NI 43-101 Technical Report on the Preliminary Economic Assessment

Converse Project, Nevada, USA

Roxmore Resources Inc.

Prepared by:

SLR Consulting (Canada) Ltd.

SLR Project No.: 233.065545.00001

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Making Sustainability Happen

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Roxmore Resources Inc. Note on Forward Looking Information

This Technical Report contains “forward-looking information” within the meaning of applicable Canadian securities legislation. Forward-looking information includes, but is not limited to: the timing of the potential advancement of the Converse Project towards pre-feasibility and feasibility, the completion of the environmental baseline work, the expansion of the pit at the Converse Project, the planned drilling metres, the capex requirements, the funding of activities in 2026 and beyond, IRR, NPV, AISC, the future price of gold, cash flow, payback period, the development of infrastructure, life-of-mine and other future financial or operating performance of the Converse Project, long-life and large-scale potential of the Converse Project, and exploration upside of the land packages. Generally, forward-looking information can be identified by the use of forward-looking terminology such as “accelerate”, “add” or “additional”, “advancing”, “anticipates” or “does not anticipate”, “appears”, “believes”, “can be”, “conceptual”, “confidence”, “continue”, “convert” or “conversion”, “deliver”, “demonstrating”, “estimates”, “encouraging”, “expand” or “expanding” or “expansion”, “expect” or “expectations”, “fast-track”, “forecasts”, “forward”, “goal”, “improves”, “increase”, “intends”, “justification”, “leading”, “plans”, “potential” or “potentially”, “pro forma”, “promise”, “prospective”, “prioritize”, “reflects”, “re-rating”, “robust”, “scheduled”, “stronger”, “suggesting” or “suggests”, “support”, “updating”, “upside”, “will be” or “will consider”, “work towards”, or variations of such words and phrases or state that certain actions, events or results “may”, “could”, “would”, “might”, or “will be taken”, “occur”, or “be achieved”.

Forward-looking information is based on the opinions and estimates of the QPs at the date the information is made, and is based on a number of assumptions and is subject to known and unknown risks, uncertainties and other factors that may cause the actual results, level of activity, performance or achievements of the issuer to be materially different from those expressed or implied by such forward looking information, including risks associated with required regulatory approvals, the exploration, development and mining such as economic factors as they effect exploration, future commodity prices, changes in foreign exchange and interest rates, global inflationary pressures, actual results of current exploration activities, government regulation, political or economic developments, the ongoing wars and their effect on supply chains, tariffs, environmental risks, pandemic risks, permitting timelines, capex, operating or technical difficulties in connection with development activities, employee relations, the speculative nature of gold exploration and development, including the risks of diminishing quantities of grades of reserves, contests over title to properties, and changes in project parameters as plans continue to be refined as well as those risk factors discussed in the issuer’s Annual Information Form for the year ended December 31, 2024, available on www.sedarplus.ca.

Although the QPs have attempted to identify important factors that could cause actual results to differ materially from those contained in forward-looking information, there may be other factors that cause results not to be as anticipated, estimated or intended. There can be no assurance that such information will prove to be accurate, as actual results and future events could differ materially from those anticipated in such information. Accordingly, readers should not place undue reliance on forward-looking information. The QPs do not undertake to update any forward-looking information, except in accordance with applicable securities laws.

Use of Non-GAAP Financial Measures

Certain financial measures referred to in this Technical Report are not measures recognized under IFRS and are referred to as non-GAAP financial measures or ratios. These measures have no standardized meaning under IFRS and may not be comparable to similar measures presented by other companies.

These measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS.

The non-GAAP financial measures used in this Technical Report and common to the gold mining industry, include, but are not limited to, the following:

- Free Cash Flow is a non-IFRS financial measure. Undiscounted and net of Initial and Sustaining Capital Expenditures (Capex), and Operating Costs (Opex).
- Sustaining Capital is a non-IFRS financial measure. Sustaining capital is defined as required capital spent in existing operation to maintain production levels such as equipment replacement, haul roads, raise of the Tailings Storage Facility (TSF), water diversion, water ponds and water management.
- All-in Sustaining Costs (AISC) includes cash costs plus sustaining capex and closure. AIC includes AISC plus initial capex.
- Earnings before interest, taxes, depreciation and amortization (EBITDA) comprises revenues less operating costs before financing costs, capital asset and intangible asset amortization, and income taxes.

The issuer does not currently have operations and therefore, there are no historical equivalent measures to compare them to, and a reconciliation of these non-IFRS financial performance measures cannot be carried out.

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1.0 Summary

1.1 Executive Summary

SLR Consulting (Canada) Ltd. (SLR) was retained by Roxmore Resources Inc. (Roxmore or the Company) to prepare an independent Technical Report on the Converse Project (the Project or the Property), located in the state of Nevada, USA. The purpose of this Technical Report is to disclose results of an updated Mineral Resource estimate, effective March 31, 2026, and a Preliminary Economic Assessment (PEA) of the Project. This Technical Report was prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). Two Qualified Persons (QP) from SLR visited the Property on November 4, 2025.

The Property is located in north-central Nevada within the Basin and Range physiographic province, one of Nevada's principal mining trends. The Property comprises 526 unpatented mining claims and seven fee land parcels, covering an area 5,365 hectares (ha) (13,275 acres). The claims span Humboldt and Perishing counties; the defined Mineral Resource lies entirely within Humboldt County.

1.1.1 Conclusions

The QPs conclude that the PEA demonstrates robustly positive economics at gold prices lower than current forecasts:

- after-tax NPV at a 5% discount rate of US$2,749 million
- after-tax IRR of 42.8%
- payback period of approximately 2.2 years from the start of production

The PEA is preliminary in nature, and is based, in part, on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Further conclusions by area are as follows:

1.1.1.1 Geology and Mineral Resources

- Effective March 31, 2026, within an optimized pit shell and above cut-off grades ranging from 0.18 grams per tonne (g/t) to 0.20 g/t gold (Au), the Converse Project Mineral Resource is estimated to total:
    - Indicated: 103 million tonnes (Mt) at an average grade of 0.65 g/t Au, totalling 2,162 thousand ounces (koz); and
    - Inferred: 218 Mt at an average grade of 0.43 g/t Au, totalling 3,035 koz.
- The deposit remains open at depth and along key structural trends, indicating potential for additional discoveries and resource growth.
- The Converse Project is characterized as a reduced gold skarn deposit.
- The Project area exhibits a well-constrained geological framework, characterized by Havallah sequence sedimentary units that are intruded by a central porphyritic stock. Gold mineralization is primarily localized parallel to bedding planes and further focused

by a series of southwest- and southeast-trending structural corridors that acted as conduits for hydrothermal fluid flow.

- Protocols for drilling, sample preparation and analysis, data verification, and security meet industry standard practices and are appropriate for use in a Mineral Resource estimate.
- Mineral Resource classification was based primarily on drill hole spacing, applied to designate contiguous zones of similar classification.

1.1.1.2 Mining Methods

- The mine plan demonstrates the technical viability of exploiting a 1.18 billion tonne deposit to deliver 299.8 Mt of mineralized material to the leach pad over a 14.5-year life of mine (LOM).
- Strategic scheduling is designed to maintain a consistent steady-state production rate of 22.5 Mtpa from Year 2 through Year 14, supported by a peak annual mining capacity of 90 Mtpa.
- Net present value (NPV) is optimized through a phased mining approach that front loads higher grade material, maintaining gold grades above the 0.51 g/t LOM average through Year 8.
- Economic resilience is supported using a $3,000/oz gold price and the adoption of conservative reporting cut-off grades between 0.18 g/t and 0.20 g/t Au, providing a significant buffer above the marginal breakeven limit.
- The conversion of approximately 93% of the optimized pit shell resource into the final 299.8 Mt production schedule demonstrates high design efficiency while accounting for necessary operational constraints such as ramp geometries and geotechnical berms.
- The utilization of an owner-operator model with 240-tonne trucks and large-scale shovels provides the necessary economies of scale to manage the 2.93:1 LOM strip ratio with high operational efficiency.
- The increase in the LOM strip ratio to 2.93:1—representing a 12.5% variance from the theoretical optimization—ensures that the Project’s cost model and 14.5 year schedule are grounded in a realistic, mineable geometry. Crucially, following the pre-production phase, the active mining schedule achieves a more efficient production strip ratio of 2.71:1.
- Geotechnical and logistical stability is maintained through engineered designs featuring overall slope angles (OSA) of 38° in alluvium and 45° in bedrock, alongside optimized haulage gradients of 10% in-pit and 8% ex-pit.
- All labour requirements and operating costs are referenced against the SHERPA 2025 Dataset, ensuring that the Project's economic model reflects current industry-standard benchmarks.
- Clear paths for further project optimization and de-risking have been established for the pre-feasibility stage, specifically regarding expanded geotechnical characterization of the Redline porphyry and deep pit hydrogeology.

1.1.1.3 Mineral Processing

- The distribution of metallurgical samples collected to date adequately covers the oxide, transition, and sulphide domains. In addition, the test work includes a balanced representation of both high-grade and low-grade mineralized zones. The QP notes, however, that there is insufficient sampling at the lower portion of the deposit and additional sampling is required to ensure comprehensive characterization of the deposit.
- Material from the Converse deposits is amenable to cyanide leaching for recovery of gold and silver values, providing overall good recoveries with moderate reagent requirements at a high pressure grinding roll (HPGR) crushed product size of 80% passing 6.3 mm. Based on the current minable resource, the overall average gold recovery is estimated at 70.9%.
- Mineralized material will be heap leached at a rate of 22.5 million tonnes per year (Mtpa) on a multi-lift heap that will be developed in three phases. Crushed material will be drum-agglomerated with cement to provide stability and pH buffering, then stacked and leached with a low-concentration cyanide solution. Pregnant solution from the heap leach will be processed in an Adsorption-Desorption-Recovery (ADR) recovery plant to produce doré bars.

1.1.1.4 Infrastructure

- Services are readily available in nearby towns including Lovelock, Winnemucca and Battle Mountain where skilled labour, suppliers, and experienced contractors can also be sourced.
- Grid electrical power is available on site via an existing transmission line that bisects the property. A substation drop will be placed at the north side of the mine and the existing powerline infrastructure will be rerouted around the mine perimeter.
- In 2019, Converse Resources LLC (CRL) purchased 2,560 acre-ft of irrigation water rights from the New Nevada Lands, LLC, which was subsequently converted to mining and milling use. The acquired water rights will support the construction and operation of a future mine at the Property.

1.1.1.5 Environment

- The Converse Project is in the very early stages of environmental and social permitting. With the exception of water rights and exploration permits, no environmental approvals have been secured.
- Because much of the Project area is situated on Bureau of Land Management (BLM) land, a Plan of Operations (PoO) will be required. Pre-Feasibility Study (PFS) level planning and at least one year of baseline studies will be required prior to submittal of a draft PoO to BLM.
- Due to the scope of the Project, a detailed Environmental Impact Statement (EIS) performed under the *National Environmental Policy Act* (NEPA) will likely be required for environmental project approval.
- Previous studies indicate a low potential for acid generating waste. Metallurgical work by Kappes, Cassiday & Associates (KCA) indicates relatively low sulphide content (0.5% to 5%).

- The BLM identifies mining as one of the primary threats contributing to sagebrush habitat loss for Greater Sage-Grouse in Nevada; however, the Project generally falls outside currently mapped Greater Sage-Grouse habitat, thus the need for habitat loss mitigation may be low or nonexistent.
- The existing water rights and aquifer yield may not be sufficient for operations and hydrological characterization work will form an important part of the next phase of work. As of the date of this writing, the water right is not fully vested under Nevada law because Proof of Completion and Proof of Beneficial Use filings have not been completed.

1.1.1.6 Capital and Operating Costs

- Mining capital and operating estimates are validated by the SHERPA 2025 Dataset and reconciled against regional operating benchmarks and recent PFS- or Feasibility Study (FS)-level studies to ensure cost competitiveness.
- Initial capital intensity is strategically managed through a 20% equipment down payment ($39 million) and a $116 million capitalized stripping program, providing the necessary pit access to reach the 22.5 Mtpa target by Year 2.
- Mining-specific infrastructure is budgeted at $38 million, covering truck shops, mine site haul roads, and fuel/explosive storage; this excludes general project infrastructure such as the main access road, site power substation, and administration buildings.
- The Project maintains a resilient $2.36/t LOM unit mining cost, which accounts for the transition from low cost alluvial stripping to the extended haulage cycles and increased fuel consumption associated with the 620 m deep bedrock pit.
- Long-term capital stability is supported by a 6 year lease amortization of the initial fleet ($193 million), followed by $177 million in staggered sustaining capital for replacements and expansions.
- Process plant capital costs are based on recent budgetary quotes from similar projects in KCA’s files for all major and most minor equipment. Where recent quotes were unavailable, reasonable cost estimates or allowances were provided based on cost guide data. All capital cost estimates were based on the purchase of equipment quoted new from the manufacturer or to be fabricated new.
- Process mechanical equipment installation estimates were based on the equipment type and included all installation labour, tools, and equipment usage at an average hourly installation rate of $125.00, based on KCA’s experience from recent projects.
- Process operating costs were developed by first principles and test work results based on KCA’s experience in this jurisdiction.

1.1.2 Recommendations

The Project should be advanced to the next stage of engineering study.

1.1.2.1 Geology and Mineral Resources

1 Continue exploration with emphasis on infill drilling inside the resource pit shell to support the conversion of Inferred to Indicated Mineral Resources.

2 Continue to refine the redox model by integrating relogged and newly collected geology data, cyanide solubility assays, and additional geochemical indicators.

3 Relogging efforts have further differentiated the prograde and retrograde skarn assemblages. Incorporate these updated classifications into a comprehensive skarn model to evaluate the spatial and temporal interactions between prograde and retrograde mineralizing processes and gold recovery within these domains.

4 Undertake careful logging of the alluvium with attention to the competency, hardness, and a generalized composition of the matrix and the cobbles, particularly if sulphides are present.

5 Continue collecting density measurements, with attention to the oxide zone and other areas of the deposit where sample coverage is limited. In intervals characterized by highly broken or porous rock, the use of a wax emulsion coating is recommended to ensure accurate volume determination.

6 Evaluate the database to identify samples lacking silver assays and those analyzed only by two-acid digestion. Using statistical and spatial criteria, select a representative subset for re-analysis using a four-acid digestion to generate complete silver values. Once a sufficiently robust silver dataset has been established, undertake a Mineral Resource estimate for silver.

1.1.2.2 Mining Methods

1 Execute an expanded geotechnical drilling program to characterize the Redline Porphyry and deep bedrock units at the 620 m ultimate pit floor. While the current pit design applies conservative engineering parameters to manage the limited five-hole dataset, this expanded program is required to optimize slope angles and reduce risk for a project of this billion-tonne scale.

2 Build a three-dimensional (3D) structural domain model using oriented core data to determine if the dominant west-dipping structural fabric persists or changes orientation across the wider pit footprint.

3 Conduct specialized stability modelling and detailed mapping for the sharp "nose" geometry located between the North and South Redline deposits to mitigate identified risks of tensile failure.

4 Prioritize a comprehensive hydrogeological investigation, including the installation of piezometers, to establish the phreatic surface, quantify the impact of pore water pressure on slope stability at depth, inform the dewatering requirements, and inform the site water balance.

5 Implement real-time in situ mapping and additional geotechnical testing of weak clay units during the early phases of alluvium excavation to refine conceptual slope stability and trafficability assumptions.

6 Quantify mining dilution and mineralized material loss through a trade-off study evaluating the relationship between the 29.8 m^3 hydraulic shovel selectivity, bench geometries, and contact mineralization to move beyond the 0% PEA assumption.

7 Perform advanced scheduling and phasing trade-off studies to balance vertical advance rates and optimize the sequence of pushbacks, specifically addressing the Year 11 feed shortfall and narrowing pit floor constraints.

8 Execute dedicated blast fragmentation and metallurgical trade-off studies to optimize hole diameters, confirm drill rig configurations, and refine the efficiency of wall-control drilling for final highwalls.

9 Conduct additional bulk density testing across all alluvial and bedrock material types to validate the 25% swell factor and the spatial capacity of the 507 Mm^3 waste rock storage facilities (WRSF).

10 Refine the mining fleet requirements by transitioning from SHERPA benchmarks to vendor-specific performance data and site-specific haulage simulations tailored to the 10% in-pit and 8% ex-pit gradients.

1.1.2.3 Mineral Processing

1 Complete additional metallurgical testing in the deeper areas of the deposit to ensure its comprehensive characterization.

2 Complete additional metallurgical test work on HPGR and conventionally crushed material.

3 Conduct variability testing to better characterize redox units.

4 Compacted permeability testing and Sulphidization-Acidification-Recycling-Thickening (SART) tests should be included in the test program.

5 Perform pulp agglomeration test work with high-grade and low-grade material to evaluate a potential hybrid flow sheet.

1.1.2.4 Infrastructure

1 Evaluate site-wide water balance and discharge requirements.

2 Confirm power requirements and supply availability.

3 Develop site-specific design criteria for the heap leach pad and perform field investigations and laboratory testing to support advancing the heap leach pad design.

4 Develop a PFS-level heap leach pad design that incorporates site specific data into the engineering calculations and analysis for such items as grading plan, slope stability analysis, water balance, settlement, and closure.

1.1.2.5 Environment

1 Commence baseline studies as soon as practical. Of urgency are hydrogeologic, surface water (if any), wildlife, and air quality studies, as these require at least one year of field data collection.

2 Conduct additional geochemical characterization using the updated mine plan and geologic block model.

3 Begin stakeholder engagement activities. Include federal, state and local regulatory agencies, tribal groups and NGOs. Consider commissioning a stakeholder engagement study.

4 Perform an updated cultural resource survey of the entire Project area of influence.

5 Identification of a significant wildlife or cultural resource that requires mitigation. Early commencement of baseline and other studies can reduce this risk.

6 Finalize Proof of Completion and Proof of Beneficial Use for existing water rights.

1.1.2.6 Capital and Operating Costs

1. Transition from benchmarked estimates to a formal Request for Proposal (RFP) with OEMs to secure binding quotes and guaranteed maintenance terms for the primary 240 t haulage and shovel fleet.
2. Conduct a formal RFP for contractor mining services as an alternative to the owner mining model.
3. Conduct site-specific geotechnical testing on the alluvium to maximize "free-dig" zones, potentially reducing the initial drilling and blasting capital and operating requirements.
4. Perform localized studies to confirm that the assumed diesel ($0.90/L), power ($0.08/kWh), and hourly labour rates align with current regional market conditions to de-risk the LOM unit cost.
5. Perform a trade-off study on autonomous haulage (AHS) to determine if the long-term operating cost savings justify the higher initial capital outlay, given the Project's scale and 14.5 year life.
6. Advancement to the next level of study should include the development of SRCE-level inputs and a site-specific closure plan to further refine this estimate and timeline, as part of permitting and financial assurance requirements in Nevada.

1.1.3 Risks

- Metallurgical recoveries
  - Confirm metallurgical recoveries with more variability test work for all mining areas.
  - Confirm HPGR heap leach recoveries versus conventional crushing.
- Copper in pregnant leach solution may require removal with a SART plant to recover cyanide; conduct more testing to understand copper implications, if any.
- Water use and water quality protection will be key environmental issues under scrutiny by BLM and NDEP. It is currently unclear whether the existing water rights and effective aquifer yield will be sufficient for operational demand.
- Proof of Completion and Proof of Beneficial Use for the existing water rights may not have been completed, putting the rights at risk of being forfeit or cancelled.

1.1.4 Opportunities

- Geotechnical Slope Refinement: Utilize high-density data in future PFS/FS studies to safely steepen inter-ramp pit slopes, reducing the LOM strip ratio and lowering waste mining costs.
- Optimized Phase Scheduling: Refine the current 3-phase pit sequencing during subsequent PFS/FS stages to further accelerate high-grade material extraction, optimize waste deferral, and maximize project NPV ahead of reserve declaration.
- Evaluation of pulp agglomeration processing to explore the potential to improve metal recovery and overall project economics.

- Silver as a by-product:
  - The current economic model does not include silver, so it is not treated as contributing to project revenues.
  - While silver is less valuable than gold, recovering it as a by-product could improve the Project's economics, and its processing would likely not require additional infrastructure.
- Early Stakeholder Engagement: Begin engaging with stakeholders (NDEP, BMRR, BLM, Humboldt County officials, NGOs, and interested indigenous organizations). A Stakeholder Engagement Study may be a beneficial approach at this stage.
- Exploration Permitting: Notice level exploration permitting with BLM for areas of potential interest beyond the existing permitted exploration limits.

1.2 Economic Analysis

The economic analysis in this Technical Report was prepared in accordance with paragraphs 2.3(1)(b) and (c), subsections 2.3(3) and (4), and paragraph 3.4(e), of the Instrument, including any required cautionary language, which is stated with equal prominence.

The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this Preliminary Economic Assessment is based will be realized.

SLR notes that the economic analysis presented in this section is based on revenue from gold only. An after-tax Cash Flow Projection has been generated from the LOM production schedule and capital and operating cost estimates, and is summarized in Table 1-1.

A summary of the key parameters and assumptions for the economic analysis is provided below.

1.2.1 Economic Criteria

1.2.1.1 Revenue

- Mine life: Starts with 9 months of mine pre-production development, followed by 13.75 years of production.
- Peak mining rate of 90 Mtpa per year (mineralized material and waste).
- Total LOM mineralized material feed to process: 299,795 kt, with an average of 59,000 tpd processed (considering full production years) at LOM average grade of 0.51 g/t Au.
- Contained gold: 4,910 koz Au.
- Average LOM process gold recovery: 70.9%.
- Recovered gold over the LOM totals 3,482 koz Au, with an average gold production of 246 koz/yr (years of full production).
- Gold payable is assumed at 99.5% based on the benchmark from nearby projects.
- Annual average gold sales of 245 koz Au payable per year (full production years).

- Gold price is based on analyst consensus price forecasts from March 2026. For the economic analysis was assumed:
  - Y-1 and Y1: US$4,000/oz Au
  - Y2 to Y14: US$3,600/oz Au
- Transportation, logistic and refining charges assumed at US$1.14/oz Au, based on benchmark from nearby projects.
- NSR royalty of 6% payable to Royalty Consolidation Company, LLC (RCC Royalty).
- Revenue is recognized at the time of gold production. Gold production starts in the pre-production period.
- LOM net revenue is US$11,803 million (after royalty, and transportation and refining charges).

1.2.1.2 Costs

- Pre-production capital costs were estimated at $829 million, including a 25% contingency. See Table 1-1 for the pre-production capital costs breakdown.
- Mine life sustaining capital costs were estimated at $514 million.
- Mine equipment lease. Roxmore plans to use a mine equipment lease strategy to obtain the initial equipment for the pre-production period. This strategy assumes a 20% downpayment and lease payments at a 6% annual rate over 6 years. Annual lease payment was estimated at US$31.4 million.
- Mine equipment additions and replacement over the LOM were estimated by SLR at US$177 million.
- Process sustaining estimated by KCA at US$148 million over the LOM.
- Capitalized operating costs for processing and administrative activities during pre-production.
- Final reclamation costs assumed at US$100 million at the end of the LOM.
- Average LOM operating cost is US$16.01/t processed.
- Open pit operating costs of US$2.36/t mined ($8.75/t processed).
- Processing operating costs of US$5.96/t processed.
- Site services and general and administrative (G&A) costs of US$27.9 million/yr (LOM average of $1.29/t processed).
- Total All-in Sustaining Cost (AISC) is approximately US$1,769/oz Au.

1.2.1.3 Taxation and Royalties

- The Project is subject to a state income tax rate of 5.0% and a federal income tax rate of 21.0%.
- LOM taxes total approximately US$974 million.
- The entire Property is subject to an NSR production royalty payable to RCC Royalty on the sale of any minerals from the Property. The RCC Royalty rate is 6%, except as to

those portions of the Property that were subject, as of the date of the RCC Royalty grant, to existing royalty obligations, in which case the RCC Royalty rate is the difference between 6% and the rate of the existing royalty obligations. Effectively, the RCC Royalty means that the Property is subject to a blanket 6% NSR royalty on the production of all minerals.

1.2.2 Cash Flow Analysis

SLR developed, for the Converse Project, an unlevered after-tax cash flow model that consolidates physicals, costs, and revenue. The inputs for the cash flow model for Converse were developed by the following considerations:

- Production schedule by SLR.
- Metallurgy assumptions by KCA.
- Revenue is based on analyst consensus price forecasts from March 2026.
- Capital and operating costs were developed by KCA (processing) and SLR (others).

All costs are in Q1 2026 US$ dollars with no allowance for inflation.

The Converse unlevered after-tax cash flow model excludes the following components:

- Escalation beyond the date of our source inputs.
- Effect of tariffs on non-US products or supplies.
- Financing costs.
- Insurance.
- Overhead cost for a corporate office.

The annual after-tax cash flow summary is presented in Table 1-1.

Table 1-1: After-Tax Cash Flow Summary

Economic Model Annual Summary																						
SLR	Company	Roxmore Resources Inc.																				
	Project Name	Converse OP Project																				
	Scenario Name	Base Case																				
	Analysis Type	PEA & NI-43-101																				
Calendar Year				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Project Stage				Pre-Prod.	Pre-Prod.	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Closure	Closure	Closure
Time Until Closure in Years		US$ & Metric Units	LoM Avg / Total	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2	-3
Market Prices																						
Gold, Forecast		US$/oz	$3,626	-	4,000	4,000	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600
Physicals																						
Total Mineralized Material Mined		kt	399,795	-	2,623	13,949	22,500	23,500	31,602	22,500	22,500	23,103	22,500	22,500	22,500	18,434	22,500	22,500	16,585	-	-	-
Total Waste Mined		kt	877,578	-	72,377	71,051	62,500	55,745	63,425	67,500	67,500	66,897	67,500	65,500	66,613	71,566	45,392	26,765	7,248	-	-	-
Total Material Mined		kt	1,177,373	-	75,000	85,000	85,000	79,245	95,027	90,000	90,000	90,000	90,000	88,000	89,113	90,000	67,892	49,265	23,833	-	-	-
Stripping Ratio		W:O	2.93	-	27.59	5.09	2.78	2.37	2.94	3.00	3.00	2.90	3.00	2.91	2.96	3.88	2.02	1.19	0.44	-	-	-
Total Mineralized Material Processed		kt	399,795	-	2,623	13,949	22,500	23,500	31,602	22,500	22,500	23,103	22,500	22,500	22,500	18,434	22,500	22,500	16,585	-	-	-
Gold Grade, Processed		g/t	0.51	-	0.81	0.82	0.57	0.59	0.51	0.51	0.52	0.55	0.53	0.47	0.31	0.34	0.41	0.59	0.61	-	-	-
Contained Gold, Processed		koz	4,910	-	68	276	411	449	352	367	378	407	385	338	226	202	297	430	328	-	-	-
Average Recovery, Gold		%	70.9%	-	61.7%	64.5%	63.7%	64.2%	69.5%	76.0%	76.5%	75.0%	74.9%	72.0%	71.1%	66.8%	71.1%	72.4%	71.0%	-	-	-
Recovered Gold, Processed		koz	3,482	-	42	178	262	288	244	282	287	305	288	246	161	136	212	311	234	-	-	-
Finished Goods - Gold		koz	3,482	-	42	178	262	288	244	282	287	305	288	246	161	136	212	311	234	-	-	-
Payable Gold, Total		koz	3,465	-	42	177	260	288	243	281	286	305	287	245	160	136	211	310	233	-	-	-
Cash Flow																						
Gold Gross Revenue	100.00%	$000s	12,580,665	-	166,025	707,996	937,754	1,031,297	875,263	1,010,302	1,029,381	1,102,234	1,033,117	882,088	576,606	495,459	758,078	1,114,924	839,241	-	-	-
Total Gross Revenue		$000s	12,580,665	-	166,025	707,996	937,754	1,031,297	875,263	1,010,302	1,029,381	1,102,234	1,033,117	882,088	576,606	495,459	758,078	1,114,924	839,241	-	-	-
Mining Cost		$000s	(2,801,418)	-	-	(139,669)	(173,536)	(172,302)	(189,295)	(187,830)	(192,390)	(212,130)	(220,231)	(210,196)	(224,372)	(234,487)	(172,573)	(176,145)	(90,193)	-	-	-
Process Cost		$000s	(1,772,036)	-	-	(83,176)	(134,168)	(140,131)	(128,011)	(134,166)	(134,167)	(137,762)	(134,168)	(134,167)	(134,168)	(109,920)	(134,168)	(134,168)	(98,596)	-	-	-
G&A Cost		$000s	(383,825)	-	-	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(20,925)	-	-	-
Refining and Freight Cost		$000s	(3,963)	-	(193)	(245)	(277)	(287)	(270)	(285)	(287)	(294)	(287)	(271)	(239)	(230)	(258)	(296)	(266)	-	-	-
Royalties		$000s	(753,401)	-	(9,998)	(42,485)	(56,249)	(61,981)	(52,501)	(60,601)	(61,748)	(66,118)	(61,970)	(52,909)	(34,507)	(29,714)	(45,469)	(66,879)	(50,330)	-	-	-
Subtotal Cash Costs Before By-Product Credits		$000s	(5,514,464)	-	(10,191)	(293,475)	(392,131)	(402,560)	(398,747)	(410,793)	(416,489)	(444,202)	(444,555)	(401,443)	(421,285)	(402,251)	(380,366)	(405,388)	(259,618)	-	-	-
By-Product Credits		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Cash Costs After By-Product Credits		$000s	(5,514,464)	-	(10,191)	(293,475)	(392,131)	(402,560)	(398,747)	(410,793)	(416,489)	(444,202)	(444,555)	(401,443)	(421,285)	(402,251)	(380,366)	(405,388)	(259,618)	-	-	-
Operating Margin	56%	$000s	7,066,201	-	155,834	414,521	545,622	628,737	476,516	599,519	612,892	658,032	588,562	480,644	155,421	93,208	377,712	709,539	579,623	-	-	-
Other Admin Expenses		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EBITDA		$000s	7,066,201	-	155,834	414,521	545,622	628,737	476,516	599,519	612,892	658,032	588,562	480,644	155,421	93,208	377,712	709,539	579,623	-	-	-
Depreciation Allowance		$000s	(1,342,588)	-	(9,979)	(40,665)	(74,945)	(85,436)	(78,704)	(96,558)	(106,834)	(119,320)	(117,469)	(103,794)	(70,677)	(63,609)	(102,522)	(150,781)	(113,496)	-	-	-
Earnings Before Taxes		$000s	5,703,613	-	145,855	373,856	470,677	543,302	397,832	502,961	506,058	538,712	471,093	376,851	84,745	29,600	275,190	558,757	466,126	-	-	-
Federal & State Income Tax		$000s	(974,062)	-	(104)	(66,573)	(64,444)	(99,273)	(68,466)	(89,946)	(89,599)	(95,632)	(81,192)	(55,507)	(10,573)	(3,693)	(41,991)	(100,187)	(86,524)	-	-	-
Net Income		$000s	4,729,551	-	145,471	300,984	406,233	444,030	329,366	413,016	414,659	443,080	389,900	291,343	74,170	25,907	233,200	458,571	379,601	-	-	-
Non-Cash Add Back - Depreciation		$000s	1,342,588	-	9,979	40,665	74,945	85,436	78,704	96,558	106,834	119,320	117,469	103,794	70,677	63,609	102,522	150,781	113,496	-	-	-
Working Capital		$000s	(0)	(6,101)	(16,743)	(17,118)	(6,294)	(4,340)	4,066	(6,662)	(2,949)	(2,496)	1,923	5,496	12,307	1,173	(11,730)	(14,525)	4,556	61,437	-	-
Operating Cash Flow		$000s	6,072,118	(6,101)	139,707	330,830	484,285	525,134	412,134	502,916	526,544	559,904	509,292	400,633	157,155	90,689	323,991	594,827	496,657	61,437	-	-
Initial Capital		$000s	(826,907)	(270,045)	(556,941)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalized Opex		$000s	(29,593)	-	(29,593)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$000s	(513,601)	-	-	(83,460)	(70,130)	(31,404)	(68,756)	(49,999)	(76,001)	(16,500)	(31,066)	(16,500)	(16,500)	(16,690)	(10,596)	-	-	-	-	-
Closure/Reclamation/Monitoring Costs		$000s	(100,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(40,000)	(60,000)	-
Total Capital		$000s	(1,472,100)	(270,045)	(586,534)	(83,460)	(70,130)	(31,404)	(68,756)	(49,999)	(76,001)	(16,500)	(31,066)	(16,500)	(16,500)	(16,690)	(10,596)	-	-	(40,000)	(60,000)	-
Cash Flow Adj./Reimbursements		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LoM Metrics																						
Economic Metrics																						
Discount Rate	BOP	5%		0.9524	0.9070	0.8638	0.8227	0.7835	0.7462	0.7107	0.6768	0.6446	0.6139	0.5847	0.5568	0.5303	0.5051	0.4810	0.4581	0.4363	0.4155	0.3957
a) Pre-Tax																						
Free Cash Flow		$000s	5,574,020	(270,146)	(448,643)	313,943	461,198	592,993	411,844	542,857	531,942	637,036	559,419	437,640	149,229	75,690	347,357	695,044	583,101	21,437	(60,000)	-
Cumulative Free Cash Flow		$000s		(270,146)	(726,709)	(412,846)	48,352	641,344	1,053,188	1,596,046	2,127,988	2,765,024	3,324,443	3,762,083	3,911,311	3,987,001	4,334,358	5,029,402	5,612,503	5,634,020	5,574,020	5,574,020
NPV @ 5%		$000s	3,391,119	(264,901)	(406,932)	271,196	379,429	464,626	307,324	385,789	360,039	410,659	343,434	255,879	83,096	40,140	175,464	334,314	267,182	9,353	(24,931)	-
Cumulative NPV @ 5%		$000s		(264,901)	(406,932)	8,294	365,723	850,349	1,157,673	1,543,471	1,903,511	2,314,150	2,657,584	2,913,464	2,996,560	3,036,700	3,212,154	3,546,467	3,813,629	3,822,982	3,798,051	3,798,051
IRR		%	60.5%																			
Payback Period		years	1.9			1.0	0.8	-														
b) After-Tax																						
Free Cash Flow		$000s	4,599,936	(270,146)	(448,027)	247,070	376,754	493,720	343,378	452,911	442,543	541,405	478,226	382,133	138,657	71,997	306,397	594,827	496,657	21,437	(60,000)	-
Cumulative Free Cash Flow		$000s		(270,146)	(720,974)	(479,903)	(103,149)	390,571	733,949	1,186,860	1,629,203	2,170,608	2,648,834	3,030,967	3,169,623	3,241,621	3,547,017	4,141,844	4,638,501	4,659,938	4,599,938	4,599,936
NPV @ 5%		$000s	2,749,445	(264,901)	(407,100)	213,429	309,957	386,840	256,234	321,876	299,395	348,994	293,509	223,426	77,209	38,182	154,348	286,122	227,824	9,353	(24,931)	-
Cumulative NPV @ 5%		$000s		(264,901)	(672,001)	(458,572)	(148,616)	238,227	494,461	816,336	1,115,731	1,464,726	1,758,315	1,981,740	2,058,950	2,097,131	2,251,377	2,537,499	2,765,023	2,774,376	2,749,445	2,749,445
IRR		%	42.0%																			
Payback Period		years	2.2			1.0	1.0	0.3														
Operating Metrics																						
Mine Life		Years	14																			
Average Daily Mining Rate		t/d mined	241,600	-	205,482	242,857	242,857	220,413	242,934	257,143	257,143	257,143	257,143	251,429	254,607	257,143	193,976	140,758	91,604	-	-	-
Average Daily Processing Rate		t/d placed	59,000	-	9,539	38,219	61,644	64,384	59,103	61,644	61,644	63,295	61,644	61,644	61,644	50,503	61,644	61,644	60,309	-	-	-
Mining Cost		$ / t mined	$2.38	-	-	1.64	2.04	2.18	2.23	2.09	2.14	2.36	2.45	2.48	2.52	2.61	2.54	3.58	3.78	-	-	-
Mining Cost		$/t process	$8.75	-	-	10.01	7.71	7.34	8.76	8.35	8.55	9.18	9.79	9.61	9.97	12.72	7.67	7.83	5.44	-	-	-
Processing Cost		$/t process	$5.96	-	-	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	-	-	-
G&A Cost		$/t process	$1.29	-	-	2.00	1.24	1.19	1.29	1.24	1.24	1.21	1.24	1.24	1.24	1.51	1.24	1.24	1.26	-	-	-
Subtotal Direct Operating Costs		$/t process	$16.01	-	-	17.98	14.92	14.49	16.02	15.55	15.75	16.35	16.99	16.81	17.18	20.20	14.87	15.03	12.66	-	-	-
Refining and Freight Cost		$/t process	$0.01	-	0.07	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02	-	-	-
NSR Royalty		$/t process	$2.54	-	3.01	3.04	2.50	2.63	2.43	2.69	2.74	2.86	2.75	2.35	1.54	1.61	2.02	2.97	3.04	-	-	-
Total Operating Cost		$/t process	$18.56	-	3.88	21.04	17.43	17.13	18.46	18.26	18.51	19.23	19.76	19.19	18.72	21.82	16.91	18.02	15.71	-	-	-
Sales Metrics																						
Au Sales		koz	3,465	-	42	177	260	288	243	281	288	306	287	245	160	136	211	310	233	-	-	-
Total Cash Cost (LOM)		$ / oz Au	1,593																			
Total AISC (LOM)		$ / oz Au	1,768			2,130	1,805	1,515	1,923	1,642	1,739	1,511	1,657	1,836	2,745	3,059	1,895	1,309	1,118	-	-	-
Avg. LOM Annual Au Sales (excl. rinsing phase)		koz/yr	248																			

SLR prepared a LOM unlevered after-tax cash flow model to confirm the economics of the Converse Project over the LOM. Economics have been evaluated using the discounted cash flow method, considering annual processed tonnages and the associated gold grades. The process gold recovery, gold price forecast, operating costs, refining and transportation charges, royalties, and initial and sustaining capital expenditures were also considered.

The economic analysis demonstrates that the Project's Mineral Resources have reasonable prospects for economic extraction at the LOM net average realized prices of $3,625/oz Au, with a long-term price of $3,600/oz Au, and that further advancement of Project studies is warranted.

The base discount rate assumed in this Technical Report is 5% as per standard industry practice for evaluating precious-metal projects in North America. Discounted present values of annual cash flows are summed to arrive at the Project's Base Case NPV.

The Project's pre-tax NPV at a 5% discount rate is approximately US$3,391 million, and the pre-tax internal rate of return (IRR) is approximately 50.3%. The Project's after-tax NPV at a 5% discount is approximately US$2,749 million, the after-tax IRR is approximately 42.8%, and the payback period is approximately 2.2 years from the start of production.

The Project's undiscounted pre-tax cash flow is approximately US$5,574 million, and the undiscounted after-tax cash flow is approximately US$4,600 million.

SLR has also run a stand-alone economic analysis for the Project using flat resource metal prices of US$3,000/oz for gold, and the analysis demonstrates that the Project's Mineral Resources also have reasonable prospects for economic extraction at these prices.

The World Gold Council Adjusted Operating Cost (AOC) is US$1,592/oz Au. The mine life sustaining capital unit cost is US$177/oz Au, for an AISC of US$1,769/oz Au.

The Project's average annual gold sales during the LOM are approximately 245 koz Au per year, considering years of full production.

1.2.3 Sensitivity Analysis

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Gold price
- Gold recovery
- Gold head grade
- Operating costs
- Capital costs (Initial, Sustaining, and Closure costs)

Where possible, the after-tax NPV 5% and IRR sensitivities relative to the base case have been calculated for -20% to +20% variations in head grade and recovery, and -30% to 40% in gold price. Operating and capital cost sensitivities have been calculated at -15% to +35% variations. The sensitivities are shown in Table 1-2 and Figure 1-1 and Figure 1-2.

Table 1-2: After-Tax Sensitivity Analyses

Variance	Head Grade (g/t Au)	NPV at 5% (US$ million)	IRR (%)
80%	0.41	1,534	28.3%
90%	0.46	2,143	35.7%
100%	**0.51**	**2,749**	**42.8%**
110%	0.56	3,358	49.5%
120%	0.61	3,967	56.1%
Variance	**Recovery (% Au)**	**NPV at 5% (US$ million)**	**IRR (%)**
80%	57%	1,534	28.3%
90%	64%	2,143	35.7%
100%	**71%**	**2,749**	**42.8%**
110%	78%	3,358	49.5%
120%	85%	3,967	56.1%
Variance	**Metal Prices (US$/oz Au)**	**NPV at 5% (US$ million)**	**IRR (%)**
69%	$2,500	884	19.6%
85%	$3,060	1,840	32.1%
100%	**$3,600**	**2,749**	**42.8%**
120%	$4,320	3,967	56.1%
139%	$5,000	5,111	68.1%
Variance	**Operating Costs (US$/t)**	**NPV at 5% (US$ million)**	**IRR (%)**
85%	$13.61	3,226	48.3%
93%	$14.81	2,988	45.5%
100%	**$16.01**	**2,749**	**42.8%**
118%	$18.81	2,193	36.2%
135%	$21.61	1,637	29.3%
Variance	**Capital Costs (US$000)**	**NPV at 5% (US$ million)**	**IRR (%)**
85%	$1,251	2,934	51.4%
93%	$1,362	2,842	46.7%
100%	**$1,472**	**2,749**	**42.8%**
118%	$1,730	2,534	35.3%
135%	$1,987	2,319	29.7%

Figure 1-1: After-Tax NPV 5% Sensitivity Analysis

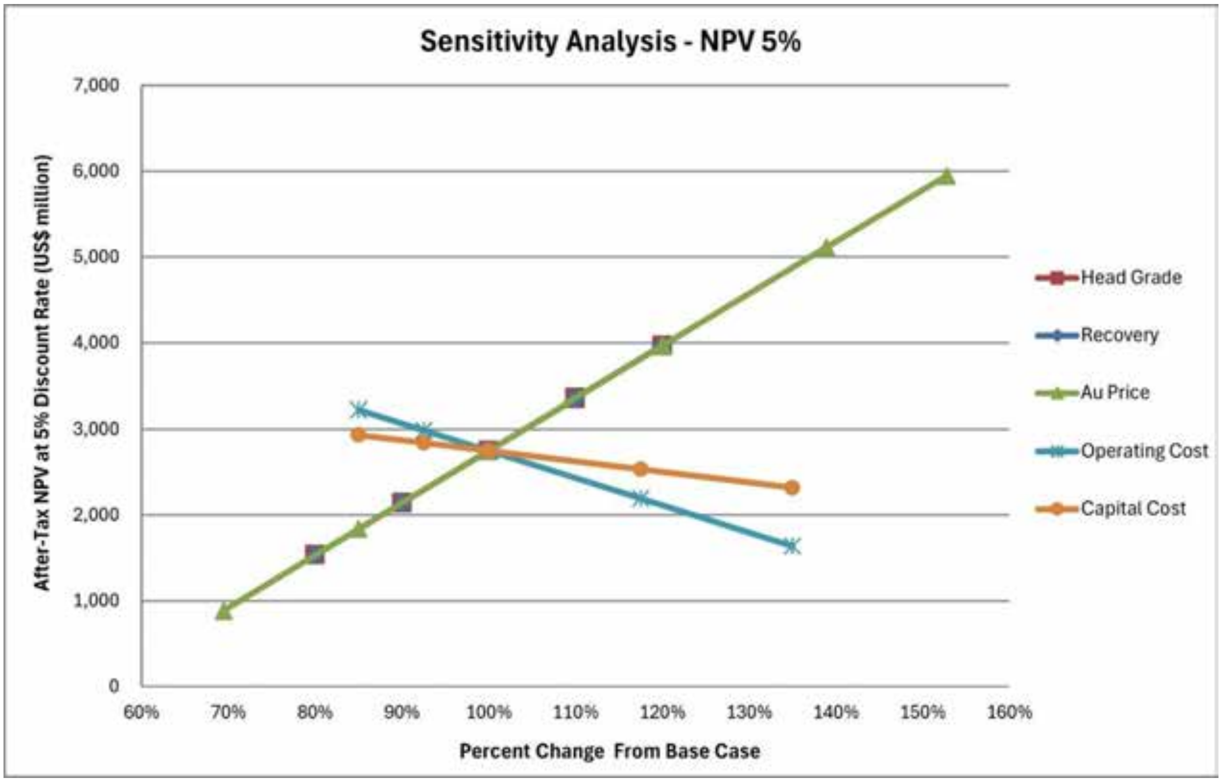
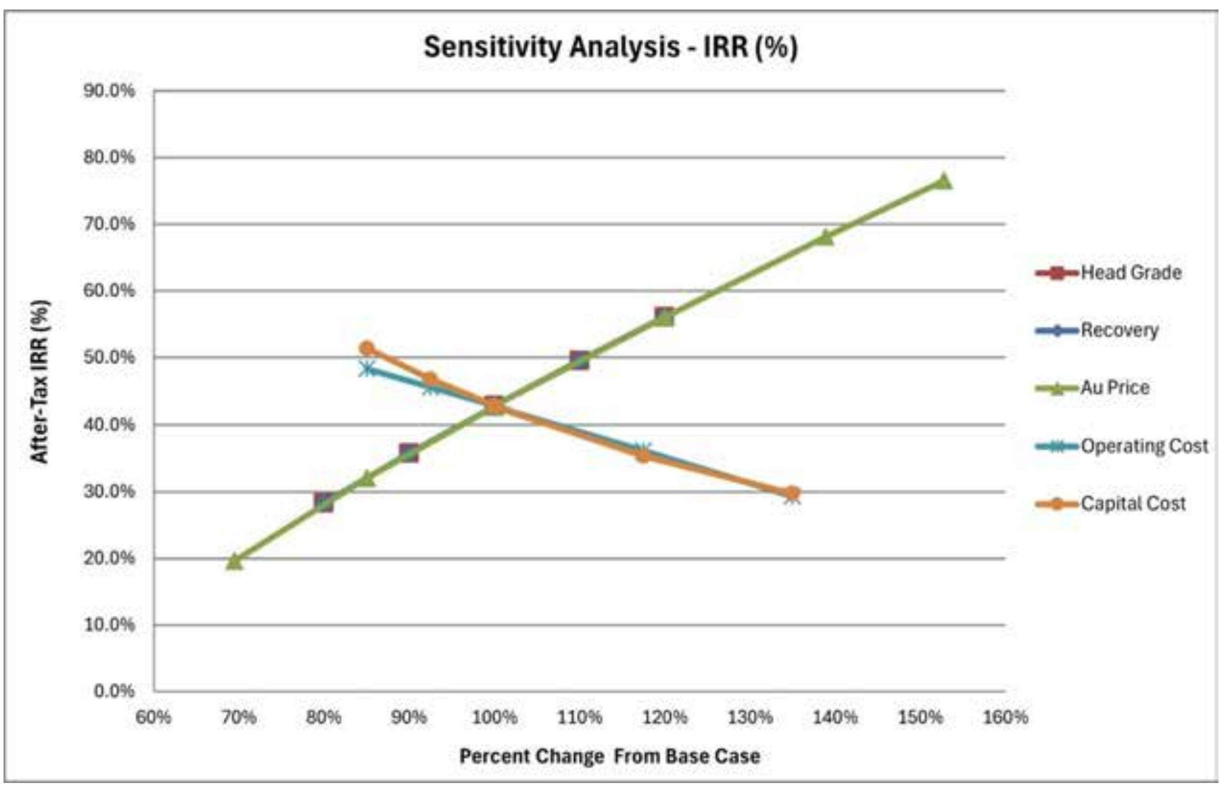


Figure 1-2: After-Tax IRR Sensitivity Analysis



1.3 Technical Summary

1.3.1 Property Description and Location

The Property is in Buffalo Valley in southeastern Humboldt County, Nevada, with a small portion extending into Pershing County. It lies approximately 17 km south of the Valmy I-80 exit, 40 km west-northwest of Battle Mountain, 80 km southeast of Winnemucca, and 330 km northeast of Reno. The site is located at approximately 40°43' N, 117°16' W within Township 33N, Range 42E.

1.3.2 Land Tenure

The Property represents a consolidated package of 526 unpatented mining claims (lode and placer) and seven fee land parcels, representing a total land area of 13,275.3 acres.

The unpatented mining claims are administered by the BLM and give the holder rights to extract minerals from hard rock (lode) or unconsolidated ground (placer) from federal public land; however, the federal government retains ownership of the surface, and annual maintenance (assessment work or fees) is required. For the current year, annual maintenance fees at Converse total $215,523. All claims are in good standing.

Through its subsidiary, Roxmore holds title to 276 of the unpatented claims, and all fee land parcels. The remainder of the claims (250) are held by Nevada North Resources (USA) (NNR) and leased to Roxmore, requiring annual advance royalty payments from $50,000 to $100,000, fully creditable against future net smelter return (NSR) payments. There are three active royalty agreements at the Project, effectively encumbering the Property with a total NSR royalty of 6% on the production of all minerals.

The fee land parcels (4,436.5 acres) give private ownership of both surface and subsurface rights with no maintenance or fee obligations outside of property tax. One fee land parcel is currently leased to Nevada Gold Mines (NGM), consequently the mineral rights are not currently available to Roxmore.

1.3.3 History

Exploration of the Property began in 1988 when NNR staked the original claim block. Early drilling was limited, with Kennecott completing two reverse circulation (RC) holes that same year. From 1989 to 1992, Chevron Resources and later Cyprus Mines Corp. advanced the project through geological mapping, geochemical sampling, geophysical surveys, and multiple RC drilling campaigns, several of which returned anomalous gold values.

Independence Mining Co. continued work in 1993 and 1994, completing a bulk leach extractable gold (BLEG) survey and mud-rotary drilling that intersected notable gold mineralization. Through the mid-1990s, Uranerz U.S.A. Inc. and Romarco Nevada Inc. formed the Nike Joint Venture, expanding geophysical coverage, drilling extensively, and discovering several significant gold intercepts. Additional agreements with Santa Fe Pacific Gold (later acquired by Newmont USA Ltd.) facilitated further drilling, metallurgical testing, and the development of early block models and historical resource estimates through 1999.

From 2001 to 2009, Metallic Ventures Gold Inc. (and later its successors) advanced the project with additional drilling, Controlled Source Audio Magnetotellurics (CSAMT) surveys, and multiple rounds of metallurgical test work. Several Mineral Resource estimates were prepared during this time period and disclosed within Technical Reports prepared in accordance with NI

43-101. Drilling by Newcrest Resources also occurred between 2003 and 2005, though details are limited.

International Minerals Corp. acquired the project in 2010 and undertook substantial core and RC drilling from 2011 to 2012, culminating in an updated Mineral Resource estimate and a PEA. Following corporate transactions in 2013 and 2014, the Property was transferred to Chaparral Gold Corp. and subsequently to CRL. CRL completed metallurgical drilling in 2017, followed by comprehensive metallurgical testing in 2018 and the acquisition of water rights in 2019.

Most recently, in 2025, Axcap Ventures Inc.(Axcap) acquired CRL and the Property, and subsequently published an amended and restated NI 43-101 Technical Report by Apex Geoscience. The report was originally drafted by CRL prior to the acquisition and finalized by Axcap before its release. Later in 2025, Axcap rebranded as Roxmore Resources. Operations and permits for the Property continue to be held and administered by CRL, which now operates as a wholly owned subsidiary of Roxmore.

The estimates prepared by previous owners are considered to be historical in nature and should not be relied upon; however, they do give an indication of mineralization at the site, as well as the nature of exploration and study work completed on the Property.

1.3.4 Geology and Mineralization

The Converse Property contains two gold-rich skarn deposits, North Redline and South Redline, which are concealed beneath alluvium of 20 m to 275 m depth. Beneath this cover, drilling has confirmed a thick (up to 800 m) package of Mississippian to Permian-aged Havallah sequence sediments dipping 20° to 35° west, which form the primary host unit to the Redline deposits.

This sedimentary package is intruded by the Redline porphyry stock (Main Stock) and associated dykes and sills. The Main Stock is a 450 m to 600 m wide dacite-to-granodiorite intrusion that grades into tonalite along its southern margin and is regionally dated at approximately 41 Ma.

Gold mineralization is primarily hosted within the favourable units of the Havallah sequence, with the highest grades commonly along southeast or southwest trending faults and in late-stage breccia structures.

At North Redline, gold mineralization covers an area with dimension approximately 670 m long by 200 m to 350 m wide, extending from 350 m to 650 m below surface. South Redline has a similar footprint, approximately 670 m by 260 m, and reaches comparable depths. Gold is interpreted to be associated with retrograde stages of skarn alteration and is accompanied by silver and, to a lesser extent, copper. Notable drill results include 194.5 m averaging 0.71 g/t Au in hole CV25-007C from North Redline and 134.1 m averaging 1.0 g/t Au in hole CNR-MT17-004 from a breccia zone at South Redline. Sulphide minerals—including pyrrhotite, chalcopyrite, pyrite, sphalerite, and molybdenite—formed during both prograde and retrograde skarn development. Minor galena, arsenopyrite, and bismuth-tellurium minerals are also present; however, not in economically significant quantities.

1.3.5 Exploration Status

Roxmore completed seven drill holes in 2025, including two RC holes and five diamond drill holes. These drill holes tested both North and South Redline deposits and were instrumental in defining new mineralized breccia zones in both the north and south areas.

Prior to Roxmore’s ownership, various owners and operators completed 84,129 m of drilling from 348 holes between 1989 and 2017. Because the Property is largely concealed beneath alluvial cover, conventional soil and rock chip sampling is not helpful. Prior operators conducted a variety of geophysical surveys, including gravity, gradient array induced polarization (IP), ground and airborne magnetics, and CSAMT.

1.3.6 Mineral Resources

The Mineral Resource estimate at the Converse Project was completed by SLR between November 2025 and March 2026 and has an Effective Date of March 31, 2026 (Table 1-3).

A series of low and high grade mineralization shapes define the estimation domains for the North and South Redline deposits and are constructed in Leapfrog Geo based on geological and gold grade trend interpretations. Within each domain, gold assays are capped, composited, and interpolated into 10 m cubic blocks using a multi-pass ordinary kriging (OK) method in Leapfrog Edge.

Classification is assigned based on drill hole spacing criteria referencing observed and modelled continuity, adjusted as needed to ensure consistent and geologically coherent class volumes.

The Mineral Resource estimate is constrained within optimized pit shells developed considering a long-term gold price of US$3,000/oz, variable gold recoveries, and area and redox domain dependent cut-off-grades of 0.18 g/t or 0.20 g/t Au.

Validation of the block model includes statistical comparisons with composites and nearest neighbour (NN) estimates, swath plots, visual reviews in three-dimensional (3D) and longitudinal, cross-sectional and plan views.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 1-3: Summary of Mineral Resources for the Converse Project – March 31, 2026

Classification	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	103,102	0.65	2,162
Inferred	218,446	0.43	3,035

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported within an optimized pit shell at a cut-off grade of 0.18 g/t Au or 0.2 g/t Au, dependent on zone and redox domain (See Table 14-13).
3. Mineral Resources are estimated using a long-term gold price of US$3,000 per ounce.
4. Gold recoveries are based on equations derived from metallurgical studies.
5. Recoveries are variable and based on equations derived from metallurgical testing.
6. Bulk density is variable by rock type and redox domain and ranges from 2.62 t/m^3 to 2.71 t/m^3. Overburden was assigned a value of 1.8 t/m^3.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. A minimum mining width of 50 m was used.
9. Numbers may not add or multiply correctly due to rounding.

1.3.7 Mining Method

The Project is designed as a high capacity open pit operation utilizing a conventional truck-and-shovel fleet to extract 299.8 Mt of mineralized material over a 14.5 year LOM. The mining strategy is centered on a steady-state production profile of 22.5 Mtpa, supported by a peak total material movement of 90 Mtpa. To optimize the Project's economic performance and accelerate capital payback, the mine plan utilizes a phased sequencing strategy (three pushbacks) that front loads higher grade material, maintaining gold grades above the 0.51 g/t LOM average throughout the first 8 years of operation.

The open pit design incorporates specific geotechnical parameters to manage the depth of the alluvium and the underlying bedrock, featuring 10 m benches with OSAs of 38° in alluvium and 45° in bedrock. Final pit limits were determined using a Whittle™ Pseudoflow algorithm at a base gold price of $3,000/oz. The final engineered pit design achieves a 93.2% reconciliation of mineralized material relative to the optimal pit shell, with the 6.8% variance attributed to the practical incorporation of a 30 m wide dual-lane ramp system and geotechnical berms. This design captures a total of 877.6 Mt of waste, resulting in a LOM strip ratio of 2.93:1.

Development is scheduled across three stages: Phase 1 (North Redline) and Phase 2 (South Redline) act as independent starter pits that provide early access to mineralized material, while Phase 3 represents the final ultimate pit expansion reaching a maximum depth of 620 m. Operational execution follows an owner-operator model utilizing 33.6 m³ cable shovels and 29.8 m³ hydraulic shovels paired with 240-tonne payload haul trucks. Logistics are further optimized through a dual-gradient ramp system (10% in-pit; 8% ex-pit) to enhance fleet efficiency, while waste storage is designed with an overall 18.3° slope to facilitate progressive reclamation and long-term stability.

1.3.8 Mineral Processing

Metallurgical test work for the Project to date includes almost 1,000 bottle roll leach tests, 75 column leach tests, gravity concentration tests, and flotation test work between 1997 and 2018. The test work indicates that recoveries of gold and silver from the Converse deposits are sensitive to several parameters, with particle size having the greatest impact on recovery. A conventional heap leach with three-stage crushing with HPGR, cement agglomeration, and an ADR recovery plant has been selected for the process flow sheet. The metallurgical test work forms the basis of the following conclusions:

- Crush size of 80% passing 6.3 mm
- Overall average gold recovery of 70% (based on variable recovery applied to the minable resource on a block-by-block basis).
- Design leach cycle of 120 days.
- Cement consumption of 6 kg/t.
- Cyanide consumption of 0.641 kg/t.

Mineralized material mined from the pit will be hauled to the process facilities and crushed in a three-stage crushing circuit to a product size of 80% passing 6.3 mm at an average throughput of 61,644 tonnes per day (tpd). The crushed material will then be stockpiled, reclaimed, and agglomerated with cement prior to being conveyed to the heap stacking system by an overland conveyor and a series of grasshopper conveyors feeding the radial stacking conveyor on the leach pad, where the material will be stacked in 10 m lifts and irrigated with dilute cyanide solution. The pregnant solution will drain by gravity to the pregnant solution pond and will then be pumped to a carbon adsorption circuit. Gold will be adsorbed onto activated carbon, periodically stripped in a desorption circuit, and recovered by electrowinning. The resulting precious metals sludge will be processed in a retort to recover mercury values prior to smelting to produce doré.

1.3.9 Project Infrastructure

The Project is designed to produce 228,500 tonnes per day (tpd) to 257,100 tpd from open pit operations, 61,644 tpd of which will be mineralized material delivered to the leach pad. Heap leaching will require a comprehensive infrastructure network. Power will be supplied via an onsite substation tied into the local electrical grid, operated by the regional utilities, NV Energy and Idaho Power, who jointly own the North Valmy Generating Station. Given the energy-intensive nature of mining, processing, and pumping operations, the substation will be designed to accommodate peak demands including mining, dewatering, and process plant requirements. Communications infrastructure will include a site-wide fiber-optic or microwave backbone to support operational controls, supervisory control and data acquisition (SCADA) systems, and worker communications, with redundancy for reliability. Potable and process water will be supplied from wells, permitted under Nevada Office of the State Engineer, and treated on site. Given the arid climate, process water recycling will be maximized, to minimize makeup water needs.

The processing infrastructure will center around a lined heap leach pad, solution conveyance corridor, external ponds (pregnant solution pond, barren pond and storm event pond), and an adsorption-desorption and recovery plant. The heap leach pad will be constructed in phases with a synthetic liner. Pregnant leach solution will be pumped from the preg pond to the ADR plant for stripping via carbon-in-column (CIC) process and barren solution will be returned to the heap leach pad.

Support infrastructure will include a permanent administration complex with mine offices, change houses (mine dry facilities), a safety/training center, and a first-aid station. A centralized warehouse will handle spare parts and consumables, located near an all-weather laydown yard and explosives magazine site (separately permitted). Secure fencing, access control, and environmental monitoring stations (air, groundwater) will be established to comply with state permitting requirements under Nevada regulations.

1.3.10 Market Studies

The principal commodity for the Converse Project is gold, which is freely traded at prices that are widely known and is a highly liquid commodity. As such, prospects for the sale of Converse gold production are virtually assured.

The gold prices used in this report have been provided to SLR by Roxmore and are aligned with the long-term views of several reputable market analysts and with SLR's internal pricing guidelines.

The pit optimization analysis and the Mineral Resource estimation are based on a long-term gold price of US$3,000 per ounce. Refer to Sections 14.0 and 16.0 of this report for more details.

SLR notes that for the economic analysis in this report, the gold prices used are based on analyst market consensus as of March 31, 2026. The QP considers the selected metal prices acceptable for the economic analysis, given the current mine life. The prices used for the economic analysis were assumed to be:

- Year -1 and Year 1: US$4,000/oz Au
- Year 2 to Year 14: US$3,600/oz Au

Contracts for the Project are expected to include design and construction contracts for the facilities and mine development; however, none of these contracts are in place at this time.

Contracts for the transportation, refining, and commercialization of the gold produced will also be required. Contracts will be negotiated and executed as necessary.

1.3.11 Environmental, Permitting and Social Considerations

The Project exists on private and public lands administered by the BLM. As a result, the majority of environmental studies related to mining activities will need to be conducted under the BLM authority as part of the NEPA regulations, which require various degrees of environmental impact analyses dictated by the scope of the proposed action. The Nevada Division of Environmental Protection (NDEP) will also be issuing state level permits. Due to the scope of the Project, a detailed Environmental Impact Statement (EIS) performed under NEPA will likely be required for environmental project approval.

Minimal environmental studies and project permitting efforts have been performed to date. The Project holds water and mineral rights. Existing environmental conditions are summarized from earlier assessments, including physiography, geology, hydrology, wildlife, and cultural resources. The area lacks permanent surface water, with groundwater occurring at approximately 350 ft depth and flowing south toward Alkali Lake. Wildlife studies indicate typical Great Basin fauna, with Greater Sage-Grouse (GSG) habitat located in the vicinity. GSG habitat is not currently mapped within the project boundary, so the risk of habitat loss mitigation requirements is low. Baseline studies should be started as soon as practical. Hydrogeologic, surface water, wildlife, and air quality studies all require at least one year of field data collection.

Permitting requirements span federal, state, and local jurisdictions. A BLM Plan of Operations and NEPA review will be central federal approvals, while Nevada agencies will oversee water pollution control, air quality, and reclamation permits. Existing permits relate primarily to exploration. The document emphasizes that reclamation and closure planning will require detailed cost estimates and bonding under both BLM and state regulations.

No major social or community issues have been identified, though stakeholder engagement has not yet begun. An updated cultural resource survey and stakeholder engagement should also be performed. Stakeholders include federal, state and local regulatory agencies, tribal groups, and NGOs.

Based on the review summarized in this report, and analogous projects in northern Nevada, there do not appear to be any red flags or insurmountable hurdles to developing the Project.

1.3.12 Capital and Operating Cost Estimates

The capital and operating costs presented in this Report include the costs required for mining and processing Mineralized Material from the Converse Project. The capital and operating costs presented in this Technical Report were estimated in Q1 2026 US dollars.

The Converse capital costs estimate meets the requirements of an AACE International (AACE) Class 5 estimate with an accuracy range of -20% to -50% and +30% to +100%. The cost estimates are considered reasonable for an IA level of study, considering that Class 5 estimates typically have 0% to 2% project definition per the AACE classification system.

The Converse Project's pre-production capital costs are estimated to be $829 million, as summarized in Table 1-4.

Table 1-4: Pre-Production Capital Cost Summary

Cost Category	Initial Capital Cost (US$ million)
Mine Pre-Production Development	116
Mine Equipment Purchase (down payment)	39
Mine Facilities (Including truck shop)	38
Process - Heap Leach Pad & Process Plant	312
Site Powerlines & Substation	16
Administration Building	8
Access Road	3
Sub-total Direct Cost	**531**
Engineering, Procurement, and Construction Management (EPCM) / Indirect Costs / Spares & First Fills	79
Owner's Costs	53
Sub-total Indirects & Owners Cost	**132**
Total excluding contingency	**663**
Contingency	166
Total Capital Costs	**829**

The sustaining capital cost estimate for the Converse Project was developed on the same basis as the initial capital cost. KCA developed the processing sustaining capital, and SLR estimated the mining sustaining capital. SLR found the sustaining capital costs to be an acceptable representation of the Project.

The total sustaining capital costs are estimated to be $514 million over the LOM as summarized in the Table 1-5.

Table 1-5: Sustaining Capital Cost Summary

Sustaining Capital Costs	US$ million
Mine Equipment Lease	188
Mine Equipment Sustaining	177
Processing Sustaining	148
Total LOM Sustaining Capital Costs	**514**

The final reclamation and closure costs are assumed at US$100 million at the end of the LOM.

The LOM mine operating costs total $4,757 million during the production period, and are estimated to be $16.01/t processed, as summarized in Table 1-6. The costs were estimated in Q1 2026 US dollars.

Table 1-6: Operating Cost Summary

Operating Costs	LOM Costs (US$ million)	LOM Average (US$ million / year)	Unit Costs (US$/t processed)
Open Pit Mining Costs	2,601	186	8.75
Processing Costs	1,772	127	5.96
G&A and Support Costs	384	27	1.29
Total Operating Costs	**4,757**	**340**	**16.01**

Notes:

1. Table values may not sum due to rounding.
2. Considers operating costs during the 14 years of the production period.

2.0 Introduction

SLR Consulting (Canada) Ltd. (SLR) was retained by Roxmore Resources Inc.(Roxmore or the Company) to prepare an independent Technical Report on the Converse Project (the Project or the Property), located in the state of Nevada, USA. The purpose of this Technical Report is to disclose results of an updated Mineral Resource estimate, effective March 31, 2026, and a Preliminary Economic Assessment (PEA) for the Converse Project as announced in the Company's press release dated April 20, 2026. This Technical Report was prepared in accordance with to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Roxmore Resources is a publicly traded mineral exploration company focused on advancing the Converse Project and is listed on the Toronto Stock Exchange under the symbol TSX:RM. The Project was acquired by Axcap Ventures Inc. through the purchase of Converse Resources LLC (CRL) from Waterton Splitter LLC, prior to Axcap's subsequent rebranding to Roxmore Resources. Roxmore now holds its interest in the Project through CRL, its wholly owned U.S. subsidiary, while title to the Property, permits, and certain operational aspects continue to be held and administered under CRL.

This Technical Report is considered by SLR to meet the requirements of a PEA as defined in Canadian NI 43-101 regulations. The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

2.1 Sources of Information

The Qualified Persons (QP), as defined under NI 43-101 guidelines, responsible for the various sections of this Technical Report are listed in Table 2-1.

Ms. April Barrios, P.Geo., and Mr. Balaji Subrahmanyan, SME(RM), both of SLR, conducted a site visit to the Project and associated facilities on November 4, 2025. During this site visit, they inspected multiple 2025 drill sites and verified collars relevant to the Mineral Resource estimate using a handheld Global Positioning System (GPS) unit. Ms. Barrios and Mr. Subrahmanyan also reviewed the general property layout, observed active drilling operations, and assessed the logging environment and procedures used for geological data collection and sampling. Both historical and recently drilled drill core were examined, and mineralization was visually compared with recorded lithology and redox classifications as well as corresponding analytical gold assay results. Ms. Barrios and Mr. Subrahmanyan were provided full access to all facilities and personnel during the visit and were accompanied by Blake McLaughlin (Executive Vice President) and Zsolt Molnar (Exploration Manager) of Roxmore.

Discussions were held with the following Roxmore personnel:

- Blake Mclaughlin, P.Geo., Executive Vice President – Development
- Paul Criddle, Director of Roxmore Resources
- Zsolt Molnar P.Geo., Exploration Manager, North America
- Isaac Riddle P.Geo., Senior Geologist
- Vance Spalding, Executive Vice President - Exploration

Table 2-1: Qualified Persons and Responsibilities

QP, Designation, Title	Company	Responsible for
April Barrios, P.Geo.	SLR	1.1, 1.1.1.1, 1.1.2.1, 1.3.1 to 1.3.6, 2 to 12, 14, 23, 24, 25.1, 26.1, 30
Balaji Subrahmanyan, SME(RM)	SLR	1.1.1.2, 1.1.2.2, 1.3.7, 1.3.10, 15, 16, 19, 21 (except 21.1.2.2, 21.1.3.2, and 21.2.2), 25.2, 26.2
Caleb Cook, P.E.	Kappes, Cassiday & Associates (KCA)	1.1.1.3, 1.1.2.3, 1.3.8, 13, 17, 21.1.2.2, 21.1.3.2, 21.2.2, 25.3, 26.3
Mark Trevor, PG, CPG, SME(RM)	SLR	1.1.1.5, 1.1.2.5, 1.3.11, 20, 25.5, 26.5
Matthew Behling, P.E.	SLR	1.1.1.4, 1.1.2.4, 1.3.9, 18, 25.4, 26.4
Jason J. Cox, P.Eng.	SLR	1.1.1, 1.1.1.6, 1.1.2.6, 1.1.3, 1.1.4, 1.2, 1.3.12, 22, 25.0, 25.6, 25.7, 25.8, 26.6
All		27

A site visit to the Converse Project was conducted on November 4, 2025, by April Barrios, P.Geo., and Balaji Subrahmanyan, SME(RM), of SLR, to support the Mineral Resource estimate and mining aspects of this Technical Report.

