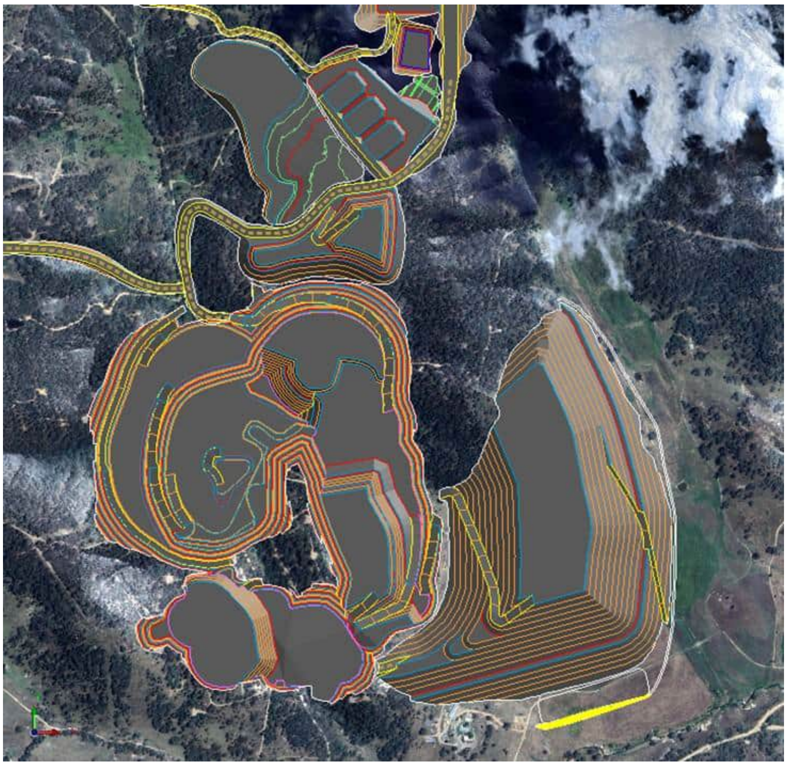
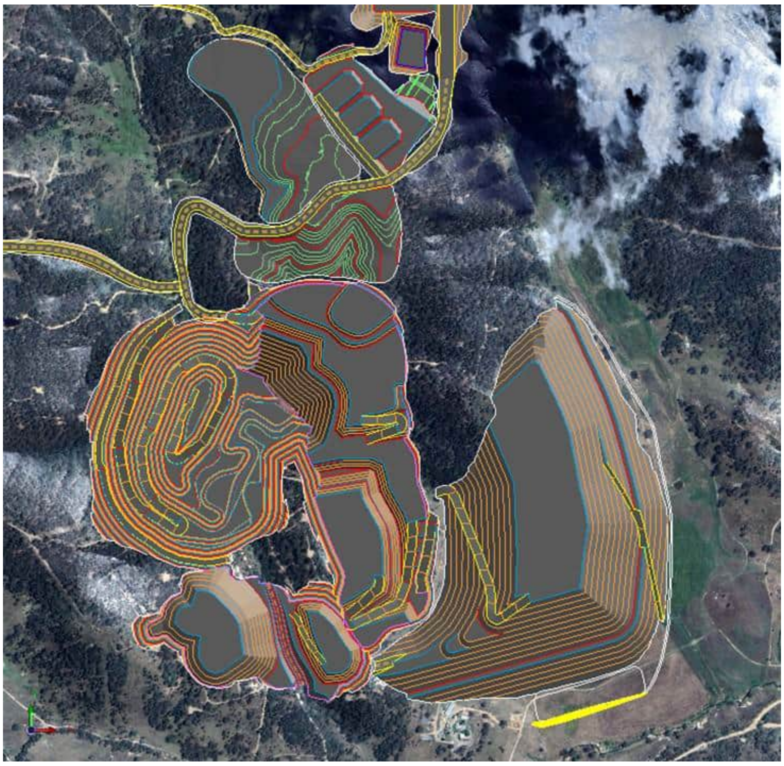


Figure 16.2.12 Mine Development Year 23



17.0 ORE PROCESSING – MINERAL RECOVERY

17.1 Overview

The DFS process plant design has been modified from the previous 2024 optimisation based on up-to-date information and testwork, design review outcomes, and capital cost optimisation. The basic flowsheet is summarised as two-stage crushing, primary and secondary milling, bulk flotation incorporating rougher, concentrate regrind and two cleaning stages. Tailings are dewatered by pressure filtration, and subsequently deposited and compacted into a stable, engineered landform.

17.2 Process Design

17.2.1 Design Philosophy

The DFS process plant design is based on a robust metallurgical flowsheet designed for optimum recovery at minimum operating costs, while minimising fines generation. The flowsheet consists of unit operations that are well proven in the industry and supported by results from testwork conducted on samples collected from the Bowdens resource.

The key criteria for equipment selection were suitability for the duty over the projected mine life of the operation, reliability and efficiency, operability and operating cost, ease and cost of maintenance and future expandability. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a compact footprint to minimise construction costs. Sufficient physical space for future additional items of plant has also been allowed for to facilitate potential expansion opportunities.

The plant will process a range of ore lithology types (breccia, crystal tuff, and welded tuff) with variable ore characteristics, minerals and metal grades.

The key project and ore specific criteria relating to the silver recovery circuit are as follows:

- 2 Mtpa (5,479 tpd) of ore.
- Plant operating schedule of 365 d/y.
- Plant operating schedule basis:
  - 5,256 h/y for the crushing plant
  - 8,000 h/y for the process plant
  - 6,666 h/y for the concentrate filtration plant
  - 6,666 h/y for the tailings filtration plant.
- Plant ROM feed maximum lump size of 900 mm.
- Crushing circuit product P_{80} of 35 mm.
- Grinding circuit product P_{80} of 106 µm.
- Sufficient control and automation to minimise continuous operator input but allow manual override and control as required.

The DFS design has been prepared based on the process design criteria derived from the results of metallurgical testwork managed by Bowdens. Refer to Section 13 for a summary of the metallurgical testwork incorporated into the DFS design.

17.2.2 Process Flowsheet

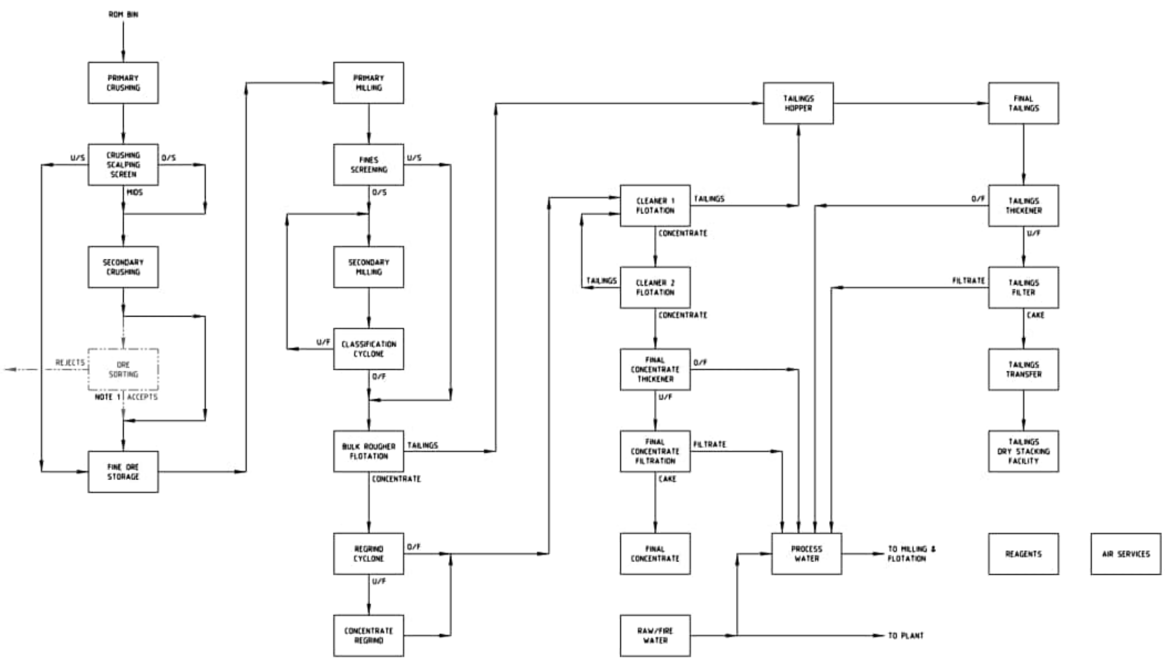
The plant is a conventional flotation concentrator for bulk silver-lead-zinc concentrate production.

The process flowsheet developed for the DFS is summarised by unit processes as follows:

- Two-stage crushing comprising primary jaw crushing followed by secondary cone crushing to produce a crushed product with a particle size of P_{80} 35 mm.
- Primary and ball mill grinding circuit operating in SAB mode (SAG mill and ball mill in series), with classification via fine screening and hydrocyclones.
- Bulk rougher flotation of sulphide ore comprising a train of rougher flotation cells with reagent addition at the conditioning stage and selected cells.
- Rougher concentrate regrind using a stirred mill operating in open circuit.
- Two-stage cleaner flotation comprising Cleaner 1 tank cells followed by Cleaner 2 flotation utilising a Jameson Cell to produce a final concentrate.
- Bulk concentrate thickening, pressure filtration, undercover storage and loadout for transport.
- Tailings thickening followed by filtration and disposal to a compacted, stable filtered tailings landform, with recovery of filtrate to the process water system.
- Reagent preparation and distribution systems supporting flotation and dewatering operations.
- Process plant and tailings treatment utilities including water services and air services.

The overall block flow diagram for the plant is shown in Figure 17.2.1.

Figure 17.2.1 Process Plant Block Flow Diagram



17.2.3 Key Process Design Criteria

The key process design criteria are listed in Table 17.1.1.

Table 17.2.1 Summary of Key Process Design Criteria

Parameter	Units	Design	Source
Plant Throughput	tpa	2,000,000	Client
Head Grade, Life of Mine – LOM Weighted Average			
• Silver	g/t	67.5	Client
• Lead	%	0.25	Client
• Zinc	%	0.34	Client
Overall Metal Recovery, Life of Mine – LOM Weighted Average			
• Silver	%	82.4	Client
• Lead	%	80.8	Client
• Zinc	%	90.5	Client
Mass Recovery, Life of Mine – LOM Weighted Average	%	1.24	Client
Crushing Plant Overall Utilisation	%	60.0	OMC
Process Plant Overall Utilisation	%	91.3	Client
Filtration Plant Overall Utilisation	%	76.1	Client
Crushing Work Index (CWi)	kWh/t	10.5	Testwork
Bond Rod Mill Work Index (RWi)	kWh/t	18.3	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	18.4	Testwork
SMC Test Parameter, A*b		40.0	Testwork
Bond Abrasion Index (Ai)		0.223	Testwork
Grinding Circuit Product Size, P_{80}	µm	106	Testwork
Regrind Circuit Product Size, P_{80}	µm	20.0	Testwork
Flotation Residence Time (Lab)			
• Rougher Flotation	min	9.0	Testwork
• Cleaner 1 Flotation	min	8.0	Testwork
• Cleaner 2 Flotation	min	8.0	Testwork
Plant Flotation Residence Time Scale-Up Factor		2.5	Client
Design Mass Recovery to Bulk Flotation Concentrate	%	1.88	Client
Bulk Concentrate Thickener U/F Density	% solids (w/w)	63.0	Testwork
Bulk Concentrate Specific Thickener Rate	tph/m^2	0.14	Testwork
Bulk Concentrate Specific Filtration Rate	$kgDS/m^2.h$	148	Vendor
Bulk Concentrate Moisture	% H_2O w/w	10.0	Client
Tailings Thickener U/F Density	% solids (w/w)	56.0	Testwork
Tailings Specific Thickener Rate	tph/m^2	1.5	Testwork
Tailings Specific Filtration Rate	$kgDS/m^2.h$	96.0	Vendor Testwork
Tailings Moisture	% H_2O w/w	19.7	Vendor

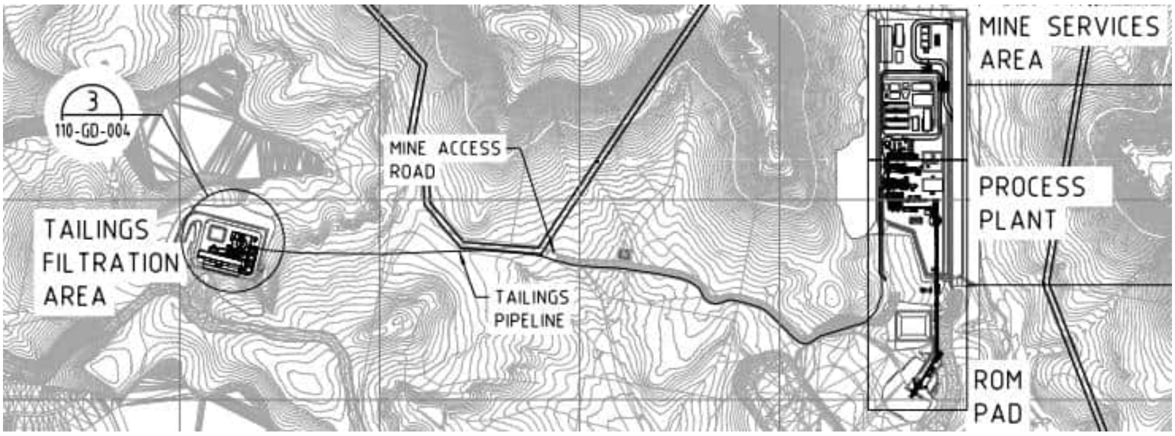
17.3 Process and Plant Description

The ROM pad and main process plant are located to the north of the pit area. This location visually shields the process plant from the nearby township of Lue.

The tailings filtration plant is located approximately 2 km to the west of the main process plant, close to the TSF in order to minimise hauling distance of the filtered tailings.

The relative locations of the main process plant and the tailings filtration plant are shown in Figure 17.3.1.

Figure 17.3.1 Location of Process Plant and Tailings Filtration Plant



17.3.1 Major Equipment

A summary of the major equipment selected for the process plant is provided in Table 17.3.1.

Table 17.3.1 Major Equipment List

Equipment Description	Details
Primary Crusher	C150 jaw crusher
Primary Screen	1.8 x 4.8 m double deck vibrating screen, 70 mm / 40 mm aperture
Secondary Crusher	HP 450e cone crusher
Fine Ore Bin	1,302 m^3 live capacity steel bin
Primary Mill	6.5 m dia. x 3.7 m EGL, grate discharge with 3,100 kW variable speed drive
Primary Mill Discharge Screen	2.4 x 4.8 m single deck vibrating screen, 6 mm aperture
Fines Screen	Derrick Superstack 8 deck screen, 210 μm aperture
Ball Mill	5.2 m dia. x 8.22 m EGL, grate discharge with 3,640 kW variable speed drive
Milling Cyclones	14 x 250 mm cyclone cluster – 8 duty, 6 standby
Bulk Rougher Flotation Cells	5 x 50 m^3 tank cells
Regrind Mill	SMD 355
Regrind Cyclones	8 x 150 mm cyclone cluster – 6 duty, 2 standby
Cleaner 1 Flotation Cells	4 x 20 m^3 tank cells
Cleaner 2 Flotation Cell	Jameson E1714/2
Concentrate Thickener	7 m diameter high rate
Concentrate Filter	Metso VPA 1010 plate and frame filter
Tailings Thickener	22 m diameter high rate
Tailings Filters	3 x Metso FFP 3512 plate and frame filters

17.3.2 Ore Receiving and Primary Crushing

ROM ore from the open pit, at maximum lump size of 900 mm, is transported to the plant by 100 t capacity rear dump trucks. The trucks can tip directly into the crusher ROM bin or onto the ROM stockpile, as required at the time. The purpose of the ROM stockpile is to absorb natural fluctuations in mining rate relative to processing rate, and ore blending according to grade and material properties. ROM ore is reclaimed from the stockpile to the ROM bin by a front-end loader.

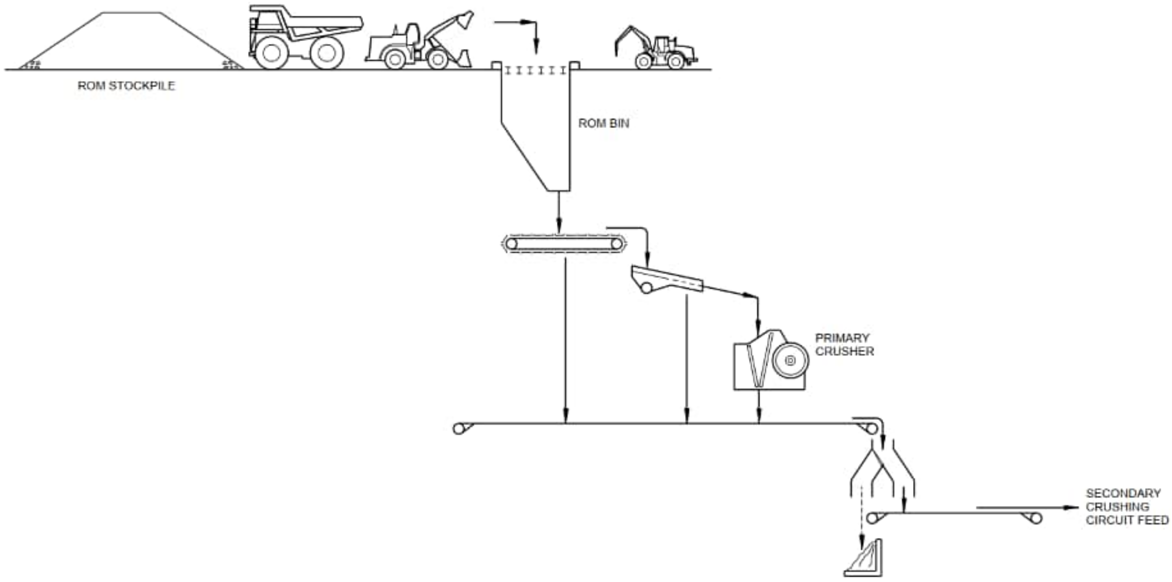
A static grizzly (900 mm aperture) mounted above the ROM bin limits the maximum rock size presented to the jaw crusher. A mobile rock breaker is available to assist in breaking down oversize material retained on the static grizzly, for any occasion that the loader is unable to clean it off. Ore is fed from the ROM bin into the 200 kW Nordberg C150 jaw crusher via a variable speed apron feeder and vibrating grizzly. Oversize from the grizzly reports to the crusher, which operates in open circuit. Crushed ore, together with undersize from the grizzly report directly onto the 1,200 mm wide primary crusher discharge conveyor, prior to being conveyed to the secondary crusher via the 1,200 mm wide screen feed conveyor. The primary crusher discharge conveyor is fitted with a weightometer to monitor the crusher throughput and control the apron feeder variable speed drive.

The ROM bin is covered by a structure with insulated cladding on the walls and roof to minimise noise emissions to the surroundings. Additional sound-attenuating and dust minimisation controls will line the ROM bin internal structure to further attenuate noise and minimise dust emissions.

The primary crusher is incorporated into the same structure, also with insulated cladding to minimise noise emissions to the surroundings.

The crusher is serviced by an overhead crane with a drop-down bay within the structure. The primary crusher flowsheet is shown in Figure 17.3.2.

Figure 17.3.2 Primary Crusher Flowsheet



The primary crusher building is serviced by a dust collection system, with collected dust discharged onto the primary crusher discharge conveyor and finely sprayed with water for dust suppression.

17.3.3 Screening and Secondary Crushing

Primary crushed ore is conveyed from the primary crusher to the screen feed conveyor and presented to a vibrating scalping screen (primary screen) for size classification. The screen undersize reports directly to the 900 mm wide crushed product conveyor, thus bypassing the secondary crusher and be transferred to the fine ore bin feed conveyor for storage ahead of the grinding circuit.

Screen oversize is fed to the 315 kW Metso Nordberg HP450e secondary crusher via a surge bin and vibrating feeder. The secondary crusher operates in open circuit, with the crushed product discharging onto the product conveyor, where it recombines with the scalping screen undersize material.

The combined crushed product is conveyed to the fine ore bin, which provides approximately 8 h of live capacity to ensure stable feed to the milling circuit and to decouple the crushing and grinding operations.

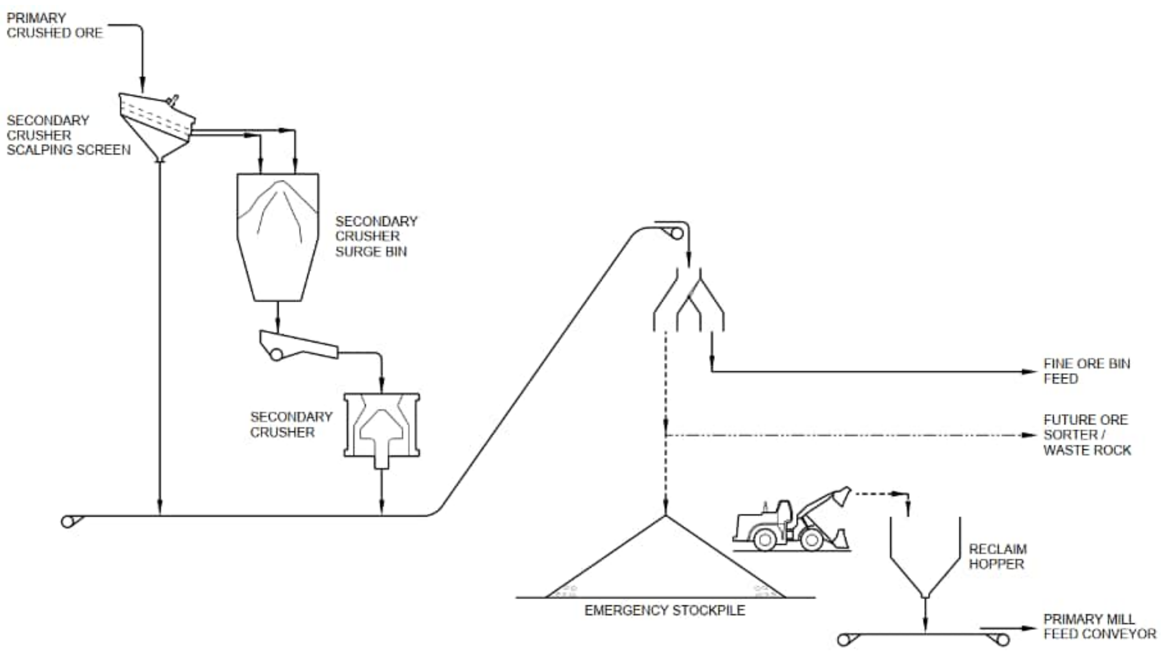
Instrumentation includes belt weightometers for throughput monitoring and control, and a metal detector is installed on the screen feed conveyor to protect downstream equipment.

Operational flexibility is incorporated through provision of diverter gates, temporary stockpile and emergency reclaim arrangements, allowing for versatile stockpile management and future integration of additional processing stages such as ore sorting if required.

The scalping screen and secondary crusher are housed inside a sheeted building with a dust collection system and insulated cladding to manage dust and noise emissions. Collected dust is returned to the conveyor system.

The scalping screen and secondary crusher are housed inside a sheeted building with a dust collection system and insulated cladding to further reduce the risk of dust emissions and provide noise attenuation. Collected dust is returned to the conveyor system. The screening and secondary crushing area is shown in Figure 17.3.3.

Figure 17.3.3 Screening and Secondary Crushing Flowsheet



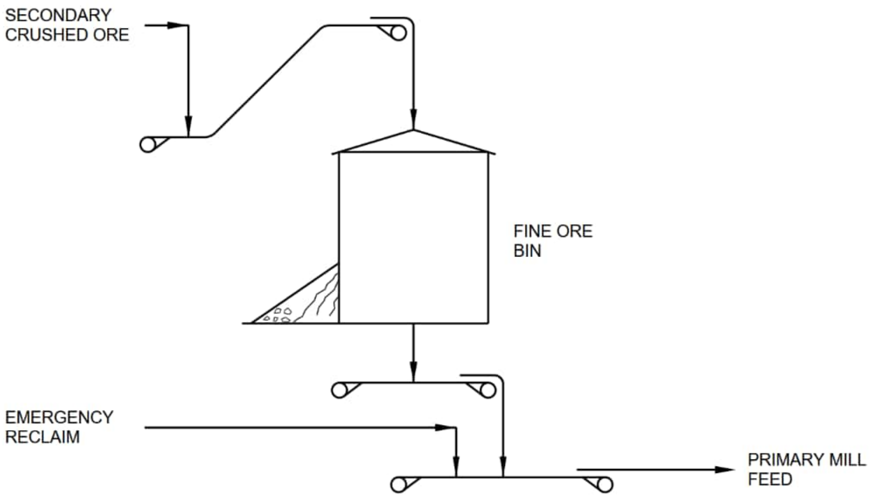
17.3.4 Fine Ore Bin

The fine ore bin provides surge capacity between the crushing and milling circuits. A fine ore bin has been chosen in preference to an open (or partially enclosed) stockpile to minimise dust emissions.

The fine ore bin has a live capacity of approximately 2,000 t (equivalent to 8 h of mill feed at 2 Mtpa). Crushed ore is reclaimed from the fine ore bin via a reclaim feeder located beneath the bin and onto the 900 mm wide bin discharge conveyor, which in turn discharges onto the 900 mm wide mill feed conveyor. The mill feed conveyor transfers the reclaimed ore to the primary mill. A belt weightometer is installed on the mill feed conveyor to monitor and control the feed rate to the grinding circuit.

A dust collection system at the fine ore bin and reclaim conveyor controls dust emissions, with collected dust deposited onto the conveyor belt. Provision is made for an emergency reclaim arrangement associated with the fine ore bin reclaim system. The fine ore bin flowsheet is shown in Figure 17.3.4.

Figure 17.3.4 Fine Ore Bin Flowsheet



17.3.5 Milling and Classification

The grinding circuit comprises a primary mill and ball mill operating in series, with screening and classification. The primary mill is a low aspect ratio SAG mill, designed to reduce overgrinding and improve operability and power efficiency when compared to a high aspect ratio SAG mill.

From the fine ore bin, ore is conveyed to the primary mill feed chute. The primary mill receives the crushed ore, with process water addition to facilitate slurry formation. A belt weightometer on the mill feed conveyor monitors and controls the feed rate to the grinding circuit.

Bowdens ore contains a combination of minerals, ranging from very hard to very soft, and metal recovery is grind-sensitive. The circuit is designed to treat varying ratios of hard and soft minerals with maximum consistency and efficiency, by having:

- Secondary crushing prior to SAG milling rather than pebble crushing post-SAG milling.
- Adequate size and power for hard ore.
- Turn-down capacity (variable speed drives (VSDs)) on both mills for softer ore.
- Fines screening prior to the ball mill circuit to prevent soft, fine material entering the ball mill circuit.

The primary and secondary milling circuits are designed to operate in series, with an intermediate screening stage to maximise early removal of soft gangue minerals and liberated sulphides prior to the ball milling circuit. Early removal of ground material (via screening rather than hydrocyclones) has the purpose of:

- Improving metallurgical performance by reducing viscosity in the ball mill circuit, and especially in the hydrocyclone classifiers.
- Reducing overgrinding of soft gangue minerals, in turn improving tailings filtration rates.

Primary mill discharge is classified by a vibrating screen, with oversize conveyed to the primary mill feed conveyor for further grinding in the primary mill. Lime slurry is added to the primary mill discharge hopper for the initial stage of pH modification. The primary mill discharge screen undersize reports to the next screening stage comprising of scalping and fines screens. Oversize material is directed to the ball mill for further grinding. The fines screen undersize is at the final product size and reports directly to the rougher flotation circuit.

The ball mill operates in a typical closed-circuit configuration with the cyclones providing classification. Ball mill discharge slurry is pumped from the milling cyclone feed hopper to the cyclone cluster. Cyclone underflow returns to the ball mill feed for further size reduction. Cyclone overflow reports to the rougher flotation circuit as final grinding product.

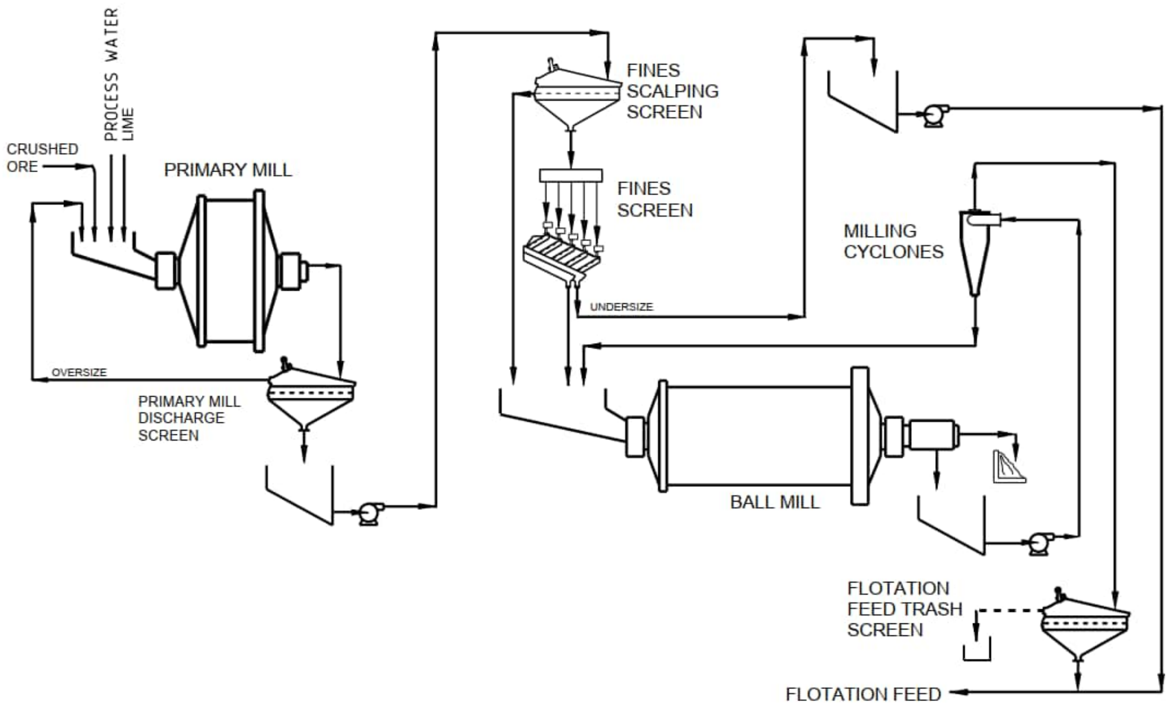
The final product of the grinding circuit is the combined fines screen undersize and cyclone overflow from the ball mill circuit. The design grinding product size is P_{80} 106 µm.

The circuit design includes operational flexibility to bypass the primary mill scalping and fines screens if the primary mill fines classification is not required, depending upon the feed blend. For the bypass configuration, the primary mill discharge slurry is pumped to the milling cyclone feed hopper and the milling circuit will be operated in a traditional SAB mode.

The milling circuit includes process control and monitoring systems, including particle size analysis and pH measurement on the flotation feed streams, and sampling of cyclone overflow and fines screen undersize. Particle size analysis is a key means of controlling the milling circuit operational parameters, particularly the mill variable speed drives.

The milling and classification circuit flowsheet is depicted in Figure 17.3.5.

Figure 17.3.5 Milling and Classification Circuit



Ancillary systems include process water addition at various points in the circuit, and sump pumps for collection and return of spillage within the milling area. Associated mill services include compressed air and lubrication. Overhead gantry cranes are used for lifting and moving heavy equipment.

The milling and classification building is sheeted with insulated cladding to minimise noise emissions to the surroundings.

17.3.6 Bulk Rougher Flotation

The final product from the grinding circuit flows to the rougher flotation conditioning tank via the trash removal screen and metallurgical sampler. Reagents including collector, activator and lime slurry are added to the conditioning tank. The conditioned slurry is pumped to the rougher feed box where frother is added ahead of the five 50 m^3 rougher flotation tanks cells arranged in series.

The rougher flotation cells are elevated on a steel support structure, with gravity flow through the stepped-down cells. Slurry level is controlled by dart valves in each cell while blower air supplied to individual cells is controlled to an adjustable setpoint. Concentrate from each cell reports via a common launder to the rougher concentrate hopper.

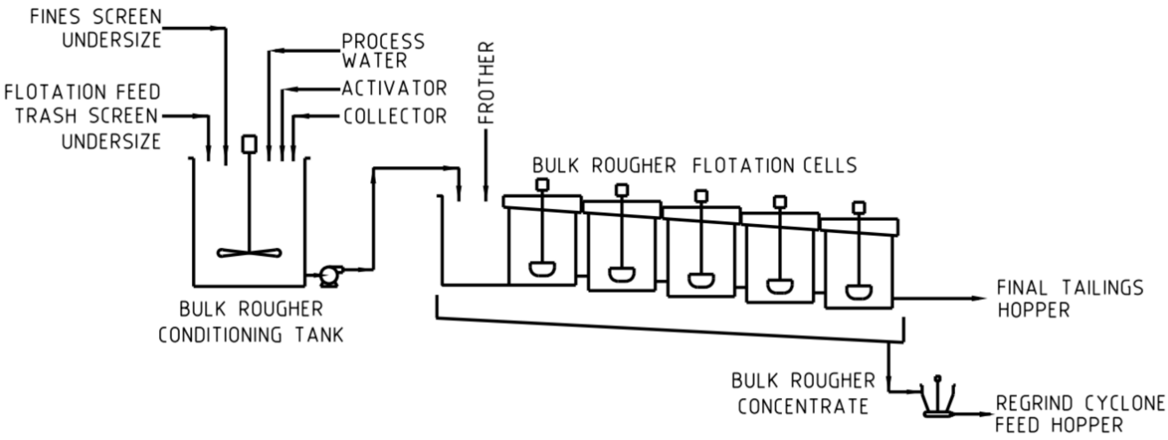
The area under the flotation cells is kept open to provide access for clean-up. A concrete bund is included to capture any spillage. Flotation cells, pumps and associated equipment is maintained with the assistance of mobile lifting equipment.

The rougher concentrate is pumped to the regrind cyclone feed hopper.

Tailings from the final discharges to the final tailings hopper via the rougher flotation tailings metallurgical accounting sampler.

The bulk rougher flotation flowsheet is shown in Figure 17.3.6.

Figure 17.3.6 Bulk Rougher Flotation Flowsheet



17.3.7 Rougher Concentrate Regrind

Rougher concentrate reporting to regrind cyclone feed hopper is combined with dilution water as required, prior to classification by the nest of 150 mm diameter hydrocyclones. Cyclone overflow discharges via launder to the cleaner flotation conditioning tank. Cyclone underflow reports to the regrind mill feed hopper for pumping to the regrind mill for size reduction.

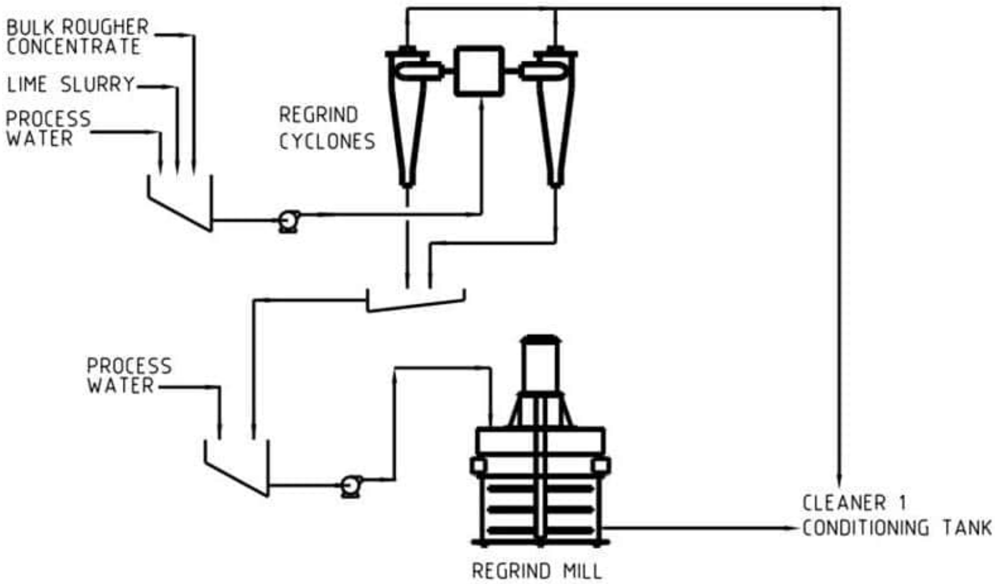
The variable speed 355 kW Stirred Media Detritor (SMD) concentrate regrind mill operates in open circuit. Ceramic grinding media is added via a ball charging system consisting of a kibble and hoist arrangement. Maintenance of the cyclones and regrind mill utilises a fixed jib crane.

The design product size after regrinding is P_{80} 20 µm, which is required for improved liberation of sulphide minerals.

Process water and gland water services are provided to the regrind circuit, with sump pumps to manage spillage and return slurry to the process.

The concentrate regrind flowsheet is shown in Figure 17.3.7.

Figure 17.3.7 Concentrate Regrind Flowsheet



17.3.8 Bulk Cleaner Flotation

The cleaner flotation circuit consists of a Cleaner 1 stage comprising four 20 m^3 tank cells arranged in series, followed by a Cleaner 2 stage utilising a Jameson Cell. Cleaning stages are required to remove entrained and liberated gangue minerals, as well as reduce recovery of gangue sulphides such as those containing iron and arsenic.

The combined regrind cyclone overflow and regrind mill discharge reports to the Cleaner 1 conditioning tank, where reagents including collector, frother, depressant and lime slurry are added as required. The conditioned slurry is then pumped to the Cleaner 1 flotation feed box, from where it flows to the first of the four tank cells arranged in series.

Similar to the rougher circuit, slurry flows by gravity through the cleaner cells, with level control and blower air provided to each cell. Cleaner 1 concentrate is recovered via a common launder and pumped forward to the Cleaner 2 stage. Tailings from the Cleaner 1 circuit reports to a hopper and is pumped to the final tailings hopper via a sampler.

Cleaner 2 (re-cleaner) is a single E1714/2 Jameson Cell and upgrades the Cleaner 1 concentrate stream. The feed is conditioned as required prior to entering the Jameson Cell, where self-aspirated flotation produces the final concentrate.

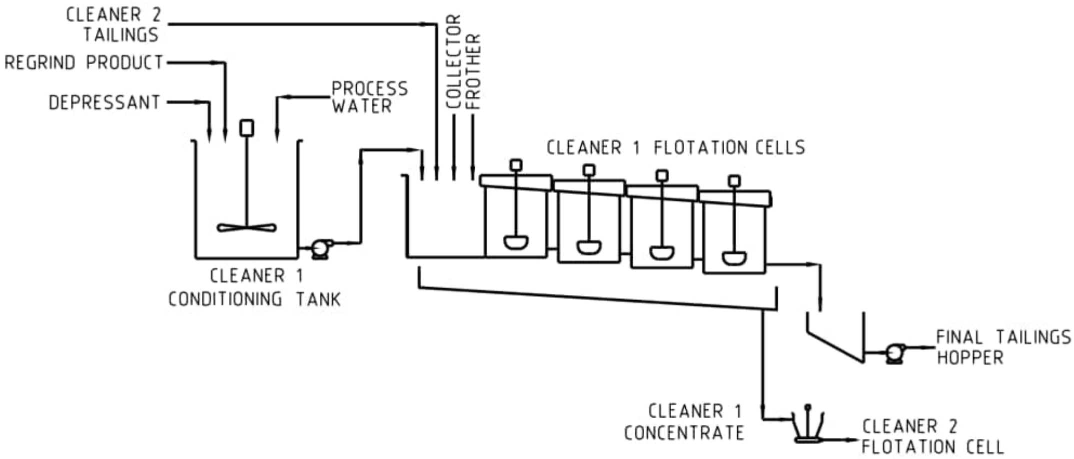
Tailings from the Jameson Cell are pumped back to the Cleaner 1 conditioning tank.

Final concentrate from the Jameson Cell is pumped to the concentrate thickener via a trash screen and metallurgical sampler.

Ancillary systems include process water services, gland water, blower air, and sump pumps for collection and return of spillage within both Cleaner 1 and Cleaner 2 flotation areas.

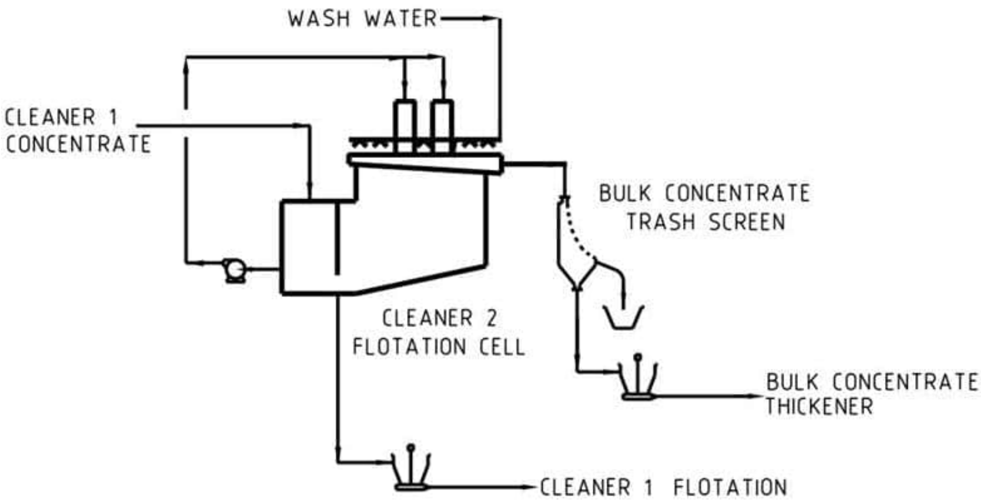
The Cleaner 1 flotation flowsheet is shown in Figure 17.3.8.

Figure 17.3.8 Cleaner 1 Flotation Flowsheet



The Cleaner 2 flotation flowsheet is shown in Figure 17.3.9.

Figure 17.3.9 Cleaner 2 Flotation Flowsheet



17.3.9 On-Stream Analysis

The flotation plant is equipped with a Courier on-stream analysis system to support metallurgical accounting and process control. The system comprises sampling points and analysers installed on the following key streams within the flotation circuit:

- Flotation feed
- Rougher concentrate
- Rougher tailings
- Cleaner 1 concentrate
- Cleaner 1 tailings
- Cleaner 2 tailings
- Final concentrate.

These samples are collected via dedicated samplers and directed to the on-stream analyser system.

The on-stream analyser measures (assays) slurry composition from the selected process streams to provide real-time data for monitoring and control of flotation performance. Sample streams are processed through the analyser, with sampled material returned to the process via sample return pumps. Samples will also be collected for daily reporting composites.

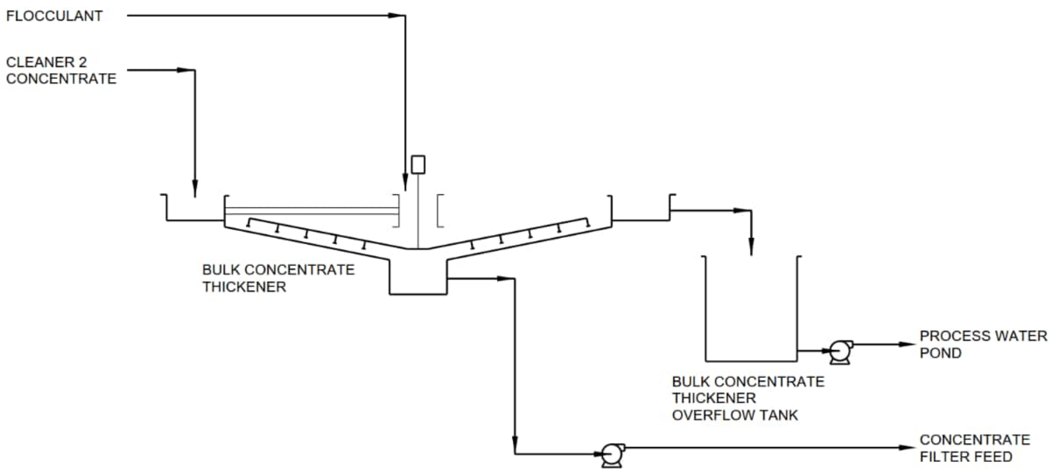
Particle size analysis is conducted on key streams including flotation feed, fines screen undersize and regrind product to support circuit optimisation and control. The on-stream analyser and particle size analysis systems are integrated with the plant control system to enable continuous monitoring of metallurgical performance and assist in process optimisation.

17.3.10 Bulk Concentrate Thickening

A 7-m diameter high-rate thickener is used for the first stage of flotation concentrate de-watering. Cleaner 2 (final) concentrate reports to the concentrate thickener feed box via the metallurgical sampler and trash screen. Flocculant solution is added to the concentrate slurry via the flocculant mixing and dosing system to promote solids settling within the thickener and to maintain overflow clarity.

The thickener includes an auto dilution feed well, froth sprays to break any froth build-up on the surface and bridge supported drive mechanism. The concentrate thickener overflow stream is pumped to the process water pond for reuse in the process. Thickener underflow is pumped to the filter feed tank in the concentrate filtration area. Any spillage from the concentrate thickener area is pumped to the concentrate thickener feed box by the concentrate thickener area sump pump. The bulk concentrate thickening flowsheet is shown in Figure 17.3.10.

Figure 17.3.10 Concentrate Thickening Flowsheet



17.3.11 Bulk Concentrate Filtration

The concentrate filter feed tank provides 12 h of surge capacity at the average concentrate production rate, allowing filter maintenance to be conducted without impacting mill throughput.

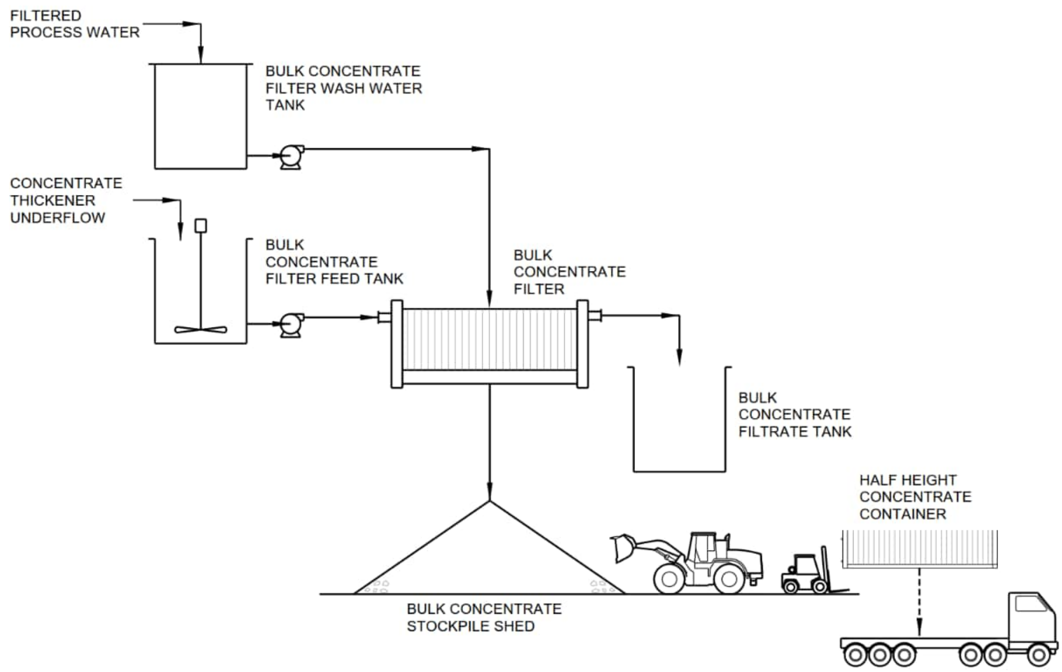
Thickened concentrate slurry at 62.7% solids is pumped from the filter feed tank to the Metso VPA 1010 vertical plate pressure filter, producing a filter cake at a design moisture content of 10% which discharges directly into a bunker located below the filter. The filter is housed in an enclosed building and supported by a structure that allows drainage and collection of filtrate and wash water.

Filtered water is used for flushing and cloth washing duties. Filtrate and cloth wash water is collected below the filter and flows to the filtrate tank to be pumped back to the process; to the filter press as wash water and to the concentrate thickener feed.

Compressed air is used to assist filter operation during the pressing and drying cycles and is supplied by a dedicated filter air system comprising compressors and receivers. An overhead gantry crane facilitates maintenance of the filter and associated equipment.

The bulk concentrate filtration and loadout flowsheet is shown in Figure 17.3.11.

Figure 17.3.11 Concentrate Filtration & Loadout Flowsheet



17.3.12 Bulk Concentrate Storage and Loadout

The filter cake discharge bunker is accessed for clearing by a front-end loader (FEL) from inside the concentrate storage shed, where the material is transferred to the concentrate stockpile(s) for storage.

The stockpiled concentrate is reclaimed using a FEL and loaded into containers for transport. Provision for container handling and weighing is included within the storage area, with container movements handled by a forklift.

The concentrate storage area is inside an enclosed shed on a concrete slab, with a dedicated dust collection system to further guard against safety and environmental impacts.

Provision has been made for a future concentrate bagging facility, allowing concentrate to be packaged in bags if required. The bagging facility is housed in the concentrate storage and load-out area, with allowance in the layout for installation as part of a future scope.

17.3.13 Tailings Thickening

Tailings from the flotation circuit is pumped from the final tailings hopper to the thickener feed tank at the tailings treatment facility. The tails pipeline is approximately 2 km and utilises dual-stage centrifugal pumps at the hopper.

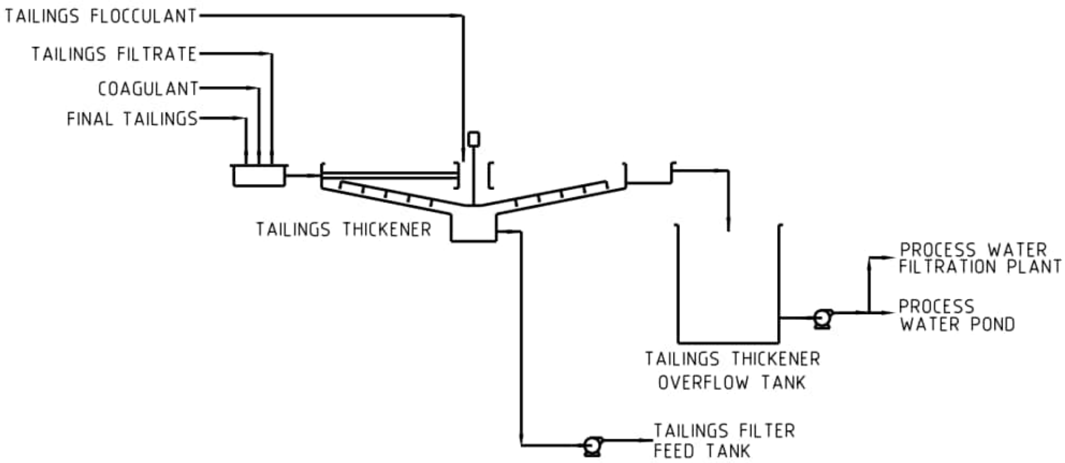
The 24 m diameter high-rate thickener increase slurry density from approximately 28% solids to 55% solids in preparation for the subsequent filtration stage. Recovered thickener overflow water is collected in an overflow tank and pumped to the process water pond for reuse within the plant. Thickener underflow is pumped to the tailings filter feed tank.

Flocculant and coagulant are dosed into the tailings slurry to promote solids settling and supernatant (overflow) clarity within the thickener. The tailings thickener has a dedicated flocculent mixing plant.

Supporting systems include process and gland water services, as well as sump pumps for the collection and return of spillage within the thickener area. A containment pond collects any excess water or spillage in the area.

The tailings thickening flowsheet is shown in Figure 17.3.12.

Figure 17.3.12 Tailings Thickening Flowsheet



17.3.14 Tailings Filtration and Disposal

Thickened tailings are maintained under agitation prior to filtration in the two tailings filter feed tanks, with a combined nominal 8-h storage capacity at full tonnage.

The filter feed pumps deliver the tailings from the feed tanks directly into the three Larox FFP 3512 plate and frame pressure filters operating in parallel. Each filter press has 98 plates (3.5 x 2.5 m each) and a nominal active surface area of 1,314 square metres (m^2). After each cycle, the filter presses discharge the filtered tailings into the bunkers below reach of them at a nominal moisture content of 19.7%. The filtration system is fully automated for maximum control and minimal operator intervention.

The recovered water (filtrate) is collected in the tailings filtrate tank and returned to the tailings thickener.

Tailings filter cake is loaded by front end loader (up to Cat 988) from the discharge bunkers into articulated mine trucks and delivered to the filtered tailings storage area, where the material is stacked and compacted in the filtered tailings landform.

The filtration system is supported by ancillary systems including filter feed pumps, cloth wash water systems, and compressed air systems for pressing and drying cycles. Compressed air is supplied by dedicated compressors, receivers and air treatment systems.

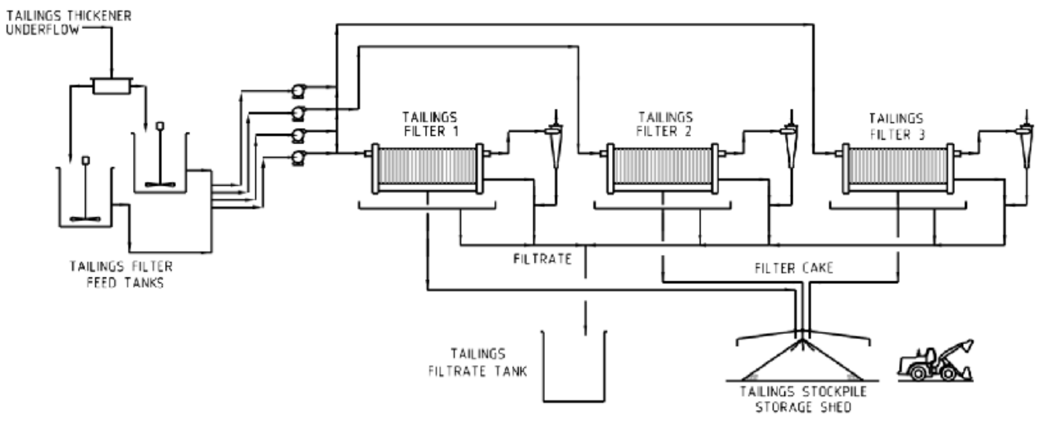
The tailings filters are housed indoors for shelter and noise abatement. Filter maintenance inside the building is facilitated by a gantry crane. Mobile cranes are utilised for heavy lifting in the area outside the filter building.

Integration with the overall tailings handling system includes management of process water return and provision for event ponds associated with the process plant and tailings system.

Sump pumps within the tailings filtration area collect and return spillage to the tailings thickener. Maintenance access is facilitated by provision of lifting equipment including cranes.

The tailings filtration flowsheet is shown in Figure 17.3.13.

Figure 17.3.13 Tailings Filtration Flowsheet



17.3.15 Process Plant Reagents

The flotation reagent regime has been developed based on the metallurgical testwork results. The regime comprises a combination of reagents that are prepared on site through mixing systems and other reagents that are dosed directly from intermediate bulk containers (IBCs) at full strength.

The flotation reagent mixing and storage area is located in close proximity to the flotation circuit to minimise distribution distances to the reagent addition points, while the concentrate thickener flocculant mixing and storage facility is located in the concentrate thickener and storage area.

Reagents are stored, under cover, in a nearby area with access for delivery trucks. Forklifts are used to transferred reagents from the storage to their mixing and usage areas. The exception is hydrated lime, which is stored in a silo.

Flotation Collector (Aero 4037 or Equivalent)

Flotation collector (Aero 4037) is delivered to site as a 100% concentration liquid in 1-tonne (t) capacity intermediate bulk containers (IBCs). The collector is stored in the IBCs and transferred to a storage tank, from where dedicated metering pumps deliver the reagent to the required addition points within the flotation circuit. Reagent transfer is by gravity feed from a rack above the storage tank.

Dosing points include the rougher conditioning tank, rougher flotation Cell 4, and the Cleaner 1 conditioning tank.

The flotation collector area has a dedicated floor sump and pumping arrangement for containment and recovery of spillage.

Flotation Depressant - Sodium Silicate (Na_2SiO_3)

Sodium silicate is also delivered to site in IBCs and transferred to the storage tank as required. The reagent is transferred from the IBCs to the storage tank and dosed to the process using dedicated metering pumps.

Sodium silicate is distributed to the cleaner flotation circuit, including addition to the Cleaner 1 conditioning tank as required.

Flotation Activator – Copper Sulphate ($CuSO_4$)

Flotation activator (copper sulphate) is delivered to site in bulk bags and transferred to a dedicated mixing system via a bag breaker. The reagent is mixed with water in an agitated mixing tank to form a solution, which is transferred to a storage tank and dosed to the process using dedicated metering pumps.

The copper sulphate solution is distributed to the flotation circuit, including addition to the bulk rougher conditioning tank and other required addition points.

Flotation Frother (MIBC)

MIBC is delivered to site as a 100% liquid in IBCs. The frother is stored in the IBCs and transferred to a storage tank, from where it is delivered by metering pumps to the required addition points within the flotation circuit. Addition points include the rougher and cleaner feed boxes.

Flotation pH Modifier - Hydrated Lime ($Ca(OH)_2$)

Hydrated lime is delivered to site in bulk tankers and pneumatically offloaded into the storage silo.

The lime is added to the lime mixing tank via a screw feeder at the base of the silo, and a pre-wetting system. Lime and process water addition is controlled to produce a lime slurry of consistent density reading for dosing into the plant. Lime is mixed in batches, based on tank storage levels.

After mixing, the lime slurry is transferred to the lime storage tank, from which pumps circulate the slurry throughout the milling and flotation circuits via a ring main pipeline. Lime slurry is supplied to process areas including the primary mill, rougher conditioning tank, regrind cyclone feed hopper, and Cleaner 1 conditioning tank.

The lime mixing and storage area has a dedicated sump for spillage recovery and clean-up.

Concentrate Flocculant

Concentrate flocculant powder is delivered to site in 25 kg bags on pallets. Bags are emptied into the feed hopper of the flocculant mixing system. The dry flocculant is added to the mixing system via a screw feeder and mixed with water in the mixing tank to create a consistent flocculent solution. The flocculent solution is aged, then transferred to the storage tank prior to dosing into the concentrate thickener to promote solids settling rate and supernatant (overflow) clarity.

The flocculant mixing and storage has a dedicated sump and pumping arrangement for spillage containment and clean-up.

17.3.16 Tailings Treatment Facility Reagents

The reagents at the tailings treatment facility are flocculant and coagulant. Coagulant is required to assist with the settling of ultra-fine particles from the softer gangue minerals, whilst the flocculant drives settling rate in the thickener.

Tailings Flocculant

Tailings flocculant powder is delivered to site in bulk bags, which are stored in a dry area. Bags are taken to the mixing tank and emptied into the dry storage tank via an enclosed bag breaker. The flocculant powder is discharged from the dry storage tank through a screw feeder into an agitated mixing tank, where it is combined with water to prepare a solution.

The prepared flocculant solution is aged and transferred to the storage tank and dosed to the tailings thickener feed using dosing pumps.

The flocculant handling area has a dedicated sump and pumping arrangement for containment of spillage. Any spillage generated within this area is collected and returned to the process via the slurry handling system.

Tailings Coagulant

Coagulant is delivered to site and stored in IBCs. It is transferred from the IBCs to the storage tank and dosed to the tailings feed system using metering pumps.

17.3.17 Water Services

The plant has an integrated water system, with services at both the main process plant area and the tailings treatment facility.

Process Plant Water Services

The process plant water services system comprises:

- Raw water,
- Process water,
- Filtered process water,
- Gland water,
- Potable water,
- Fire water systems,

with distribution to the relevant process and other necessary areas.