The remaining QPs did not conduct site visits. Their contributions relate to metallurgical processing, environmental and permitting, infrastructure, and economic analysis, which are based on the review of technical data, test work, and engineering information. In their opinion, a site visit was not required, and its absence does not materially impact the reliability of their work or conclusions.

The documentation reviewed, and other sources of information, are listed at the end of this Technical Report in Section 27 References.

2.2 List of Abbreviations

Units of measurement used in this Technical Report conform to the metric system. All currency in this Technical Report is US dollars (US$ or $) unless otherwise noted.

μ	micron
μg	microgram
a	annum
A	ampere
bbl	barrels
Btu	British thermal units
°C	degree Celsius
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
d	day
dia	diameter
dmt	dry metric tonne
dwt	dead-weight ton
°F	degree Fahrenheit
ft	foot
ft^2	square foot
ft^3	cubic foot
ft/s	foot per second
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
Gpm	Imperial gallons per minute
g/t	gram per tonne
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
Hz	hertz
in.	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km^2	square kilometre
km/h	kilometre per hour
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
lb	pound
L/s	litres per second
m	metre
M	mega (million); molar
m^2	square metre
m^3	cubic metre
MASL	metres above sea level
m^3/h	cubic metres per hour
mi	mile
min	minute
μm	micrometre
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
oz	Troy ounce (31.1035g)
oz/st, opt	ounce per short ton
ppb	part per billion
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
wt%	weight percent
yd^3	cubic yard
yr	year

3.0 Reliance on Other Experts

This Technical Report has been prepared by the QPs for Roxmore. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to SLR at the time of preparation of this Technical Report.
- Assumptions, conditions, and qualifications as set forth in this Technical Report.

For the purpose of this Technical Report, the QPs have relied on ownership information provided by Roxmore. The client has relied on an opinion by Wolcott dated May 2, 2026 entitled *Record Title Examination, Land Status and Mining Claim Review Unpatented Mining Claims, Converse Mine, Perishing and Humboldt Counties, Nevada*, and this opinion is relied on in Sections 4.2, 4.3, and 4.4 and the Summary of this Technical Report. The QPs have not researched property title or mineral rights for the Converse Project and expresses no opinion as to the ownership status of the property.

The QPs have relied on Roxmore for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income as summarized in Sections 4.2, 4.3, and 4.4 from the Converse Project.

Except for the purposes legislated under provincial securities laws, any use of this Technical Report by any third party is at that party's sole risk.

4.0 Property Description and Location

4.1 Location

The Property is located in the Buffalo Valley, in the southeast corner of Humboldt County, Nevada, with a small portion extending into Pershing County at the southwest corner of the Property (Figure 4-1). The Property lies approximately 40 km (25 mi) west-northwest of Battle Mountain, and 80 km (49.6 mi) southeast of Winnemucca accessible via the Valmy exit off Interstate I-80. Reno is located approximately 330 km (205 mi) to the southwest by highway. The Property is located at approximately 40°43' N latitude and 117°16' W longitude, within Township 33 North, Range 42 East.

Figure 4-1: Location Map

Source: SLR 2026.

4.2 Land Tenure

The Property represents a consolidated package of 526 unpatented mining claims (lode and placer) and seven fee land parcels, representing a total land area of 13,275.3 acres.

The unpatented mining claims are administered by the Bureau of Land Management (BLM) and give the holder rights to extract minerals from hard rock (lode) or unconsolidated ground (placer) from federal public land, but the federal government retains ownership of the surface, and annual maintenance (assessment work or fees) is required. Most (500) of the claims are unpatented mining lode claims. The remainder (26) are unpatented placer claims and overlap the lode claims.

Unpatented mining claims (both lode and placer) are created and maintained in accordance with the U.S. General Mining Law of 1872, as amended. While the federal government retains title to the land within an unpatented mining claim, the claimant, through compliance with federal and state laws, has the right to explore for and exploit certain minerals (including precious metals) within the claim area. No specific mining or exploitation license or permit is required to hold the claims.

An individual unpatented lode claim is limited to a maximum size of 20.66 acres, and most claims at the Property meet this size threshold. Placer claims can vary from 20 acres to 160 acres in size; at Converse, all placer claims are 20 acres and overlap with the lode claims.

Through its subsidiary CRL, Roxmore holds title to 276 of the unpatented lode claims, and all fee land parcels. The remainder of the claims (224 lode and all 26 placer) are held by Nevada North Resources (USA) (NNR) and leased to Roxmore, pursuant to an amended and restated mining lease dated March 29, 2013, but effective August 31, 2012 (the Converse Lease). The Converse Lease grants Roxmore via its subsidiary CRL the right to explore, develop, and mine on the claims. The Converse Lease had an initial term of 10 years (until August 31, 2022), with extensions possible so long as there are mining related activities occurring (development, mining, processing, reclamation or closure). The term has been extended twice in 10 year blocks; the current expiry date is August 31, 2032.

An additional annual fee, payable to the BLM, is required to maintain the remaining 500 claims in good standing. At present, the fee is $200 per unpatented mining claim and $20/acre for placer claims. For the current year, annual maintenance fees to BLM at Converse total $110,512. Currently, there are no federal royalties imposed on mineral production from unpatented mining claims.

Remaining payments stemming from the acquisition of the Property are staged annually to July 15, 2028, and total $7.5 million in cash or shares at the discretion of the current owner.

The fee land parcels (4,436.5 acres) give private ownership of both surface and subsurface rights with no maintenance or fee obligations outside of annual property tax, which is $5,011 Each fee land parcel covers approximately one square mile or 640 acres. One fee land parcel is under lease to Nevada Gold Mines LLC (NGM), consequently, the mineral rights are not currently available to Roxmore.

A property map showing ownership, tenure type, and tenure status is presented in Figure 4-2. A description of the fee land parcels is presented in Table 4-1, and a summary of the unpatented claims is presented in Table 4-2. A detailed list of land tenure is provided in Appendix 1.

Table 4-1: Fee Land Parcels

Section	Township	Range	Area		APN #	% SURF	% MIN
			Acres	Hectares			
05	33	42	646.5	261.6	07-0451-02	100	100
17[1,3]	33	42	635.6	257.2	07-0451-14	100	100
19	33	42	566.7	229.3	07-0451-19	100	100
21[2]	33	42	653.9	264.6	07-0451-21	100	100
29[2]	33	42	642.0	259.8	07-0451-26	100	100
33[2]	33	42	657.0	265.9	07-0451-33	100	100
05[2]	32	42	634.8	256.9	07-0481-02	100	100
Total			4,436.5	1,795.4			

Source: Roxmore 2026.

Notes:

1. Currently, mineral rights are leased to NGM.
2. Subject to Triple Flag Precious Metals royalty payment.
3. Subject to Nevada Land and Resource Company LLC royalty.

Table 4-2: Unpatented Mining Claims

Lode Claim Owner	No. Claims	Area		Placer Claim Owner	No. Claims	Area	
		Acres	Hectares			Acres	Hectares
CRL	276	5000.2	2023.5	CRL	0	0.0	
NNR	224	3838.6	1553.4	NNR	26	525.6	213.1
Total Lode	500	8838.8	3576.9	Total Placer	26	526.6	213.1

Source: Roxmore 2026.

4.3 Royalties

The following royalties are active on the Property:

1 The entire Property is subject to an NSR production royalty payable to Royalty Consolidation Company LLC (RCC) on the sale of any minerals from the Property. The royalty rate is agreed at the higher of 6%, or 6% minus existing royalty agreements. Effectively, this agreement encumbers the Property with a total NSR royalty of 6% on the production of all minerals.

2 The Converse Lease is subject to a sliding scale net smelter return (NSR) royalty from 3% to 5% on all minerals produced, indexed to the price of gold. At current gold prices, and so long as the gold price is at or above $375/oz, the Converse Lease royalty rate is 5%. The Converse Lease requires annual advance royalty payments of $50,000, $75,000, or $100,000 depending on the average gold price for the 12-month period ending July 31 of each year. The annual advance royalty payment for 2026 is $100,000. Roxmore, through its subsidiary CRL, to date has paid approximately $2,060,000 in

annual advance royalties, all of which can be credited in full toward future royalty payments with production.

3 Triple Flag Precious Metals (TFPM) is entitled to a gold price-related sliding scale NSR royalty of 3% to 5% on the production of gold, and 3% on the production of other minerals, from four of the five fee land parcels. At gold prices of US$400/oz and above, the royalty rate is 5%. The royalty originated under Newmont which was then sold to Maverix in 2020 and Maverix was subsequently acquired by TFPM in 2023.

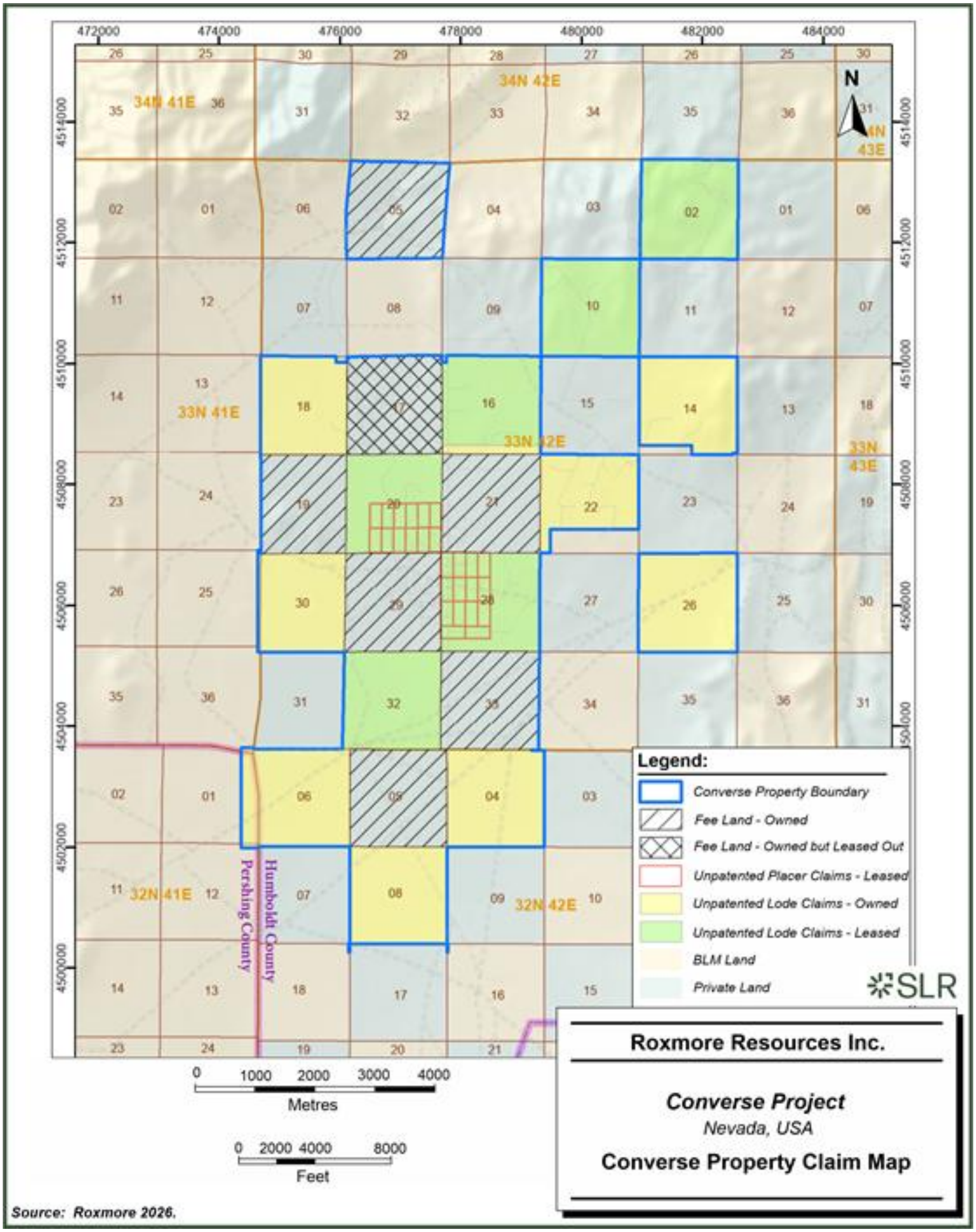
4 Nevada Land and Resource Company LLC (NLRC) is entitled to a 1% NSR royalty on the production of minerals from the fee land parcel currently leased to NGM, and is therefore excluded from the tenure.

The current land holding costs of the Property total approximately $215,523 annually. This includes the annual advance royalty payment due under the Converse Lease, as well as BLM maintenance fees and county recording fees for the claims and property taxes on the fee land parcels.

4.4 Encumbrances

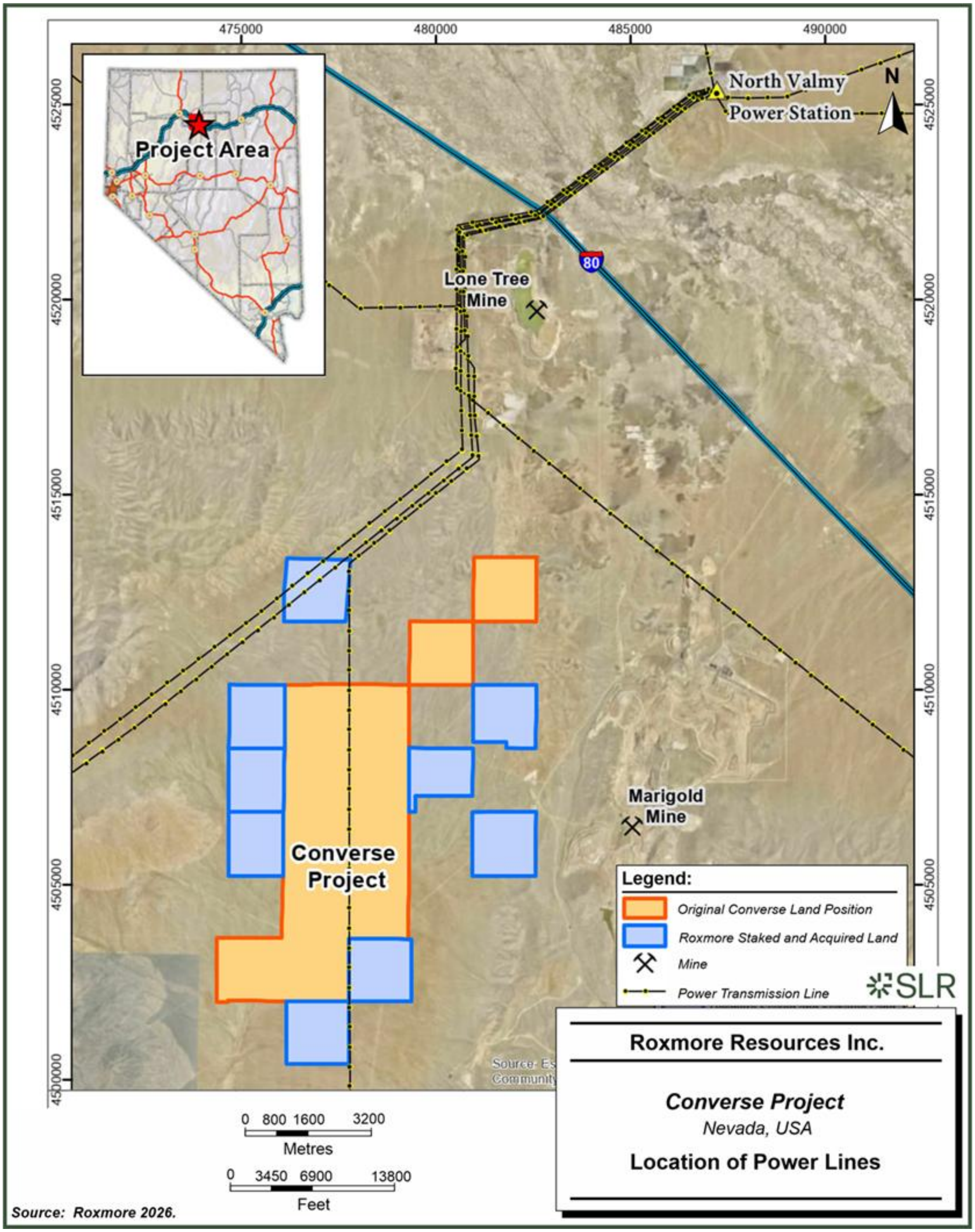
A transmission line traverses the Project area and crosses the interpreted mineralized zone and is expected to require relocation to accommodate mine development. Available mapping indicates the line operates at approximately 120 kV. Figure 4-3 shows the location of power lines relative to the Property.

Figure 4-2: Converse Property Claim Map



Note: For more information on the individual claims, see Appendix 1.

Figure 4-3: Location of Power Lines



Source: Roxmore 2026.

4.5 Environmental Permitting, Regulatory Requirements, and Water Rights

The Converse Project is a greenfield site. All exploration, development and production activities are subject to regulation under one or more of the various state and federal environmental laws and regulations. Many of the regulations require Roxmore to obtain permits for its activities. Roxmore must update and review its permits periodically and may be subject to environmental impact analyses and public review processes prior to approval of any additional activities. Roxmore expects to make significant expenditures in the future to expand the scope of its current permits.

Exploration at the Property is carried out under an Exploration Plan of Operations NVN065461, approved by the BLM pursuant to Environmental Assessment (EA) N20-98-001P and Reclamation Permit #0122 approved by the Nevada Division of Environmental Protection (NDEP). There is a US$139,508 reclamation bond currently associated with the existing permits.

In 2019, CRL purchased 2,560 acre-ft of irrigation water rights from New Nevada Lands, LLC (Permit 71715 and 71716). Once converted to mining and milling use, the acquired water rights will support the construction and operation of a future mine at the Property. An application requesting a change in the water rights' point of diversion, place of use and manner of use was submitted to the Nevada State Engineer on October 29, 2020. The change was granted and is still valid.

The QP is not aware of any environmental liabilities on the Property. Roxmore has all the required permits to conduct the proposed work on the Property. The QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the Project.

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the Converse Property is via a well maintained county gravel road extending approximately 17 km (10.6 mi) from Exit 216 (Valmy) on Interstate I-80 (Figure 5-1). The road passes the Marigold mine and the turnoff to the Trenton Canyon mine, both located near the Property. The Valmy exit lies 24 km (15 mi) northwest of Battle Mountain and 63 km (39 mi) southeast of Winnemucca by road in northwestern Nevada. Additional access within the Property area is by unimproved dirt roads and tracks.

5.1 Climate

The Project area is characterized by a cold semi-arid (BSk) climate, with low precipitation, moderate winter snowfall, and a large seasonal temperature range. The coldest month of the year, December, has average low and high temperatures of -7°C and 5°C, respectively (WeatherSpark 2026). In the hottest month, July, the monthly average low and high temperatures are 12°C and 33°C. Average annual precipitation is approximately 150 mm to 220 mm (6 in. to 9 in.), with annual snowfall on the order of 25 cm to 40 cm (10 in. to 16 in.). The Property has no significant climatic issues and work can be completed throughout the year.

5.2 Local Resources

The Property lies near the regional population centres of Battle Mountain, 40 km (25 mi) to the east and Winnemucca, 80 km (49.6 mi) to the west, where commercial services, suppliers, accommodation, and medical facilities are available. In addition, both communities support several nearby active mining operations, and are a source of experienced labour as well as mining industry-specific support services.

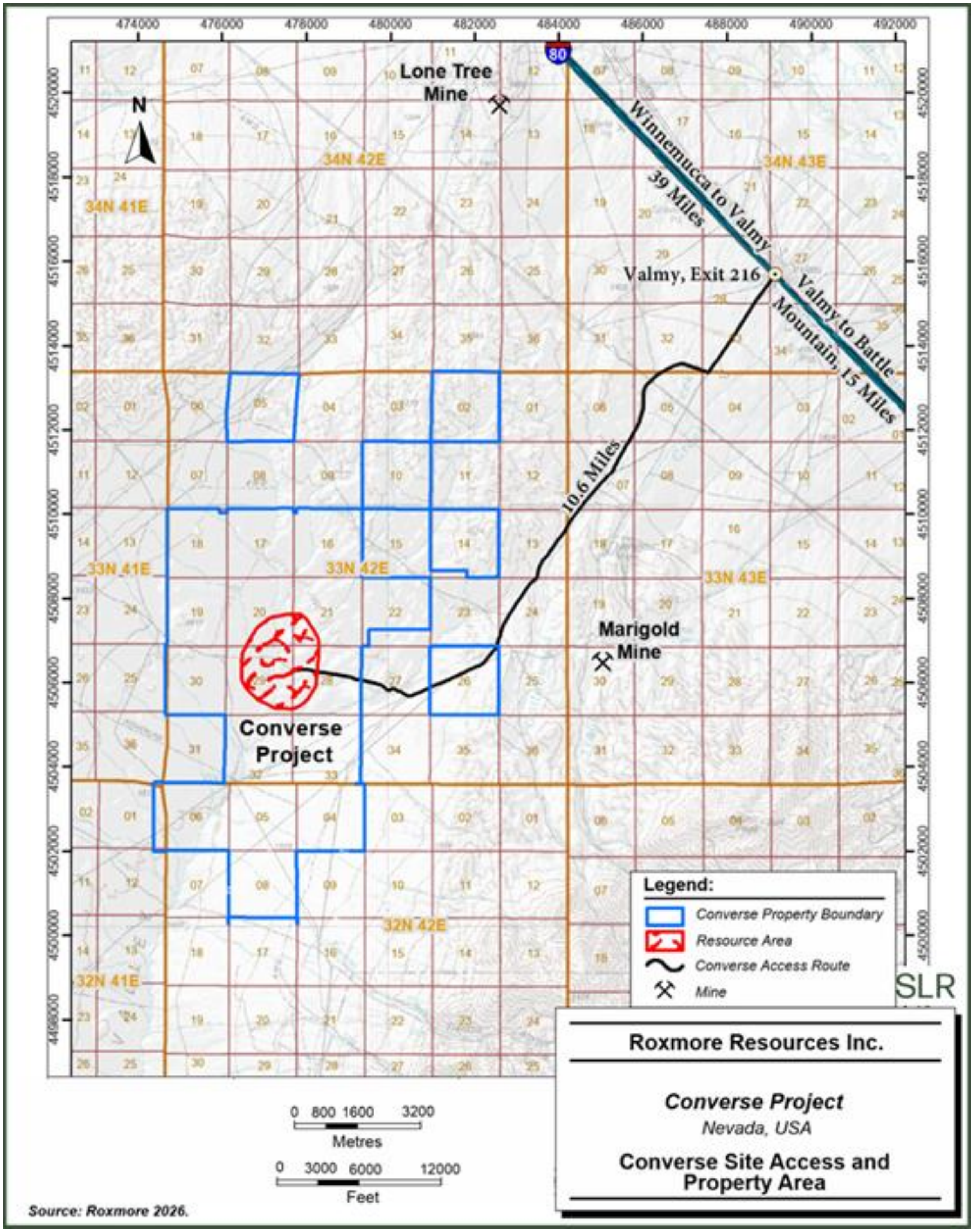
5.3 Infrastructure

Electricity and natural gas infrastructure do not currently exist on the Property. NV Energy operates the main powerline that runs along the I-80 corridor, from where power could be obtained. Water required for exploration drilling is supplied by two near by sources, the Marigold Mine's well (owned by SSR) or by the Road and Highway Builders well. The Property is sufficient size to accommodate potential exploration and mining facilities, including waste rock disposal and processing infrastructure.

5.4 Physiography

The Property is in a relatively flat valley with Buffalo Mountain to the west-northwest, the Havallah Hills to the north, and the Battle Mountain Range to the southeast. The Property covers a gently southwest-sloping alluvial plain with elevations ranging from 1,494 masl to 1,585 masl (4,900 ft to 5,200 ft above sea level) (topographic relief is approximately 91 m [300 ft]). Vegetation consists of sagebrush and desert grasses, with areas of desert hardpan without vegetation.

Figure 5-1: Converse Site Access and Property Area



Source: Roxmore 2026.

6.0 History

6.1 Prior Ownership

The earliest recorded exploration activities date back to 1988, which is also when systematic exploration began. Since that time, the Project has been operated by numerous companies, as summarized in Table 6-1. Since acquiring the Project, Roxmore has expanded the land holdings.

6.2 Exploration and Development History

Historical exploration up to 2019 is summarized in Table 6-1 from a previous Technical Report on the Property (Apex Geoscience 2025). There has been no production from the Property.

Table 6-1: Converse Property Ownership and Exploration Summary (1988 to 2019)

Operator	Period	Comments
Nevada North Resources (NNR)	1988	Staked 315 unpatented lode mining claims, known as the “Nike Property”.
Kennecott Corp.	1988	Completed two reverse circulation (RC) holes totalling 178 m (585 ft).
Chevron Resources (Chevron)	1989-1991	Optioned the Nike Property and carried out reconnaissance geological mapping, geochemical sampling, gravity and gradient array induced polarization (IP) surveys.
		Completed 11 RC holes totalling 1,466 m (4,810 ft), of which three holes all failed to reach bedrock.
Cyprus Mines Corp.	1991-1992	Acquired the option from Chevron and carried out limited geological mapping and acquired ground magnetic data.
		Completed 17 RC holes, totalling 1,454 m (4,770 ft), with anomalous gold values intercepted in bedrock material.
Independence Mining Co.	1993-1994	Leased the Property from NNR and carried out a bulk leach extractable gold (BLEG) survey.
		Completed 10 mud-rotary holes, totalling 1,509 m (4,950 ft), and intersected significant mineralization in two holes with 0.411 g/t (0.012 oz/st) Au over 13 m (45 ft) (IN-1) and 0.480 g/t (0.014 oz/st) Au over 23 m (75 ft) (IN-4A).
Uranerz USA Inc. (UUI)	1994	Executed a lease agreement for the Property with NNR.
Romarco Nevada Inc (Romarco NV).	1995	Executed a joint venture (JV), known as the Nike Venture, agreement with UUI, who remained the operator, on a 50%-50% basis.
		Completed further gravity surveys as well as an enzyme leach geochemical survey.
		Completed 10 mud-rotary holes, totalling 1451 m (4,760 ft).
	1996	The Nike JV staked 36 unpatented lode claims.

Operator	Period	Comments
		The Nike JV entered into an exploration agreement with Santa Fe Pacific Gold (SFPG, now Newmont), who leased adjacent fee land.
		Sixteen RC holes (totalling 3,190 m [10,465 ft]) and six core holes (totalling 1,963 m [6,440 ft]) were completed with significant intercepts including 1.06 g/t (0.031 oz/st) Au over 85 m (280 ft) (NK-021) and 1.06 g/t (0.031 oz/st) Au over 160m (525 ft) (NK-031).
	1997	Newmont acquired SFPG.
		Thirty RC holes (totalling 6,446 m [21,148 ft]) and three core holes with RC pre-collars (totalling 1,100m [3,611 ft]) were completed with significant intercepts, including 0.028 oz/st Au over 682 ft (NK-042C), 0.042 oz/st Au over 250 ft (NK-054) and 0.039 oz/st Au over 250 ft (NK-065).
		Initial metallurgical test work was completed.
		A historical block model was completed by the Nike JV.
	1998	Cameco acquired UUI and changed names to UUS Inc. (UUS)
		Fifty-two RC holes (totalling 12,805 m [42,012 ft]) were completed, and further metallurgical test work was carried out.
	1999	Fifteen RC holes (totalling 2,845 m (9,335 ft]) were completed.
		An updated historical mineral estimate was prepared by the Nike JV.
Metallic Ventures Gold Inc. (MVG)	2001	Romarco NV was acquired by MVG.
	2002	Romarco NV acquired USS' interest in the Nike JV and Converse Agreement, as well as acquired Newmont's interest in the Converse Agreement.
		Mine Development Associates (MDA) prepares an updated Mineral Resource estimate and discloses results within a NI 43-101 Technical Report.
	2003	Zonge Geoscience completed 4.8-line km (3 line miles) of Controlled Source Audio Magnetotellurics (CSAMT) on the Property.
		Eighteen RC holes (totalling 4,568 m [14,988 ft]) and eight core holes with mud-rotary pre-collars (totalling 1,618 m [5,307.2 ft]) were completed.
	2004	Twenty-eight RC holes (totalling 7,505 m [24,622.5 ft]) were completed.
		Metallurgical test work at Kappes, Cassiday & Associates (KCA) was initiated.
		Watts Griffs and McOuat (WGM) prepare an updated Mineral Resource estimate and disclose results within a NI 43-101 Technical Report.
	2007	Fifty-three RC holes (totalling 11,424 m [37,480 ft]) and eight core holes (totalling 2,235 m [7,332.2 ft[) were completed.

Operator	Period	Comments
	2008-2009	Metallurgical test work at McClelland Laboratory Inc. (MLI), Reno, and geotechnical evaluations were completed.
		FSS Canada generated an updated historical mineral estimate.
Newcrest Resources	2003	Four RC holes (totalling 1149 m (3,770 ft) hole IDs have prefix BV - no data on operator.
	2004	Thirteen RC holes (totalling 3,758 m (12,328 ft)) hole IDs have prefix BV - no data on the operator.
	2005	Three RC holes (totalling 1,335 m (4,380 ft)) hole IDs have prefix BV - no data on the operator.
International Minerals Corp. (IMC)	2010	MVG was acquired by International Minerals Corp. (IMC).
	2011	Eight core holes (totalling 4,251 m [13,945.5 ft]) and six core holes with RC pre-collars (totalling 2,347 m (7,700.4 ft)) were completed.
	2012	Four core holes (totalling 1,533 m [5,028.6 ft]) and 10 core holes with RC pre-collars holes (totalling 4,896 m [16,064.2 ft]) and six RC holes (totalling 1,236 m [4,058 ft]) were completed.
		Micon completes an updated Mineral Resource estimate and PEA on behalf of IMC, disclosing results within a NI 43-101 Technical Report.
Chaparral Gold Corp. (Chaparral)	2013	IMC was acquired by Hochschild Mining plc and the Converse Property, along with the other Nevada assets, was spun out into Chaparral Gold Corp.
	2014	RedDot3D completes an updated Mineral Resource Estimate and NI 43-101 Technical Report.
Converse Resources LLC (CRL)	2014	Chaparral was acquired by CRL (through Waterton Global Resource Management)
	2017	Completed seven metallurgical core drill holes on the Property totalling 1,812 m (5,944 ft)
	2018	Completed metallurgical test work that included bottle rolls, agglomeration, and compaction tests and column leach tests.
	2019	CRL purchased 2,560 acre-ft of irrigation water rights from New Nevada Lands LLC (Permit 71715 and 71716).
CRL and Axcap Ventures Inc. (Axcap)	2025	RedDot3D prepares an Amended and Restated NI 43-101 Technical Report; Axcap purchases the Property from CRL.

Modified from Apex Geoscience 2025.

6.3 Historical Drilling

This section summarizes the drilling carried out at the Converse Project by historical operators.

Roxmore has compiled information for a total of 84,129 m drilled in 348 historical holes at the Project, summarized in Table 6-2. Approximately 83% of the holes and metres were drilled using conventional rotary or mud-rotary (MR) and reverse circulation (RC) methods and 17% of the holes were diamond drilled (DD) including those holes with a RC or MR pre-collar through the overburden.

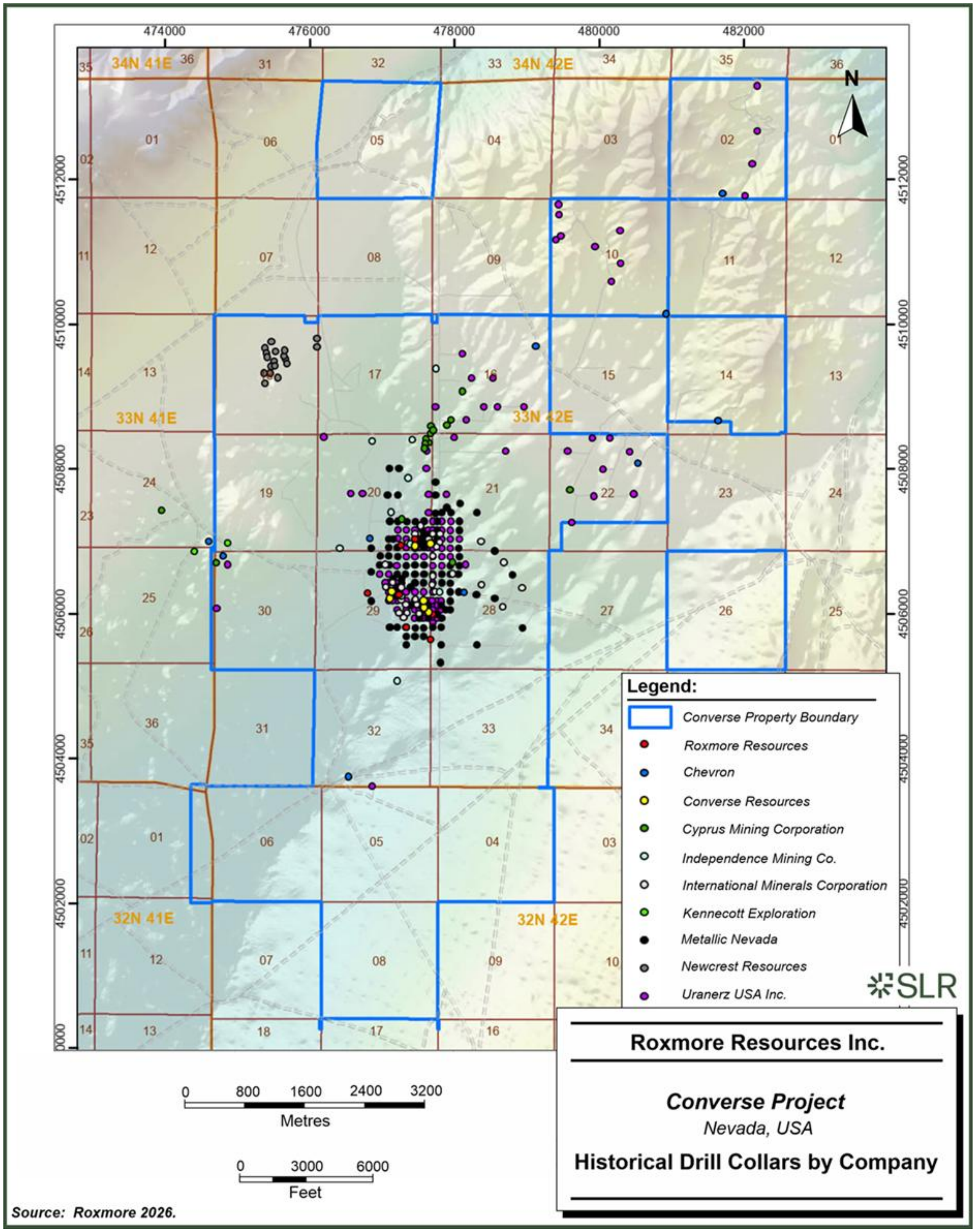
Table 6-2: Summary of the Converse Project Historical Drilling

Company	Year	RC or MR		MR-RC/DD		DD		Total	
		No.	Metres	No.	Metres	No	Metres	No	Metres
Chevron	1989, 1991	11	1,466					11	1,466
Kennecott Corp.	1989	2	178					2	178
Cyprus Mining Corp.	1992	17	1,454					17	1,454
Independence Mining Co	1994	10	1,509					10	1,509
Uranerz USA Inc.	1995-1998	123	26,792	9	3,064			132	29,856
Unknown	2003-2005	20	6,242					20	6,242
Metallic Ventures Gold Inc.	2003-2004, 2007	99	23,497	8	1,618	8	2,235	115	27,350
International Minerals Corp.	2011-2012	6	1,236	16	7,243	12	5,783	34	14,262
CRL	2017					7	1,812	7	1,812
Total		**288**	**62,374**	**33**	**11,925**	**27**	**9,830**	**348**	**84,129**

Most historical holes were drilled vertically, or within 10°of vertical. Roughly 30% have been drilled as angle holes (<80°dip). The geometry of gold mineralization varies but appears steeply dipping, associated with fault structures, or more gently dipping, associated with bedding. Steeply dipping mineralized features are not effectively tested at true widths by steeply oriented holes.

Figure 6-1 shows the location of historical drill hole collars within the Project.

Figure 6-1: Historical Drill Collars by Company



Source: Roxmore 2026.

The QP is not aware of the details regarding the drilling contractors, drilling methods, sampling procedures, collar-survey methods, and types of drill rigs utilized in the historical Converse drilling programs other than those summarized in the following subsections.

6.3.1 RC Drilling

Between 1995 and 1999, Uranerz USA Inc. (UUI) / Romarco Nevada Inc (Romarco NV) conducted RC drilling using the Eklund Drilling Company of Reno, NV. RC holes drilled at Converse were between 6 in. (15.2 cm) and 5¼ in. (13.3 cm) in diameter, using a standard configuration of downhole hammer and interchange; as is common practice, the alluvial portion of the holes were drilled using rotary tricone bits. A booster and auxiliary compressors were used in the deeper portions of most holes, typically below approximately 900 ft (274.3 m). RC drilling was completed using TH-100A and Explorer 1500 rigs and was completed wet using a cyclone and rotary wet splitter. Mud rotary drilling was completed by B & B Drilling, of Grand Junction, Colorado, with hole diameters of 5½ in. to 5¼ in. (12.7 cm to 13.3 cm).

Between 2003 and 2007, Metallic Ventures Gold Inc. (MVG)'s RC drilling was performed by Eklund Drilling Company of Reno, NV and Rimrock Drilling of Reno, NV, generally using MPD 1,500 rubber-tire Explorer and truck-mounted deep-hole IR-75C rigs. RC drilling was carried out using a 5 ¾ in. (14.6 cm) diameter hammer bit.

In 2011 and 2012, International Minerals Corp. (IMC)'s RC drilling was performed by Rimrock Drilling of Reno, NV, generally using a TH75E drill rig. RC drilling was carried out using a 5 ¾ in. (14.6 cm) diameter hammer bit. The hole conditions, any drilling problems, and the water depth and flow were reported by the driller on the driller's log.

6.3.2 Core Drilling

Between 1996 and 1997, UUI/Romarco NV's core drilling was performed by Connors Drilling using a Longyear 44 core rig. Core sizes were HQ and reduced to NQ where necessary. Most of the core holes were pre-collared using an RC rig.

Between 2003 and 2007, MVG's core drilling was performed by Boart Longyear Drilling Company (Boart Longyear) using an LS 244 truck-mounted core rig. Core sizes were NQ and HQ (1.87 in. and 2.5 in.). Most of the core holes were pre-collared using an RC rig.

In 2011 and 2012, IMC's core drilling was performed by American Drilling and Boart Longyear. The type of rigs used is unknown. Core sizes were HQ. Most of the core holes were pre-collared using an RC rig.

In 2017, CRL's core drilling was performed by Major Drilling of Salt Lake City, Utah, using a truck mounted LF-230 rig. The purpose of the 2017 core drilling program was to collect samples for metallurgical studies. A total of seven PQ size (3.35 inch or 8.5 cm) core diameter) diamond core holes were drilled, totalling 1,812 m (5,944.0 ft).

6.3.3 Geological Logging

The historical RC logging campaigns collected a variety of information that predominantly included mineralogy, lithological unit, colour, alteration, metamorphic assemblages, quartz vein intensity, and oxide state with intensity of iron oxides and sulphides.

The historical core logging campaigns collected a variety of information, predominantly including recovered core length mineralogy, lithological unit, colour, alteration, metamorphic assemblages, oxide state with intensity of iron oxides and sulphides, and vein type and abundance.

The 2017 CRL core was transported to a secure logging facility in Lovelock, NV, where the CRL geologists and technicians completed the following:

- Core boxes were arranged sequentially on the logging tables and drill mud was washed from the core.
- Geotechnical measurements recovery, rock quality designation (RQD) and rock mass rating (RMR) were captured.
- Geological data, including mineralogy, lithological unit and texture, colour, structural type and style, redox, alteration type and intensity, mineralization type, and percentage were captured by CRL geologists directly into a Microsoft Excel logging template.
- Sample boundaries were marked with wax pen, and sample tags were stapled to the inside of the core box at the beginning of the interval. Digital core photographs were taken of wet core with the sample tags visible and the box number and interval length (from – to) labeled.

Following logging, sample markup and photography, the core boxes were placed on pallets, wrapped in plastic, and stored within the secured laydown yard at the Lovelock facility. An independent transportation company transported pallets of core with a signed inventory list to the ALS Global facility for sample cutting, bagging, and analysis, as described in Section 11.0.

6.3.4 Recovery

MVG core drilling recoveries averaged approximately 90%. Approximately 92% of the core was from the bedrock where the overall recovery was 94%. IMC core recovery was 94% in bedrock and 70% in alluvium. The combined average recovery was 92% for the entire drill program.

The 2017 CRL core recovery values averaged 98% in the bedrock and 82% in the overlying alluvium units. The combined average recovery was 95% for the entire drill program.

6.3.5 Bulk Density

Historical operators have collected 380 bulk density or specific gravity measurements from across the deposit. Over half of the samples were collected from RC cuttings sampling both alluvium and bedrock, while the remainder were taken from core rock samples. Historical records indicate a variety of methods used, including water immersion methods for cuttings, wax coating, and acrylic coating methods to accommodate highly fractured and porous material.

6.3.6 Historical Collar Surveys

Prior to MVG drilling in 2003, no information is known on how collars were surveyed. MVG drill collars were surveyed by a registered contract land surveyor; however, it is unknown who completed the work and with what instrument. IMC drill collars were surveyed but it is also unknown by whom and with what instrument. The 2017 CRL drill collars were surveyed by Daniel Park of Elko Mining Group using a high accuracy Real Time Kinematic (RTK) GPS equipment with centimetre accuracy.

6.3.7 Historical Downhole Surveys

Downhole surveys for the Romarco NV / UUI drilling were completed by Silver State Surveys Inc. of Tucson, Arizona (AZ), and by Wellbore Navigation Inc. (Wellbore) of Elko, NV. Both companies used gyroscopic instruments with measurements recorded on 15.2 m (50 ft) 15.2 m intervals. Downhole surveys for the MVG drillholes were collected using a gyroscopic instrument operated by Wellbore. Measurements were recorded on 15.2 m (50 ft) intervals. Downhole surveys for the IMC drill holes were collected using a gyroscopic instrument operated by International Directional Services, with measurements recorded on 15.2 m (50 ft) intervals.

The 2017 CRL drill holes were downhole surveyed using a Reflex EZ-Trac gyroscopic tool operated by Major, with measurements recorded on 15.2 m (50 ft) intervals. Downhole survey deviations were recorded on a tablet and sent via electronic mail to CRL personnel. The REFLEX GYRO is not affected by magnetic interference and can be used within steel drill rods. Surveys were checked for erroneous records, such as large deviations between readings. Questionable survey data was flagged in the database and excluded from the Company's database exports.

6.4 Previous Resource Estimates

Several Mineral Resource estimates were completed for the Property between 1999 and 2014. These estimates predate the implementation of the standards set forth in NI 43-101 and Canadian Institute of Mining (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014) and CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines (November 2019).

An updated Mineral Resource estimate prepared by RedDot3D in 2021 and amended in 2025 was prepared and documented within a NI 43-101 Technical Report (Apex Geoscience, 2025).

These estimates are considered to be historical in nature and should not be relied upon. Neither SLR nor Roxmore are treating the historical estimates as current Mineral Resources or Mineral Reserves. The Company does not view these historical estimates to be relevant for this Technical Report, as they have been superseded by the current Mineral Resource estimate presented in Section 14.0.

6.5 Past Production

The QP is not aware of any historical mining or mineral production having occurred on the Property.

7.0 Geological Setting and Mineralization

The content of this section has been updated, although certain figures and source references from the previous technical report (Apex Geoscience 2025) have been retained.

7.1 Regional Geology

The Converse Property is located in north-central Nevada within the Basin and Range physiographic province in the Battle Mountain district. The Property lies near the northern end of the Battle Mountain-Eureka Trend, a northwest-trending belt of precious metal deposits with current reserves and past production exceeding 50 Moz Au (Holley et al. 2015).

7.1.1 Geological and Tectonic History of North-Central Nevada

Paleozoic basement rocks of north-central and northeastern Nevada are generally divided into four distinct tectonostratigraphic assemblages (Roberts 1964) that have recorded a complex history of compressional and extensional tectonic events. These events have affected the western margin of North America from the early Paleozoic to the present, and formed the following assemblages:

- The eastern carbonate assemblage
- The slope or transitional assemblage
- The western siliceous and volcanic assemblage
- The overlap assemblage

Several authors have proposed that the linear alignment of Nevada's major gold belts, including the Carlin Trend and the Battle Mountain Eureka Trend, reflects the reactivation of Paleozoic and early Mesozoic tectonic structures situated along a broad Paleozoic suture zone. This suture formed during the accretion of the Archean and Paleoproterozoic terrain on the westward margin of the Wyoming craton during the breakup of Rodinia (Cline et al. 2005, Muntean et al. 2007).

Rifting in the late Proterozoic established a passive continental margin along the proto-Pacific edge of western North America, leading to the accumulation of an 8 km to10 km-thick continental margin sequence of siliciclastic and carbonate rocks (Muntean et al. 2007). These deposits range from nearshore sandstone to shale and offshore carbonate reef and lagoonal deposits (Cook and Carboy 2004). Farther west, debris flow deposits, turbidites, and lime mudstone of the transitional assemblage accumulated along the continental slope, while siliceous and volcanic rocks of the western assemblage were deposited in the deeper basin plain (Roberts et al. 1965; Cook and Corboy 2004).

Evidence of a major Late Devonian–Early Mississippian tectonic event, the Antler Orogeny, is recorded by folding and thrusting of Ordovician western-assemblage rocks and the development of the Antler highland (Roberts 1964). During this event, western assemblage rocks were emplaced eastward, over eastern assemblage carbonates along the Roberts Mountains Thrust (RMT). Subsequent Pennsylvanian uplift and erosion of the Antler highland produced the Pennsylvanian–Permian overlap assemblage, derived from western assemblage detritus shed into a broad foreland basin. The Antler Orogeny continued into the Permian, after which the Golconda allochthon was emplaced during the Sonoma Orogeny in the Late Permian–Early Triassic, thrusting deep water Paleozoic sediments over rocks of the Roberts Mountains allocthon (Silberling and Roberts 1962). Deformation during both orogenies migrated progressively westward as successive thrust plates were emplaced (Price 2010). By the Middle Triassic, an east-dipping subduction zone had developed along the western margin of North America (Cline et al. 2005).

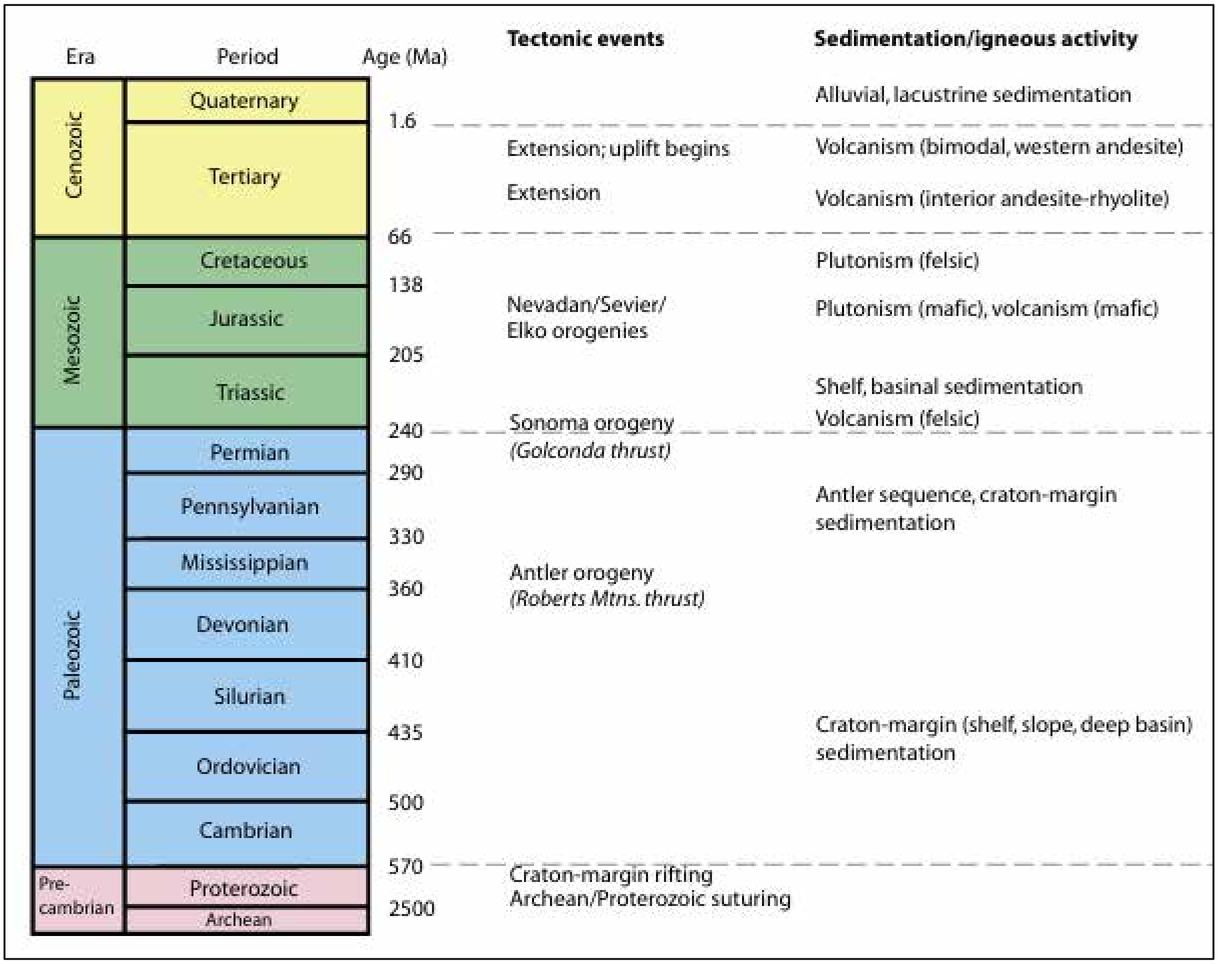
Jurassic to Early Cretaceous compression along the western margin of North America produced subduction-related magmatism, generating large volumes of intermediate to felsic melts that led to the emplacement of the Sierra Nevada batholith. Continued compression resulted in the accretion of oceanic arc terranes, development of thrust belts, and formation of ophiolite sequences, collectively referred to as the Nevadan Orogeny (Wyld et al. 2002). A major phase of felsic plutonism occurred in Nevada during the Late Jurassic (approximately 155 Ma to 160 Ma) (du Bray 2007). Late Jurassic to Cretaceous compression farther east formed the Sevier fold-and-thrust belt (DeCelles 2004), followed by flat-slab subduction of the Farallon plate in the Late Cretaceous–Eocene, which drove thick-skinned deformation and uplift of the Rocky Mountains during the Laramide Orogeny (du Bray 2007). A second major felsic magmatic pulse in Nevada (approximately 90 Ma to 95 Ma) is associated with porphyry-style base-metal mineralization (du Bray 2007).

The Eocene was a time of significant igneous activity in the Great Basin. Widespread calc-alkaline arc-related magmatism in northern Nevada from approximately 43 Ma to 36 Ma resulted in the emplacement of stocks and dykes and eruption of volcanic rocks. A southwestward sweep through time of this magmatism has been proposed to be related to roll-back of the Farallon slab (Humphreys 1995). Eocene magmatism is widely thought to be related to mineralization in the Great Basin, including Carlin-type gold deposits (e.g., Henry and Ressel 2000) and metalliferous deposits in the Battle Mountain district (Huff et al. 2025).

At the end of the Laramide orogeny in the Eocene, Nevada shifted from a compressional tectonic setting to one dominated by extension. From the late Eocene onward, extensional stresses—expressed through widespread block faulting and tilting—became the primary force shaping the region. This tectonic regime produced the characteristic basin-and-range landscape of central Nevada, where north-northeast trending mountain ranges alternate with broad, flat-floored valleys. During this prolonged period of extension, several porphyry copper-gold systems formed, and many of northern Nevada’s Carlin-type gold deposits are also interpreted to have developed in this interval (approximately 36 Ma to 42 Ma; Cline et al. 2005).

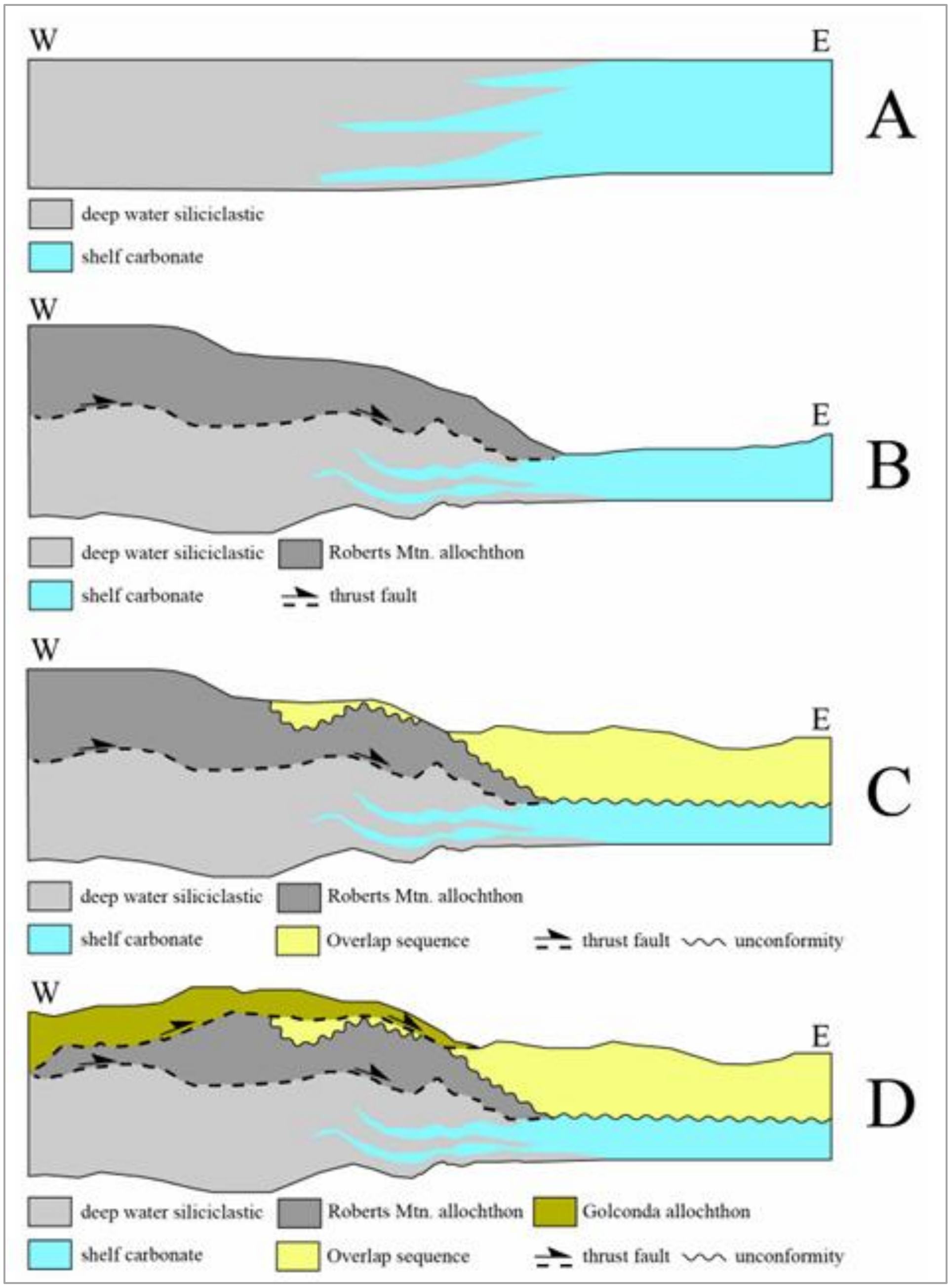
A timeline of the major geological and stratigraphic events in northern Nevada is shown in Figure 7-1 and a summary of tectonostratigraphic events is shown in Figure 7-2.

Figure 7-1: Timeline of Geological Events in Northern Nevada



Source: Modified from Wallace et al. 2024.

Figure 7-2: Tectonostratigraphic Events of Northern Nevada

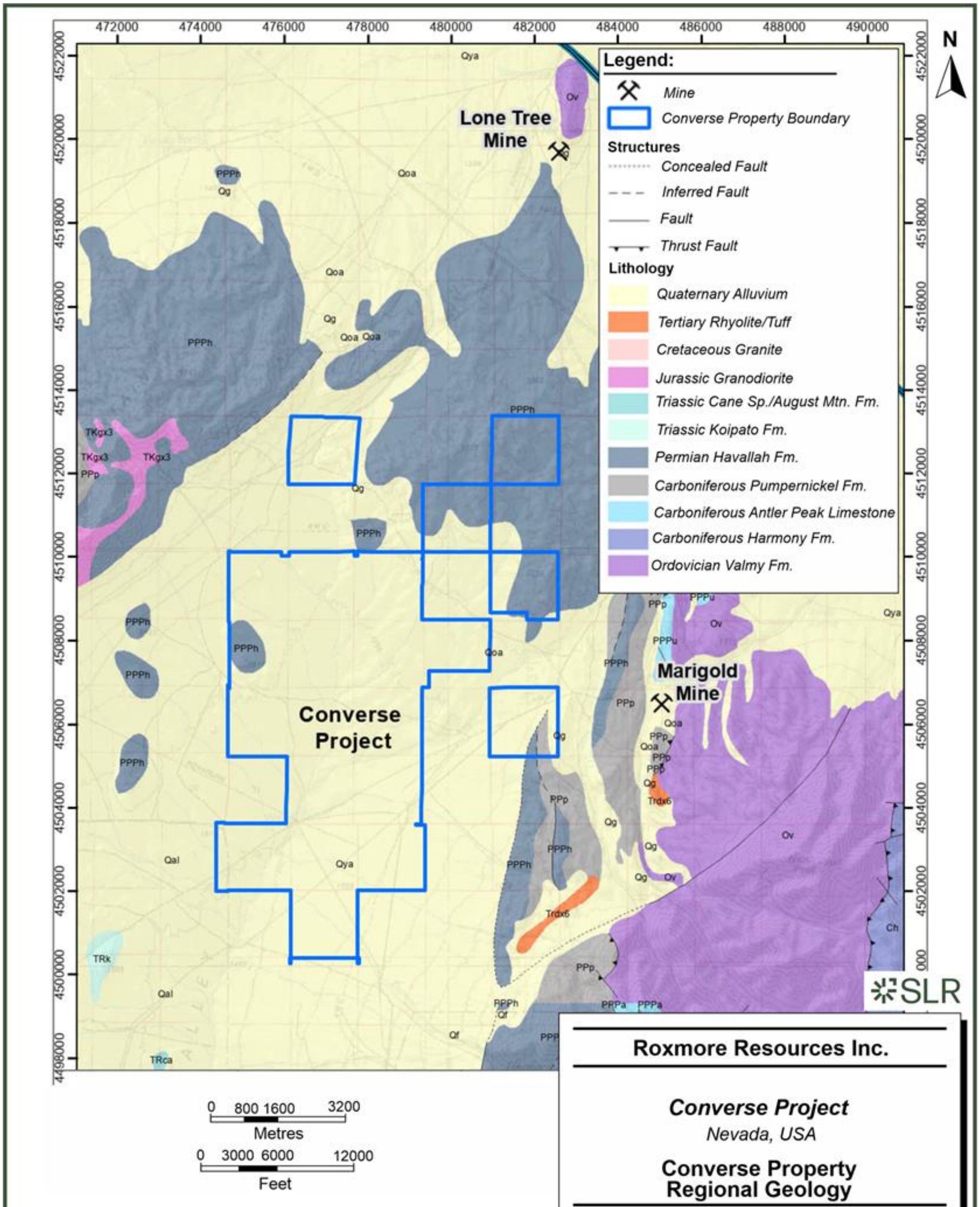


Source: Fithian 2005.

Note: A) Devonian, B) Devonian to Mississippian, C) Mississippian to Permian, and D) Permian to Triassic

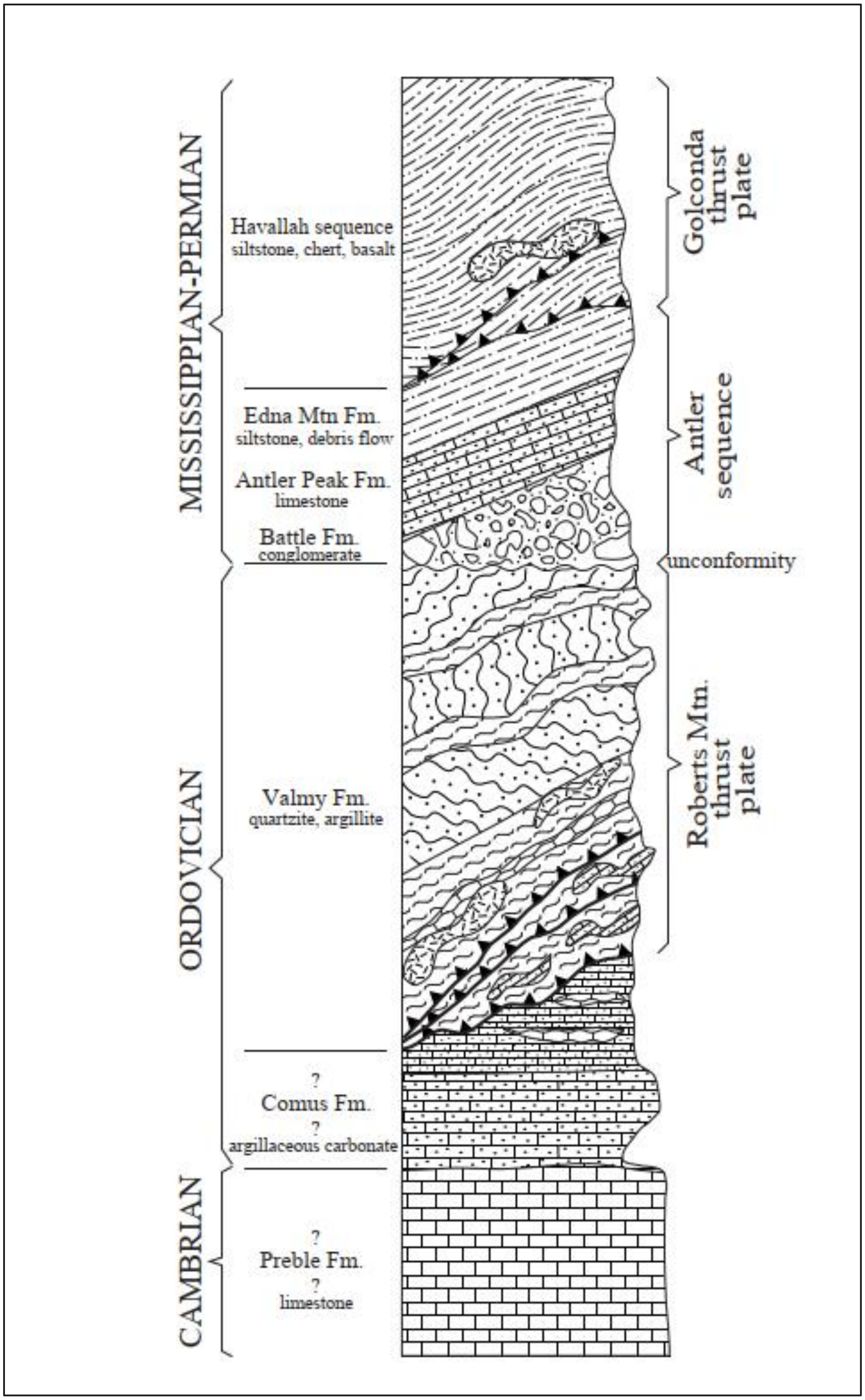
The regional geological setting of the Converse Property is shown in Figure 7-3. The Property is underlain by the structurally complex Havallah sequence that comprises the upper plate of the Golconda allochthon (Cleveland 2000). The Havallah sequence consists of basinal sedimentary rocks, including calcareous sandstone and siltstone, quartzite, pebbly limestone, siliceous siltstone, chert, argillite, and variable amounts of basalt and greenstone. The Havallah sequence overlies the Antler sequence, which includes the Middle Pennsylvanian Battle Formation, the Pennsylvanian and Permian Antler Peak Limestone, and the Permian Edna Mountain Formation. The Antler sequence overlies the siliciclastic sedimentary rocks of the Roberts Mountains allochthon. The stratigraphic location of the Havallah units are shown in Figure 7-4.

Figure 7-3: Converse Property Regional Geology



Source: Roxmore 2026.

Figure 7-4: Stratigraphic Location of the Havallah Sequence



Source: Fithian 2005, modified after Teal and Jackson 2002.

7.2 Property Geology

The following description of geology and mineralization at the Converse Property is based on previous studies and technical reports by Cleveland (2000), Srivastava et al. (2012), and Smith (2001), together with Roxmore's drill core logging and relogging observations and interpretations completed in 2025 and early 2026.

The Property is covered by approximately 20 m to 275 m of weakly to moderately consolidated Quaternary and late Tertiary alluvial and colluvial material. Surficial cover consists of variable proportions of sand, silt, clay, gravel, pebble gravel, cobbles, and local boulders. Clasts are derived from nearby bedrock units and include sedimentary rocks of the Havallah sequence, quartzite, chert, intrusive rocks, and locally volcanic material.

Beneath the surficial cover, the principal host rocks to the Redline deposits are siliciclastic and calcareous sedimentary rocks of the Havallah sequence. The Havallah package consists of interbedded calcareous sandstone, sandstone, siltstone, calcareous siltstone, quartz arenite, quartzite, mudstone, shale, chert, sandy limestone, limestone, and local conglomeratic intervals. Bedding is commonly preserved and generally dips shallowly to moderately west. The variable calcareous content of the Havallah sequence is an important control on alteration style, with calcareous units forming calc-silicate hornfels and skarn, and more siliciclastic units forming biotite hornfels and quartz hornfels.

The Havallah sequence is intruded by the Redline Stock and numerous dykes and sills. The Redline Stock is a large porphyritic granitoid, centrally located between North and South Redline zones. Dykes have been encountered throughout the project area and include mafic to intermediate, feldspar-hornblende ± quartz porphyry, hornblende-bearing porphyritic intrusive rocks locally logged as andesite and granodiorite.

One deep drill hole, CV25-002C, intersected a thick limestone-dominant unit below a broad low-angle damage zone interpreted as the Golconda Thrust or a related thrust structure. This limestone unit is interpreted as the Antler Peak Limestone and is composed primarily of limestone and subordinate sandy limestone, with minor cherty limestone and chert-bearing intervals. Where intersected, the unit is predominantly altered to pyroxene and pyroxene-garnet skarn and is locally cut by granodiorite dykes, dykelets, and/or sills that exhibit extensive endoskarn alteration.

Gold mineralization at Converse is interpreted to represent a reduced gold-skarn system. Mineralization is primarily hosted by prograde and retrograde skarn assemblages developed within reactive calcareous sedimentary rocks of the Havallah sequence and, locally, within endoskarn-altered intrusive rocks. Hornfelsed siliciclastic sedimentary rocks, hydrothermal breccia, potassic-altered dykes, and late vein systems also form locally important secondary hosts or overprints.

7.2.1 Converse Stratigraphy

Although drilling at the Project has largely targeted the Havallah sequence, one deep stratigraphic hole completed in 2025 indicates that units of the Antler Peak, Battle Mountain, and Valmy formations are preserved at depth and described below.

Valmy Formation (Ov): The Valmy Formation consists mainly of interbedded chert, quartzite/quartz arenite, shale, siltstone, sandstone, and heterolithic sandstone. Local calcareous sandstone, sandy limestone, or limestone intervals are present in CV25-002C, but carbonate-bearing beds are subordinate and are not considered defining features of the unit.

Multiple chaotic and deformed zones suggest tectonic disruption, and strong intermittent skarn alteration was still noted at the bottom of the hole at 1,446 m depth.

Battle Mountain Fm (PNBt): The Battle Formation is logged in CV25-002C below the Antler Peak Limestone and consists of a mixed conglomeratic, siliciclastic, and locally calcareous sedimentary package. The unit includes pebble to boulder conglomerate, calcareous sandstone, sandstone, siltstone, chert, shale/argillite, sandy limestone, and local limestone. Conglomeratic intervals commonly contain chert, quartzite, sandstone, and heterolithic clasts in a sandy to silty matrix. Calcareous sandstone and sandy limestone intervals are locally altered to calc-silicate and skarn assemblages. The interpreted Battle formation has only been intercepted in one drill hole to date on the Project.

Antler Peak Limestone (PNap): The Antler Peak Limestone is a carbonate-dominant sedimentary unit composed primarily of limestone and subordinate sandy limestone, with minor local cherty limestone and chert-bearing intervals. Sandy limestone intervals locally contain approximately 1 mm quartz grains and may grade into pebble-bearing or conglomeratic beds along the lower margin of the formation. Where intersected, the unit has been predominantly altered to pyroxene skarn and pyroxene-garnet skarn. Skarn-altered limestone and sandy limestone are typically dark green to black, whereas less intensely altered or hornfelsed intervals are pale green to grey. The unit is commonly massive to thick-bedded, with minor local medium-bedded sandy limestone intervals. Relict sedimentary features are locally preserved, including fossil hash beds and rare worm burrows.

The unit is locally cut by granodiorite dykes, dykelets, and/or sills, commonly with brecciated intrusive margins. These intrusive phases are predominantly endoskarned. A thin interval of mudstone, shale, chert, calcareous siltstone, and lesser limestone is logged above the Antler Peak Limestone and below the interpreted Golconda thrust, this may correlate with the Edna Mountain Formation. This interpretation is consistent with the regional stratigraphic position of the Edna Mountain Formation above the Antler Peak Limestone.

Havallah Sediments (PNhv): Havallah sediments comprise a heterogeneous package of interbedded siliciclastic and calcareous sedimentary rocks. Because little to no unaltered Havallah material is preserved within the deposit area, lithologic descriptions are interpreted primarily as protolithologies based on preserved primary textures, sedimentary features, and alteration/mineral assemblages. The dominant interpreted protolithologies within the Havallah package include calcareous sandstone, siltstone, sandstone, and calcareous siltstone. Subordinate interpreted protolithologies include quartz arenite, mudstone, quartz-rich sandstone or quartzite, sandy limestone, conglomerate, shale, limestone, chert, and calcareous shale. Overall, the Havallah sediments represent a compositionally variable siliciclastic–calcareous sedimentary sequence that provided favourable reactive horizons for variable hornfels, and skarn alteration at the Property.

Calcareous sandstones are variably pale to dark green, olive green, and yellow-green, depending on the intensity of metamorphism and/or metasomatism and the abundance of calc-silicate minerals present. Within the deposit area, most calcareous units have undergone at least hornfels metamorphism, and locally skarnization. Grain size ranges from fine to coarse grained, and the sand fraction varies from heterogeneous to quartz dominated. Bedding ranges from thin to thick bedded and is locally massive. Minor, normally graded beds are locally preserved. Quartz-rich sandstone and quartz arenite intervals are common within this package and locally contain pods, ribbons, or interbeds of dark green sandy limestone.

Siltstones are variably pale brown-grey to dark brown-grey, depending on the intensity of biotite hornfelsing and/or quartz hornfelsing. Bedding is predominantly thin, with minor medium to thick beds. Minor normally graded beds are locally preserved. Where more strongly metamorphosed, siltstones are commonly logged as biotite hornfels or quartz hornfels.

Sandstones are highly variable in colour, ranging from beige, brown, orange, and yellowish tones to grey. They are dominantly fine grained, with subordinate medium-grained and lesser coarse-grained intervals. Overall, sandstones are commonly massive, with minor medium- to thick-bedded intervals. Quartz arenite and quartz-rich sandstone intervals are locally common within the Havallah package and are generally quartz dominated, although heterolithic arenite and heterolithic pebble-bearing intervals are locally present. Quartzite and quartz hornfels are commonly logged where quartz-rich sandstone or siltstone protoliths have been recrystallized during contact metamorphism.

Calcareous siltstones are dominantly pale green to dark green, depending on the intensity of metamorphism and/or metasomatism. They are dominantly massive to medium bedded, with lesser thin- and thick-bedded intervals. Like the calcareous sandstones, these units are commonly altered to calc-silicate hornfels and locally to skarn assemblages.

Subordinate carbonate- and chemical-sedimentary lithologies include sandy limestone, limestone, calcareous shale, shale, and chert. These units occur as discrete beds, interbeds, pods, or ribbons within the broader siliciclastic package. Conglomeratic intervals are locally present and include quartz-rich and heterolithic pebble-bearing beds. The carbonate-bearing units are significant because they form the most reactive horizons for calc-silicate and skarn alteration.

The Havallah sequence is locally fractured, sheared, faulted, and broken, particularly in structurally disrupted intervals. Oxidation and weathering are variable and are commonly associated with fractured or carbonate-bearing intervals.

Redline Stock (Ti): The Redline Stock is a large, slightly elongate, steeply cylindrical intrusive body located in the central portion of the deposit area. It is modelled as approximately 500 m by 690 m in plan view, measured in the X and Y directions, respectively. In drill core, the Redline Stock is logged primarily as granodiorite porphyry, including in holes CONV-029C and CONV-034C.

The intrusive is generally porphyritic and varies from pale green to light grey, grey-brown, olive green, and locally orange-brown where oxidized. The Redline Stock has undergone variable alteration, ranging from weak to strong endoskarn overprinting, commonly superimposed on potassic alteration. Strongly endoskarned intervals are commonly pale green to light grey and locally bleached, whereas less altered intervals preserve a darker grey-brown colour.

Where primary intrusive textures are preserved, the granodiorite porphyry consists of a very fine-grained, medium brown-grey groundmass containing approximately 1 mm to 4 mm subhedral plagioclase feldspar and amphibole phenocrysts, with local quartz and biotite. Primary hornblende and biotite are locally preserved in less intensely altered intervals. The intrusive is locally fractured, oxidized, or broken along fault and vein zones.

Intrusive Rocks (Ti): Numerous intrusive units intrude the Havallah sediments, and Redline Stock. Ranging from what appear as mafic-intermediate composition fine grained dykes, amphibole-biotite porphyry dykes, and feldspar-quartz dykes.

Mafic to intermediate dykes occur as dark green, commonly aphanitic, thin dykes and/or sills that are typically strongly overprinted by endoskarn alteration. They locally contain millimetre-scale plagioclase feldspar crystals or feldspar-rich fragments, interpreted as possible relict phenocrysts, xenocrysts, or incorporated wall-rock material. This intrusive phase has been observed cutting the Redline Stock. Feldspar-hornblende ± quartz porphyry dykes are most common in the South Redline deposit, with minor occurrences observed in the North and West Redline areas. These intrusive rocks are characterized by feldspar and hornblende phenocrysts, with local quartz, hosted in a fine-grained to locally aphanitic groundmass. Colours are variable and range from grey, dark grey, and brownish-grey to grey-green and locally, these porphyries are bleached or pale green were affected by strong endoskarn alteration.

In the western portion of the Property, a distinct hornblende-bearing porphyritic intrusive phase is locally logged as andesite. This unit is characterized by quartz, feldspar, and hornblende phenocrysts within a fine-grained to aphanitic groundmass. In CV25-005C, the unit is observed cutting and incorporating a granodiorite porphyry phase that is texturally and mineralogically similar to the Redline Stock, with local granodiorite xenoliths present within the andesite. Similar andesite to hornblende-phyric porphyry material is observed within Breccia Pipe 01 (BP-01), where it occurs as clasts and locally as matrix within hydrothermal to possible magmatic breccia textures. These units are commonly affected by moderate to strong potassic alteration, expressed as biotite alteration of the groundmass and local potassium feldspar alteration.

A large porphyritic granodiorite dyke has been observed west of the main deposit area, including in CONV-030C and CV25-005C. The dyke is approximately 60 m to 70 m thick and is interpreted to dip steeply to the west. It is mineralogically and texturally similar to the Redline Stock and is characterized by feldspar phenocrysts with lesser primary hornblende and biotite. The dyke has undergone variable alteration, including extensive endoskarn alteration and local to pervasive potassic alteration. Potassic alteration is particularly evident where the granodiorite is cut by a quartz-feldspar-hornblende porphyry phase, and is expressed as secondary biotite with local potassium feldspar. The unit is generally grey-brown, pale green where strongly endoskarned, and red-brown to pinkish where potassically altered.

7.2.2 Structure

Several high- and low-angle faults have been identified at the Project through drill core logging, geological relationships, and mineralization patterns. Southwest- and south-southeast-striking, steeply dipping faults have been interpreted from high-confidence structural measurements collected from oriented diamond drill core. South-southeast-striking structures may represent an earlier structural zone that was subsequently reactivated, as supported by offsets to geological domains and the presence of gold mineralization developed along this orientation. Southwest-striking structures are interpreted to represent later basin and range-style faults. These interpretations have been extrapolated into areas of historical, non-oriented drill core were supported by logged cataclastic damage zones, lithologic relationships, alteration patterns, and grade-shell discontinuities.

Fault block domains are recognized from changes in bedding orientation measured in oriented core, indicating local rotation between structural blocks. This is particularly evident across interpreted late basin and range faults. Additional support for these domains comes from disruptions or offsets in elemental grade shells, including copper and gold. In drill core, these steeply dipping structures commonly appear as cataclastic damage zones ranging from centimetre-scale features to decametre-scale intervals. These damaged zones locally contain clay gouge, rubble, and cataclastic textures. One low angle shear zone was intersected in oriented diamond drill core in 2025. High confidence structural measurements indicate that this feature dips shallowly to the west. The structure is interpreted as a possible early thrust that may have been reactivated during later extension; however, there is currently insufficient evidence to confirm its origin or reactivation history.

In hole CV25-002C, a broad low angle shear zone interpreted to represent the Golconda Thrust or a related thrust structure was intersected at a vertical depth of approximately 900 m. The structure is expressed as a broad fault-damage zone extending across approximately 70 m of drill core, from approximately 856.8 m to 924.6 m. The zone includes multiple splays, cataclastic breccia, rubble, clay gouge, shearing, and localized sigmoidal textures. The most strongly developed fault intervals occur near 878.8 m to 882.4 m, 897.1 m to 901.4 m, and 907.0 m to 917.5 m, where chaotic breccia, gouge matrix, rubble, and discrete fault splays are logged.

Oriented core measurements indicate that slip surfaces within the zone are generally shallowly dipping, commonly between approximately 10° and 25°, with slickenlines indicating predominantly dip-slip movement. Locally steeper slip surfaces, dipping approximately 20° to 45°, indicate an oblique-slip component. The lowest interpreted slip surface dips approximately 10°, consistent with a very low angle shear structure. The interpretation of this zone as a possible Golconda Thrust or related structure is supported by the presence of Antler Peak Limestone below the shear zone, consistent with the regional position of the Golconda Thrust system.

Breccia and multi-phase breccia bodies were observed in several drill holes in the southwest Redline deposit (Figure 7-5 and Figure 7-6). These breccias are thought to have developed syn- to post-skarn development. This is supported by the presence of clasts of hornfels, and various degrees of skarnized clasts, as well as clasts of intrusive material that exhibit similar textural and mineralogical characteristics to the Redline Stock and associated intrusions.

Breccia Pipe 01 (BP-01) is characterized predominantly by hydrothermal breccia, with local intrusion-related or possibly magmatic breccia textures. Core logging shows that the breccia ranges from clast-supported to matrix-supported and contains angular to subrounded clasts of quartz hornfels, biotite hornfels, pyroxene hornfels, pyroxene skarn, calcareous sandstone, siltstone, sandstone, and minor intrusive material. The breccia matrix and infill commonly consist of quartz/silica, actinolite, tremolite, carbonate, clay/nontronite, and locally chlorite or biotite.

Local intrusion-breccia textures occur where hornblende-phyric and endoskarn intrusive phases form part of the breccia matrix. These features suggest that BP-01 includes a subordinate intrusive or magmatic breccia component in addition to the dominant hydrothermal breccia.

The breccia body is interpreted to record multiple brecciation and hydrothermal events. This interpretation is supported by discrete breccia intervals within the broader BP-01 domain, locally separated by more competent sedimentary, hornfelsed, or skarn-altered intervals. These competent zones may represent internal screens or larger wall-rock blocks preserved within the breccia body. Variability in gold grades across breccia intervals further indicates that not all brecciation events were equally mineralizing.

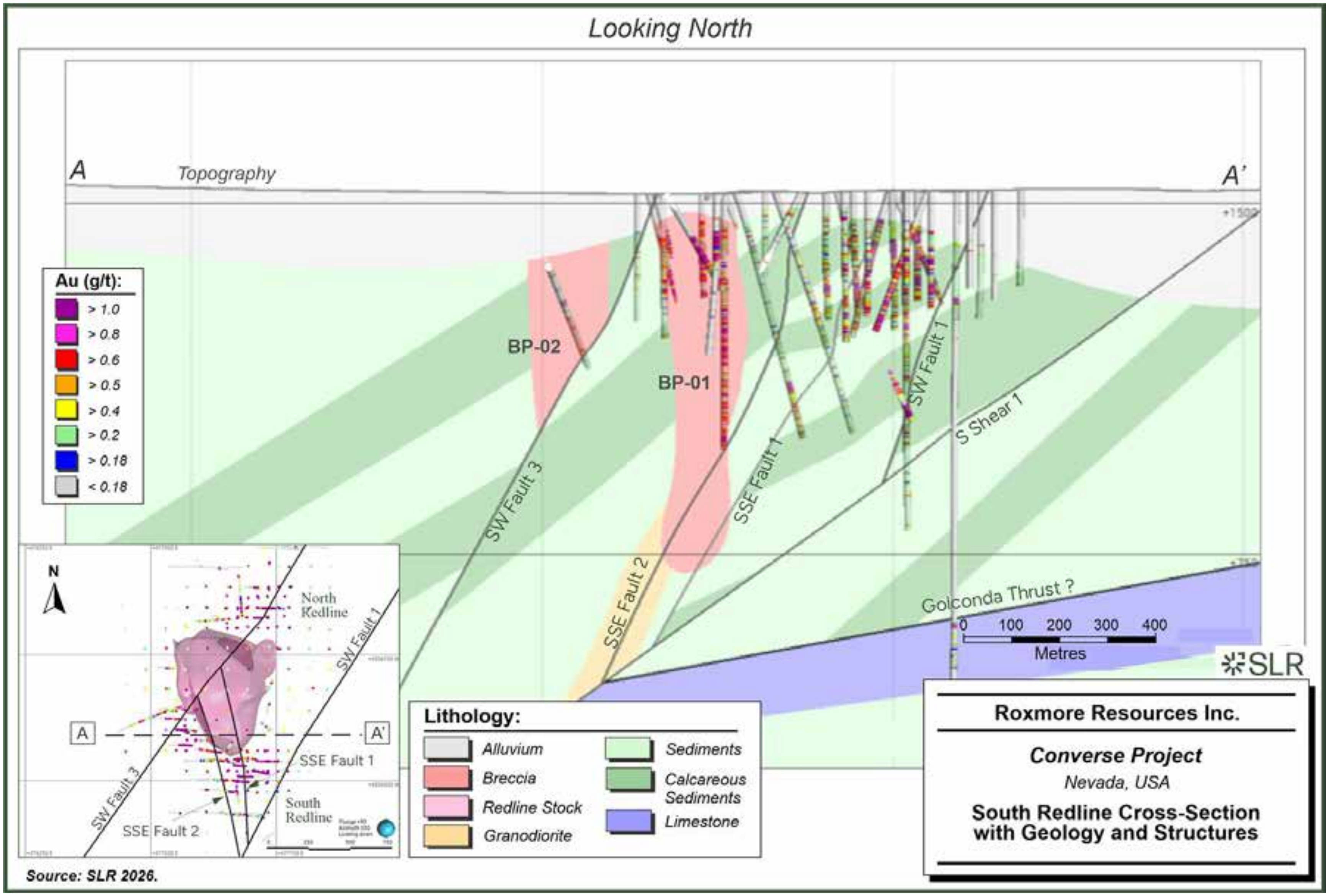
Breccia Pipe 02 (BP-02) is interpreted as the structurally displaced upper portion of BP-01, separated from the main breccia body by displacement along the Southwest-Fault 3 basin and range fault (Figure 7-6). BP-02 is characterized predominantly by hydrothermal breccia and exhibits stronger weathering and oxidation than BP-01. Core logging shows that the breccia ranges from clast-supported to matrix-supported, but original textures are commonly degraded by oxidation and weathering. Several intervals preserve silica ± carbonate hydrothermal cement and matrix infill, indicating that the unit represents a more weathered and oxidized equivalent of the adjacent BP-01 breccia system.

A major northwest-southeast-trending structural zone is interpreted from historical aeromagnetic data and is considered to pre-date mineralization (Figure 7-7). Cleveland (2000) suggested that the Redline Stock was emplaced near the intersection of this pre-existing northwest-southeast structural zone and one or more south- to southwest-striking structures. Current geological interpretation supports this relationship, as South Redline mineralization, the associated dyke complex, skarn alteration, and BP-01 are broadly aligned along this northwest-southeast-trending aeromagnetic feature. This relationship suggests that the structure may have influenced intrusive emplacement, hydrothermal fluid flow, and the localization of mineralization.

Figure 7-5: Breccia (BP-02) from Drill Hole CV25-005 at 748 m Depth

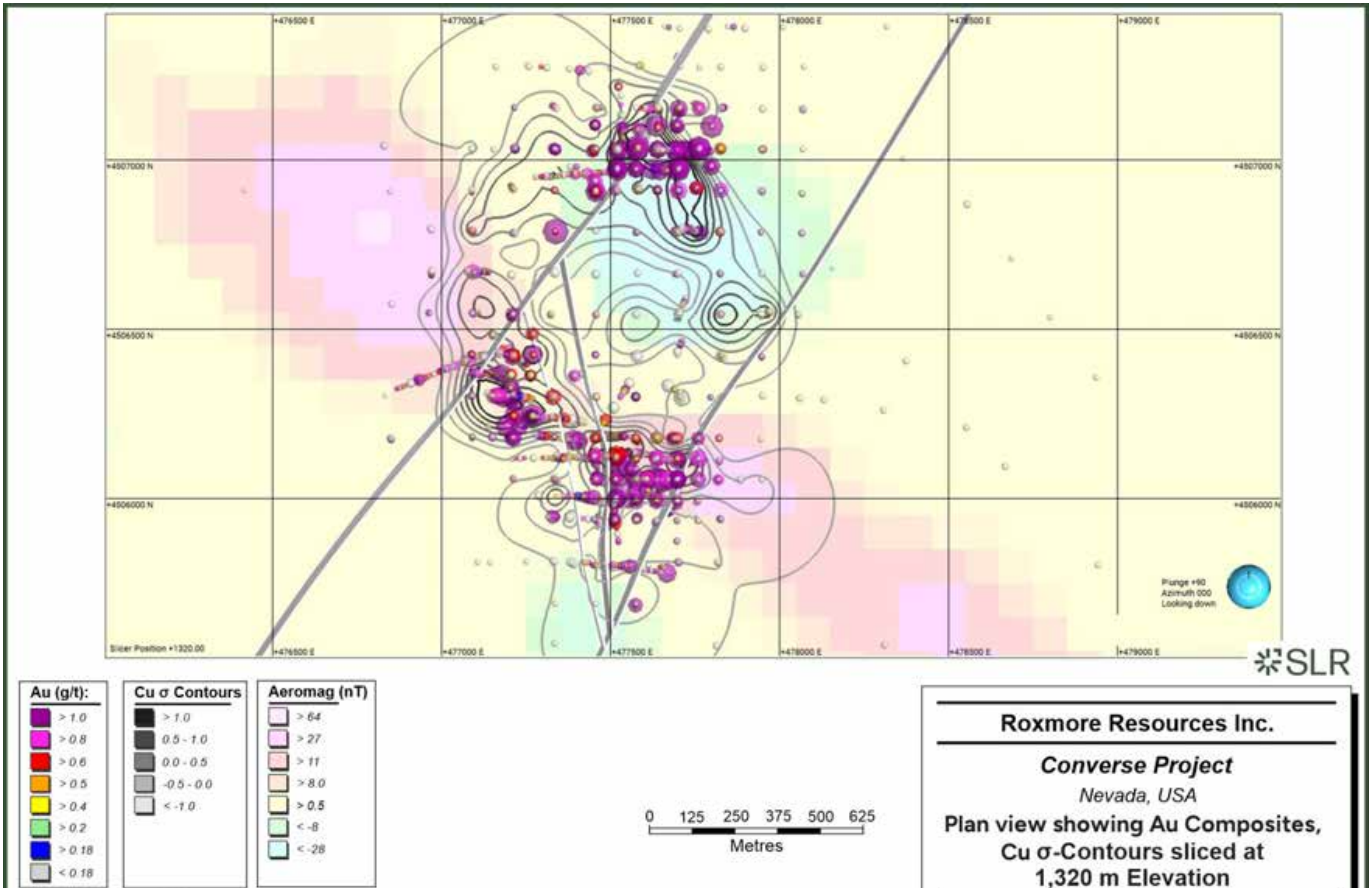
Source: SLR 2025.

Figure 7-6: South Redline Cross Section with Geology and Structures



Source: SLR 2026.

Figure 7-7: Plan View showing Au Composites, Cu σ-Contours sliced at 1,320 m Elevation



Source: Roxmore 2026.

Notes: Fault-surface intersections shown at the 1,320 m elevation, and regional aeromagnetic data. The relationship supports a broad NW-SE structural control on mineralization.

7.2.3 Metamorphic/Alteration

Numerous metamorphic/alteration styles have been observed at the Property, including early hornfels contact metamorphism, endoskarn alteration, prograde skarn alteration, retrograde skarn alteration, and potassic alteration.

Hornfels represents the earliest recognized thermal metamorphic overprint of the Havallah sedimentary package at Converse. Hornfels assemblages are strongly protolith-dependent and are interpreted to have formed during emplacement of the Redline Stock, which thermally recrystallized the original sedimentary rocks and produced a contact metamorphic aureole around the intrusive complex (Smith 2001).

In the deposit area, calcareous sandstone and calcareous siltstone are commonly metamorphosed to calc-silicate hornfels, whereas more argillaceous and siliceous sedimentary rocks are commonly metamorphosed to biotite hornfels and quartz hornfels. These hornfels assemblages generally reflect the composition of the original sedimentary protolith, with calcareous units producing calc-silicate-rich assemblages and more siliciclastic units producing biotite- and quartz-rich hornfels assemblages.

Smith (2001) with reference to Meinert (1992) described hornfels development as early in the genetic history of the South Redline skarn system. Biotite hornfels was interpreted as a contact metamorphic assemblage related to emplacement of the Redline Stock. This was followed by pyroxene-bearing calc-silicate hornfels, which Smith interpreted as transitional between largely isochemical hornfels development and metasomatic skarn formation. Pyroxene hornfels developed from carbonate-cemented sandstone was interpreted to be primarily metamorphic in origin, whereas pyroxene hornfels developed in siltstone may have required minor metasomatic addition of calcium (Ca), iron (Fe), and silica (Si). Calc-silicate hornfels is distinguished from metasomatic skarn alteration by its generally lighter colour and finer-grained texture, reflecting lower iron, more diopsidic pyroxene compared with later skarn pyroxene.

Based on current and historical logging, moderate to strong hornfels metamorphism extends approximately 350 m north and at least 400 m south of the Redline Stock. Drilling farther south is limited, so the full southern extent of the hornfels aureole is not yet well constrained. Weak hornfels metamorphism is locally observed more than 500 m north of the stock.

Prograde skarn alteration overprints earlier hornfels assemblages and is dominated by pyroxene with lesser garnet. Pyroxene is light to dark green, locally pale green to white or olive green, and is dominantly fine grained. Garnet is much less abundant than pyroxene and ranges from light red-pink to dark red-brown, medium brown, and locally pale green. Garnet is commonly fine grained, with local medium grained intervals, and is generally best developed proximal to interpreted fluid pathways, dykes or sills, intrusive contacts, and zones of stronger skarn development.

Prograde skarn mineralogy and intensity are strongly controlled by host rock composition and proximity to fluid pathways, including structures, dykes, sills, and intrusive contacts. In North Redline, prograde skarn is interpreted to be focused around a central feeder-style structural zone characterized by massive skarn, although much of this assemblage has been overprinted or replaced by later retrograde alteration. In South Redline, prograde skarn is spatially associated with a porphyritic dyke/sill complex and related fluid pathways. Moderate prograde skarn is also developed along the contact of the Redline Stock, where it extends outward into the Havallah sedimentary package for up to approximately 30 m.

Retrograde skarn alteration overprints earlier hornfels, calc-silicate hornfels, and prograde skarn assemblages. Retrograde alteration is dominantly expressed as chlorite-carbonate replacement of earlier pyroxene- and garnet-bearing assemblages, and occurs as irregular blebs, patches, veinlets, and fracture-controlled selvages within sedimentary units and skarn-altered intervals. Proximal to interpreted feeder structures and causative dykes, the retrograde assemblage locally contains greater amounts of actinolite and epidote.

Retrograde alteration is typically medium to dark green where chlorite and actinolite are dominant, and light green where epidote is present. Textures are commonly very fine to fine grained, although coarser fracture- and vein-controlled alteration is locally developed. The intensity of retrograde alteration is strongly controlled by host lithology, permeability, and proximity to interpreted fluid pathways, including structures, dykes, and skarn feeder zones.

Endoskarn alteration is common within intrusive phases throughout the Project area. Smith (2001) noted that all intrusive phases examined at South Redline exhibited some degree of endoskarn alteration, although that study was limited to a relatively small portion of the broader Project area. Endoskarn alteration is typically expressed as patchy to pervasive replacement of primary intrusive minerals by pyroxene, locally accompanied by biotite, actinolite, chlorite, epidote, carbonate, and subordinate garnet. In drill core, endoskarn-altered intrusive rocks are commonly pale green to dark green, with alteration intensity ranging from weak, patchy replacement to pervasive overprinting of the original intrusive texture.

Potassic alteration is observed throughout the Project area and is most commonly developed within intrusive rocks. It is typically expressed as fine-grained to aphanitic secondary biotite, producing a light red to reddish-brown pervasive wash through altered intrusive intervals. Local potassium feldspar alteration is also observed.

The timing relationship between potassic alteration and endoskarn alteration is variable. In some intervals, potassic alteration appears to be overprinted by endoskarn alteration, whereas in other intervals potassic alteration appears to overprint earlier endoskarn assemblages. These relationships suggest either temporally overlapping alteration events or multiple phases of potassic and/or endoskarn alteration.

Locally, potassic alteration is associated with elevated gold grades. For example, in CONV-030C, a potassic-altered amphibole-phyric dyke returned 1.1 g/t Au over 1.5 m. Additional work is required to determine whether this association is systematic or locally controlled by lithology, structure, or fluid pathways.

7.3 Mineralization

The Property hosts two gold-rich skarn deposits known as North Redline and South Redline (Figure 7-8). Gold mineralization at North Redline is approximately 670 m long and 200 m to 350 m wide, with mineralization extending to depths of up to 650 m below surface. The south Redline footprint is approximately 670 m x 260 m and extends to a depth up to 650 m below surface. Mineralization at the Project is interpreted to represent a reduced gold-skarn system. Gold mineralization is primarily hosted by prograde and retrograde skarn assemblages developed within reactive calcareous sedimentary rocks of the Havallah sequence and, locally, within endoskarn-altered intrusive rocks. Hornfelsed siliciclastic sedimentary rocks also host gold locally, but skarn-altered domains generally contain the strongest and most continuous mineralization.

Sulphide abundance is generally low, commonly less than 0.5%, and is dominated by pyrrhotite, with subordinate chalcopyrite and pyrite, and local molybdenite, sphalerite, galena, and arsenopyrite. Smith (2001) described pyrrhotite as the most abundant sulphide at South Redline, occurring as disseminations and veinlet-hosted aggregates, with deposition beginning early in the skarn system and continuing through prograde and retrograde alteration stages.

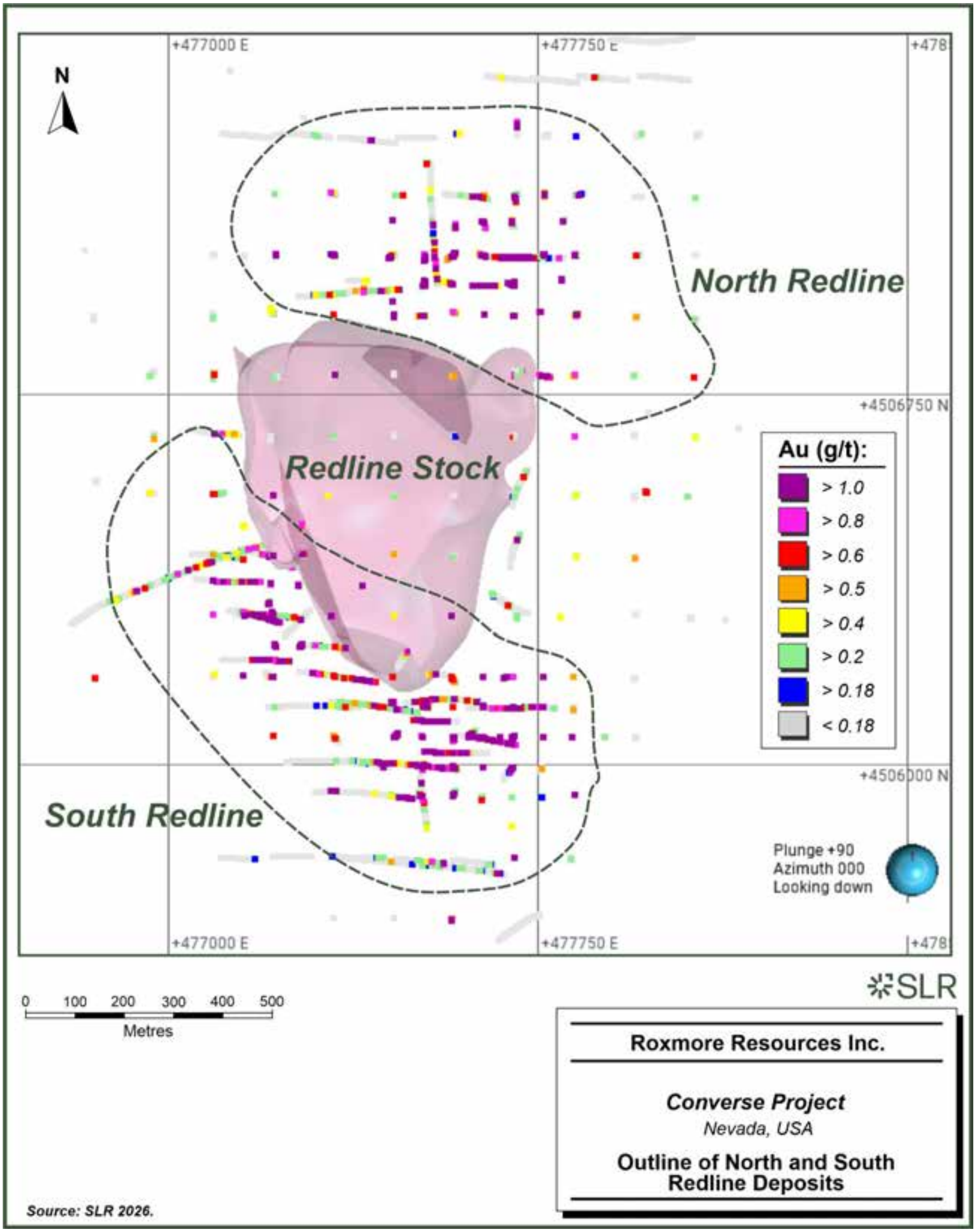
Chalcopyrite commonly accompanies pyrrhotite but is also observed with later pyrite-dominant sulphide assemblages. Pyrite is more commonly associated with retrograde veinlets and locally replaces pyrrhotite on a limited scale. These relationships are consistent with the positive Au-Cu relationships observed in several alteration domains and suggest that copper-bearing sulphide mineralization occurred during multiple stages of the hydrothermal system.

Gold is most strongly and continuously developed within skarn-altered domains. Both prograde and retrograde skarn assemblages contain elevated gold; however, Smith (2001) interpreted the main stage of gold deposition to post-date earlier pyroxene hornfels and pyroxene-garnet skarn assemblages. Smith described gold occurring as its native form and electrum along gangue grain boundaries in quartz-sulphide veinlets, with bismuth and tellurium minerals accompanying each gold occurrence. Smith further described the main stage of gold deposition as fracture-controlled fluid propagation that cuts earlier hornfels and later pyroxene-garnet skarn. These gold-bearing veinlets contain sulphide and retrograde assemblage minerals, and their selvages locally replace prograde pyroxene and garnet in the wall rock.

Gold mineralization intersected in alluvium material that is situated immediately above the contact with the bedrock and is assumed to be contained in clast-supported horizons.

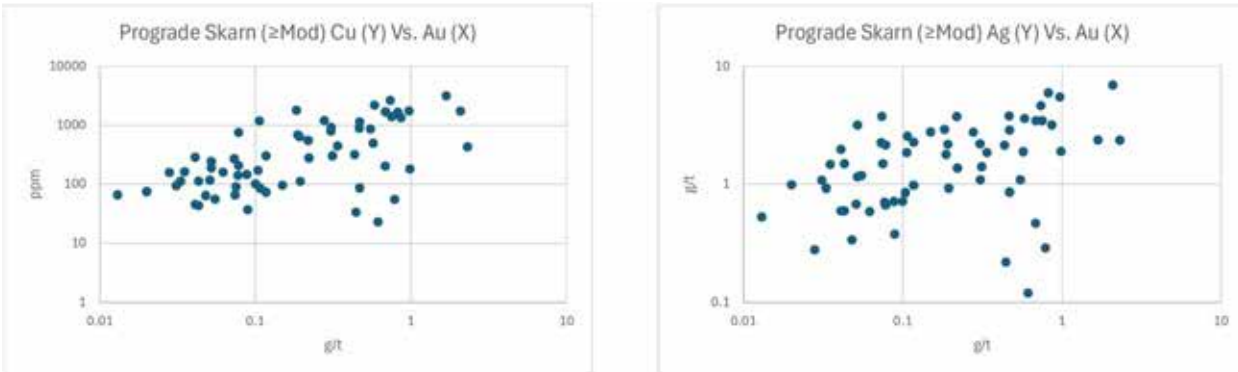
Preliminary review of multi-element geochemistry indicates that gold mineralization at Converse is commonly associated with copper enrichment. Positive Au-Cu relationships are observed in prograde skarn, retrograde skarn, quartz hornfels, biotite hornfels, and calc-silicate hornfels developed in silty units. These relationships, shown in Figure 7-9 and Figure 7-10, suggest that gold is closely associated with copper-bearing sulphide mineralization across multiple alteration domains.

Figure 7-8: Outline of North and South Redline Deposits



Source: SLR 2026.

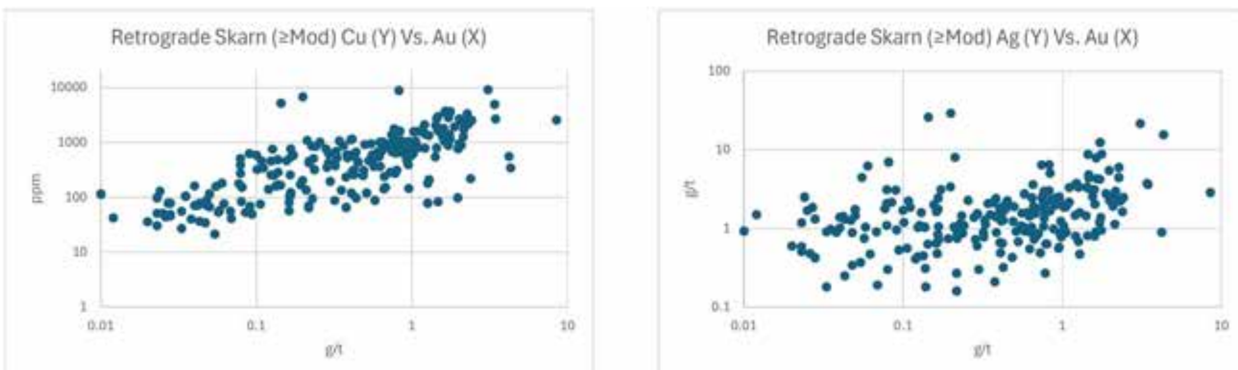
Figure 7-9: Log-log Scatter Plots Showing Au-Cu and Au-Ag Relationships for Moderate to Strong Prograde skarn Intervals



Source: Roxmore 2026.

Notes: Prograde skarn shows a positive Au-Cu relationship, indicating an association between gold and copper-bearing sulphide mineralization.

Figure 7-10: Log-log Scatter Plots showing Au-Cu and Au-Ag relationships for moderate to strong retrograde skarn intervals.



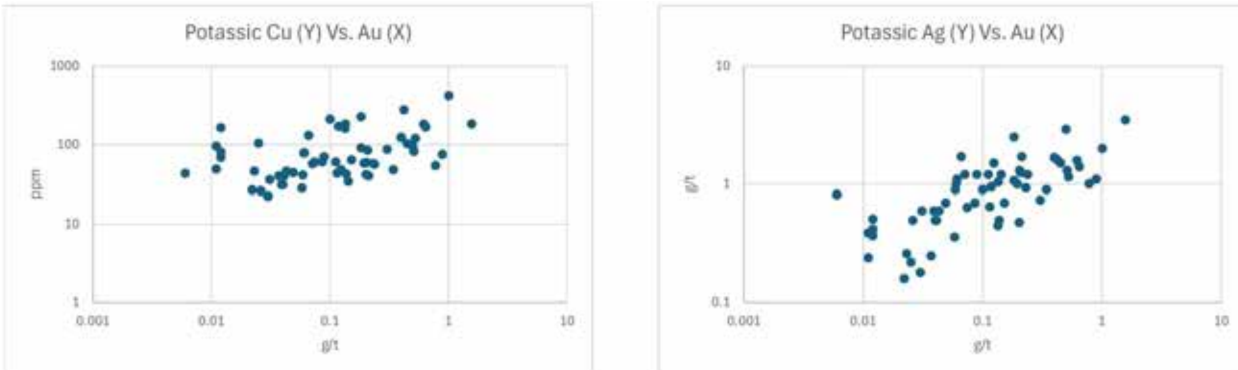
Source: Roxmore 2026.

Notes: Retrograde skarn shows a positive Au-Cu relationship, indicating an association between gold and copper-bearing sulphide mineralization.

Hornfelsed sedimentary rocks host lower-grade but locally significant gold mineralization. Quartz hornfels and biotite hornfels show positive Au-Cu relationships, suggesting that competent siliciclastic hornfels units may have acted as brittle hosts or conduits for Cu-Au-bearing hydrothermal fluids. Gold mineralization in hornfels is generally more variable and less continuous than in skarn-altered domains.

Silver shows a positive but more variable and domain-dependent relationship with gold than the Au-Cu relationship. Globally, Au-Ag values show a weak to moderate positive relationship, but the strength of the relationship varies between alteration domains. Calc-silicate hornfels, as well as potassic-altered dykes and magmatic breccia west of the main deposit area, show stronger Au-Ag relationships than Au-Cu relationships (Figure 7-11). Review of logged redox domains indicates that silver grades are generally higher in oxide and transition intervals than in primary sulphide intervals, suggesting possible redox-related redistribution or supergene enrichment of silver; however, silver is also associated with hydrothermal and vein-related mineralization, and Smith (2001) noted the presence of electrum, although its relative abundance was not quantified. Additional work is required to determine the relative importance of primary hydrothermal silver mineralization versus later redox-related redistribution.

Figure 7-11: Log-log Scatter Plots Showing Au-Cu and Au-Ag Relationships in Potassic-Altered Intervals



Source: Roxmore 2026.

Notes: Potassic alteration shows a stronger Au-Ag relationship than Au-Cu relationship, suggesting a distinct or overlapping Au-Ag association relative to the main copper-bearing skarn domains.

Vein-controlled sulphide mineralization includes arsenopyrite ± sphalerite ± galena-bearing quartz-carbonate stringers and molybdenite-bearing quartz veins. Arsenopyrite-bearing quartz-carbonate stringers are typically centimetre-scale and appear to represent a relatively late vein event, as they are observed cross-cutting earlier skarn and hornfels domains. These veins are most abundant in the North Redline deposit area and are commonly associated with elevated gold and silver grades. Molybdenite occurs in quartz veinlets throughout the project area and is locally associated with chalcopyrite. Smith (2001) described molybdenite in quartz veins cutting altered granodiorite dykes, locally associated with calc-silicate minerals, and in quartz-sulphide veinlets cutting skarn assemblages and earlier hornfels, suggesting both early and late stages of molybdenite deposition.

Overall, gold mineralization at Converse is primarily controlled by skarn development, sulphide deposition, structural permeability, and proximity to intrusive phases and interpreted feeder zones. Skarn-altered domains represent the principal mineralized host, while hydrothermal breccia, hornfelsed sedimentary rocks, potassic-altered dykes, and late vein systems form locally important secondary hosts or overprints.

Gold mineralization intersected in alluvium material that is situated immediately above the contact with the bedrock and is assumed to be contained in clast-supported horizons.

7.4 Redox

Three redox zones are observed at the Converse Property and include an oxide, transition, and primary or sulphide zone.

Oxide is defined at the Property as zones where no sulphides are present and where the host rock is pervasively oxidized, and/or strongly oxidized along fractures, with iron oxidized to various reddish-brown and brown-yellow colours. The oxide zone has a variable vertical depth profile ranging from 10 m to greater than 100 m below the base of alluvium, with localized oxide zones along interpreted structural zones extending to depths greater than 400 m below the bedrock contact. Goethite is the dominant iron-oxide mineral (Muerhoff et al. 2002).

The transition zone comprises both iron oxide and sulphide minerals and underlies the oxide zone. The vertical depth profile of this zone ranges from several meters to greater than 800 m along interpreted structural zones.

The primary, or sulphide, zone consists of sulphide minerals and generally underlies the transition zone. Isolated pods of sulphide zone material may also occur within oxide material, observed in the North Redline deposit.

The current redox model was developed using the logged redox zones described above. Roxmore added 1,135 gold cyanide leach assays (AuCN) from drilling completed in 2025, which has contributed to further refinement of the Redox model. The cyanide leach to fire assay ratio (AuCN/AuFA) has been used as a qualitative guide and was not applied to define the current redox domains as the spatial distribution of AuCN assays is not sufficiently dense to support domain modelling.

8.0 Deposit Types

The Converse Property is being explored primarily for skarn mineralization and, specifically, gold skarn mineralization, which includes the currently identified Redline deposits. The following is a brief summary of a skarn deposits, and precious metal skarns in particular, after Meinert (1993).

Skarns are a category of intrusion-related mineral deposits that occur world-wide and have been mined for a wide variety of commodities including Fe, tungsten (W), Cu, lead (Pb), zinc (Zn), molybdenum (Mo), Au, Ag, uranium (U), rare earth elements (REE), fluorine (F), boron (B) and tin (Sn). Skarns can develop in shallow and deep crustal levels in a variety of geological settings. The common characteristic of skarn deposits is the predominance of calc-silicate mineralogy, which normally includes garnet and pyroxene. Skarn formation is a dynamic process affected by many variables, including temperature, pressure, and host-rock chemistry as well as the chemistry of the intrusion(s) and the mineralizing fluids they generate. Large skarn systems are typically characterized by several phases of 'skarn' development from early, normally isochemical, hornfels phase; followed by structurally- and/or stratigraphically-controlled reaction skarn development; then a main phase of proximal, metasomatic, coarse-grained skarn development at peak temperatures; followed by retrograde skarn development as temperatures cool.

Precious metal skarns are often related to ilmenite-bearing granodioritic plutons or intrusive complexes. The skarn mineralogy is generally dominated by iron-rich mineralogy including hedenbergitic pyroxene and intermediate (grossular to andraditic) garnets. Other common minerals include potassium feldspar, scapolite, vesuvianite, apatite and high-chlorine aluminous amphibole. Distal, or early-stage alteration, can often include significant potassic (k-feldspar ± biotite) hornfels development. Arsenopyrite and pyrrhotite are the most common sulphide minerals associated with precious metal mineralization.

The Redline deposits at Converse are interpreted as gold-rich skarn deposits. Smith (2001) interpreted that gold mineralization occurs during the propagation of retrograde fluids along fracture networks. The skarn assemblages developed subsequent to the emplacement of a Redline intrusive stock. Alteration minerals occur mainly as replacements of carbonate minerals in the matrix of calcareous sandstones and also as cross-cutting veinlets. As observed in drill core, much of the prograde skarn replaces bedding planes (dipping shallowly to the west).

9.0 Exploration

9.1.1 Historical Data Compilation and Project Database Construction

In October 2024 the Datashed database was acquired from Waterton and GeoSequel was purchased. All database objects were migrated from Datashed to GeoSequel by GeoSequel staff using the following processes:

- A careful validation of all drill tables began in November 2024. Full digitization and validation of all drill data were undertaken by Sanford Information Systems, LLC.
- All paper records were first organized and scanned at high resolution. Following scanning, assay certificates were processed using optical character recognition (OCR) software, exported to Microsoft Excel (CSV), and subsequently validated. All certificate information and associated metadata—including analytical methods, quality control data, laboratory name, report dates, and units of measure—were captured. The resulting assay CSV files were then imported into the database. ALS Geochemistry certificates from 2007 to 2017 were downloaded directly from Webtrieve and processed within GeoSequel.
- All collar coordinate surveys (local grid) were located and transformed to NAD1983 UTM Zone 11N using ArcGIS software and the local grid definition file by Loyle Olson. Once transformed to UTM metres, all collar coordinates were imported into GeoSequel and verified against various generations of air photos.
- All downhole survey files were OCR'd, converted to csv, and validated. All surveys, survey dates, and metadata were captured in GeoSequel.
- All geologic logs were digitized into GeoSequel Logger using standard Alteration, Lithology, Mineral, and Structure tables. All comments were also digitized.

The database compilation project was completed in February 2026.

9.1.2 Surface Exploration

The Property is mostly undercover; however, there is minimal outcrop exposure located in the northern part of the claim area. No geologic mapping has been completed on the Property, and only limited rock sampling has been conducted. A soil orientation survey was completed in 1995 and is summarized in Table 9-1. The survey was conducted over the Redline deposit, and samples were analyzed by US Mineral Labs using enzyme leach method. Some samples returned elevated gold values, the study area was restricted to the known deposit area. Additional lag sample programs were undertaken during this period, but the data has not been compiled.

Table 9-1: Summary of Historical Soil Sampling Survey

Operator	Period	Survey Notes	Survey Details
Romarco Nevada Inc. (Romarco)	1995	Completed an enzyme leach orientation geochemical survey.	3 lines, 121 samples

9.1.3 Geophysics

Several geophysical surveys have been conducted at the Project, as summarized in Table 9-2. Historical documents have been reviewed, including survey work dating back to the 1990s. The raw data from surveys completed by earlier operators is available only in paper format. Not all historical survey lines and geophysical data have been digitized, so their locations over the Converse property are unknown. The 2003 Zonge Geoscience survey was largely conducted south of the current property boundary.

Table 9-2: Historical Geophysical Surveys

Operator	Period	Survey Notes
Chevron Resources (Chevron)	1989-1991	Carried out gravity and gradient array induced polarization (IP) surveys.
Cyprus Mines Corp. (Cyprus)	1991-1992	Acquired ground magnetic data.
Romarco Nevada Inc. (Romarco)	1995	Completed targeted gravity surveys.
Private Survey	1995	PRJ flew a magnetic survey over its Battle Mountain survey grid; the survey covered the Converse claim block.
Metallic Ventures Gold Inc. (MVG)	2003	Zonge Geoscience completed CSAMT surveys. Only one line crossed the current property boundary.

10.0 Drilling

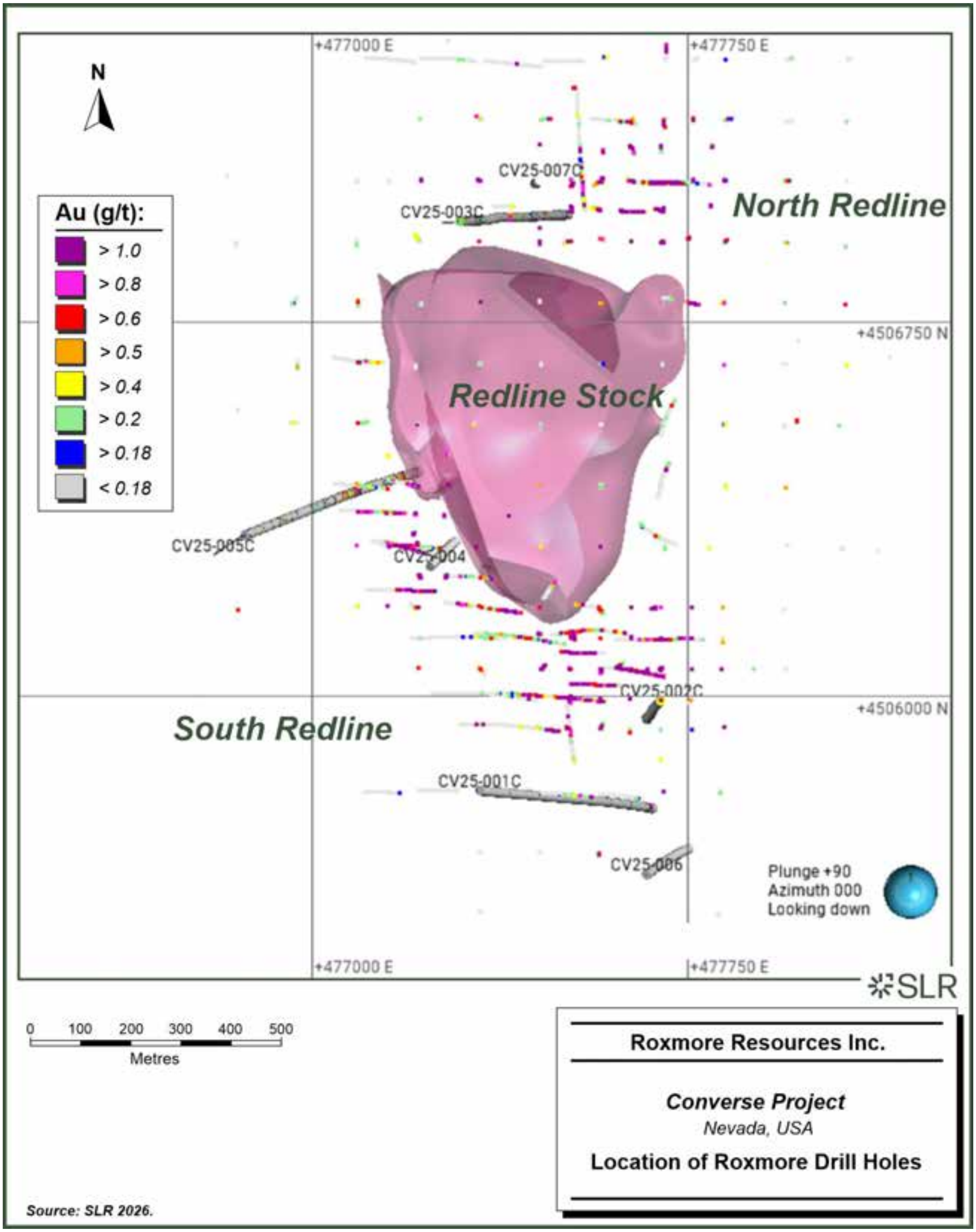
10.1 Summary

Roxmore, formerly operating under the name Axcap, drilled 4,614 m from seven holes in 2025. Drilling is summarized in Table 10-1 and collar locations are shown in Figure 10-1. Several core holes were drilled with RC pre-collars through the overburden.

Table 10-1: Summary of Roxmore Resources Converse Project Drilling

Company	Year	RC		RC/DD		DD		Total	
		No.	Metres	No.	Metres	No.	Metres	No.	Metres
Roxmore	2025	2	285	3	3,243	2	1,086	7	4,614
Total		**2**	**285**	**3**	**3,243**	**2**	**1,086**	**7**	**4,614**

Figure 10-1: Location of Roxmore Drill Holes

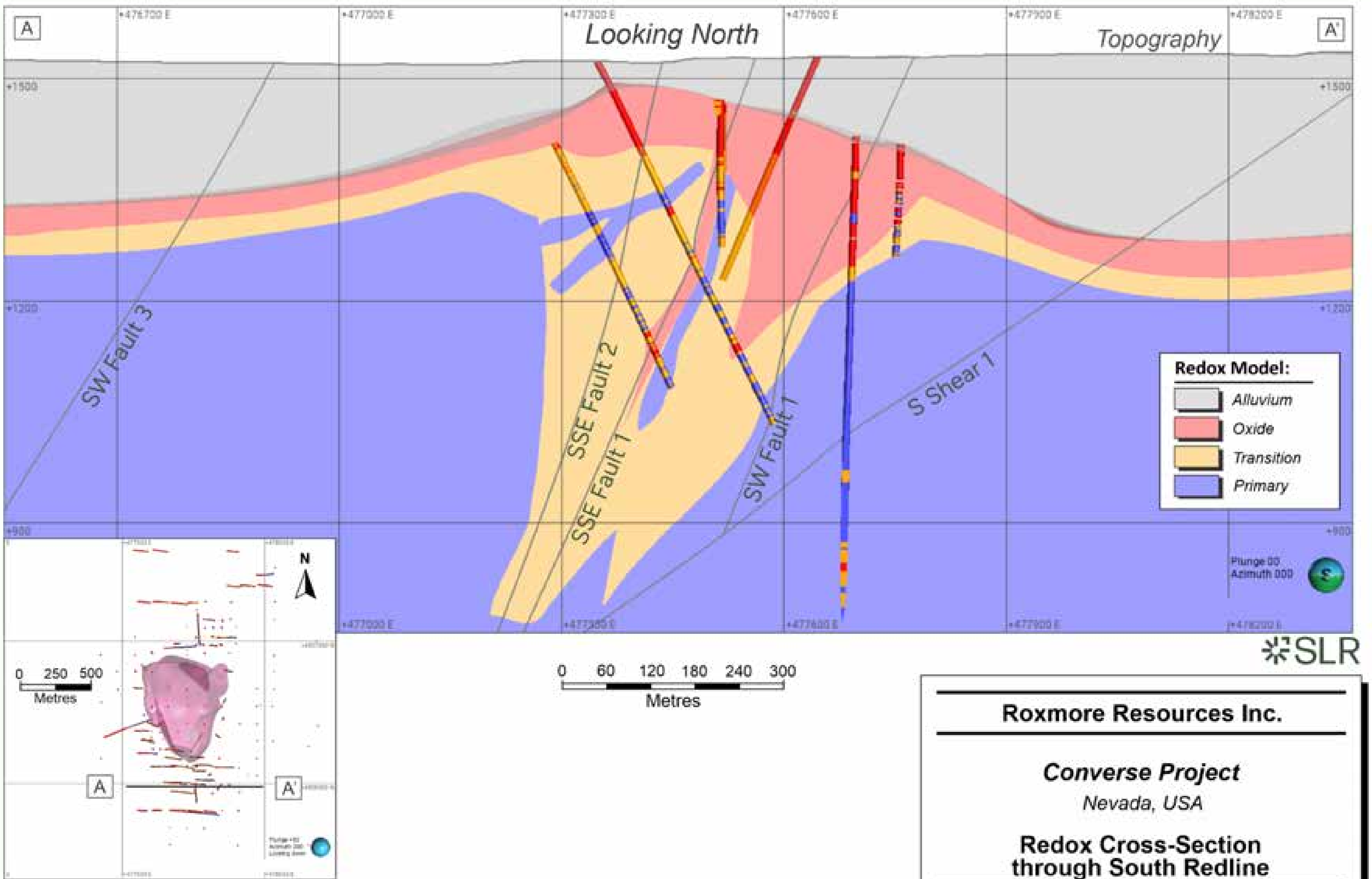


Source: SLR 2026.

Roxmore's 2025 drill program focused on validating historical results through the drilling of twin holes, and on testing new exploration concepts. The Company completed oriented core drilling to accurately measure structural orientations, providing a more robust structural interpretation and geological model. Drill hole CV25-007C was a twin of hole NK-087, both vertically drilled located in the North Redline deposit. Both holes were downhole surveyed, giving the QP confidence that any deviation has been recorded. The twin hole stayed within 5 m of the original hole.

Revised structural models were also key to interpreting zones of deep oxidation encountered in several holes and allowing the Company to refine earlier redox interpretations Figure 10-2.

Figure 10-2: Revised Redox and Structural Modelling at South Redline



Source: SLR 2026.

10.2 Drilling Description

Roxmore drilled seven holes in 2025. RC drilling was done by National EWP of Elko, Nevada, using a track-mounted Schramm T685 rig. Drill holes CV25-002C, CV25-003C, and CV25-005C were RC precollared. Drill hole CV25-001C was drilled from surface to completion with a coring rig. RC hole, CV25-006 was abandoned before reaching bedrock due to overshooting its target in overburden; the location had not previously been drilled. RC hole CV25-004 was terminated before reaching bedrock and left open for future continuation with a coring rig. Core holes CV25-001C, CV25-002C, CV25-003C, and CV25-005C were drilled by Timberline Drilling Inc., and hole CV25-007C was drilled by Major Drilling. Timberline Drilling of Elko Nevada used a truck-mounted Atlas Copco CS4002, and Major Drilling of Salt Lake City used a track-mounted LF-90 rig. Core holes drilled in 2025 were drilled using PQ3 (83 mm diameter core), HQ3 (61 mm diameter core) and NQ3 (45 mm core diameter) tools, as was appropriate for ground conditions.

10.3 Drill-Hole Collar Surveys

Roxmore staff capture an initial collar location using either a differential or handheld GPS in the field once the drill has vacated the drill pad. The staff ensure that the collar location is well marked. A third-party company (Humboldt Land Surveying) subsequently surveys all collars using a Trimble R8 with a Model 3 Receiver; these coordinates are used in the final database.

10.4 Down-Hole Surveys

International Directional Services (IDS) of Elko, Nevada was contracted to survey holes CV25- 001C, CV25-002C, CV25-003C, and CV25-005C. IDS used a truck-mounted north-seeking gyro (NSG) instrument and surveyed the pre-collar holes prior to core drilling, as well as the core holes at final depth. The pre-collared holes were surveyed at the top and bottom to ensure the holes had not deviated from the planned dip and azimuth. Readings for the core holes were taken at the top and bottom of each hole, as well as at 15.2 m intervals along the length of each hole.

Various core drilling companies used a north-seeking OMNIx42 instrument on hole CV25-004, CV25-006 and CV25-007C. The tool was set to capture a reading every 30.48 m (100 ft) both in and out of the hole.

11.0 Sample Preparation, Analyses, and Security

This section describes the procedures for sample preparation and analysis, the implementation of quality assurance and quality control (QA/QC) protocols, and the measures adopted to maintain the security and chain of custody of samples for the Converse Project. These procedures were initially established during early campaigns and have been consistently applied and refined through successive phases operated by UUI/Romarco (1995–1999), MVG (2003–2007), IMC (2011–2012), and CRL (2017), culminating in the current Roxmore program initiated in 2025. The methods summarized herein align with industry best practices and the protocols ensuring continuity across all campaigns. ALS Global has served as the primary analytical laboratory in recent programs, with historical support from other accredited laboratories, including American Assay Laboratories (AAL), Actlabs, and Cone Geochemical. All analytical laboratories used at the Project are independent of the Company.

11.1 Sample Preparation and Analysis

11.1.1 Reverse Circulation (RC) Sampling

UUI/Romarco (1995-1999)

RC drilling during the UUI/Romarco campaigns was conducted using wet drilling techniques with a cyclone and rotary wet splitter. Samples were collected at 1.5 m (5 ft) intervals, with single splits taken in alluvial material and duplicate splits (designated A and B) in bedrock. The A split was submitted for assay, while the B split was retained on site for potential metallurgical or check testing. Typical sample weights ranged from approximately 3.6 kg to 4.5 kg (8 lb to 10 lb) in early programs, increasing to 4.5 kg to 7.0 kg (10 lb to 15 lb) in later years. Assays were generally performed on all bedrock intervals and the lower portion of the alluvium. Mud rotary holes drilled in 1995 were sampled by allowing cuttings to settle in a trough before bagging; these intervals were also standardized at 1.5 m (5 ft).

MVG (2003-2007)

MVG implemented systematic RC sampling at 1.5 m (5 ft) intervals using a cyclone and wet splitter mounted on the drill rig. The splitter was calibrated to produce samples weighing between 5 kg and 10 kg (11 lb and 22 lb), which were placed in pre-numbered bags. Excess moisture was drained prior to dispatch to the laboratory. A representative portion of each interval was collected for geological logging and stored in chip trays at the company’s facility in Sparks, Nevada.

11.1.2 Core Sampling

UUI/Romarco (1995-1999)

Core recovered during these programs was transported to a secure warehouse for logging and sampling. Intervals were selected based on lithological and structural criteria, typically ranging from 0.15 m (0.5 ft) to 2.3 m (7.5 ft) with an average interval length of approximately 1.2 m (3.9 ft). Core was split longitudinally using a hydraulic splitter, with one half submitted for analysis and the other archived. Chain-of-custody was maintained by laboratory personnel collecting samples directly from the storage facility.

MVG (2003-2007)

MVG processed core at its Sparks facility, where it was washed, photographed, and logged. Sampling was generally performed in 1.5 m (5 ft) intervals, adjusted for mineralization or geological boundaries. Competent core was cut using a diamond saw, with half-core (HQ/NQ) or quarter-core (PQ) submitted for assay. Broken intervals were sampled by combining representative fragments and fines.

IMC (2011-2012)

Core was transported to the Sparks, NV facility for processing. After washing and photographing, intervals were typically sampled at 1.5m (5 ft), with adjustments for geological features. Competent core was split using a diamond saw—½ for HQ/NQ sizes and ¼ for PQ size—with the remainder archived. Broken intervals were sampled by combining representative fragments and fines; overburden material was sampled to obtain a proportional split. Samples were bagged with bar-coded labels, marked by hole ID and interval, and delivered by company personnel to ALS Chemex in Reno for analysis.

CRL (2017)

Core was moved daily from the drill site to a secure logging facility in Lovelock, Nevada. After geotechnical and geological logging, sampling intervals (typically 1.5 m (5 ft), adjusted for geology) were marked and tagged. Core was photographed, palletized, wrapped, and stored in a controlled yard. An independent carrier delivered the pallets with an inventory list to ALS Global, where samples were cut, bagged, and analyzed. Bulk density specimens were collected before cutting for later determination.

Roxmore (2025)

Core was delivered daily to the core shack, arranged on benches in descending order, and photographed prior to processing. Sampling intervals ranged from 0.3 m (1 ft) to 1.5 m (5 ft), adjusted to respect lithological and structural boundaries.

Core was cut lengthwise; the half containing the orientation line remained in the box, while the other half was sent for analysis to ALS Global. Broken intervals were sampled representatively, including all size fractions and fines. Samples were packaged in rice bags (10 per bag), labeled with identification details, and palletized for shipment. Additional measurements included magnetic susceptibility readings every meter and specific gravity determination using the paraffin-coated specimen method.

11.1.3 Density Determinations

The current database comprises density measurements from 634 samples collected from 1995 to the present. Of these 151 are from cuttings and 483 from core and are summarized in Table 11-1.

Table 11-1: Summary of Density Samples at the Converse Project

Company	Year	Alluvium		Rock		Total	
		Cuttings	Core	Cuttings	Core	Cuttings	Core
UUI	1995-1999	0	0	27	0	27	0
MVG	2003-2007	0	118[1]	124	1	124	119
IMC[2]	2012	0	0	0	32	0	32
CRL	2017	0	1	0	77	0	78
Roxmore	2025	0	0	0	254	0	254
					Total	**151**	**483**

Notes:
1. 118 core samples were submitted by CRL
2. Density samples collected by Big Rock Exploration (subcontractors to Roxmore) on historical core, in early 2025.

UUI (1995–1999)

Samples analyzed by KCA (Reno) using the wax coating method.

MVG (2003–2007)

Core samples analyzed by both KCA using the wax coating method and by McClelland Laboratory Inc. (MLI) using the acrylic coating method.

Determination on rock or alluvium samples used water immersion method at ALS Global (ALS code OA-GRA08a) and applied to core samples.

CRL (2017)

Whole-core samples collected approximately every 30 ft; analyzed by ALS Global using paraffin wax-coated and water immersion methods (ALS code OA-GRA09a).

Roxmore (2025)

Roxmore collected specific gravity measurements from drill core following AASHTO T 275 (2008) using the paraffin-coated specimen method. Each specimen was cut to approximately 15.2 cm (6 in.), dried to constant mass, and cooled to 25°C ± 5°C before weighing. Porous samples were sealed with paraffin to prevent water ingress. Submerged weights were recorded in a temperature-controlled water bath (25°C ± 1°C) using a weigh-below apparatus. Calculations incorporated paraffin density corrections to ensure accurate bulk density values for resource modelling.

11.1.4 Analysis and Test Laboratories

A summary of all laboratories contracted, as well as analyses and protocols used, is outlined in Table 11-2. A more detailed description follows:

Chevron and Cyprus (Pre-1995)

Drilling samples from the Chevron and Cyprus campaigns were submitted to Cone Geochemical for preparation and analysis. Specific preparation procedures were not documented; however, gold was analyzed using a one assay-ton aliquot (approximately 29.2 g) subjected to fire assay digestion with an atomic absorption spectroscopy (AAS) finish. Silver and other elements were determined using wet chemical techniques available at the time.

UUI/Romarco (1995–1999)

Samples collected during the UUI and Romarco programs were prepared by accredited laboratories following industry standard protocols. Core and RC samples were dried, crushed to achieve more than 50% passing 10 mesh, and pulverized until at least 90% passed 200 mesh. Gold was analyzed by fire assay with AAS finish, and gravimetric methods were employed for over-limit determinations. Multi-element analyses were performed using inductively coupled plasma atomic emission spectroscopy (ICP-AES) following aqua regia digestion. Laboratories involved included ALS Global, AAL in Sparks, Nevada, and Bondar-Clegg (acquired in 2001 by ALS Chemex, the minerals geochemistry division of ALS Global).

MVG (2003–2007)

MVG submitted samples to ALS Chemex and AAL for preparation and analysis. Procedures included drying, crushing, and pulverizing to produce fine pulps suitable for assay. Gold was determined by fire assay with AAS finish, with gravimetric checks for high-grade intervals. Multi-element analysis was conducted using four-acid digestion followed by ICP-AES for a suite of 34 elements.

IMC (2011–2012)

IMC employed ALS Global for all analytical work. Sample preparation consisted of drying, crushing, and pulverizing to achieve at least 90% passing 200 mesh. Gold was analyzed by fire assay with AAS finish, and gravimetric methods were used for over-limit samples. Multi-element determinations were completed using four-acid digestion and ICP-AES instrumentation.

CRL (2017)

CRL submitted samples to ALS Global in Reno, Nevada. Preparation protocols included drying, fine crushing to 70% passing 2 mm, and pulverizing to 85% passing 75 µm. Gold was analyzed using fire assay with AAS finish, and gravimetric checks were performed for high-grade material. Multi-element analysis was completed using four-acid digestion and ICP-AES.

Roxmore (2025)

Samples from the Roxmore program were processed by ALS Global facilities in Elko, Reno, and Vancouver. Preparation included drying at a maximum temperature of 60°C, fine crushing to 70% passing 2 mm, rotary splitting, and pulverizing up to 250 g to achieve 85% passing 75 µm. Gold was analyzed using method Au-AA23, which involves a 30 g fire assay charge with AAS finish. Multi-element analysis was performed using method ME-ICP61, a four-acid digestion followed by ICP-AES for a 34-element suite.

Table 11-2: Summary of Analytical Methods and Laboratory Protocols to 2025

Phase	Gold Method	Multi-Element Method	Sample Preparation	Laboratory	Accreditations
Chevron/Cyprus (Pre-1995)	fire assay-AAS (1 assay-ton aliquot ~29.2 g)	Wet chemical / ICP (limited)	Unknown	Cone Geochemical (Colorado, USA) Bondar-Clegg (Nevada, USA)	Historical laboratories, unknown accreditation
UUI/Romarco (1995–1999)	fire assay-AAS; Gravimetric for over-limit	ICP-AES (aqua regia)	Dry, crush >50% -10 mesh; pulverize >90% -200 mesh	ALS Global (Nevada, USA), AAL (Nevada, USA), Bondar-Clegg (Nevada, USA),	ALS Global: ISO/IEC 17025; AAL: ISO accredited; Bondar-Clegg: historical
MVG (2003–2007)	fire assay-AAS; Gravimetric for high-grade	ICP-AES (four-acid, 34 elements)	Dry, crush <70% -2 mm; pulverize +85% -75 µm	ALS Chemex (Reno, Nevada, USA) AAL (Sparks, Nevada, USA)	ALS Chemex: ISO/IEC 17025; AAL: ISO accredited
IMC (2011–2012)	fire assay-AAS; Gravimetric for over-limit	ICP-AES (four-acid, 51 elements)	Dry, crush <70% -2 mm; pulverize +85% -75 µm	ALS Global (Reno, Nevada, USA)	ALS Global: ISO/IEC 17025
CRL (2017)	fire assay-AAS; Gravimetric for over-limit	ICP-AES (four-acid)	Dry, crush 70% <2 mm; pulverize 85% <75 µm	ALS Global (Reno, Nevada, USA)	ALS Global: ISO/IEC 17025
Roxmore (2025)	Au-AA23 (30 g fire assay with AAS finish)	ME-ICP61 (four-acid, 34 elements)	Dry max 60°C; crush 70% <2 mm; split; pulverize 250 g to 85% <75 µm	ALS Global (Reno, USA Vancouver, Canada)	ALS Global: ISO/IEC 17025

11.2 Sample Security

For the MVG drill program, samples were stored at the drill site until collected by AAL, with pickups scheduled to align with the drilling contractor’s rotation. During the IMC and CRL drill programs, samples remained at the drill site under the drillers’ custody until transported to the respective company’s storage facilities in Reno and Lovelock. For the 2025 Roxmore campaign, core was cut in half lengthwise, with one half retained and the other sent for analysis. Broken core was sampled representatively across all size fractions. Samples were packaged in labelled rice bags, sealed, and stacked on pallets for shipment, with records maintained to ensure traceability.