Raw water is supplied from a combination of groundwater bores and clean run-off catchment dams and stored in the raw water tank. Pumps distribute water for various requirements including make-up water to the process water system and filtered process water systems, and supply to the potable water production plant.

Process water is recovered directly from the tailings and concentrate thickener overflow streams (recovered water from the filters reports back to the thickeners) and stored in the process water pond for reuse within the plant. Process water is distributed via pumps to the required addition points, mainly in the milling and flotation circuits.

Process water is also be used for spray applications and services throughout the plant, including screen sprays, launder sprays, conditioning tanks, and dust suppression systems. Collection of run-off water and return to the process water pond is provided via event ponds and low-point drains within the process plant.

Some applications such as gland water, concentrate washing, reagent mixing and filter washing require water that is cleaner than process water, but the total plant water balance does not allow additional raw water to be added into the plant. To clean water for these uses, some process water is passed through a sand filter which removes the majority of turbid particles. The process water filter is located at the tailings treatment facility.

Gland seal water is supplied to centrifugal pumps throughout the process plant, including milling, flotation, regrind, thickening, filtration and tailings areas.

Potable water is generated by the water treatment plant from raw water and stored in the site potable water tank. Potable water is distributed site wide for consumption. It is also distributed to plant facilities including safety showers, buildings, offices and laboratory services.

The fire water system includes fire water pumps, hydrants and hose reels to support fire protection requirements across the plant site. The fire water system uses raw water.

Integration of the water services system includes return water from the tailings dewatering circuits, and provision of event and storage ponds for containment and pumping of process water and storm water in the plant area.

Tailings Treatment Facility Water Services

The tailings treatment facility water services system comprises the same water types as in the main process plant – raw, process, filtered process, gland, potable and fire water; with distribution to the required applications and usage points. Filtrate from the tailings filters is also re-circulated as flushing water for the filter presses.

Raw water is pumped to the tailings treatment facility from the site raw water storage facilities.

The tailings thickener overflow tank is the main collection point for recovery of process water for re-use in the milling and flotation circuits. The recovered process water is pumped back to the process water pond, with some split off and passed through sand filters to become the filtered process water stream.

Filtered process water is stored in a dedicated tank and distributed to process areas in the tailings treatment and main plant areas, including the tailings filters, reagent mixing systems, flotation wash water and gland water for centrifugal pumps.

The tailings area water services system also includes management of runoff water from the tailings filtration, process and loading areas. Run-off water is captured via drains and sumps and directed to the tailings thickener and/or the tailings event pond for containment and subsequent recycling back into to the process.

The fire water system for the tailings treatment facility includes pumps, hydrants and hose reels to support fire protection requirements.

17.3.18 Air Services

The separate main plant and tailings treatment facilities means that compressed air services are installed in both areas. Low-pressure air blowers are situated in the main plant only.

Process Plant Air Services

The process plant air services system comprises:

- Plant air supplied from a central compressed air system.
- Instrument quality compressed air.
- Low-pressure flotation blowers.
- Air receivers and dryers.
- Distribution network.

Plant and instrument air is distributed throughout the process plant to all necessary areas and equipment, and general service points.

Air supply to the flotation circuit is provided via air low-pressure blowers to the rougher and cleaner flotation cells, and the concentrate filter has a dedicated compressed air supply.

The air services system also supports ancillary equipment including dust collection systems installed across the plant, in location including the crushers, fine ore bin, lime handling area, and concentrate storage facility.

Tailings Treatment Facility Air Services

The tailings treatment facility air services comprise of compressed air and compressed instrument-quality air. The system includes compressors, receivers, and dryers. Compressed air is used in the tailings filters (especially in the filtration drying stages) and general service points throughout the tailings treatment facility. Instrument air is also used throughput the general area for pneumatically controlled instruments.

17.3.19 Plant Control

The plant control system consists of:

- A plant control system (PCS) designed around a moderate level of automation and monitoring, consistent with modern process plant operations.
- Instrumentation.
- Radio telemetry for the western borehole, and harvestable rights property M pumps.
- Communications equipment and licenses for external communications to company head office.

There is one central plant control room for the main process plant located adjacent to the mill platform.

Within the plant control room are two operator interface terminals (OITs), an engineering workstation, and a server OIT. The control system is provided with an uninterruptable power supply (UPS).

Plant programmable logic controllers (PLCs) are integrated into the motor control centres (MCCs). All the field input-output (I/O) signals are connected to the plant PLCs via hard wired signals to remote I/O panels.

There are also some vendor supplied PLCs for proprietary equipment. Communication between the plant PLCs and the vendor PLCs is via an Ethernet compatible protocol using either CAT6 cabling or fibre optic cable.

Network panels are provided in each switchroom and control room to accommodate switches, fibre optic break out trays (FOBOTs), and UPSs.

There is an additional control room inside the tailings filter building with two further OITs.

Operation of most electrical drives and actuated valves is performed from the distributed control system (DCS) OITs. There are no Remote / Local selector switches in the field. However, field operators may start equipment local to the drive if the control room has selected Local at the OIT. Local mushroom head style Lock-Off-Stop stations are installed which are hardwired to the drive starter circuit to enable local stop / emergency stop regardless of drive mode selection.

In general, the use of actuated isolation or control valves is implemented around the plant for automatic control loops or sequencing as part of the plant control sequence. All actuated valves and control valves are operated from the OITs with remote position indication available. Automatic control valves are controlled by proportional, integral and derivative (PID) loops within the DCS.

The DCS performs all digital and analogue control functions, including PID control. Faceplates on the DCS displays facilitate the entry of setpoints, readout of process variables (PVs) and controlled variables (CVs) and entry of the three PID parameters.

The majority of equipment interlocks are software configurable. However, selected drives are hard wired to provide the required level of personal safety protection, e.g. the emergency stop buttons associated with each and every motor and the pull wire switches associated with conveyors.

All alarm and trip circuits from field or local panel mounted contacts are based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of power supply, the equipment will return to a de-energized state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops are not incorporated with non-packaged plant equipment. However, in any sequential process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g. trip of mill feed conveyor will result in stop of fine ore bin reclaim feeder).

18.0 PROJECT INFRASTRUCTURE

18.1 Roads and Layout

18.1.1 External / Public Roads

Access to the Project site is currently provided via Lue Road, Pyangle Road and Maloneys Road. Lue Road is the main road between Mudgee and Rylstone whilst Pyangle Road and Maloneys Road are local roads.

During the early stages of the site establishment and pre-construction, the existing site office and workshop will continue to be used and accessed by the existing road network, using Pyangle and Maloneys roads. The intersection of Lue and Pyangle roads, and sections of Pyangle road and Maloneys road will be upgraded to handle necessary loads.

The existing Maloneys Road traverses through the project operational area and will be relocated during construction. Once the realigned Maloneys Road is completed, the existing section of Maloneys Road inside the project boundary will be closed to the public. As the new realigned Maloneys Road will service both the public and mine traffic, the engineering design has been completed to an AACE Class 1 standard to ensure safety and all weather trafficability.

Operational site access will be via the new realigned Maloneys Road to the northwest corner of the Project boundary, then along a section of the original Maloneys Road (upgraded) to a new site access road to the operating offices.

18.1.2 Internal Site Roads

The project site includes a number of internal roads, including:

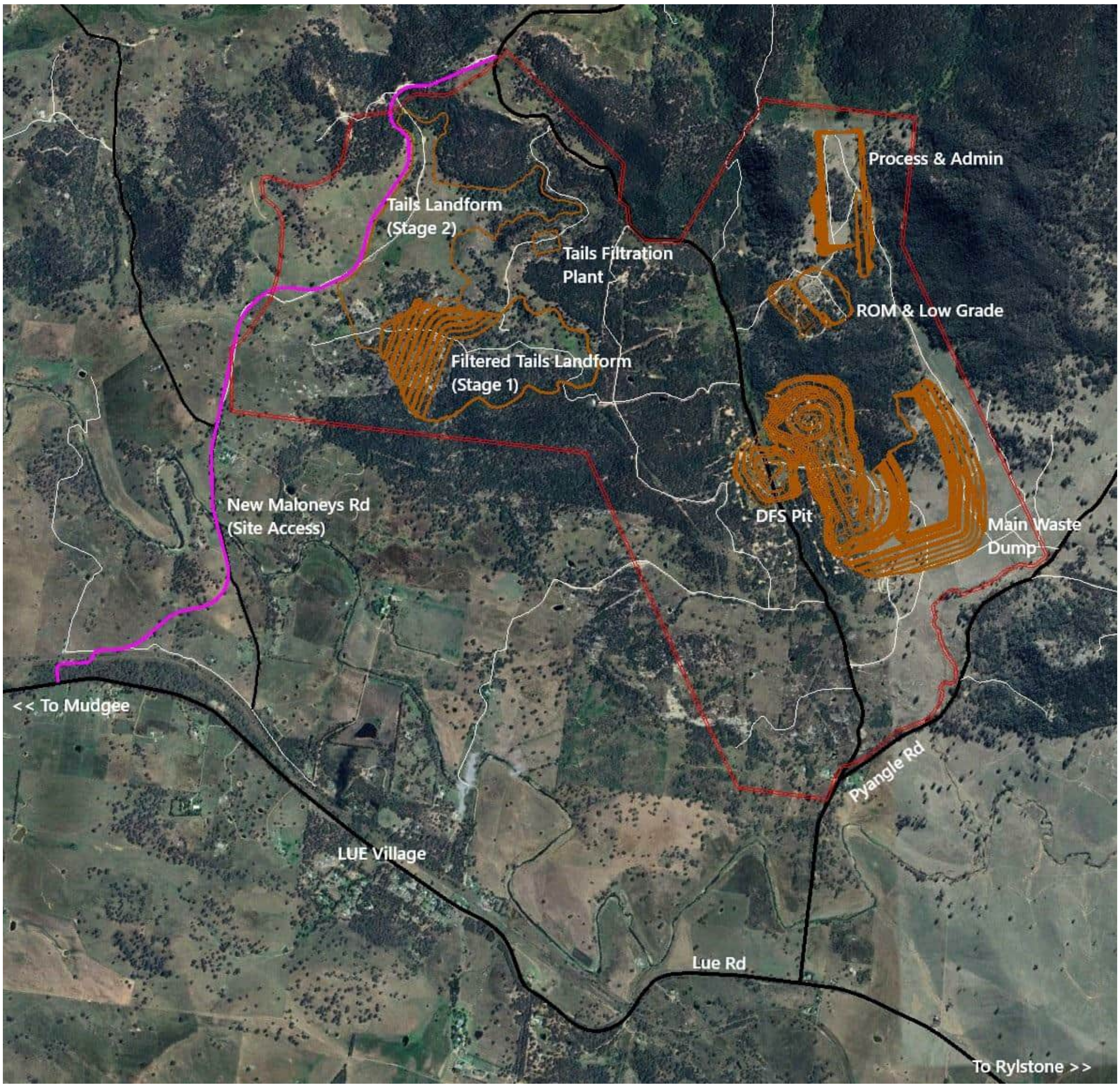
- Site access road connecting the main office, process and mining facilities, warehouse, and workshops to the public Maloneys Road.
- Haul roads from the open pit to waste storage facilities, ROM pad, low grade stockpile, mine offices and workshop, and the FTL.
- Service roads along internal power and pipelines.
- Access to water management infrastructure.
- General access / service roads for other site activities, such as environmental monitoring.

Most of the roads will be built during construction by earth moving, construction and mining contractors. Specifications for these roads will vary depending on the intended use of the road and types of equipment planned to operate on them.

18.1.3 Overall Site Layout

The overall site layout is shown in Figure 18.1.1. The drawing shows the major features of the Project and its infrastructure including the mine pit, main process plant, tailings filtration plant, FTL and mine lease boundary. Project infrastructure and pads are shown in brown, main public roads in black, small or private tracks in white, the new re-aligned Maloneys road in magenta and the Project boundary in red.

Figure 18.1.1 Overall Site Layout



18.2 Power Supply

18.2.1 Construction Power

The Project site is currently serviced in different locations by a number of 11 kV and 22 kV rural overhead powerlines (OHPL). While these OHPLs have limited capacity, they can service pre-construction and construction offices and minor workshop facilities. In later stages of construction, diesel gensets will be required where main construction area loads increase and in peripheral areas.

Large gensets purchased for construction power at the process plant will be retained for backup power during operations. Smaller gensets purchased for peripheral loads, such as remote pump locations, will also be retained for operational power supply.

18.2.2 Operational Power

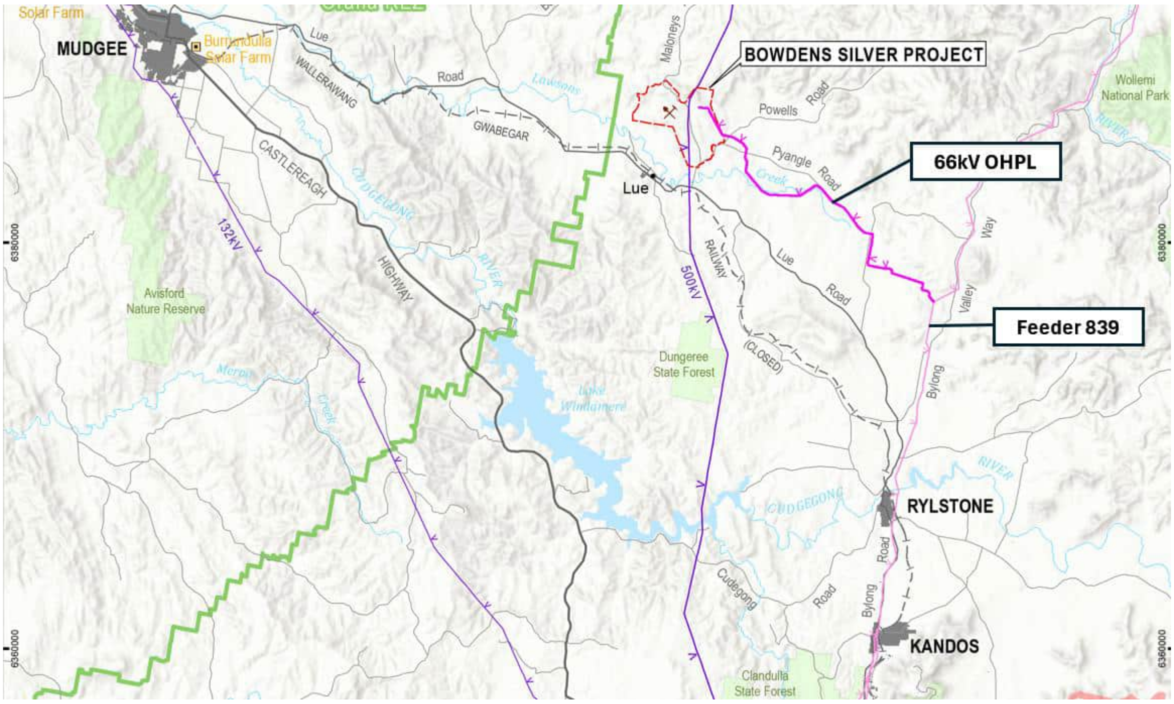
Electrical connection and load details are provided below:

- Connected load – 15.0 megawatts (MW).
- Maximum Demand – 12.3 MW.
- Average annual demand – 10.5 MW.
- Average energy consumption – 82.8 gigawatt-hours per year (GWh/y).
- Largest single motor – 3.5 MW (ball mill).
- Average grid power price A$.19 /kWh.

Site power will be provided by a new 66 kV transmission line connecting site to the existing Endeavour Energy owned 66 kV feeder 839 running from Kandos to Bylong. A 66 kV combined overhead / underground transmission line approximately 19.5 km long will connect site to feeder 839.

Outlying locations requiring intermediate power (such as pumps for harvestable water from dams) will either be powered by remote diesel generators or internal 11 kV.

Figure 18.2.1 OHPL Connection to Endeavour Energy



18.3 Power Distribution

The onsite power distribution consists of:

- A 66/11 kV substation, including an 11 kV switchboard, an 11 kV capacitor bank and a station services transformer.
- An emergency diesel generator with an installed load capable of supplying the nominated emergency loads estimated at an installed load of approximately 1,200 kilovolt-amperes (kVA).
- A pre-fabricated, transportable switchroom which houses the main 11 kV switchboard for the processing plant as well as the medium voltage variable speed drives for the primary mill and ball mill.
- Medium voltage (MV) cables from the 66/11 kV substation and the emergency generator to the main 11 kV switchboard.
- MV cables from the main 11 kV switchboard to the 11 kV / 433 V transformers located in the plant load areas, primary mill and call mill motors, and the two overhead powerlines.
- 11 kV / 433 V transformers at the plant load areas.
- Pre-fabricated, transportable electrical switchrooms to house the low voltage MCC, LV VSD and associated equipment.
- Outdoor MCC for the office complex loads.
- Overhead powerlines to the tailing filtration area, northeast borefield, FTL valley catchment dam, TFL runoff dam, WRE seepage dam, and pit dewatering pumps.

The main site switchboards and related load information is provided in Table 18.3.1.

Table 18.3.1 Main Site Switchboard Loads

Switchboard Name	Voltage V	Connected Load kW	Annual Average kW	Annual Energy MWh/y
Plant 11 kV Switchboard	11,000	14,955	9,412	82,450
Primary Crusher Area 415 V MCC	415	659	242	2,117
Secondary Crushing Area 415 V MCC	415	700	232	2,032
Milling Area 415 V MCC	415	1,464	954	8,356
Flotation Area 415 V MCC	415	1,866	1,215	10,646
Concentrate and Tailings Area 415 V MCC	415	1,893	1,225	10,729
Services Area 415 V MCC	415	1,160	673	5,899
Buildings Area 415 V MCC	415	9	4	34
Site 11 kV Switchroom L&SP Distribution Board	415	60	44	387
Site 11 kV Switchboard	11,000	15,015	9,456	82,836

18.3.1 66/11 kV Substation

The 66/11 kV substation located adjacent to the process plant has a 66 kV circuit breaker, and a single 66/11 kV, 16/20 MVA transformer. An 11 kV cable exits the 66/11 kV substation to supply the main 11 kV switchboard within the process plant.

18.3.2 Emergency Diesel Generator

A single emergency diesel generator (EDG) provides power the maximum demand load of the critical process equipment load of approximately 700 kW. Additional loads fed from the EDG include area lighting, small power requirements and the administration area. The EDG has a design standby power output of 1,936 kW which includes derating factors for temperature and altitude.

18.3.3 MV Switchroom

The MV switchroom houses an 11 kV switchboard which feeds seven separate 11/0.433 kV transformers, each servicing a dedicated MCC. The primary mill (2,900 kW), ball mill (3,500 kW) and two 11 kV overhead powerlines are also fed from the MV switchboard. The switchroom is equipped with a spare 11 kV feeder panel for potential future expansion.

18.3.4 11kV Overhead Powerlines

Two 11 kV OHPLs have been included. The first OHPL supplies the northeast borefield, tailings filtration area, TFL valley catchment dam, and TFL runoff dam with multiple droppers to various transformers along the route. The second OHPL supplies the pit dewatering pumps and WRE seepage dam with multiple droppers to various transformers along the route.

Optical ground wire will be strung at the top of the powerline, providing grounding and fibre optical cable for communication back to the plant control system.

18.3.5 Distribution Transformers

The design includes for 11 kV / 433 V oil-filled, distribution transformers at the plant load centres and installed in concrete bunded areas with fire walls on three sides. Transformers have been designed with 20% spare capacity for any future additional equipment.

The rating of the transformers will vary according to the calculated maximum demand for each area; however, some standardisation has been applied to limit the number of different ratings of the transformers.

18.3.6 Primary Mill and Ball Mill Motor Variable Speed Drives

The primary mill and ball mill motors are powered by MV VSDs installed in switchrooms. An 11 kV supply for each mill connects to an external 11/3.3 kV oil filled transformer installed in a concrete bunded area.

18.3.7 Pre-Fabricated Switchrooms

Dedicated pre-fabricated, transportable switchrooms will be located at each major load centres, being:

- Primary crushing area.
- Secondary crushing area.
- Milling area.
- Flotation area.
- Services area.
- Tailings filtration area.

Switchrooms are single storey, air-conditioned, 2-h fire rated and mounted on supports approximately 2,000 mm above the finished concrete level.

18.3.8 Motor Control Centres

The 415 V MCC's will be installed within pre-fabricated, transportable switchroom buildings and will be configured for bottom entry cables.

All electrical motor drives in the plant are rated at either three phase 415 volt alternating current (VAC) or single phase 240 VAC for smaller drives. The largest LV drives in the plant are:

- 315 kW drive motor for the secondary crusher.
- 355 kW milling cyclone feed pumps.
- 355 kW drive motor for the regrind mill.

All drives have a local emergency stop button (latched) at the local control station (LCS), as well as padlockable isolators at the associated MCC. Drives can be selected into either manual start mode or remotely started from the plant control system in the control room.

Equipment isolation for maintenance will be carried out at the motor starter module located within the MCC. A design allowance of 15% spare capacity or one full spare tier is provided in each MCC.

18.4 Control Systems

The control systems consists of:

- A PCS.
- Instrumentation.
- Radio telemetry for the borefields and harvestable rights pumps.
- Communications equipment and licenses for external communications.

18.4.1 Control System Design

A single central plant control room, located on the mill planform, services the process plant. Two operator interface terminals (engineering workstation and a server operator interface are included in the plant control room. The control system will be provided with a UPS.

Plant PLCs will be integrated into the MCCs. All the field I/O signals will be connected to the plant PLCs via hard wired signals to remote I/O panels. Some are vendor supplied PLCs for proprietary equipment. Communication between the plant PLCs and the vendor PLCs will be via an ethernet compatible protocol using either CAT6 cabling or fibre optic cable.

There will be an additional control room inside the tailings filtration building which will also contain two operator interface terminals.

18.5 Water Supply

SRK developed a site-wide water balance and an updated groundwater numerical model (SRK, 2026a; SRK, 2026b) as part of the surface water and groundwater impact assessments for the Project.

18.5.1 Water Demand and Water Balance Modelling

After internal recycling and use of contact water are taken into account, the modelled average make-up water requirement is <10 litres per second (L/s), with the 95th percentile demand being <30 L/s.

A predictive, probabilistic water balance was developed using a dynamic system model (DSM) implemented in the GoldSim™ software platform (version 15). The DSM incorporates:

- Climate inputs (rainfall, evaporation).
- Surface water runoff from disturbed and undisturbed catchments.
- Infiltration and recharge to groundwater.
- Process plant, dust suppression and miscellaneous operational demands.
- Site storage facilities and water recycling circuits.

The GoldSim™ platform allows Monte Carlo simulation of parameter uncertainty and climate variability. Key results from the water balance are summarised below, with details provided in the water balance report (SRK, 2026a).

The water balance was developed for the following principal objectives:

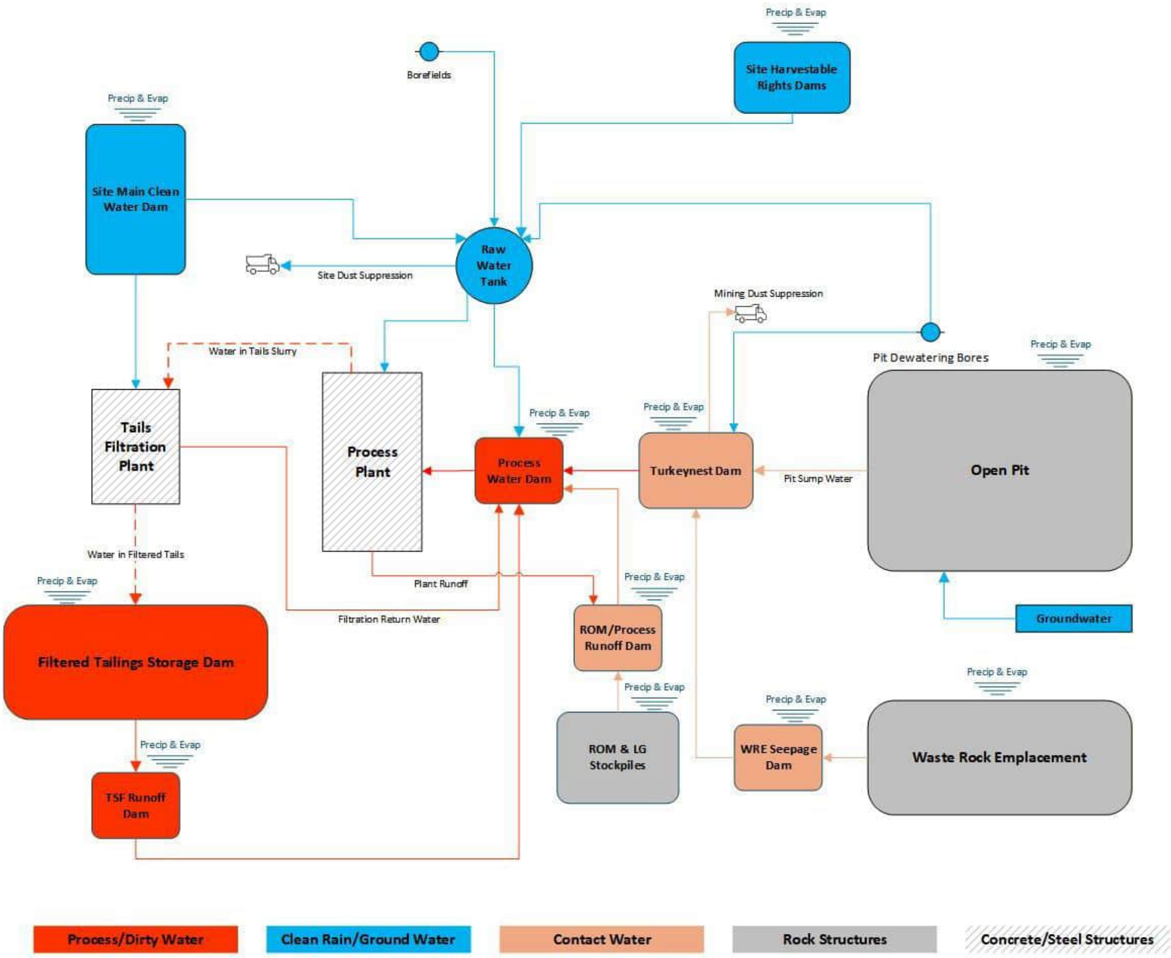
- Estimation of flow to-and-from individual mine infrastructure areas (e.g. pit, waste rock emplacement (WRE), FTL, low-grade stockpile).
- Estimation of combined flows from catchment and mine facilities where water can be controlled, recycled or re-used.
- Quantification of water demands for the process and filter plant, dust control, and other operational uses.
- Estimation of water inventory on site over operational and post-closure phases.

The Project's water management strategy is based on differentiating water from disturbed and undisturbed catchments:

- Mine effected (contact) water is contained within the operational zones and reused in the process plant and for dust suppression where possible.
- Run-off from undisturbed catchments is either captured in harvestable rights dams or diverted for discharge to the environment in accordance with regulatory requirements.

The relationship between these components and the proposed flow paths for contact and non-contact water, including water return circuit, is illustrated in Figure 18.5.1.

Figure 18.5.1 Conceptual View of Project Water Movement



Source: SRK, 2026a.

18.5.2 Groundwater Modelling and Pit Dewatering

The groundwater numerical model was constructed using the MODFLOW-USG finite difference code, implemented via Groundwater Vistas graphical user interface (SRK, 2026b). The model simulates groundwater levels and inflows to the planned open pit over the life of mine and provides boundary conditions for the site wide water balance.

The current mine plan incorporates a 14-year pit operating period, with progressive backfilling of the eastern section of the pit from Year 9 onwards. During operations, the pit will be maintained in a dewatered condition through pumping to the Turkey Nest Dam.

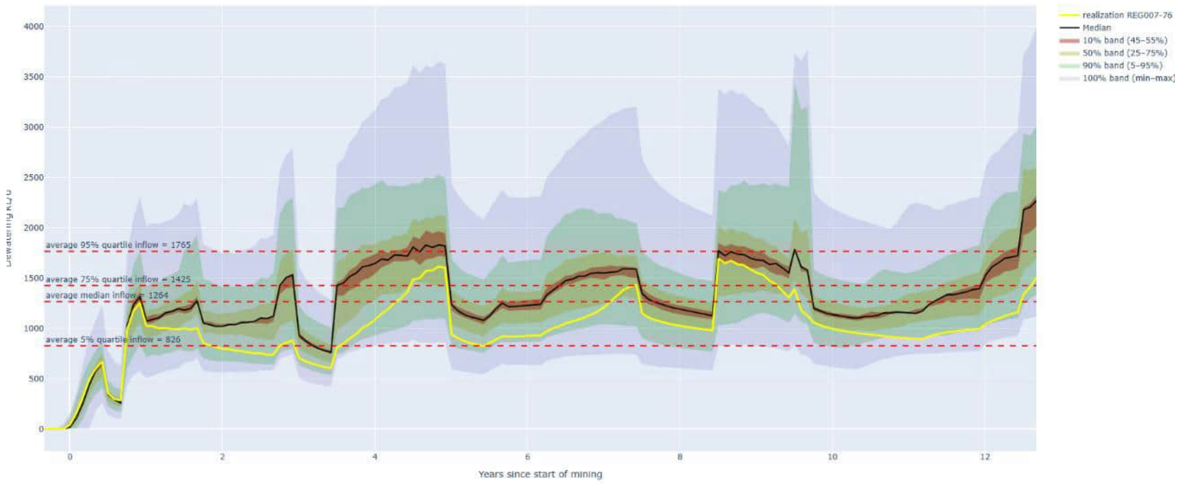
Results from the numerical model (SRK, 2026b) indicate:

- Average groundwater inflows to the pit range from 826 to 1,765 kilolitres per day (kL/d) (approximately 9.6 - 20.4 L/s), with an average inflow of 1,425 kL/d or less (approximately 16.5 L/s).

- Peak inflow rates higher than the long-term average are expected as the pit deepens and intersects more transmissive bedrock and fault zones.
- Short duration temporary peak inflows of up to 2,100 kL/d (approximately 24.3 L/s) are likely, and inflows up to 3,500 kL/d (approximately 40.5 L/s) are possible but considered unlikely.