11.3 Quality Assurance and Quality Control

QA/QC procedures were implemented to monitor potential contamination, analytical accuracy and precision, and possible short- or long-term laboratory bias. The program included the insertion of blank materials and certified reference materials, the collection of field duplicates, and the submission of check assays to an umpire laboratory. A summary of the QA/QC program from 1998 to 2025 is shown in Table 11-3.

11.3.1 QA/QC Protocols

Chevron and Cyprus (Pre-1995)

Documentation of QA/QC practices for these early programs is limited. There is no evidence of systematic insertion of blanks, standards, or duplicates, and no check assay data has been located.

UUI/Romarco (1995–1999)

QA/QC procedures were introduced during these campaigns. Certified reference materials (CRMs) and blanks were inserted at regular intervals, typically at a rate of one control sample per 20 to 30 primary samples (approximately 3% to 5%). Historical records indicate that the 1997–1998 programs included 186 check assays, and the 1999 program included 12 check assays, performed by independent laboratories to verify primary results.

MVG (2003–2007)

QA/QC protocols followed industry standards, with insertion of blanks and CRMs at approximately 1 in 20 samples (approximately 5%). Field duplicates were collected from RC splits and core intervals where possible. Analytical performance was monitored through control charts, and failures were re-assayed as required.

IMC (2011–2012)

QA/QC measures included systematic insertion of blanks and CRMs at a frequency of approximately 10%, with occasional field duplicates. Independent check assays were performed on selected batches to confirm accuracy of ALS Global results.

CRL (2017)

QA/QC consisted of blanks and CRMs inserted at approximately 1 in 10 samples (approximately 10%). No field duplicates were collected because core was retained for metallurgical testing. ALS Global provided internal QA/QC checks, and external verification was limited to pulp repeats.

Roxmore (2025)

The current program (CVWP012) applies a balanced QA/QC insertion rate of 3.33% blanks, 3.33% CRMs, and 3.33% half-core duplicates, equivalent to one control sample every 30 primary samples. For a batch of 100 samples, this represents approximately three blanks, three CRMs, and three duplicates. Earlier 2025 drilling (CVWP002) did not include duplicates but maintained blanks and CRMs at similar rates.

Table 11-3: Summary of Control Insertion by Year (Historical–2025)

Phase	Primary Samples	Controls			Submission Rate		
		Blank	CRM	Duplicates	Grand Total	QC Total	QC Rate
Historical (1989-1999)	15,976	0	0	202	16,178	202	1.2%
MVG (2003-2008)	16,403	249	379	127	17,158	755	4.4%
IMC (2011-2012)	8,266	157	612	1	9,036	770	8.5%
CRL (2017)	919	34	61		1,014	95	9.4%
Roxmore (2025)	2,995	148	166		3,309	314	9.5%
Grand Total	44,559	588	1,218	330	46,695	2,136	

Source: SLR 2025.

11.3.1.1 Blanks

Blank samples were routinely inserted at a frequency of approximately one in every 20 to 30 samples across all phases to monitor potential contamination during sample preparation and analysis. For assessment purposes, the QP defined a failure threshold as 10 times the limit of detection (LoD). For gold, with a typical fire assay LoD of 0.005 ppm, any blank returning a value ≥0.05 ppm Au was considered a failure. For silver, with a LoD of 0.5 ppm, the failure threshold was set at 5 ppm Ag. For copper, the LoD used in this assessment was 1 ppm, resulting in a failure threshold of 10 ppm Cu. Figure 11-1 illustrates the performance of 128 blank material inserted in the 2025 drill program, all of which returned results below their respective thresholds, indicating no failures.

Between 2003 and 2025, a total of 587 blank samples for gold exhibited low and acceptable contamination rates, as detailed in Table 11-4. Coarse blank material for copper inserted during the 2025 phase (n = 148) also showed acceptable contamination levels, although 10 failures were recorded (6.8%), predominantly with values below 25 ppm Cu (Figure 11-2). These results suggest minor carry-over for copper, which is not considered to materially affect mineralized intervals. For silver, no failures were recorded across all phases, indicating no evidence of contamination events.

Table 11-4: Summary of Blank Failure Rates by Phase in ALS Global (2003–2025)

Phase	Element	Type	Samples	Failures	Failures Rate (%)	Limit (ppm)	Min (ppm)	Max (ppm)
MVG (2003-2008)	Au	Coarse Blank	248	4	1.6	0.05	0	0.64
		Fine Blank	1	0	0.0	0.05	0	0
	Ag	Coarse Blank	1	0	0.0	5	1	1
	Cu	Coarse Blank	1	0	0.0	10	10	10
IMC (2011-2012)	Au	Fine Blank	157	0	0.0	0.05	0	0.02
CRL (2017)	Au	Coarse Blank	34	1	2.9	0.05	0	0.06
	Ag	Coarse Blank	40	0	0.0	5	0	0.2
Roxmore (2025)	Au	Coarse Blank	148	0	0.0	0.05	0	0.01
	Ag	Coarse Blank	148	0	0.0	5	0	0.5
	Cu	Coarse Blank	148	10	6.8	10	0	50

Source: SLR 2025.

Figure 11-1: Gold Blank Samples: 2025 Coarse Blanks Analyzed by ALS Global

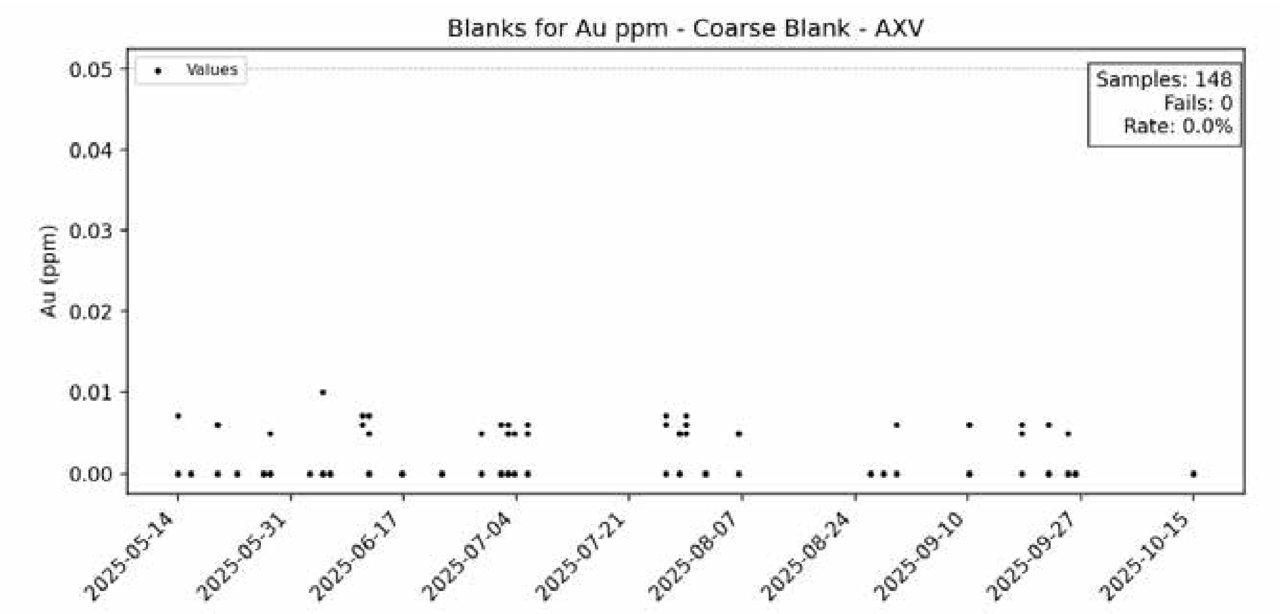
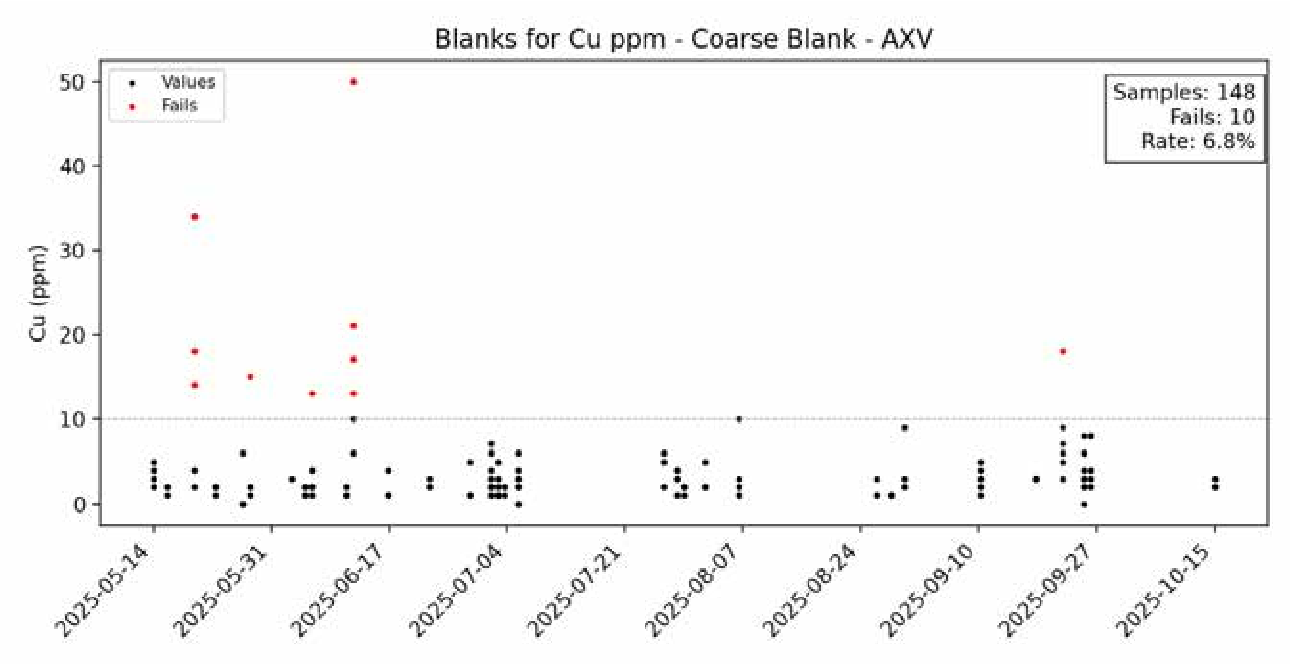


Figure 11-2: Copper Blank Samples: 2025 Coarse Blanks Analyzed by ALS Global



11.3.1.2 Certified Reference Materials

CRMs from OREAS (Ore Research & Exploration, Victoria, Australia), Rocklabs (Scott Technology, Auckland, New Zealand), and CDN Resource Laboratories (British Columbia, Canada) were used to evaluate analytical accuracy and identify potential bias trends. The CRM suite was selected to bracket anticipated gold grade ranges. Acceptance limits were defined as the certified expected value ± three standard deviations (EV ± 3SD).

A total of 1,218 CRMs were inserted into the sample stream at a frequency of approximately 5% to 6% between 2012 and 2025, and at a lower rate of 2% noted during MVG phases (2003-2008). This included 29 CRM types certified for gold, six CRM types for silver, and eight for copper. All 1,218 CRMs were analyzed by Bureau Veritas (BV) Laboratories.

Gold CRM results demonstrate overall strong accuracy as detailed in Table 11-5, with most samples falling within the threshold of EV ± 3SD and global biases typically below ±5%. Two isolated cases merit discussion: during the MVG phases, standard S104004X returned a bias of -11.2% at 0.03 ppm Au, and during the IMC phases (2011–2012), CDN-GS-1F showed a bias of 10.1% at 1.16 ppm Au, influenced by a single outlier at 3.2 ppm Au. Removing this outlier reduces the bias to approximately 6%. These deviations are considered insignificant because they occur at very low grades, where relative bias is amplified.

Silver CRMs analyzed from holes drilled in 2025 performed within acceptable limits overall. Higher biases of silver and occasional exceedances beyond EV ±3SD are restricted to CRMs with nominal silver values near the analytical detection limit (e.g., OREAS 263b and OREAS 236b at approximately 0.5 ppm Ag). This behavior is expected due to reduced precision at sub-ppm concentrations and does not indicate systematic bias.

Copper CRMs demonstrated excellent performance, with most samples within EV ±2SD and showing appropriate scatter across all phases, indicating natural variability without excessive spread or over-tight clustering. No failures were recorded.

Table 11-5: Summary of CRM Samples Analyzed by BV: 2020 to 2025

Phase	CRM	Period in Use	No. of Samples	Mean (ppm)	EV (ppm)	SD (ppm)	No. of Outliers	Bias (%)	Failure Rate (%)
MVG	S105002X	2007	40	0.42	0.44	0.02	3	-4.8	7.5
	S105004X	2007	34	3.85	3.75	0.2	1	2.7	2.9
	S104004X	2007	37	0.03	0.03	0.01	3	-11.2	8.1
	S105005X	2007	37	2.46	2.42	0.08	3	2	8.1
	S105001X	2007	41	1.85	1.84	0.08	4	0.6	9.8
	S105006X	2007	36	4.51	4.52	0.1	7	-0.2	19.4
	C404002X	2007	38	1.31	1.31	0.05	1	-0.5	2.6
	S104008X	2007	40	0.66	0.66	0.02	8	-0.4	20
	S105003X	2007	37	0.51	0.52	0.03	7	-3.8	18.9
	S104007X	2007	39	0.76	0.75	0.02	7	1.5	17.9
IMC	SE44	2011-2012	34	0.61	0.61	0.02	0	0.2	0
	CDN-GS-1F	2011-2012	42	1.28	1.16	0.06	6	10.1	14.3
	CDN-GS-2G	2011	30	2.38	2.26	0.1	3	5.3	10
	CDN-GS-7B	2011-2012	127	6.45	6.42	0.23	3	0.5	2.4
	CDN-GS-2J	2011-2012	123	2.47	2.36	0.1	4	4.8	3.2
	CDN-GS-1H	2011-2012	122	0.99	0.97	0.05	2	2.1	1.6
	SE58	2011-2012	122	0.59	0.61	0.02	4	-2.4	3.3
	CDN-GS-2F	2011	12	2.24	2.16	0.12	1	3.6	8.3
CRL	CDN-GS-2L	2017	18	2.38	2.34	0.12	0	1.5	0
	OXG99	2017	22	0.92	0.93	0.02	2	-1.5	9.1
	SE68	2017	21	0.59	0.6	0.01	0	-0.9	0
Roxmore	OREAS 263b	2025	20	0.22	0.21	0.01	0	2	0
	OREAS 251c	2025	21	0.5	0.51	0.01	0	-0.6	0
	OREAS 231b	2025	34	0.56	0.56	0.02	0	0	0
	OREAS 152c	2025	34	0.14	0.13	0.01	0	5.3	0
	OREAS 236b	2025	23	1.93	1.92	0.04	0	0.3	0
	OREAS 253b	2025	4	1.25	1.24	0.04	0	0.5	0
	OREAS 701	2025	10	1.07	1.11	0.05	0	-3.7	0
	OREAS 502d	2025	20	0.5	0.5	0.01	0	0	0

Source: SLR 2025.

Notes:

1. Element: Au
2. EV: Expected Value
3. SD: Standard Deviation

Figure 11-3 illustrates Z-score distributions for CRMs by phase and element. Gold results are largely within EV ±3SD with no systematic drift; MVG phase failures are linked to very low-grade CRMs where relative bias is amplified but not material. Silver shows greater scatter and occasional failures, mainly for CRMs near detection limits, reflecting expected precision limitations rather than analytical bias. Copper demonstrates excellent accuracy, with most samples within EV ±2SD and appropriate scatter.

The QP is of the opinion that CRM performance for gold demonstrates good accuracy and supports the reliability of the assay data used in the 2025 Mineral Resource estimate.

Figure 11-3: Z-Score Distribution for CRMs by Phase and Element

Source: SLR 2025.

11.3.1.3 Duplicates

Duplicate samples were inserted during the historical and MVG drill phases; however, there is no documentation regarding the specific core fractions collected in these programs. Overall, duplicates were inserted during core logging at a rate of approximately 3% (one in every 30 samples).

For the Historical phase, a total of 23 field duplicate (FD) pairs for gold were analyzed, showing a very strong correlation ($R^2 = 0.991$), and 167 coarse duplicate (CD) pairs for gold exhibited moderate correlation ($R^2 = 0.455$). These results, presented in Table 11-6, indicate that, despite the lack of clarity regarding the exact sampling fractions, historical duplicates provided reasonable reproducibility at the field level, whereas coarse duplicates introduced additional variability, which may also be influenced by the inherent nature of gold mineralization and its local-scale variability.

In contrast, the MVG phase included 124 FD pairs for gold, which showed an almost null correlation (R^2 = -0.028), indicating poor reproducibility and suggesting that the duplicate protocol applied during this phase did not achieve adequate representativity.

Roxmore did not collect duplicate samples during the first phase of its 2025 drilling program. During the most recent site visit, new protocols for half-core duplicate sampling were observed; however, the number of samples collected at the time of reporting was insufficient to support meaningful analysis.

The QP recommends continuing the systematic half-core duplicate sampling to improve representativity and comparability between original and duplicate samples. Additionally, implementing coarse reject and pulp duplicate programs will enable more robust monitoring of precision during sample preparation and analysis.

If Roxmore elects to include silver in the next MRE, all analysis should be done using a four-acid digestion. In addition, applying a lower detection limit analysis for silver (e.g., 0.1 ppm) would likely reduce the number of analytical failures at the low-grade end.

Table 11-6: Summary of Duplicate Sample Analysis for Gold – Historical and MVG Drill Phases

Lab	DupType	Element	Pairs	Fails	Corr	Mean (ppm)	Min (ppm)	Max (ppm)	Mean (ppm)
MVG	FD	Au	124	63	-0.028	-0.03	0	2.15	0.06
Historical	FD	Au	23	2	0.991	0	0	0.76	0.1
Historical	CD	Au	167	47	0.455	0.06	0.02	10.8	1.06

Source: SLR 2025.
Note: Pairs number fewer than three were not accounted for in this evaluation

11.3.1.4 Umpire Check Assays

A total of 235 check assays were completed during the UUI/Romarco drilling phases. Of these, only 18 pulps had copper assays (analyzed in 1996 by ActLabs and submitted to Cone Geochemistry as the umpire laboratory). These copper results show good correlation between laboratories. For the remaining samples, only mercury assays were available, and no gold or silver data could be incorporated into this review.

The QP recommends implementing a periodic submission of pulp check assays at a rate of 4% to 5% to an accredited third-party laboratory. This protocol is intended to validate the accuracy and precision of the primary laboratory and should be implemented together with the use of appropriate control samples and consistent analytical methods.

11.4 QP Opinion

The QP has reviewed sample preparation, analytical methods, sample security, chain of custody, and QA/QC results for the 2003–2025 period.

Procedures employed by Roxmore are consistent with industry best practices for exploration and mineral resource estimation.

Blank sample data do not indicate material contamination during preparation or analysis.

Certified reference materials demonstrate very good analytical accuracy for the three elements evaluated: gold, silver, and copper. Minor biases or failure incidents seem to be associated with low gold and silver concentrations near their limit of detection.

Duplicate results from historical and MVG phases show inconsistent reproducibility, likely influenced by fraction representativity and the nugget effect. The QP recommends adopting half-core duplicates for better representativity and implementing pulp duplicates to improve precision and reduce nugget effect impacts.

No current umpire check assay data were available for this review. Routinely, pulp-sample checks are recommended to be sent to a third-party laboratory along with blanks and CRMs.

The QP is of the opinion that:

- Roxmore's sample preparation procedures are appropriate for the deposit type and mineralization style.
- The analytical methods used, including fire assay and metallic fire screen assays, are suitable for determining gold grades at the Project.
- The QA/QC program, which includes CRMs and blanks, is well structured, meets industry standards, and provides confidence in the assay data.
- Sample security measures and chain of custody protocols are sufficient to ensure the integrity of the data.

Based on the foregoing, the QP considers the assays to be of sufficient quality to support their use in the 2026 Mineral Resource estimate for the Converse Project.

12.0 Data Verification

12.1 Roxmore Data Verification

Roxmore’s technical staff independently verify the accuracy, completeness and reliability of the data collected. This verification process includes the evaluation of collar locations, downhole surveys data, geological and structural information and assay results.

12.1.1 Collars

Drill hole collars are initially positioned using a handheld GPS. Roxmore staff capture an initial collar location using a field GPS once the drill has vacated the drill pad, and ensure the collar location is well marked. A third-party company (Humboldt Land Surveying) subsequently surveys all collars using a Trimble R8 with a Model 3 Receiver; these are the coordinates that are used in the final database.

12.1.2 Surveys

Roxmore engaged International Directional Services (IDS) of Elko Nevada or had the onsite drilling company complete downhole surveys on drill holes using north-seeking gyro instruments. Holes were typically surveyed at intervals of 15.2 m, occasionally 30.48 m, both in and out of the hole. Additional downhole survey checks were conducted on precollared hole segments.

12.1.3 Geological Data

Geological logging was conducted at Roxmore’s core logging facility in Lovelock NV, by experienced geologists and technicians. Core and RC logging included detailed lithological and alteration descriptions including prograde and retrograde skarn distinctions, mineralization styles, vein descriptions and redox classifications. Roxmore is also collecting orientated structure measurements from their core drilling programs. Logging is done in an Access database with curated dropdown picklists.

As of the effective date Roxmore has relogged approximately 30% or the historical core holes.

Once the data is loaded into the SQL database, it is uploaded into Leapfrog, where built-in validation routines, such as checking for overlapping intervals, intervals exceeding drill hole depth and mandatory fields containing null values, are automatically flagged and corrected by the database manager.

12.1.4 Assays

Assay results are received via email from the laboratory. All result certificates are imported in SQL Access Database by the database manager.

12.2 SLR Data Verification

12.2.1 Site Visit

April Barrios, P.Geo., and Balaji Subrahmanyan, SME(RM), of SLR conducted a site visit to the Project and associated facilities on November 4, 2025. During this site visit, they inspected several 2025 drill sites and verified collars relevant to the MRE. Ms. Barrios and Mr. Subrahmanyan also reviewed the general property layout, observed active drilling operations, and assessed the logging environment and procedures used for geological data collection and sampling. Both historical and recently drilled drill core were examined, and mineralization was visually compared with recorded lithology and redox classifications as well as corresponding analytical gold assay results. Ms. Barrios and Mr. Subrahmanyan were provided full access to all facilities and personnel during the visit and were accompanied by Blake McLaughlin (Executive Vice President) and Zsolt Molnar (Exploration Manager) of Roxmore.

12.2.2 Collars

The QPs visited a number of Roxmore's 2025 drill pads. The coordinates of several drill pads were collected with a handheld GPS while on site and compared to the coordinates in the database Figure 12-1. Historical pads have been reclaimed and very few hole markers remain.

Figure 12-1: Drill Site CV25-002C

Source: SLR 2025.

12.2.3 Surveys

Downhole surveys were visually reviewed in Leapfrog Geo. Approximately 30% of the drill holes informing the MRE (77 of 262) lack downhole surveys. These unsurveyed holes were all vertically drilled in the mid to late 1990s and did not exceed 250 m depth. More recent drill holes of comparable depths (250 m to 300 m) that were downhole surveyed show negligible deviation from vertical. Accordingly, the QP considers the risk of significant deviation in the unsurveyed, relatively short, vertical holes to be low and has retained them in the MRE database.

12.2.4 Geological and Redox Data

Roxmore staff have been systematically re-logging the lithology, alteration, structural features, and redox characteristics in historical drill holes. The updated lithologic and redox logs were reviewed against their respective geological models and were found to exhibit strong correlation with the interpreted domains.

12.2.5 Assay Database

SLR reviewed the drill hole database for the Converse Project, which is maintained in GeoSequel. As of the cut-off date, the database contained 44,800 samples collected between 1989 and 2026. A total of 32,257 samples, representing approximately 72% of the database, were cross-checked against 598 original assay certificates. The remaining samples could not be verified due to missing certificates from historical campaigns.

No significant discrepancies were identified during the review, except for one interval that was not found in the available certificates. Minor differences in sample naming conventions were noted but do not materially impact data integrity. The verification covered 243 of the 358 drill holes completed to date.

The QP concludes that the drill hole database for the Converse Project is reliable, consistent, and meets industry best practices for data management and summarized in Table 12-1. This verification covered assays completed between the historical drill campaigns and 2025.

Table 12-1: Summary of Data Verification and Cross-Check Rates

Year	No. Samples	No. Samples Compared	Rate Comparison (%)	No. Discrepancies
Historical	99	5	5%	
1989	357	357	100%	
1991	217	217	100%	
1992	732	732	100%	
1994	288	288	100%	
1995	274	274	100%	
1996	719	719	100%	
1997	4,053	2,256	56%	
1998	7,731	1065	14%	
1999	1,522	120	8%	
2003	1,077	1,077	100%	
2004	2,100	2,077	99%	
2005	795	795	100%	
2007	9,989	7,756	78%	
2008	2,426	2,339	96%	
2011	2,890	2,890	100%	1
2012	5,376	5,375	100%	
2017	919	919	100%	0
2025	2,995	2,995	100%	0
Total	44,559	32,256	72%	1

SLR conducted a number of digital and visual queries on the resource database and found no material discrepancies that would impact the validity of the Mineral Resource estimate. In the opinion of the QP, the verification procedures confirm the reliability of the assay database, ensuring its suitability for use in the Mineral Resource estimate.

13.0 Mineral Processing and Metallurgical Testing

Metallurgical test work review and process design and development of capital and operating costs for the process plant were completed by KCA in Reno, Nevada. The following KCA documents were developed for the PEA.

- Summary Review of Metallurgical Test Data Converse Mining Project
- Converse Process PEA CAPEX OPEX_Option1 Rev D
- Converse Design Criteria Option 1_Heap Case Rev
- Converse Flowsheet Option 1 (9828-0113-11-100-C)
- Converse Design Criteria Option 2_Pulp Agglomeration Rev A
- Converse Flowsheet Option 2 (9828-0113-11-200-C)
- Converse Process PEA CAPEX OPEX_Option 2 Rev A

KCA reviewed and verified metallurgical test sample locations and test procedures to ensure the data and conclusions derived from the data are in line with industry standards. It is the QP's opinion that the metallurgical test data available are sufficient to support the selected processes and conclusions regarding recoveries and costs for the PEA level.

13.1 Introduction and Summary

The Converse deposit has undergone an extensive series of metallurgical test programs between 1997 and 2018, designed to characterize gold recovery potential under various processing methods. Several laboratories performed test work, notably MLI, Newmont Metallurgical Services (NMS), and KCA. Collectively, these studies evaluated the performance of cyanidation, heap leaching, comminution, gravity concentration, and flotation. All test work was conducted on samples that were considered representative at the time.

The Converse deposit has historically been divided into North and South Redline. The deposits are characterized as a skarn-type gold system with oxide, transition and sulphide redox zones. The sulphide content is generally low, less than 0.5% in most samples, though some samples reach as high as 4.8%. The dominant sulphide minerals are pyrrhotite and chalcopyrite. Test work has included the following, with different parameters regarding particle size distribution, leach time, and leach conditions:

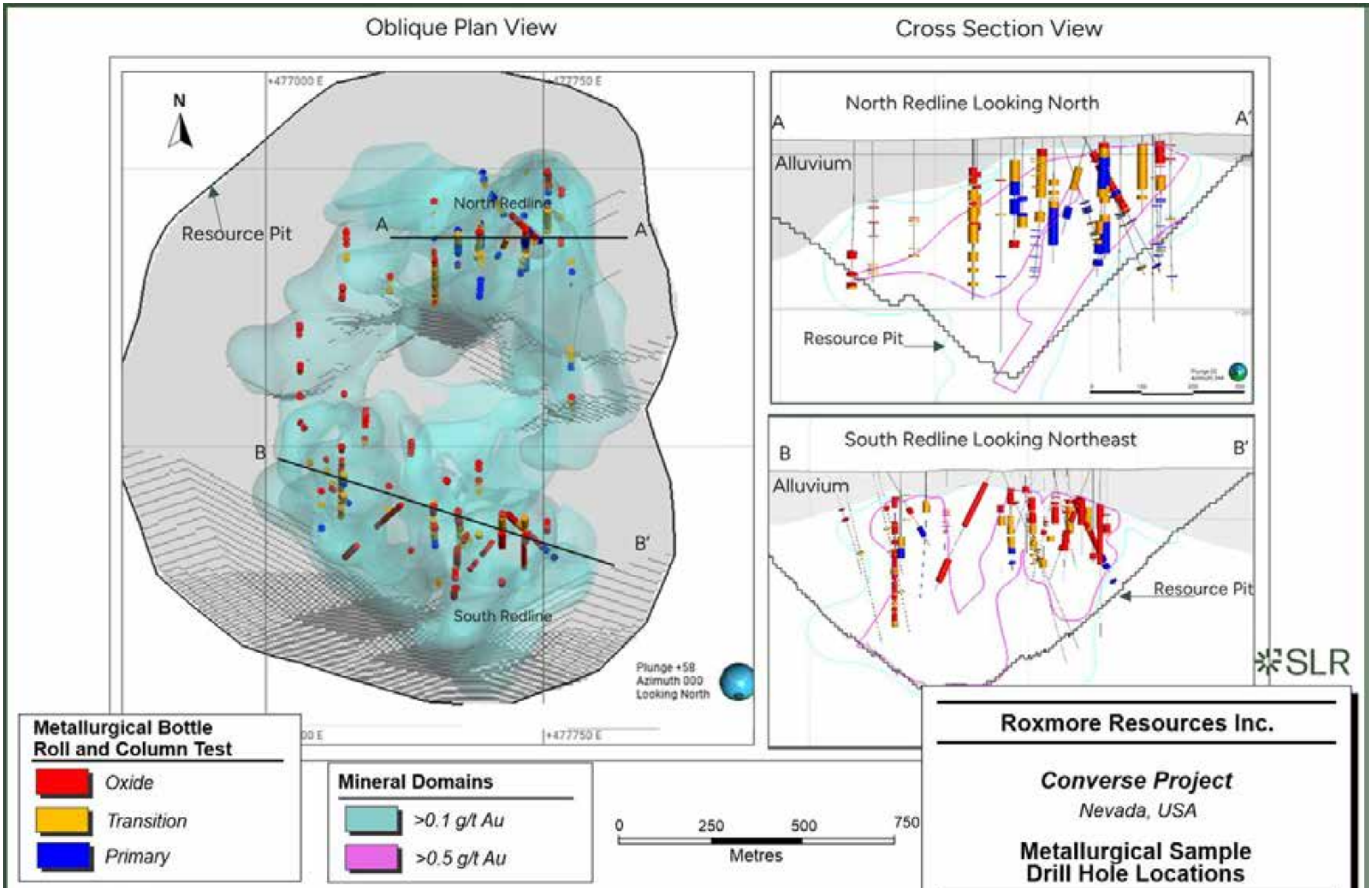
- Bottle roll tests that evaluated the amenability of the materials to cyanidation.
- Column leach tests that evaluated the amenability of the materials to conventional heap leaching.
- Permeability testing that evaluated the amenability of the material to compressive loads that simulate heap stacking.
- Comminution testing that evaluated the amenability of the material to be optimally crushed or milled, including a high pressure grinding roll (HPGR) option.
- Gravity concentration testing that evaluated the amenability of the material to concentrate the gold-containing particles.
- Flotation testing to evaluate the amenability of the material to flotation treatment.

A summary of all available test work programs is presented in Table 13-1. The spatial distribution of the drill holes selected for metallurgical sampling across all programs is shown in Figure 13-1 and demonstrates a broad, well distributed coverage of the deposit. Relevant portions of the test work are summarized chronologically in the following sections.

Table 13-1: Test Work Program Summary

Report Date	Provider	Client	Description	Bottle Roll Quantity	Column Test Quantity
August 21, 1997 April 3, 1998	MLI	Uranerz USA, Inc.	Bottle Rolls	19	-
February 2, 1999	NMS	Cameco (U.S.) Inc.	Bottle Rolls and Column Tests	4	2
February 2005, August 2005	KCA a	Metallic Ventures Gold Inc.	Rock Density, Gravity Concentration, Bottle Rolls, Column Tests, Comminution Tests	27	27
June 8, 2005	KCA b	Metallic Ventures Gold Inc.	Bottle Rolls on Assay Reject	750	-
June 27, 2009	MLI	Metallic Ventures Gold Inc.	Bottle Rolls, column tests, Gravity recoverable gold (GRG) test, Gravity cyanidation, sulphide Floatation	54	26
August 1, 2013	MLI	International Minerals Corp.	Bottle Rolls	100	-
August 2018	KCA	Converse Resources Ltd.	Comminution tests, bottle rolls, column tests, HPGR crushing/agglomeration, compaction permeability	42	20

Figure 13-1: Metallurgical Sample Drill Hole Locations



Source: SLR 2026.

13.2 MLI Metallurgical Services 1997-1998

Two initial sets of test work programs for the Project commenced in 1997 and 1998, conducted by MLI. The programs included cyanide bottle roll tests on 11 (1997) and 8 (1998) RC cuttings, composited from the Converse Project on material considered representative at the time. These bottle roll tests were conducted at a nominal crush size of minus 10 mesh (Tyler). The lowest gold recoveries of 50% were observed from two oxide composites, No. 1 and No. 2 (1997), associated with holes NK-24 and NK-31a, after being leached for 96 hours. The highest gold recovery of 84% was achieved with sulphide composite No. 11 (1997), sourced from hole NK-54b, and was leached for 96 hours. The cyanide consumptions for all 19 composites ranged from 0.14 kg/t to 2.6 kg/t, with an average of 0.63 kg/t. The high cyanide consumption of 2.6 kg/t observed with composite No. 7 (1998) may be attributed to the high copper content (2,780 ppm) observed. Lime addition ranged from 0.7 kg/t to 6.1 kg/t, with an average of 2.36 kg/t. The complete results summary is presented in Table 13-2.

The bottle-roll test results indicated that the Converse composites tested were amenable to direct cyanidation at the nominal 10 mesh (Tyler) feed size, with moderate gold recoveries. Gold recovery rates generally were fairly rapid. Additionally, cyanide consumption was low to moderate, except where high copper concentration is present. Lime requirements were low to moderate for all the composites, except for No. 5 (1997), which required more than 6 kg/t of material.

Table 13-2: Bottle Roll Result Summary (MLI 1998-1999)

Composite No.	Interval (ft)	Oxide	Grade	Hole ID	Calc. Head Grade Au (g/t)	Tail Grade Au (g/t)	Au Extracted (%)	Leach Time (days)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
1 (1997)	345-500	Oxide	Low grade	NK-24	0.312	0.156	50%	4	0.14	3.12
2 (1997)	265-405	Oxide	High grade	NK-31a	1.187	0.594	50%	4	0.15	3.18
3 (1997)	405-500	Oxide	Medium grade	NK-31b	0.812	0.344	58%	4	0.67	3.64
4 (1997)	150-210	Oxide	Medium grade	NK-33	0.687	0.25	64%	4	0.15	2.87
5 (1997)	260-370	Oxide	High grade	NK-38	1.344	0.656	51%	4	0.28	6.06
6 (1997)	330-490	Mixed	Medium grade	NK-45a	0.719	0.312	57%	4	0.22	2.52
7 (1997)	515-585	Mixed	Low grade	NK-45b	0.5	0.219	56%	4	0.37	1.84
8 (1997)	600-650	Sulphide	Low grade	NK-46	0.344	0.094	73%	4	0.14	2.66
9 (1997)	810-885	Sulphide	Medium grade	NK-49	0.656	0.281	57%	4	0.51	3.95
10 (1997)	580-680	Mixed	High grade	NK-54a	0.844	0.187	78%	4	0.81	2.55
11 (1997)	685-775	Sulphide	High grade	NK-54b	2.156	0.344	84%	4	2.45	2.31
1 (1998)	320-445	Oxide		NK-21	1.094	0.406	63%	7	0.23	1.35
2 (1998)	400-490	Oxide		NK-36	0.562	0.25	56%	7	0.3	1.7
3 (1998)	500-585	Mixed		NK-44	0.594	0.219	63%	7	0.47	0.95
4 (1998)	755-815	Sulphide		NK-47	1.344	0.344	74%	7	0.22	0.7
5 (1998)	570-665	Mixed		NK-62a	1	0.375	63%	7	0.76	1.3
6 (1998)	440-490	Sulphide		NK-62b	1.125	0.375	67%	7	0.82	1.1
7 (1998)	45-225	Mixed		NK-65a	1.531	0.437	71%	7	2.59	1.7
8 (1998)	225-295	Oxide		NK-65b	0.656	0.156	76%	7	0.68	1.25
Average:					**0.919**	**0.316**	**64%**		**0.633**	**2.356**

13.3 Newmont Metallurgical Services 1999

Metallurgical test programs for the Converse Project continued in 1999 at NMS. The program included bottle roll and column leach tests conducted on a single drill hole, NKC-43, compiled into two composites based on depth interval. The column leach test included 1.5 kg/t of cement for agglomeration and pH control. NMS's observations included the lack of preg-robbing tendencies. Additionally, assays indicated no measurable sulphide sulphur content, despite the expectation that the deeper composite would have contained sulphides. The copper content for Holes NKC-43a and NKC-43b was 200 ppm and 700 ppm, respectively. The higher copper content of NKC-43b may be the reason for the lower column recovery of 49% compared to 62% for NKC-43a. The summarized results for the bottle roll and column leach tests are presented in Table 11-4 and Table 11-5, respectively.

As part of the test program, NMS also performed bulk density calculations and gravity concentration using a Gemini Table. The calculated bulk densities were 1519 kg/m^3 and 1446 kg/m^3 for composites 1-C and 2-F, respectively. Gravity concentration was conducted on the composites, which were crushed to approximately 60% minus 200 mesh (Tyler), yielding concentrate weight percentages of 42.3% and 33.5% for composites 1-C and 2-F, respectively.

Results from the NMS test program suggest that heap leaching may be a viable processing method, with recoveries comparable to those from the coarse bottle roll tests. Recoveries appear to be sensitive to particle size, with the finer products of -200 mesh (Tyler) resulting in higher gold recoveries in the range of 87% to 88%, compared to the 61% to 64% recoveries for minus 10 mesh (Tyler). The gravity concentration test work suggests that the material would not benefit from gravity gold concentration.

Table 13-3: NMS Bottle Roll Result Summary (NMS 1999)

Comp.	Interval (ft)	Oxide	Hole ID	Crush Size	Calc. Head Grade Au (g/t)	Tail Grade Au (g/t)	Au Extracted (%)	Leach Time (days)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
1-C	317-576.4	Oxide	NKC-43a	-10 mesh	1.125	0.404	64%	4	0.13	1.09
1-C	317-576.4	Oxide	NKC-43a	80% -200M	1.469	0.185	87%	4	0.14	0.68
2-F	576.4-865.8	Oxide	NKC-43b	-10 mesh	0.812	0.311	61%	4	0.25	0.85
2-F	576.4-865.8	Oxide	NKC-43b	80% -200M	0.75	0.093	88%	4	0.19	0.44
Average					**1.039**	**0.248**	**75%**		**0.18**	**0.767**

Table 13-4: NMS Column Leach Test Result Summary (NMS 1999)

Comp.	Interval (ft)	Oxide	Hole ID	Crush Size	Calc. Head Grade Au (g/t)	Tail Grade Au (g/t)	Au Extracted (%)	Leach Time (days)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
1-C	317-576.4	Oxide	NKC-43a	3/8"	1.094	0.437	62%	88	0.63	1.50
2-F	576.4-865.8	Oxide	NKC-43b	3/8"	0.687	0.344	49%	88	0.76	1.79
Average					**0.891**	**0.391**	**55%**		**0.697**	**1.645**

13.4 KCA Test Program 2005a

In 2005, KCA initiated three series of metallurgical test work programs for the Converse Project. Two sets of the test work (2005a) with reports dated February 2005 and August 2005, respectively, were conducted on a series of nine representative composite samples from eight HQ core holes, formed to examine the variability in gold and silver extraction with respect to the following parameters:

- North and South;
- Head grade; and
- Degree of oxidation (oxide, mixed, and sulphide).

The first phase of the work included rock density, gravity, bottle roll leach tests at 200 mesh (Tyler), and column leach tests at minus 9.5 mm. The second Phase included additional bottle roll leach tests, column leach tests at minus 38 mm, and subsequently crushed and re-leached at minus 6.3 mm, and a grindability evaluation.

The remaining test work program (2005b) was conducted on various assay reject materials, and is discussed in Section 13.5.

The nine composites' identification, grade designation, oxidation type, hole ID, and deposit location are presented in Table 13-5.

Table 13-5: KCA 2005 Composite Classifications (KCA 2005a)

KCA Sample No.	Description	Hole ID	Area
32101	Mixed, Low grade	NK-134/135/136/140	North
32102	Mixed, Medium grade	NK-133/135/136/140	North
32103	Oxide, Medium grade	NK-136/140	North
32104	Sulphide, Medium grade	NK-133/134/135/140	North
32105	Sulphide, Medium grade	NK-133/135/136/140	North
32106	Mixed, Low grade	NK-138/139	South
32107	Mixed, Medium grade	NK-138/139	South
32108	Oxide, Low grade	NK-137/139	South
32109	Sulphide, Low grade	NK-137	South

Carbon and sulphur levels were measured to identify potential preg-robbing carbonaceous material and assess sulphide sulphur levels that inhibit gold recoveries. Composites of note include the highest value of sulphide sulphur (0.19%) observed in sample 32109 and the highest total carbon (1.2%) observed in sample 32102. The complete set of composite values is presented in Table 13-6.

Table 13-6: Composite Head Analysis – Carbon and Sulphur (KCA 2005a)

KCA Sample #	Description	Total Carbon (%)	Total Sulphur (%)	Sulphide Sulphur (%)	Sulphate Sulphur (%)
32101	Mixed, Low grade	0.4%	0.03%	<0.01%	0.03%
32102	Mixed, Medium grade	1.2%	0.22%	0.09%	0.13%
32103	Oxide, Medium grade	0.6%	0.07%	<0.01%	0.07%
32104	Sulphide, Medium grade	0.6%	0.24%	<0.01%	0.24%
32105	Sulphide, Medium grade	0.7%	0.22%	0.05%	0.17%
32106	Mixed, Low grade	0.1%	0.01%	<0.01%	0.01%
32107	Mixed, Medium grade	0.1%	0.02%	0.01%	0.01%
32108	Oxide, Low grade	0.1%	0.02%	0.01%	0.01%
32109	Sulphide, Low grade	0.3%	0.28%	0.19%	0.10%

Whole-rock analysis was also conducted using standard geochemical procedures; the sample was assayed for 11 elements. The following observations are noted based on the whole-rock analysis of the composites:

- KCA Sample No. 32101, Mixed Low grade North, showed high barium.
- KCA Sample No. 32102, Mixed Medium grade North, showed high amounts of carbon, lead, and tungsten.
- KCA Sample No. 32105, Sulphide Medium grade North showed high arsenic.
- KCA Sample No. 32107, Mixed Medium grade South showed low total copper.

13.4.1 Rock Density (KCA 2005a)

Rock density tests were completed on selected pieces of waste rock from various drill holes and depth intervals (54 ft to 770 ft) for the Converse Project. The rock density test results averaged 2567 kg/m^3.

13.4.2 Concentration (KCA 2005a)

Two gravity concentration tests were conducted on each composite: one at 10 psig and the other at 2 psig. The gravity tests were conducted on samples crushed to 65 mesh (Tyler) at 25% solids.

The weight percent of gold in the concentrate ranged from 16% to 59% at just 2% to 9% of the total initial weight for the 10 psig, and 71% to 86% at 20% to 24% (excluding test no. 32811 E) of the total initial weight for the 2 psig. The results of the gravity concentration tests are summarized in Table 13-7.

Table 13-7: KCA Summary of Falcon Gravity Concentration Tests

KCA Sample No.	Pressure (psig)	Sample Weight (g)	Concentrate (s)						Tailings					
			Weight (g)	Weight (%)	Au (oz/st)	Au (%)	Ag (oz/st)	Ag (%)	Weight (g)	Weight (%)	Au (oz/st)	Au (%)	Ag (oz/st)	Ag (%)
32101	10	999.11	15.37	2%	0.192	16%	0.08	1%	983.38	98%	0.017	84%	0.1	99%
32102	10	986.57	35.04	4%	0.642	52%	1.03	14%	951.53	96%	0.022	48%	0.24	86%
32103	10	999.32	56.64	6%	0.276	44%	0.38	22%	942.68	94%	0.021	56%	0.08	78%
32104	10	998.78	55.4	6%	0.239	45%	0.33	29%	943.38	94%	0.018	55%	0.05	71%
32105	10	995.72	42.38	4%	0.527	59%	0.75	13%	953.34	96%	0.016	41%	0.23	87%
32106	10	998.53	42.8	4%	0.24	42%	0.18	16%	955.73	96%	0.015	58%	0.04	84%
32107	10	1005.77	47.47	5%	0.143	26%	0.13	15%	958.3	95%	0.02	74%	0.04	85%
32108	10	1006.82	76.89	8%	0.102	33%	0.05	14%	929.93	92%	0.018	67%	0.03	86%
32109	10	1000.14	90.61	9%	0.132	44%	0.07	22%	909.53	91%	0.017	56%	0.03	78%
32101	2	999.17	197.84	20%	---	71%	---	66%	801.33	80%	0.006	29%	0.03	34%
32102	2	499.1	244.14	49%	---	86%	---	53%	254.96	51%	0.012	14%	0.26	47%
32103	2	998.86	229.03	23%	---	73%	---	35%	769.83	77%	0.012	27%	0.17	65%
32104	2	999.49	219.39	22%	---	85%	---	63%	780.1	78%	0.006	15%	0.06	37%
32105	2	998.11	208.68	21%	---	83%	---	27%	789.43	79%	0.008	17%	0.2	73%
32106	2	998.82	241.51	24%	---	76%	---	40%	757.31	76%	0.007	24%	0.05	60%
32107	2	998.51	211.57	21%	---	73%	---	43%	786.94	79%	0.011	27%	0.05	57%
32108	2	996.66	207.08	21%	---	75%	---	56%	789.58	79%	0.008	25%	0.04	44%
32109	2	995.38	230.7	23%	---	81%	---	31%	764.68	77%	0.007	19%	0.09	69%

13.4.3 Comminution Test (KCA 2005a)

KCA sent three composites, grouped by redox type, to Hazen Research Inc. for comminution testing to determine the Bond rod mill work index (RWi) and Bond mall mill work index (BWi), as well as the abrasion index (Ai). The comminution results indicate that the material is hard and moderately to very abrasive. Results are shown in Table 13-8.

Table 13-8: Bond Ball and Rod Work Index, Abrasion Index Results

HRI No.	KCA Composite No.	Rock Composition	RWi (kWhr/st)	BWi (kWhr/st)	Ai (g)
50958-1	33103	Oxide	15.7	14.6	0.490
50958-2	33104	Mixed	18.0	15.4	0.662
50958-3	33105	Sulphide	18.0	15.5	0.701

13.4.4 Bottle Roll Leach Test (KCA 2005a)

Bottle roll leach tests were conducted during Phases 1 and 2 of the 2005a program. Phase 1 included bottle roll tests for each composite at 75 µm grind size. Phase 2 included bottle roll leach tests for each composite at 1.7 mm. The summarized results of the bottle roll leach tests are presented in Table 13-9.

From the bottle roll tests, the 1.7 mm material had gold recoveries ranging from 47% to 63% with cyanide consumptions ranging from 0.2 kg/t to 1.5 kg/t. The 75 µm material had gold recoveries ranging from 95% to 98% with cyanide consumptions ranging from 0.2 kg/t to 2.8 kg/t. The north deposit samples showed significantly higher soluble copper levels (676 g/t versus 132 g/t average), resulting in substantially greater cyanide consumption than the south deposit (1.48 kg/t versus 0.38 kg/t average). The sulphide samples also exhibited higher cyanide consumption compared to the other sample types (1.48 kg/t versus 0.38 kg/t average), which may be attributed to above-average soluble copper levels (523 g/t versus 390 g/t average).

Table 13-9: KCA Bottle Roll Leach Tests – Summary of Results

KCA Sample #	Phase	North / South	Redox	Crush Size P_{80} (mm)	Calc. Head Grade Au (g/t)	Calc Head Grade Ag (g/t)	Cyanide Soluble Cu (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (days)	NaCN Consumption (kg/t)	Lime Addition (kg/t)
32101 B	P-1	North	Mixed	0.075	0.656	3.035	228	95%	61%	4	0.7	1
32101 C	P-2	North	Mixed	1.7	0.625	2.971	228	52%	40%	7	0.7	0.5
32102 B	P-1	North	Mixed	0.075	1.375	7.189	1048	97%	84%	4	2.8	1
32102 C	P-2	North	Mixed	1.7	1.469	7.046	1048	55%	40%	7	1.5	0.5
32103 B	P-1	North	Oxide	0.075	1.125	7.171	860	98%	69%	4	1.7	1.5
32103 C	P-2	North	Oxide	1.7	1.125	5.498	860	58%	37%	7	1.4	0.5
32104 B	P-1	North	Sulphide	0.075	0.812	4.203	463	96%	59%	4	1.5	1
32104 C	P-2	North	Sulphide	1.7	0.937	2.383	463	53%	67%	7	0.8	0.5
32105 B	P-1	North	Sulphide	0.075	1.344	4.74	783	98%	60%	4	2.5	1
32105 C	P-2	North	Sulphide	1.7	0.937	1.953	783	49%	41%	7	1.2	0.5
32106 B	P-1	South	Mixed	0.075	0.687	2.798	105	95%	58%	4	0.3	1.5
32106 C	P-2	South	Mixed	1.7	0.781	2.969	105	48%	39%	7	0.2	0.5
32107 B	P-1	South	Mixed	0.075	1.031	2.66	54	97%	56%	4	0.1	1.5
32107 C	P-2	South	Mixed	1.7	0.812	4.822	54	63%	14%	7	0.2	0.5
32108 B	P-1	South	Oxide	0.075	0.719	2.568	46	96%	54%	4	0.2	1.5
32108 C	P-2	South	Oxide	1.7	1.125	1.356	46	56%	63%	7	0.4	0.5
32109 B	P-1	South	Sulphide	0.075	0.719	1.992	323	96%	41%	4	1.2	1
32109 C	P-2	South	Sulphide	1.7	1.156	1.443	323	47%	46%	7	0.4	0.5

13.4.5 Column Leach Tests (KCA 2005a)

Column leach tests for Phase 1 and Phase 2 were conducted on the nine sample composites at three nominal crush sizes of minus 9.5 mm (Phase 1) and 37.5 mm (Phase 2). The 37.5 mm material had gold recoveries ranging from 21% to 56% with cyanide consumptions ranging from 0.3 kg/t to 1.3 kg/t after leaching for 62 days. The 9.5 mm material had gold recoveries ranging from 53% to 74% with cyanide consumptions ranging from 1.2 kg/t to 2.3 kg/t after leaching for 120 days.

The oxide samples for both North and South deposits exhibited higher cyanide consumption than the other sample types (1.5 kg/t versus 1.0 kg/t average). Overall, the sulphides had the lowest average gold recovery of 40%, and the North deposit sulphide samples had a lower recovery than the South at 43% versus 52%. Silver recoveries showed no significant trend with composite type and crush size. Gold and silver recoveries from each of the column tests are summarized in Table 13-10. Material crush size versus recovery for gold and silver is presented in Figure 13-2 and Figure 13-3, respectively.

Leach kinetics were relatively slow, with all of the columns appearing to still be leaching at the end of the test. Gold recovery versus days is presented in Figure 13-4.

Table 13-10: KCA Column Leach Test Summary of Carbon Data

KCA Sample #	KCA Test #	North / South	Redox	Target Crush Size (mm)	Calc. Head Grade (Au, g/t)	Calc Head Grade (Ag g/t)	Cyanide Soluble Copper (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (days)	NaCN Consumption (kg/t)	Lime Addition (kg/t)
32101 B	32121	North	Mixed	9.5	0.69	1.94	228	59%	36%	120	1.39	0.5
32101 C	32826	North	Mixed	37.5	0.75	5	228	28%	5%	62	0.51	0.5
32102 B	32124	North	Mixed	9.5	1.41	5.15	1048	60%	45%	120	2.13	0.5
32102 C	32829	North	Mixed	37.5	1.34	10.31	1048	28%	10%	62	0.78	0.5
32103 B	32127	North	Oxide	9.5	1.31	5.21	860	60%	34%	120	2.32	0.5
32103 C	32832	North	Oxide	37.5	1.37	9.06	860	32%	10%	62	1.27	0.5
32104 B	32130	North	Sulphide	9.5	1	3.16	463	53%	41%	120	1.54	0.5
32104 C	32835	North	Sulphide	37.5	1	6.56	463	26%	10%	62	0.37	0.5
32105 B	32133	North	Sulphide	9.5	1.28	3.71	783	54%	41%	120	1.78	0.5
32105 C	32838	North	Sulphide	37.5	1.12	3.12	783	21%	17%	62	0.42	0.5
32106 B	32136	South	Mixed	9.5	0.78	1.81	105	67%	31%	120	1.23	0.5
32106 C	32841	South	Mixed	37.5	0.91	3.44	105	35%	10%	62	0.47	0.5
32107 B	32139	South	Mixed	9.5	0.97	1.71	54	74%	27%	120	1.35	0.5
32107 C	32844	South	Mixed	37.5	1.22	1.56	54	56%	15%	62	0.52	0.5
32108 B	32142	South	Oxide	9.5	0.72	2.71	46	74%	19%	120	1.52	0.5
32108 C	32847	South	Oxide	37.5	0.84	1.56	46	44%	21%	62	0.87	0.5
32109 B	32145	South	Sulphide	9.5	0.75	1.93	323	54%	19%	120	1.19	0.51
32109 C	32850	South	Sulphide	37.5	0.81	1.25	323	22%	10%	62	0.3	0.5

Figure 13-2: Material Size versus Recovery - Gold

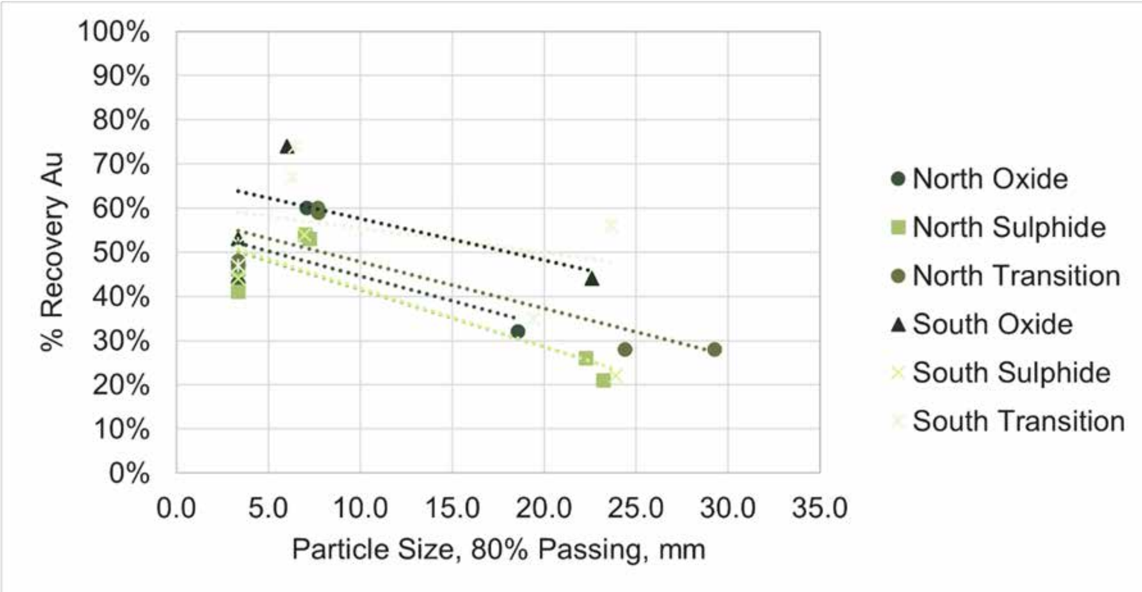


Figure 13-3: Material Size versus Recovery - Silver

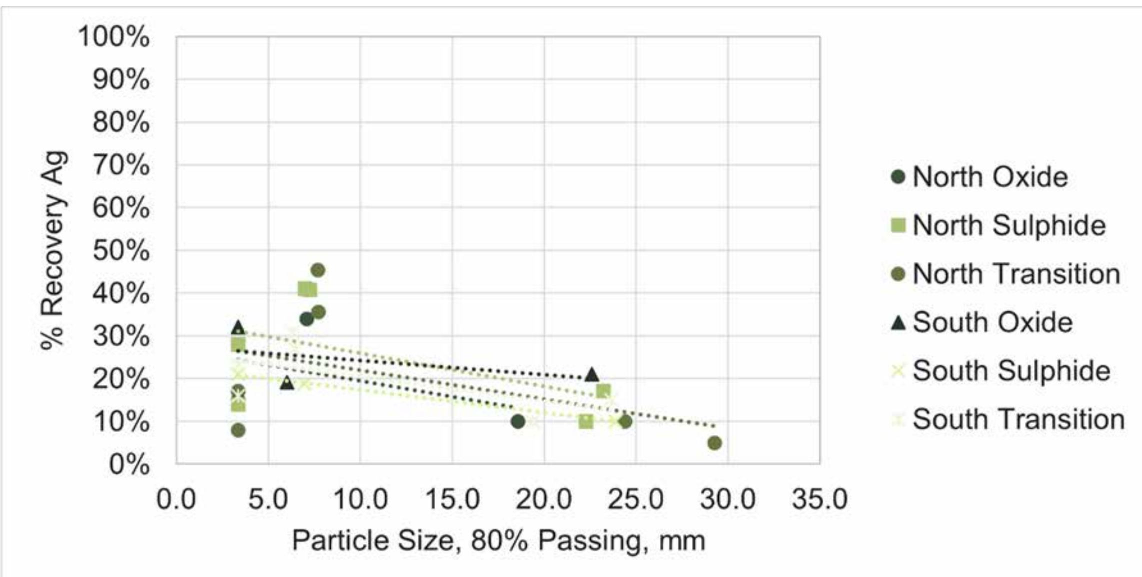
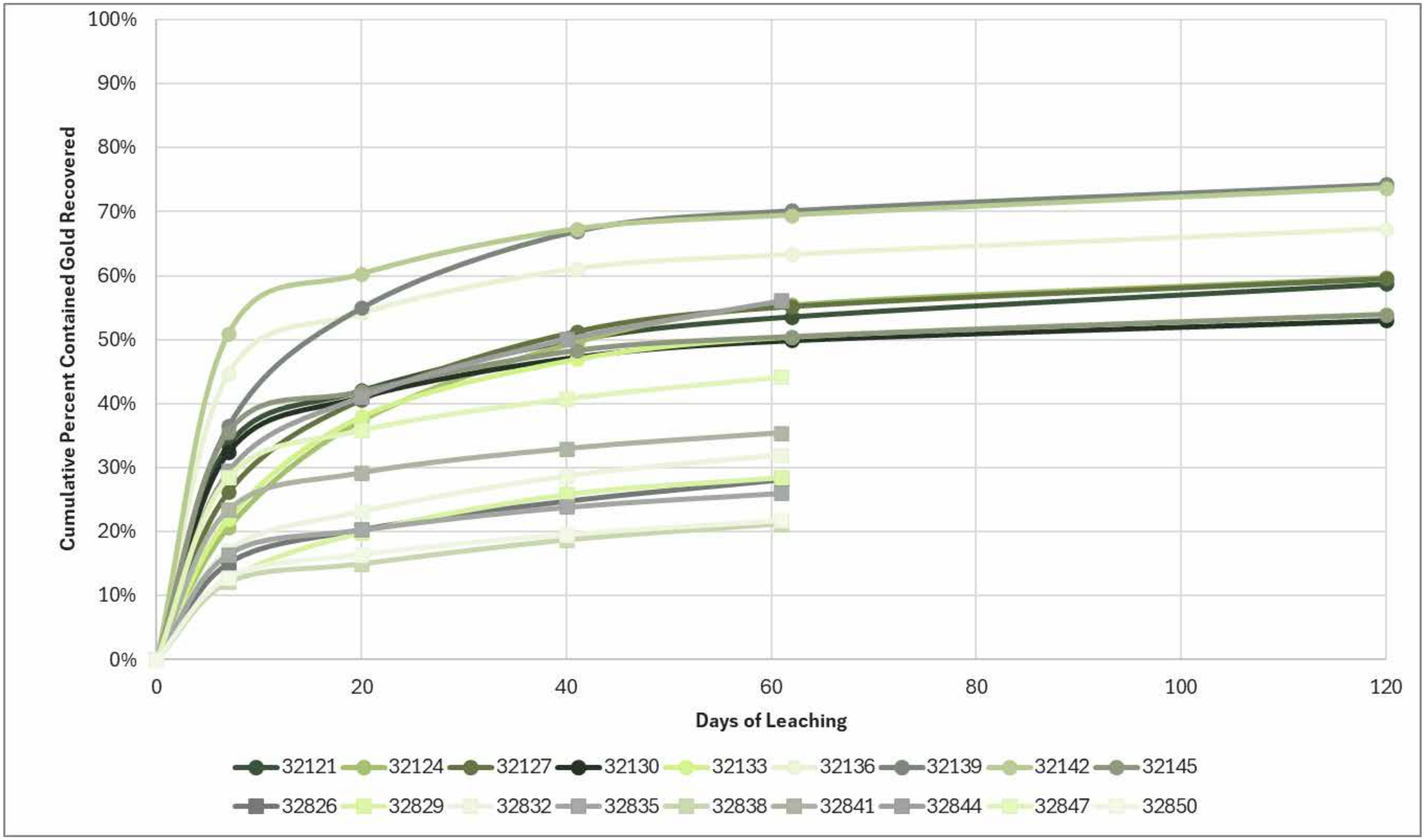


Figure 13-4: Column Leaching Rate on Composite Samples - Gold



13.4.5.1 Bottle Roll Tests on Column Tails (KCA 2005a)

Bottle roll tests were conducted on the 38 mm column leach tail samples following 62 days of leaching. The summarized results of the bottle roll leach tests are presented in Table 13-11. Gold extraction ranged from 1% to 10%, while NaCN consumption ranged from 0.1 kg/t to 0.9 kg/t. These results indicate that some column tests had not reached ultimate leach recovery at the time the tests were terminated.

Table 13-11: KCA Bottle Roll Leach Tests on minus 37.5 mm Column Tails

KCA Sample #	Redox	Calc. Head Grade Au (g/t)	Calc Head Grade Ag (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (days)	NaCN Consumption (kg/t)
32101 C	Mixed	0.344	5.312	4%	1%	4	0.6
32102 C	Mixed	0.875	8.125	1%	4%	4	0.7
32103 C	Oxide	0.687	6.875	3%	2%	4	0.1
32104 C	Sulphide	0.719	2.500	2%	4%	4	0.9
32105 C	Sulphide	0.719	5.000	1%	2%	4	0.7
32106 C	Mixed	0.719	4.375	3%	2%	4	0.7
32107 C	Mixed	0.719	2.812	7%	1%	4	0.7
32108 C	Oxide	0.719	1.875	10%	2%	4	0.3
32109 C	Sulphide	0.719	2.187	2%	1%	4	0.6

13.4.5.2 Column Tests on Crushed Column Tails (KCA 2005a)

Portions of each 38 mm column leach tail sample were re-crushed to 100% passing 6.3 mm and then subjected to an additional 41 days of column leaching. The summarized results of the column leach tests are presented in Table 13-12. Gold extraction ranged from 42% to 53%, indicating that a finer crush size could have a positive effect on recovery.

Table 13-12: Column Leach Tests on Re-Crushed Column Tails passing 6.3 mm

KCA Sample #	KCA Test #	Redox	Calc. Head Grade Au (g/t)	Calc Head Grade Ag (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (days)	NaCN Consumption (kg/t)
32101 C	33801	Mixed	0.531	4.69	47%	7%	41	0.74
32102 C	33804	Mixed	0.969	9.37	48%	17%	41	1.14
32103 C	33807	Oxide	0.937	8.12	43%	15%	41	1.60
32104 C	33810	Sulphide	0.750	5.94	42%	16%	41	0.73
32105C	33813	Sulphide	0.875	2.50	46%	25%	41	0.83
32106	33816	Mixed	0.594	3.12	47%	20%	41	0.70
32107	33819	Mixed	0.531	1.25	53%	25%	41	0.56
32108	33822	Oxide	0.469	1.25	53%	25%	41	0.78
32109	33825	Sulphide	0.625	1.25	45%	25%	41	0.60

13.4.5.3 Leach Solution – Copper (KCA 2005a)

Interim pregnant (effluent) cyanide leach solutions from column tests were assayed periodically for copper content. The lowest and highest copper in solution data obtained over the leach period are summarized in Table 13-13, Table 13-14, and Table 13-15. Multiple composite samples had relatively high copper values, which could be a cause of the higher cyanide consumption observed in several column tests.

Table 13-13: Copper Concentration in 37.5 mm Column Leach Solutions

KCA Sample No.	Designation	Location	KCA Test No.	Low Cu (mg/L)	High Cu (mg/L)
32101 C	Mixed, Low grade	North	32826	49.5	127
32102 C	Mixed, Medium grade	North	32829	187	547
32103 C	Oxide, Medium grade	North	32832	28.2	404
32104 C	Sulphide, Medium grade	North	32835	63.4	198
32105 C	Sulphide, Medium grade	North	32838	134	293
32106 C	Mixed, Low grade	South	32841	20.5	104
32107 C	Mixed, Medium grade	South	32844	14.3	24.7
32108 C	Oxide, Low grade	South	32847	12.3	24
32109 C	Sulphide, Low grade	South	32850	30.1	58.7

Table 13-14: Copper Concentration in 9.5 mm Column Leach Solutions

KCA Sample No.	Designation	Location	KCA Test No.	Low Cu (mg/L)	High Cu (mg/L)
32101 B	Mixed, Low grade	North	32121	116	340
32102 B	Mixed, Medium grade	North	32124	357	1104
32103 B	Oxide, Medium grade	North	32127	273	933
32104 B	Sulphide, Medium grade	North	32130	77	397
32105 B	Sulphide, Medium grade	North	32133	224	710
32106 B	Mixed, Low grade	South	32136	41.6	206
32107 B	Mixed, Medium grade	South	32139	14.2	84
32108 B	Oxide, Low grade	South	32142	17.1	64.8
32109 B	Sulphide, Low grade	South	32145	49.1	209

Table 13-15: Copper Concentration in 6.3 mm Column Leach Solutions

KCA Sample No.	Designation	Location	KCA Test No.	Low Cu (mg/L)	High Cu (mg/L)
32826	Mixed, Low grade	North	33801	1.51	204
32829	Mixed, Medium grade	North	33804	7.39	636
32832	Oxide, Medium grade	North	33807	16.5	497
32835	Sulphide, Medium grade	North	33810	17.1	184
32838	Sulphide, Medium grade	North	33813	89.7	267
32841	Mixed, Low grade	South	33816	1.34	182
32844	Mixed, Medium grade	South	33819	1.33	51.4
32847	Oxide, Low grade	South	33822	3.09	23.9
32850	Sulphide, Low grade	South	33825	1.44	120

The conclusion for the 2005a KCA test program indicates that the materials is amenable to either agitated cyanide or heap leach treatment. Based on the gravity test work, the material could undergo gravity concentration if an agitated leach circuit is selected.

13.5 KCA Test Program 2005b

The second KCA test program included 750 bottle roll tests on assay reject samples from exploration drilling in both the South and North zones, including core and RC samples crushed to a nominal -1.7 mm size and 48 hour leach. The highest gold recovery was observed in hole NK-151, with a value of 72% gold recovered, an associated 35% silver recovery, and a moderately lower cyanide consumption of 0.39 kg/t. The lowest gold recovery was observed in NK-110, with an average of only 10% gold recovered, associated with 18% silver recovered, and the highest cyanide consumption of 1.3 kg/t. The average recoveries and reagent requirements for the assay reject bottle rolls based on Hole ID are presented in Table 13-16.