Simulated inflows corresponding to the DFS mine schedule are shown in Figure 18.5.2.

Figure 18.5.2 Groundwater Inflows During Operations

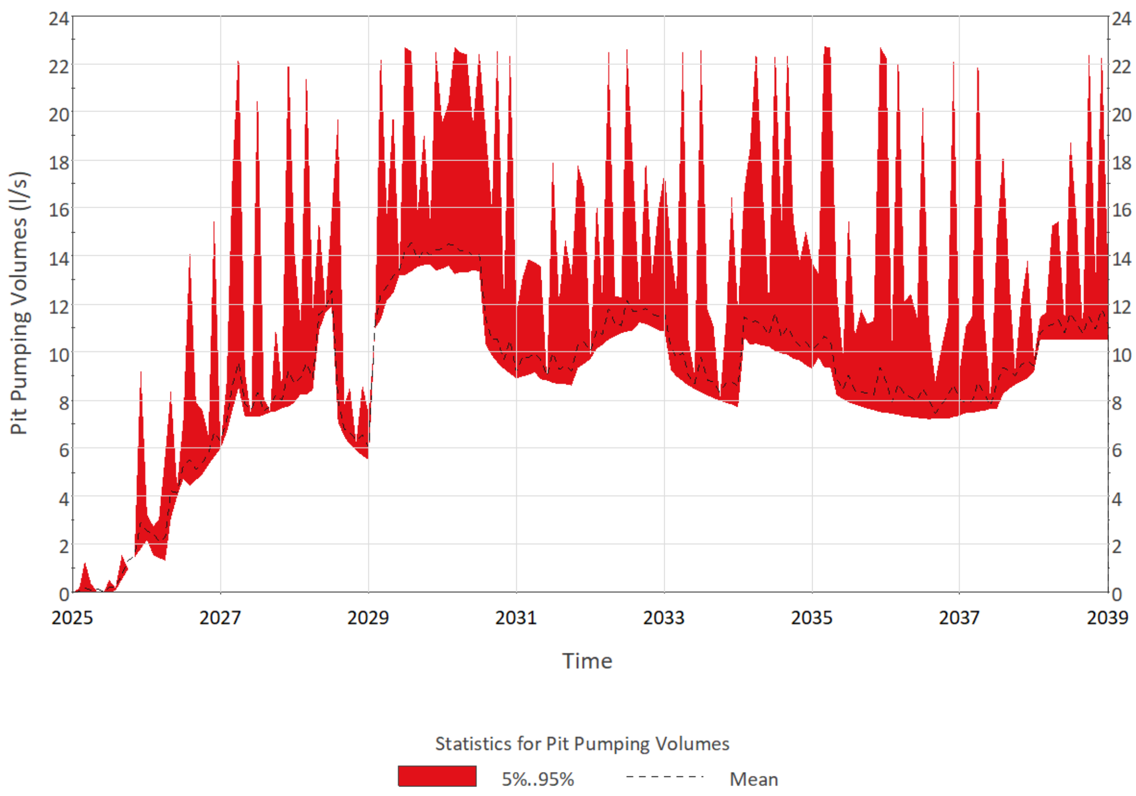


Source: SRK, 2026b.

In the water balance, pit pumping rate is simulated based on a sump capacity of 1,600 m^3, with a minimum retained volume of 200 m^3, pumped to the 8,000 m^3 Turkey Nest Dam during operation.

The required pit pumping rate during operation is estimated to be less than 25 L/s (Figure 18.5.3), including allowance for peak inflows. Dewatering pumps and associated infrastructure are designed for the peak inflows rather than average inflows.

Figure 18.5.3 Simulated Pumping Volumes from the Main Pit



Source: SRK, 2026a.

18.5.3 Water Storage

The main contact and non-contact water storages included in the site-wide water balance are the pit sump, Turkey Nest Dam, process pond, WRE seepage pond, FTL Run-off Dam, and Main Valley Dam.

Storage capacities and operating behaviour were assessed using volume-elevation relationships and simulated under stochastic climate conditions (WGEN) and dynamic operational conditions in GoldSim™. Key findings are:

- The Turkey Nest Dam (8,000 m^3) and process pond (700 m^3) provide sufficient buffering capacity to manage variability in contact water inflows from the pit, WRE and FTL Run-off Dam, while maintaining process water supply within operating ranges.
- The WRE seepage pond (94,000 m^3) provides a substantial buffer relative to predictive seepage and run-off volumes. Simulated water levels remain well within freeboard during operations, with only moderate drawdown during extended low-inflow periods.
- The FTL Run-off Dam can be maintained at an average volume of 50,000 m^3, which is below the maximum operating volume of 118,000 m^3, using a pumping capacity of 23 L/s to the Turkey Nest Dam and process water circuits.
- The Main Valley Dam is managed as the main non-contact raw water storage. It receives approximately 100 – 200 m^3/d of combined runoff and direct rainfall during operations and has more than adequate storage and transfer capacity relative to modelled demands.

Model results indicate that, under the simulated climate conditions and operating strategy, the proposed storages provide adequate operational buffering and are highly unlikely to overtop. Residual overtopping risk are manageable through operational controls.

18.5.4 Operational Water Supply and Make-Up Requirements

The site-wide water balance indicates that:

- Operational water demands can be met largely through recycling of mine affected (contact) water and pit dewatering inflows, supplemented by a relatively modest volume of external make-up water.
- Dust suppression demand averages about 600 m^3/d over the operating period (approximately 6.9 L/s), varying with climate conditions and applied across haul roads, the FTL, open pit, low-grade stockpile and WRE. Pit dust suppression demand is preferentially supplied from the Turkey Nest Dam.
- After internal recycling and use of contact water are taken into account, the modelled average make-up water requirement of <10 L/s, with the 95th percentile demand of <30 L/s.

Make-up water is planned to be supplied from a combination of:

- The Main Valley Dam.
- Onsite and offsite production bores.
- Harvestable rights dams on Bowdens owned properties (if required).

Within the overall water supply strategy, the Main Valley Dam is intended to function as a balancing and contingency storage rather than a critical supply source.

18.5.5 Summary

Based on the current mine plan, water balance modelling and groundwater inflow analysis, the Project is expected to meet its operational water requirements predominantly through internal recycling, pit dewatering, and planned surface and groundwater supplies. Further work (groundwater exploration drilling and bore installation) is planned to confirm and secure the required make-up water capacity.

18.6 Waste Rock Management

SRK developed a Class 3 design for the WRE and NAF waste dump (ND) for BSP (SRK, 2026c, SRK, 2026d).

18.6.1 Waste Rock Geochemistry

Static geochemical and mineralogical assessment of waste rock has been undertaken on core samples from various drill programs between 2012 and 2025 by Graeme Campbell and Associates, and Okane Consultants. Kinetic geochemical testing has been undertaken on waste rock samples since 2012 in the form of small-scale (1.5 – 2.0 kg, 28 samples) and large-scale (18 kg, 10 samples) laboratory-based kinetic leach columns. The 2025 kinetic tests are still running and continue to provide ongoing data to inform acid mine drainage (AMD) management for the Project.

SRK reviewed the acid and metalliferous drainage potential classification criteria for waste rock and recommended three classifications based on the oxidation state and sulphur content of the waste rock:

- Type 1 (non-acid forming, NAF).
- Type 2 (NAF with neutral mine drainage potential).
- Type 3 (potentially acid forming, PAF).

Current AMD management plans involve selective handling and placement of waste rock (including backfill to the open pit and encapsulation of Type 3 waste rock within the waste rock emplacement), seepage capture and onsite management during mine operation, and progressive rehabilitation plans that include placement of a cover on the waste rock emplacement to reduce long-term seepage and oxygen ingress.

18.6.2 Waste Rock Emplacement

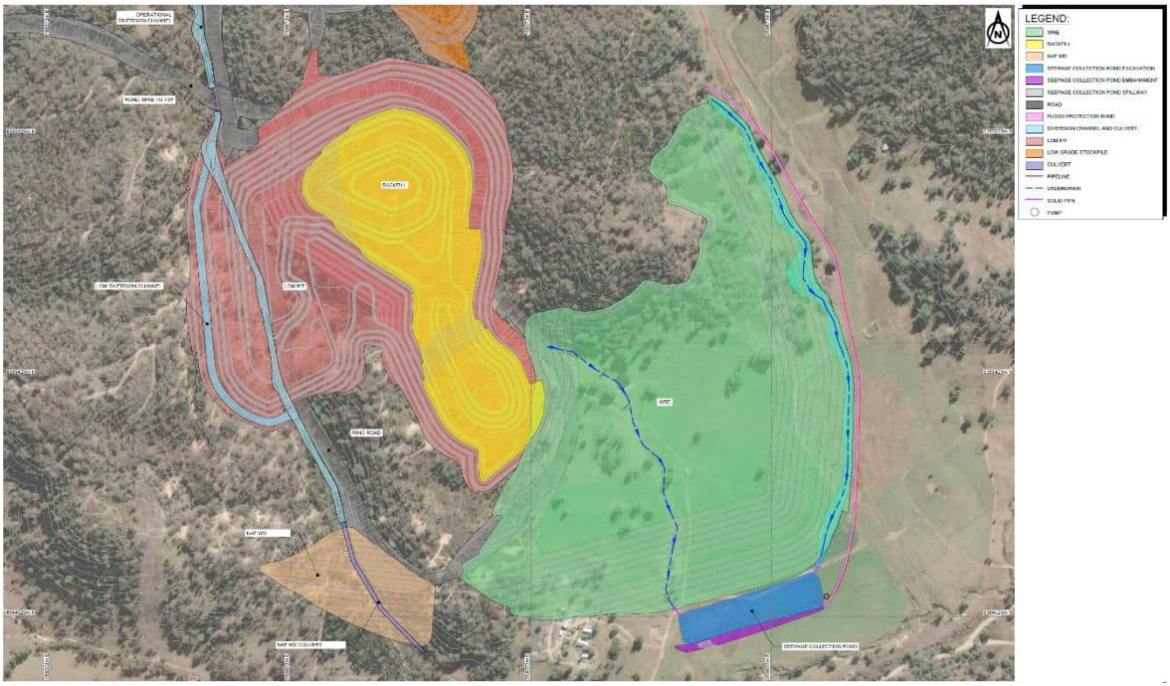
Approximately 11.6 Mm^3 (placed volume) waste rock materials, comprising the three types of waste materials, will be deposited within the WRE. The WRE design incorporates features to reduce AMD potential at source, including paddock dumping and encapsulation of Type 3 waste rock, and will be lined with low permeability material at the base (basal liner).

The WRE encapsulation includes:

- A minimum 10 m thick layer (measured horizontally) of Type 1 material on the outer wall.
- A minimum 5 m thick Type 1 waste cap above the top surface of the Type 3 material.

During operations, WRE seepage and surface runoff will be captured in a seepage collection pond located south of the WRE via two underdrains (central and toe underdrains) and flood protection bund. A general arrangement presenting the LOM water management measures is shown in Figure 18.6.1 and described in the following subsections.

Figure 18.6.1 WRE General Arrangement



Source: SRK, 2026c.

Low Permeability Basal Liner (600 mm Thick)

After general surface preparation, a layer of compacted, low permeability material will be placed over the footprint of the WRE and associated water management infrastructure to mitigate against seepage. The layer will be 600 mm thick and constructed in two 300 mm compacted lifts. This excludes the seepage collection pond basin, which will have a layer of 300 mm thick low permeability material, overlain by an impervious HDPE geomembrane liner. This will allow the contact water to follow the designed drainage characteristics within the footprint, flowing downstream to the seepage collection pond.

Central Underdrain

A central underdrain will be constructed along the natural drainage line to promote and direct seepage towards the seepage collection pond. Since the seepage from the central underdrain catchment is expected to report to the seepage collection pond via existing natural drainage pathways (even without the underdrain), the central underdrain provides additional design redundancy to further enhance seepage collection efficiency and control.

Toe Underdrain

A toe underdrain will be constructed to provide a preferential pathway for the seepage generated in the eastern portion of WRE catchment to flow within the WRE footprint and discharge to the seepage collection pond. In the absence of the toe underdrain, seepage generated in this catchment would be expected to flow eastwards along existing natural drainage pathways, daylight at the WRE toe, and then be intercepted by the flood protection bund before being directed to the seepage collection pond.

The toe underdrain is intended to support progressive closure of the landform by improving (or maintaining) surface water quality along the eastern toe of the WRE, and by providing additional design redundancy for seepage control. The WRE performance can therefore be demonstrated and optimised during operations, rather than being deferred to closure.

Seepage Collection Pond

A seepage collection pond will be constructed downstream of the WRE to collect contact water (seepage and runoff from the WRE). The collected contact water will be pumped to the plant for re-use in the processing circuit. The pond will comprise an excavated basin, an earthen embankment and a spillway.

The pond has been sized to include allowances for sediment and settling accumulation zones. These zones represent the operational volume for the pond. To maintain sufficient capacity for ongoing settled sediment, periodic dredging of the impoundment may be required.

The method of sizing for the seepage collection pond has been adopted from the *Managing Urban Stormwater: Soils and Construction – Volume 1 and Volume 2E Mines and Quarries* (Landcom, 2004; DECC, 2008) guidelines.

The embankment will be constructed mainly with Type 1 waste material and will have a 5 m wide crest, with an upstream slope of 1:2 (v:h) and a downstream slope of 1:3 (v:h). The water depth at the embankment toe due to water level rising in Hawkins Creek during 1:100 annual exceedance probability (AEP) storm event is estimated to be 0.8 m. A 1 m thick layer of rip rap will be placed on the downstream face of the embankment and along the spillway for erosion protection.

The pond basin, including the upstream face of the embankment, will be lined with a 300 mm thick layer of low permeability material overlain by a HDPE liner to mitigate potential seepage through the embankment and foundation.

An emergency spillway will be constructed to accommodate a 1:100 AEP flood. The spillway will have a base width of 20 m, with side slopes of 1:3 (v:h). The spillway depth will transition from 1 m at the crest to 0.5 m with a longitudinal slope of 1:6 (v:h) along the spillway chute. A lining system comprising geofabric on either side of an HDPE liner, overlain by 1 m thick layer of rip rap will be placed on the spillway face to mitigate the risk of the phreatic surface rising in the embankment and potential erosion due to spillway flows.

Flood Protection Bund

The WRE footprint currently encroaches into the 1:100 AEP flood extents of both Price Creek and Hawkins Creek. It is estimated that the maximum peak flood depth at the flood protection bund is approximately 0.4 m.

A flood protection bund, nominally 1 m high, will be constructed along the eastern toe of the WRE. The WRE basal liner will be extended to the upstream crest of the bund to mitigate potential seepage of contact water into the ground and for clean and contact water separation, as follows:

- Contact water from the WRE on the western side of the bund will be captured and conveyed to the seepage collection pond.
- Clean water from Price Creek will flow along the eastern side of the bund and continue southwards into Hawkins Creek.

Geotechnical Stability

A geotechnical assessment was undertaken by SRK in accordance *with Guidelines for Mine Waste Dump and Stockpile Design* (Hawley et al., 2017) and *Guidelines for Open Pit and Waste Dump Closure* (de Graaf et al., 2025). The results indicate the proposed WRE geometries are adequately stable and can be implemented as designed. The opportunity to optimise the configuration will be reviewed once production begins and the as-constructed materials can be further characterised.

18.6.3 NAF Waste Dump

Approximately 1.8 Mm^3 of Type 1 waste material will be placed in the NAF Waste Dump (ND). The ND is located in Blackmans Gully, with the direction of surface water drainage being from north to south. A culvert has therefore been designed to convey surface water as detailed in Section 18.8.

A geotechnical assessment was undertaken by SRK (2026c) in accordance with *Guidelines for Mine Waste Dump and Stockpile Design* (Hawley et al., 2017) and *Guidelines for Open Pit and Waste Dump Closure* (de Graaf et al., 2025). The results indicate the proposed ND geometries are adequately stable and can be implemented as designed. The opportunity to optimise the configuration will be reviewed once production begins and the as-constructed materials can be further characterised.

18.6.4 Closure

All materials in the ND will be rehandled at closure and a conceptual closure design has been developed for the WRE. During closure, water collected in the WRE seepage pond will be transferred to the pit via pumping and/or passive drainage. Once the closure criteria are achieved, the water management infrastructure (seepage collection pond and flood protection bund) will be decommissioned and removed, leaving the WRE as a permanent landform on site.

Closure cover

The WRE cover system incorporates (from top to bottom):

- 30 cm thick topsoil.
- 30 cm thick subsoil.
- 40 cm thick Type 1 waste rock (gravels and cobbles).
- 40 cm thick Type 1 waste rock (boulders).
- 2 cm thick geosynthetic clay liner.

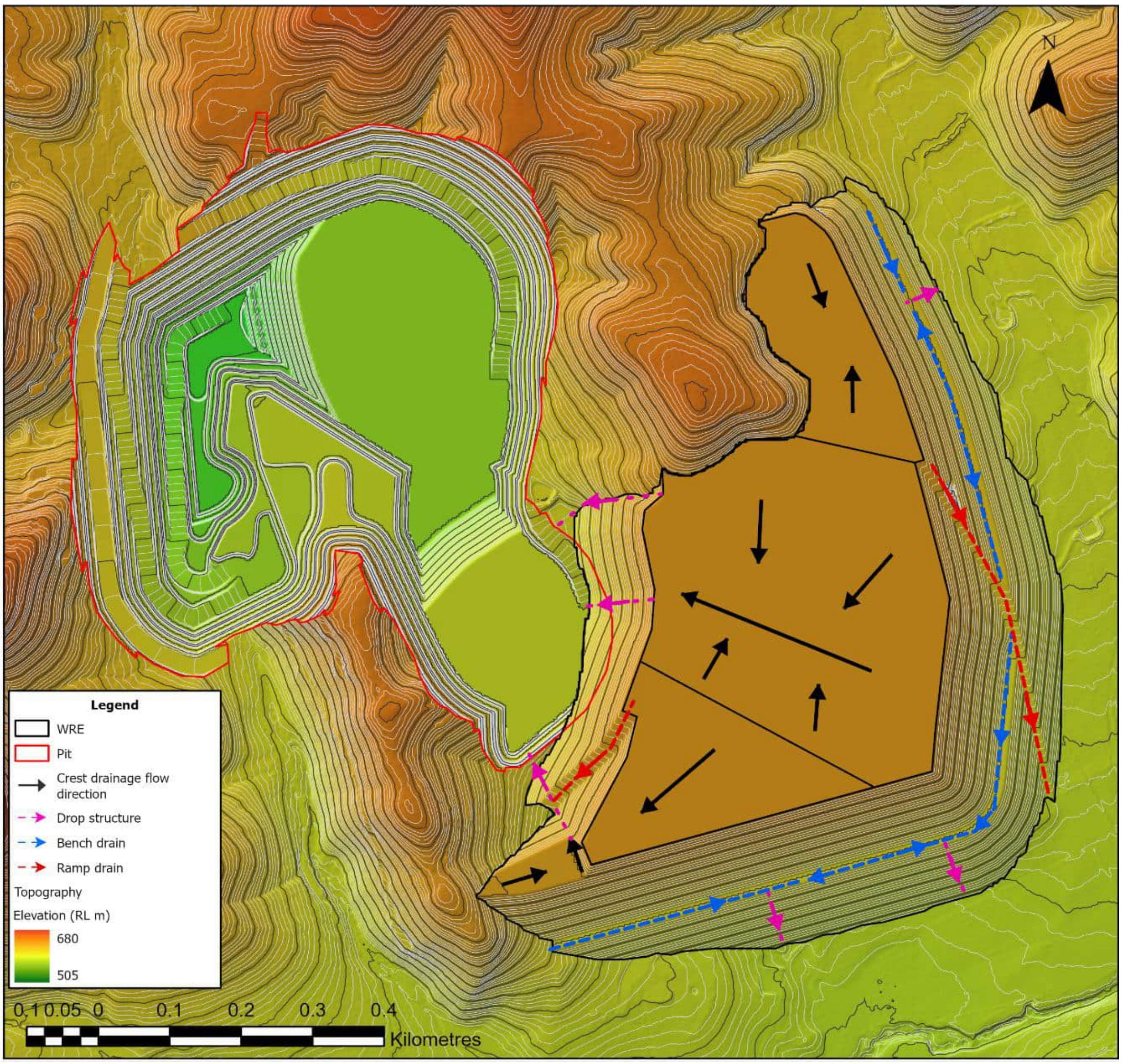
Closure Surface Water Management

A conceptual closure surface water management (Figure 18.6.2) has been developed for the WRE. The management plan includes the following control features:

- Ramp drain.
- Drop structures.
- Bench drains.
- WRE crest shaped to drain to the pit.

Run-off from the upper lift of the dump will be conveyed towards the drop structures and ramp drain via bench drains. The ramp drains and drop structures, where highly concentrated flows are expected during storm events, require competent and durable rock armour to be placed as erosion protection.

Figure 18.6.2 WRE Closure Surface Water Management



18.7 Tailings Storage

SRK developed an AACE Class 3 design for Stage 1 of the FTL for the Project (SRK, 2026e). As discussed in Section 1.1.1, the DFS has been limited to match the Stage 1 design of the FTL, which holds approximately 30 Mt of filtered compacted tailings. A Stage 2 FTL will be required once Stage 1 is filled at the end of the DFS mine schedule. The Stage 2 FTL has only been designed at a pre-feasibility level.

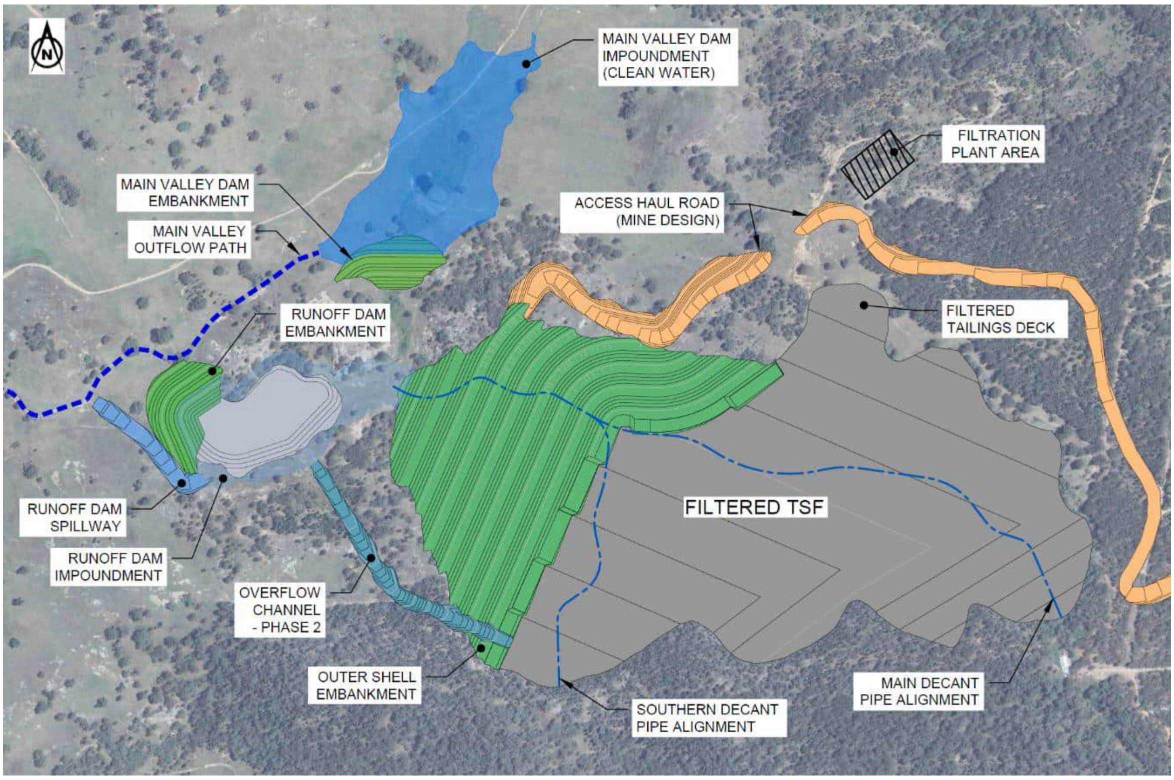
18.7.1 Stage 1 FTL Landform

Filtered (dewatered) tailings will be placed and compacted in a valley fill operation to form a stable FTL. The Stage 1 FTL will be approximately 70 m above the existing ground level at the thickest point. The landform will be located in a steep valley located 2 km east of the open pit and process plant area. The valley is orientated approximately east to west and is approximately 1,200 m long from entry to head.

The tailings filtration plant will be located on a raise to the north of the valley. Filtered tailings, dewatered to be between -2 and +4% of the optimum moisture content, will be hauled to the landform via trucks, before being placed and compacted using conventional earthwork equipment and methods.

A layout plan for the FTL is shown in Figure 18.7.1. Also shown are the nearby or associated surface water management structures and the auxiliary structures that support the operations of the FTL.

Figure 18.7.1 Stage 1 FTL Layout

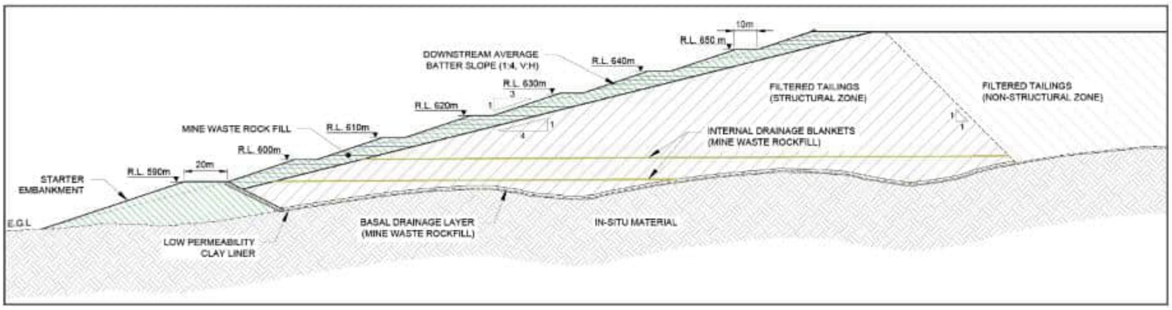


The FTL is designed such that it does not include any flowable materials and is self-supporting. The landform consists of two filtered tailings zones, with a downstream structural zone compacted to a higher density then the upstream non-structural zone.

The outer downstream face of the landform (shell) will consist of an incrementally raised embankment constructed from NAF mine waste rockfill. Note that the compacted filtered tailings form a stable landform and therefore the primary purpose of the 'embankment' is erosion control. The development of the outer shell embankment will comprise of a sizeable stater embankment, followed by subsequent multiple smaller raises, placed using methods analogous to upstream raising with each raise being partial founded on structural zone filtered tailings. As placement within the landform will be continuous so will the outer shell embankment raises with 2 m nominal lifts.

The outer profile of the embankment will consist of a bench and batter arrangement using 10 m wide benches with 10 m high batter between each bench; the slope of the batter will be 1:3 (V:H) resulting in an overall external slope of 1:4 (V:H). An internal profile with a continuous 1:4 (V:H) slope will be offset from the external profile so that a minimum 20 m thickness of NAF mine waste rockfill shell (measured horizontally) will be maintained throughout the embankment section. Figure 18.7.2 shows a cross section through of the outer shell embankment and FTL landform.

Figure 18.7.2 Outer Shell Embankment Cross Section



18.7.2 Underdrainage and Seepage Control

The underdrainage seepage capture system comprises a low permeability clay liner over the base of the valley with a network of subsoil drains overlain and extending up into the minor gullies on the valley sides.

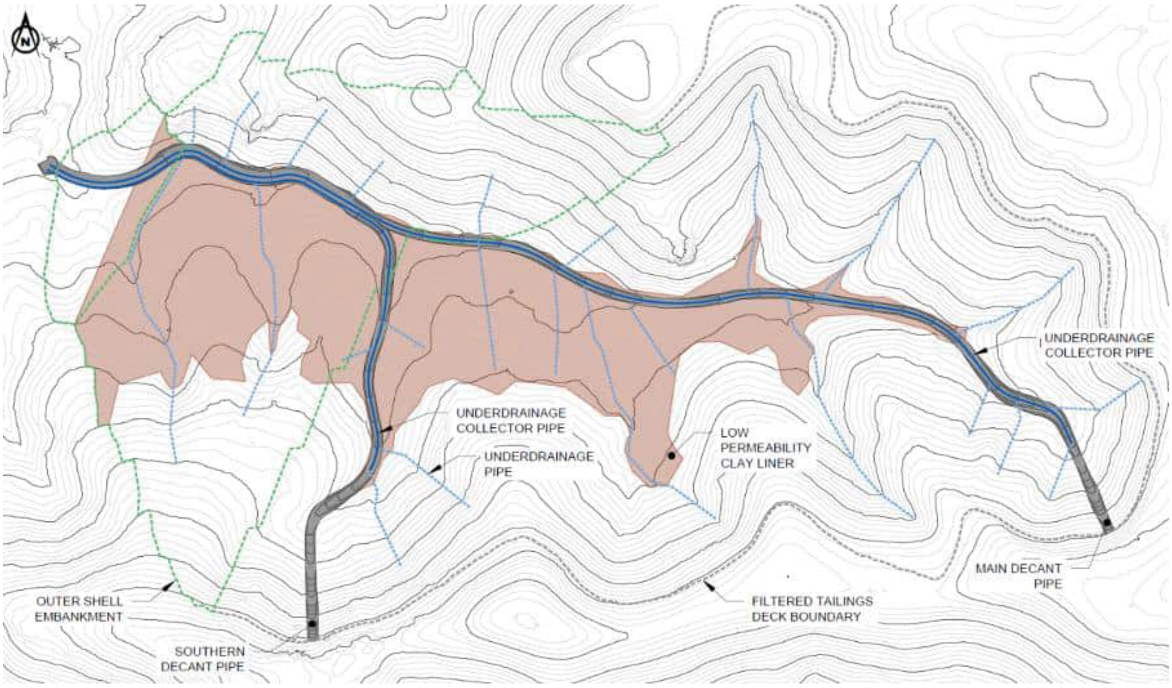
The low permeability clay liner is included to reduce the potential for any rain/runoff infiltration to report to the foundation soils. The liner is to be constructed using clay soils sourced from nearby borrow areas and will cover approximately 24% of the total landform's footprint. The designed extent of the clay liner is limited by constructability with the liner placed in all locations where slopes are 20% or less.