Table 13-16: Average Bottle Roll Results by Hole Summary

Hole	North/ South	Number of Samples	Avg. Calc. Head, Au (g/t)	Avg. Tail, Au (g/t)	Avg. Ext. Au (%)	Avg. Calc. Head, Ag (g/t)	Avg. Tail, Ag (g/t)	Avg. Ext. Ag (%)	Avg. Cons NaCN (kg/t)	Avg. Addition $Ca(OH)_2$ (kg/t)	Avg. Solution AAS (mg Cu/L)	Avg. Extraction Calc. (mg Cu/kg)
NK-37	North	3	0.577	0.292	50%	1.22	0.78	33%	0.477	0.51	5.6	5.76
NK-44	South	3	0.675	0.271	57%	1.02	0.78	23%	0.469	0.89	11.4	13.05
NK-52	South	4	0.426	0.172	49%	3.54	2.27	26%	0.709	1.4	147.09	151.87
NK-57	South	9	0.666	0.212	64%	4.72	4.03	16%	0.429	1.02	18.63	19.08
NK-63	North	8	0.565	0.215	61%	8.66	7.52	13%	0.205	1.46	12.13	12.47
NK-75	North	6	0.888	0.531	47%	11.05	9.64	12%	0.996	3.5	202.87	216.06
NK-86	North	11	0.778	0.293	63%	4.34	3.53	28%	0.481	0.64	30.81	31.08
NK-88	North	7	1.81	1.04	36%	6.9	6.61	6%	0.729	0.93	176.16	176.95
NK-92	South	7	0.575	0.259	57%	4.58	4.13	15%	0.574	0.65	93.46	93.87
NK-101	South	4	0.657	0.32	51%	10.67	9.06	15%	0.549	0.51	17.53	18.15
NK-102	South	2	0.369	0.172	53%	6.82	6.56	4%	0.446	0.51	4.12	4.22
NK-110	North	8	1.162	1.027	10%	44.59	36.21	18%	1.291	2.14	134.03	135.13
NK-118	North	73	0.623	0.311	50%	3.15	2.35	28%	0.539	1.07	93.03	113.17
NK-120	North	111	0.848	0.374	56%	2.98	2.29	29%	0.505	1.47	175.61	181.17
NK-121	North	12	0.344	0.141	56%	3.18	2.77	19%	0.308	1.01	75.93	80.92
NK-123	South	94	0.673	0.285	58%	1.84	1.3	33%	0.257	1.35	15.51	15.82
NK-130	North	60	0.788	0.248	60%	10.49	4.28	32%	0.502	1.49	117.77	119.44
NK-132	North	11	0.945	0.625	38%	8.04	6.86	13%	0.67	0.73	37.35	37.93
NK-138	South	73	0.908	0.577	43%	4.93	4.53	14%	0.541	1.18	12.66	12.84
NK-140	North	79	0.903	0.574	41%	3.84	3.38	18%	0.451	1.02	102.77	102.83
NK-144	South	52	0.976	0.454	53%	3.04	2.81	17%	0.209	1.36	17.06	17.29

Hole	North/ South	Number of Samples	Avg. Calc. Head, Au (g/t)	Avg. Tail, Au (g/t)	Avg. Ext. Au (%)	Avg. Calc. Head, Ag (g/t)	Avg. Tail, Ag (g/t)	Avg. Ext. Ag (%)	Avg. Cons NaCN (kg/t)	Avg. Addition $Ca(OH)_2$ (kg/t)	Avg. Solution AAS (mg Cu/L)	Avg. Extraction Calc. (mg Cu/kg)
NK-146	South	15	0.64	0.231	62%	2.13	1.25	26%	0.593	0.65	62.52	65.26
NK-147	South	7	0.903	0.308	69%	1.92	1.65	17%	0.239	0.66	9.27	9.75
NK-148	South	4	0.293	0.098	66%	1.14	0.98	14%	0.266	1.43	3.57	3.7
NK-150	South	36	0.617	0.235	63%	2.62	2.04	30%	0.412	1.21	23.98	24.27
NK-151	North	22	0.369	0.085	72%	4.51	3.01	35%	0.387	1.51	39.83	40.08
NK-160	South	18	0.318	0.162	49%	1.33	0.87	32%	0.404	1.02	87.86	93.13
NK-164	South	7	0.467	0.183	59%	1.87	1.21	39%	0.346	0.58	24.03	25
NK-167	North	4	0.276	0.164	36%	10.3	7.23	39%	0.695	1.25	68.36	68.51
Average North			0.804	0.444	47%	9.3	7.14	24%	0.596	1.2	95.6	98.84
Average South			0.616	0.274	57%	3.57	3.03	21%	0.403	0.93	30.96	31.94

13.6 MLI Variability Testing 2009

In 2009, MLI conducted a metallurgical testing program on 50 representative drill core composites from the Converse Project to verify earlier results. The initial phase involved bottle roll tests on 39 composites, selected based on mineral zone, oxidation, depth, gold and copper grades, and rock type to assess variability in whole-rock cyanidation.

The preliminary 39 composites crushed to a P_{80} of -1.7 mm were subjected to bottle roll leach tests. The results, sorted by oxide rating and lithology, provided further insight into recovery trends. The highest gold recoveries were observed in the sulphide materials at 69%, along with the highest cyanide consumption of 1.49 kg/t. The North and South samples had approximately equivalent gold recoveries; however, the North deposit's high copper content led to cyanide consumption higher than that in the South. These results follow trends similar to those of the KCA test program from 2005. The summary of the bottle roll results is presented in Table 13-17.

Table 13-17: Bottle Roll Average Result Summary (MLI 2009)

Material Type	No. of Composites	Au Recovery (%)	Au (g/t)			Reagent Requirements (kg/t)	
			Extracted	Tail	Calculated Head	NaCN Consumed	Lime Added
North	17	66%	1.39	0.61	2.00	1.10	2.50
South	22	65%	0.59	0.30	0.89	0.55	2.20
Oxide	14	68%	0.46	0.19	0.64	0.29	3.10
Mixed	22	63%	1.22	0.61	1.83	1.02	1.90
Sulphide	3	69%	1.10	0.34	1.44	1.49	1.80
Siltstone	7	64%	0.90	0.39	1.29	0.94	2.60
Sandstone	8	67%	0.73	0.32	1.05	0.74	2.40
Chert	5	61%	0.80	0.53	1.33	1.16	1.60
Porphyry	3	54%	0.29	0.28	0.57	0.62	1.80

Results from the bottle roll tests informed the creation of 11 larger metallurgical composites as well as additional master composites for more detailed analysis. Testing on these larger composites included column tests to examine the effects of crush size, cyanide feed levels, and lime addition on four lower grade samples and on one master composite representing the overall material grade. Higher grade composites were tested for gravity concentration, combined gravity and cyanidation, and bulk sulphide flotation methods. The composite characterization is presented in Table 13-18 and the carbon and sulphur content in Table 13-19.

Table 13-18: MLI 2009 Additional Composites Characterization

North / South	Composite ID	Oxide	Au Grade	Copper Grade
North	NLGOx	Oxide	Low	Med.
South	SLGOx	Oxide	Low	Med.
North	NLGSulf	Sulphide	Low	Med.
South	SLGSulf	Sulphide	Low	Med.
Both	LGSulf Master	Sulphide	Low	Med.
Both	Master	Mix	Med.	Med.
North	NHGOx	Oxide	High	Med.
South	SHGOx	Oxide	High	Med.
North	NHGSulf	Sulphide	High	High
South	SHGSulf	Sulphide	High	High
North	NLGHiCu	Mix	Low	High
South	SLGHiCu	Mix	Low	High
Both	LGHiCu Master	Mix	Low	High
North	NHGHiCu	Mix	High	High
South	SHGHiCu	Mix	High	High

Table 13-19: MLI 2009 Additional Composites Carbon and Sulphur Content

Composite ID	Oxide	Total Carbon (%)	Total Sulphur (%)
NLGOx	Oxide	0.29%	0.07%
SLGOx	Oxide	0.27%	0.04%
NLGSulf	Sulphide	0.46%	0.83%
SLGSulf	Sulphide	0.30%	0.57%
LGSulf Master	Sulphide	0.41%	0.81%
Master	Mix	0.28%	0.15%
NHGOx	Oxide	0.25%	0.11%
SHGOx	Oxide	0.40%	0.08%
NHGSulf	Sulphide	0.60%	0.61%
SHGSulf	Sulphide	0.49%	0.47%
NLGHiCu	Mix	0.41%	0.47%
SLGHiCu	Mix	0.28%	0.18%
LGHiCu Master	Mix	0.32%	0.29%
NHGHiCu	Mix	0.54%	0.36%
SHGHiCu	Mix	0.60%	0.28%

The secondary set of bottle rolls was crushed to a nominal minus 1.7 mm with 1 g NaCN/L. The observed lowest gold recovery was 49% from the North high-grade sulphide composite (NLGSulf), and the highest of 81% from the South low-grade high copper composite (SLGHiCu), which also saw high cyanide consumption. The summary of the composite bottle roll test results is presented in Table 13-20.

Table 13-20: MLI 2009 Additional Composites Bottle Roll Result Summary

Composite ID	Oxide	Calc. Head Grade Au (g/t)	Calc. Head Grade Ag (g/t)	Total Copper (g/t)	Au Extracted (%)	Ag Extracted (%)	Calc Cu Extraction (%)	Leach Time (days)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
NLGOx	Oxide	0.93	3	651	72%	33%	14%	7	0.66	1.9
SLGOx	Oxide	0.61	3	648	69%	33%	13%	7	0.51	2.9
NLGSulf	Sulphide	0.64	4	1163	53%	25%	14%	7	1.72	2.4
SLGSulf	Sulphide	0.51	5	843	67%	20%	29%	7	1.64	2.6
LGSulf Master	Sulphide	0.59	2	1109	63%	50%	28%	7	2.48	2.8
Master	Mix	1.06	3	940	68%	33%	18%	7	1.06	1.4
NHGOx	Oxide	7.19	7	744	69%	29%	14%	7	0.62	2.6
SHGOx	Oxide	2.25	5	1301	68%	20%	22%	7	1.29	2.2
NHGSulf	Sulphide	3.21	11	1606	49%	46%	11%	7	1.25	1.9
SHGSulf	Sulphide	1.91	5	1529	70%	60%	25%	7	2.35	1.3
NLGHiCu	Mix	1.45	8	2099	66%	25%	18%	7	2.09	2.8
SLGHiCu	Mix	0.89	4	1719	81%	50%	18%	7	1.47	2.9
LGHiCu Master	Mix	1.14	7	1850	65%	29%	19%	7	1.89	2.8
NHGHiCu	Mix	2.33	6	3488	70%	50%	27%	7	4.06	2.3
SHGHiCu	Mix	2.53	3	3052	71%	33%	23%	7	3.1	2

13.6.1 MLI 2009 Column Leach Tests

Column tests were conducted on the additional composites to assess grade, copper content, and oxide rating, all crushed to a P_{80} of minus 9.5 mm. Additional column leach tests at various crush sizes were also performed on a master composite. Gold recovery values ranged from 32.8% for the low-grade high sulphide composite, possibly due to sulphide encapsulation, to 65.3% for the South low-grade oxide composites. The average gold recoveries for the master composites increased at finer crush sizes, going from 35.1% at 37.5 mm to 78.2% at 1.7 mm. The column test results are presented in Table 13-21.

Table 13-21: Column Test Results Summary, Converse Heap Leach Composites (MLI 2009)

Composite	Feed Size P_{80} (mm)	Leach Time (days)	Au Recovery (%)	Au (g/t)			Reagent Requirements (kg/t)	
				Extracted	Tails	Calculated Head	NaCN Consumed.	Lime Added
NLGOx	9.5	139	45%	0.21	0.26	0.47	1.4	1.7
NLGOx	9.5	140	46%	0.25	0.29	0.54	1	1.7
NLGOx	9.5	141	48%	0.25	0.27	0.52	0.6	1.7
NLGOx	9.5	141	51%	0.25	0.24	0.49	1.7	1.1
NLGOx	9.5	142	47%	0.22	0.25	0.47	1.1	1.1
NLGOx	9.5	141	46%	0.22	0.26	0.48	0.6	1.1
SLGOx	9.5	141	48%	0.24	0.26	0.5	1.5	2.6
SLGOx	9.5	142	49%	0.27	0.28	0.55	1	2.6
SLGOx	9.5	142	47%	0.23	0.26	0.49	0.6	2.6
SLGOx	9.5	142	65%	0.32	0.17	0.49	1.9	1.7
SLGOx	9.5	140	56%	0.27	0.21	0.48	1.1	1.7
SLGOx	9.5	141	54%	0.27	0.23	0.5	0.7	1.7
LGHiCu Master	9.5	188	53%	0.56	0.49	1.05	3.2	2.5
LGHiCu Master	9.5	188	52%	0.51	0.47	0.98	2.3	2.5
LGHiCu Master	9.5	188	44%	0.44	0.57	1.01	1.2	2.5
LGHiCu Master	9.5	188	55%	0.57	0.47	1.04	3.4	1.7
LGHiCu Master	9.5	189	50%	0.54	0.55	1.09	2.3	1.7
LGHiCu Master	9.5	188	46%	0.5	0.6	1.1	1.4	1.7
LGSulf Master	9.5	121	33%	0.18	0.36	0.54	0.9	2.5
LGSulf Master	9.5	118	33%	0.2	0.41	0.61	1	1.7
LGSulf Master	9.5	118	35%	0.19	0.36	0.55	1.5	1.7
Master Comp	37.5	120	35%	0.26	0.48	0.74	1.1	1

Composite	Feed Size P_{80} (mm)	Leach Time (days)	Au Recovery (%)	Au (g/t)			Reagent Requirements (kg/t)	
				Extracted	Tails	Calculated Head	NaCN Consumed.	Lime Added
Master Comp	13	122	46%	0.4	0.48	0.88	1.3	1
Master Comp	9.5	117	51%	0.45	0.44	0.89	1.4	1
Master Comp	6.3	122	60%	0.53	0.36	0.89	1.4	1
Master Comp	1.7	124	78%	0.68	0.19	0.87	1.6	1

13.6.2 Gravity Recoverable Gold Tests (MLI 2009)

Gravity recoverable gold (GRG) tests were conducted on six high-grade mineral composites using a Knelson concentrator. The purpose was to assess their response to gravity concentration treatment. The composites responded reasonably well, with total GRG content ranging from 55.6% to 84.3%. Notably, the gold was generally liberated at relatively coarse grind sizes between minus 20 and 65 mesh (Tyler). These results indicate the potential to produce high-grade gravity concentrates from the high-grade Converse material. The results for the GRG test of high-grade material are presented in Table 13-22.

Table 13-22: Summary GRG Test Results, High- Grade Oxide Core Composites

Sample	Au Recovery, % of Contained Values Nominal Grind Size					Calculated Head Grade (g/t)
	0.85 mm	0.212 mm	0.15 mm	0.075 mm	Total	
NHGOx	50%	25%	7%	3%	84%	0.843
SHGOx	36%	20%	4%	3%	63%	0.634
NHGHiCu	32%	24%	7%	4%	67%	0.667
SHGHiCu	35%	22%	6%	3%	66%	0.662
NHGSulf	21%	23%	8%	4%	57%	0.566
SHGSulf	29%	16%	6%	4%	56%	0.556

13.6.3 Gravity Cyanidation Test (MLI 2009)

Gravity concentration tests were performed on the same six high-grade composites from the GRG test, using agitated cyanidation of the gravity tailings. These tests were conducted at feed sizes of 80% at minus 0.212 mm and 80% at minus 0.075 mm to gather data on the potential of the higher grade material for this hybrid processing method and to assess the potential for "pulp agglomeration" heap leaching of the gravity tailings. The results of the tailings gravity cyanidation are presented in Table 13-23.

Combined gravity and leach testing resulted in high overall recoveries ranging between 85% and 98%, and a hybrid flowsheet may be a viable option for the Project.

Table 13-23: Gravity/Cyanidation Tests, High Grade Oxide Core Composites

Sample ID	Feed Size P_{80} (mm)	Gold Extraction, % of total				Silver		Reagents	
		by Gravity[1]	by Cyanidation[2] (72 hrs)	Total (Gravity/ Cyanidation)	Calculated Head (g/t)	Extraction % of total	Calculated Head (g/t)	NaCN Consumed (kg/t)	Lime Added (kg/t)
NHGOx	0.212	58%	39%	97%	5.312	67%	3	0.15	1.6
	0.212	47%	47%	94%	4.344	75%	4	0.08	2
	0.075	61%	37%	98%	3.562	67%	3	0.21	1.8
SHGOx	0.212	20%	70%	90%	1.875	50%	2	0.37	1.3
	0.212	14%	75%	89%	2.062	67%	3	0.38	1.6
	0.075	32%	64%	96%	1.75	50%	2	0.38	1.3
NHGHiCu	0.212	44%	41%	85%	2.656	50%	6	1.97	2.5
	0.075	48%	44%	92%	2.562	50%	6	2.14	2.5
SHGHiCu	0.212	40%	51%	91%	2.781	50%	2	1.03	1.9
	0.075	44%	50%	94%	2.125	50%	2	1.06	2.3
NHGSulf	0.212	51%	37%	89%	3.375	58%	12	0.74	1.9
	0.075	50%	42%	92%	2.969	58%	12	1.08	2.2
SHGSulf	0.212	34%	51%	85%	2.031	50%	4	0.98	2.3
	0.075	40%	48%	88%	1.75	67%	3	0.96	1.6

Notes:

1. Gold values reporting to gravity concentrate do not account for any potential gold losses incurred during subsequent processing of the gravity concentrate for gold recovery.
2. Cyanidation of recombined gravity (rougher & cleaner) tailings.
3. Average of triplicate assays

13.6.4 Sulphide and Copper Floatation Test (MLI 2009)

Bulk flotation tests were conducted on each of the high-grade composites as well as the high copper composites. The bulk flotation tests were conducted on material at P_{80} passing 75 µm using and 30% solids by weight using Denver laboratory scale flotation units in five stages. The resulting rougher concentrate was combined and cleaned in a single stage to produce a cleaner concentrate. Results from the flotation tests are presented in Table 13-24.

The flotation tests resulted in reasonable recoveries for gold (88% to 89%) and silver (56% to 72%) with moderate mass pulls for a rougher concentrate. Copper recoveries were lower at 49% to 76%. As with the gravity and leach test work, the flotation results indicate that a hybrid flowsheet may be viable for treating the high-grade minerals.

Table 13-24: Flotation Concentration Test Results, 80% -200 mesh Feed Size

Sample ID	Product	Weight (%)	Assays			Au		Ag		Cu	
			(Au g/t)	(Ag g/t)	(% Cu)	(%)	(Cum. %)	(%)	(Cum. %)	(%)	(Cum. %)
NHGHiCu	Cl. Conc.	1.80%	132	222	9.20%	82.00%	82.00%	65.40%	65.40%	42.30%	42.30%
	Cl. Tail	3.70%	5.75	6	0.70%	7.40%	89.40%	3.60%	69.00%	6.80%	49.10%
	Ro. Tail	94.50%	0.34	2	0.20%	10.60%	100.00%	31.00%	100.00%	50.90%	100.00%
	Composite	100.00%	2.91	6	0.39%	100%		100%		100%	
SHGHiCu	Cl. Conc.	4.70%	38	39	4.90%	73.50%	73.50%	61.50%	61.50%	70.30%	70.30%
	Cl. Tail	6.50%	5.28	4	0.30%	14.20%	87.70%	8.70%	70.20%	5.50%	75.80%
	Ro. Tail	88.80%	0.34	1	0.10%	12.30%	100.00%	29.80%	100.00%	24.20%	100.00%
	Composite	100.00%	2.41	3	0.33%	100%		100%		100%	
NHGSulf	Cl. Conc.	2.00%	116	316	7.00%	77.20%	77.20%	49.40%	49.40%	77.90%	77.90%
	Cl. Tail	4.60%	2.34	19	0.30%	3.60%	80.80%	6.80%	56.20%	6.60%	84.50%
	Ro. Tail	93.40%	0.62	6	0.00%	19.20%	100.00%	43.80%	100.00%	15.50%	100.00%
	Composite	100.00%	3	13	0.18%	100%		100%		100%	
SHGSulf	Cl. Conc.	1.50%	100	143	9.10%	71.40%	71.40%	60.40%	60.40%	77.00%	77.00%
	Cl. Tail	2.10%	7.69	21	0.60%	7.70%	79.10%	12.40%	72.80%	6.60%	83.60%
	Ro. Tail	96.40%	0.44	1	0.00%	20.90%	100.00%	27.20%	100.00%	16.40%	100.00%
	Composite	100.00%	2.09	4	0.18%	100%		100%		100%	

13.7 MLI Grind Size Optimization 2013

The 2013 MLI program (Job 3765) specifically focused on grind size optimization. The metallurgical testing was conducted on 25 drill core composites using bottle roll testing at four crush sizes: 212 µm, 150 µm, 100 µm, and 75 µm. The average gold recoveries for most of the composites fit within a small range of 83% to 86%, and silver recoveries between 50% and 52%. The average cyanide consumption was between 0.61 kg/t and 0.72 kg/t. The samples tested appeared to be more amenable to cyanidation, with minimal interference from cyanide-soluble copper. Individual sample results generally showed that gold recoveries improved consistently with finer particle size, with higher recoveries in samples crushed to 75 µm compared with coarser fractions; however, silver extraction remained largely insensitive to grind size. Cyanide solubility tests indicated that portions of the gold were locked in sulphide, explaining the incomplete recoveries in some samples even under fine-grind conditions. The summary of the bottle roll average recoveries and reagent consumptions is presented in Table 13-25, and individual sample results are presented in Table 13-26.

Table 13-25: Average Grind Size Optimization Bottle Roll Tests Result Summary

Feed Size P_{80} (mm)	Avg Au Grade (g/t)	Avg Ag Grade (g/t)	Avg Soluble Cu (g/t)	Avg NaCN Cons (kg/t)	Avg. Lime Addition (kg/t)	Avg Au (%)	Avg Ag (%)
0.212	0.687	4.25	158	0.613	0.632	83%	50%
0.15	0.656	4.00	158	0.697	0.632	85%	52%
0.1	0.656	4.06	158	0.645	0.672	84%	51%
0.075	0.656	4.44	158	0.723	0.696	86%	50%

Table 13-26: Grind Size Optimization Bottle Roll Tests Result Summary

Composite	Feed Size P_{80} (mm)	Au Recovered (%)	Au Tail (g/t)	Au Calculated Head (g/t)	Ag Recovered (%)	Ag Tail (g/t)	Ag Calculated. Head (g/t)	NaCN Consumed (kg/t)	Lime Added (kg/t)
CONV-002C (260-270 ft.)	0.212	54%	0.159	0.344	28%	7.8	10.9	0.58	0.8
CONV-002C (260-270 ft.)	0.150	58%	0.159	0.375	27%	9	12.4	0.77	0.8
CONV-002C (260-270 ft.)	0.100	56%	0.150	0.344	29%	9.3	13.1	0.63	0.9
CONV-002C (260-270 ft.)	0.075	55%	0.150	0.344	32%	8.3	12.2	0.66	0.9
CONV-002C (480-490 ft.)	0.212	38%	0.981	1.594	52%	2.1	4.4	0.98	1.0
CONV-002C (480-490 ft.)	0.150	39%	0.959	1.562	53%	2.1	4.5	1.00	1.0
CONV-002C (480-490 ft.)	0.100	40%	0.941	1.562	56%	1.9	4.3	0.96	1.2
CONV-002C (480-490 ft.)	0.075	40%	0.931	1.531	57%	1.9	4.4	1.15	1.2
CONV-002C (875-885 ft.)	0.212	87%	0.131	0.969	9%	2.1	2.3	0.39	0.6
CONV-002C (875-885 ft.)	0.150	93%	0.059	0.844	10%	1.9	2.1	0.59	0.5
CONV-002C (875-885 ft.)	0.100	91%	0.081	0.906	13%	1.4	1.6	0.36	0.8
CONV-002C (875-885 ft.)	0.075	93%	0.059	0.812	13%	1.4	1.6	0.36	0.9
CONV-008C (280-290 ft.)	0.212	93%	0.031	0.437	>81.8%	<0.02	<1.1	0.51	0.5
CONV-008C (280-290 ft.)	0.150	96%	0.019	0.469	>84.6%	<0.02	<1.3	0.37	0.5
CONV-008C (280-290 ft.)	0.100	91%	0.041	0.437	52%	1	2.1	0.47	0.5
CONV-008C (280-290 ft.)	0.075	88%	0.059	0.469	>85.7%	<0.02	<1.4	0.26	0.7
CONV-008C (355-365 ft.)	0.212	79%	0.141	0.656	35%	8.5	13.1	3.39	0.5
CONV-008C (355-365 ft.)	0.150	79%	0.141	0.656	34%	8.9	13.4	4.45	0.5
CONV-008C (355-365 ft.)	0.100	75%	0.169	0.687	35%	7.9	12.2	3.21	0.7
CONV-008C (355-365 ft.)	0.075	83%	0.109	0.625	33%	9.3	13.9	3.66	0.7
CONV-009C (500-510 ft.)	0.212	91%	0.100	1.125	23%	1.7	2.2	0.88	0.5
CONV-009C (500-510 ft.)	0.150	91%	0.100	1.094	26%	1.4	1.9	0.95	0.5
CONV-009C (500-510 ft.)	0.100	94%	0.059	1.031	25%	1.2	1.6	1.14	0.5

Composite	Feed Size P_{80} (mm)	Au Recovered (%)	Au Tail (g/t)	Au Calculated Head (g/t)	Ag Recovered (%)	Ag Tail (g/t)	Ag Calculated. Head (g/t)	NaCN Consumed (kg/t)	Lime Added (kg/t)
CONV-009C (500-510 ft.)	0.075	95%	0.050	0.969	24%	1.3	1.7	0.98	0.7
CONV-013C (320-330 ft.)	0.212	91%	0.059	0.656	42%	5.5	9.5	0.44	0.7
CONV-013C (320-330 ft.)	0.150	93%	0.041	0.531	79%	1.1	5.3	0.59	0.7
CONV-013C (320-330 ft.)	0.100	93%	0.041	0.594	49%	4.7	9.2	0.53	0.7
CONV-013C (320-330 ft.)	0.075	93%	0.041	0.562	61%	3.2	8.2	0.89	0.5
CONV-013C (890-900 ft.)	0.212	85%	0.059	0.406	>40.0%	<0.3	<0.05	0.24	0.5
CONV-013C (890-900 ft.)	0.150	91%	0.041	0.406	40%	0.3	0.5	0.31	0.5
CONV-013C (890-900 ft.)	0.100	91%	0.041	0.437	>50.0%	<0.02	<0.4	0.36	0.5
CONV-013C (890-900 ft.)	0.075	90%	0.041	0.406	>50.0%	<0.02	<0.4	0.24	1.0
CONV-014C (450-460 ft.)	0.212	83%	0.041	0.250	65%	1.2	3.4	0.35	0.5
CONV-014C (450-460 ft.)	0.150	83%	0.041	0.250	>91.7%	<0.02	<2.4	0.30	0.5
CONV-014C (450-460 ft.)	0.100	82%	0.059	0.344	81%	0.6	3.1	0.35	0.5
CONV-014C (450-460 ft.)	0.075	82%	0.050	0.281	>73.0%	<0.1	<3.7	0.43	0.5
CONV-014C (480-490 ft.)	0.212	89%	0.059	0.562	32%	1.7	2.5	0.17	0.8
CONV-014C (480-490 ft.)	0.150	90%	0.050	0.531	44%	1	1.8	0.25	0.8
CONV-014C (480-490 ft.)	0.100	91%	0.050	0.531	47%	1	1.9	0.25	0.8
CONV-014C (480-490 ft.)	0.075	95%	0.031	0.594	46%	1.3	2.4	0.25	0.8
CONV-014C (750-760 ft.)	0.212	94%	0.091	1.437	89%	0.2	1.8	0.78	0.5
CONV-014C (750-760 ft.)	0.150	92%	0.119	1.437	76%	0.5	2.1	0.73	0.5
CONV-014C (750-760 ft.)	0.100	93%	0.100	1.500	>89.5%	<0.02	<1.9	0.73	0.6
CONV-014C (750-760 ft.)	0.075	94%	0.081	1.375	>85.0%	<0.3	<2	0.68	0.8
CONV-014C (825-835 ft.)	0.212	88%	0.069	0.594	>50.0%	<0.02	<0.4	0.25	0.5
CONV-014C (825-835 ft.)	0.150	90%	0.069	0.656	>50.0%	<0.02	<0.4	0.21	0.5
CONV-014C (825-835 ft.)	0.100	91%	0.059	0.656	>50.0%	<0.02	<0.4	0.21	0.5
CONV-014C (825-835 ft.)	0.075	94%	0.041	0.656	>60.0	<0.02	<0.5	0.20	0.6

Composite	Feed Size P_{80} (mm)	Au Recovered (%)	Au Tail (g/t)	Au Calculated Head (g/t)	Ag Recovered (%)	Ag Tail (g/t)	Ag Calculated. Head (g/t)	NaCN Consumed (kg/t)	Lime Added (kg/t)
CONV-019C (530-540 ft.)	0.212	92%	0.019	0.250	71%	1	3.4	0.98	1.0
CONV-019C (530-540 ft.)	0.150	92%	0.019	0.250	71%	1	3.5	0.97	1.0
CONV-019C (530-540 ft.)	0.100	95%	0.009	0.219	67%	1.3	3.9	1.04	1.0
CONV-019C (530-540 ft.)	0.075	94%	0.009	0.187	>93.3%	<0.02	<3	1.26	1.0
CONV-019C (730-740 ft.)	0.212	88%	0.081	0.687	17%	3.4	4.1	0.55	0.5
CONV-019C (730-740 ft.)	0.150	88%	0.081	0.656	14%	3.8	4.4	0.52	0.5
CONV-019C (730-740 ft.)	0.100	89%	0.069	0.656	15%	2.8	3.3	0.61	0.5
CONV-019C (730-740 ft.)	0.075	90%	0.069	0.687	13%	3.3	3.8	0.82	0.5
CONV-019C (1110-1120 ft.)	0.212	88%	0.150	1.281	>60.0%	<0.02	<0.5	0.27	0.5
CONV-019C (1110-1120 ft.)	0.150	91%	0.119	1.312	>71.4%	<0.02	<0.7	0.37	0.5
CONV-019C (1110-1120 ft.)	0.100	94%	0.081	1.312	20%	1.2	1.5	0.37	0.5
CONV-019C (1110-1120 ft.)	0.075	95%	0.069	1.375	50%	0.3	0.6	0.47	0.5
CONV-028C (70-80 ft.)	0.212	88%	0.091	0.781	81%	0.6	3.1	0.37	0.8
CONV-028C (70-80 ft.)	0.150	92%	0.059	0.781	53%	2.2	4.7	0.35	1.3
CONV-028C (70-80 ft.)	0.100	91%	0.069	0.750	66%	1.3	3.8	0.44	1.0
CONV-028C (70-80 ft.)	0.075	96%	0.031	0.687	67%	1.3	3.9	0.52	1.0
CONV-028C (155-165 ft.)	0.212	91%	0.159	1.781	89%	0.4	3.7	1.25	1.1
CONV-028C (155-165 ft.)	0.150	95%	0.091	1.625	92%	0.3	3.8	1.60	0.7
CONV-028C (155-165 ft.)	0.100	95%	0.081	1.625	90%	0.4	3.9	1.34	1.2
CONV-028C (155-165 ft.)	0.075	97%	0.059	1.687	90%	0.4	4.0	1.66	0.7
CONV-028C (540-550 ft.)	0.212	89%	0.019	0.187	52%	1.1	2.3	0.39	1.0
CONV-028C (540-550 ft.)	0.150	89%	0.031	0.281	56%	1.1	2.5	0.55	0.9
CONV-028C (540-550 ft.)	0.100	83%	0.031	0.187	48%	1.5	2.9	0.35	0.9
CONV-028C (540-550 ft.)	0.075	88%	0.019	0.156	64%	0.9	2.5	0.66	0.8
CONV-028C (905-915 ft.)	0.212	48%	0.150	0.281	59%	1.1	2.7	0.56	0.6

Composite	Feed Size P_{80} (mm)	Au Recovered (%)	Au Tail (g/t)	Au Calculated Head (g/t)	Ag Recovered (%)	Ag Tail (g/t)	Ag Calculated. Head (g/t)	NaCN Consumed (kg/t)	Lime Added (kg/t)
CONV-028C (905-915 ft.)	0.150	42%	0.141	0.250	57%	1.2	2.8	0.47	0.6
CONV-028C (905-915 ft.)	0.100	40%	0.150	0.250	34%	3.3	5.0	0.62	0.5
CONV-028C (905-915 ft.)	0.075	44%	0.150	0.281	60%	1.2	3.0	0.81	0.5
CONV-030C (750-760 ft.)	0.212	84%	0.069	0.437	67%	0.7	2.1	0.38	0.4
CONV-030C (750-760 ft.)	0.150	87%	0.059	0.469	68%	0.7	2.2	0.28	0.5
CONV-030C (750-760 ft.)	0.100	85%	0.069	0.469	89%	0.2	1.8	0.39	0.5
CONV-030C (750-760 ft.)	0.075	91%	0.041	0.406	>92.0%	<0.02	<2.5	0.27	0.5
CONV-030C (760-770 ft.)	0.212	81%	0.059	0.312	55%	2.7	6.0	0.64	0.5
CONV-030C (760-770 ft.)	0.150	86%	0.041	0.281	68%	1.5	4.7	0.73	0.5
CONV-030C (760-770 ft.)	0.100	87%	0.041	0.312	>93.8%	<0.02	<3.2	0.73	0.5
CONV-030C (760-770 ft.)	0.075	89%	0.031	0.281	74%	1	3.8	0.69	0.6
CONV-033C (585-595 ft.)	0.212	86%	0.019	0.125	31%	0.9	1.3	0.24	0.5
CONV-033C (585-595 ft.)	0.150	>93.3%	<0.009	<0.15	>66.7%	<0.02	<0.06	0.32	0.5
CONV-033C (585-595 ft.)	0.100	89%	0.019	0.187	>66.7%	<0.02	<0.06	0.32	0.5
CONV-033C (585-595 ft.)	0.075	95%	0.009	0.187	>66.7%	<0.02	<0.06	0.14	0.6
CONV-033C (600-610 ft.)	0.212	85%	0.131	0.875	>66.7%	<0.02	<0.06	0.20	0.5
CONV-033C (600-610 ft.)	0.150	94%	0.050	0.875	67%	0.2	0.6	0.21	0.5
CONV-033C (600-610 ft.)	0.100	94%	0.050	0.844	67%	0.2	0.6	0.07	0.5
CONV-033C (600-610 ft.)	0.075	94%	0.050	0.875	67%	0.2	0.6	0.26	0.4
CONV-034C (350-360 ft.)	0.212	87%	0.059	0.469	>66.7%	<0.3	<0.9	0.22	0.5
CONV-034C (350-360 ft.)	0.150	88%	0.050	0.437	>75.0%	<0.02	<0.8	0.21	0.5
CONV-034C (350-360 ft.)	0.100	90%	0.041	0.375	>75.0%	<0.02	<0.8	0.31	0.5
CONV-034C (350-360 ft.)	0.075	93%	0.031	0.437	>75.0%	<0.02	<0.8	0.39	0.5
CONV-034C (390-400 ft.)	0.212	87%	0.041	0.312	60%	0.8	2.0	0.32	0.5
CONV-034C (390-400 ft.)	0.150	87%	0.041	0.312	44%	1.5	2.7	0.33	0.5

Composite	Feed Size P_{80} (mm)	Au Recovered (%)	Au Tail (g/t)	Au Calculated Head (g/t)	Ag Recovered (%)	Ag Tail (g/t)	Ag Calculated. Head (g/t)	NaCN Consumed (kg/t)	Lime Added (kg/t)
CONV-034C (390-400 ft.)	0.100	91%	0.031	0.312	88%	0.2	1.6	0.35	0.5
CONV-034C (390-400 ft.)	0.075	88%	0.031	0.250	>86.7%	<0.02	<1.5	0.38	0.5

13.8 KCA HPGR and Variability Testing Program 2018

The most recent test program (KCA 2018) expanded on previous work to include HPGR crushing, agglomeration, and permeability. Column leach tests produced results consistent with prior studies for both North (PN, VN) and South (PS, VS) deposits: oxide composites recovered 67% to 85% gold, transition/mixed materials 49% to 65%, and sulphides 37% to 53%. Cyanide consumption was consistently higher in composites with higher copper content.

A total of 10 composites, five from the South zone and five from the North zone (PN, PS), were conventionally crushed and subjected to head analyses, head screen analyses with size-fraction assays, bottle roll leach tests, agglomeration tests, and column leach tests. Portions of each composite were also used for HPGR crushing tests, with the HPGR crushed material undergoing the same suite of tests. In addition to these 10 composites, 11 variability composites (VN, VS) were produced from the drill core samples received for head analyses, bottle roll leach tests, and agglomeration tests. Four composites were prepared for comminution testing, and five for optical sorting and head analyses. Head analysis for gold and silver for the North and South composites and variability composites is presented in Table 13-27 and Table 13-28, respectively. Head analyses for copper, carbon, and sulphur for the North and South composites and variability composites are presented in Table 13-29 and Table 13-30, respectively. Mercury levels for all composites were low, ranging from 0.02 mg/kg to 0.03 mg/kg.

As part of later reviews, composite PN4 was reclassified as oxide and is considered in the oxide redox zone for metallurgical test work conclusions.

Table 13-27: PS and PN Composites

KCA Sample No.	Description	Average Head Assay, Au (g/t)	Conv. Crushed Weighted Avg. Head Assay, Au (g/t)	HPGR Crushed Weighted Avg. Head Assay, Au (g/t)	Average Head Assay, Ag (g/t)	Conv. Crushed Weighted Avg. Head Assay, Ag (g/t)	HPGR Crushed Weighted Avg. Head Assay, Ag (g/t)
80501 B	PS1	0.732	0.663	0.663	1.2	1.02	1.34
80502 B	PS2	0.89	0.879	0.839	5.11	4.73	5.13
80503 B	PS3	0.504	0.484	0.528	1.61	1.29	1.64
80504 B	PS4	1.157	1.183	1.211	1.41	1.32	1.54
80505 B	PS5	0.667	0.628	0.631	1.82	1.36	1.94
80506 B	PN1	0.855	1.323	1.109	5.31	4.6	5.48
80507 B	PN2	1.353	1.253	1.425	6.7	5.7	6.4
80508 B	PN3	0.535	0.504	0.557	4.39	3.93	4.51
80509 B	PN4	0.993	0.92	1.05	5.21	4.79	5.39
80510 B	PN5	0.989	1.058	1.199	4.49	3.81	4.37

Table 13-28: VS and VN Composites

Comp.	Comp. Objective	Au Avg. (g/t)	Ag Avg. (g/t)
VS1	Base	1.019	2.91
VS2	High Au	2.04	3.6
VS3	Mid Cu	0.362	3.39
VS4	Low Au	0.591	6.51
VS5	Sulphide	1.178	1.82
VS6	Base	0.83	2.71
VS7	High Cu	0.537	7.3
VN1	High Cu	1.572	7.61
VN2	Sulphide	0.974	2.19
VN3	Low Au	0.408	3.39
VN4	Mid Cu	0.792	2.61

Table 13-29: PS and PN Composites Head Analyses – Copper, Carbon, and Sulphur

KCA Sample No.	Description	Total Carbon (%)	Organic Carbon (%)	Inorganic Carbon (%)	Total Sulphur (%)	Sulphide Sulphur (%)	Sulphate Sulphur (%)	Total Copper (mg/kg)	Cyanide Soluble Copper (mg/kg)	Cyanide Soluble Copper (%)
80501 B	PS1	0.13	0.07	0.06	0.02	0.01	0.01	91	29	0.32
80502 B	PS2	0.19	0.07	0.12	0.24	0.08	0.16	933	203	0.22
80503 B	PS3	0.09	0.03	0.06	0.01	0.01	0	254	26	0.1
80504 B	PS4	0.25	0.05	0.2	0.02	0.01	0.01	76	15	0.2
80505 B	PS5	0.18	0.13	0.05	2.74	2.39	0.35	1020	253	0.25
80506 B	PN1	0.57	0.07	0.51	0.27	0.12	0.16	811	304	0.37
80507 B	PN2	0.86	0.08	0.78	0.26	0.11	0.16	1470	652	0.44
80508 B	PN3	0.44	0.07	0.37	0.13	0.02	0.11	500	246	0.49
80509 B	PN4	0.41	0.04	0.36	0.04	0.01	0.04	766	181	0.24
80510 B	PN5	0.36	0.07	0.29	0.25	0.13	0.13	1260	659	0.52

Table 13-30: Variability Composites Head Analyses – Copper, Carbon, and Sulphur

Description	Total Carbon (%)	Organic Carbon (%)	Inorganic Carbon (%)	Total Sulphur (%)	Sulphide Sulphur (%)	Sulphate Sulphur (%)	Total Copper (mg/kg)	Cyanide Soluble Copper (mg/kg)	Cyanide Soluble Copper (%)	Description
80528 B	VS1	0.15	0.02	0.13	0.01	<0.01	0.01	265	51.5	0.19
80529 B	VS2	0.11	0.02	0.08	0.02	<0.01	0.01	837	55	0.07
80530 B	VS3	0.04	0.03	0	0.03	<0.01	0.03	396	39	0.1
80531 B	VS4	0.03	0.02	0.01	0.03	<0.01	0.03	860	123.7	0.14
80532 B	VS5	0.08	0.05	0.02	5.23	4.79	0.44	621	246.5	0.4
80533 B	VS6	0.06	0.05	0.01	0.12	<0.01	0.12	571	50.3	0.09
80534 B	VS7	0.22	0.04	0.18	0.04	0.01	0.04	1120	207.5	0.19
80535 B	VN1	0.9	0.05	0.85	0.01	<0.01	0.01	2800	291.5	0.1
80536 B	VN2	0.22	0.05	0.17	0.2	0.08	0.12	1300	389	0.3
80537 B	VN3	0.23	0.02	0.21	0.03	<0.01	0.03	506	95.5	0.19
80538 B	VN4	0.18	0.02	0.17	0.13	0.05	0.08	942	336.5	0.36

13.8.1 Comminution Test Work (KCA 2018)

A portion of the head material was submitted to Hazen Research, Inc. in Golden, Colorado, for comminution testing. Test work was completed to provide Bond Abrasion and Bond Crusher Work (CWi) indices, as well as an abrasion index, for the sample. This resulted in an average abrasion index of 0.675 g and an average ball impact work index of 8 kWh/t. The test work results are presented in Table 13-31. The results indicate that the material is very abrasive.

Table 13-31: KCA 2018: Comminution Results Summary

KCA Sample No.	Hole	Rock Type	Ai, (g)	CWi, (kWh/t)
80539 A	CNR-MET17-004 & 5	SsBx	SsBx	SsBx
CNR-MET17-004 & 5	CNR-MET17-003 & 5	Ss/Qm	Ss/Qm	Ss/Qm
CNR-MET17-003 & 5	CNR-MET17-001 & 2	Ss	Ss	Ss
CNR-MET17-001 & 2	CNR-MET17-006 & 7	Ss/Qm	Ss/Qm	Ss/Qm

13.8.2 Bottle Roll Leach Test Work (KCA 2018)

Bottle roll tests were conducted on 1,000 g portions of the North and South composites and variability composites. The bottle rolls were completed for each composite at 80% passing 1.7 mm and 80% passing 0.150 mm. The samples were leached for 144 hours for the 1.7 mm samples, and 96 hours for the 0.150 mm samples. Results for the bottle roll leach tests are presented in Table 13-32 for the North and South Composites and Table 13-33 for the variability composites.

The bottle roll tests show significant recovery improvements at 0.150 mm compared to 1.7 mm, consistent with previous observations.

Table 13-32: PS / PN Composites Summary of Bottle Roll Leach Test Work (KCA 2018)

KCA Sample No.	Description	Target P_{80}/P_{100} Size (mm)	Calculated Head, Au (g/t)	Calculated Head, Ag (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (hr)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
80501 B	PS1	1.7	0.584	1.24	75%	20%	144	0.24	1
		0.15	0.613	1.2	95%	42%	96	0.21	1.25
80502 B	PS2	1.7	0.749	5.22	75%	25%	144	0.56	1.5
		0.15	0.799	5.2	94%	52%	96	0.79	2
80503 B	PS3	1.7	0.453	1.49	64%	20%	144	0.17	0.75
		0.15	0.476	1.57	92%	50%	96	0.21	1
80504 B	PS4	1.7	1.101	1.35	54%	27%	144	0.21	0.75
		0.15	0.954	1.34	95%	41%	96	0.24	1
80505 B	PS5	1.7	0.643	1.76	60%	38%	144	0.65	1
		0.15	0.59	1.75	88%	43%	96	3.86	1.75
80506 B	PN1	1.7	0.988	5.26	65%	28%	144	0.5	0.75
		0.15	1.113	5.28	95%	51%	96	1.21	0.75
80507 B	PN2	1.7	1.246	6.28	69%	35%	144	1.22	0.75
		0.15	1.525	6.68	95%	54%	96	2.06	0.75
80508 B	PN3	1.7	0.517	4.31	71%	30%	144	0.51	0.5
		0.15	0.432	4.28	89%	51%	96	0.92	0.75
80509 B	PN4	1.7	0.848	5.13	64%	20%	144	0.37	0.75
		0.15	0.795	5.21	93%	37%	96	0.56	1
80510 B	PN5	1.7	1.083	4.56	52%	36%	144	0.63	0.5
		0.15	0.996	4.44	95%	73%	96	2.34	0.5
Avg.	PS	1.7	0.706	2.21	66%	26%	144	0.37	1

KCA Sample No.	Description	Target P_{80}/P_{100} Size (mm)	Calculated Head, Au (g/t)	Calculated Head, Ag (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (hr)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
Avg.	PN	1.7	0.936	5.11	64%	30%	144	0.65	0.65
Avg.	O'all	1.7	0.821	3.66	65%	28%	144	0.51	0.83
Avg.	PS	0.15	0.687	2.21	93%	46%	96	1.06	1.4
Avg.	PN	0.15	0.972	5.18	93%	53%	96	1.42	0.75
Avg.	O'all	0.15	0.829	3.7	93%	49%	96	1.24	1.08

Table 13-33: Summary of Variability Bottle Roll Leach Test Work (KCA 2018)

KCA Sample #	Description	Target P_{80} (mm)	Calc. Head Grade, Au (g/t)	Calc. Head Grade, Ag (g/t)	Cyanide Soluble Copper (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (hr)	Consumption NaCN kg/t)	Addition $Ca(OH)_2$ kg/t)
80528 B	VS1	1.7	0.98	2.87	51	49%	16%	144	0.2	0.5
80528 B	VS1	0.15	1.06	2.87	51	93%	40%	96	0.23	0.5
80529 B	VS2	1.7	1.81	3.5	55	62%	11%	144	0.25	1.25
80529 B	VS2	0.15	2.19	3.63	55	92%	31%	96	0.31	1.25
80530 B	VS3	1.7	0.34	3.32	39	62%	25%	144	0.23	1.25
80530 B	VS3	0.15	0.35	3.5	39	92%	48%	96	0.35	1.5
80531 B	VS4	1.7	0.4	6.45	124	63%	15%	144	0.26	0.75
80531 B	VS4	0.15	0.44	6.69	124	95%	51%	96	0.42	0.75
80532 B	VS5	1.7	1.25	1.79	246	45%	21%	144	2.05	3.5
80532 B	VS5	0.15	1.19	1.87	246	90%	25%	96	6.7	4
80533 A	VS6	1.7	0.78	2.65	50	72%	17%	144	0.33	1.5
80533 A	VS6	0.15	0.85	2.55	50	96%	53%	96	0.41	1.75
80534 B	VS7	1.7	0.66	7.17	207	65%	23%	144	0.58	1
80534 B	VS7	0.15	0.52	7.19	207	94%	54%	96	0.61	1.25
80535 B	VN1	1.7	1.64	7.6	291	71%	26%	144	0.49	1
80535 B	VN1	0.15	1.74	7.7	291	99%	61%	96	0.85	1.25
80536 B	VN2	1.7	1.35	2.49	389	38%	60%	144	0.38	0.5
80536 B	VN2	0.15	1.04	2.96	389	91%	86%	96	0.83	0.5
80537 B	VN3	1.7	0.4	3.3	95	58%	18%	144	0.3	0.5
80537 B	VN3	0.15	0.41	3.34	95	86%	37%	96	0.39	0.5
80538 B	VN4	1.7	0.85	2.54	336	58%	18%	144	0.31	0.5
80538 B	VN4	0.15	0.75	2.58	336	84%	30%	96	0.68	0.5

KCA Sample #	Description	Target P_{80} (mm)	Calc. Head Grade, Au (g/t)	Calc. Head Grade, Ag (g/t)	Cyanide Soluble Copper (g/t)	Au Extracted (%)	Ag Extracted (%)	Leach Time (hr)	Consumption NaCN kg/t)	Addition $Ca(OH)_2$ kg/t)
Avg.	VS	1.7	0.89	3.96	110	60%	18%	144	0.56	1.39
Avg.	VN	1.7	1.06	3.99	278	56%	31%	144	0.37	0.62
Avg.	Overall	1.7	0.95	3.97	171	58%	23%	144	0.49	1.11
Avg.	VS	0.15	0.94	4.04	110	93%	43%	96	1.29	1.57
Avg.	VN	0.15	0.99	4.14	278	90%	54%	96	0.69	0.69
Avg.	Overall	0.15	0.96	4.08	171	92%	47%	96	1.07	1.25

13.8.3 Column Leach Test Work (KCA 2018)

Column leach tests were conducted using conventionally crushed and HPGR-crushed material for the North and South composites. Prior to the column leach tests, preliminary agglomeration work was conducted with samples at 0 kg, 2 kg, 4 kg, and 8 kg of cement per tonne of dry material. Based on the preliminary agglomeration tests, the HPGR crushed samples were agglomerated with 4 kg/t cement, and the conventional crushed columns received no cement. Results from the column leach tests are presented it Table 13-34. The tail screen analysis is presented in Figure 13-5.

The column tests showed mostly similar gold recoveries between the HPGR and conventional crushed columns, with the HPGR tests averaging 3% higher recoveries. Silver recoveries showed an opposite trend with the conventional columns averaging 1% higher.

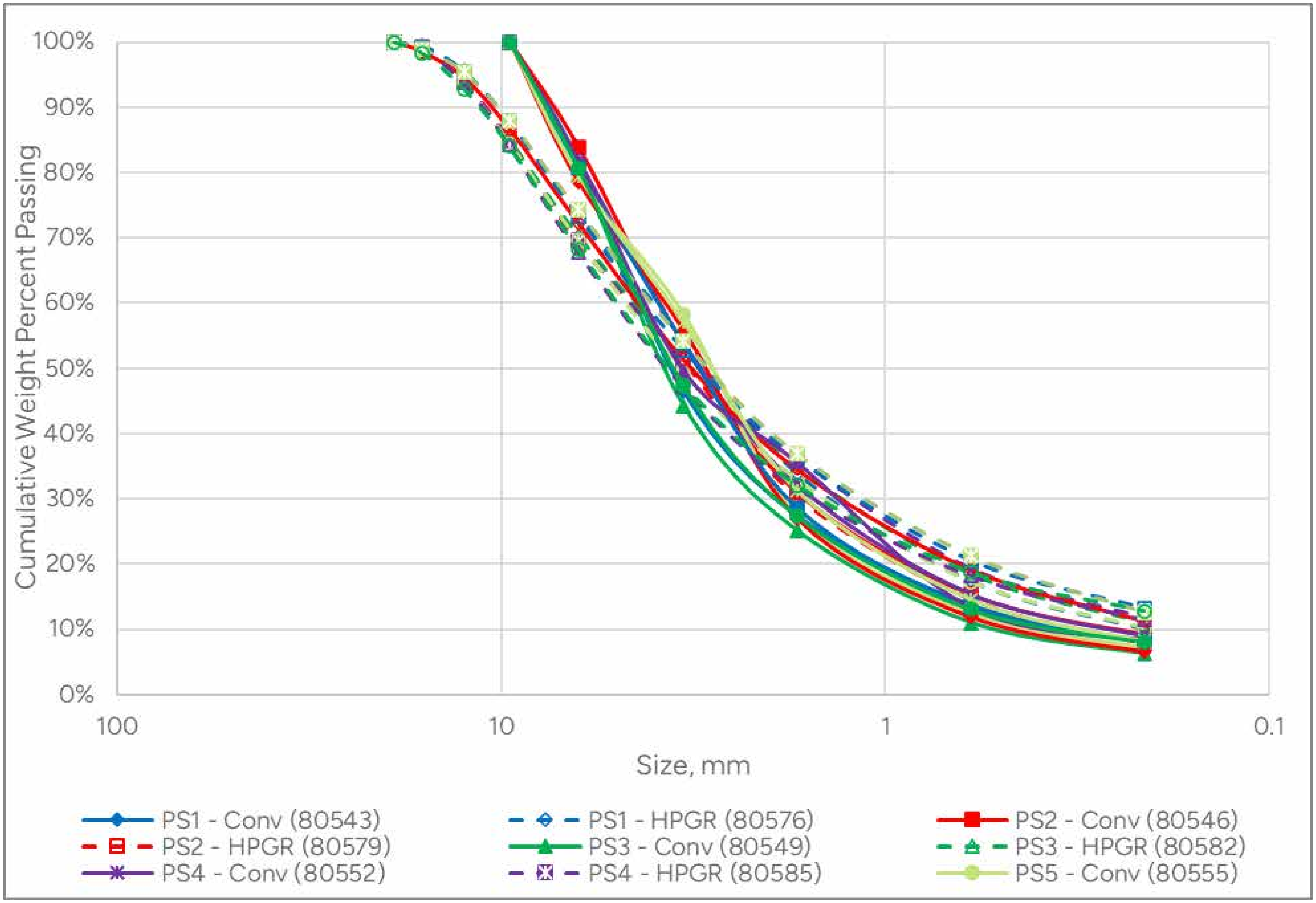
Generally, recovery benefits from HPGR crushing are due to the generation of additional fines material, as observed in the tailings screen analysis; however, in most composites, only 5% to 10% more of the minus 1 mm material was generated. Additional work is required to determine whether the slight recovery benefit is sufficient to cover the additional agglomeration requirements for the HPGR-crushed material.

Table 13-34: PS and PN Composite Summary of Column Leach Test Work (KCA 2018)

KCA Sample No.	Description	Crush Size (mm)	Redox	Head Cu (mg/kg)	Calculated Head (g Au/t)	Calculated Head (g Ag/t)	Extract % Au	Extract % Ag	Calc. Copper Extraction (%)	Days of Leach	Cons. NaCN (kg/t)	Add. Hydrated Lime (kg/t)	Add. Cement (kg/t)
80501 B	PS1	9.5	Ox	91	0.614	1.06	81%	24%	30%	133	1.74	1.01	0
80511 C	PS1	HPGR	Ox	91	0.609	1.28	81%	21%	27%	132	2.09	0	4.03
80502 B	PS2	9.5	Ox	933	0.798	5.11	84%	28%	33%	133	2.86	1.52	0
80512 C	PS2	HPGR	Ox	933	0.78	5.09	86%	29%	27%	132	2.38	0	4.11
80503 B	PS3	9.5	Ox	254	0.487	1.31	74%	22%	11%	133	1.96	0.75	0
80513 C	PS3	HPGR	Ox	254	0.489	1.35	74%	17%	11%	132	1.86	1.01	0
80504 B	PS4	9.5	Ox	76	1.097	1.54	69%	33%	37%	133	2.04	0.76	0
80514 C	PS4	HPGR	Ox	76	1.058	1.32	75%	28%	30%	132	1.55	1.02	0
80505 B	PS5	9.5	Sul	1020	0.685	1.9	56%	47%	16%	142	2.26	1	0
80515 C	PS5	HPGR	Sul	1020	0.584	2.03	57%	40%	22%	142	2.47	0	4.28
80506 B	PN1	9.5	Tr	811	1.181	5.01	62%	30%	18%	142	2.26	0.75	0
80516 C	PN1	HPGR	Tr	811	1.172	5.51	63%	31%	25%	142	2.3	0.95	0
80507 B	PN2	9.5	Tr	1470	1.343	5.67	68%	34%	31%	143	2.9	0.76	0
80517 C	PN2	HPGR	Tr	1470	1.355	6.17	68%	34%	23%	142	2.9	1.03	0
80508 B	PN3	9.5	Tr	500	0.573	3.7	56%	34%	14%	143	2	0.51	0

KCA Sample No.	Description	Crush Size (mm)	Redox	Head Cu (mg/kg)	Calculated Head (g Au/t)	Calculated Head (g Ag/t)	Extract % Au	Extract % Ag	Calc. Copper Extraction (%)	Days of Leach	Cons. NaCN (kg/t)	Add. Hydrated Lime (kg/t)	Add. Cement (kg/t)
80518 C	PN3	HPGR	Tr	500	0.46	4.25	55%	28%	23%	142	2.11	0	3.95
80509 B	PN4	9.5	Tr	766	0.968	4.42	60%	21%	12%	132	2.4	0.76	0
80519 C	PN4	HPGR	Tr	766	0.901	4.73	67%	20%	9%	133	2.08	0	4.08
80510 B	PN5	9.5	Sul	1260	1.148	4.47	43%	34%	25%	132	2.36	0.51	0
80520 C	PN5	HPGR	Sul	1260	1.15	4.62	50%	35%	19%	133	1.88	0.77	0
Avg.	PS	9.5	--	475	0.736	2.18	73%	31%	25%	135	2.17	1.01	0
Avg.	PN	9.5	--	961	1.043	4.65	58%	31%	20%	138	2.38	0.66	0
Avg.	Overall	9.5	--	718	0.889	3.42	65%	31%	23%	137	2.28	0.83	0
Avg.	PS	HPGR	--	475	0.704	2.21	75%	27%	23%	134	2.07	0.41	2.48
Avg.	PN	HPGR	--	961	1.008	5.06	61%	30%	20%	138	2.25	0.55	1.61
Avg.	Overall	HPGR	--	718	0.856	3.64	68%	28%	22%	136	2.16	0.48	2.05

Figure 13-5: Tails Screen Analysis (KCA 2018)



13.8.3.1 Compacted Permeability Test Work (KCA 2018)

Compacted permeability tests on column tails were performed to evaluate cement addition requirements under simulated load conditions. The compacted permeability tests were evaluated at equivalent heap heights of 20 m, 40 m, 60 m, 80 m, 100 m, 120 m, and 140 m with 4 kg/t cement addition for HPGR material and 0 kg/t for conventionally crushed material. Compacted permeability test results are presented in Table 13-35.

Based on the results, the HPGR columns had a higher failure rate at heights above 100 m compared to the unagglomerated conventionally crushed columns. Cement agglomeration would likely be required for conventionally crushed material at heap heights greater than 140 m.

Table 13-35: PS and PN Composite Summary of Column Leach Test Work (KCA 2018)

KCA Sample No.	Sample Description	Test Phase	Cement Added (kg/t)	Effective Height (m)	Flow Rate (LpHr/m^2)	Flow Result	Saturated Permeability (cm/sec)	Incremental Slump (%)	Cumulative Slump (%)	Slump Result	Overall Pass/Fail
80543	PS1-Conv.	Primary	0	20	6,823	Pass	0.190	0%	0%	Pass	Pass
		Stage Load		40	6,370	Pass	0.180	2%	2%	Pass	Pass
		Stage Load		60	5,811	Pass	0.160	3%	5%	Pass	Pass
		Stage Load		80	4,672	Pass	0.130	2%	7%	Pass	Pass
		Stage Load		100	3,724	Pass	0.100	0%	7%	Pass	Pass
		Stage Load		120	2,801	Pass	0.078	2%	9%	Pass	Pass
		Stage Load		140	2,020	Pass	0.056	2%	11%	Fail	Fail
80576	PS1-HPGR	Primary	4	20	6,859	Pass	0.190	0%	0%	Pass	Pass
		Stage Load		40	6,126	Pass	0.170	3%	3%	Pass	Pass
		Stage Load		60	5,414	Pass	0.150	3%	6%	Pass	Pass
		Stage Load		80	4,585	Pass	0.130	2%	8%	Pass	Pass
		Stage Load		100	4,024	Pass	0.110	1%	9%	Pass	Pass
		Stage Load		120	3,217	Pass	0.089	2%	11%	Fail	Fail
		Stage Load		140	2,612	Pass	0.073	1%	12%	Fail	Fail
80546	PS2-Conv.	Primary	0	20	6,283	Pass	0.170	0%	0%	Pass	Pass
		Stage Load		40	4,740	Pass	0.130	3%	3%	Pass	Pass
		Stage Load		60	3,658	Pass	0.100	2%	5%	Pass	Pass
		Stage Load		80	2,729	Pass	0.076	2%	7%	Pass	Pass
		Stage Load		100	2,000	Pass	0.056	2%	9%	Pass	Pass
		Stage Load		120	1,454	Pass	0.040	2%	11%	Fail	Fail
		Stage Load		140	1,060	Pass	0.029	1%	12%	Fail	Fail

KCA Sample No.	Sample Description	Test Phase	Cement Added (kg/t)	Effective Height (m)	Flow Rate (LpHr/m²)	Flow Result	Saturated Permeability (cm/sec)	Incremental Slump (%)	Cumulative Slump (%)	Slump Result	Overall Pass/Fail
80579	PS2-HPGR	Primary	4	20	2,444	Pass	0.068	1%	1%	Pass	Pass
		Stage Load		40	702	Pass	0.020	4%	5%	Pass	Pass
		Stage Load		60	461	Pass	0.013	3%	8%	Pass	Pass
		Stage Load		80	322	Pass	0.009	2%	10%	Pass	Pass
		Stage Load		100	233	Pass	0.007	1%	11%	Fail	Fail
		Stage Load		120	178	Pass	0.005	1%	12%	Fail	Fail
		Stage Load		140	134	Pass	0.004	1%	13%	Fail	Fail
80555	PS5-Conv.	Primary	0	20	7,440	Pass	0.210	0%	0%	Pass	Pass
		Stage Load		40	7,312	Pass	0.200	2%	2%	Pass	Pass
		Stage Load		60	7,223	Pass	0.200	2%	4%	Pass	Pass
		Stage Load		80	7,097	Pass	0.200	1%	5%	Pass	Pass
		Stage Load		100	7,025	Pass	0.200	2%	7%	Pass	Pass
		Stage Load		120	6,946	Pass	0.190	1%	8%	Pass	Pass
		Stage Load		140	6,849	Pass	0.190	1%	9%	Pass	Pass
80588	PS5-HPGR	Primary	4	20	5,329	Pass	0.150	1%	1%	Pass	Pass
		Stage Load		40	3,939	Pass	0.110	3%	4%	Pass	Pass
		Stage Load		60	3,173	Pass	0.088	1%	5%	Pass	Pass
		Stage Load		80	2,677	Pass	0.074	2%	7%	Pass	Pass
		Stage Load		100	2,248	Pass	0.062	1%	8%	Pass	Pass
		Stage Load		120	1,880	Pass	0.052	1%	9%	Pass	Pass
		Stage Load		140	1,530	Pass	0.043	0%	9%	Pass	Pass

KCA Sample No.	Sample Description	Test Phase	Cement Added (kg/t)	Effective Height (m)	Flow Rate (LpHr/m²)	Flow Result	Saturated Permeability (cm/sec)	Incremental Slump (%)	Cumulative Slump (%)	Slump Result	Overall Pass/Fail
80562	PN2-Conv.	Primary	0	20	7,177	Pass	0.200	0%	0%	Pass	Pass
		Stage Load		40	6,892	Pass	0.190	3%	3%	Pass	Pass
		Stage Load		60	6,688	Pass	0.190	2%	5%	Pass	Pass
		Stage Load		80	6,324	Pass	0.180	2%	7%	Pass	Pass
		Stage Load		100	6,022	Pass	0.170	1%	8%	Pass	Pass
		Stage Load		120	5,558	Pass	0.150	2%	10%	Pass	Pass
		Stage Load		140	5,114	Pass	0.140	1%	11%	Fail	Fail
81204	PN2-HPGR	Primary	0	20	1,268	Pass	0.035	0%	0%	Pass	Pass
		Stage Load		40	873	Pass	0.024	2%	2%	Pass	Pass
		Stage Load		60	1,082	Pass	0.030	2%	4%	Pass	Pass
		Stage Load		80	942	Pass	0.026	1%	5%	Pass	Pass
		Stage Load		100	700	Pass	0.019	1%	6%	Pass	Pass
		Stage Load		120	647	Pass	0.018	1%	7%	Pass	Pass
		Stage Load		140	601	Pass	0.017	1%	8%	Pass	Pass
80565	PN3-Conv.	Primary	0	20	6,983	Pass	0.190	0%	0%	Pass	Pass
		Stage Load		40	4,433	Pass	0.120	3%	3%	Pass	Pass
		Stage Load		60	4,204	Pass	0.120	2%	5%	Pass	Pass
		Stage Load		80	3,621	Pass	0.100	1%	6%	Pass	Pass
		Stage Load		100	3,271	Pass	0.091	2%	8%	Pass	Pass
		Stage Load		120	2,962	Pass	0.082	1%	9%	Pass	Pass
		Stage Load		140	2,557	Pass	0.071	1%	10%	Pass	Pass

KCA Sample No.	Sample Description	Test Phase	Cement Added (kg/t)	Effective Height (m)	Flow Rate (LpHr/m²)	Flow Result	Saturated Permeability (cm/sec)	Incremental Slump (%)	Cumulative Slump (%)	Slump Result	Overall Pass/Fail
81207	PN3-HPGR	Primary	4	20	6,378	Pass	0.180	1%	1%	Pass	Pass
		Stage Load		40	4,119	Pass	0.110	3%	4%	Pass	Pass
		Stage Load		60	3,412	Pass	0.095	3%	7%	Pass	Pass
		Stage Load		80	2,627	Pass	0.073	1%	8%	Pass	Pass
		Stage Load		100	2,305	Pass	0.064	2%	10%	Pass	Pass
		Stage Load		120	1,978	Pass	0.055	1%	11%	Fail	Fail
		Stage Load		140	1,554	Pass	0.043	0%	11%	Fail	Fail
80571	PN5-Conv.	Primary	0	20	7,075	Pass	0.200	1%	1%	Pass	Pass
		Stage Load		40	7,240	Pass	0.200	2%	3%	Pass	Pass
		Stage Load		60	7,033	Pass	0.200	2%	5%	Pass	Pass
		Stage Load		80	7,023	Pass	0.200	2%	7%	Pass	Pass
		Stage Load		100	2,890	Pass	0.080	1%	8%	Pass	Pass
		Stage Load		120	2,960	Pass	0.082	1%	9%	Pass	Pass
		Stage Load		140	2,464	Pass	0.068	2%	11%	Fail	Fail
81213	PN5-HPGR	Primary	0	20	6,141	Pass	0.170	2%	2%	Pass	Pass
		Stage Load		40	5,354	Pass	0.150	2%	4%	Pass	Pass
		Stage Load		60	4,766	Pass	0.130	2%	6%	Pass	Pass
		Stage Load		80	4,420	Pass	0.120	1%	7%	Pass	Pass
		Stage Load		100	3,902	Pass	0.110	2%	9%	Pass	Pass
		Stage Load		120	3,417	Pass	0.095	1%	10%	Pass	Pass
		Stage Load		140	2,964	Pass	0.082	1%	11%	Fail	Fail

13.9 Metallurgical Test Work Interpretations

Metallurgical test work to date includes almost 1,000 bottle roll leach tests and 75 column leach tests, as well as gravity concentration and flotation test work. The test work indicates that recoveries of gold and silver from the Converse deposits are sensitive to several parameters, with particle size having the greatest impact on recovery. Based on the information available a stand-alone heap leach with three-stage crushing with HPGR, cement agglomeration and an ADR recovery plant is recommended for the Project moving forward.

A summary of recommended heap leach parameters is presented in Table 13-36.

Table 13-36: Heap Leach Design Criteria

Description	Heap Leach with HPGR, P_{80} 6.3 mm, 120 Day Leach Cycle			
	Au (%)	Ag (%)	NaCN Consumed (kg/t)	Lime Added (kg/t)
North Oxide	60%	27%	0.6	0.5
North Trans.	69%	36%	0.84	0.5
North Sulphide	58%	42%	0.68	0.4
South Oxide	79%	25%	0.64	1
South Trans.	76%	30%	0.42	0.4
South Sulphide	64%	39%	0.76	0.6

Lime addition is shown for reference. In the QP's experience, HPGR crushing almost always requires cement agglomeration with a recommended cement addition of 6 kg/t. Lime will not be required based on the cement addition as discussed in Section 13.9.3.2.

Gold and silver recoveries are estimated based on the target crush size of 80% passing 6.3 mm, with a recovery correction for HPGR crushing and includes a 2% recovery discount for gold and a 4% recovery discount for silver.

Additional HPGR test work is recommended as part of future studies to better define the potential recovery benefits. The current test work indicates minor improvements in recovery with HPGR crushing.

13.9.1 Heap Recovery Estimates

To determine recovery estimates for the North and South Converse deposits, several parameters were analyzed, including material crush size, gold head grade, copper head grade, sulphide content, and their effects on recovery. Recovery Analyses are presented in Figure 13-6 through Figure 13-15.

From the analysis, gold and silver recoveries are most sensitive to crush size (Figure 13-6 through Figure 13-9) with only minor impacts from cyanide soluble copper content and total copper content (Figure 13-11 and Figure 13-12) and no material impacts from gold grade (Figure 13-10) or percent sulphur (Figure 13-13). To minimize the effect of particle size, all other recovery parameters were evaluated using column tests at approximately 9.5 mm crush size unless otherwise noted.

Cyanide concentration for the column leach was only varied in MLI 2009's test program on oxide materials for both the North and South zones. The North material did not present any impact from the increasing cyanide concentration; however, the South samples had a minor increase in recovery for one of the samples treated with 1000 ppm of NaCN (Figure 13-14).

There does appear to be a correlation between cyanide consumption and gold recovery, with the cyanide consumption increasing with higher gold recovery (Figure 13-15). It is assumed that this is a result of other cyanide consumers (such as copper) being liberated along with gold and should be further evaluated.