To promote drainage the low permeability layer will be overlain by a 1 m layer of screened (low fines) mine waste rockfill overlain by a graded sand filter to prevent fines migration (and blocking) of the rock. This is referred to as the basal drainage layer.

An underdrainage pipe system is included in the basal drainage layer to intercept seepage that has infiltrated downwards through the tailings and any seepage entering the tailings from the interface with the surrounding landform. The network of pipes will be constructed in trenches through the existing topography and will extend through the basal drainage layer to the outfall.

Figure 18.7.3 shows a schematic of the underdrainage system and the extents of the low permeability seepage liner.

Figure 18.7.3 Stage 1 Underdrainage System and Seepage Liner



Seepage Interception

A series of extraction bores is included as part of the design to monitor groundwater and collect any seepage that potentially bypasses the underdrainage. Seven monitoring / extraction bores are planned, five along the crest of the outer shell starter embankment and two additional monitoring / extraction bores located downstream of the Run-off Dam to intercept any seepage by passing the upstream bores or emanating from the auxiliary structures. Any collected seepage from the seven bores will be pumped to the Run-off Dam.

18.7.3 Tailings Geochemistry

Static geochemical and mineralogical assessment of tailings generated by bench-scale metallurgical testing has been undertaken by Graeme Campbell and Associates in 2020 and by SRK Consulting in 2025. Kinetic geochemical testing has been undertaken on four tailings samples between 2025 and 2026 in the form of small scale (1 kg) laboratory-based kinetic leach columns and customized multi-step leaches with deionized water.

Sulphide minerals in the tailings are predominantly pyrite, with accessory galena, sphalerite, arsenopyrite, pyrrhotite, chalcopyrite and covellite. Carbonate minerals are predominantly ferroan rhodochrosite with compositional variation within the Ca–Mg–Fe–Mn carbonate series.

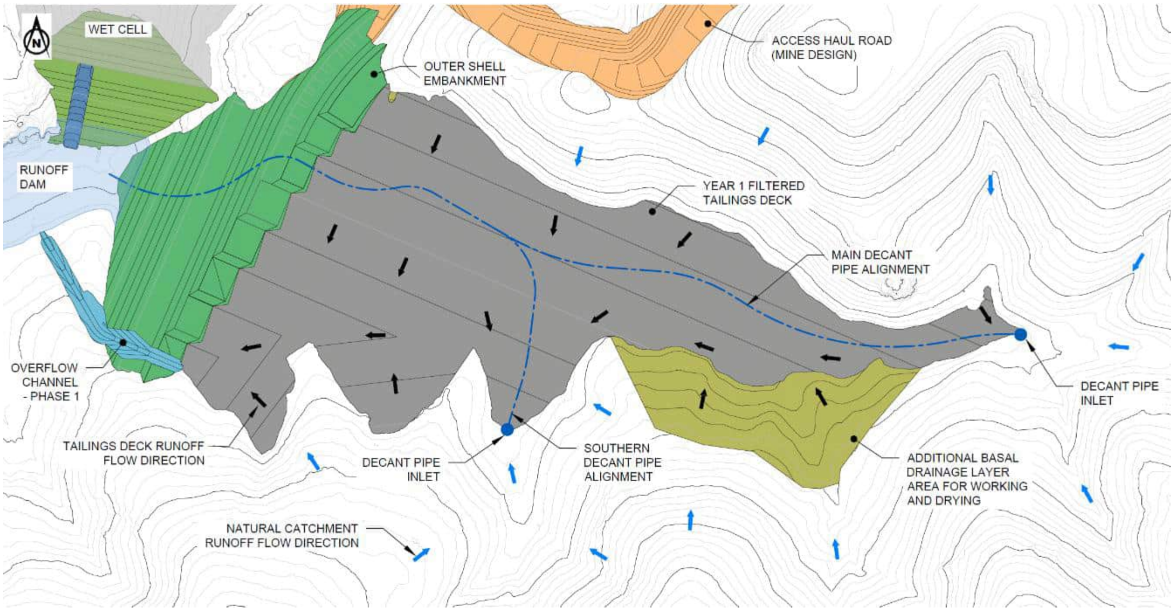
Tailings samples sourced from the Main and West ore zones are classified as NAF according to the AMIRA (2002) classification, whereas tailings samples sourced from the South ore zone are PAF.

The tailings are to be subject to dewatering, compaction and storage in a valley-fill FTL. Oxygen supply to active oxidising zones within the dewatered tailings is expected to be controlled by diffusion due to the fine grain size of the tailings (median grain size 40 µm). Regular placement of lifts will limit oxygen supply to deeper tailings, and at closure, emplacement of a low-permeability cover could further limit oxygen ingress. During operations, seepage will be mostly porewater expelled due to consolidation of the tailings and is expected to be near-neutral in pH. During the post-closure period, seepage volumes from the consolidated tailings are expected to be very low (near zero) due to the low permeability of the tailings and cover placement. The low volumes of long-term post-closure seepage are expected to be saline but to remain near-neutral in pH.

18.7.4 Surface Water Management

The FTL is designed to have no surface water retention capacity. During operations the tailings deck will be shaped to direct all surface water flows to an external overflow channel and two sets of underneath decant pipes located, as shown (at the Year 1 stage of landform development) in Figure 18.7.4. Each set of decant pipes are sized to accommodate a 1:10 annual exceedance probability (AEP) rainfall event. Should a rainfall event greater than 1:10 AEP occur (or the decant pipes become wholly or partially blocked) then excess flows will be managed by the external overflow channel, which has been sized to accommodate a 1:100 AEP rainfall event.

Figure 18.7.4 Stage 1 Surface Water Management Plan Layout



Both the external overflow channel and decant pipes will report to the Run-off Dam, which has been designed to collect contact water from the catchment and is located adjacent to the downstream toe of the FTL. Throughout the life of the facility, all runoff from the southern valley catchment will report to the FTL and be treated as contact water, with the Run-off Dam being sized accordingly.

Decant Pipes

Each branch of the decant pipes will be sized to accommodate the 1:10 AEP design storm for the full FTL catchment – this will provide some flexibility for the surface water management strategy and redundancy in the case of an inlet blocking or becoming inoperable. The decant inlets will be raised incrementally with the tailings deck, and use of interim bunds and channels will help to direct surface flows towards these inlets. The design of the inlets will also include sediment traps to reduce ingress into the pipelines.

To avoid charging of surface water into the underdrainage network, the decant pipes and underdrainage pipe network will remain as two separate systems.

Overflow Channel

The overflow channel will be developed in two discrete phases to align with the landform construction progression and the natural topography of the valley. Both phases are integral to the surface water management strategy and are designed to safely remove surface water should the decant system become inundated during heavy rainfall.

Phase 1 of the overflow channel will be constructed within the downstream batter of the outer shell embankment, running perpendicular to the embankment crest. Phase 1 of the overflow channel will be sized to accommodate the 1:100 AEP design storm for the full FTL catchment.

Phase 2 of the overflow channel will be active approximately 12 years into the operational life of the FTL. The channel will follow a new alignment, located approximately 200 m south of the Phase 1 channel. Reshaping of the tailings deck will be required to divert run-off towards the Phase 2 channel alignment and inlet. The Phase 2 channel will be predominantly constructed across natural topography and make use of a shallow gully down the western face of the southern ridge of the southern valley. The structure is intended to be used in the post-closure period for long-term surface water management of the capped tailings deck. As such the channel has been sized to accommodate the 1:10,000 AEP design storm for the full FTL catchment.

18.7.5 Geotechnical Stability

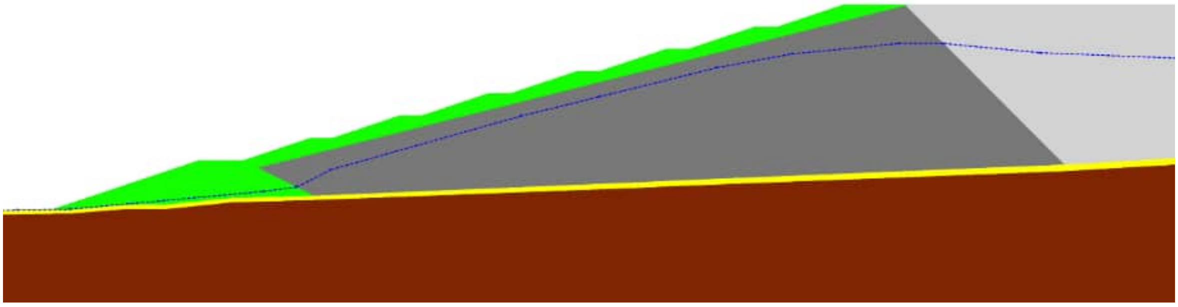
The scenarios analysed as part of the geotechnical stability assessment all included conservative material parameters and were undertaken to investigate the effect of the drainage on the factor of safety (FoS), a key control for excess pore pressure.

Modelled Scenarios

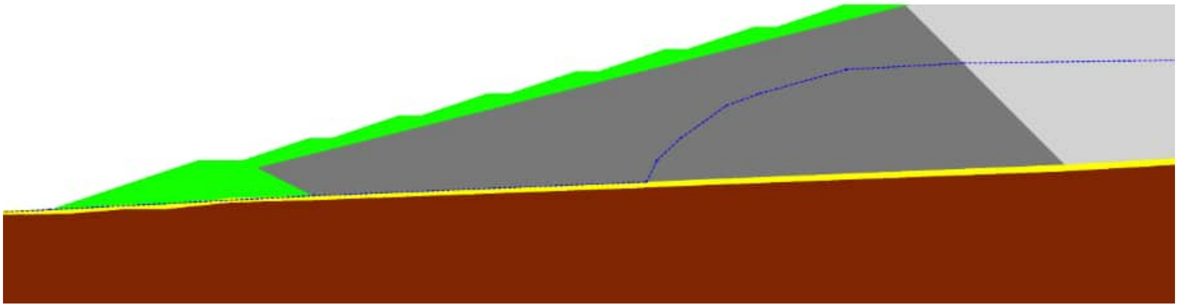
A total of four scenarios were modelled, with each model run adopting a phreatic level determined from a corresponding PLAXIS 2D model developed as part of the numerical modelling analysis. The modelled scenarios are shown in Figure 18.7.5.

Figure 18.7.5 Modelled Scenarios

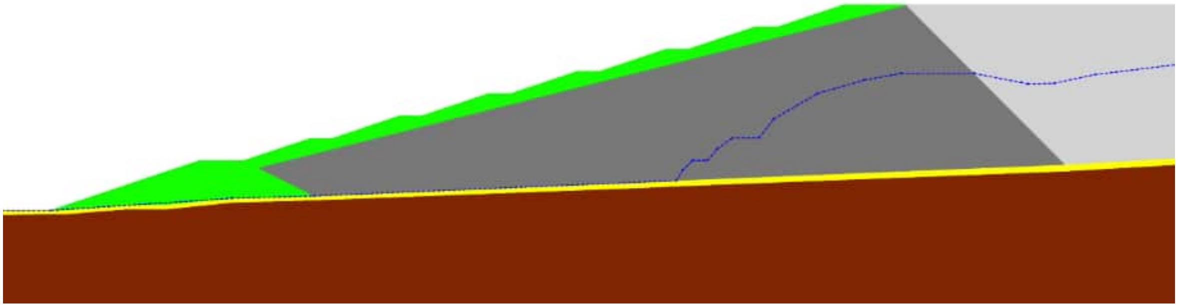
1. FTL with no drainage



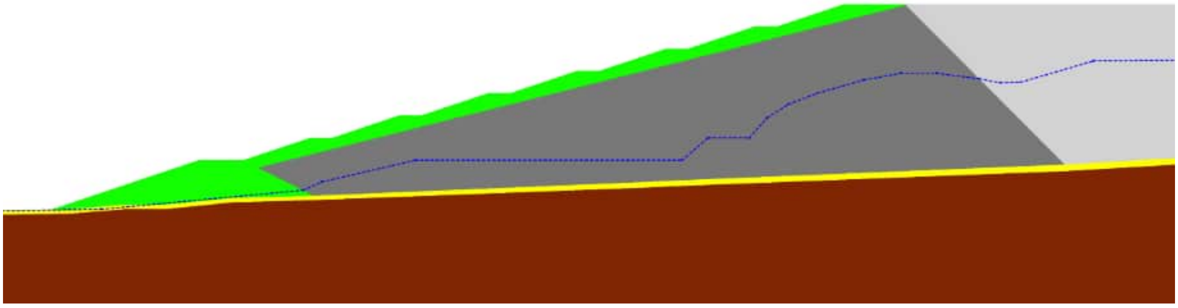
2. FTL with underdrainage



3. FTL with mid-height blanket drains and underdrainage



4. FTL with mid-height blanket drains and blocked underdrainage



Slope Stability Results

A summary of the slope stability results is presented in Table 18.7.3.

Table 18.7.1 Stability Results

ID	Description	Long-Term Drained[1]	Short-Term Undrained[2]	Static Liquefaction[3]
1	No drainage	1.59	1.23	0.48
2	Underdrainage	2.04	1.55	1.08
3	Underdrainage + blanket drains	2.05	1.55	1.22
4	Blocked underdrainage + blanket drains	2.04	1.56	0.67

Notes:

1. **Long-term drained analysis target FoS: 1.5.**
2. **Short-term undrained analysis target FoS: 1.5 (potential loss of containment).**
3. **Static liquefaction analysis target FoS: 1.0 - 1.2 (to be related to confidence in selection of residual shear strength).**

The FoS results where the underdrainage is included are acceptable for all cases and these are marginally improved where the blanket drains are included. Where the underdrainage is modelled to be completely blocked there is a reduction in the FoS for the static liquefaction case. However, it is noted that a situation where the underdrainage becomes fully blocked is considered to be effectively implausible (there may be a reduction in permeability but not total blockage of the whole layer). The number and location of the blanket drains will be optimised as part of the final engineering design.

The results from the analyses show that the FoS values against instability for the proposed FTL are demonstrated to be acceptable even for a range of cases that are outside the design intent and expected outcomes during operations.

18.7.6 Instrumentation and Monitoring

A monitoring network consisting of 89 vibrating wire piezometers (VWPs), grouped in horizontal arrays and arranged across five monitoring sections will be installed in the FTL landform. The network has been designed to collect piezometric data from across the FTL landform to evaluate phreatic levels and pore water pressure conditions through the structural tailings zone, internal drainage blankets and basal drainage layers.

VWPs installed within the structural tailings zone are used to monitor the accumulation and dissipation of pore water pressure conditions during placement of the filtered tailings, whereas VWPs installed within the internal drainage blanket and basal drainage layer will be used to assess drainage functionality.

The layout of the monitoring sections is shown in Figure 18.7.6 and an example monitoring section is shown in Figure 18.7.7.

Figure 18.7.6 Stage 1 Monitoring Sections Layout

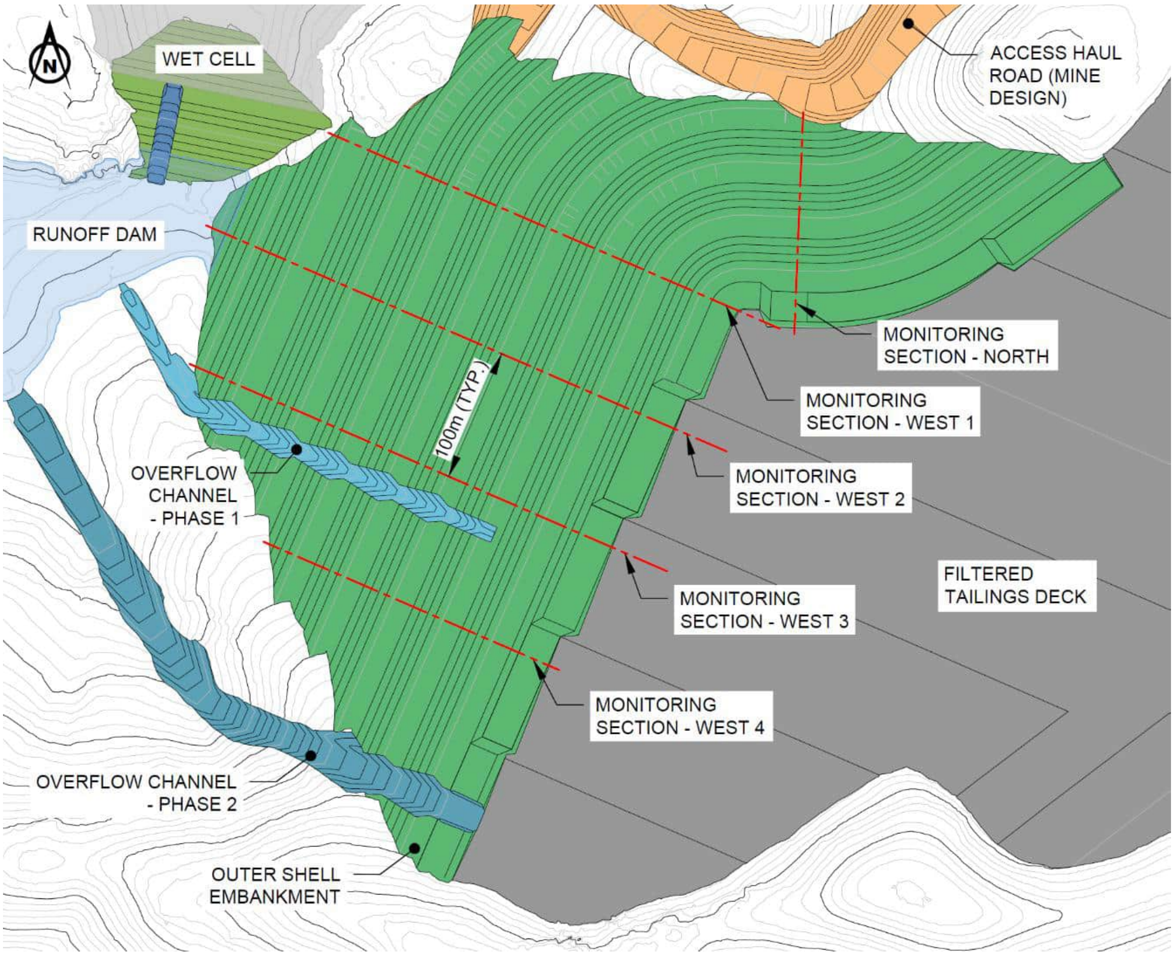
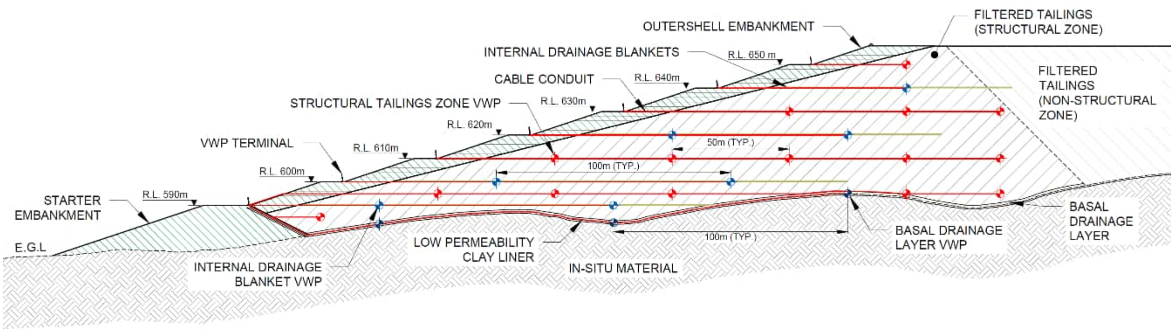


Figure 18.7.7 Stage 1 Monitoring Section



18.7.7 Closure

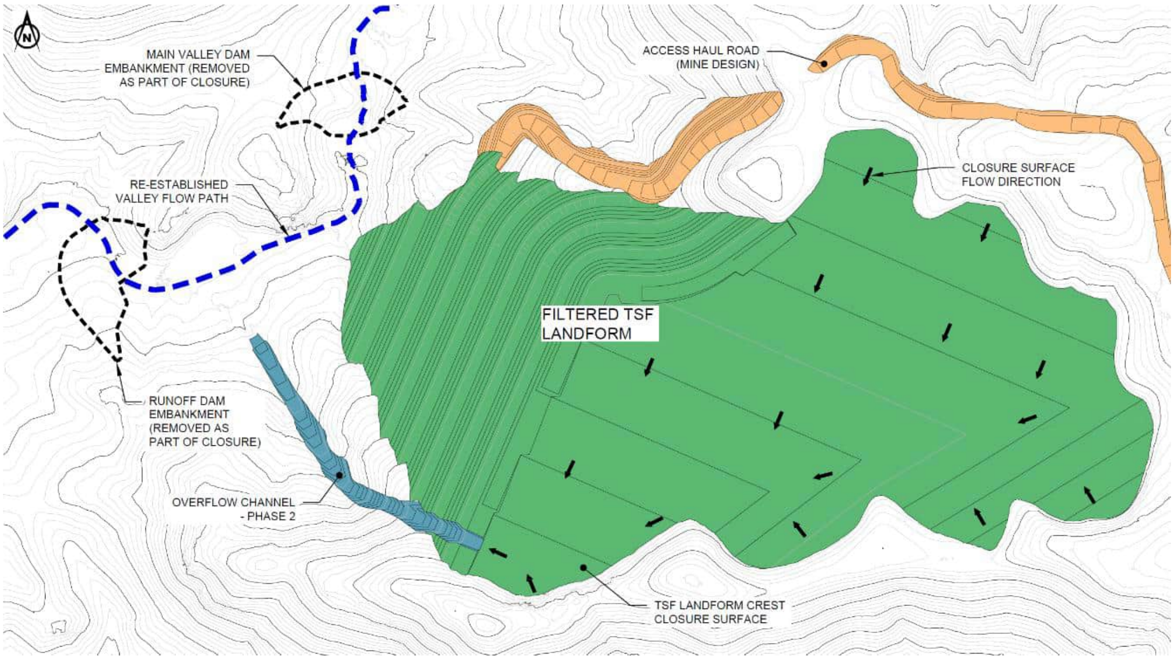
The closure cover system is the same as developed for the WRE, with the layers detailed below (from top to bottom):

- 30 cm thick topsoil.
- 30 cm thick subsoil.

- 40 cm thick Type 1 waste rock (gravels and cobbles).
- 40 cm thick Type 1 waste rock (boulders).
- 2 cm thick geosynthetic clay liner (GCL).

The closure surface water management plan is shown in Figure 18.7.8, with the management plan resembling the operations plan in the final years of the landform development. Modifications for closure include the decommissioning of the decant pipes and as such the crest surface of the landform will need to be reshaped so that all crest areas drain towards the Phase 2 overflow channel.

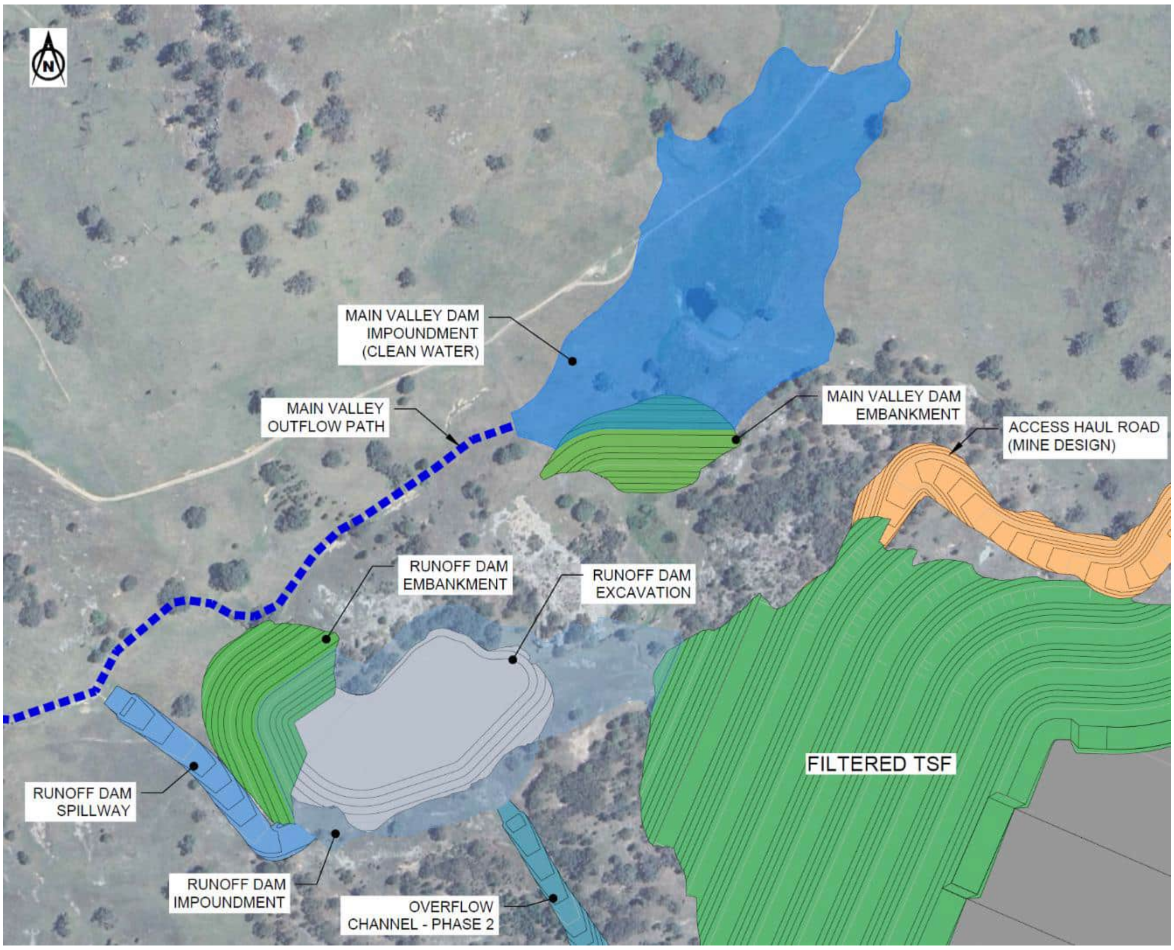
Figure 18.7.8 Stage 1 Closure Surface Water Management Plan



18.7.8 Auxiliary Structures

To support the operations of the FTL, there will be two functionally associated dam structures located to the west of the FTL landform. A layout of the auxiliary structures is shown in Figure 18.7.9 and the structures are described in greater detail in the proceeding sections.

Figure 18.7.9 Stage 1 Auxiliary Structures Layout Plan



Preliminary Consequence Category Assessment

The preliminary consequence category assessment (CCA) completed for the previous slurry cell design (ATC Williams, 2020; REC, 2024), was reappraised in the context of the auxiliary structures. The review determined that the population at risk (PAR) of >1 to 10 was still applicable to the auxiliary structures. Non-itinerant loss of life is not expected; however, a dam break analysis would need to be undertaken to assess the risk to site personnel and itinerant persons.

Under Dams Safety New South Wales (DSNSW) regulations, tailings facilities that store acid contents are prescribed a minimum 'Severity of Damage and Loss' level of 'Major'. As some of the tailings are expected to be PAF, the Runoff Dam is considered to meet the threshold for a severity damage and loss level of Major under DSNSW regulations. The Main Valley Dam will impound clean water runoff from a natural catchment, however if the embankment structure was to fail it would be expected to significantly impact the Runoff Dam, which is downstream of it, potentially resulting in a cascading failure. As such the Severity of Damage and Loss level is considered to be 'Major' under DSNSW regulations.

Australian National Committee on Large Dams (ANCOLD) guidelines are not as prescriptive in regard to the impoundment contents, and therefore it is likely a minor to medium classification for damage severity would be applicable, however the DSNSW guidelines are considered to supersede those of the ANCOLD.

Similar to the DSNSW regulations, the Global Industry Standard on Tailings Management (GISTM) guidelines for CCA include consideration of the toxicity of the process water. Given the assumed presence of PAF tailings material and risk of cascading failure, it has been conservatively assumed that the two auxiliary structures would be assessed as 'High' consequence category under the GISTM guidelines.

The outcomes of the preliminary CCA for the auxiliary structures are summarised in Table 18.7.4.

Table 18.7.2 Preliminary CCA outcomes

Impoundment	PAR	Severity of Damage and Loss	Consequence Category		
			Dams Safety NSW (2022)	ANCOLD (2012; 2019)[1]	GISTM (2020)[2]
Run-off Dam	>1 to 10	Major	High C		High
Main Valley Dam	>1 to 10	Major	High C		High

Notes:

1. **DSNSW CCA is considered to supersede the ANCOLD CCA.**
2. **GISTM guidelines do not define a Severity of Damage and Loss rating level; the CCA is defined as the highest consequence for any given 'Incremental Losses' category.**

Run-Off Dam

The Run-off Dam will be the collection point for all surface runoff and any collected seepage from the FTL and is intended to prevent any mine affected water from entering the environment.

The dam will comprise an earthen embankment across the existing main valley outflow channel, an excavated basin and a spillway. The entire impoundment will be lined with an impermeable membrane to mitigate potential loss of containments via seepage through the embankment and foundation layers. The impounded area will be approximately 5.9 ha and will be bounded to the west by the Run-off Dam embankment, to the east by the FTL landform and by natural topography along the northern and southern perimeters.

As the runoff from the FTL is expected to be sediment laden to some degree, allowances for sediment and settling accumulation zones have been included in the sizing of the dam.

An overflow spillway will be located in the southwestern corner of the Runoff Dam in natural ground adjacent to the embankment. The spillway will report to the existing flow path and discharge to the west. As per the requirements of a 'High C' category structure, the spillway has been designed to safely pass a 1:100,000 AEP design storm and include an appropriate allowance for wind wave run-up to ensure that overtopping of the embankment crest does not occur during the design events. As part of the capacity design for the runoff dam, an appropriately sized extreme storm storage buffer has been included.

Main Valley Dam

The Main Valley Dam will be formed by construction of an earthen embankment located approximately 100 m north of the toe of the FTL landform, upstream of the Run-off Dam.

The Main Valley Dam has a dual function to both capture surface water flows from the main valley for use by the mining operations and to divert catchment flows away from the toe of the FTL landform. The proposed structure has a reporting catchment of 211 ha and a storage capacity of 267 ML.

When the Main Valley Dam is at full storage capacity any excess inflow will be diverted to the downstream environment through an overflow spillway located at a natural saddle in the rise to the west of the main valley. The downstream flow path extends around the Run-off Dam before merging with the existing natural flow path downstream.

The crest of the Main Valley Dam embankment was set to an elevation such that the structure would not be over topped during a design storm event of 1:100,000 AEP, as per the requirements of a 'High C' category structure. As part of the freeboard design an appropriate allowance for wind wave run-up was also included.

The upstream batter of the embankment will be lined with an impervious liner, tied into an excavated seepage cut-off wall extending down to relatively impermeable bedrock. The cut-off wall is intended to remove the potential seepage gradient between the Main Valley Dam and the Run-off Dam.

Closure

At closure the impermeable liner in both of the auxiliary structures will be removed and the containment embankments removed so the existing natural outflow path can be re-established.

18.8 Surface Water Flood Management

18.8.1 Introduction and Approach

SRK conducted a surface water impact assessment for the Project (SRK, 2026f). The assessment included flood modelling and development of a surface water management plan (SWMP) covering the following areas:

- Open pit.
- Haul road and mine access road.
- Filtered tailings landform.
- Waste rock emplacement (WRE).
- Sound barrier.
- Process plant pad.
- Seepage and run-off ponds.
- Runoff harvesting dams.

18.8.2 Summary of Major Creeks and Local Drainage

All run-off from the project area ultimately discharges to Lawsons Creek, with the downstream catchment area estimated at approximately 273 km². Within the project boundary, local drainages include Price Creek, Blackmans Gully, an unnamed southern drainage line, and Walkers Creek. Run-off from the eastern areas (via Price Creek, Blackmans Gully, and the unnamed southern drainage) is collected by Hawkins Creek, which then conveys flows to Lawsons Creek at their confluence.

Summary of the Lawsons Creek and the local catchment drainages are shown below in Figure 18.8.1 and Figure 18.8.2.

Figure 18.8.1 Summary of Lawsons Creek Catchment and Drainages

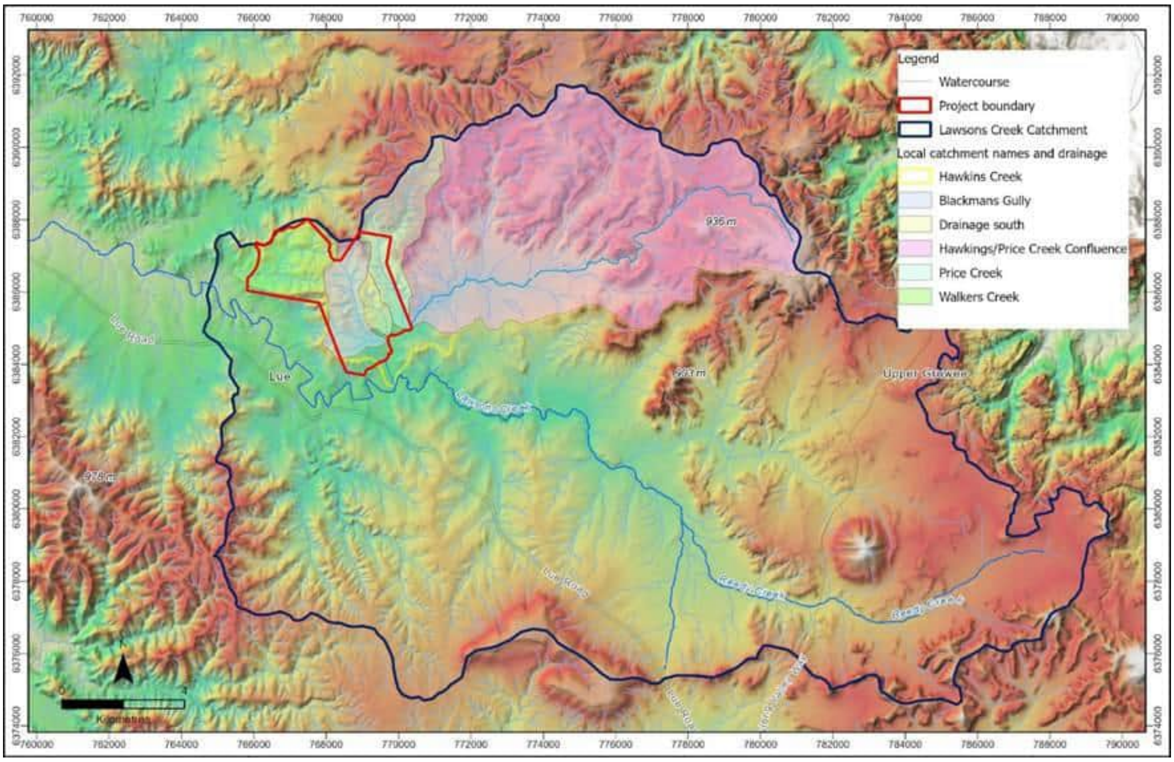
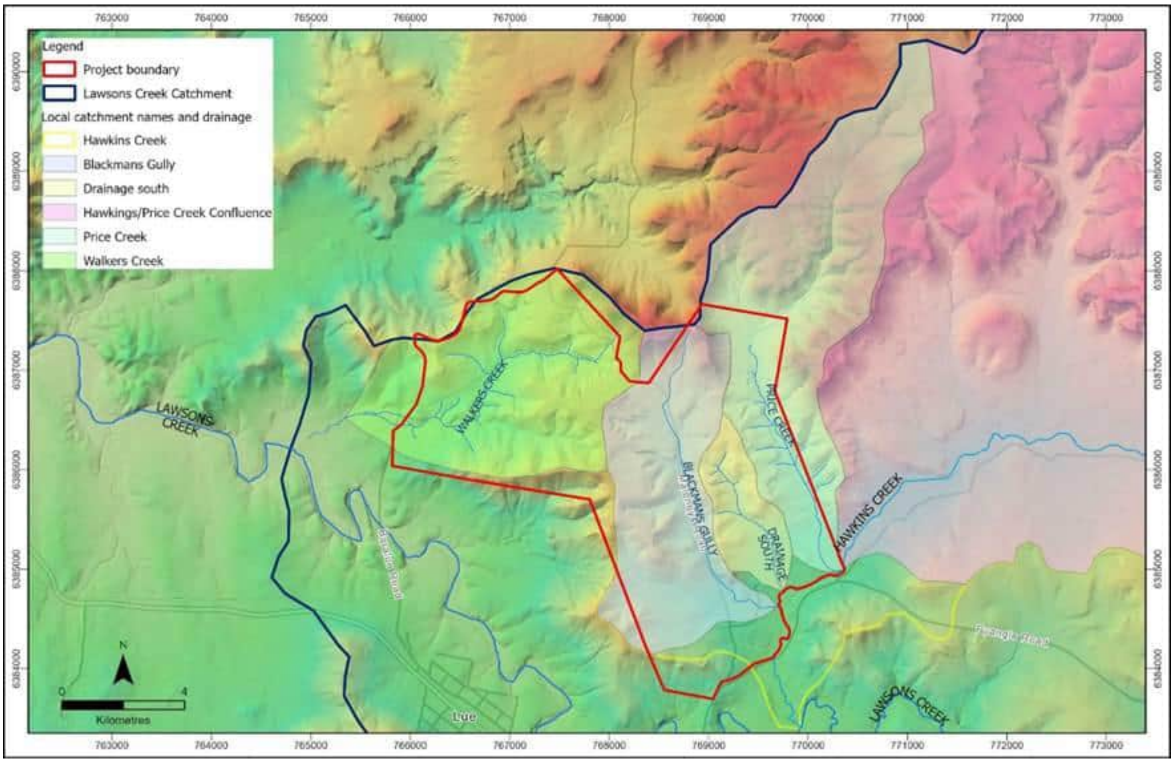


Figure 18.8.2 Summary of Project Site Catchment and Drainage



18.8.3 Hydrological and Hydraulic Modelling Summary

Hydrological and hydraulic modelling was undertaken to assess surface water risks, using the TUFLOW, RORB and HydroCAD modelling software which factored in climate adjustment as per the Shared Socioeconomic Pathways SSP3-7.0 scenario (a high-emissions climate scenario projecting approximately 3.5–4.0°C of global warming by 2100). Time horizons considered for SSP3-7.0 include 2050 (operations) and 2100 (closure).

18.8.4 Surface Water Risks and Proposed Management Measures

Summary of all surface water risks identified are summarised in the Table 18.8.1 and Figure 18.8.6.

Table 18.8.1 Summary of Surface Water Flood Risks Identified (1:100 AEP) – Operations

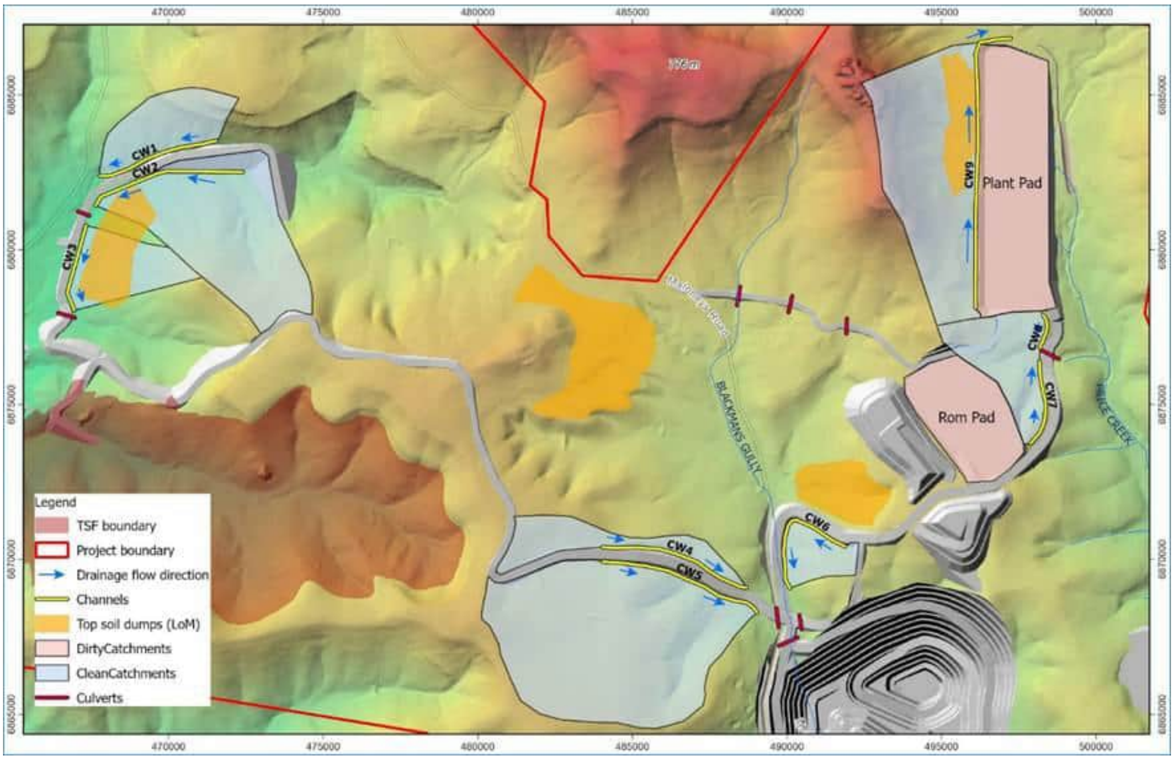
Location	Reference ID on Map	Risk	Potential Impact
Plant Pad	A	Flood inundation against the Plant Pad	Damage to Plant Pad, infrastructure
Haul / Mine access road	B	Overflows unto road	Increased downtime due to flooding of roads and corresponding repairs.
Open Pit	C	Flooding of Pit	Mining downtime due to flooded pit, benches and access ramps
WRE	D	Flood inundation of WRE at southwestern toe	Potential erosion of the WRE toe from high velocity flows
Sound Barrier	E	Extended ponding against upstream end of Sound Barrier	Potential slope failure due to piping
WRE Seepage Pond embankment	F	Flood inundation of WRE Seepage Pond embankment	Potential erosion of the WRE Seepage Pond Embankment

Management measures for risks highlighted in Table 18.8.1 regarding surface water management (clean and dirty water separation) and flood risk would include:

- A diversion along Price Creek to mitigate flood inundation along the eastern boundary of the plant pad.
- 1:20 AEP channels along roads to convey flows to the clean water system.
- A 1:100 AEP channel along the western Plant Pad to direct run-off to Price Creek.
- 1:10 AEP culverts along drainage intersections with mine access and haul roads to ensure overtopping of road surfaces are minimised.
- Erosion protection at culvert and diversion and channel outlets as per the Drainage Design Guide (Austroads, 2023).
- Light armouring of the WRE seepage pond and minor bunding (up to 1 m high) along the southern and eastern WRE boundary flood water contact.

Clean and dirty water separation strategy and channels are shown in Figure 18.8.9.

Figure 18.8.3 Summary of Clean and Dirty Water Separation



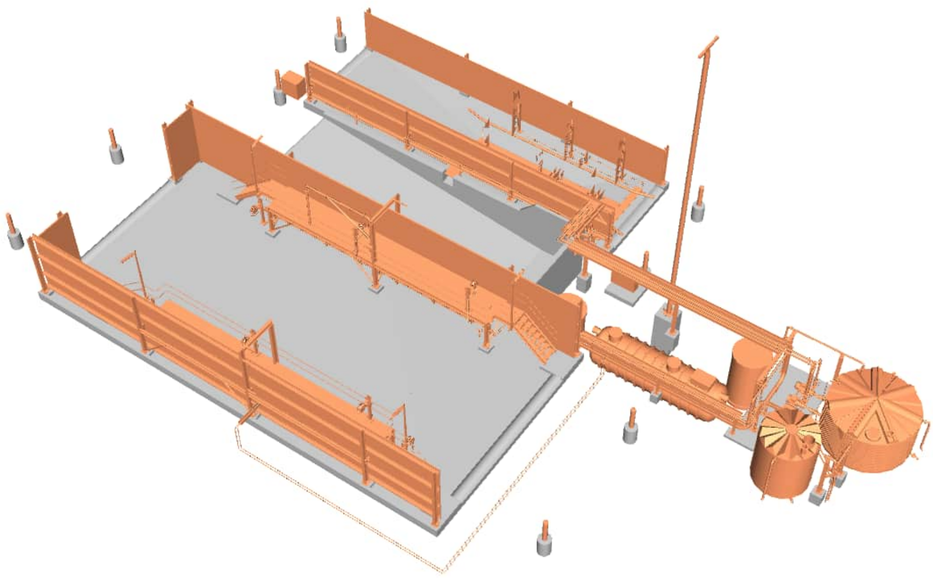
18.9 General Site Infrastructure

As shown in Figure 18.1.1, a single pad contains the process, mining and administration facilitates. A number of infrastructure components are shared between all three functions including:

- Parking lot.
- Office complex.
- Training rooms.
- First aid / medical centre.
- Warehouse and laydown yards.
- HV and LV fuel storage and dispensing.
- HV and LV washdown pad.
- Laboratory.

Figure 18.9.1 shows the shared HV / LV washdown station. A bund (not shown) separates HV and LV access roads. In this example, the combining of the HV and LV washdown results in reduced Capex and Opex as only a single sump and water treatment facility is required.

Figure 18.9.1 HV / LV Washdown Station



18.10 Process Plant Infrastructure

18.10.1 Geotechnical

A preliminary geotechnical investigation has been conducted in the process, administration and mining area (Barnson 2024). Additional geotech will be conducted in the locations of major foundations prior to final engineering designs.

Where possible, major equipment, and specifically the mills have been located in cut sections of the process plant pad, rather than fill. Based on the available geotech information, shallow foundations have been allowed for the majority of equipment and building foundations. Piling has been allowed under the secondary crushing and screening building and the fine ore bin which are located on fill.

18.10.2 Process Plant Roads and Carpark

Roads internal to the process plant area and tailings filtration plant area are provided for general maintenance accessibility as well as access between the plant office areas and reagent storage areas. These roads will generally be 8 m wide unsealed roads and will be constructed flush with bulk earthworks pads to ensure that storm water sheet flow is achieved across the site, thereby minimising the need for deep surface drains and culvert crossings within the plant area.

An unsealed hardstand carpark with capacity for approximately 144 cars is located adjacent to the process plant office building.

18.10.3 Ponds

The process water pond is a high density polyethylene (HDPE)lined pond with a stilling section to remove silt located within the main process plant pad.

A HDPE lined combined stormwater / event pond is located between the main process plant and the primary crusher. This pond will capture any stormwater run-off from the main process plant and mine services area from where it can be returned as process water. This pond also serves as an emergency capture for any spillages that overtop concrete bunds within the process plant. There is a similar HDPE lined stormwater / event pond at the tailings filtration plant.

There is also a small HDPE lined catchment pond adjacent to a low point in the tailings slurry line between the process plant and the tailings filtration plant. In the rare event where the pumping of tailings slurry stops and the line is unable to be fully flushed with water, there is the ability to open a manual valve at the low point in the line and drain the slurry to this pond.

18.10.4 Non-Process Buildings

Non-process buildings within the process plant are as listed in Table 18.10.1.

Table 18.10.1 Process Plant Non-Process Buildings

Description	Comments	Style	Area m^2
Process Plant			
Process Plant Offices	Includes training room, conference room. First aid facility and security	Prefabricated	900
Change Room / Shower Block		Prefabricated	400
Plant Crib Room		Prefabricated	80
Plant Ablutions Block		Prefabricated	40
Mill Control Room	Including kitchenette and ablutions	Prefabricated	40
Reagents Store		Steel framed shed	300
Plant Workshop / Warehouse	With fenced hardstand area	Steel framed shed	1,200
Plant Workshop / Warehouse Offices	Within / adjacent to workshop / warehouse	Prefabricated	80
Laboratory		Prefabricated	800
Tailings Filtration Plant			
Filter Control Room	Including kitchenette and ablutions	Prefabricated	40
Reagents Store		Steel framed shed	160

18.11 Mine Support Infrastructure

The mining area will contain dedicated maintenance, crib and ablution facilities.

18.12 Sewage Treatment

Effluent from all water fixtures in the process / admin / mining area will drain to gravity sewerage systems. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a vendor package sewage treatment plant.

18.13 Accommodation

18.13.1 Construction Accommodation

A total workforce of up to 320 personnel are estimated during site construction, although some of these personnel will be engaged in off-site activities such as procurement, engineering, drafting and administration. The instantaneous workforce numbers (both on and off-site) will vary depending on the actual work being performed at any point.

The site establishment and construction workforce will predominately comprise of persons engaged under one the following employment arrangements:

- Employed by the contractor appointed to construct the processing plant engaged from local, state and inter-state locations.
- Employed by other contractors or service providers undertaking specific tasks such as clearing, bulk earthworks, waste rock and tailings foundation preparations, road construction and electrical reticulation engaged primarily from local and state locations.
- Employed directly by Bowdens engaged primarily from local areas such as Kandos, Rylstone and Mudgee.

Non-local personnel may be accommodated in a variety of facilities including company owned housing, commercial temporary facilities, and rental agreements.

A temporary workers accommodation (TWA) facility will be constructed to accommodate the majority of the non-local personnel engage in project construction. The TWA will be constructed in close proximity to the Bowdens on company owned land and will be removed and rehabilitated following commissioning. The TWA will be a 200-bed facility that will include parking, messing, laundry, entertainment, recreation, power supply, water supply and sewage treatment.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The final product will be a HPM concentrate, which will be transported to a smelter for further processing. Payable metals in BSP HPM concentrate are Ag, Au, Zn, and Pb. Table 19.1.1 lists the average life of mine concentrate metal grades for the DFS and LOM mine plans.

Table 19.1.1 Concentrate Grades

HPM Average Grades	Unit	DFS	LOM
Ag Grade in HPM Concentrate	g/dmt	4,494	3,692
Au Grade in HPM Concentrate	g/dmt	0.62	1.08
Zn Grade in HPM Concentrate	%	24.46%	24.18%
Pb Grade in HPM Concentrate	%	16.25%	15.90%

There are no offtake agreements in place and no formal market studies have been undertaken. However, the Company monitors market conditions and has engaged the services of Calgacus AG (Switzerland) to provide advice on concentrate marketing.

There is a high market demand for HPM concentrates, especially those with >2,500 g/t Ag content. In response, some imperial smelters and Pb / Zn smelters have added circuits to increase the recovery of byproduct metals.

19.2 Handling, Transport and Shipping

As there are no agreements in place, all options remain open to the marketing of the HPM concentrates. However, for the purposes of this study, it has been assumed that the concentrates will be shipped to China or Korea for smelting. Concentrates will be direct loaded from beneath the concentrate filter press into half-height sealed concentrate containers. The containers will then be road transported to Port Botany for shipment to overseas.

Transport and shipping costs incorporated in the financial model are provided in Table 22.1.2. Australian land-based concentrate handling and transport costs were provided by Qube Holdings Ltd. Shipping costs have been benchmarked against other concentrate shipments from Australia to China and Korea.

Table 19.2.1 Transport and Shipping Costs

Transport and Shipping Costs	A$ /wmt	US$ /wmt
Container Hire and Loading	16.98	11.89
Inland Freight to Port Botany	84.29	59.00
Freight Insurance	2.03	1.42
Sea Freight Cost	78.57	55.00
Sampling and Assaying	1.42	0.99
Transport and Shipping Costs	**183.29**	**128.31**

19.3 Commercial Terms

Indicative terms received from various parties were relatively consistent. Sensitivity analyses indicate that project economics are not sensitive to market related costs.

A summary of the commercial terms applied in the financial model are listed in Table 19.3.1.

Table 19.3.1 Commercial Terms

Description	Unit	Value
HPM Treatment Charge	US$ /dmt	60.00
Ag Refinement Charge	US$ /oz	0.45
Au Refinement Charge	US$ /oz	15.00
As Penalty Threshold	>	0.20%
As Penalty Increments, After Threshold	each	0.10%
As Penalty, Per Increment	US$ /dmt	1.50
Ag Payability	%	95%
Deduction (if Ag grade <2,500 g/t)	g/t	50.0
Au Payability	%	95%
Minimum Deduction	g/t	1.0
Pb Payability	%	95%
Minimum Deduction	%	3%
Zn Payability, con grade <10%	%	10%
Zn Payability, con grade >10%, <15%	%	20%
Zn Payability, con grade >15%, <20%	%	25%
Zn Payability, con grade >20%, <30%	%	30%
Zn Payability, con grade >30%	%	35%

19.4 Contracts

There are no concentrate marketing contracts in place yet.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Bowdens is currently pursuing redetermination of its Development Application (DA) for the BSP through the NSW DPHI planning approvals system. To achieve this, Bowdens recently provided DPHI with updated assessment information for their consideration. It is anticipated that DPHI will subsequently provide an update of their assessment report to the NSW IPC for their consideration prior to making a redetermination of the DA for the Project.

20.2 Environmental Setting

The Project is located near the village of Lue, which lies between the regional towns of Mudgee and Rylstone in the Central Tablelands Region of NSW, Australia. The Project is located in an established mining district with outstanding logistics, local suppliers and operators available to support future mine development. The Project is located within a rural area principally used for cattle and sheep grazing, with some cropping and small production of olives and grapes.

The Project is located within the 507 km^2 Lawsons Creek catchment, in the eastern headwaters of the Macquarie River basin. The major regional drainage feature in the area surrounding the Project is Lawsons Creek, a tributary of the Cudgegong River. Lawsons Creek flows in a north westerly direction and flows into the Cudgegong River at Putta Bucca, immediately north of Mudgee. Hawkins Creek, a tributary of Lawsons Creek with a catchment area of 61 km^2 flows in a south-westerly direction parallel to, but set back from, the southeastern boundary of the Project. The Project is traversed by a number of tributaries of Hawkins and Lawsons Creeks.

The Project is located on the foothills of the Great Dividing Range with forested high ridges, rocky escarpments and deep gullies widening to the south into open low relief cleared grazing land with a mosaic of paddock trees.

20.3 Environmental Studies

Numerous environmental and social baseline studies, impact assessments and monitoring programs associated with the Project have been undertaken alongside exploration activities since 1996. Accordingly, the environmental characteristics of the Project area are well understood.

Environmental and social aspects that have been subject of monitoring, survey, study and assessment include:

- Surface and ground water.
- Meteorological.
- Biodiversity.
- Stygofauna.
- Aboriginal cultural heritage.
- Historic heritage.

- Soils and land capability.
- Air quality.
- Noise and vibration.
- Greenhouse gas.
- Human health.
- Traffic and transport.
- Social impact.
- Economic impact.
- Visibility, lighting and sky glow.
- Agricultural impact.

The environmental assessments been subject to reviews by numerous regulatory authorities and third-party independent expert reviews.

20.4 Environmental Management Approach

As part of the government approvals process, a strategic framework will be documented in an environmental management strategy (EMS) that guides environmental management during the planning, development and operation of the Project. The EMS will account for the statutory approvals that apply to the Project and set out the roles, responsibilities, authorities and accountabilities of key project personnel involved in the satisfaction of statutory requirements.

The EMS will set out procedures that will be implemented on the Project in order to protect the environment from harm and also provide information to the local community and other stakeholders including relevant government bodies to keep them informed about the performance of the project. Procedures will cover complaints response, dispute resolution, incident and noncompliance reporting and management and emergency response.

The EMS will detail all of the strategies, plans and programs required to deliver the Project and will also illustrate the locations of all sites where monitoring is required to satisfy statutory requirements of the Project.

20.5 Social Setting and Community Requirements

20.5.1 Social Setting

The Project is located approximately 26 km east of Mudgee and approximately 2 km northeast of Lue. The Project is proximal to the State suburbs of Lue, Pyangle, Bara, Havilah and Monivae, all of which fall within the MWRC LGA. The MWRC LGA is located in the central-west of NSW with the township of Mudgee being the major commercial and urban centre. The MWRC LGA also includes a number of small villages and towns including Lue, Windeyer, Gulgong, Ulan, Wollar, Bylong, Rylstone and Kandos.

The MWRC LGA has strong and diverse industries including mining, tourism and agriculture and a Council that actively promotes support for projects that are intended to create new jobs in the region and help to build a diverse and multi-skilled workforce as well as the expansion of essential infrastructure and services to match business and industry. The business services sector is strongly represented in the region. Barriers to economic advancement in the region include labour force competition due to mining activity and an increasing retirement age population that reduces the skilled employee base. Notwithstanding, at a more localised level, there are high levels of unemployment in Kandos, Gulgong and Rylstone, compared to MWRC LGA and NSW.

Some elements of infrastructure and services and other characteristics that have shaped the regions social profile from a physical perspective includes:

- Ongoing investment in road and stormwater infrastructure.
- The regions historic character.
- Youth related infrastructure development.
- Foot path and cycleway infrastructure.
- Medical facilities.
- Absence of commercial flights.

The close-knit nature of the mid-western region can also be characterised as being made up of individuals willing to volunteer, who provide support for the arts and cultural development with programs supported by the local council to improve housing and the tourism industry including support for a number of art galleries. Conversely, risks to the social standing of the region include isolation, substance abuse, family breakdown, lack of youth awareness of existing support services and facilities, under-utilisation of existing facilities, lack of regular recreational events especially in outlying areas, and youth access to existing recreational / cultural activities. There is a shortage of short and long-term accommodation (that is perceived to be connected to mining industry growth) and temporary accommodation options often segregate people from the local community.

20.5.2 Community Engagement

Bowdens has undertaken substantial engagement with a broad range of stakeholders over a long period of time in a variety of forums and formats regarding the Project. Keeping the community and regulators properly informed about the Project along with the implementation of the Projects Community Investment Program (CIP) has led to an advanced understanding of and keen interest in the Project from community members. The Project also benefits from advanced brand recognition as a result of the scale of the CIP. Community engagement initiatives implemented by Bowdens include:

- Operation of a Community Consultation Committee.
- Establishment of Registered Aboriginal Party consultation group.
- Hosting information days on site.
- Hosting town hall style meetings.
- Face to face meetings with local community members.
- Operating stalls at local agricultural shows.
- The operation of a project related website.
- Distribution of a monthly newsletter.
- Advertising in local media.
- Long term sponsorship of wide variety community / sporting events and organisations.

20.6 Site Monitoring

Bowdens have operated an extensive locally based environmental monitoring program over a long period of time. The data and information gathered during the implementation of this historical monitoring program has been used to inform environmental assessment of the project and will be used to inform baselines to which the environmental performance of the Project will be compared during development and operation phases.

The environmental performance of the Project will be measured in part via the implementation of an expanded environmental monitoring program. Results from an expanded monitoring program will also be utilised to ensure that compliance requirements of the project are satisfied.

The expanded environmental monitoring program will be developed in consultation with local community members and regulatory authorities and be subject of implementation requirements detailed in management plans. The auditing of the performance of the Projects environmental monitoring program will occur on a regular basis and be formally reported on at least annually.

20.7 Water Management

Site water balance modelling along with surface water and groundwater impact assessments performed for the Project have identified that an integrated water management and supply system is able to be developed and implemented that will adequately manage potential impacts to receiving waters and be able to supply the water demands of the Project.

The integrated water management system will manage ground water seepage and any surface water runoff from the open cut pit area, the filtered tailings landform (FTL), processing plant area, oxide ore stockpile and waste rock emplacement that will potentially have elevated concentrations of dissolved metals, within a closed water management system. Within the closed water management system any potentially acid forming leachate will be conveyed to a dedicated leachate management dam. Water accumulated within the closed water section of the water management system will be used as the first priority water source for recycling in the Projects processing operations.

The returning of run-off from area's undisturbed by project related activities outside of the closed water management system to the environment will be maximised. The implementation of clean water diversion channels will maximise the flow of clean water to the environment from within the Project area.