Figure 13-6: Column Material Particle Size (P_{80}) versus Gold Recovery

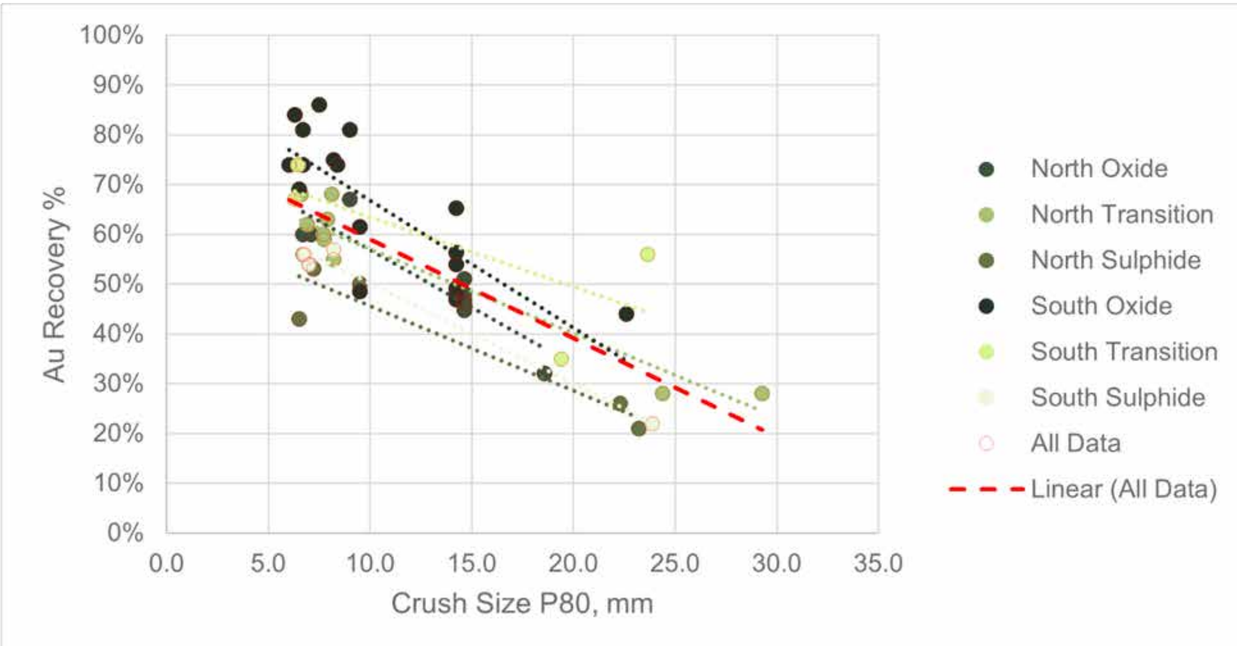


Figure 13-7: Column Material Particle Size (P_{80}) versus Silver Recovery

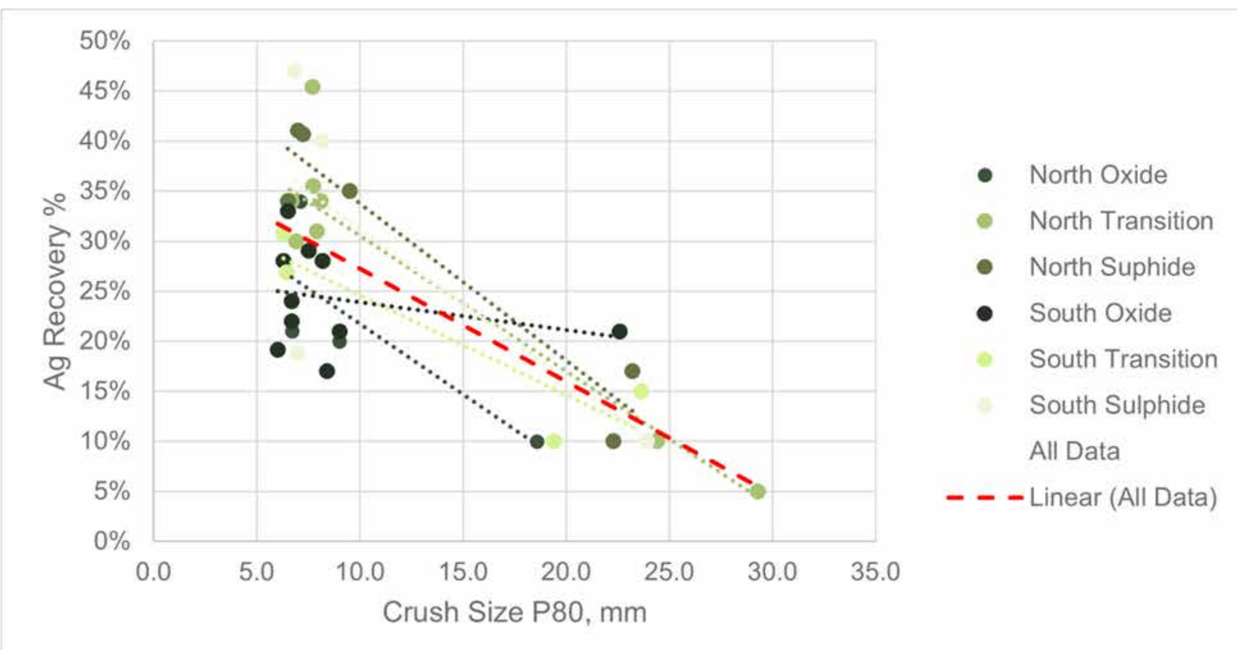


Figure 13-8: Column % Passing 1.7 mm versus Gold Recovery

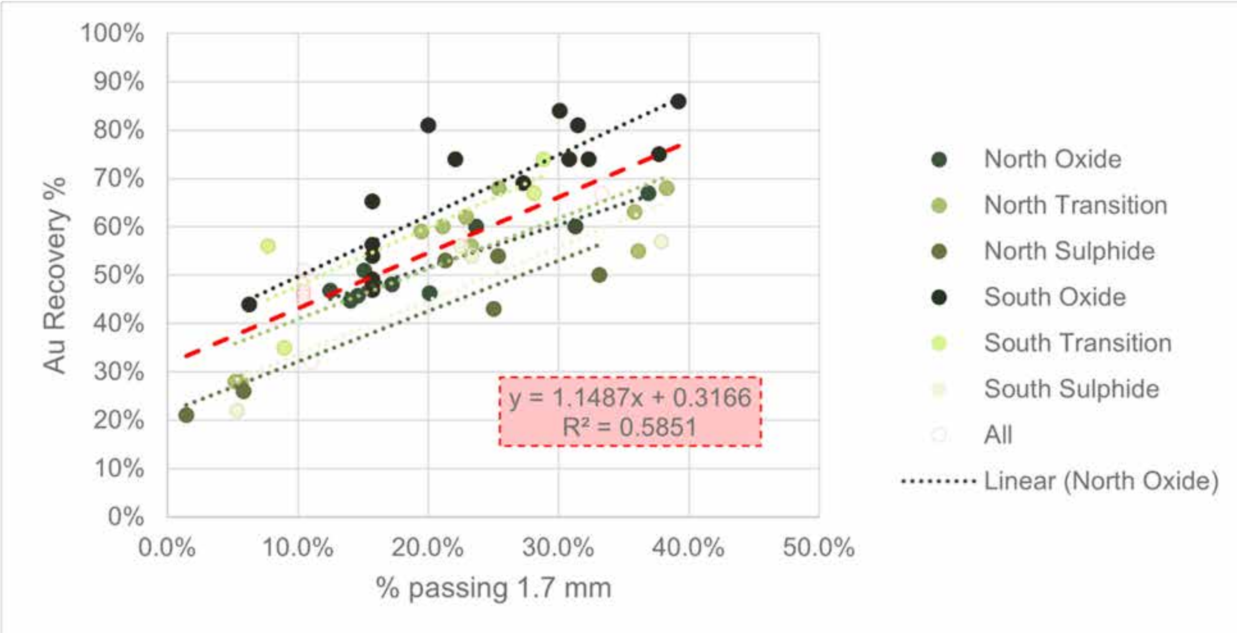


Figure 13-9: Column % Passing 1.7 mm versus Silver Recovery

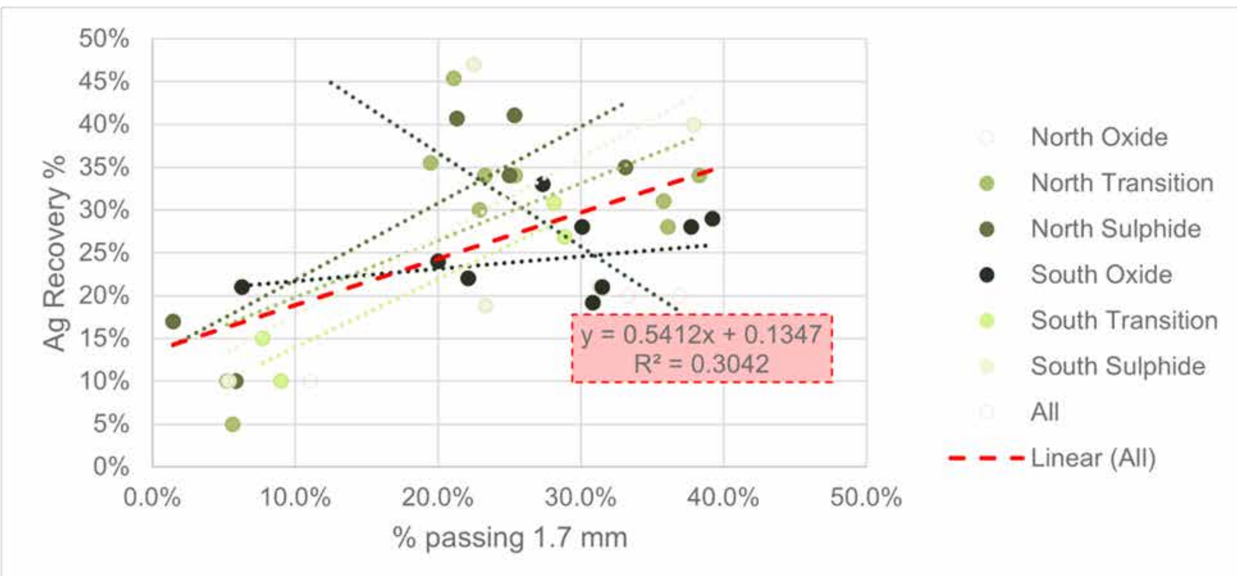


Figure 13-10: Gold Grade versus Gold Recovery @ +/- 9.5 mm

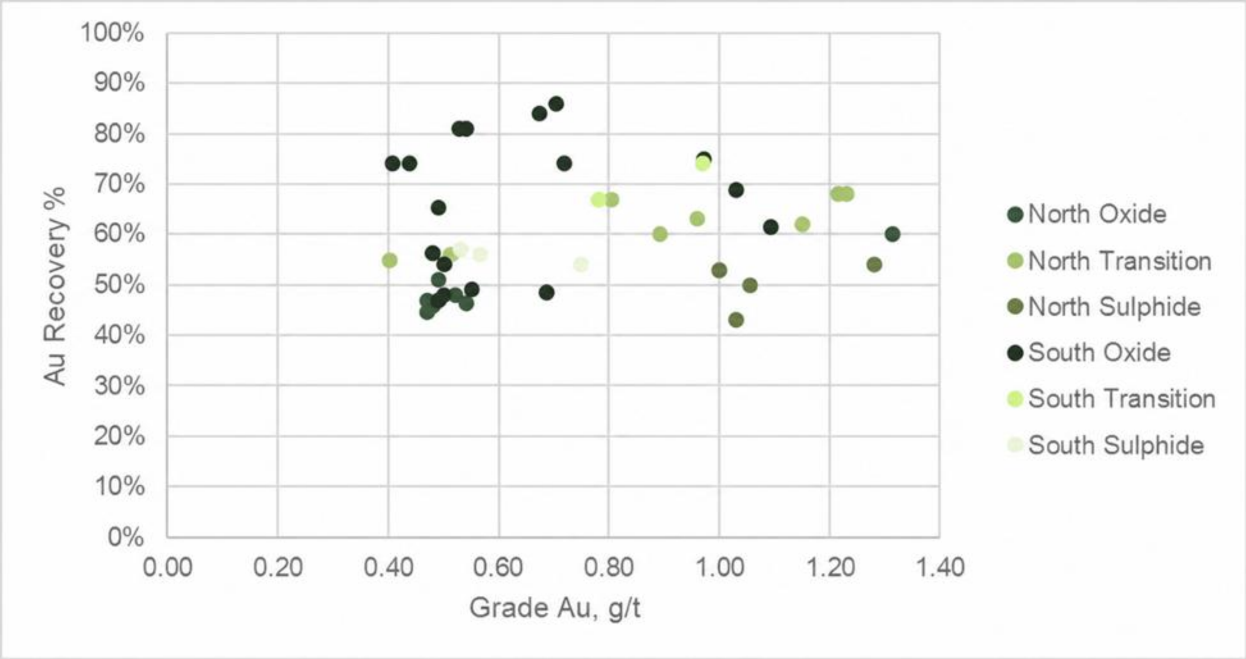


Figure 13-11: Cyanide Soluble Copper versus Gold Recovery @ +/- 9.5 mm

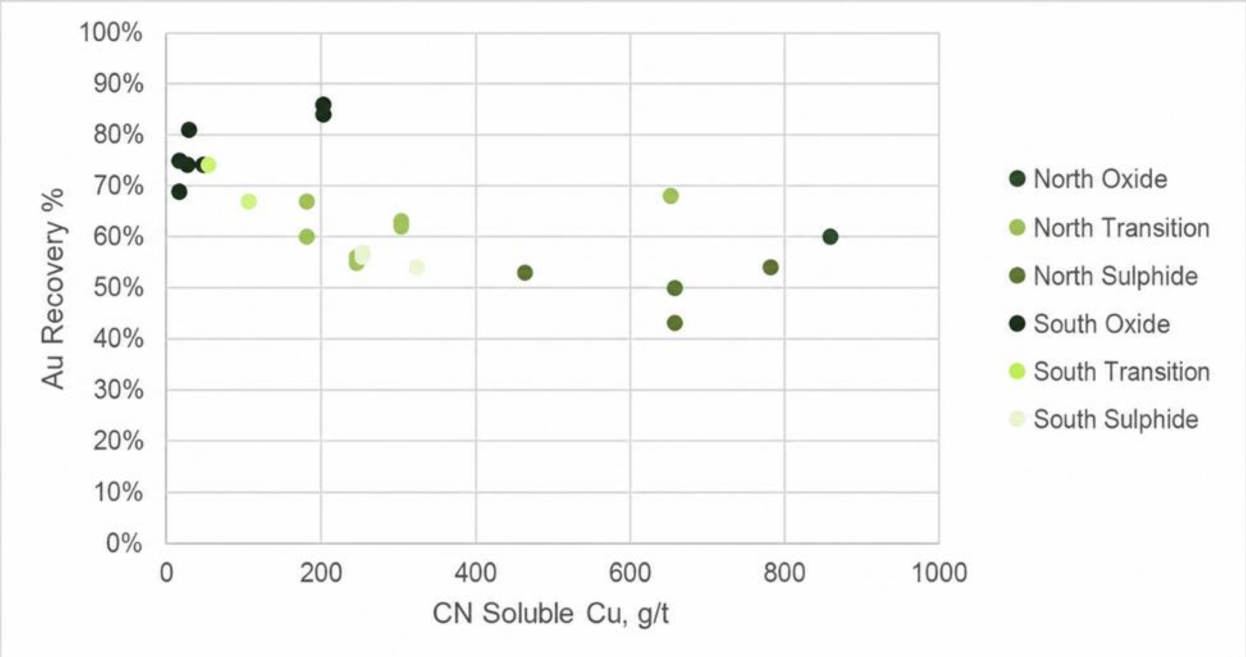


Figure 13-12: Copper Head Grade versus Gold Recovery @ +/- 6.3 mm

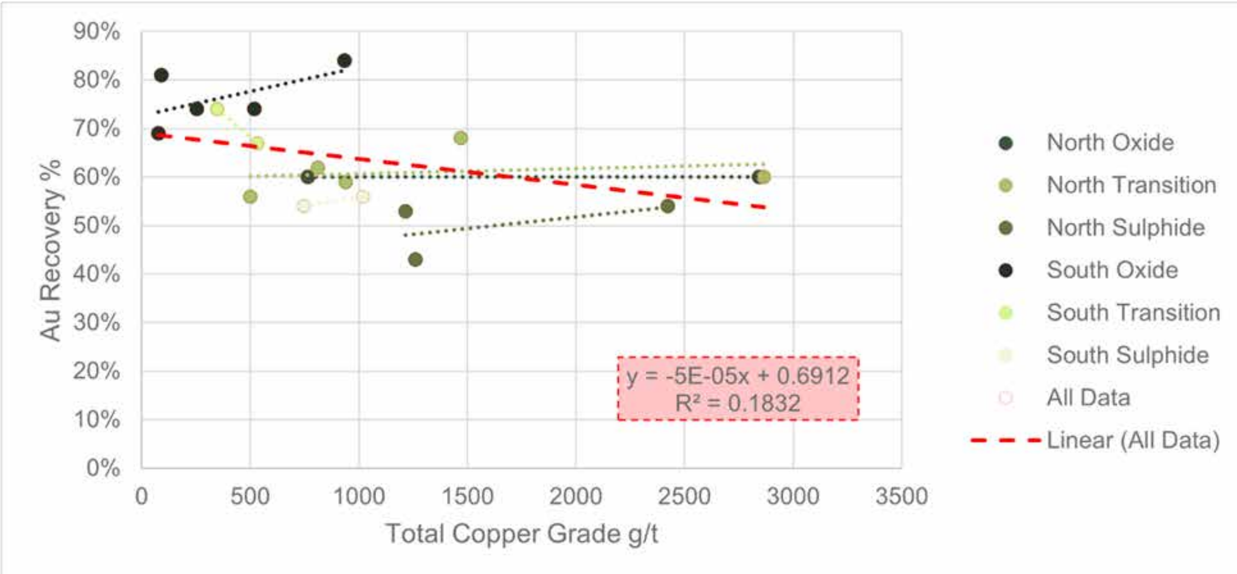


Figure 13-13: Percent Sulphur versus Gold Recovery @ +/- 9.5 mm

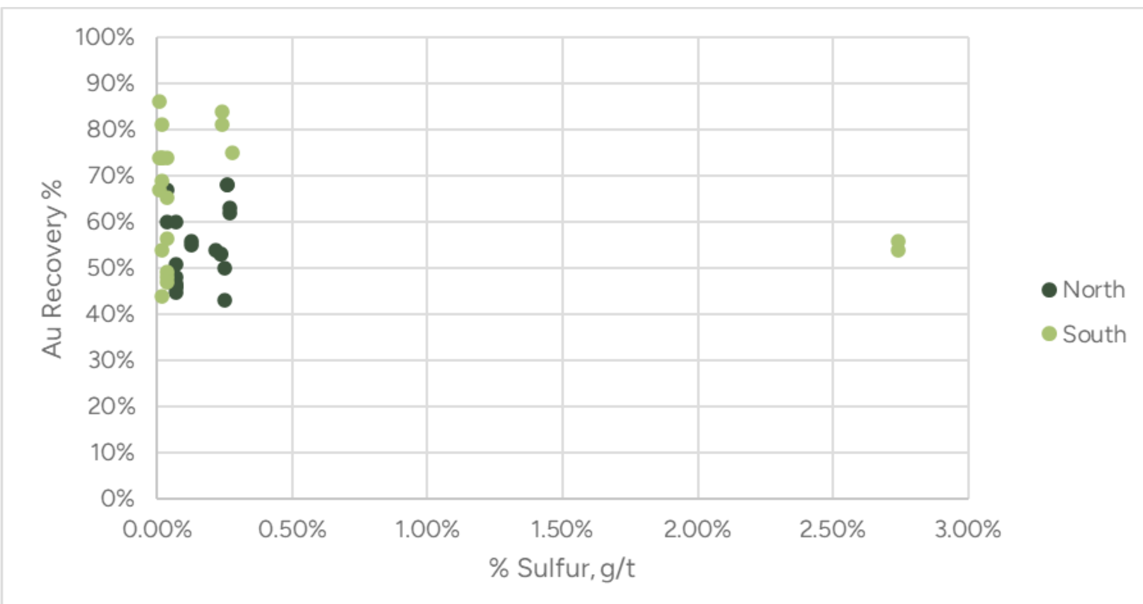


Figure 13-14: Cyanide Feed Concentration versus Gold Recovery

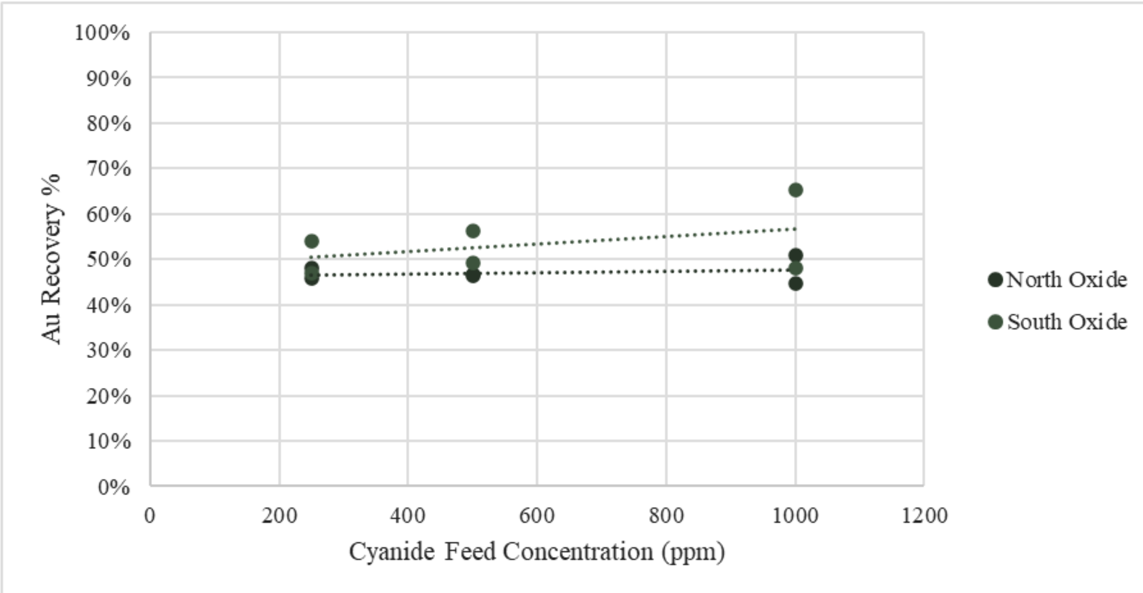
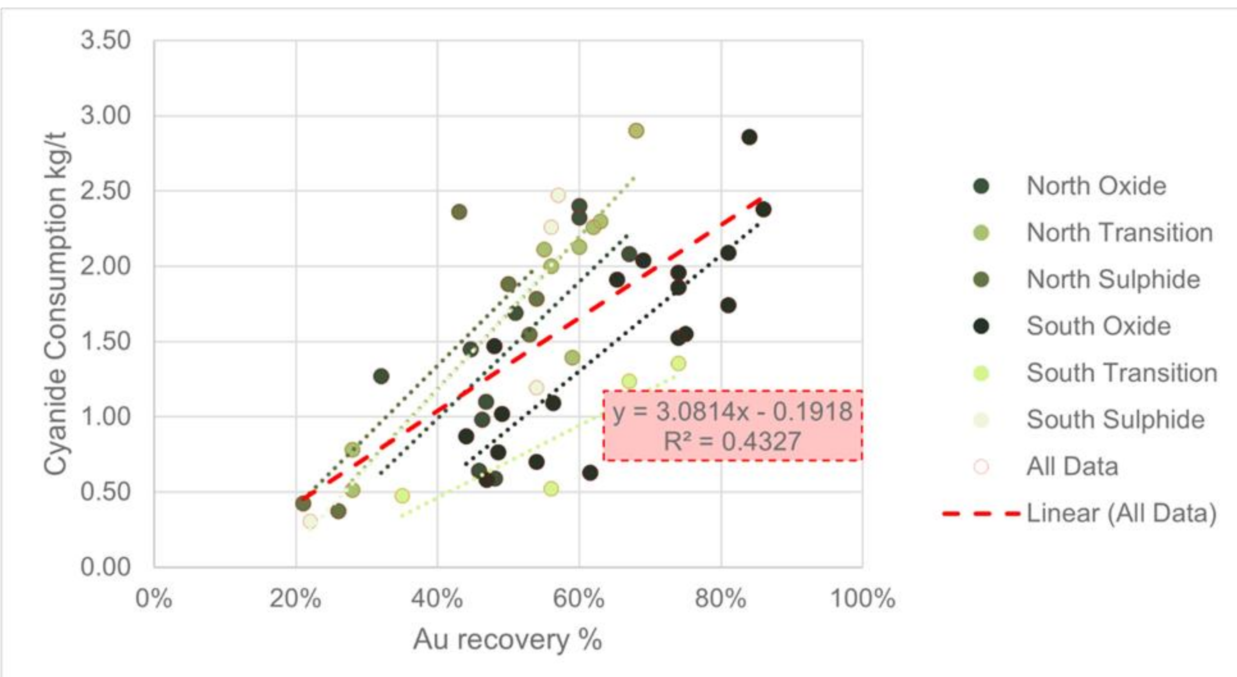


Figure 13-15: Column Cyanide Consumption versus Gold Recovery



The QP recommends proceeding with a target crush size of 80% passing 6.3 mm and tertiary HPGR crushing based on the data available. Recovery estimates have been determined using the linear relationship between P_{80} crush size and recovery at 6.3 mm, plus the estimated additional recovery from HPGR crushing.

Test work results show a noticeable recovery improvement for gold and silver with increasing percent minus 1 mm material (Figure 13-8 and Figure 13-9 for gold and silver, respectively).

The conventional crushed columns at approximately 80% passing 6.3 mm had approximately 25% minus 1.7 mm material compared to the HPGR crushed columns, which produced 36% of minus 1.7 mm material with an overall average of 28% minus 1.7 mm for all columns at the ±6.3 mm P_{80} product size. Additional recovery estimated for HPGR crushing ranged between 4% and 9%, depending on the material type for gold and 0% and 7% for silver (where a negative correlation was observed, no additional recovery was added). A 2% recovery deduction was applied to gold recovery and a 4% deduction was applied to silver recovery after the HPGR recovery additions were applied based on the QP's experience and the limited HPGR test work available.

Although copper does have a minor impact on recoveries, as suggested in previous reports on the Converse Project, the overall influence for each individual material type is significantly less than the effect of material size, and it is the QP's recommendation that copper grade not be used for recovery estimation at this time.

Additional HPGR testing is needed to properly assess the potential benefits and costs and determine whether the recovery improvements are sufficient to warrant the agglomeration costs. The limited results available suggest that the HPGR recovery improvements are minor.

The QP recommends the following recoveries for gold and silver:

- Gold recoveries of:
    - 60% for North Oxide and 79% for South Oxide
    - 69% for North Transition and 76% for South Transition
    - 58% for North Sulphide and 64% for South Sulphide
- Silver recoveries of:
    - 27% for North Oxide and 25% for South Oxide
    - 36% for North Transition and 30% for South Transition
    - 42% for North Sulphide and 39% for South Sulphide

13.9.2 Leach Cycle

A leach cycle has been estimated based on the column leach tests completed to date. The leach cycle considers tonnes of solution per tonne of material processed, as well as the time required to achieve ultimate recovery in column leach tests. The recommended leach cycle for all of the Converse material types is 120 days.

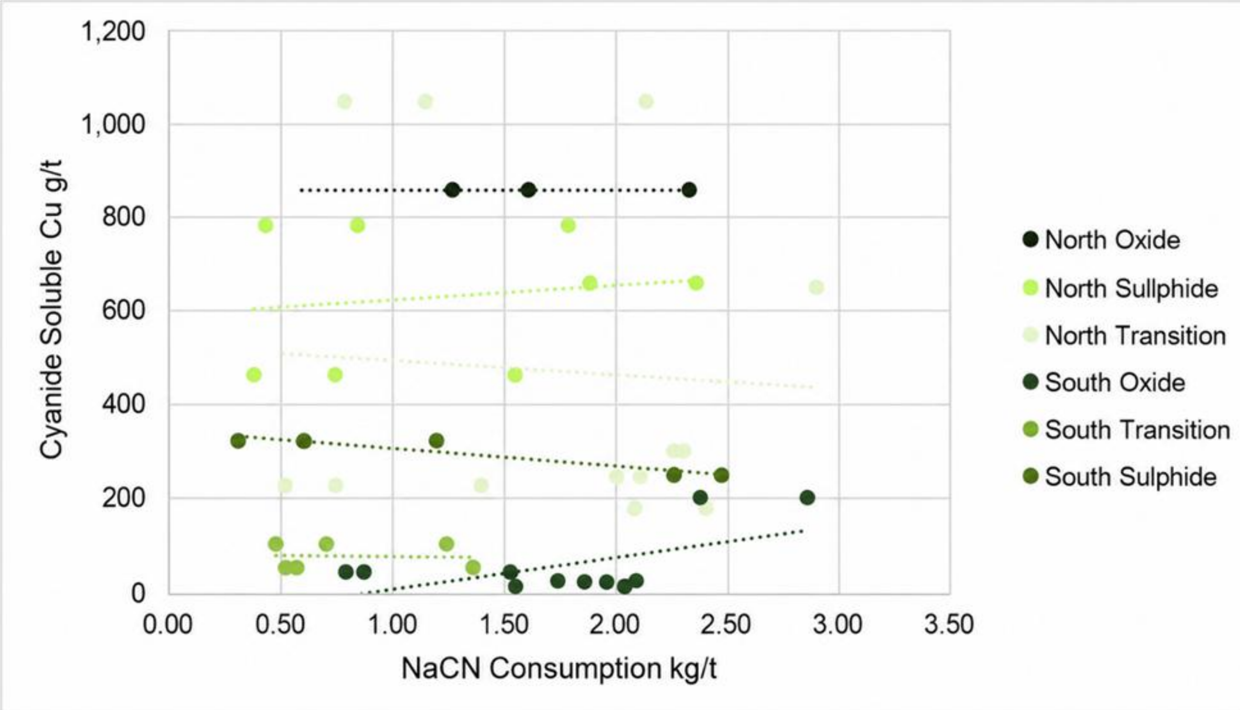
13.9.3 Heap Reagents

13.9.3.1 Cyanide

Cyanide consumptions were evaluated for each material type and adjusted to provide a basis for the expected field cyanide consumptions. In the QP's experience, field cyanide consumptions are typically 33% of the observed laboratory consumptions.

The test work did not show a strong correlation between cyanide soluble copper and cyanide consumption (Figure 13-16) but did show a correlation between gold recovery and cyanide consumption (Figure 13-15).

Figure 13-16: Column Cyanide Consumption Versus Cyanide Soluble Copper



For this analysis, KCA has estimated cyanide consumption based on the estimated gold recoveries for each material type, with a 33% laboratory-to-field factor applied. This method shows slightly higher cyanide consumption than the direct average of the column leach tests and warrants further investigation, including Sulphidization-Acidification-Recycling-Thickening (SART) for copper removal and cyanide recovery. Estimated cyanide consumptions for the material types are as follows:

- 0.6 kg/t for North Oxide and 0.64 kg/t for South Oxide
- 0.84 kg/t for North Transition and 0.42 kg/t for South Transition
- 0.68 kg/t for North Sulphide and 0.76 kg/t for South Sulphide

13.9.3.2 Cement and Lime

Lime or cement will be required to control pH during leaching. Lime consumption is a 1:1 ratio with the laboratory lime consumption. Cement has a lower pH-buffering potential than lime, and typically requires three times as much cement to achieve the same pH buffering potential as lime.

Lime consumption has been based on the average consumption from the column leach tests for each material, using pebble lime. Estimated lime requirements are as follows:

- 0.5 kg/t for North Oxide and 1.0 kg/t for South Oxide
- 0.5 kg/t for North Transition and 0.4 kg/y for South Transition
- 0.4 kg/t for North Oxide and 0.6 kg/t for South Sulphide

Cement agglomeration will be required for HPGR crushed material for heap heights greater than 100 m. The QP recommends a cement addition of 6 kg/t, which should provide sufficient pH buffering for all material types without the need for additional lime.

13.10 Recommendations and Conclusions

In conclusion, significant amounts of test work have been completed on converse material, including evaluation of both heap leach and milling options, using variability testing regarding crush and grind size. It is the QP’s opinion that the current test work available is sufficient for the evaluation of the Project at a PEA level, with recommended metallurgical design parameters presented in this section.

As part of future programs, the QP recommends:

- Additional column leach tests, both on conventionally and HPGR crushed material at variable crush sizes from 80% passing 12.5 mm (1/2”) to 6.3 mm (1/4”) to better understand the recovery versus size relationship and if HPGR crushing is economically beneficial.
- Compacted permeability tests should also be conducted to define potential cement agglomeration requirements (and agglomeration costs) based on the planned ultimate heap heights.
- For a milling case study, conduct additional bottle roll test work, especially on the south sulphide materials. Pulp agglomeration tests should also be considered. All test work should be completed on grade-representative samples.

Some of the material tested has contained high levels of cyanide soluble copper. In the column leach tests, the high copper did not appear to be directly correlated with increased cyanide consumption, unlike the bottle roll tests, which showed higher cyanide consumption at higher copper values. In any case, high copper levels can affect the process, and SART test work is recommended alongside column leach tests to evaluate methods for managing copper in solution. Because cyanide consumption was not excessive, the soluble copper may be manageable through mine planning and cold stripping to remove copper from carbon in the recovery circuit.

Concurrently with the test program, the QP recommends evaluating the resource and mine plan to see if there is sufficient high grade material and if this material can be mined concurrently with lower grade material to support a combined hybrid heap/mill project (such as pulp agglomeration) or a standalone mill based on the high recoveries obtained from the test work for these options. Economic trade-offs should be completed to identify the preferred processing methods.

14.0 Mineral Resource Estimates

14.1 Summary

The Mineral Resource estimate at the Converse Project was completed by SLR between November 2025 and March 2026 and has an Effective Date of March 31, 2026 (Table 14-1). Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Resource classification.

A series of low- and high-grade mineralization domains define the estimation domains for the North and South Redline deposits and are constructed in Leapfrog Geo based on geological and gold grade trend interpretations. Within each domain, gold assays are capped, composited, and interpolated into 10 m x 10 m x 10 m blocks using a multi-pass ordinary kriging (OK) method in Leapfrog Edge.

Classification is assigned based on drill hole spacing criteria referencing observed and modelled continuity, adjusted as needed to ensure consistent and geologically coherent class volumes.

The Mineral Resource estimate is constrained within an optimized pit shell based on a long-term gold price of US$3,000/oz, variable gold recoveries, and application of area and redox domain dependent cut-off-grades of 0.18 g/t or 0.20 g/t Au.

Validation of the block model includes statistical comparisons with composites and nearest neighbour (NN) estimates, swath plots, visual reviews in three-dimensional (3D) and longitudinal, cross-sectional and plan views.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

Table 14-1: Summary of Mineral Resources for the Converse Project – March 31, 2026

Classification	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	103,102	0.65	2,162
Inferred	218,446	0.43	3,035

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported within an optimized pit shell at a cut-off grade of 0.18 g/t Au or 0.2 g/t Au, dependent on zone and redox domain (See Table 14-13).
3. Mineral Resources are estimated using a long-term gold price of US$3,000/oz.
4. Gold recoveries are based on equations derived from metallurgical studies.
5. Recoveries are variable and based on equations derived from metallurgical testing.
6. Bulk density is variable by rock type and redox domain and ranges from 2.62 t/m^3 to 2.71 t/m^3. Overburden was assigned a value of 1.8 t/m^3.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. A minimum mining width of 50 m was used.
9. Numbers may not add or multiply correctly due to rounding.

14.2 Comparison to Previous Resource

The previous Mineral Resource at the Project was prepared by RedDot3D of Toronto, Ontario Canada, effective February 13, 2025. Table 14-2 provides a comparison of the previous MRE and current MRE for the open pit mining scenario.

Several factors contribute to the differences between the two estimates, listed in general order of impact:

- The 2026 MRE relied on drill hole spacing, geological confidence, and data age to assign Indicated and Inferred categories. The 2025 MRE relied primarily on spatial criteria and data quality to assign Measured, Indicated, and Inferred categories. A key difference in the 2026 approach is that a drill hole spacing of approximately 60 m was set as criteria for Indicated Mineral resources, the 2025 MRE classified this spacing as Measured.
- Updated mineral domain approach: In the 2025 estimate, a probabilistic method which referenced metamorphic grade was used. In the 2026 estimate, a discrete gold grade-based method which referenced inferred and interpreted structural and lithology controls was used, which resolved discrete structurally controlled high-grade wireframe domains.
- Geological and redox interpretations were updated in the 2026 estimate based on a 2025 core and chip relogging campaign.
- The gold price used in the 2025 estimate was US$2,000/ oz, compared to US$3,000/oz in the 2026 estimate.
- Gold recovery equations were revised.
- For the 2026 estimate, density application reference both lithology and redox characteristics, as opposed to redox only for the 2025 estimate.
- Seven new drill holes were added since the 2025 estimate, including twinned holes to confirm historical results.

Table 14-2: 2026 Converse Mineral Resource Versus 2025 Mineral Resource

Author	Year	Class	Tonnage (Mt)	Average Au Grade (g/t)	Contained Au (Moz)
RedDot3D	2025	Measured and Indicated	330.1	0.53	5.57
		Inferred	24.8	0.53	0.42
SLR	2026	Indicated	103.1	0.65	2.16
		Inferred	218.4	0.43	3.03
Change		Measured and Indicated	-69%	24%	-61%
		Inferred	780%	-19%	621%

14.3 Resource Database

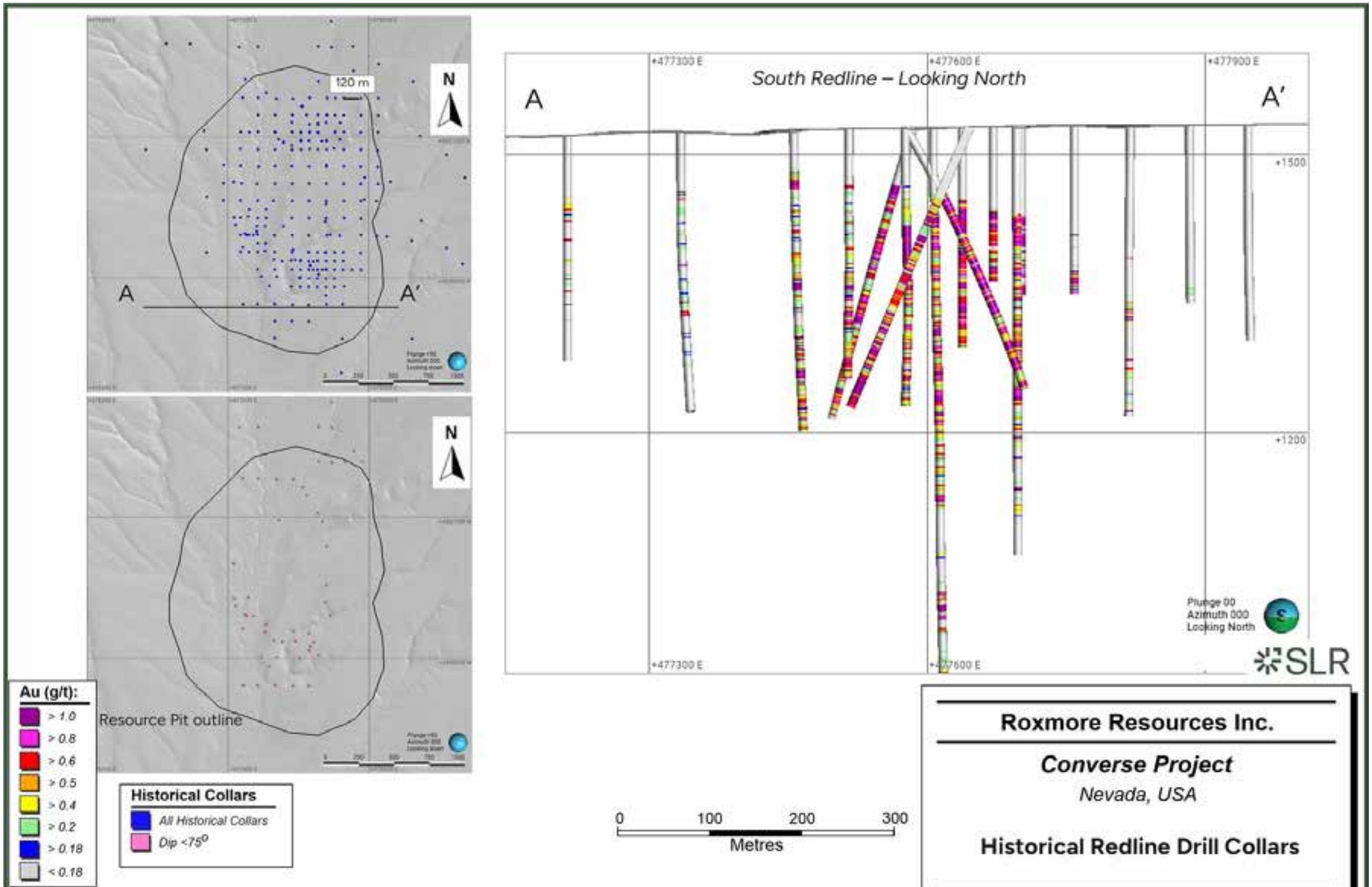
The Roxmore drilling database is maintained in a SQL Access database, with drill hole location information in NAD83 datum, UTM Zone 11N projection (in metres). The database contains information for collars, surveys, assays, lithology, redox, and density.

Various drill hole types contribute to the Converse block model, including RC, mud-rotary holes (MR), diamond drilling or core (DD), and core holes with RC or MR pre-collars (RC-MR/DD). Roxmore's predecessors completed 66,875 m of drilling in 255 drill holes, including 195 RC or MR holes, 60 DD holes, 33 drilled with an RC-MR/DD. Roxmore has drilled seven holes (five core holes with RC pre-collars and two RC holes since the RedDot3D Mineral Resource estimate was completed in February 2025. A summary of drill holes contributing to the Mineral Resource estimate is shown in Table 14-2.

Where assays were not collected, such as through the overburden, SLR assigned a value of 0 g/t Au. In zones of poor recovery through the mineralized horizons, which resulted in missing assays, SLR assigned a null gold value for grade interpolation purposes.

Historical operators primarily drilled vertical holes to a depth of approximately 250 m to 300 m on a 120 m x 120 m grid, with locally tighter spacing (50 m to 75 m) in the central portions of the North and South Redline deposits. Figure 14-1 illustrates the historical drill hole spacing in plan view and the distribution of drill holes drilled at angles less than 75°. The predominance of vertical, relatively shallow holes resulted in many holes terminating within mineralization. Since 2017, drilling activities have incorporated deeper and angled holes to better evaluate the geometry and extent of the mineralized zones.

Figure 14-1: Historical Redline Drill Collars



Source: SLR 2026.

Table 14-3: Summary of Drill Holes Supporting Block Model Estimation

Company	Year	RC or MR		RC - MR/DD[1]		DD		Total	
		No.	Metres	No.	Metres	No.	Metres	No.	Metres
Chevron	1989, 1991	2	247					2	247
Cyprus Mining Corp	1992	2	137					2	137
Independence Mining Co.	1994	6	1,199					6	1,199
Uranerz USA Inc.	1995-1999	87	20,394	9	3,064			96	23,458
Metallic Nevada	2003, 2004, 2007	95	22,607	8	1,618	8	2,235	111	26,460
Converse Resources	2017					7	1,812	7	1,812
International Minerals	2011, 2012	3	536	16	7,243	12	5,783	31	13,562
Roxmore Resources[2]	2025	2	285	3	3,243	2	1,086	7	614
Total		**197**	**45,405**	**36**	**15,168**	**29**	**10,916**	**262**	**71,489**

Notes:
1. Reverse circulation or mud-rotary pre-collar, hole finished with diamond drill core.
2. Includes drilling completed by AxCap Ventures, which was renamed to Roxmore Resources.

14.4 Geological Interpretation and Mineralized Models

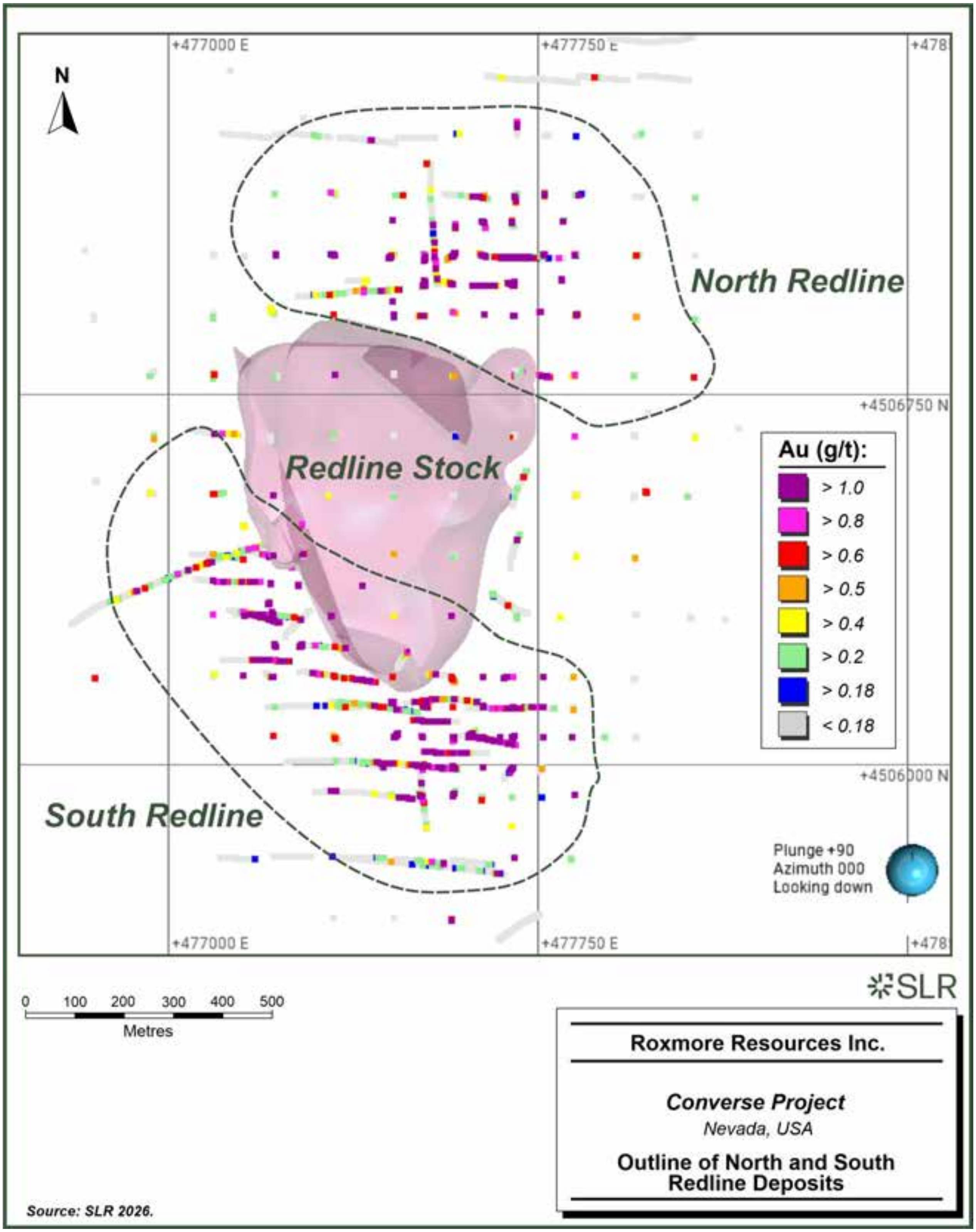
14.4.1 Lithology and Redox Models

Gold mineralization at the Project is spatially associated with the central Redline Stock. Prograde and retrograde metamorphic fluids have interacted with the calcareous and non-calcareous stratabound sediments of the Havallah sequence, depositing microscopic, disseminated gold peripheral to the Redline Stock Figure 14-2. Additional gold enrichment appears to be controlled by northeast-trending structures in the North Redline deposit and northwest-trending structures in the South Redline deposit. Following drilling and relogging efforts in 2025, Roxmore recognized and modelled several multiphase mineralized breccias; however, the current understanding of the event that caused breccia-hosted mineralization remains limited.

Roxmore shared with SLR a Leapfrog project containing digital lithology wireframe solids, redox solids, and interpreted faults that cover the extent of the Project. Lithology solids include the Main Redline Stock, sediments and calcareous sediments of the Havallah Formation, several breccia bodies, and a granodiorite intrusion present at depth in the South Redline deposit. An alluvium or overburden solid was also modelled as overlying the entire project footprint. Additional lithology solids occur outside the current resource areas, including a limestone unit interpreted to represent the Antler Peak stratigraphic horizon. Figure 14-3 shows the modelled lithology at North and South Redline, respectively.

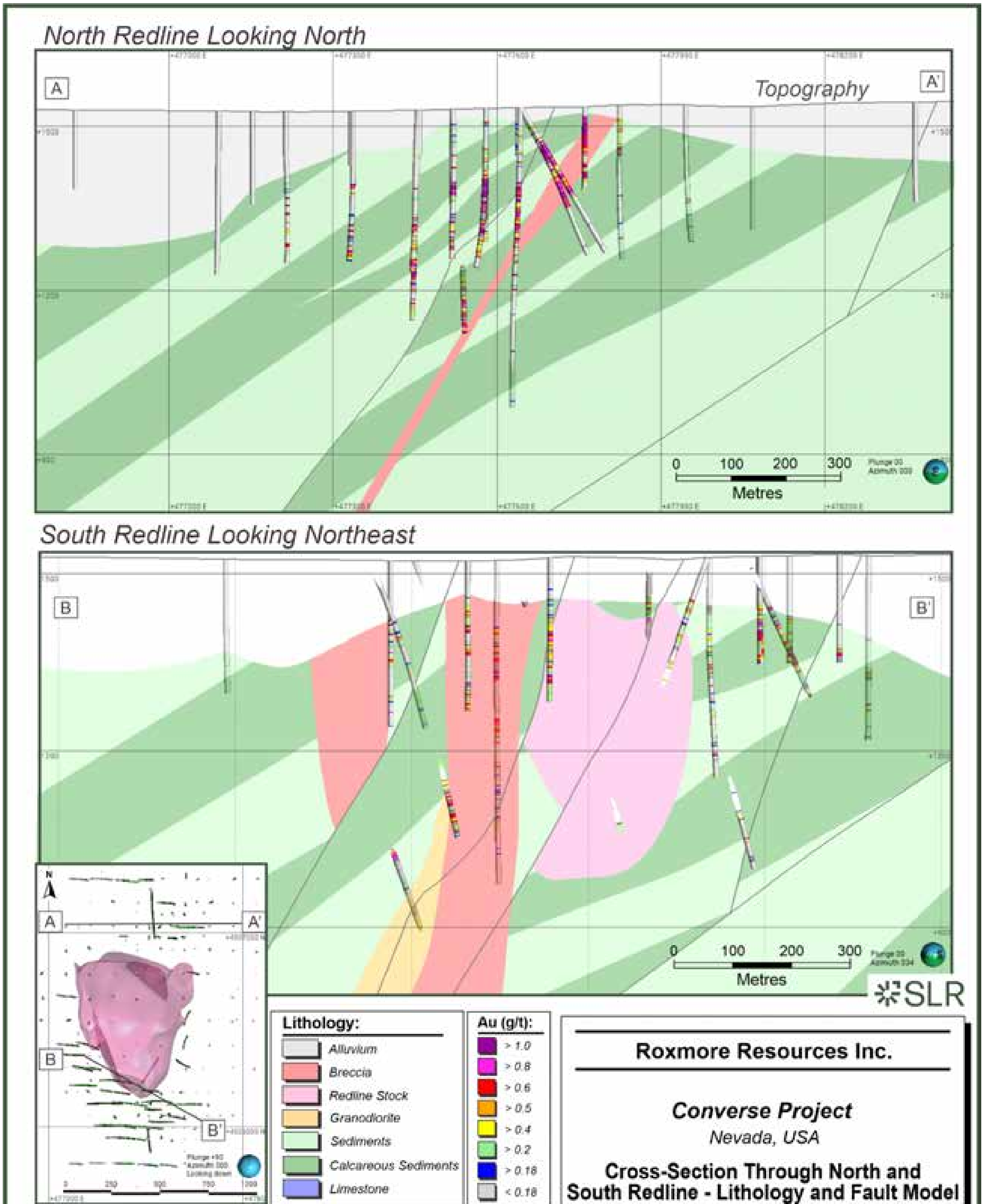
The redox model includes an oxide, transition, and primary division based on visual observations such as colour, hardness, and abundance of fresh sulphides. Roxmore has used their full multi-element suite of assays to help guide the geologic, redox and structural interpretations. The QP reviewed the downhole lithology logs and geochemistry and is of the opinion the lithologic and redox wireframes provided are acceptable for the purposes of this study. An example of the redox model is shown for the South Redline deposit in Figure 14-4.

Figure 14-2: Outline of the North and South Redline Deposits



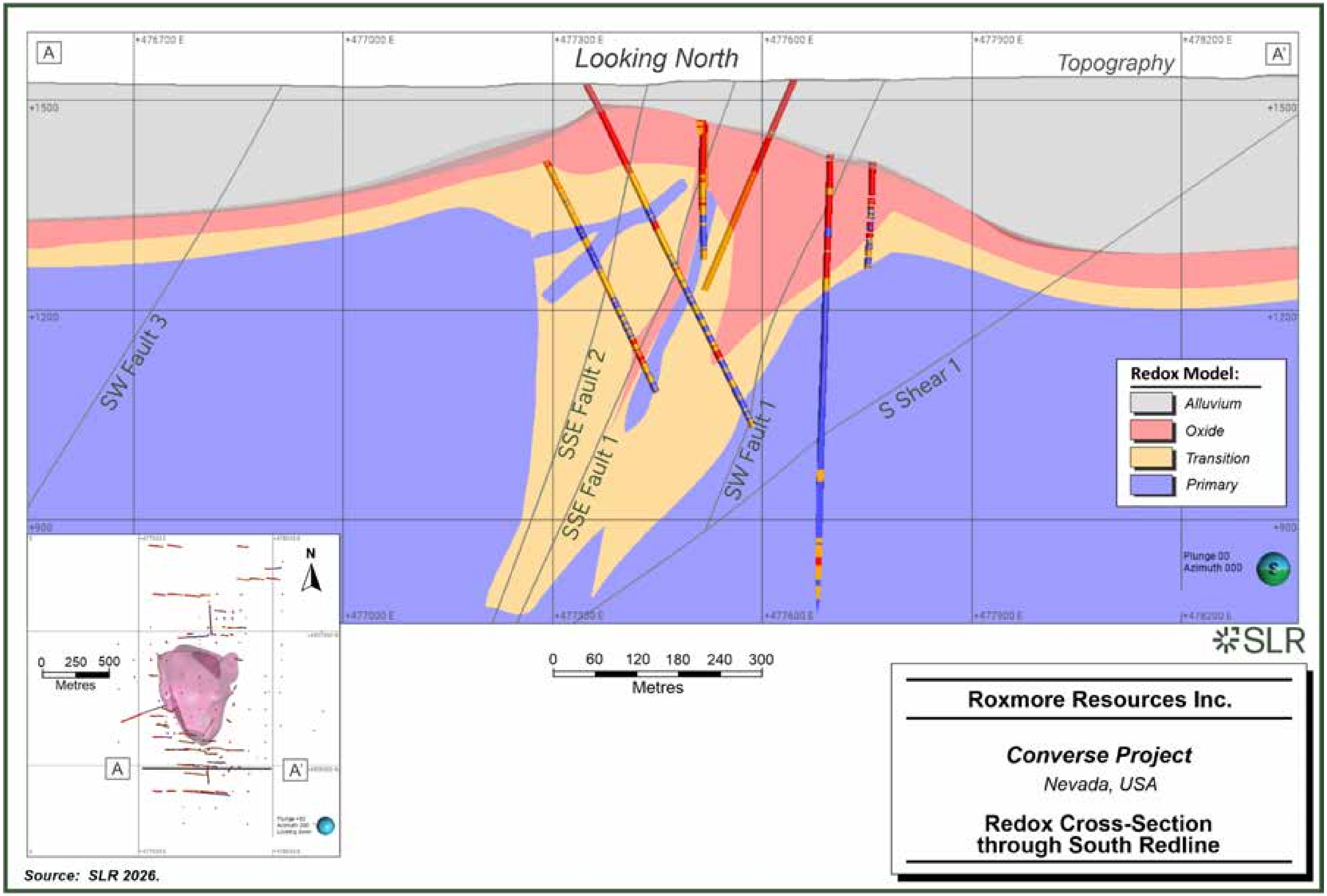
Source: SLR 2026.

Figure 14-3: Cross-Section Through North and South Redline - Lithology and Fault Model



Source: SLR 2026.

Figure 14-4: Redox Cross-Section through South Redline



Source: SLR 2026.

14.4.2 Mineral Domains

Gold assay interpolants were evaluated at various grade cut-off values, searching for cohesive trends that honour the geologic and structural interpretation. At both North and South Redline deposits, assays exceeding or equal to 0.5 g/t Au were found to form coherent mineralized volumes. Using Leapfrog Geo's intrusion and vein modelling techniques, SLR built a series of domains that maintain a gold grade of greater than 0.5 g/t Au and that honour the geologic and structural interpretations. A low-grade population was subsequently wireframed using the same methodology to capture material grading greater than or equal to 0.1 g/t Au, fully encompassing the higher grade domains. The tally of domains at both the North and South Redline deposits is shown in Table 14-4.

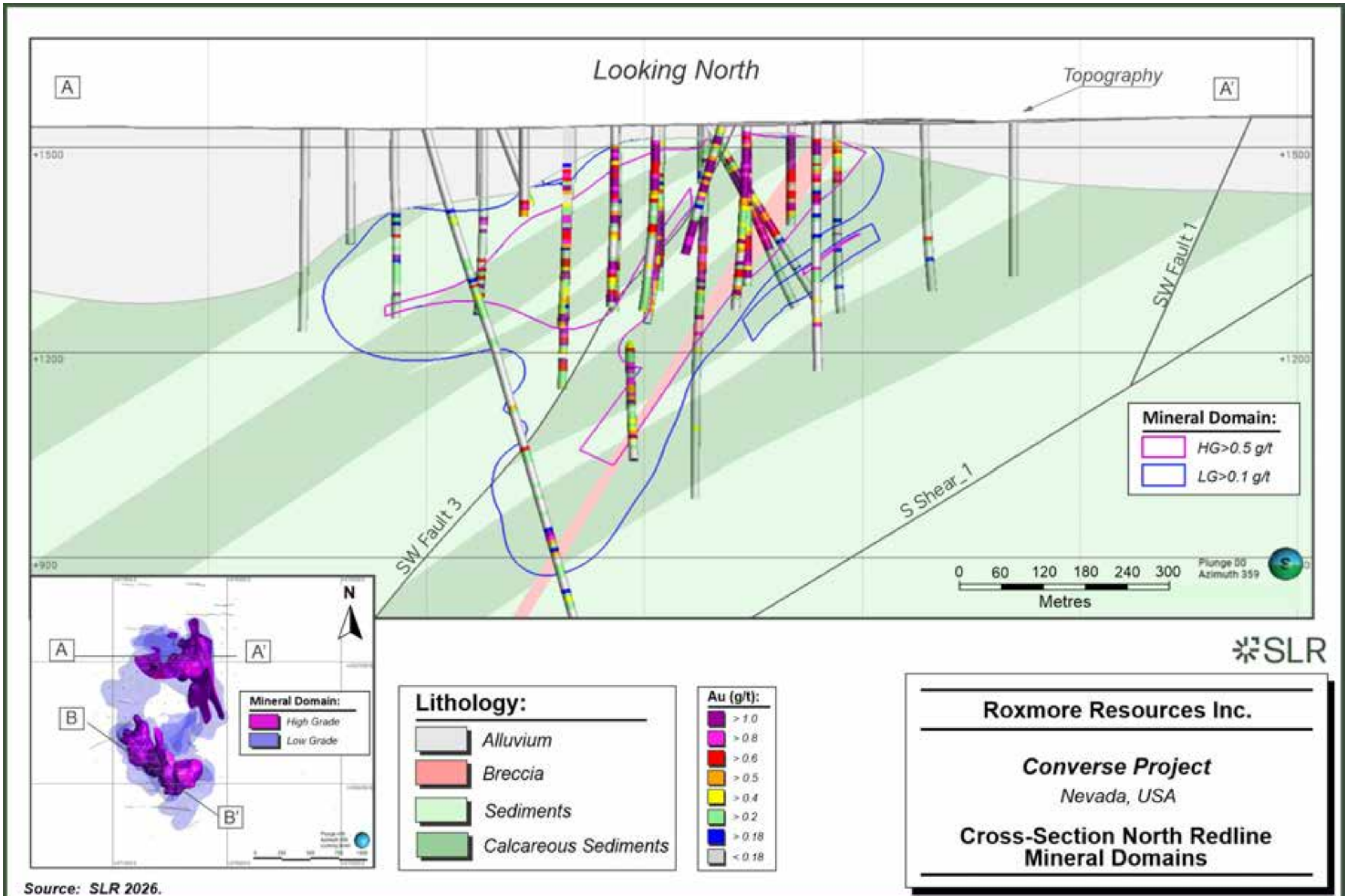
Mineralization at both North and South Redline extends over 850 m along a northwest-southeast trend and 400 m along a northeast–southwest trend orientation. At North Redline, mineralization has been defined to a vertical depth of approximately 500 m and up to 550 m depth at the South Redline deposit. Figure 14-5 and Figure 14-6 provide a cross-section view through the North and South Redline domains.

Additionally, SLR created a very low-grade buffer (vlg) domain to constrain gold mineralization outside of the low-grade domains. Gold estimated within the vlg domain is not reported in the Mineral Resource estimate due to insufficient geologic confidence and widely spaced drill holes.

Table 14-4: Converse Project Mineralized Wireframes

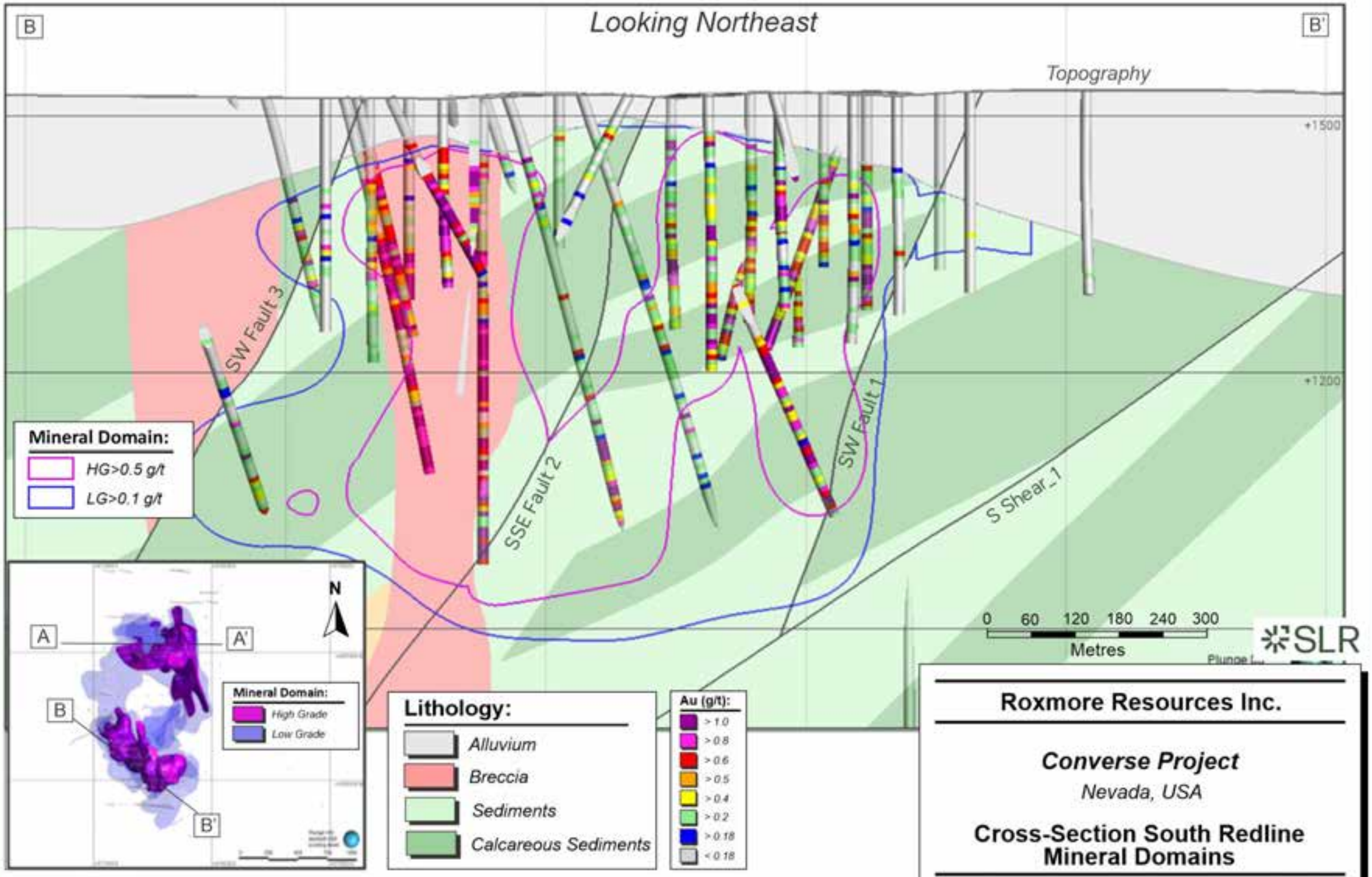
Area	Number of Low-Grade Wireframes	Number of High-Grade Wireframes
North Redline	5	5
South Redline	7	3

Figure 14-5: Cross-Section – North Redline - Mineral Domains



Source: SLR 2026.

Figure 14-6: Cross-Section - Redline South - Mineral Domains



Source: SLR 2026.

14.5 Assay Statistics, Capping and Compositing

14.5.1 Capping

To limit the influence of a small number of high assay values that could disproportionately influence grade estimation, SLR applied high-grade capping to the drill hole database. Raw assays were reviewed within both the low-grade and high-grade mineralized domains using basic statistics, histograms, log probability plots, and decile analysis to determine the presence of outliers and appropriate capping thresholds. Examples of the capping analysis are shown in Figure 14-7 to Figure 14-8.

The selected capping levels presented in Table 14-5 reflect a balance between preserving legitimate high-grade mineralization and reducing the undue influence of statistical outliers.

Figure 14-7: Selected Capping Analysis for High-Grade Domains: Capped at 15 g/t Au

Log Probability Plot **Histogram (Zoomed)**

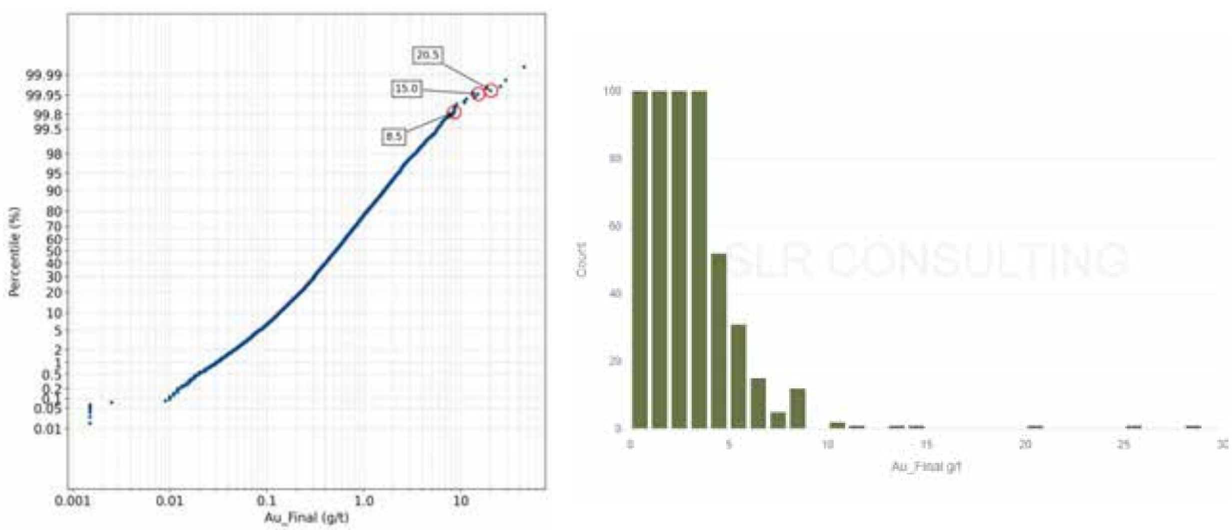


Figure 14-8: Selected Capping Analysis for Redline South Low-Grade Domains: Capped at 7.5 g/t Au

Log Probability Plot **Histogram (Zoomed)**

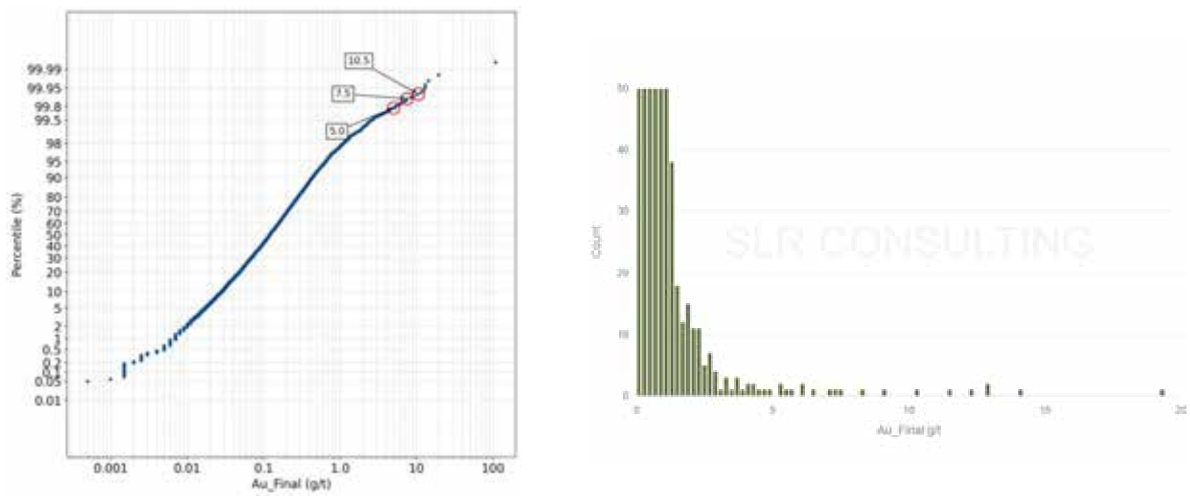


Table 14-5: Capped Gold Assay Statistics

Area	Min Group	Assays	Number of Samples	Mean (g/t)	CV	Min (g/t)	Median (g/t)	Max (g/t)	Number of Caps
North Redline	High Grade	Au	3,553	0.82	1.45	0.00	0.52	44.70	
		AuCap	3,553	0.82	1.20	0.00	0.52	15.00	1
	Low Grade	Au	3,864	0.20	1.87	0.00	0.12	11.40	
		AuCap	3,864	0.20	1.74	0.00	0.12	7.50	2
South Redline	High Grade	Au	6,246	0.73	1.34	0.00	0.50	28.80	
		AuCap	6,246	0.73	1.21	0.00	0.50	15.00	3
	Low Grade	Au	5,835	0.25	6.13	0.00	0.12	107.55	
		AuCap	5,835	0.23	2.09	0.00	0.12	7.50	8

14.5.2 Compositing

Composites were created from capped assay values using the downhole compositing function in Leapfrog's modelling software. Composite lengths used for interpolation were selected with consideration of the dominant sampling interval, the parent block dimensions, the style of mineralization, and the continuity of grade.

At Converse, the dominant sampling length is 1.524 m (5 ft). Accordingly, capped assays were composited into 6.096 m downhole intervals, honouring mineral domain boundaries. The selected composite length represents approximately four times the average sample length and approximately 60% of the parent block height (10 m). Residual composites were retained within the dataset. Raw interval length statistics versus composite length statistics by area are presented in Table 14-6. The statistical values indicate that there is less than 1% difference between total raw interval lengths versus the composited lengths for each domain group. Table

14-7 presents the capped composite statistics for gold by domain group at North and South Redline.

Table 14-6: Converse Interval Length Statistics by North and South Zones

Area	Domain Group	Total Raw Assay Length (m)	Total Composite Length (m)	% Difference, Length
North Redline	High Grade	5,331	5,344	-0.3%
	Low Grade	5,840	5,864	-0.4%
South Redline	High Grade	9,214	9,232	-0.2%
	Low Grade	8,639	8,642	0.0%

Table 14-7: Converse Composites Statistics by Domain Group

Area	Domain Group	Number of Samples	Mean (g/t)	CV	Min (g/t)	Median (g/t)	Max (g/t)
North Redline	High Grade	909	0.81	0.83	0.01	0.62	4.93
	Low Grade	1,024	0.20	1.04	0.00	0.15	2.38
South Redline	High Grade	1,540	0.73	0.78	0.03	0.58	6.10
	Low Grade	1,479	0.22	1.31	0.00	0.15	4.26

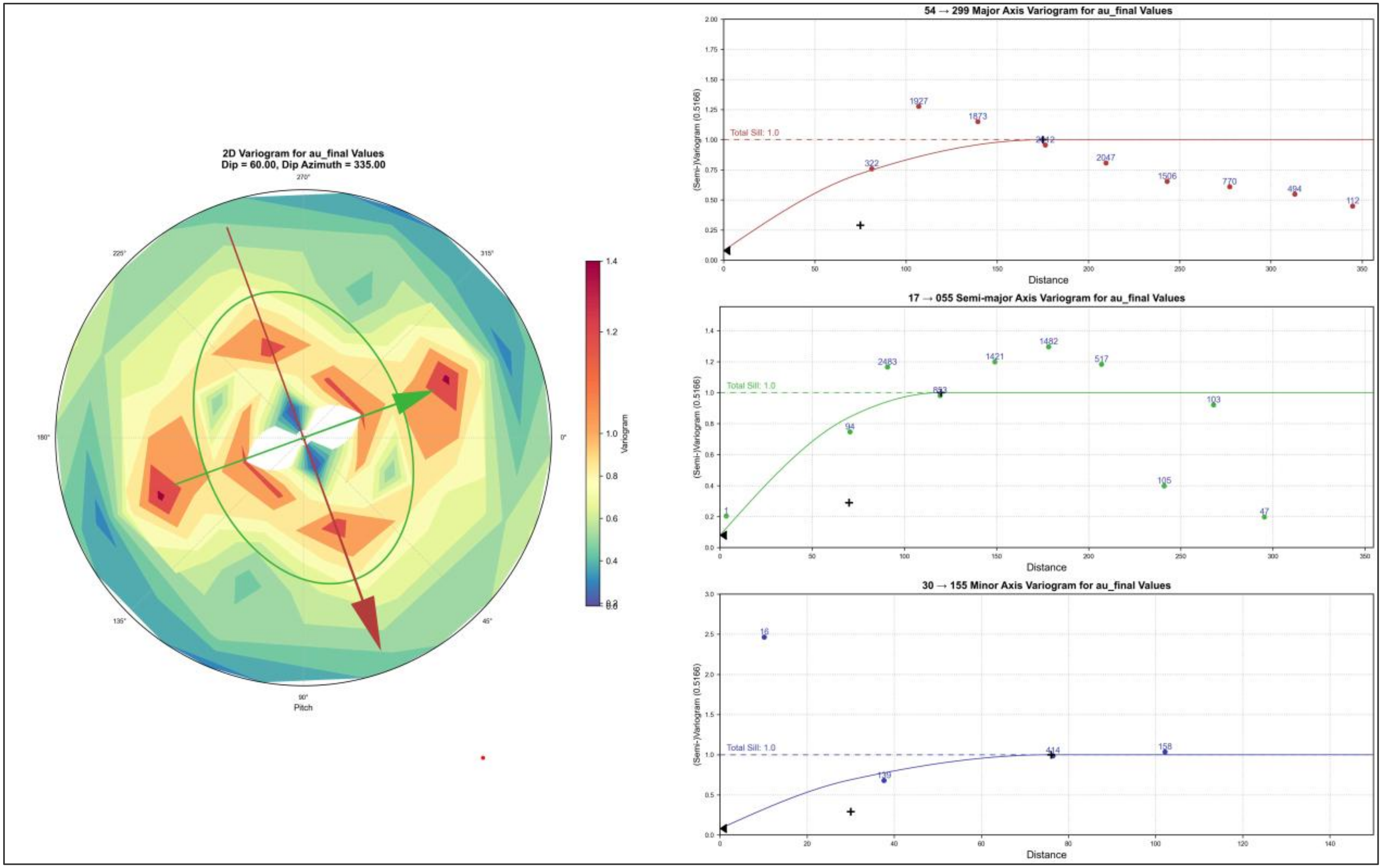
14.6 Trend Analysis

14.6.1 Variography

Experimental semi-variograms within selected individual mineralized domains were generated using the 6.096 m composited values. Mineralization trends were assessed in select domains using Leapfrog's indicator-based interpolant tools. Variogram orientations were aligned with the interpreted geometry of the mineralization, with major and semi-major axes aligned in the plane of the mineralized trends. Experimental semi-variograms were modelled using a nugget component and one or two structures, as appropriate. Downhole variograms were used to quantify and constrain nugget effect.

The mineral domains at the Converse deposit were constructed to represent the large-scale bulk tonnage, style of mineralization. As a result, individual mineral domains encompass multiple geologic and structural complexities. Variogram analysis was complicated by overlapping mineralization trends and the presence of intersecting structural fabrics incorporated into the domains, including shallow dipping bedding planes and steeply dipping structural orientations. This complexity produced experimental variograms with inconsistent ranges and limited stable directional continuity. Consequently, the final variogram models relied more on geological interpretation and observed grade continuity than purely statistical variogram fits. An example variogram model is shown in Figure 14-9 with corresponding parameters summarized in Table 14-8.

Figure 14-9: Variogram Model – North Redline Domain 201



Source: SLR 2026

Table 14-8: Select Converse Variogram Parameters

Area	Domain	Direction			Nugget	Structure 1				Structure 2			
		Dip (°)	Dip Az (°)	Pitch (°)		Sill 1	Major	Semi	Minor	Sill 2	Major	Semi	Minor
North Redline	200	40	270	56	0.15	0.58	55	55	10	0.27	120	95	20
	201	60	335	70	0.08	0.21	75	70	30	0.71	175	120	76
South Redline	800	75	260	125	0.1	0.01	80	70	10	0.89	150	140	50
	400	85	260	105	0.4	0.4	95	95	30	-	-	-	-

14.7 Search Strategy and Grade Interpolation Parameters

Gold grades were interpolated into parent blocks using a multi-pass OK approach. Search distances were guided by modelled variogram ranges and by local drill hole spacing. With each successive pass the search ellipsoid dimensions were expanded, and the minimum composite requirements were relaxed. Ellipsoid orientations were aligned with interpreted grade trends and geologic controls. Dynamic anisotropy was applied to domains 106, 202, 205, and 207, within the North Redline zone. Table 14-9 summarizes the search ellipse parameters used for the Converse deposit. In several domains, isolated high-grade samples—although within the capped limits—were found to influence grade interpolation over an unreasonably large distance. To mitigate this effect, high-grade restrictions were applied as outlined in Table 14-10.

A NN estimation (using 9.144 m composites) was prepared for comparison and validation purposes.

Table 14-9: Grade Interpolation Parameters and High-Grade Restrictions

Zone	Domains	Pass	Ellipse Size (m)	Ellipse Direction			Min Samples	Max Samples	Max per Drill Hole
				Dip (°)	Dip az (°)	Pitch (°)			
North Redline (Low Grade)	100	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	400x400x200				3	20	2
	103	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	104	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	105	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	106	1	80x80x40	Variable Orientation			6	20	2

Zone	Domains	Pass	Ellipse Size (m)	Ellipse Direction			Min Samples	Max Samples	Max per Drill Hole
				Dip (°)	Dip az (°)	Pitch (°)			
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
North Redline (High Grade)	200	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	201	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	202	1	80x80x40	Variable Orientation			6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	203	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	205	1	80x80x40	Variable Orientation			6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	207	1	80x80x40	Variable Orientation			6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
South Redline (LG)	300	1	80x80x40	70	265	90	6	20	2
		2	160x160x80				6	20	2
		3	400x400x200				3	20	2
	102	1	80x80x40	34	270	90	6	20	2
		2	160x160x80				6	20	2
		3	400x400x200				3	20	2
	301	1	80x80x40	35	255	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	302	1	80x80x40	0	0	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2

Zone	Domains	Pass	Ellipse Size (m)	Ellipse Direction			Min Samples	Max Samples	Max per Drill Hole
				Dip (°)	Dip az (°)	Pitch (°)			
	303	1	80x80x40	0	0	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	304	1	80x80x40	0	0	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	305	1	80x80x40	0	0	90	6	20	2
		2	160x160x80				6	20	2
		3	400x400x200				3	20	2
South Redline (HG)	400	1	80x80x40	70	265	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	600	1	80x80x40	70	265	90	6	20	2
		2	160x160x80				6	20	2
		3	320x320x100				3	20	2
	800	1	80x80x40	70	265	90	6	20	2
		2	160x160x80				6	20	2
		3	400x400x200				3	20	2
VLG	vlg	1	100x100x50	0	0	90	6	20	2
		2	400x400x100				6	20	2
		3	800x800x200				3	20	2

Notes:
Soft boundaries applied to domains:

- 102 and 103
- 200, 201 and 202
- 300 and 302
- 400, 600 and 800

14.7.1 High-Grade Restrictions

At the estimation level, certain domains were found to be pushing isolated high-grade samples too far in the first or second pass. To minimize grade smearing in these domains, restrictions were applied, summarized in Table 14-10.

Table 14-10: High Grade Search Restrictions

Domain	Au (g/t)	Distance (m)
100	1.0	60
102	0.6	30
202	1.0	60
301	0.6	30
305	1.0	30
800	1.7	60
Note: The outlier restriction in the 100 domain was applied to the 3rd pass; 202 domain was only applied in the 2nd and 3rd passes. All other domains had outlier restrictions applied in all passes.		

14.8 Bulk Density

A total of 634 density measurements were collected within the North and South Redline deposits, including 119 from alluvium samples. The alluvium density samples range from 1.29 g/cm^3 to 3.67 g/cm^3 with a mean value of 2.27 g/cm^3. The unconsolidated alluvium observed in the field, drill sumps, and in core photos does not support such a high average density. It is therefore interpreted that many of the reported measurements reflect cobbles and boulders rather than the true alluvial matrix material. For this reason, SLR applied an average bulk density of 1.8 g/cm^3 to the alluvium domain. The remaining 515 density measurements, representing competent bedrock, were assigned by lithology and redox and are summarized in Table 14-11.

Table 14-11: Converse Density Measurements

Lithology	Redox	Density (g/cm^3)
Alluvium	All	1.80
Sediments	Oxide	2.59
	Transition	2.70
	Primary	2.72
Calcareous Sediments	Oxide	2.63
	Transition	2.72
	Primary	2.72
Breccia	Oxide	2.59
	Transition	2.61
	Primary	2.68
Granodiorite	All	2.66
Limestone	All	2.90
Redline Stock	All	2.62
	Otherwise	2.60

14.9 Block Models

The block model and Mineral Resource estimate were completed in Leapfrog Edge software. Block model dimensions are presented in Table 14-12. No rotation was applied to the model. SLR considers the model to be appropriate for the deposit geometry and proposed mining methods.

Table 14-12: Block Model Dimensions and Location

Parameter	X	Y	Z
Base Point	476220	4505030	1567
Boundary Size (m)	2,330	2,800	840
Block Size (m)	10	10	10

14.10 Cut-off Grade and Whittle Parameters

14.10.1 Cut-off Grades

Table 16-2 and Table 16-3 present the calculation of marginal cut-off grades for the North Redline and South Redline deposits, reported by redox domain and based on a gold price of US$3,000/oz for Mineral Resource reporting. Metallurgical recoveries are domain-dependent and range from 58% to 79%, varying by zone and redox classification. The marginal cut-off grade reflects processing and G&A costs only and does not include mining costs.

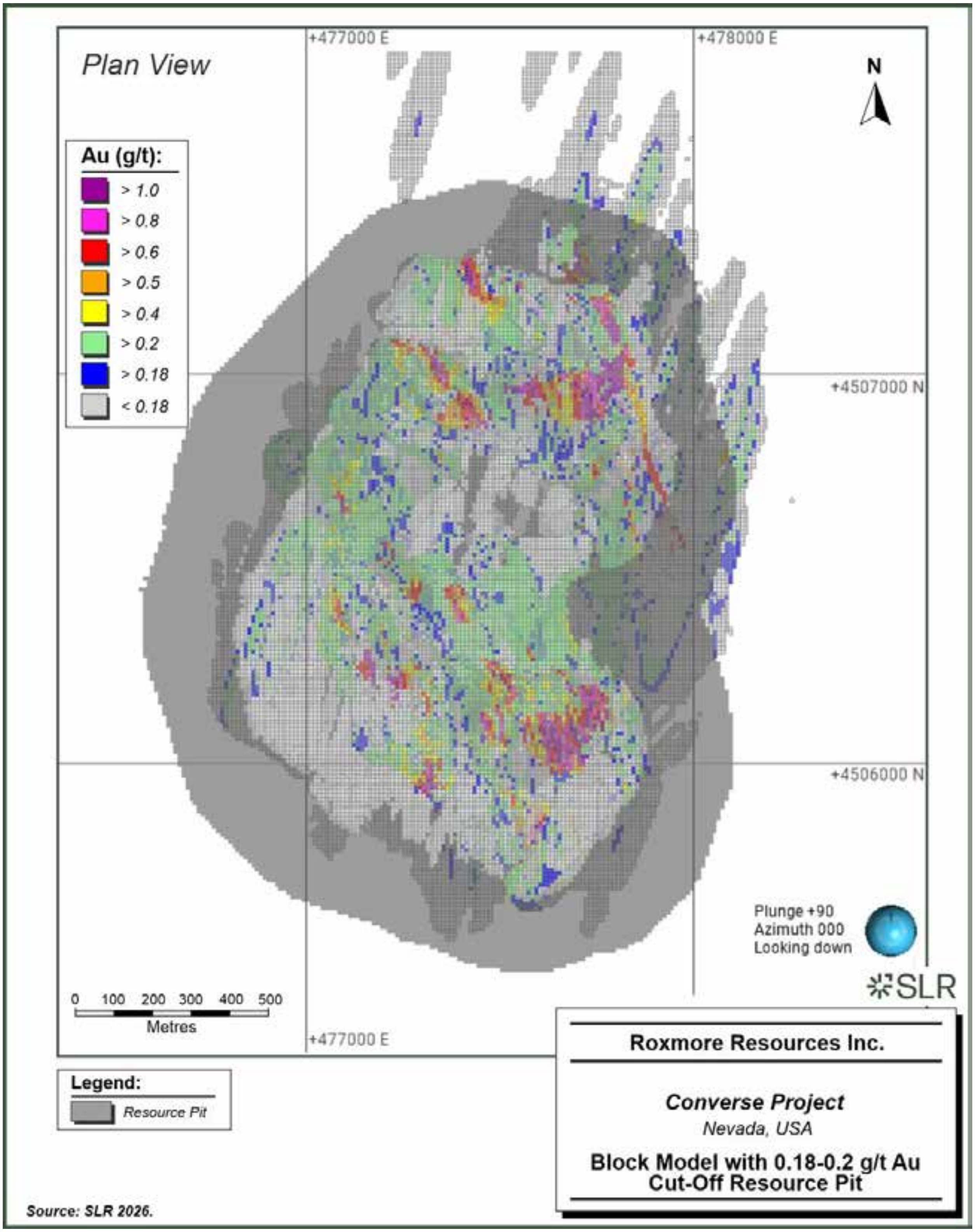
A cut-off value of 0.18 g/t and 0.20 g/t Au was selected for Mineral Resource reporting to account for uncertainties in the inputs to the cut-off grade calculation. Table 14-13 summarizes parameter inputs and highlights which domains used a 0.18 g/t or 0.20 g/t Au cut-off value. Figure 14-10 shows the conceptual pit shell with a 0.18 g/t or 0.20 g/t Au grade cut-off value as applied.

Table 14-13: Support Data for the Mineral Resource Estimate

Inputs	Area	Redox	Units	Value	Cut-off Grade (g/t Au)
Resource Metal Price	All	All	US$/oz Au	3,000	
Processing Recovery	North	Oxide	%	60	0.20
		Transition	%	69	0.18
		Primary	%	58	0.20
	South	Oxide	%	79	0.18
		Transition	%	76	0.18
		Primary	%	64	0.18
Mining Cost	All	All	US$/t mined	2.2	
Processing Cost	All	All	US$/t milled	5.5	
G&A Cost	All	All	US$/t milled	1	
Pit Slopes[1]	All	All	º	45	
Cut-off Grade	All	All	g/t Au		0.18 or 0.20

Notes:
1. Pit Slopes uses 38 degrees for overburden
2. Minimum mining width of 50 m

Figure 14-10: Block Model and 0.18-0.2 g/t Au Cut-Off Resource Pit



Source: SLR 2026.

14.10.2 Pit Optimization

To fulfill the CIM requirements of Reasonable Prospects for Eventual Economic Extraction (RPEEE), SLR prepared a conceptual open pit shell for the Project to constrain the block model for Mineral Resource reporting.

Pit optimization was conducted in Whittle software using the Pseudoflow algorithm to generate a pit shell based on a regular block model, 10 m x 10 m x 10 m, and a set of input economic and technical parameters summarized in Table 16-2 and Table 16-3. The overall slope angle (OSA) used in the optimization ranged from 38° through the overburden and 45° through the bedrock based on technical work done by Call & Nicholas, Inc. (CNI) (CNI 2008).

Whittle uses the Pseudoflow algorithm to define the blocks that can be mined at a profit and creates an RPEEE shell based on the following information:

- Topography
- Slope angle by material (overburden versus bedrock)
- Metallurgical recoveries by zone (North/South) and redox
- Gold mineralization domain model with gold grades, density, lithology, and redox
- Process and mining costs
- Incremental vertical bench mining costs
- Downstream costs, such as gold refining, royalties, freight, and marketing

14.11 Classification

Definitions for resource categories used in this Technical Report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction”. Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the “economically mineable part of a Measured and/or Indicated Mineral Resource” demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories. No Mineral Reserves have been estimated.

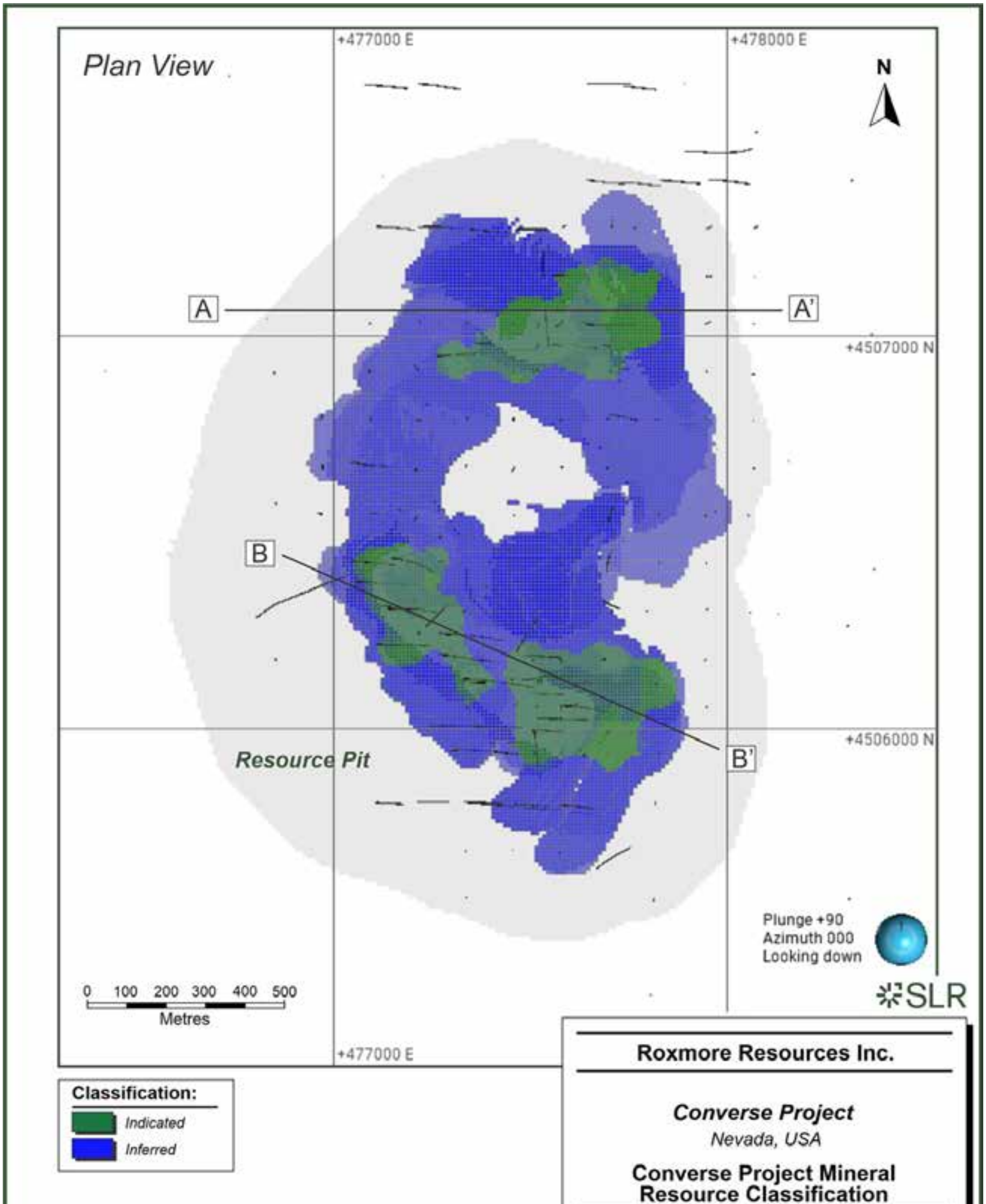
Resource classification at the Project was completed on a block-by-block basis using a combination of drill hole spacing, supplemented by post-processing to ensure continuity. Blocks with an average drill hole spacing of less than approximately 60 m from the block centre were classified as Indicated Mineral Resources where drill hole spacing average is generally less than 60 m from the estimated block centre, while blocks informed by drill hole spacing of approximately 120 m were classified as Inferred Mineral Resources. No blocks were classified as Measured.

The classification methodology is summarized as follows:

- Drill hole spacing of the three closest holes was applied to each block using Leapfrog Edge tools.
- Classification shapes were generated using Leapfrog tools guided by the drill hole spacing to create coherent and continuous classification domains.

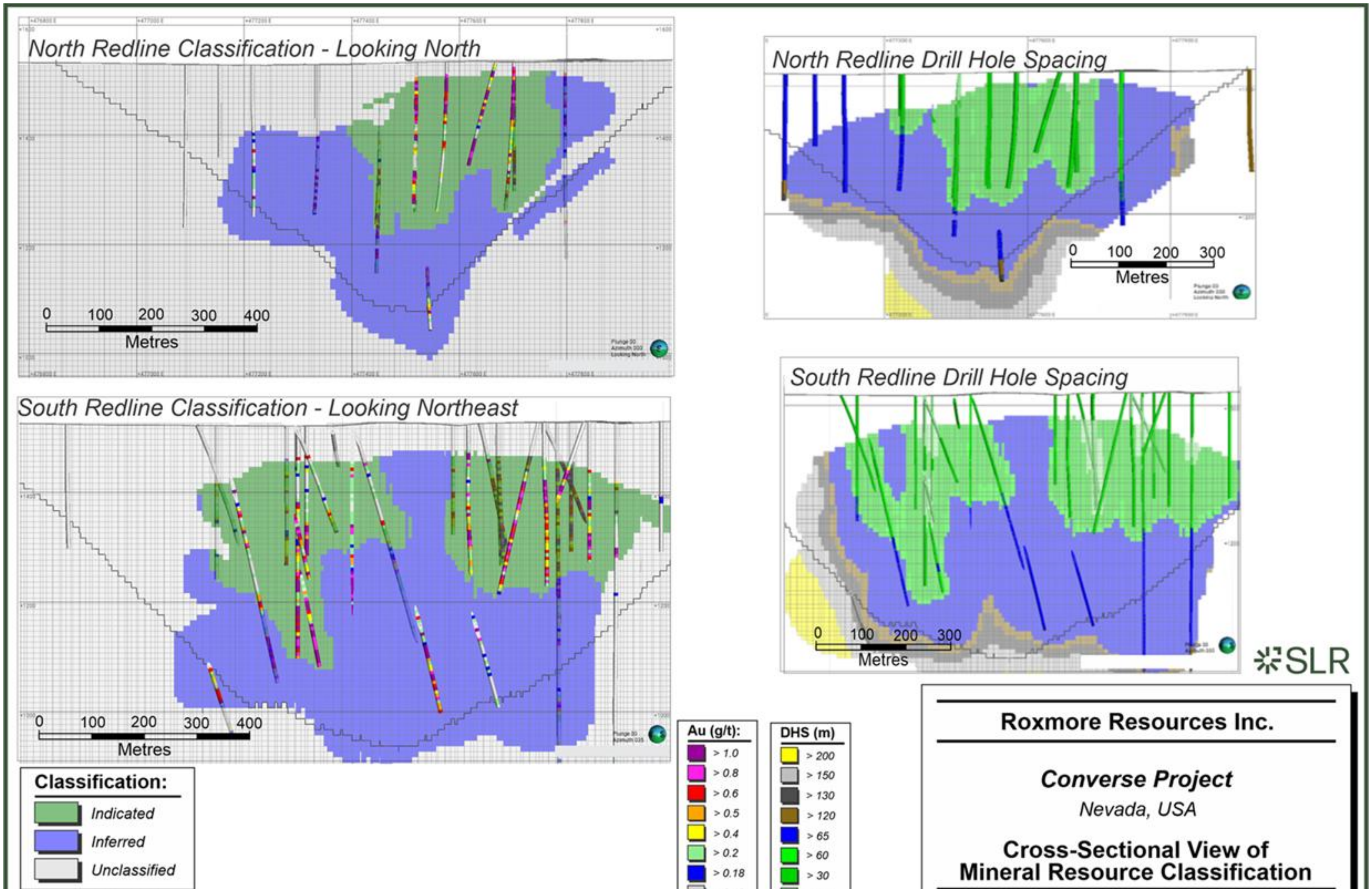
Figure 14-11 illustrates a plan view of the resource classification, and Figure 14-12 and shows cross-sections of both North and South Redline.

Figure 14-11: Converse Project - Mineral Resource Classification



Source: SLR 2026.

Figure 14-12: Cross-Sectional View of Mineral Resource Classification



Source: SLR 2026.

14.12 Block Model Validation

Blocks were validated using various techniques, including:

- Statistical comparison of assay, composite, and block statistics.
- Visual inspection of composite versus block grades.
- Wireframe and block model volume confirmation.
- Swath plots comparing OK and NN values.

Table 14-14 summarizes the comparison between assay, composites and block model estimates. Overall, SLR observes good agreement between the sample data, NN estimates, and the OK block grades, although some domains, typically with limited data, exhibit slightly higher variability, such as domain 203.

Example swath plots used to compare the OK estimate and NN estimation are shown in Figure 14-13.

Visual validation of the estimate was also completed by reviewing the composites and corresponding block grades in both plan and cross-sectional views. Representative examples are provided in Figure 14-14 to Figure 14-16.

Table 14-14: Block Model Comparison Statistics

Zone	Domain	Capped Assay Count	Capped Assay Mean (g/t)	Composite Assay Count	Composite Assay Mean (g/t)	Parent Block Count	OK (g/t)	NN (g/t)
North Redline (Low Grade)	100	3,368	0.20	880	0.20	58,438	0.19	0.22
	103	81	0.20	23	0.20	1,246	0.17	0.23
	104	141	0.21	40	0.22	1,403	0.27	0.32
	105	100	0.15	30	0.15	1,299	0.17	0.19
	106	174	0.23	51	0.23	2,613	0.19	0.22
North Redline (High Grade)	200	1,177	0.94	303	0.93	6,495	0.85	0.84
	201	1,383	0.77	345	0.77	7,646	0.72	0.72
	202	827	0.75	214	0.75	7,679	0.71	0.70
	203	35	0.67	9	0.68	345	0.67	0.57
	205	69	0.51	20	0.51	652	0.51	0.51
	207	62	0.76	18	0.76	793	0.71	0.81
South Redline (Low Grade)	300	4,248	0.22	1,066	0.21	62,273	0.20	0.22
	102	515	0.31	132	0.31	12,842	0.22	0.26
	301	115	0.35	23	0.32	1,050	0.28	0.42
	302	406	0.21	106	0.21	4,993	0.13	0.22
	303	154	0.21	44	0.21	2,300	0.15	0.18
	304	236	0.20	66	0.21	3,659	0.18	0.23
	305	161	0.22	42	0.22	5,202	0.18	0.23
South Redline (High Grade)	400	2,869	0.79	703	0.78	12,828	0.74	0.75
	600	2,317	0.69	574	0.69	17,402	0.72	0.75
	800	1,060	0.66	263	0.65	11,932	0.58	0.73

Figure 14-13: Swath Plot Across Domains 200 and 201 at North Redline

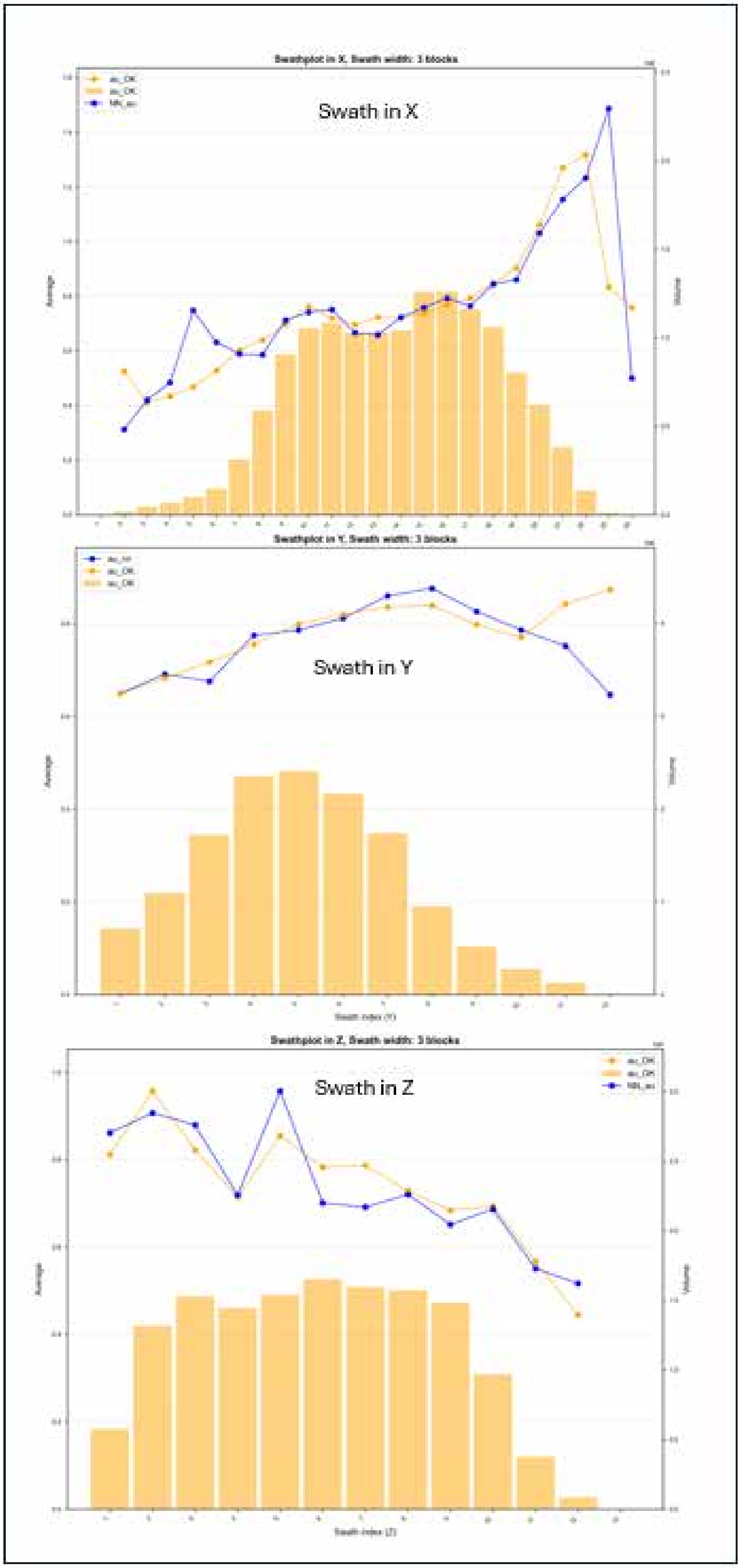
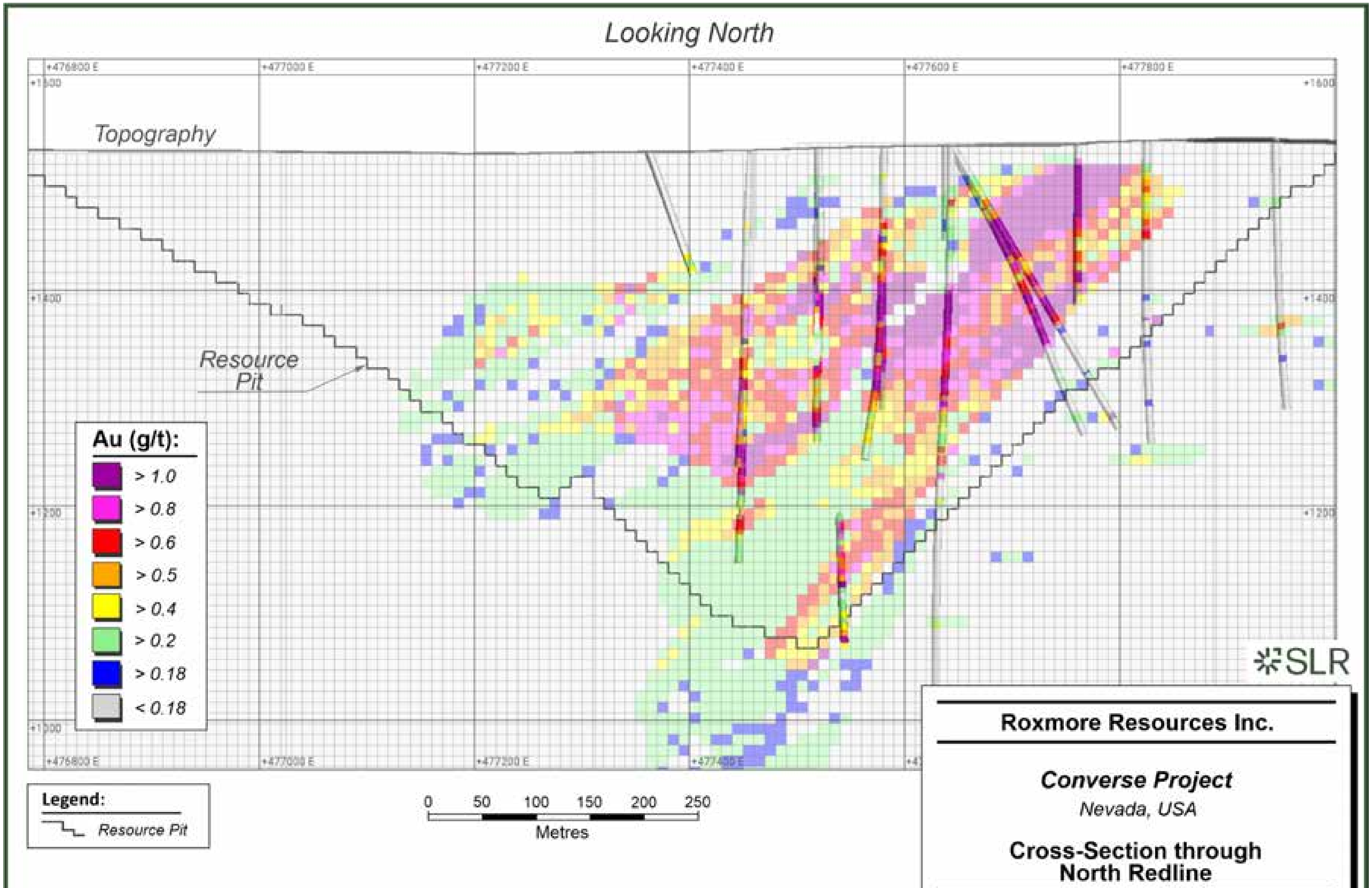
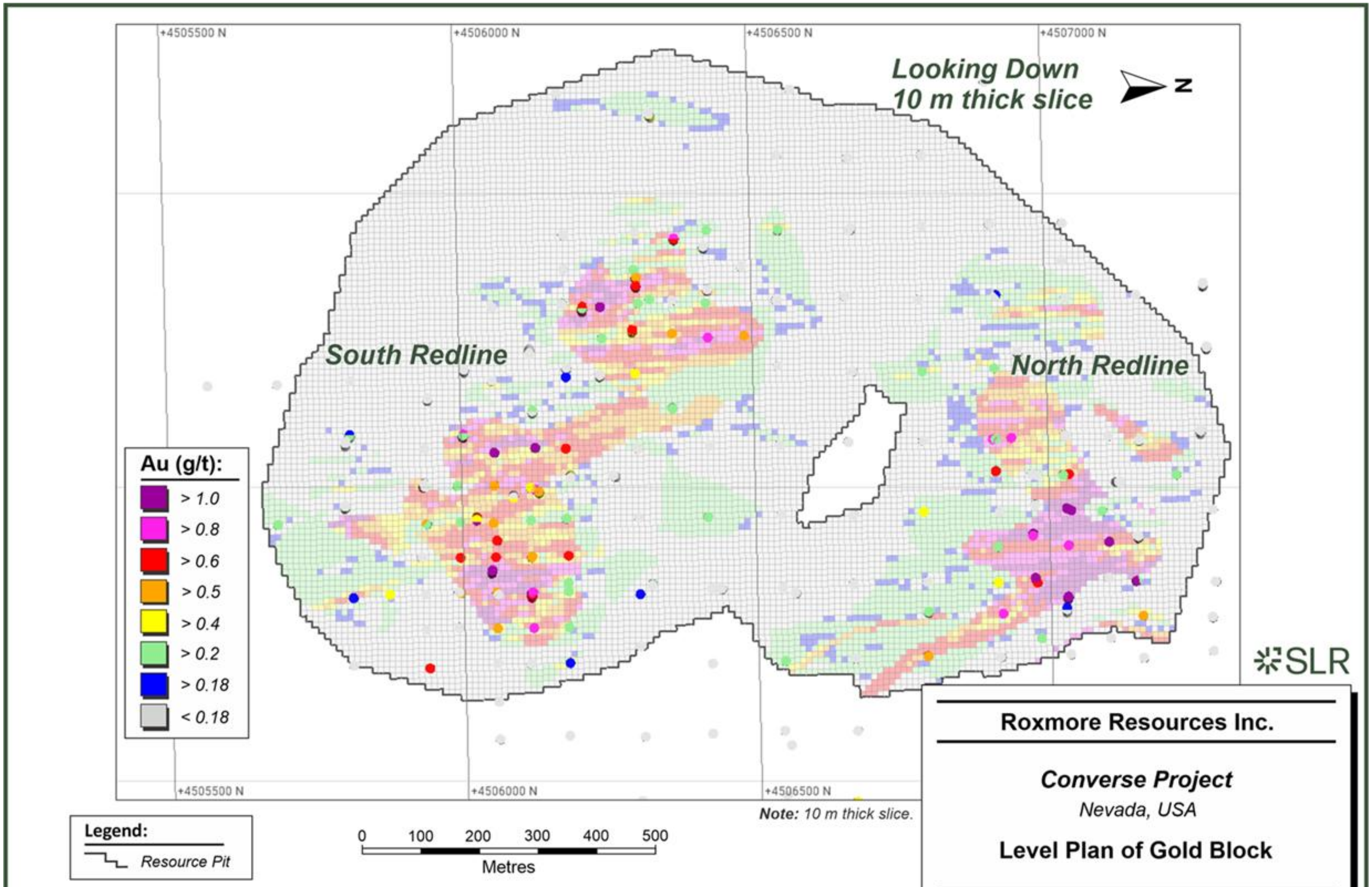


Figure 14-14: Cross-Section through North Redline



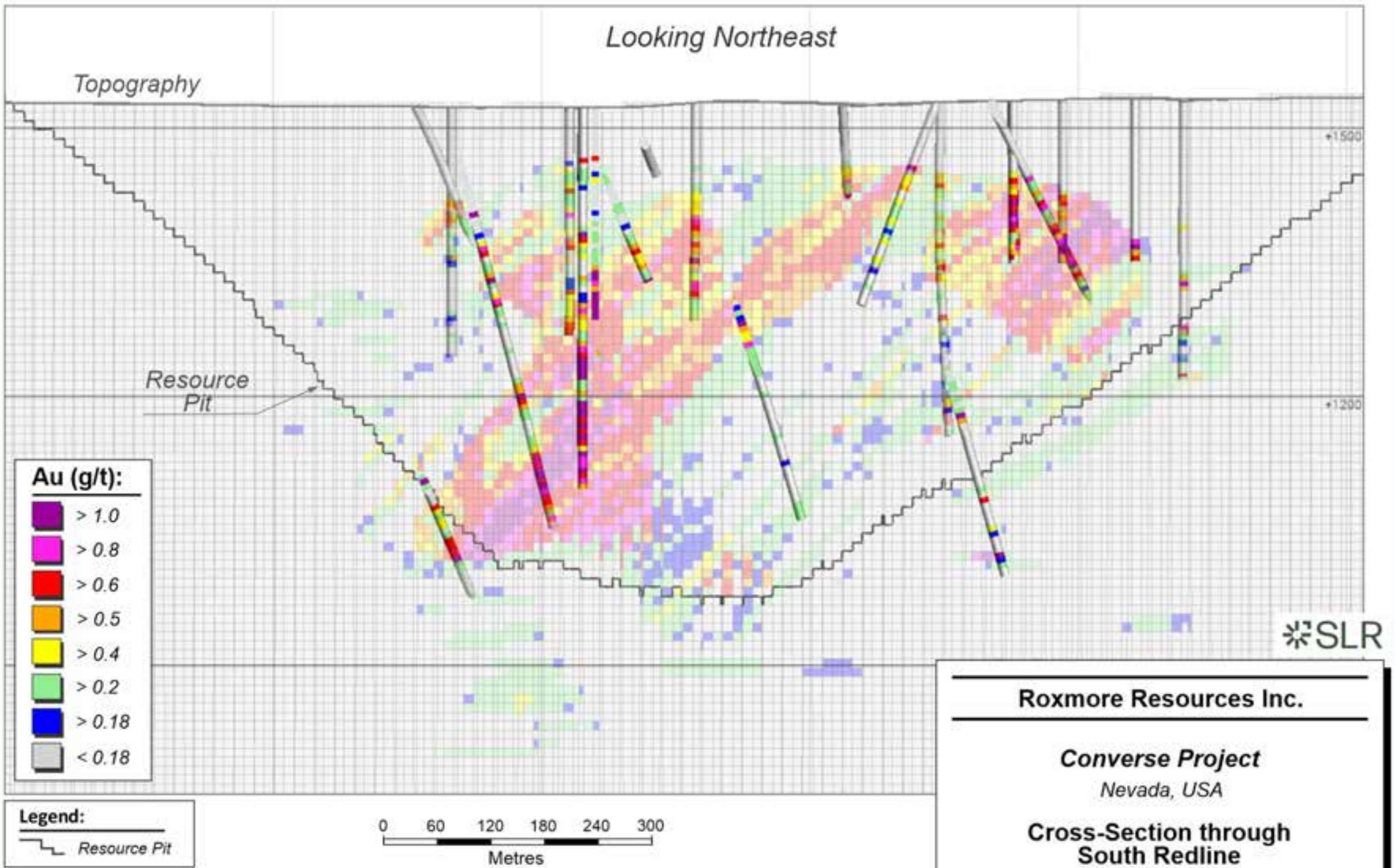
Source: SLR 2026.

Figure 14-15: Level Plan of Gold Blocks (10 m thick slice)



Source: SLR 2026.

Figure 14-16: Cross-Section through South Redline



Source: SLR 2026.

14.13 Mineral Resource Reporting

The Mineral Resource estimate for the Project is reported by North or South Zone and by redox domain, as shown in Table 14-15. Open pit Mineral Resources are constrained within an optimized pit shell and reported above a 0.18 g/t Au or 0.20 g/t Au cut-off value. In the opinion of the QP, the assumptions, parameters, and methodology applied in preparing the Converse Project Mineral Resource estimate are appropriate for the style of mineralization. The effective date of the Mineral Resource estimate is March 31, 2026.

Table 14-15: Summary by Area and Redox of the Mineral Resource – March 31, 2026

Category	Zone	Redox	Cut-Off Value (g/t Au)	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	North Redline	Oxide	0.20	5,101	0.67	110
		Transition	0.18	21,710	0.69	480
		Primary	0.20	13,628	0.77	340
		Total		**40,439**	**0.72**	**930**
	South Redline	Oxide	0.18	26,026	0.61	507
		Transition	0.18	30,158	0.60	579
		Primary	0.18	6,479	0.70	146
		Total		**62,663**	**0.61**	**1,232**
Inferred	North Redline	Oxide	0.20	13,641	0.39	172
		Transition	0.18	35,281	0.37	420
		Primary	0.20	29,205	0.47	440
		Total		**78,127**	**0.41**	**1,032**
	South Redline	Oxide	0.18	49,817	0.34	550
		Transition	0.18	72,395	0.48	1,109
		Primary	0.18	18,108	0.59	343
		Total		**140,319**	**0.44**	**2,002**

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are reported within an optimized pit shell at a cut-off grade of 0.18 g/t Au or 0.2 g/t Au, dependent on zone and redox domain (See Table 14-13).
3. Mineral Resources are estimated using a long-term gold price of US$3,000/oz.
4. Gold recoveries are based on equations derived from metallurgical studies.
5. Recoveries are variable and based on equations derived from metallurgical testing.
6. Bulk density is variable by rock type and redox domain and ranges from 2.62 t/m^3 to 2.71 t/m^3. Overburden was assigned a value of 1.8 t/m^3.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. A minimum mining width of 50 m was used.
9. Numbers may not add or multiply correctly due to rounding.

14.14 Factors Affecting the Mineral Resource

Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The QP is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issue that may have a material impact on the Converse Mineral Resource estimate.

Factors that may affect the Converse Mineral Resource estimate include:

- Revisions to the redox model, as redox domains directly inform the assignment of gold recovery parameters within the resource estimation process.
- Tighter spaced drilling may lead to revisions of the geologic and redox models, particularly in areas where historical drill holes terminated within mineralized intervals and the full extent of alteration or oxidation boundaries remains unconstrained.
- Changes to the understanding of structural controls, including the roles of fault geometries, bedding orientations, and breccia development, in governing fluid flow and mineralization patterns.
- Additional metallurgy and modelling of copper zones and their effect on gold recovery.
- Metal price and exchange rate assumptions.
- Changes to the assumptions used to generate the cut-off grades used for the construction of the mineralized wireframe domains.
- Changes to the input and design parameter assumptions that pertain to the creation of open pit volumes.
- Changes to the assumed metallurgical recoveries.

15.0 Mineral Reserve Estimates

This PEA has been prepared in accordance with NI 43-101 and CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions). The study is conceptual in nature and does not support the declaration of Proven or Probable Mineral Reserves. Mineral Reserves can only be declared upon completion of at least a PFS demonstrating economic viability. The PEA includes Inferred Mineral Resources, which are subject to a high degree of uncertainty.

16.0 Mining Methods

16.1 Overview and Mining Strategy

The Converse Project is conceptualized as a large-scale, open pit mining operation utilizing a conventional truck-and-shovel fleet. The mine plan is engineered to support a 14.5-year life of mine (LOM) through the extraction of approximately 299.8 Mt of mineralized material and 877.6 Mt of waste rock. This results in a steady-state production strip ratio of 2.71:1 (waste:mineralized material) during active operations, and a cumulative LOM strip ratio of 2.93:1 when including initial pre-production stripping.

16.1.1 Strategic Objectives and Production Targets

The primary objective of the mining strategy is to maximize the Project's net present value (NPV) by prioritizing the extraction of higher-grade material in the early years of operations while maintaining a consistent total material movement (TMM). Following a nine month pre-production period dedicated to waste stripping and stockpile establishment, the operation targets a steady-state leach pad feed rate of 22.5 Mtpa (5.625 Mt per quarter).

Key strategic drivers include:

- Front-Loaded Grade Profile: Gold grades remain above the LOM average of 0.51 g/t through Year 8, accelerating project payback.
- Operational Scale: The mining fleet is sized to manage a peak annual capacity of 90 Mtpa, ensuring sufficient vertical advance rates to sustain production.
- Metallurgical Sequencing: Initial production focuses on the North Redline sector, transitioning in Year 4 to include the South Redline sector, where gold recoveries consistently exceed 70%.

16.1.2 Pit Optimization and Phasing Logic

Mine designs are based on a resource block model utilizing 10 m x 10 m x 10 m blocks, which aligns with the selected 10 m bench height and equipment reach. Pit optimization was conducted using a gold price of $3,000/oz, employing a conservative reporting cut-off grade (ranging from 0.18 g/t to 0.20 g/t Au) to ensure a margin of safety against market volatility.

To optimize the extraction sequence and manage stripping requirements, the mine is developed in three distinct phases:

- Phase 1 (North Redline): Independent starter pit providing early, high-grade mineralized material.
- Phase 2 (South Redline): Independent southern starter pit targeting higher recovery material.
- Phase 3 (Final Pit): The ultimate pit expansion that merges the North and South sectors, reaching a maximum depth of 620 m.

16.1.3 Geotechnical and Hydrogeological Framework

The pit geometry is dictated by the depth of the alluvium and the structural characteristics of the bedrock. Stability is maintained through a single-benching configuration with a 50° bench face angle in alluvium and 80° in bedrock. Final designs incorporate inter ramp angles of 37° to 38° for alluvium and 46° to 49° for bedrock.

As the pit extends below the natural water table, the strategy incorporates active dewatering through peripheral wells and site-wide water management. This depressurization is critical for maintaining the Factor of Safety (FoS) and ensuring efficient equipment trafficability.

16.1.4 Mining Method and Equipment Selection

The Project adopts an owner-operator model for primary mining activities to ensure direct control over unit costs and schedule execution. The fleet selection leverages a "pass-match" optimization strategy between large-scale loading tools and 240-tonne payload haul trucks. The selected equipment for different stages of the mine plan include the following:

- Waste Stripping: Handled by high durability 33.6 m³ cable shovels for maximum throughput.
- Mineralized Material Recovery: Managed by 29.8 m³ hydraulic shovels, providing the breakout force and selectivity required to minimize dilution.

Haulage logistics are optimized through a dual-gradient strategy: a 10% in-pit gradient to minimize waste stripping and an 8% ex-pit gradient on the waste rock storage facilities (WRSF) to maximize mechanical availability and fuel efficiency for the 240-tonne truck fleet.

16.1.5 Labour and Costing Basis

Personnel requirements and operating costs were derived using the Costmine Intelligence SHERPA 2025 Dataset (CostMine 2025). This provides a defensible, benchmarked basis for both hourly and salaried labor. To ensure a clean economic cut-off, mining labour is strictly categorized into operations and technical oversight, while broader administrative and environmental roles are accounted for separately under general and administrative (G&A) expenses.

Detailed descriptions of mine-related facilities, including truck shops, fuel storage, and haul road infrastructure, are provided in Section 18.0.

16.2 Geotechnical Analysis and Parameters

16.2.1 Overview and Source of Data

Geotechnical slope design recommendations for the South Redline deposit within the Project were provided by CNI in a study dated February 2008. The Slope angles are conceptual and subject to change.

The recommendations are based on a 2007 geotechnical field program designed to characterize the rock mass and alluvium properties. The data collection included:

- Drilling: Five primary geotechnical holes, including two alluvium holes for 362 m (1,189 ft) and three oriented-core holes for 1,084 m (3,555 ft) to determine structural orientations.
- Supplemental Logging: Geomechanical logging of an additional 737 m (2,419 ft) of bedrock core from previous drilling campaigns.
- Testing: Comprehensive laboratory testing including uniaxial compression, Brazilian disk tension, and direct-shear tests on both alluvium and bedrock fractures.

16.2.2 PEA Design Criteria and Block Model Alignment

The following mine design parameters for this PEA were selected to ensure technical and operational alignment with the resource block model:

- Bench Height: A 10 m bench height was adopted for all pit shells. This height was selected to match the 10 m vertical block size of the resource block model.
- Sub-blocking: The resource block model was constructed without sub-blocking; therefore, the 10 m bench height is utilized to ensure that mining selectivity and grade control align exactly with the resource estimation units.
- Berm Configuration: To maintain a conservative stability profile and simplified operational management, a catch bench (berm) is placed every 10 m (single benching). No double benching was utilized for the current PEA pit optimization.

16.2.3 Alluvium Slope Design

The alluvium thickness ranges from 15 m to 244 m (50 ft to 800 ft). CNI identified that stability in these units is primarily governed by the potential for block failure through weak clay units near the bedrock/alluvium contact.

- Stability Factors: CNI's analysis of a 107 m (350 ft) high alluvium slope at a 38° inter ramp angle yielded a FoS ranging from 1.17 to 1.26.
- PEA Adoption: For the purpose of the pit optimization, an overall slope angle (OSA) of 37° to 38° was adopted for the alluvium. To achieve this at a 10 m bench height, the design utilizes a 50° bench face angle and a 4.4 m (14.4 ft) catch bench width.

16.2.4 Bedrock Slope Design

The bedrock consists of high strength Paleozoic meta-sediments with uniaxial compressive strengths (UCS) ranging from 181 MPa to 278 MPa (26,265 psi to 40,386 psi). Slope failure is expected to be controlled by structural orientations rather than rock-mass failure.

- Structural Fabric: A dominant west-dipping fault and joint fabric (dipping 55° to 75°) dictates bench stability.
- PEA Adoption: While CNI's original recommendation for the South Redline starter pit was 43° based on a 12.2 m (40 ft) bench, an OSA of 45° was selected for the bedrock during optimization. This 45° angle assumes that the 10 m single bench configuration, combined with controlled blasting and rigorous structural mapping, will achieve the necessary stability.

16.2.5 Summary of Design Parameters

Table 16-1 summarizes the geotechnical design inputs used for the PEA pit shell generation.

Table 16-1: Geotechnical Design Summary

Detail	Bench Height (m)	Overall Slope Angle (degrees)
Alluvium	10	38°
Bedrock	10	45°

16.2.6 Geotechnical Risks and Recommendations

The current geotechnical model is based on a limited dataset (five holes) originally intended for South Redline starter pit. As the Project has evolved into a large-scale operation exceeding one billion tonnes, the following recommendations are critical for advancing the Project to Pre-Feasibility Study (PFS) and Feasibility Study (FS) levels:

- Expanded Geotechnical Drilling Program: The current density of five holes is insufficient for a one billion tonne pit. A comprehensive drilling program is required to characterize the deeper bedrock units and the lateral extent of the alluvium. This program should focus on:
  - Redline Stock: Since this unit was not intersected in the 2007 program, dedicated holes are needed to confirm its strength and stability, as it will form the toe of the slope in central sectors.
  - Deep Bedrock Structures: Drilling must reach the ultimate depths of the final pit to identify any deep-seated shear zones or faults.
- Structural Domain Modeling: As the pit footprint grows, the west-dipping structural fabric may change orientation. Additional oriented core is needed to build a 3D structural domain model for the north, south, and west walls.
- East Wall "Nose" Stability: The sharp nose geometry between the North and South Redline deposits remains a high risk area for tensile failure. Detailed mapping and specialized stability modeling are required for this sector.
- Hydrogeological Investigation: With a significantly deeper pit, the impact of groundwater and pore-water pressure becomes a major factor. Piezometers must be installed to determine the phreatic surface and its effect on the FoS.
- In situ Alluvium Mapping: During the excavation of the upper benches, the alluvium must be mapped in real time to identify the lateral extent of weak lacustrine clay units.

16.3 Hydrogeology and Hydrology

16.3.1 Hydrogeology and Geotechnical Considerations

Mining is expected to extend below the natural water table. For the purposes of this PEA the following assumptions are made:

- Depressurization: Active dewatering will be required to maintain the FoS for pit slope stability.
- Data Gap: Site-specific hydraulic conductivity and storage coefficients are currently being characterized in the 2026 field program.
- Operational Impact: Blasthole conditions and equipment trafficability are assumed to be managed through standard dewatering practices.
- Mining Cost Impact: Dewatering related cost increases will be further quantified and refined during the PFS/FS stages.

16.3.2 Dewatering and Water Management

The Project will implement a site-wide water management plan to handle both groundwater inflow and surface runoff. The water management plan includes the following components:

- Pit Dewatering: A network of peripheral wells will be phased with the mine production schedule. Captured water will be the primary source for the process plant.
- Surface Hydrology: Engineered diversions will be constructed to reroute ephemeral drainages. All facilities, including the heap leach pad and WRSFs, are sited to avoid major hydrological conduits.
- Storm Events: Infrastructure is designed to contain or divert a 24-hour, 100-year storm event per Nevada Division of Environmental Protection (NDEP) regulations.

16.3.3 Process Water Supply and Rights

Roxmore has secured the necessary water rights to support the Project's peak consumption, taking into account the following considerations:

- Water Balance: The site is expected to operate on a net loss basis due to evaporation on the leach pads. This deficit will be managed by a dedicated production well field.
- System Redundancy: The well field provides a redundant supply to ensure consistent leaching cycles, independent of fluctuations in pit dewatering volumes.

16.4 Cut-Off Grade Estimation

The Project is divided into North Redline and South Redline sectors based on its geological structure, specifically the depth of the bedrock, which is shallow in the North Redline and significantly deeper in the South Redline. Within these areas, the mineralized material is further classified as Oxide, Transition, or Primary based on the degree of weathering and the chemical state of iron and sulphur minerals. These classifications are critical because they dictate the metallurgical recovery rates and the associated processing costs. To define the economic limits of the deposit, the study utilizes a gold price of $3,000/oz to calculate the marginal (internal) cut-off grade. This marginal grade represents the threshold where the value of the recovered gold covers only the costs of processing and site G&A, assuming the material has already been extracted from the pit.

The specific calculations for these economic thresholds are detailed in Table 16-2 (North Redline) and Table 16-3 (South Redline). These tables define the Marginal Cut-off Grade (COG), representing the theoretical "break-even" point where the value of recovered metal equals the cost of processing and administration. This calculation is based on the following formula:

$$\text{COG (g/t)} = \frac{\text{Cost}_{\text{Process}} + \text{Cost}_{\text{G\&A}}}{(\text{Price}_{\text{Au}} - \text{Cost}_{\text{Sell}}) * \text{Recovery} * \text{Conversion}}$$

While this formula proves the baseline for economic viability, the Project adopts a more conservative Reporting Cut-off Grade for the Mineral Resource estimate, as shown in Table 16-4. This higher threshold was selected to ensure the estimate maintains RPEEE. By elevating the COG above the marginal limit, the Project establishes a necessary safety buffer against real-world economic volatility – such as rising reagent costs, energy price fluctuations, or minor variations in metallurgical recovery – ensuring that only higher-margin mineralization is included both in the resource estimate and the subsequent mine schedule.

Table 16-2: Theoretical North Redline Cut-Off Calculation

Parameter	Unit	Oxide	Transition	Primary
Gold Price	US$/oz	3,000.00	3,000.00	3,000.00
Total Selling Cost (Refining + Royalty)	US$/oz	(182.50)	(182.50)	(182.50)
Net Smelter Return (NSR) Price	US$/oz	2,817.50	2,817.50	2,817.50
Processing Recovery	%	60.0%	69.0%	58.0%
Unit Operating Costs (Process + G&A)	US$/t	6.50	6.50	6.50
Marginal Cut-off Grade	**g/t**	**0.12**	**0.10**	**0.12**

Table 16-3: Theoretical South Redline Cut-Off Calculation

Parameter	Unit	Oxide	Transition	Primary
Gold Price	US$/oz	3,000.00	3,000.00	3,000.00
Total Selling Cost (Refining + Royalty)	US$/oz	(182.50)	(182.50)	(182.50)
Net Smelter Return (NSR) Price	US$/oz	2,817.50	2,817.50	2,817.50
Processing Recovery	%	79.0%	76.0%	64.0%
Unit Operating Costs (Process + G&A)	US$/t	6.50	6.50	6.50
Marginal Cut-off Grade	**g/t**	**0.09**	**0.09**	**0.11**

Table 16-4: Reporting Cut-off Grades

Area	Oxide Cut-off (g/t)	Transition Cut-off (g/t)	Primary Cut-off (g/t)
North	0.20	0.18	0.20
South	0.18	0.18	0.18

By adopting a reporting grade above the calculated marginal limit, the Project applies a conservative threshold that provides a margin of safety for potential mining dilution, ensuring the Mineral Resource meets the NI 43-101 requirement for RPEEE. This approach ensures the Project remains robust under various market conditions.

It should be noted that the cut-off grades presented in the preceding tables are incremental process limits; they account for crushing, leaching, and G&A requirements but exclude primary mining costs. Throughout this analysis, mining costs are treated as a separate operational variable ($2.20/metric tonne), highlighting the low cost, bulk tonnage nature of the Converse deposit while maintaining a clear distinction between the costs of excavation and the economic thresholds of the leaching facility.

16.5 Pit Optimization and Shell Selection

16.5.1 Mineral Resource Block Model

The open pit optimization and conceptual production schedule are based on the Converse resource block model provided as an export from LeapFrog as “Conv_pea_bm_02022026.csv” with the model construction parameters as provided in “Conv_pea_bm_02022026.csv.txt”. For this evaluation, these data are imported into Maptek Vulcan™ and renamed “conv_pea_bm_02022026_reg.bmf”. Additional variables were added to this model, before exporting the same to Geovia Whittle™ for optimization purposes. The model utilizes 10 m x 10 m x 10 m block size.

16.5.2 Mining Dilution and Recovery

In accordance with the standards applied for this PEA, no external factors for mining dilution or the loss of mineralized material have been applied to the resource model. The pit optimization and subsequent production schedule assume a mining recovery of 100% and 0% external dilution. These parameters are conceptual in nature and will be subject to refinement in future FS level studies as specific mining equipment specifications, bench geometries, and grade control strategies are finalized for the Project.

16.5.3 Pit Optimization Parameters

The parameters used for the optimization are listed in Table 16-1 through Table 16-3. A range of gold prices from $600/ oz to $4,500/ oz, with a $100/oz increment, was used. The base price (Revenue Factor =1) of $3,000/oz was used for the reporting of Mineral Resources in Section 14.0.

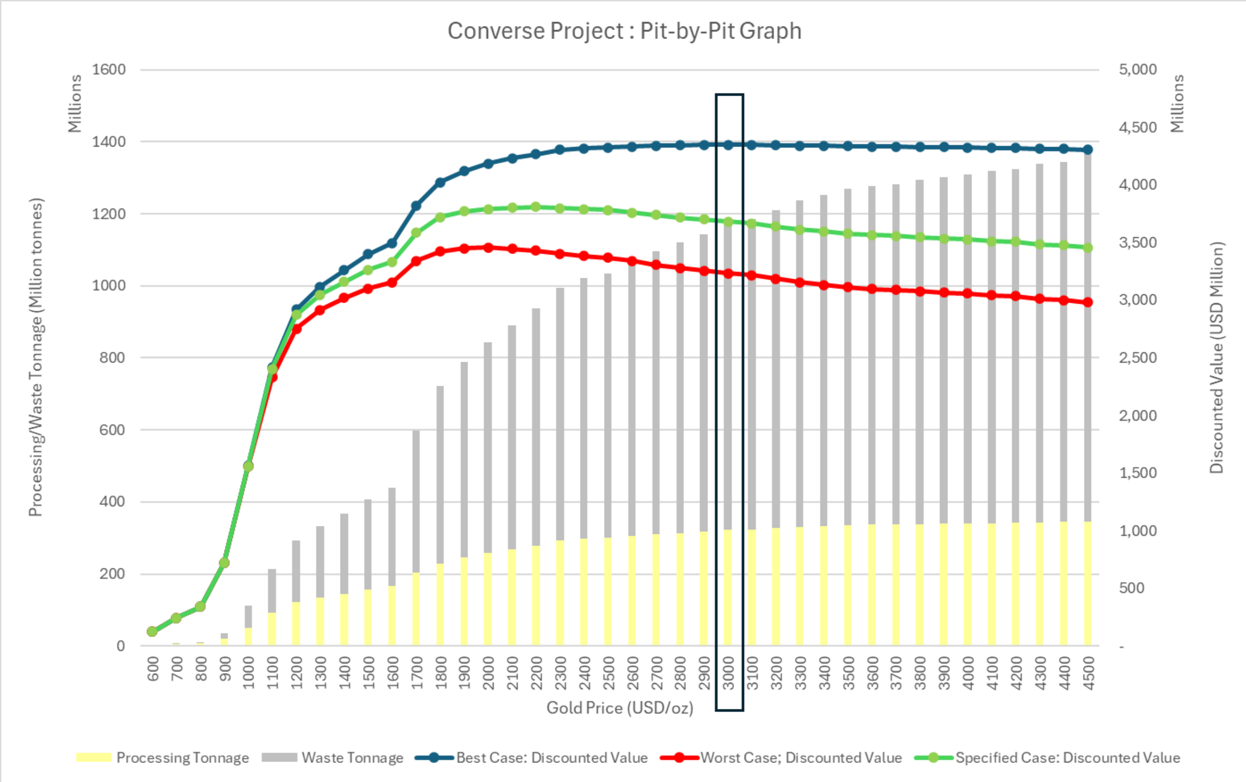
16.5.4 Pit Shell Results and Selection

The pit optimization was conducted using the Whittle™ software suite utilizing the Pseudoflow algorithm. The optimization process evaluated a range of revenue factors to define the optimal economic pit limits based on the raised cut-off grades as previously defined in Section 16.4

Key outcomes from the optimization process are shown in the pit-by-pit graph in Figure 16-1, and are described as follows:

- Pit shell geometry and total mined tonnage are largely insensitive to cut-off grade, with pit limits driven by slope and overall economics rather than marginal grades.
- Higher cut-off grades mainly affect in-pit material classification, reclassifying marginal mineralized material as waste.
- Strip ratio increases are driven by mineralized material reclassification and selective mining, not by pit expansion or additional stripping.
- Higher cut-off grades leave some lower-grade in-pit material unmined, reducing recoverable mineralized material tonnes and mine life while total material movement remains similar.