Water demand for the project is largely made up of water used for dust suppression and water used in processing activities. This water demand will be satisfied via a combination of run-off accumulated in site storages, recycling of water used in processing activities and ground water sources.

Bowdens possesses Water Access Licences that account for all water to be taken from all required water sources including water to be taken under Harvestable Rights provisions.

20.8 Permitting Requirements

Bowdens submitted its DA and associated EIS to the NSW DPHI in May 2020. In March 2021, Bowdens also submitted its MLA.

During the EIS exhibition process, no objection to the BSP were received from any Government agency and a high level of public support was experienced. At the end of December 2022, the NSW DPHI advised that assessment of BSP was in the public interest and approvable subject to conditions of consent. The NSW DPHI also referred BSP to the IPC of NSW for final determination. Consequently, on 3 April 2023, the IPC announced the approval of BSP allowing BSP to proceed to development and production, subject to conditions of consent.

The Bingman Catchment Landcare Group Incorporated (Bingman) commenced proceedings in the Land and Environment Court of NSW challenging the development consent for BSP approved by the IPC on 3 April 2023. The Proceedings challenged whether the IPC adequately considered matters relating to the construction and location of a powerline required to power the mine site.

On 14 March 2024, the Company announced that Bowdens had successfully defended the Proceedings with the Land and Environment Court of NSW, dismissing the Proceedings. The court upheld the decision of the IPC with respect to the grant of the Development Consent.

Subsequently, Bingman filed an appeal in the NSW Supreme Court, Court of Appeal. A hearing was held on 22 July 2024. The appeal challenged the decision by the Land and Environment Court and on 16 August 2024 the appeal was upheld in a 2-1 majority decision by the Court of Appeal.

Following the decision by the Court to set aside the Bowdens' Development Consent granted by the IPC, NSW legislated amendments to the Environmental Planning and Assessment Act 1979 (NSW) (EP&A Act). The amended legislation empowers the Planning Secretary to declare which part of a project does or does not form part of a single proposed development for the purposes of determining what does and does not require consent under an SSD approval. This helps to reinstate the general approach taken to the planning assessment of SSD projects, such as Bowdens, prior to the Court's decision.

In April 2025, the Company announced that it has submitted a request to the NSW DPHI, pursuant to Section 4.38(4A) of the EP&A Act, for the Planning Secretary to determine whether or not the transmission line to power BSP will form part of the single SSD. In May 2025, Bowdens was informed that the Planning Secretary had determined under Section 4.38(4A) of the EP&A Act that the 66 kV power line that will provide electricity to the Project (Transmission Line) does not form part of the single proposed development that is BSP (SSD 5765).

The outcome of this determination by the Planning Secretary essentially reinstates the general approach taken to the planning assessment of BSP prior to the August 2024 NSW Court of Appeal decision to set aside the Bowdens' development consent granted by the IPC, which always assumed that approval for the Transmission Line would be sought under an alternative planning pathway.

The NSW DPHI subsequently issued Bowdens with a request for information (RFI) for Bowdens to supply any additional necessary for the redetermination of the Project. Along with updates to relevant EIS assessments, Bowdens chose to refresh its ecological surveys and provide an updated biodiversity assessment in accordance with the latest requirements of the BC Act.

The refreshed surveys were completed in the first half 2026, with the updated biodiversity assessment recently submitted to the NSW DPHI along with other updated assessment information. It is expected that the NSW DPHI will then provide an updated assessment report to the IPC of NSW for consideration and subsequent DA redetermination.

20.9 Other Regulatory Requirements, Leases, Licences, Authorisations, Permits

The Project will be required to acquire or retain the following approvals listed below:

- Development Consent issued under the Environmental Planning and Assessment Act 1979.
- An approval from the Commonwealth Minister for the Environment and Energy under the Environment Protection and Biodiversity Conservation Act 1999 Cth.
- A Mining Lease issued under the Mining Act 1992.
- An Environment Protection Licence issued under the Protection of the Environment Operations Act 1997.
- Water Access Licences in accordance with the Water Management Act 2000.
- Permits under Section 138 of the Roads Act 1993.
- Explosive licence under the NSW Explosives Act 2003 and the NSW Explosives Regulation 2024.
- Construction and Occupation Certificates under Part 6 of the Environmental Planning and Assessment Act 1979.
- Council approval for the operation of a sewage management system under the Local Government Act 1993.
- Agreement of the Country Rail Network Manager to access the rail corridor.
- Crown Lands Licences, to occupy Crown Lands.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

All costs are expressed in A$ unless otherwise stated and are based on first quarter 2026 (1Q26) pricing. The engineering design and cost estimates are consistent with AACE International Class 3, or better.

21.1.1 Mining Capital

The mining financial model utilises capital financing rather than outright purchasing for the main mining mobile fleet. As such, the only capital purchases associated with mining are the support equipment such as light vehicles, lighting towers, sump pumps and some ancillary / backup machinery. The direct capital cost for mining is A$6.45M, with an additional A$2.50M in replacement capital spread out over the DFS mine plan. The replacement capital increases to A$4.96M in the LOM plan as more equipment is replaced over the longer mine life.

The operating costs associated with pre-production mining is capitalised, including mine pre-strip, and development of roads, dumps and stockpiles. The costs incorporate all the ownership costs of the equipment along with fixed costs and the mining owner's team costs.

21.1.2 Process Capital

The process plant capital estimate is based on the following unit processes to produce a high-grade bulk concentrate:

- Two stage crushing circuit.
- Fine ore bin storage with 8 h live storage.
- Primary and secondary grinding circuit with intermediate fines screening to reduce overgrinding.
- Rougher / scavenger flotation.
- Concentrate classification and regrind circuit.
- Two stage cleaner flotation.
- Concentrate thickening and filtration.
- Flotation tailings thickening and filtration for dry stack disposal.
- Reagents.
- Air and water services.

The process plant capital estimate also includes the following process plant related infrastructure:

- Raw water storage and distribution.
- Various pumping systems for site water management.
- Internal plant roads for light and heavy vehicle access.
- Internal plant earthworks, drainage and water containment facilities.
- Power distribution, from the point of termination of the incoming 66 kV grid supply.
- Emergency power supply to critical loads in the event of grid power interruption.
- Process plant offices, change house, workshop and warehouse facilities.

- Process plant control room.
- Mining services offices, change house, heavy vehicle and light vehicle workshops and warehouse facilities.
- Heavy vehicle and light vehicle wash bay.
- Bulk fuel storage and distribution facilities.
- General admin offices, and medical facilities.
- Reagent storage facilities.
- Site laboratory facilities.
- Potable water treatment and distribution.
- Sewage collection and treatment.

Process plant and ancillary infrastructure capital costs were compiled using the methods of calculation described in Table 21.1.1.

Table 21.1.1 Capital Cost Methods of Calculation

Discipline	Sub Items	Method
Earthworks	Quantities	Bulk earthworks quantities based on preliminary design specific for the Project looking to generally balance cut and fill quantities. General assumptions made for cut and fill batter angles in the absence of specific geotechnical information. Detailed earthworks for foundations are included in concrete costs.
	Unit Rates	Unit rates are based on market enquiry for the study in 1Q26 based on preliminary quantities. Rates from WTC were used in the estimate.
Concrete	Quantities	Quantities derived by area based on previous reference projects, factored as necessary. Piling quantities based on preliminary design specific for the Project.
	Unit Rates	Unit rates are based on market enquiry for the study in 1Q26 based on preliminary quantities and confirmed for final quantities. Rates from Australian Construction Group (ACG) were used in the estimate.
Steelwork	Quantities	Quantities derived by area based on previous reference projects, factored as necessary. Some areas, namely milling were based on preliminary design specific for the Project.
	Supply Rates	Rates from current Lycopodium Project database based on SE Asian fabrication. Rates are inclusive of workshop detailing, supply, fabrication and painting. Australian based fabrication rates were obtained for the Project, but not used due to higher prices. It is assumed fabricated steelwork can be imported free of import duty.
	Install Rates	Unit rates are based on market enquiry for the study in 1Q26 based on preliminary quantities. Rates from ACG were used in the estimate.

Discipline	Sub Items	Method
Platework	Quantities	Platework quantities based on preliminary design and sizing from the mechanical equipment list. Minor platework quantities estimated per item.
	Supply Rates	Rates from current Lycopodium Project database based on SE Asian fabrication. Rates are inclusive of workshop detailing, supply, fabrication and painting. Australian based fabrication rates were obtained for the Project, but not used due to higher prices. It is assumed fabricated platework can be imported free of import duty.
	Install Rates	Unit rates are based on market enquiry for the study in 1Q26 based on preliminary quantities. Rates from ACG were used in the estimate.
Mechanical Equipment	Supply Costs – Major Equipment	Multiple budget pricing received from international vendors against datasheets developed specifically for the Project. Pricing provided and compiled using native currencies. Preliminary evaluation conducted to select appropriate vendor (not necessarily lowest).
	Supply Costs – Minor Equipment	Minor equipment pricing obtained from Lycopodium database of previous and current project orders for similar equipment.
	Install Rates	Unit rates are based on market enquiry for the study in 1Q26 based on preliminary manhours. Rates from ACG were used in the estimate. Installation manhours per item of equipment are from Lycopodium's database.
Overland Piping	Quantities	Overland piping quantities based on preliminary design specific for this project.
	Supply Rates	Rates from current Lycopodium Project database. Supply rates quoted by SMP tenderers considered too high.
	Install Rates	Unit rates are based on market enquiry for the study in 1Q26 based on preliminary quantities. Rates from ACG were used in the estimate.
Other Piping		Costs factored as a % of mechanical equipment, based on Lycopodium experience.
Electrical, Instrumentation & Control	Quantities	Electrical quantities based on preliminary design and sizing from the mechanical equipment list and the Navis works model. Minor items quantities estimated per item based on current electrical designs.
	Supply Costs – Major Equipment	Multiple budget pricing received from international vendors against datasheets developed specifically for the Project. Pricing provided and compiled using native currencies. Preliminary evaluation conducted to select appropriate vendor (not necessarily lowest).
	Supply Costs – Minor Equipment	Minor equipment pricing obtained from ECG database of previous and current project orders for similar equipment.
	Install Rates	Unit rates are based on market enquiry for the study in 1Q26 based on unit quantities. Rates from Farelly Construction Services (FCS) were used in the estimate.

Discipline	Sub Items	Method
Freight	Total Cost	Freight cost calculation for structural steel and platework based on container rates & freight tons. Freight estimated for mechanical based on a % of supply cost for equipment.
Import Duty	Total Cost	Taxes and duties have been excluded from the estimate.
Vendor Representatives	Total Cost	Estimate prepared for study based on specialist vendor representatives expected to assist with construction verification and commissioning.
Mobile Equipment	Quantities	Process plant mobile equipment as agreed with Bowdens
	Supply Rates	Rates from current Lycopodium Project database.
Spares	Total Cost	Estimated as 8% of mechanical supply costs
Reagents and Consumables First Fills and Opening Stocks	Quantities	First fill quantities based on storage capacity / supply units. Opening stocks typically seven days.
	Supply Rates	Unit rates are based on market enquiry for the study in 1Q26.
PM Services	Total Cost	Detailed deliverable based estimate prepared for the study and benchmarked against previous reference projects.
Owners Costs	Total Cost	Owner's team costs were excluded.

The following items are not included in the Lycopodium scope but have been included in Bowdens DFS model:

- Owners project team costs to oversee and manage the Project.
- Site-wide mobile equipment.
- Pre-production costs.
- Working capital.

21.1.3 Other Capital

Other key capital costs items not included in the Mining and Process capital estimates include:

- WRE.
- WRE seepage pond.
- WRE flood protection bunding.
- NAF stockpile.
- FTL.
- FTL overflow pond.
- Powerline for site power supply.
- Site clean water dam.
- Harvestable rights dams.
- Settlement dams.
- Site clean and contact water management structures.
- Re-alignment of Maloneys Road.
- Public road intersection upgrades.
- Temporary workers accommodation.

The relevant specialist consultants and engineers provided bills of quantities for these key infrastructure and earthworks components. Bowdens issued the designs and bills of quantities to civil and earthworks contractors to obtain quotes for the work. The capital estimates for these other items are based on the quotes received.

21.1.4 Capital Summary

A summary of all project capital, including contingency, is provided in Table 21.1.2.

Table 21.1.2 DFS Capital Summary

Item / Category	2026 DFS A$	2026 DFS US$
Process and Tails Plant	180,492,517	126,344,762
Site Infrastructure	67,372,771	47,160,940
Offsite Infrastructure	45,924,101	32,146,871
Waste Management	71,041,683	49,729,178
Mobile Equipment	5,898,790	4,129,153
Indirects	52,911,944	37,038,361
Capitalised Pre-Production	30,878,385	21,614,869
Initial Capital	***454,520,191***	***318,164,134***
Infrastructure and Equipment	13,097,362	9,168,153
Waste Management	10,452,564	7,316,795
Sustaining Capital	***23,549,926***	***16,484,948***
Total Capital	**478,070,118**	**334,649,082**

21.2 Operating Costs

21.2.1 Mining Operating Costs

The mining costs are based on a first principles cost model built around owner-mining. Exceptions to owner-mining are drill and blast, and equipment maintenance. The cost model includes quoted contractor rates for a rock on ground drill and blast contract, and an OEM maintenance contract. Operating costs include:

- Mobile mining fleet.
- Mobile ancillary fleet.
- Mobile equipment maintenance.
- Ground engagement tooling (GET).
- Tyres and other consumables.
- ROM ore rehandle.
- Filtered Tailings rehandle.
- Manning to operate and maintain equipment and technical support.
- Mining Owners Team General and Admin Costs.

Table 21.2.1 provides a breakdown of the DFS mining operating costs.

Table 21.2.1 DFS Mining Operating Costs

Mining Component	A$...M	A$ /t
Fixed Charges	235.23	3.19
Clearing and Stripping	7.86	0.11
Drilling and Blasting	110.32	1.49
Load and Haul	117.76	1.59
Rehandling	14.34	0.19
Dry Stack Tailings	87.51	1.19
Total Operating Cost	**573.02**	**7.76**

21.2.2 Process and Administration Operating Costs

The process plant operating costs have been developed by Lycopodium based on a design treatment rate of 2 Mtpa of ore with the process plant operating 24 h/d, 365 d/y with a 91.3% plant utilisation (nominal 8,000 h/y) and a P_{80} primary grind size of 106 µm. Tailings filtration operating costs are estimated on a 76.1% tailings filter utilisation (nominal 6,666 h/y).

Operating costs are based on prices obtained during the first quarter of 2026. The process and tailings plant operating costs for the facilities are summarized in Table 21.2.2.

Table 21.2.2 DFS Process and Admin Operating Costs

Process and Admin Opex	A$ /t Ore	A$ /t Conc
Processing Labour	4.22	339.25
Processing Power	7.98	641.43
Reagents and Consumables	4.78	384.02
Processing Maintenance	2.99	240.29
Other	0.76	61.47
Subtotal Processing	***20.73***	***1,666.46***
Administration Labour	2.52	202.80
General and Administration Other	3.50	281.18
Subtotal G&A	***6.02***	***483.98***
Total Process and Administration	**26.76**	**2,150.44**

21.2.3 Power

The power requirements for the process plant and tailings treatment facility are based on the mechanical equipment list and adjusted for equipment load factor and utilization. Grid power unit cost of A$0.189 per kilowatt (/kWh) is based on a long-term forecast proved by Schneider Electrical.

21.3 Project Implementation

Implementation strategies are being developed in parallel with government approvals.

21.4 Implementation Schedule

The implementation schedule is subject to timing of government approval processes and the implementation strategy.

Start of on ground construction to first ore is expected to be approximately 18 months post government approvals, final investment decision, and development and execution of the implementation strategy.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

Economic analyses have been carried out for four separate cases - two mining inventories (mine designs), and two metal price decks for each mining inventory. The two mining inventories are the DFS (29.9 Mt ore) and LOM (48.29 Mt ore) as discussed in Section 16. The two metal price decks are the Base case, at less than consensus Ag price, and a Spot case. The four cases are:

- DFS Base (primary focus of this report).
- DFS Spot.
- LOM Base.
- LOM Spot.

Exchange rates and metal prices incorporated in the financial models for the Base and Spot cases are provided in Table 22.1.1.

Table 22.1.1 FX and Price Deck

Parameter	Unit	Base Case		Spot Price	
		A$	US$	A$	US$
Exchange Rate	A$:US$	0.70		0.69	
Silver Price	$ /oz	64.29	45.00	87.96	61.00
Zinc Price	$ /lb	1.93	1.35	2.34	1.62
Lead Price	$ /lb	1.29	0.90	1.24	0.86
Gold Price	$ /oz	4,929	3,450	5,977	4,145

The economic analyses include minor amounts of Inferred Resources as discussed in Section 16.1.1. The Inferred Resources represent 0.4% of the ore in the DFS mine plan and 0.6% of the ore in the LOM plan, and therefore have no material impact on the economic analyses.

The economic analyses have been carried out for the project using a financial model developed by Northshore Capital Advisors. The model has been developed using quarterly periods taking into account revenues, processed tonnages and grades, process recoveries, metal prices, operating costs, transport costs, treatment and refining charges, royalties, tax and capital expenditures (both initial and sustaining). The economic analyses have been conducted on a 100% equity (no debt financing) basis, in constant dollar terms. Sunk costs (for tax purposes) and future sale of land assets were included in the model.

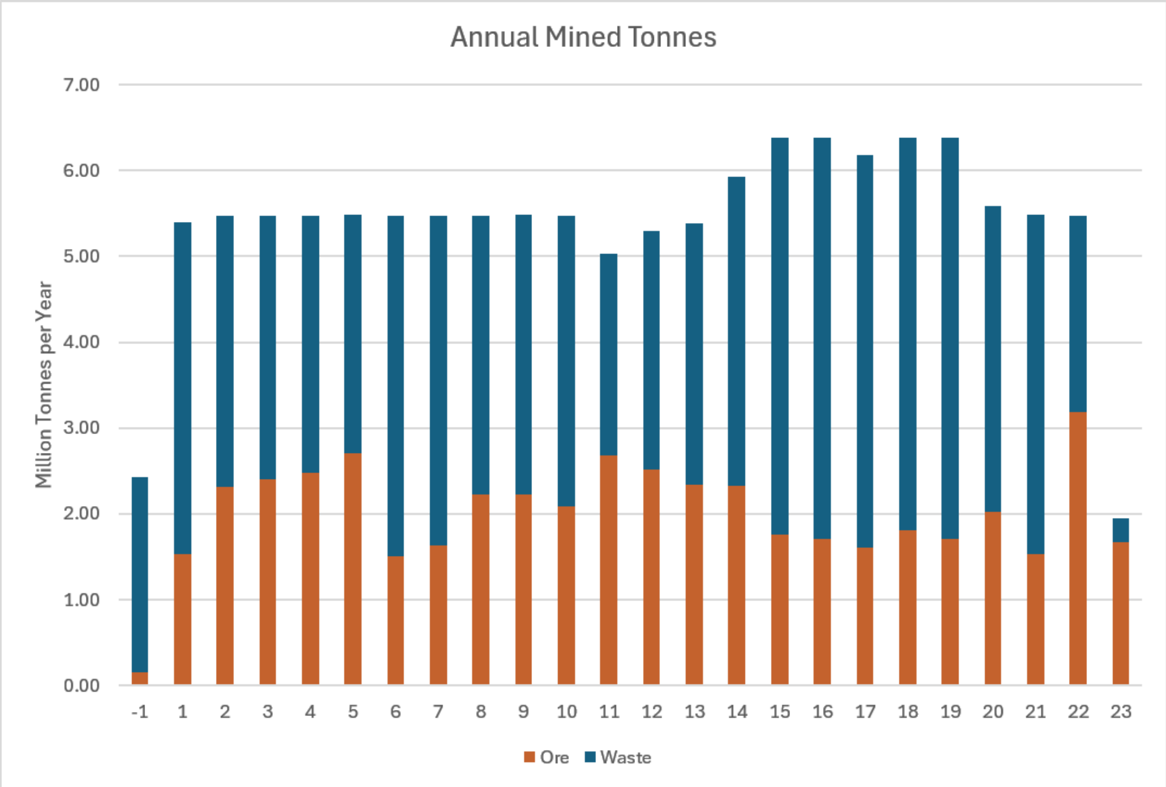
Most economic inputs and outputs are presented in A$ and US$, and outputs in pre-tax and post-tax. Royalties in NSW are an Ad Valorem system based on 4% of the ex-mine value of recovered minerals. Some processing, administration and depreciation cost are deductable.

The base case economic analyses are conservatively based on an Ag price of US$45 /oz, less that the current consensus price, which is approximately US$46.70 /oz; and the trailing two-year average price of US$46.90 /oz. Spot price cases are also provided to demonstrate the upside potential of the project. As discussed in Section 15, it is important to note that the original NSR calculation and resulting Whittle optimisation were based on a very conservative Ag price of US$35 /oz. Therefore, the financial model is highly conservative and demonstrates the robustness of the Project.

22.2 Schedules

The very low strip ratio (SR) of the pit at 1.46 and near surface high grade ore zones supports a low-cost owner operation as the same mining fleet can handle the production requirements from pre-strip through to completion. As discussed in Section 16, the staging of the pits has been scheduled to allow a very flat total production profile while maintaining mill feed with only minor low grade rehandle requirements as demonstrated in Figure 22.2.1. Due to the relative size of the final western pit cutback commencing late in Year 14 (LOM case only), the capacity of the primary production excavator may be exceeded at times during Years 15 - 19. Should this occur, the additional capacity will be picked up by the back-up / batter-cleaner excavator.

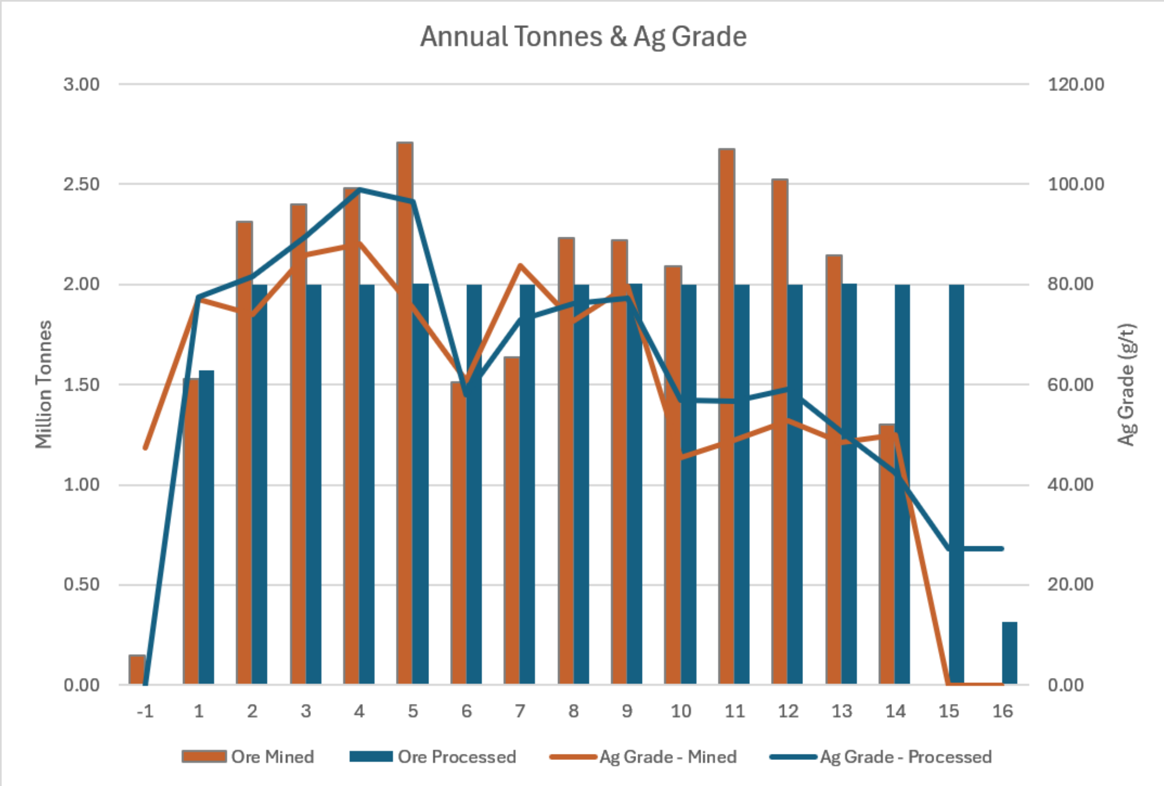
Figure 22.2.1 Annual Mined Tonnes



22.2.1 DFS Schedules

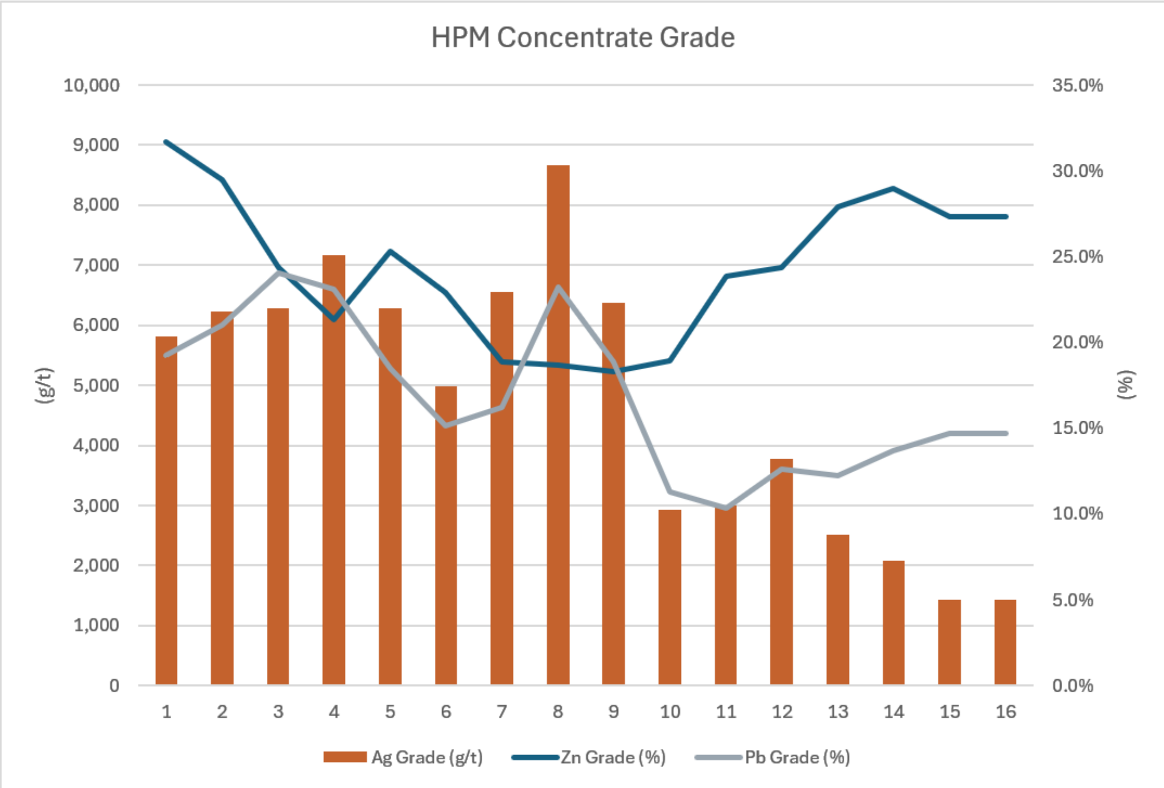
The DFS mining and processing schedules (tonnes and grade by year) are shown in Figure 22.2.2.

Figure 22.2.2 DFS Annual Schedule



The DFS HPM concentrate production grade schedule is shown in Figure 22.2.3.

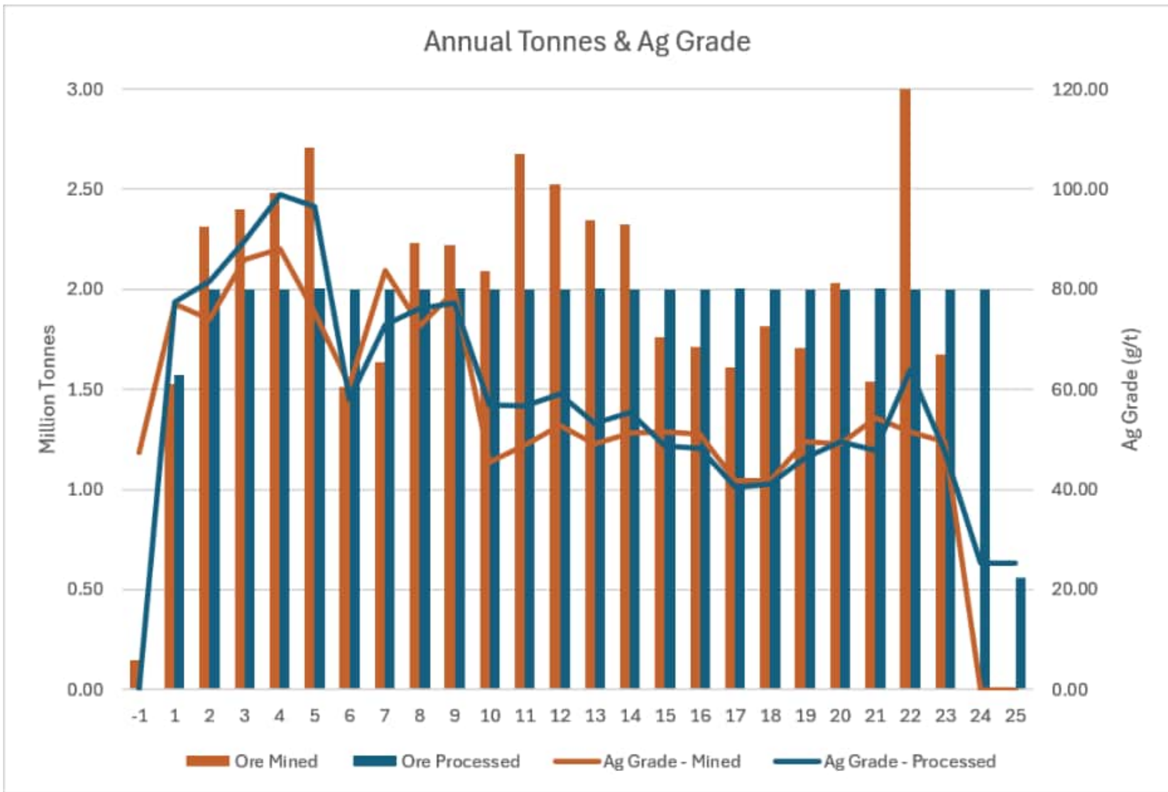
Figure 22.2.3 DFS Annual Concentrate Production



22.2.2 LOM Schedules

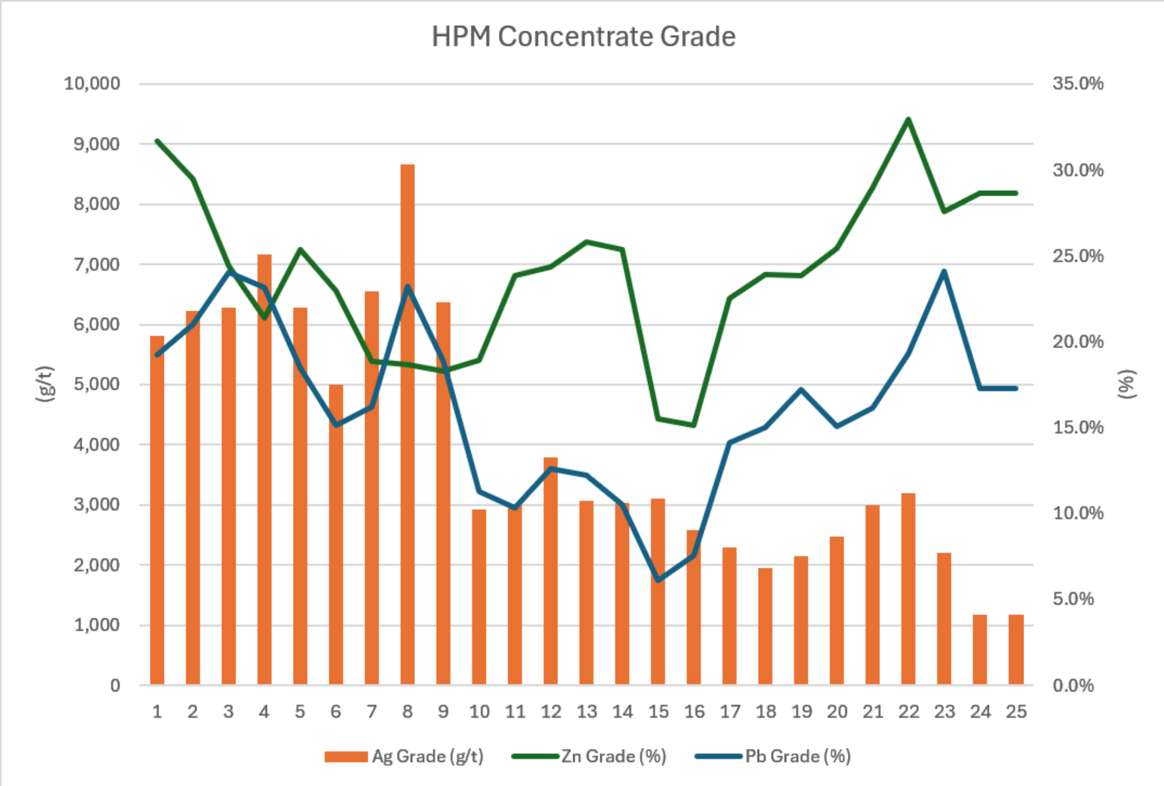
The LOM mining and processing schedules (tonnes and grade by year) are shown in Figure 22.2.4.

Figure 22.2.4 LOM Annual Schedule



The LOM HPM concentrate production grade schedule is shown in Figure 22.2.5.

Figure 22.2.5 LOM Annual Concentrate Production



22.2.3 Key Physicals

Key physicals of the DFS and LOM cases are presented in Table 22.2.1. One interesting aspect of the BSP ore body is that as the Ag grade declines in the later years of production, the Zn and Pb grades increase, resulting in slight differences in overall metal recoveries in the DFS and LOM cases.

Table 22.2.1 Key Physicals

Project Physicals	Unit	2026 DFS	2026 LOM
Project Life (incl. Construction)	Years	16.75	26.00
Operating Life	Years	15.25	24.50
Ore Mined	kt	29,908	48,162
Waste Mined	kt	43,935	81,372
Strip ratio	x	1.47	1.69
Ag Mined Grade	g/t	67.5	60.6
Zn Mined Grade	%	0.34%	0.36%
Pb Mined Grade	%	0.25%	0.26%
Au Mined Grade	g/t	0.01	0.02
Ag Recovery	%	82.8%	82.1%
Zn Recovery	%	90.8%	91.2%
Pb Recovery	%	80.9%	81.4%
Ag Recovered in Concentrate	koz	53,768	77,056
Zn Recovered in Concentrate	kt	91.02	156.96
Pb Recovered in Concentrate	kt	60.47	103.22
Au Recovered in Concentrate	koz	7.45	22.48
Ag Grade in Concentrate	g/dmt	4,494	3,692
Zn Grade in Concentrate	%	24.5%	24.2%
Pb Grade in Concentrate	%	16.2%	15.9%
Concentrate Produced	kdmt	372	649
Transport Weight	kwmt	413	721
Payable Ag	koz	51,080	73,203
Payable Zn	kt	26.6	45.8
Payable Pb	kt	49.3	83.7
Payable Au	koz	0.32	7.50

22.3 Base Case Financial Evaluations

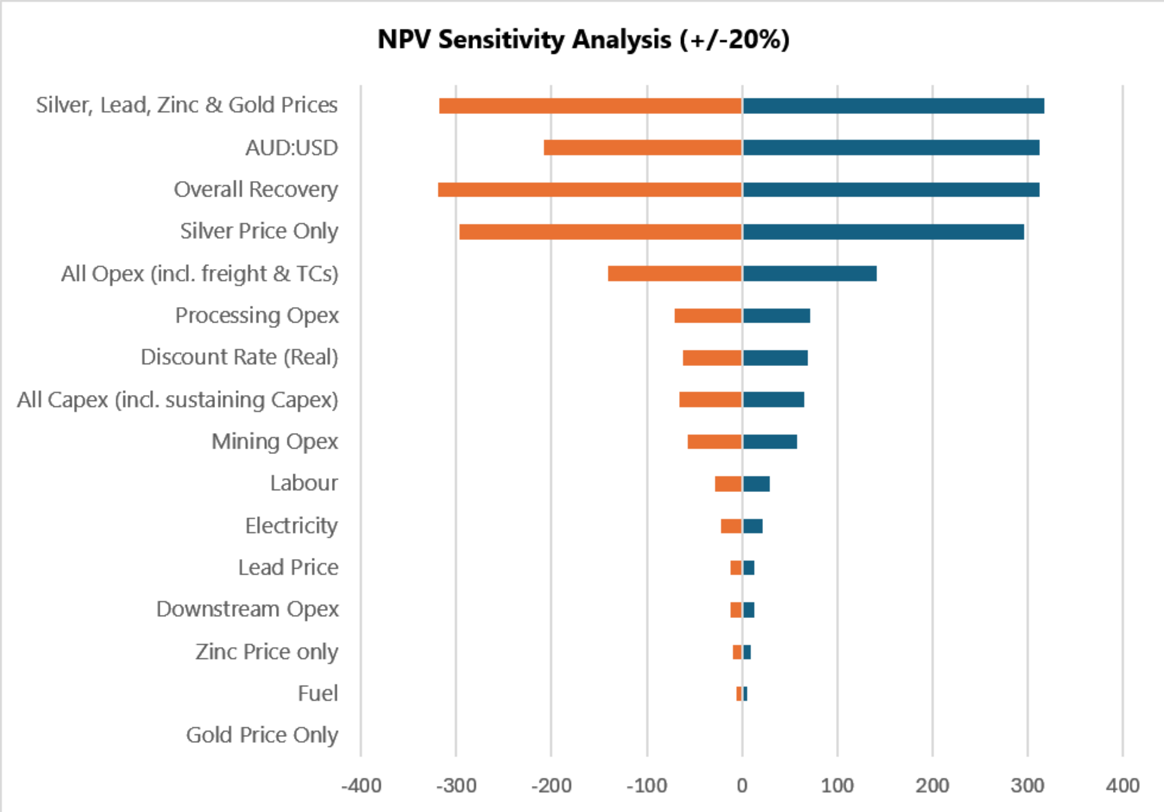
Key base case financial results of the DFS and LOM are presented in Table 22.3.1.

Table 22.3.1 Base Case Key Financial Metrics

Project Financials (Ungeared): Real Unless Stated					
Description	**Unit**	**2026 DFS**		**2026 LOM**	
		A$	**US$**	**A$**	**US$**
Revenue from Payable Metals	$...M	3.538.1	2,476.7	5,175.1	3,622.5
Operating Expenses	$...M	(1,725.9)	(1,208.1)	(2,784.7)	(1,949.3)
Operating Margin	$...M	1,812.2	1,268.6	2,390.4	1,673.3
Post Closure Sale of Land and Buildings	$...M	100.0	70.0	100.0	70.0
Initial Capex	$...M	(423.6)	(296.5)	(423.6)	(296.5)
Pre-Production Expenses	$...M	(30.9)	(21.6)	(30.9)	(21.6)
Initial Capex and Pre-Production Expenses	$...M	(454.5)	(318.2)	(454.5)	(318.2)
Sustaining Capex	$...M	(23.5)	(16.5)	(90.8)	(63.6)
Total Capital	$...M	(478.1)	(334.6)	(545.4)	(381.8)
Undiscounted Cashflow Pre-Tax	$...M	1,434.2	1,003.9	1,945.0	1,361.5
Tax Payable	$...M	(378.5)	(265.0)	(532.5)	(372.8)
Undiscounted Cashflow After Tax	$...M	1,055.6	738.9	1,412.5	988.7
C1 Cost (Ag basis)	$ /oz	26.91	18.84	29.75	20.82
C2 Cost (Ag basis)	$ /oz	36.22	25.35	37.16	26.01
C3 Cost (Ag basis)	$ /oz	37.17	26.72	39.08	27.36
All-in-Sustaining Cost (AISC): Ag Basis	$ /oz	29.32	20.53	32.91	23.04
C1 Margin (Ag Basis)	%	58.1%		53.7%	
C2 Margin (Ag Basis)	%	43.7%		42.2%	
C3 Margin (Ag Basis)	%	40.6%		39.2%	
AISC Margin (Ag Basis)	%	54.4%		48.8%	
Project NPV (Pre-Tax)	$...M	877.4	614.2	1,043.3	730.3
Project NPV (Post Tax)	$...M	625.7	438.0	735.8	515.0
Project IRR (Pre-Tax)	%	31.5%		31.5%	
Project IRR (Post Tax)	%	26.2%		26.2%	
Project Payback Period from Construction Start	Years	4.6		4.6	
Project Payback Period from Production Start	Years	3.0		3.0	
Maximum Project Drawdown (Nominal)	$...M	455.2	318.6	455.2	318.6
Breakeven Silver Price	$ /oz	37.54	26.28	37.53	26.27
Silver Price to achieve 30% After Tax IRR	$ /oz	70.31	49.22	70.31	49.22
Profitability Index	x	2.40 x		2.64 x	
Ratio of NPV to Pre-Production Capital	x	1.93		2.30	

Figure 22.3.1 shows the key project post-tax NPV sensitivities for the base case. Overall metal recovery to concentrate has nearly the same sensitivity as combined Zn, Pb, Ag and Au price, therefore are not shown in the figure as the lines effectively co-exist and are not distinguishable.

Figure 22.3.1 DFS Base Case - Sensitivity Analysis



Revenue from each payable metal in the HPM concentrate is listed in Table 22.3.2. Revenue is dominated by Ag at 92.8% and 90.9% for the DFS and LOM cases respectively. Referencing Table 22.2.1, it can be seen despite the HPM concentrate having a higher Zn grade than Pb grade, the revenue from Pb is higher. This is due to the conservative payability of Zn assumed for the HPM concentrate, with an overall average payability of only 26.6% over the LOM; although there may be opportunities to increase Zn payability over time.

Table 22.3.2 Revenue

Revenue from Payable Metals							
Metal	**Unit**	**2026 DFS**			**2026 LOM**		
		A$	**US$**	**%**	**A$**	**US$**	**%**
Revenue from Ag	$...M	3,283.7	2,298.6	92.8%	4,705.9	3,294.2	90.9%
Revenue from Au	$...M	1.6	1.1	0.0%	37.0	25.9	0.7%
Revenue from Zn	$...M	113.1	79.2	3.2%	194.8	136.4	3.8%
Revenue from Pb	$...M	139.7	97.8	3.9%	237.4	166.2	4.6%
Total		**3,538.1**	**2,476.7**	**100.0%**	**5,175.1**	**3,622.5**	**100.0%**

22.4 Spot Case Financial Evaluations

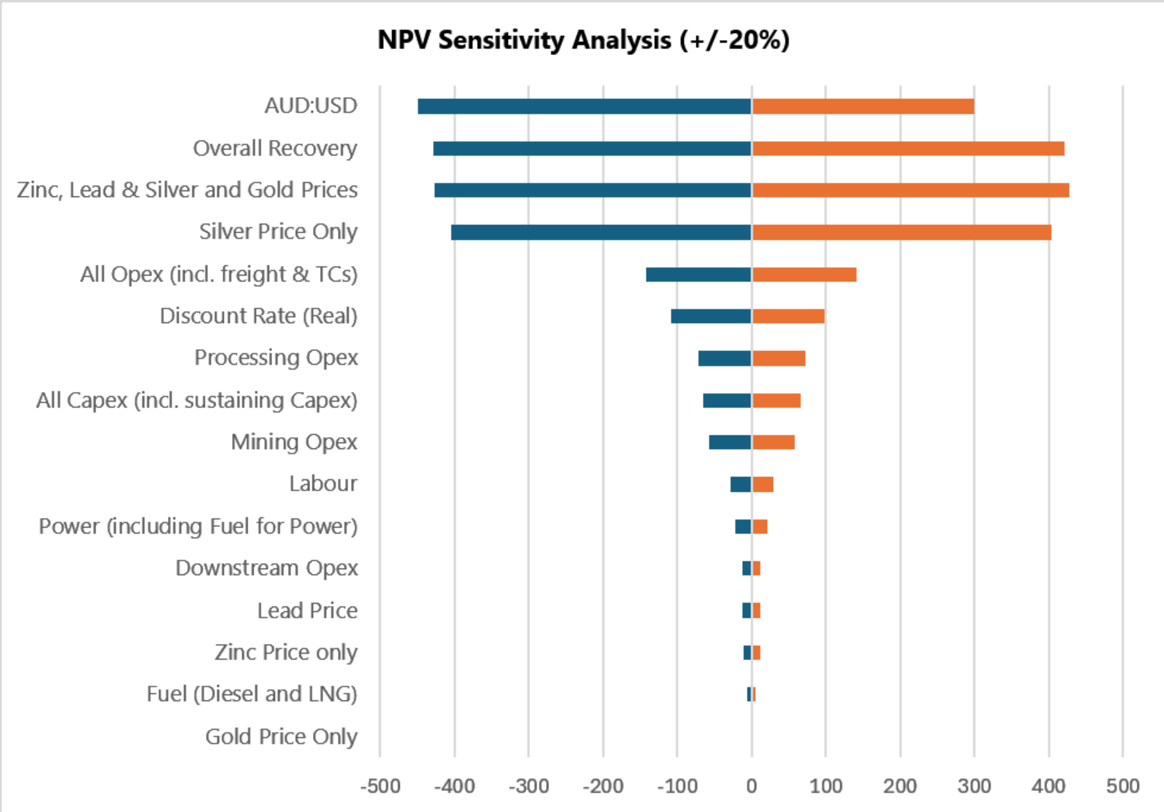
Key spot case financial results of the DFS and LOM are presented in Table 22.4.1.

Table 22.4.1 Spot Case Key Financial Metrics

Project Financials (Ungeared): Real Unless Stated					
Description	Unit	2026 DFS		2026 LOM	
		A$	US$	A$	US$
Revenue from Payable Metals	$...M	4,766.6	3,305.7	6,948.7	4,818.9
Operating Expenses	$...M	(1,812.2)	(1,256.8)	(2,911.2)	(2,018.9)
Operating Margin	$...M	2,954.4	2,048.9	4,037.4	2,800.0
Post Closure Sale of Land and Buildings	$...M	100.0	69.4	100.0	69.4
Initial Capex	$...M	(425.3)	(294.9)	(425.3)	(294.9)
Pre-Production Expenses	$...M	(30.9)	(21.4)	(30.9)	(21.4)
Initial Capex and Pre-Production Expenses	$...M	(456.2)	(316.4)	(456.2)	(316.4)
Sustaining Capex	$...M	(23.6)	(16.4)	(90.9)	(63.1)
Total Capital	$...M	(479.8)	(332.7)	(547.2)	(379.5)
Undiscounted Cashflow Pre-Tax	$...M	2,574.6	1,785.5	3,590.3	2,489.9
Tax Payable	$...M	(720.7)	(499.8)	(1,026.1)	(711.6)
Undiscounted Cashflow After Tax	$...M	1,853.9	1,285.7	2,564.2	1,778.3
C1 Cost (Ag basis)	$ /oz	27.22	18.88	29.92	20.75
C2 Cost (Ag basis)	$ /oz	36.56	25.35	37.35	25.90
C3 Cost (Ag basis)	$ /oz	39.51	27.40	40.28	27.93
All-in-Sustaining Cost (AISC): Ag Basis	$ /oz	30.64	21.25	34.09	23.64
C1 Margin (Ag Basis)	%	69.1%		66.0%	
C2 Margin (Ag Basis)	%	58.4%		57.5%	
C3 Margin (Ag Basis)	%	55.1%		54.2%	
AISC Margin (Ag Basis)	%	65.2%		61.2%	
Project NPV (Pre-Tax)	$...M	1,660.7	1,151.7	2,012.0	1,395.3
Project NPV (Post Tax)	$...M	1,174.6	814.6	1,414.6	981.0
Project IRR (Pre-Tax)	%	49.0%		49.0%	
Project IRR (Post Tax)	%	40.3%		40.3%	
Project Payback Period from Production Start	Years	2.2		2.2	
Maximum Project Drawdown (Nominal)	$...M	456.2	316.4	456.2	316.4
Breakeven Silver Price	$ /oz	37.08	25.95	37.98	26.59
Silver Price to Achieve 30% After Tax IRR	$ /oz	69.90	48.93	69.87	48.91
Profitability Index	x	3.63 x		4.17 x	
Ratio of NPV to Pre-Production Capital	x	3.64		4.41	

Figure 22.4.1 shows the key project post-tax NPV sensitivities for the Spot Case. Overall metal recovery to concentrate has nearly the same sensitivity as combined Zinc, Lead, Silver and Gold price; therefore, not shown in the figure as the lines effectively co-exist and are not distinguishable.

Figure 22.4.1 DFS Spot Case – Sensitivity Analysis



22.5 Summary

The financial analyses demonstrate that BSP is financially sound with significant upside potential at spot pricing. The Project is also robust with a breakeven Ag price of approximately US$26 /oz.

The Project is highly leveraged to the Ag price with minimal sensitivity to Zn and Pb pricing or operating costs.

23.0 ADJACENT PROPERTIES

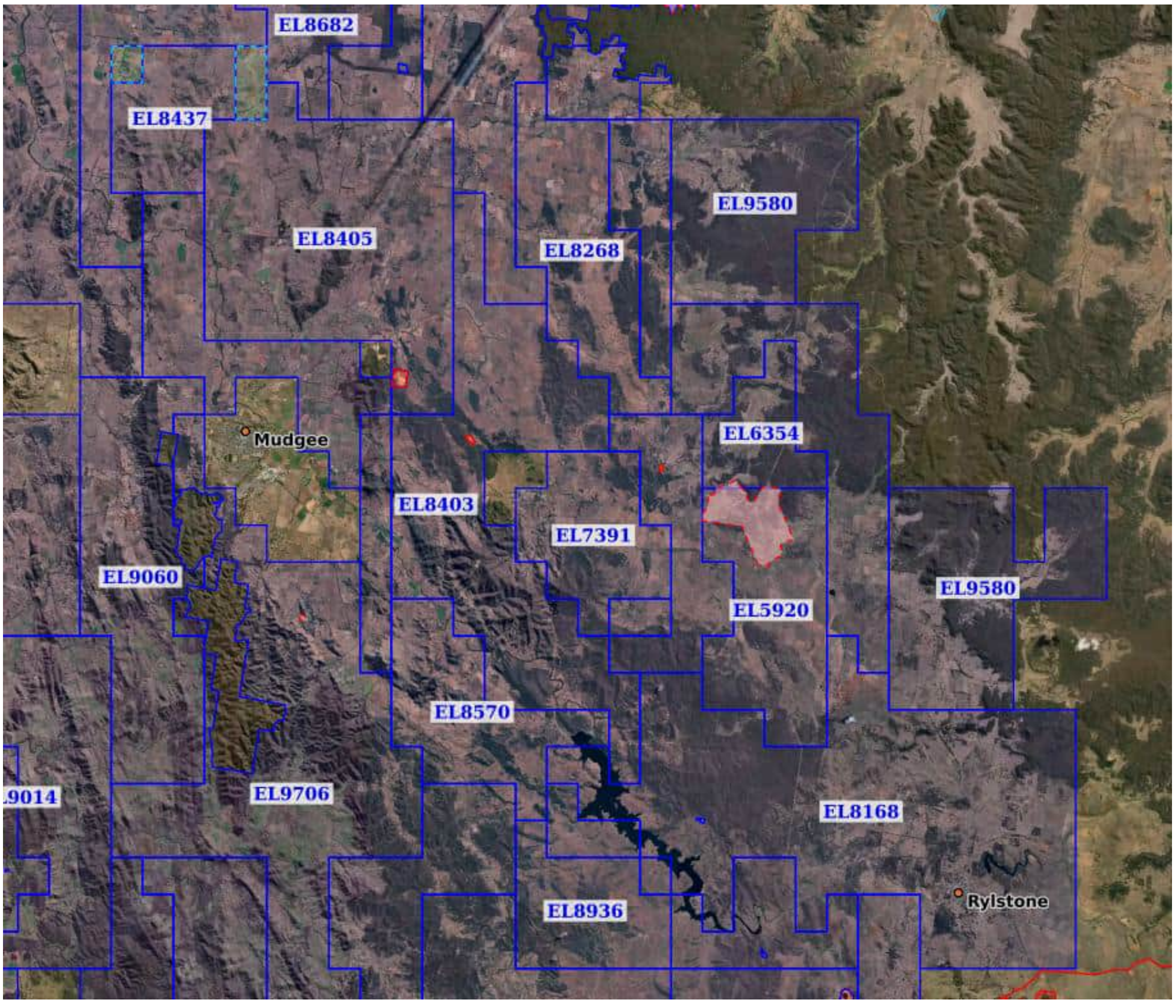
Silver Mines' subsidiary companies, Bowdens and Bowdens Agriculture Pty Ltd, own the majority of land that falls within the mine site boundary as well as those that border it. One privately owned parcel of land falls within the mine site area. The Company has an agreement in place with the landowner that allows for mining activities to occur under a commercial agreement. Bowdens also actively farms areas adjoining the site, with a mixture of sheep and cattle grazing and some small-scale fodder cropping.

Apart from Lue village itself, most land immediately surrounding the site comprises a combination of grazing, lifestyle lots, and heavily vegetated areas. Grazing is the predominant land use immediately surrounding the mine site. To a lesser degree, other properties in the area farm olives and grapes, and there is a variety of small businesses including Airbnb / farm stay accommodation. Lue also has a licenced hotel, Rural Fire Service shed, small business, and a well-established motocross complex that provides recreational and accommodation services.

A registered Native Title Claim (Warrabinga Wiradjuri #7) covers a vast tract of land of more than 14,000 km^2 in multiple LGAs. This claim covers the Bowdens mine site area. A portion of NSW Crown Land exists within MLA601 where Native Title has not been extinguished. The Company has completed an agreement in relation to this parcel with the registered Native Title Claimants. In addition, the related Section 31 Deed has subsequently been executed by the NSW Minister for Natural Resources on behalf of the State of NSW. This completed the 'Right to Negotiate' process in accordance with Section 31 of the Native Title Act 1993 (Cth).

Bowdens currently holds 11 ELs within the MWRC LGA. These are all 100% owned by the Company and comprise 2,115 km^2 (521,000 acres) of titles covering approximately 80 km of strike of the highly prospective Carboniferous Rylstone Volcanics. The Bowdens deposit sits within EL5920. Within and immediately adjacent to these tenements are other ELs held by a range of other companies and holders. Approximately 27 km to the north of the site are three well established and operating coal mines owned by Glencore, Yancoal, and Peabody.

Figure 23.1 Surrounding Leases



24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Health and Safety Management System

Bowdens has an established health and safety management system (HSMS) that will be further developed prior to the Project commencing.

The purposes and objectives of the Bowdens HSMS are to:

- Establish the systems and practices that Bowdens will apply to achieve the health and safety objectives.
- Provide the minimum requirements in relation to health and safety.
- Ensure all personnel at Bowdens understand the hazards associated with their work activities and location.
- Ensure all personnel at Bowdens understand their responsibilities in relation to health and safety.
- Meet legal requirements associated with legislation.

The Bowdens HSMS comprises the following seven levels of documentation.

- Level 1 – Occupational Health and Safety Management System Overview.
- Level 2 – Policies.
- Level 3 – Standards.
- Level 4 – Management Plans.
- Level 5 – Procedures and Registers.
- Level 6 – Forms and Electronic Files.
- Level 7 – Reference Documents.

The emergency management plan (EMP) will be further enhanced with the implementation strategy to ensure evolving needs and potential issued are addressed.

24.2 Project Opportunities

There are some opportunities identified to reduce the capital cost of the process plant. They include:

- It is possible that further filtration testwork on the tailings product may result in being able to reduce the size or quantity of the tailings filters.
- Review conveyor angles to see if can be increased and therefore reduce plant footprint.
- Reduce mill heights to reduce concrete and steel quantities in the milling area, subject to more detailed design.
- Reduce suspended concrete slab in mill area to a practical minimum and replace with steel structure and gridmesh.
- Review location of trash screen and flotation feed sampler. May be better to locate in flotation area rather than milling area.
- May be possible to reduce the size of cleaner conditioner tanks.
- Review if some metallurgical accounting samplers could be replaced with cheaper process samplers.

- Replace existing square pump hoppers with conical design. Requires less platework in terms of stiffeners, etc.
- Review pipe-rack design to see which can be on-ground racks rather than elevated racks.

These opportunities can be readily addressed during the detailed design phase.

24.3 Project Risks

Some risks identified for the project include:

- The BSP scope is split among a number of consultants. A review of the battery limits associated with each package of work needs to be completed as part of the compilation of the completed project estimate, to ensure there are no unidentified scope gaps and therefore underestimation of project costs.
- Potential delay to project commencement or completion due to external issues including:

  - delay in obtaining necessary government approvals
  - delay in establishing grid power connection due to reliance on external parties and associated permitting issues, although this can be mitigated by a hired diesel power station.

- Unfavourable changes in commodity prices or exchange rates.
- Increased labour and/or supply costs.
- Changes to concentrate marketing terms.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Technical

The DFS satisfies the requirements of an AACE International Class 3 engineering design and cost estimate, with some components completed to a higher level. The DFS report has been compiled by Bowdens with Lycopodium as the lead consultant, with inputs from other sources as provided in Section 3.

Comments on the technical adequacy of various components of the DFS are:

- The MRE is compliant with the Australian JORC guidelines. While this report as a whole is not compliant with or being issued as an NI 43-101 Technical Report, the MRE complies with all disclosure requirements set out in the NI 43-101 *Standards of Disclosure for Mineral Project* (2023) and adheres to the 2019 CIM *Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (2019 CIM Guidelines).
- The ORE is compliant with the JORC guidelines. While this report as a whole is not compliant with or being issued as an NI 43-101, the Reserve complies with all disclosure requirements set out in the NI 43-101 *Standards of Disclosure for Mineral Project* (2023) and adheres to the 2019 CIM *Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines* (2019 CIM Guidelines).
- The metallurgical testwork library is sufficiently large and varied to completed to date provides a robust and internally consistent basis for the process design adopted in the 2026 DFS. However, as is typical for a project at the DFS stage, a limited number of residual gaps remain that should be addressed during subsequent stages of project development to further reduce technical risk and support detailed engineering, commissioning, and early operations, as detailed in Section 13.
- The process plant design is based on a significant library of metallurgical testwork. A robust metallurgical flowsheet has been designed for optimum recovery with minimum operating costs. The flowsheet consists of unit operations that are well proven in the industry and are based on results from testwork conducted on samples collected from the BSP resource.
- Surface water, groundwater, geochemistry, waste rock and tailings management designs have been completed by the same consultancy to ensure interactions and co-dependencies are accounted for.
- Site infrastructure components (WRE and seepage pond, FTL and overflow catchment, clean and contact water management facilities, public and internal roads, temporary stockpiles and operational pads) have been designed to a AACE International Class 3 or better level with detailed bills of quantities developed for cost estimates.

25.2 Commercial

The capital and operating costs for most areas have been prepared by various consultants that are experts in their field and satisfy AACE International Class 3 estimates. Bowdens developed general and administration costs on a first principles basis, incorporating known and estimated fixed and variable cost components.

The Project financial model has been developed using quarterly periods taking into account revenues, processed tonnages and grades, process recoveries, metal prices, operating costs, refining charges, royalties, tax and capital expenditures (both initial and sustaining). The economic analyses have been conducted on a 100% equity (no debt financing) basis and in constant dollar terms. Sunk costs (for tax purposes) and future sale of land assets were included in the model.

The DFS has comprehensively evaluated BSP and found it to be technically feasible and economically robust.

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