Figure 16-1: Converse Project – Pit-by-Pit Graph



16.5.5 Mine Production Rate and Capacity Optimization

To optimize the extraction of the 321.5 Mt mineralized deposit, a strategic evaluation was conducted to determine the ideal annual production rate. This selection process balanced the Project's strip ratio and NPV at a 5% discount rate against the physical constraints of the site.

16.5.5.1 Optimization Methodology

SLR performed a series of sensitivity analyses using Whittle software, maintaining a peak mining capacity of 90 Mtpa. To identify the point of maximum economic efficiency, the processing limits were tested in the following increments:

- Range: 15 Mtpa to 25 Mtpa
- Incremental Step: 2.5 Mtpa
- Primary Metric: NPV

The iterations aimed to find the "sweet spot" between two competing pit shell strategies:

- Maximizing Mine Life: Utilizing the Revenue Factor 1 (RF = 1) pit shell.
- Maximizing Value: Selecting the pit shell where incremental value creation no longer exceeds the 5% discount rate.

16.5.5.2 Final Selection and Practical Constraints

By refining the internal pit shell selections using both manual design and Whittle's Auto Pushback features, the Project incorporated a 50 m minimum mining width between stages.

Based on these economic and physical parameters, the target capacity was set at 22.5 Mtpa of mineralized material for leach pad loading.

Operational Validation

The selection of 22.5 Mtpa is substantiated by its alignment with the Project's logistical capabilities:

- Quarterly Output: Maintains a consistent production rate of 5.625 Mt per quarter.
- Vertical Advance: The rate is compatible with achievable bench sinking rates, ensuring that development stays ahead of production requirements (detailed in Section 16.7).

16.6 Pit Design and Phasing

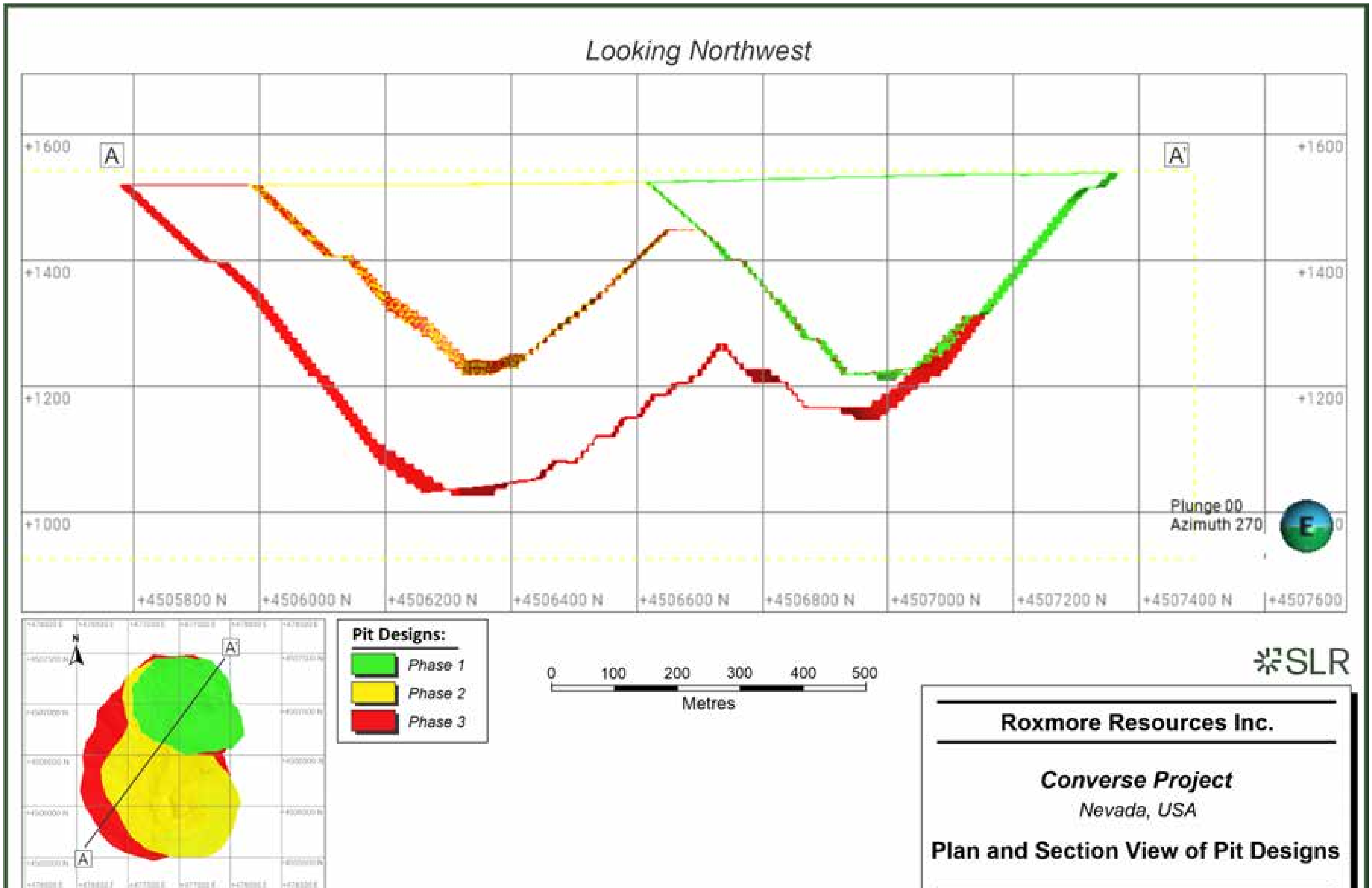
The open pit design and the initial pit phases have been chosen based on the objective of providing a feed rate of 22.5 Mtpa (as described in Section 16.5.5) as early in the LOM as possible and to sustain the same rate as long as possible through the end of the LOM schedule.

The design is executed in three distinct mining phases (pushbacks) to optimize the NPV and manage stripping requirements for the estimated 300 Mt of leach feed. Phases 1 and 2 are for North Redline and South Redline, respectively, and are independent of each other. The final phase is the Final Pit covering both the North and South Redline areas. This phased approach allows for early access to higher grade material while deferring capital-intensive waste stripping. The layout and cross-sectional geometry of these phases are illustrated in Figure 16-2, with detailed design parameters summarized in Table 16-5.

Table 16-5: Pit Design Parameters

Design Parameters	Unit	Phase1 - North Redline		Phase 2 - South Redline	Phase 3 - Final Pit		
Material Type		Alluvial	Bed Rock	Alluvial	Bed Rock	Alluvial	Bed Rock
Haul Road Width	m	30	30	30	30	30	30
Haul Road Gradient	%	10	10	10	10	10	10
Bench Height	m	10	10	10	10	10	10
Batter Angle	degrees	50.0	80.0	50.0	80.0	50.0	80.0
Berm Width	m	4.4	7.9	4.4	7.9	4.4	6.9
Inter-Ramp Angle	degrees	38.0	46.0	38.0	46.0	38.0	49.1

Figure 16-2: Plan and Section View of Pit Designs



Source: SLR 2026.

16.6.1 Design Geometry and Stability

The design of the pit is engineered to provide stability given the varying depth of alluvium present in the deposit. As detailed in Table 16-4, the architecture utilizes a 10 m bench height and 7.9 m catch berms in the interim phases. To manage the alluvium and the associated structural risks associated, a bench face angle of 50° along with a 4.4 m catch berm has been used, resulting in a 38° inter ramp angle. These parameters need to be confirmed and modified based on the geotechnical studies to be carried out during the PFS.

Primary access is provided via a 30 m wide dual-lane ramp system at a constant 10% gradient. To minimize unproductive waste stripping at depth, the ramp width is designed to reduce to 20 m (single lane) as the phases reach the bottom of the pit.

16.6.2 Sequencing Logic and Operational Constraints

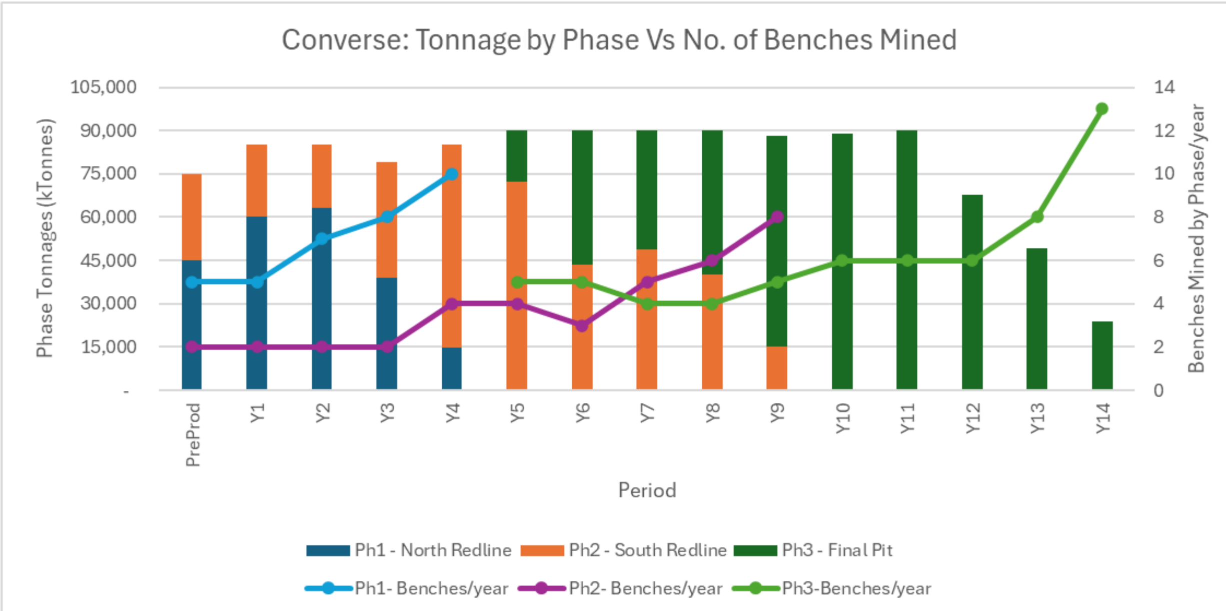
The design incorporates specific sequencing logic to ensure a stable production profile (as detailed in Section 16.7). The tonnage distribution by phase is shown in Table 16-6. Key design considerations include the following:

- Vertical Advance Limits: The phases are designed to accommodate a sinking strategy of not more than 10 benches per annum as shown in Figure 16-3. This physical constraint ensures the vertical development does not outpace the available working room; however, vertical advance naturally accelerates during the final year as the phase reaches its narrowest geometry at the pit bottom.
- Phase Staggering: The design utilizes staggered integration, allowing newer phases to provide waste stripping capacity while maturing phases provide the primary mill feed.

Table 16-6: Converse Project – Phase Details

Phase	Leach Feed Tonnes (kt)	Au (g/t)	Waste Tonnes (kt)	Total Tonnes (kt)	LOM Strip Ratio	Min RL	Max RL	Pit Depth (m)	Start Year	End Year
Ph1 - North Redline	73,110	0.60	148,954	222,065	2.04	1197	1547	350	PreProd	Y4
Ph2 - South Redline	108,412	0.54	298,495	406,907	2.75	1157	1537	380	PreProd	Y9
Ph3 - Final Pit	118,273	0.43	430,129	548,401	3.64	927	1547	620	Y5	Y14
Total	**299,795**	**0.51**	**877,578**	**1,177,373**	**2.93**	**927**	**1547**	**620**	**PreProd**	**Y14**

Figure 16-3: Tonnage Mined and Sinking Rate per Phase



16.6.3 Pit Design Reconciliation

The conceptual mine plan is based on an engineered pit design that translates economic optimization into a practical operational layout. This design incorporates haul road gradients, safety berms, and minimum mining widths.

The strategic guidance for the design was provided by the maximum value pit shell ($2,450 gold price / RF=0.817); however, the requirement for a 30 m wide dual-lane ramp system and stable bench geometries necessitated an expansion of the final pit's physical footprint. Consequently, there is a scheduled variance between the $3,000 pit shell (RF=1.0) used for Mineral Resource reporting and the final engineered design used for the production schedule, as detailed in Table 16-7.

Table 16-7: Converse Project – Pit Shell versus Pit Design Variance

Metric	Units	Pit Shell used for Mineral Resource Reporting (RF=1.0)*	Final Engineered Pit Design used for Production Schedule	Variance
Mineralized Material	Mt	321.5	299.8	-6.8%
Gold grade	g/t	0.50	0.51	1.3%
Waste Tonnage	Mt	837.0	877.6	4.9%
Total Tonnage	Mt	1,158.5	1,177.4	1.6%
LOM Strip Ratio	w:o	2.60	2.93	12.5%

*Note: Pit Shell (RF=1.0) tonnages are derived from Leapfrog™ volumetric reporting of the $3,000 pit shell. These tonnages vary slightly (<1%) from the raw Whittle™ optimization outputs shown in Figure 16-1 due to differences in block model interpolation and surface triangulation between software suites.

The final design captures 299.8 Mt of mineralized material, representing 93.2% of the resources identified within the $3,000 optimal pit shell. This -6.8% variance in mineralized tonnes is primarily due to the exclusion of marginal material along the pit walls to maintain geotechnical stability and accommodate the ramp system.

Notably, while the design was guided by a lower price-factor shell to maximize value, the spatial requirements for infrastructure pushed the total design tonnage (1,177.4 Mt) slightly above the total tonnage of the $3,000 Resource shell (1,158.5 Mt). This expansion increased total waste movement to 877.6 Mt, resulting in a LOM strip ratio of 2.93:1, a 12.5% increase over the base shell.

While the design captures fewer mineralized tonnes than the $3,000 shell, the slight increase in gold grade to 0.51 g/t confirms the successful exclusion of lower grade peripheral material in favor of the high value material. Ultimately, this reconciliation ensures that the production schedule is grounded in a mineable, geotechnically sound geometry rather than a theoretical optimization shell.

16.7 Mine Schedule

SLR generated the conceptual production schedule in Runge XPAC™ using their Target scheduling feature to ensure alignment with the physical constraints of the pit design and the mine production rate and capacities as described in Section 16.5.5. The schedule is engineered to deliver a consistent 5.625 Mt per quarter from the beginning of Year 2 to the leach pad. This is achieved while keeping the total material mined in the range of 18.7 Mt to 24.7 Mt per quarter while ensuring the annual totals do not exceed 90 Mtpa. There is a shortfall in the tonnages delivered to the leach pad in Year 11, which can be optimized further during the upcoming higher level studies on the Project.

16.7.1 Schedule Summary and Pre-Stripping

The estimated LOM is 14.5 years. The initial nine months are dedicated to pre-production and the establishment of a leach feed stockpile (utilizing Measured, Indicated and Inferred Resources). This ensures a continuous feed for processing, which is scheduled for commissioning in Q3 of the pre-production period. From that point, the production ramps up reaching a targeted leach feed of 5.625 Mt per quarter from the start of Year 2. The annualized strategic extraction sequence and production targets of 22.5 Mtpa of leach feed are summarized in Table 16-8.

Table 16-8: Strategic Mine Schedule

Detail	Units	LOM Total	PreProd	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Converse LOM																	
Total Mined	Mt	1,177.4	75.0	85.0	85.0	79.2	85.0	90.0	90.0	90.0	90.0	88.0	89.1	90.0	67.9	49.3	23.8
Waste Total	Mt	877.6	72.4	71.1	62.5	55.7	63.4	67.5	67.5	66.9	67.5	65.5	66.6	71.6	45.4	26.8	7.2
Waste - Alluvial	Mt	376.1	66.8	47.4	25.1	36.3	45.2	33.3	45.9	37.3	28.5	9.8	0.6	0.0	0.0	0.0	0.0
Waste -- Bed Rock	Mt	501.5	5.5	23.7	37.4	19.5	18.2	34.2	21.6	29.6	39.0	55.7	66.0	71.6	45.4	26.8	7.2
LOM Strip Ratio	w:o	2.93:1	27.6:1	5.1:1	2.8:1	2.4:1	2.9:1	3:1	3:1	2.9:1	3:1	2.9:1	3:1	3.9:1	2:1	1.2:1	0.4:1
Mineralized Material	Mt	299.8	2.6	13.9	22.5	23.5	21.6	22.5	22.5	23.1	22.5	22.5	22.5	18.4	22.5	22.5	16.6
Au Grade	g/t	0.51	0.81	0.62	0.57	0.59	0.51	0.51	0.52	0.55	0.53	0.47	0.31	0.34	0.41	0.59	0.61
Contained Gold	Moz	4.91	0.07	0.28	0.41	0.45	0.35	0.37	0.38	0.41	0.39	0.34	0.23	0.20	0.30	0.43	0.33
Recovered Gold	Moz	3.48	0.04	0.18	0.26	0.29	0.24	0.28	0.29	0.31	0.29	0.25	0.16	0.14	0.21	0.31	0.23
Met Recovery	%	70.9%	61.7%	64.5%	63.7%	64.2%	69.5%	76.8%	76.5%	75.6%	74.9%	72.8%	71.1%	68.6%	71.1%	72.4%	71.8%
Ph1- North Redline																	
Total Mined	Mt	222.1	45.0	60.0	63.2	39.0	14.9										
Waste Total	Mt	149.0	42.4	46.1	40.7	16.5	3.3										
Waste - Alluvial	Mt	65.1	38.9	22.4	3.8	0.0	0.0										
Waste -- Bed Rock	Mt	83.8	3.4	23.7	36.9	16.5	3.3										
LOM Strip Ratio	w:o	2:1	16.2:1	3.3:1	1.8:1	0.7:1	0.3:1										
Mineralized Material	Mt	73.1	2.6	13.9	22.5	22.5	11.5										

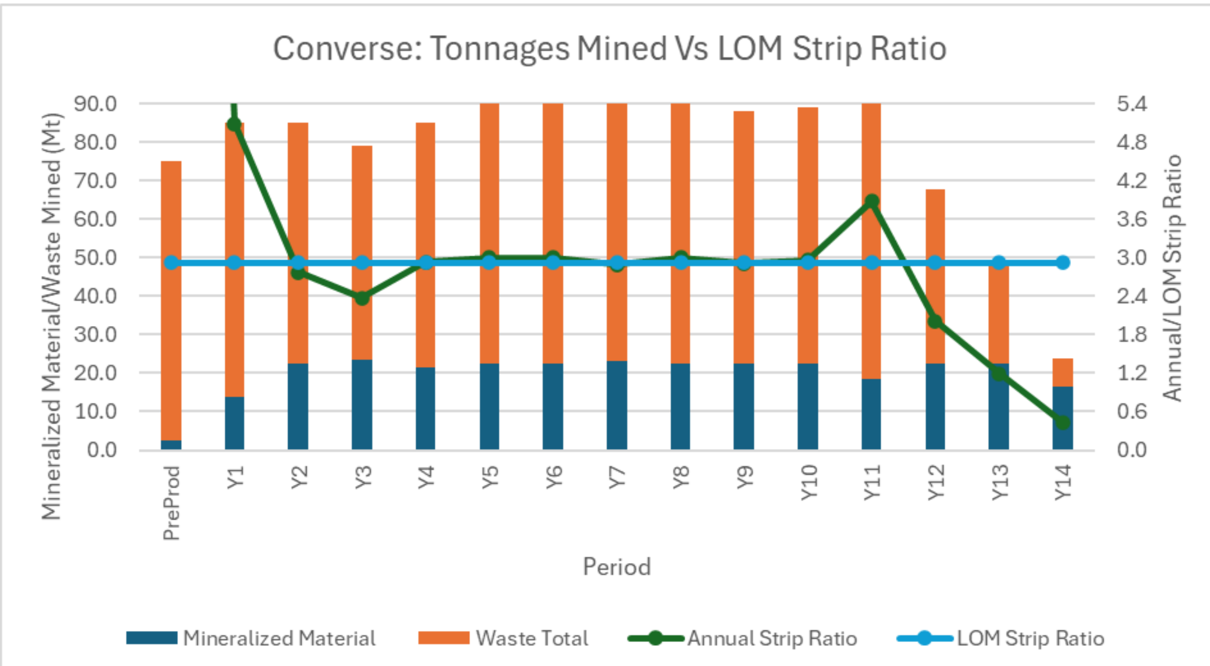
Detail	Units	LOM Total	PreProd	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Au Grade	g/t	0.60	0.81	0.62	0.57	0.61	0.57										
Contained Gold	Moz	1.41	0.07	0.28	0.41	0.44	0.21										
Recovered Gold	Moz	0.90	0.04	0.18	0.26	0.28	0.13										
Met Recovery	%	63.7%	61.7%	64.5%	63.7%	63.9%	63.3%										
Ph2- South Redline																	
Total Mined	Mt	406.9	30.0	25.0	21.8	40.2	70.2	72.2	43.6	48.8	40.2	15.0					
Waste Total	Mt	298.5	30.0	25.0	21.8	39.2	60.1	49.7	21.1	26.3	21.6	3.7					
Waste - Alluvial	Mt	171.6	27.9	25.0	21.2	36.3	45.2	15.9	0.1	0.0	0.0	0.0					
Waste -- Bed Rock	Mt	126.9	2.1	0.0	0.6	3.0	14.9	33.8	21.0	26.3	21.6	3.7					
LOM Strip Ratio	w:o	2.8:1	No Feed	No Feed	No Feed	39.2:1	6:1	2.2:1	0.9:1	1.2:1	1.2:1	0.3:1					
Mineralized Material	Mt	108.4	0.0	0.0	0.0	1.0	10.1	22.5	22.5	22.5	18.6	11.2					
Au Grade	g/t	0.54				0.31	0.43	0.51	0.52	0.56	0.58	0.63					
Contained Gold	Moz	1.87				0.01	0.14	0.37	0.38	0.40	0.35	0.23					
Recovered Gold	Moz	1.43				0.01	0.11	0.28	0.29	0.30	0.26	0.17					
Met Recovery	%	76.3%				79.0%	78.8%	76.8%	76.5%	75.7%	75.8%	75.2%					
Ph3- Final Pit																	
Total Mined	Mt	548.4						17.8	46.4	41.2	49.8	73.0	89.1	90.0	67.9	49.3	23.8
Waste Total	Mt	430.1						17.8	46.4	40.6	45.9	61.8	66.6	71.6	45.4	26.8	7.2
Waste - Alluvial	Mt	139.4						17.4	45.8	37.3	28.5	9.8	0.6	0.0	0.0	0.0	0.0

Detail	Units	LOM Total	PreProd	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Waste -- Bed Rock	Mt	290.7						0.4	0.6	3.4	17.5	52.0	66.0	71.6	45.4	26.8	7.2
LOM Strip Ratio	w:o	3.6:1						No Feed	No Feed	67.4:1	11.9:1	5.5:1	3:1	3.9:1	2:1	1.2:1	0.4:1
Mineralized Material	Mt	118.3						0.0	0.0	0.6	3.9	11.3	22.5	18.4	22.5	22.5	16.6
Au Grade	g/t	0.43								0.28	0.30	0.30	0.31	0.34	0.41	0.59	0.61
Contained Gold	Moz	1.63								0.01	0.04	0.11	0.23	0.20	0.30	0.43	0.33
Recovered Gold	Moz	1.16								0.00	0.02	0.07	0.16	0.14	0.21	0.31	0.23
Met Recovery	%	71.0%								68.7%	66.7%	67.8%	71.1%	68.6%	71.1%	72.4%	71.8%

16.7.2 Material Movement and LOM Strip Ratio

As illustrated in Figure 16-4, the annual tonnage versus strip ratio profile demonstrates a stable material movement throughout the LOM. From Year 2 through Year 11, the operation maintains a steady-state leach feed production target of 22.5 Mtpa. This is achieved by adhering to a consistent strip ratio close to 2.93 (LOM strip ratio) for most of the mine life, indicating the importance of keeping ahead on waste-stripping to enable predictable material movement throughout the majority of the schedule.

Figure 16-4: Tonnage Mined versus LOM Strip Ratio

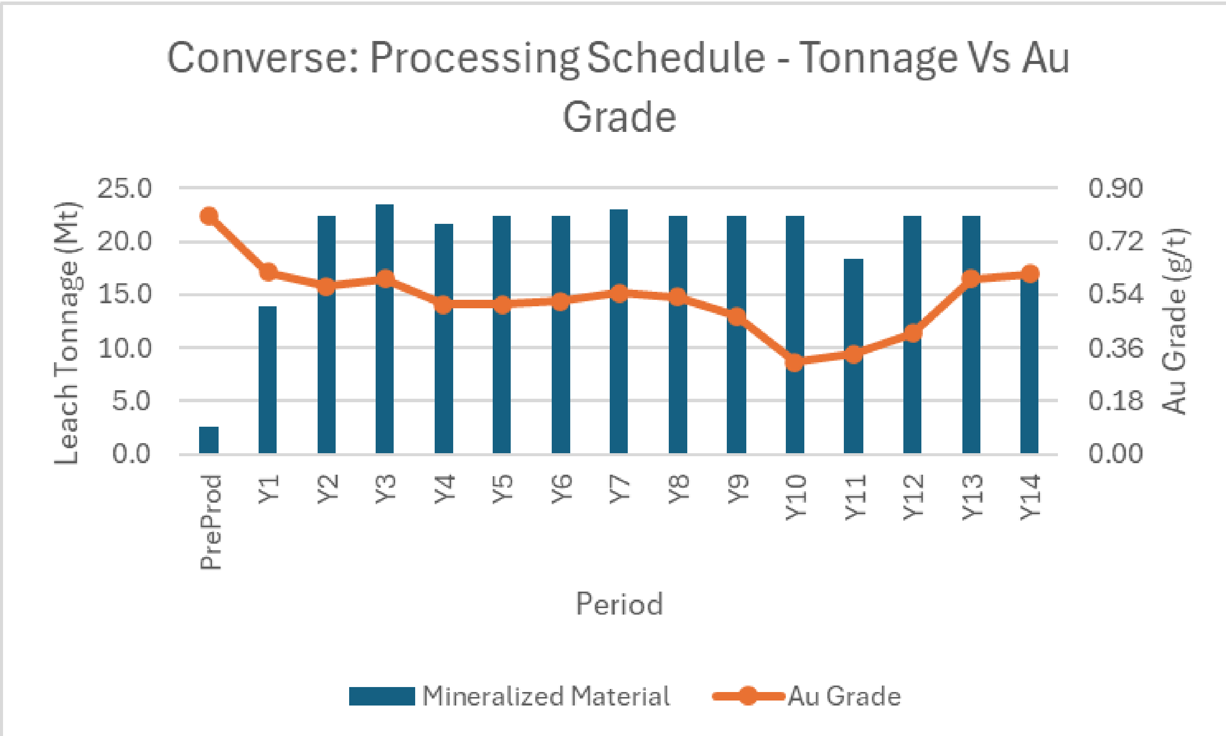


16.7.3 Mineralized Material tonnage and Grade Profile

The mine schedule, as illustrated in Figure 16-5, has been developed to support a consistent and sustainable mineralized material delivery profile throughout the central operating period. Leach feed tonnage is maintained near the nominal processing rate of approximately 22.5 Mtpa from Year 2 through Year 10, reflecting a sequencing strategy designed to ensure reliable throughput and efficient utilization of the processing infrastructure. Pre-production and Year 1 tonnages are modestly lower due to normal operational ramp-up, while Years 11 through 14 exhibit minor variability associated with the final stages of pit development.

As shown in Figure 16-5, gold grades remain above the LOM average grade of 0.51 g/t through approximately Year 8. This front loaded grade profile enables the Project to bring higher grade material forward in the schedule, resulting in increased contained gold during the early years of operation. Delivering above-average grades during this period strengthens early cash flow, accelerates project payback, and enhances overall economic performance. The combination of stable tonnage delivery and strategically sequenced higher grade feed establishes a strong foundation for the Project's economic robustness during the core years of the mine life.

Figure 16-5: Tonnage and Grade Profile

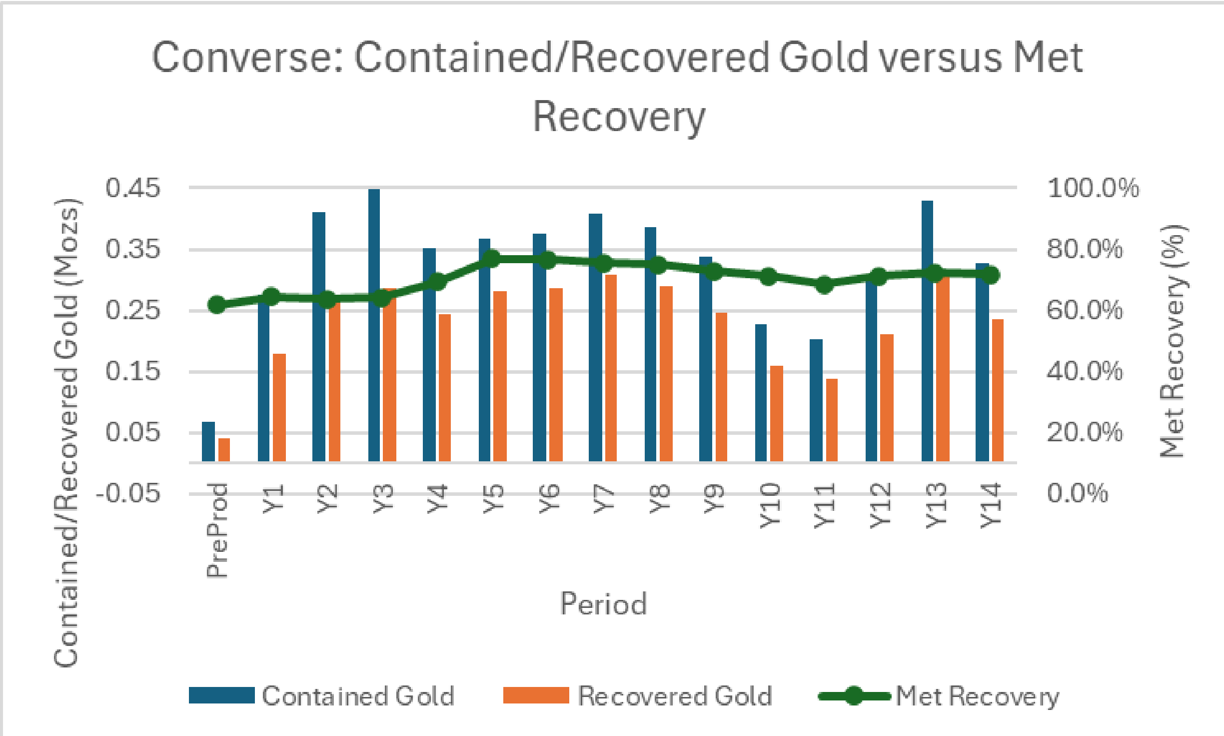


16.7.4 Gold Production and Metallurgical Recovery

Recovered gold production remains above 240 koz/yr through the central operating period (Years 2 to 9), as shown in Figure 16-6. Metallurgical recoveries are lower in the early years when mining is primarily focused on the North Redline area, which provides the initial supply of mineralized material. Beginning in Year 4, production benefits from the inclusion of material from South Redline, where recoveries consistently exceed 70% (refer to Section 16.4). This transition contributes to the increase in annual gold output during the mid-mine life period.

From the start of Year 10, mining is limited to the Phase 3 final pit. This phase initially progresses through zones characterized by lower grades and reduced recoveries before advancing to the deeper portions of the pit, where both grade and metallurgical performance improve toward the end of the mine life.

Figure 16-6: Contained/Recovered Gold and Metallurgical Recovery



16.8 Waste Rock Storage Facilities

The waste rock management strategy for the Project, developed by SLR, centers on a primary WRSF located to the north and west of the ultimate pit. This placement was selected to optimize haulage efficiency and was integrated with the broader site infrastructure layout described in Section 18.0. The facility is designed to be constructed in phases to align with the mining sequence.

16.8.1 Waste Rock Storage Facility (WRSF) Design

The WRSF design utilizes a standard sequential lift-and-berm construction methodology. The facility has been positioned to minimize haulage distances and associated operational costs—a critical factor given the LOM strip ratio of 2.93:1 (waste: mineralized material). Managing such a significant volume of waste requires a highly disciplined, spatial strategy to prevent cost overruns as the pit deepens.

Based on the engineering parameters summarized in Table 16-9, the WRSF maintains an OSA of approximately 18.3°. This relatively shallow profile is intentionally designed to enhance long-term geotechnical stability and significantly reduce future closure costs, as the configuration facilitates efficient progressive reclamation and minimizes the earthmoving required for final re-contouring.

Furthermore, the haul road design incorporates a maximum gradient of 8%, which is optimized for the hauling fleet. This gradient ensures that 240-tonne payload haul trucks can maintain efficient uphill speeds and optimal cycle times. In the context of a 2.93:1 LOM strip ratio, these marginal gains in cycle time efficiency accumulate into substantial operational savings over the LOM by reducing fuel consumption and mechanical strain on the drivetrain and braking systems.

Table 16-9: WRSF Parameters

Design Parameters Waste Dump	Unit	Value
Haul Road Width	m	40
Haul Road Gradient	%	8
Lift Height	m	30
Batter Angle	degrees	35
Berm Width	m	48

16.8.2 Material Volume and Storage Capacity

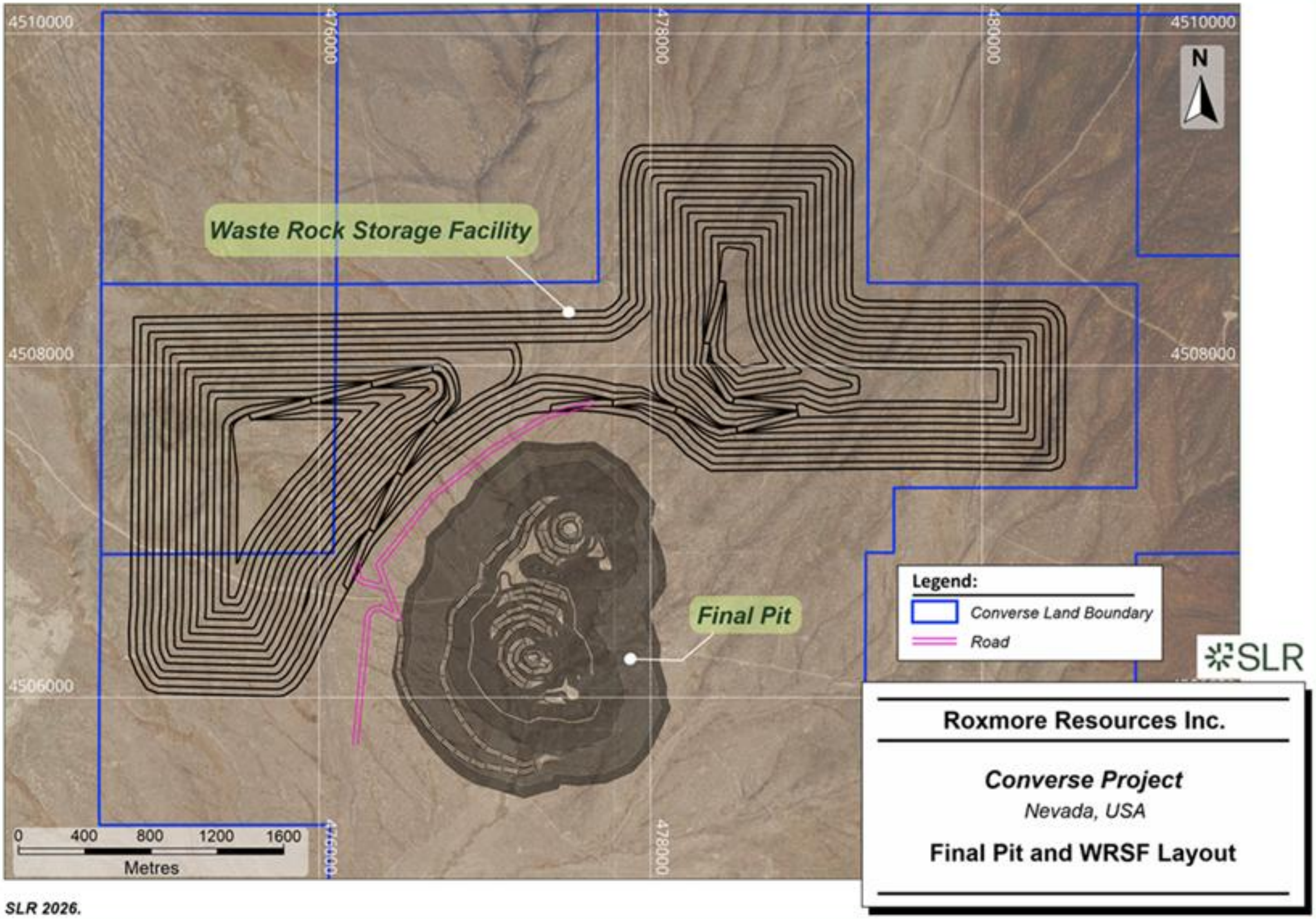
The waste rock quantities and planned storage requirements are detailed in Table 16-8. The mine plan accounts for the storage of two primary waste streams:

- Alluvial Waste**:** 376 Mt with an average in situ density of 1.8 t/m³.
- Bedrock Waste**:** 502 Mt with an average in situ density of 2.55 t/m³.

The WRSF design provides a total storage volume of 507 Mm³, which incorporates a swell factor of 25%.

The spatial relationship between the final pit design and the adjacent WRSF is shown in Figure 16-7. Based on the current site layout, additional storage requirements can be accommodated through expansion to the south of the western dump or further to the west. These areas provide flexibility to adjust for material-specific swell factors to be refined in future study phases.

Figure 16-7: Final Pit and WRSF Layout



Source: SLR 2026.

16.9 Mining Equipment

The equipment selection and operational strategy for the Project are designed to execute a 14.5 year LOM schedule. The fleet requirements were determined by SLR using a combination of first-principles engineering and the Costmine Intelligence Sherpa Estimation tool for Surface Mining. Various combinations of drilling, blasting, loading, and hauling configurations were evaluated to identify an optimal balance between initial capital expenditure (CAPEX) and long-term operating costs (OPEX).

The selected fleet configuration and associated productivities are engineered to sustain a TMM ranging from approximately 80 Mtpa to 90 Mtpa (228,500 tpd to 257,100 tpd). This capacity ensures that the mining operation can meet the required sinking rates and production targets across multiple active phases simultaneously.

16.9.1 Operational Strategy and Owner-Operated Model

The Project will utilize an owner-operator mining model from the commencement of operations. This strategic decision was made to ensure direct control over unit costs, operational precision, and the consistent execution of the engineered mine schedule. Internal management of the mining fleet provides the Project with long-term cost stability and the flexibility to adapt to operational requirements.

To mitigate initial capital intensity, the equipment selection process assumed a leasing model for the primary fleet, the economic details of which are further described in Section 21.0.

16.9.2 Equipment Fleet Selection

The primary loading and hauling fleet has been sized to manage the high volume of material movement inherent in the 2.93:1 LOM strip ratio. Key considerations for the selection of the primary and ancillary equipment include the following:

- Production Capacity: Ability to maintain a peak TMM of 90 Mtpa.
- Compatibility: Matching loading tool capacities (e.g., large-scale hydraulic shovels) with the 240-tonne payload haul trucks.
- Operational Reliability: Selection of proven equipment with established mechanical availability in similar mining environments.
- Efficiency: Optimization of fuel consumption and cycle times, specifically tailored to the 8% haul road gradients.

Details regarding the specific quantities and specifications for the primary mining fleet are provided in the following sub-sections.

16.9.3 Drilling and Blasting

The drilling and blasting parameters have been standardized across the operation to ensure consistent fragmentation and high productivity loading. The Project will utilize rotary drill rigs capable of drilling 200 mm to 229 mm diameter holes for both mineralized material and waste on 10 m high benches.

To ensure the structural integrity of the final highwalls, pre-split drilling will be implemented during the final excavation phases. For the purposes of this PEA, it is assumed that the primary rotary rigs will be utilized for wall control drilling by adjusting bit sizes and blast patterns as required. This strategy minimizes capital expenditure by maintaining fleet commonality. This assumption will be further refined in subsequent study phases through dedicated geotechnical programs and blast fragmentation trade-off studies to determine the optimal hole diameter and spacing. Blasting parameters are based on bedrock conditions, which govern the majority of drilling and blasting requirements.

The mine plan is based on a 10 m bench height, which aligns with the reach of the selected loading units and the geotechnical parameters of the bedrock. Blasting operations will primarily utilize Ammonium Nitrate/Fuel Oil (ANFO), with emulsion utilized in wet conditions. For initiation, the Project has selected electronic detonators to ensure precise timing, which optimizes fragmentation and minimizes ground vibration.

Production blastholes are assumed to be drilled on a nominal 6.5 m burden x 7.5 m spacing pattern, with approximately 1.0 m of sub-drill. Stemming is assumed at approximately 4.0 m, resulting in a loaded charge length of approximately 7.0 m per hole.

Given the anticipated groundwater conditions and the requirement for active pit dewatering, wet blasthole conditions are expected to be common. Accordingly, blasting is assumed to primarily utilize bulk emulsion or heavy ANFO–emulsion blends, rather than dry ANFO. Dry ANFO may be used selectively where dry hole conditions are confirmed.

Blasting parameters are based on bedrock conditions, which govern most of the drilling and blasting requirements within the pit.

Assuming an average in situ rock density of 2.55 t/m³, the selected pattern yields approximately 1,240 t of rock broken per blasthole. With an assumed explosive density of approximately 1.15 t/m³, the charge concentration is approximately 47 kg/m of loaded hole, resulting in approximately 330 kg of explosive per blasthole. This corresponds to an estimated powder factor of approximately 0.27 kg of explosive per tonne of rock blasted.

Electronic detonators are assumed for production blasting to improve timing precision, fragmentation control, vibration management, and final wall performance. Wall control blasting, including pre-split or trim blasting, will be implemented where required near final pit limits.

Although the Project is planned as an owner-operated mining operation, explosives supply, storage, delivery, and down-the-hole loading are expected to be provided by a qualified regional blasting contractor. Blast designs will be refined in future study phases based on geotechnical data, groundwater conditions, fragmentation requirements, and operating experience.

16.9.4 Loading and Hauling

The loading fleet is specialized to balance the differing requirements of mineralized material selectivity and high volume waste movement:

- Mineralized Material Loading: Hydraulic shovels equipped with 29.8 m³ bucket capacities will be used for mineralized material extraction. These units provide the breakout force and maneuverability required for maintaining material quality and bench precision.
- Waste Removal: Large-scale cable shovels with 33.6 m³ bucket capacities will be the primary movers for waste rock, providing the durability and high-volume throughput necessary to manage the LOM strip ratio.

Pass-Match Optimization

The primary hauling fleet consists of 240-tonne (payload) rear dump trucks, standardized across both mineralized material and waste streams to simplify maintenance and parts inventory. A key driver for the equipment selection was the optimization of the loading cycle through efficient pass-matching:

- Waste Loading: The 33.6 m³ cable shovel is sized to load the 240-tonne trucks in a 3-to-4 pass configuration. This high intensity match is designed to minimize truck "hang time," ensuring the high volume waste movement required by the 2.93:1 LOM strip ratio is maintained.
- Mineralized Material Loading: The 29.8 m³ hydraulic shovel is configured for a 4-to-5 pass match. The additional pass allows for greater operator control and selectivity at the mineralized material-waste interface, reducing dilution while still maintaining an efficient load time profile.

This synchronized pass-match strategy ensures that the loading units operate at peak rated capacity while minimizing the queuing times of the haul fleet, directly contributing to the achievement of the targeted 80 Mtpa to 90 Mtpa total material movement.

16.9.5 Productivity and Throughput

The primary loading fleet is comprised of two cable shovels dedicated to high volume waste stripping and one hydraulic shovel focused on mineralized material recovery and selective mining. This configuration is engineered to achieve a TMM of 80 Mtpa to 90 Mtpa.

These productivities are calculated based on 10 m bench heights and the material characteristics of the 1.18 billion tonnes of total material moved over the LOM. This includes the management of 376 Mt of alluvial waste, which exhibits lower in situ density and higher diggability compared to the bedrock waste, allowing for optimized cycle times in the upper sequences of the pit.

By utilizing large-scale equipment, the Project benefits from lower unit costs through economies of scale and reduced labour requirements per tonne moved. The fleet configuration has been modelled to maintain the necessary sinking rates across multiple active phases while providing the volumetric capacity required for the 2.93:1 LOM strip ratio.

16.9.6 Haulage Logistics and Ramp Geometry

The in-pit ramp system is engineered to facilitate safe and efficient cycle times for the 240-tonne payload haul fleet. Primary pit access is provided via a 30 m wide dual-lane ramp system designed at a constant 10% gradient. This gradient is a global industry standard for in-pit haulage, striking an optimal balance between the vertical rate of advance and the mechanical performance of the haul trucks.

The 30 m road width is specifically designed based on the 8.6 m overall canopy width of the selected 240-tonne truck class. This provides a safety factor of approximately 3.5 times the truck width, which is the industry standard for allowing safe dual-lane traffic, a safety berm, and a drainage ditch.

To optimize the Project’s economic performance and manage the 2.93:1 LOM strip ratio, a narrowing ramp strategy is employed. As the mining phases approach the final benches of the pit, the ramp width is reduced from 30 m to 20 m (single lane). This design choice is a critical volume reduction measure; by transitioning to a single-lane configuration at depth, the Project minimizes the requirement for unproductive waste stripping along the final pit walls. This ensures that the final tonnes of the mineralized material can be extracted with the lowest possible incremental strip ratio, supporting the Project's overall NPV.

Outside of the pit, the haulage network transitions to an 8% gradient for the WRSF. This shallower gradient is intentionally selected for long-term waste haulage for the following purposes:

- Maintain Peak Mechanical Availability: Reducing the thermal load on engines and wheel motors during the long, sustained uphill climbs typical of WRSF construction.
- Optimize Fuel Burn: Allowing the 240-tonne trucks to operate in a more efficient gear range, critical for the movement of the 1.18 billion tonnes of total material.
- Enhance Safety: Providing greater control for fully loaded trucks descending toward the primary crusher or waste dump tipping points.

16.9.7 Fleet Inventory

The preliminary mining fleet requirements are summarized in Table 16-10. This inventory represents a PEA-level estimate based on the current LOM production schedule and assumed equipment productivities. The equipment count presented for each period correspond directly to the annual material movement detailed in the Mine Production Schedule (Table 16-7), ensuring fleet capacity matches the loading and haulage demands for both mineralized material and waste.

This inventory focuses on the primary and key ancillary mining equipment and is not an exhaustive list of all minor support items. Process plant equipment is excluded in this section and is covered separately in Table 21-6.

The equipment count and specific configurations presented are subject to further refinement. As the Project progresses into feasibility-level studies, fleet requirements will be updated to reflect optimized haulage profiles, refined geotechnical constraints, and updated vendor-specific performance data.

Table 16-10: Converse Project – Mine Equipment Plan

Equipment	Make	Model	Size	Pre-Prod	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Mineralized Material																		
Hydraulic Shovel	Caterpillar	6060	29.8 m^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rear-Dump Truck	Caterpillar	793	240 t	3	4	8	8	9	9	10	10	11	11	11	11	11	11	11
Rotary Drill	Epiroc	PV 231	200 mm	2	3	4	4	4	4	4	4	4	4	4	4	4	4	4
Waste																		
Cable Shovel	Caterpillar	7495 HD	33.6 m^3	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Rear-Dump Truck	Caterpillar	793	240 t	11	15	17	17	21	21	25	25	26	26	26	26	12	12	1
Rotary Drill	Epiroc	PV 231	200 mm	4	9	8	8	9	9	9	9	9	9	9	9	5	5	1
Ancillary																		
Bulldozer	Caterpillar	D11	634 kW	4	5	6	6	6	6	6	6	6	6	6	6	4	4	4
Grader	Caterpillar	24	375 kW	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2
Water Tanker	Caterpillar	775G	53,000 L	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	Ford	F-600	22,000 lb	6	8	10	10	12	12	14	14	15	15	15	15	11	11	7
Bulk Truck	Orica	Bulk Master 7	450kg/min	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1
Lighting Plant	Allmand	Maxi-Lite	10.1 kW	5	5	5	5	5	5	5	5	5	5	5	5	4	4	4

16.10 Mine Personnel

The labour requirements for the Project were estimated using the Costmine Intelligence SHERPA software, utilizing the 2025 Dataset. This methodology employs an engineering-based approach to determine both hourly and salaried personnel counts.

16.10.1 Hourly Labor

The hourly labour count is derived from the primary equipment fleet requirements and the projected annual operating hours. SHERPA calculates the required headcount by applying the following parameters:

- Operating Factors: Direct operator requirements are based on the number of active equipment units and the established shift schedule.
- Maintenance Ratios: Maintenance personnel, including mechanics, electricians, and welders, are estimated based on equipment-specific labour-to-parts ratios and mechanical availability requirements.
- Availability Allowances: Headcounts are adjusted to account for typical industry availability, including vacation, training, and absenteeism, ensuring consistent coverage of the production schedule.

16.10.2 Salaried Labour

Salaried staff requirements are focused on the direct management and technical oversight of the mining operation, and include the following positions:

- Management and Supervision: This includes roles essential for daily pit operations, such as the Mine Manager, Mine Superintendent, and shift foremen. These counts are determined using standard "span of control" ratios relative to the size of the hourly workforce.
- Technical Services: Professional staffing for mine engineering, geology, and surveying is benchmarked against industry standards for surface mining operations of similar tonnage and complexity, as defined in the SHERPA 2025 Dataset.
- Exclusions: To avoid double-counting, administrative and support functions (e.g., environmental compliance, human resources, and accounting) are excluded from the mining labour count. These roles are accounted for separately as a fixed cost within the G&A expenses (expressed in $/t).

16.10.3 Cost Basis

All labour costs, including wages, salaries, and burden, are derived directly from the SHERPA 2025 Dataset. This ensures that the labour economic model is consistent with the most recent industry standard benchmarks available at the time of this report.

17.0 Recovery Methods

Metallurgical test work review and process design and development of capital and operating costs for the process plant were completed by KCA in Reno, Nevada. The following KCA documents were developed for the PEA.

- Summary Review of Metallurgical Test Data Converse Mining Project
- Converse Process PEA CAPEX OPEX_Option1 Rev D
- Converse Design Criteria Option 1_Heap Case Rev
- Converse Flowsheet Option 1 (9828-0113-11-100-C)
- Converse Design Criteria Option 2_Pulp Agglomeration Rev A
- Converse Flowsheet Option 2 (9828-0113-11-200-C)
- Converse Process PEA CAPEX OPEX_Option 2 Rev A

Test work results completed to date indicate that the minable Mineral Resource for the Project are amenable to cyanide leaching for the recovery of gold and silver. Based on the minable Mineral Resource of 299.8 Mt and established processing rate of 61,644 tpd, the Project has an estimated life of 14.5 years including ramp-up.

Mineralized material from the pit will be hauled to the processing site and crushed to 80% passing 6.3 mm at an average rate of 61,644 tpd using a three-stage crushing circuit. The crushed product will be stockpiled, reclaimed, and agglomerated with cement before being conveyed to the heap stacking system via an overland conveyor and grasshopper transfer conveyors. Agglomerated material will be stacked in 10 m lifts and leached with a dilute cyanide solution. Solution will flow by gravity to a pregnant solution pond before being pumped to a carbon adsorption circuit. Gold values will be loaded onto activated carbon and then periodically stripped from the carbon in a desorption circuit and recovered by electrowinning. The resulting precious metal sludge will be treated in a retort to recover mercury values, then smelted to produce the final doré product.

A summary of the processing design criteria is presented in Table 17-1.

Table 17-1: Process Design Criteria Summary

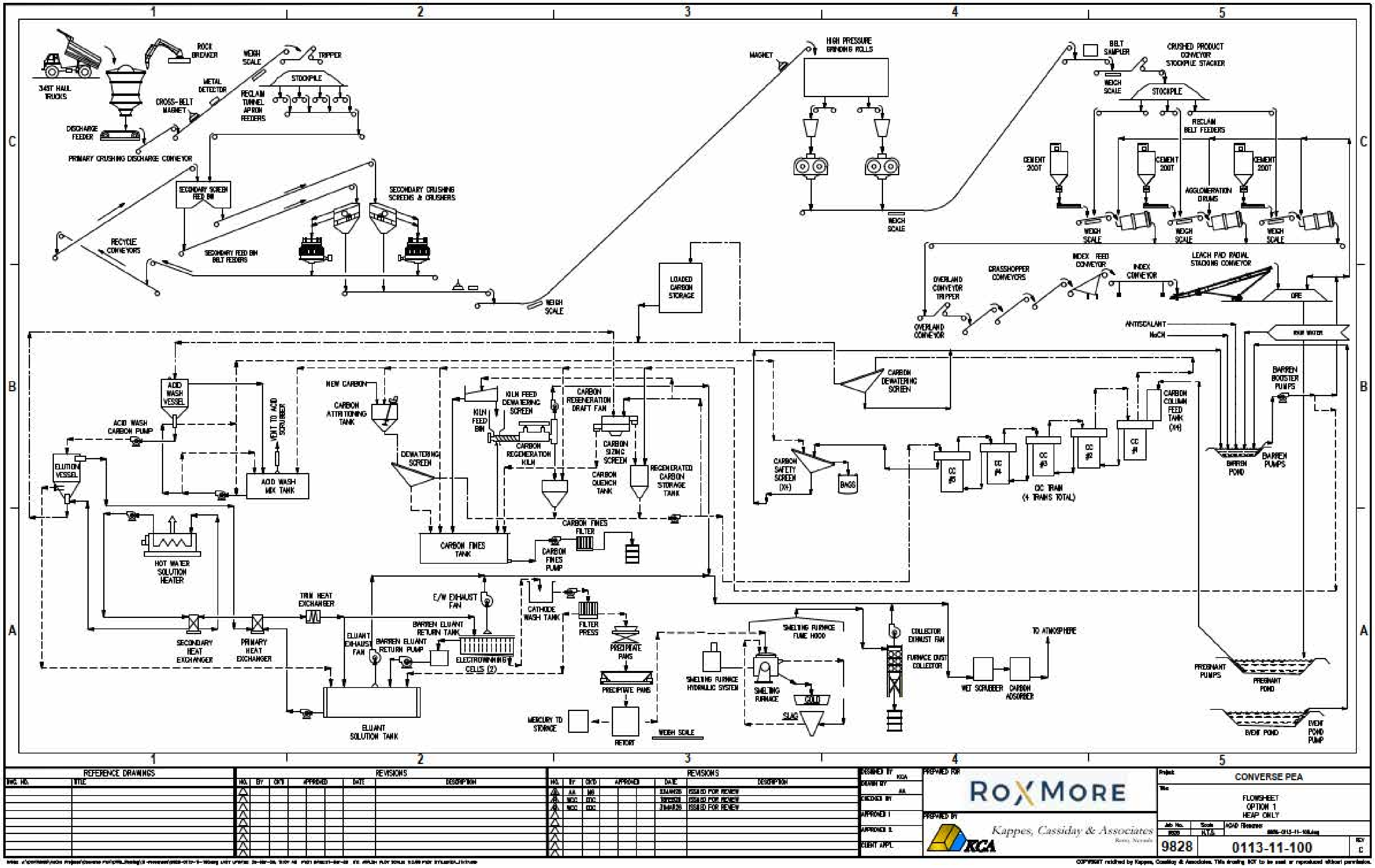
Item	Design Criteria
Annual Tonnage Processed	22.5 Mtpa
Crushing Production Rate	61,644 tpd average
Crushing Operation	12 hours/shift, 2 shifts/day, 7 days/week, 365 days/year
Crusher Availability	75%
Crushing Product Size	80% passing 6.3 mm
Conveyor Stacking System Availability	80%
Leaching Cycle	120 days
LOM Average Sodium Cyanide Consumption	0.64 kg/t
LOM Average Cement Consumption	6 kg/t
LOM Average Gold Recovery	70.9%

The overall process flowsheet is presented in Figure 17-1. The process layout is presented in Figure 17-2.

Electric power for the Project will be supplied from the grid.

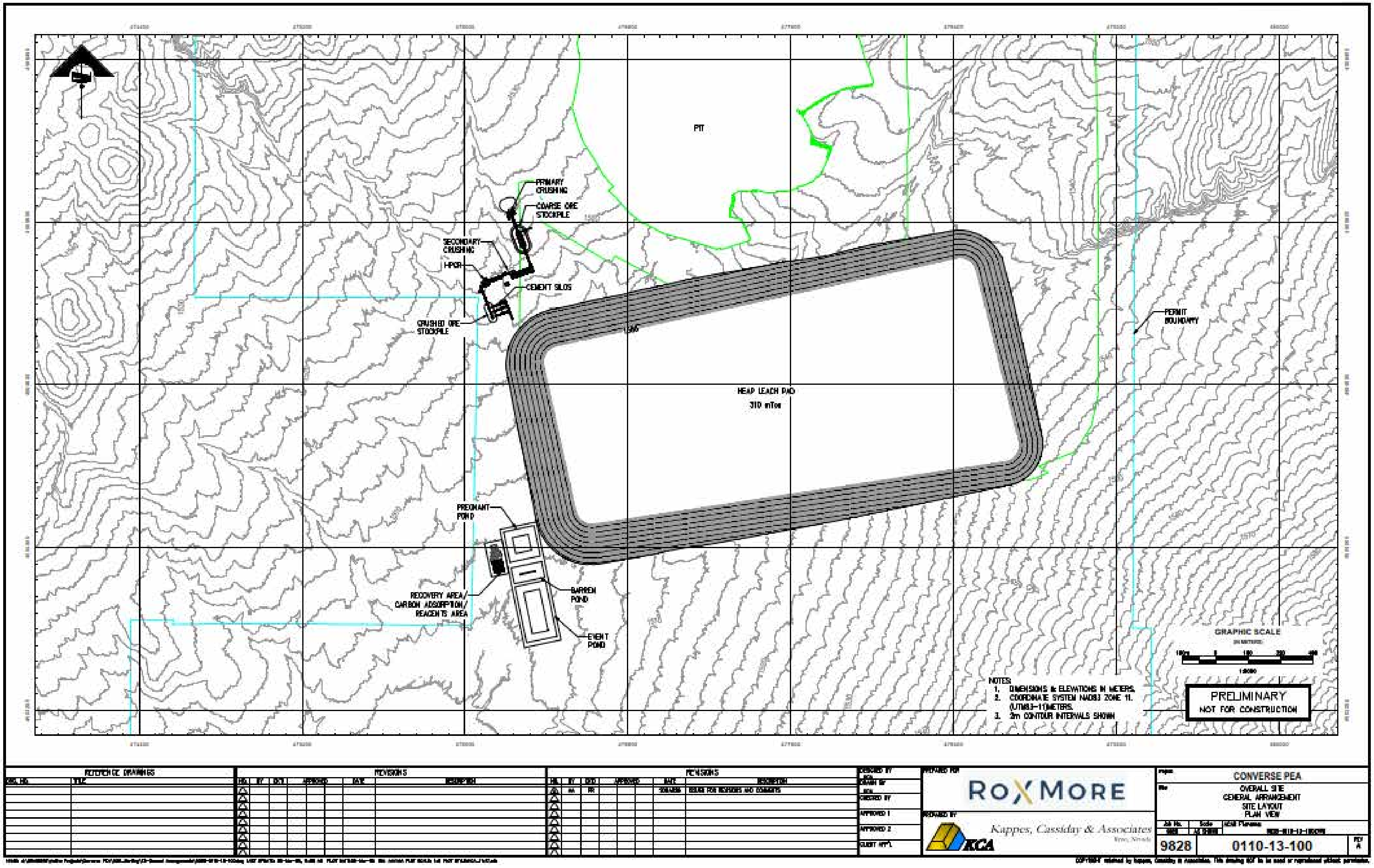
A pond is included to manage contact solution from storm events. Solution collected in the storm event pond will be returned to the process as makeup solution.

Figure 17-1: Overall Process Flowsheet



Source: KCA 2026.

Figure 17-2: Overall Process Layout



Source: KCA 2026

17.1 Crushing

The crushing circuit for the Project is designed to process 3,425 t/hr of material with an overall availability of 75% and will operate 365 days per year.

Run of mine (ROM) material will be transported from the mine pits in surface haul trucks and will either be directly dumped into the crusher dump hopper or stockpiled in a ROM stockpile near the primary crusher. Stockpiled ROM material will be reclaimed by a front-end loader and fed to the dump hopper as needed. The crushing plant will process an average of 61,644 tpd of material.

Material will primarily be fed directly into the dump hopper by haul trucks, which then feeds the primary gyratory crusher. The primary gyratory crusher will operate with a 165 mm discharge setting. Primary crushed material will be collected in the discharge pocket below the primary crusher. An apron feeder will regulate the primary crushed material discharge rate at an average rate (average rate including availability) of 3,425 t/hr onto the primary crushing discharge conveyor, which transfers material onto the primary stockpile feed conveyor. The primary stockpile feed conveyor will be equipped with a self-cleaning magnet and metal detector to protect downstream equipment from any tramp metal. Fogger type water sprays at the crusher discharge pocket and material transfer points will be used for dust control.

Primary crushed material will be stockpiled using a tripper conveyor mounted on the primary stockpile feed conveyor. The coarse material stockpile will have a live capacity of 17,979 t, approximately 7 hours at the nominal feed rate. Material will be reclaimed by apron feeders onto a primary stockpile reclaim conveyor and conveyed to the secondary crushing and screening circuit. The primary stockpile reclaim conveyor will be equipped with a self-cleaning magnet and metal detector to protect downstream equipment from any tramp metal.

The secondary crushing system will operate in a closed circuit with a product size of 100% passing 50 mm. Primary crushed material from the coarse material stockpile will be reclaimed and transferred to the secondary screen feed bin, where it is combined with the cone crusher product, reclaimed by the secondary feed bin belt feeders, and conveyed to the two secondary screens by the secondary screen feed conveyors. The secondary feed bins will be equipped with variable frequency drives (VFDs) to enable control and an accurate split of the crushed material between the screens. The secondary screening circuit includes two double deck vibrating screens with 100 mm and 50 mm top and bottom deck openings, respectively. Oversized material (+50 mm) will be fed to the secondary standard cone crushers, which will operate with a closed side setting of 50 mm. The cone crushers will discharge onto the secondary cone discharge conveyor, which will recycle the material back to the secondary screen feed bin via two secondary recycle conveyors (secondary recycle conveyors #1 and #2). Undersize material from the secondary screens will be transferred to the HPGR feed bin feed conveyor by the secondary screen undersize conveyor. The secondary screen undersize conveyor will be equipped with a self-cleaning magnet and metal detector to protect downstream equipment from any tramp metal.

The tertiary crushing system will include two HPGR crushers in parallel, operating in open circuit and producing a final crushed product with 80% passing 6.3 mm. Material from the HPGR feed bin will be reclaimed by two belt feeders, each feeding an HPGR crusher. The HPGR crushers will be fitted with variable-speed drive motors and will be choke-fed by material from the HPGR feed bin. Crushed material from the HPGRs will discharge onto the HPGR crushed product conveyor, then be transferred to the crushed product transfer conveyor, and finally stockpiled using the crushed product stockpile conveyor. Material from the crushed product stockpile will be reclaimed, agglomerated with cement, and conveyed to the heap stacking system.

All the conveyors will be interlocked so that if one conveyor is tripped, all upstream conveyors and the apron feeder will also stop. These features are considered necessary for safe operation and to meet the system's design utilization.

Water sprays will be located at all material transfer points to reduce dust generation by the crushing circuit.

17.2 Reclamation and Conveyor Stacking

Crushed material from the crushing circuit will be stacked onto a crushed product stockpile using a crushed product stacker conveyor equipped with a belt tripper. The crushed product stockpile will have three reclaim tunnels with three belt feeders, each feeding an agglomeration drum feed conveyor. Cement, which is required for heap permeability and pH control, will be metered from three cement silos (one for each agglomeration drum feed conveyor) directly onto the crushed material ahead of the agglomeration drums. Each cement silo will be equipped with a bin activator, variable speed rotary feeder, screw feeder, and dust collectors. Belt weigh scales will be installed on each agglomeration drum feed conveyor to control the cement feed rate.

Agglomeration of the material with cement will be accomplished using three agglomeration drums operated in parallel. The agglomeration drums will mix the crushed material and cement with barren process solution to produce agglomerates. The cement will bind fine particles to coarser particles, increasing the permeability and stability of the material stacked on the heap. The agglomeration drums will be installed within a lined containment area to contain any process solution or material that may be contacted by the process solution in the event of a spill or other disruption.

Agglomerated material from the agglomeration drum will discharge onto a common agglomeration drum discharge conveyor, where it is then transferred to the conveyor stacking system via an overland conveyor.

Agglomerated material from the agglomeration drum discharge conveyor will be fed onto an overland conveyor. The overland conveyor will run along the north side of the heap leach pad and will be equipped with a tripper conveyor, which will feed the material to the conveyor stacking system at the active stacking site of the heap.

The mobile conveyor stacking system will consist of twenty-two 46 m long grasshopper conveyors, six 46 m long ramp grasshopper conveyors, an index feed conveyor, a horizontal index conveyor, and a radial stacker. The horizontal index conveyor and radial stacker can retreat and stack material onto the heap. The number of grasshopper conveyors required will vary depending on the area of the heap being stacked, with a maximum of 28.

Each grasshopper and stacking conveyor will include an onboard transformer and an interlocked programmable logic controller (PLC) to allow for the addition or removal of conveyors. The master PLC will be installed on the radial stacker to initiate the conveyor start sequence. Each stacking system conveyor will include a strobe and horn alarm that will sound before the equipment starts up. The movement of the radial stacker and horizontal index conveyor will be controlled manually at the equipment. Each conveyor will be equipped with pull cords and emergency stops. If one conveyor in the stacking line is tripped, all upstream conveyors will also stop.

Once a lift of cells has finished leaching and is sufficiently drained, a new lift can be stacked over the top of the old lift. The old lift will be cross-ripped prior to stacking new material on top to break up any compacted or cemented sections. Stacked lifts will progress in a stair-step manner. The planned leach pad will have a total of eight lifts and a maximum planned height of 80 m.

17.3 Leach Pad Design

The Converse leach pad will be a single-use, multi-lift type heap and has been designed with a lining system in accordance with International Cyanide Code requirements and meets or exceeds the North American and NDEP standards and practices for lining systems, piping systems, and process ponds to minimize the environmental risk of the facilities impacting local soils, surface water, and groundwater in and around the site. The final pad design includes eight lifts and 310 Mt of material. Pad drainage will be constructed to transfer process solution to the pregnant solution pond.

The leach pad will be constructed by clearing the pad area, stripping vegetation and growth media, and grading to ensure drainage and heap stability. The leach pad liner will be composed of the following lining system from top to bottom:

- Overliner consisting of 600 mm of crushed and screened gravel
- 2 mm single side textured linear low-density polyethylene (LLDPE) geomembrane
- 600 mm of compacted soil liner with a minimum permeability of $1x10^{-7}$ cm/s
- Prepared subgrade

The planned leach pad will be constructed in multiple phases and will include 1.1 Mm^2 of initial lined area with a total LOM area of 3.4 Mm^2. Perforated gravity solution collection pipes will be installed on top of the geomembrane liner and covered with overliner material. These pipes are designed to operate at 50% full to contain the design production flows from the upgradient tributary area, allowing additional capacity to accommodate excess solution from storm events. The primary solution collection pipes will exit the heap through a concrete weir to the solution collection channel. The pipes will be solid walled as they enter the solution collection channel that flows into the pregnant pond.

Should solution flows exceed the capacity of the heap outlet pipes, solution head will build at the leach pad discharge area, causing excess solution to overflow the concrete weir into the solution collection channel.

The overliner material will act as a protective layer that resides above the LLDPE geomembrane. The main purpose of this material is to promote solution collection, reduce hydraulic heads over the liner, and protect the composite liner system and solution collection piping from damage during material placement.

A summary of the heap design parameters is presented in Table 17-2.

Table 17-2: Heap Leach Design Parameters

Parameter	Value
Material Feed Rate, tpd	61,644
Total Capacity, t	
Initial (Phase 1) Heap, Mt	103.3
Phase 2 Heap, Mt	103.3
Phase 3 Heap, Mt	103.3

Parameter	Value
LOM Heap Capacity, Mt	310
Lift Height, m	10
Quantity of Lifts	8
Maximum stacking height, m	80
Stacked Material Density, t/m^3	1.8
Front of Heap Slope, H:V	3.0
Side and Back Slopes of Heap, H:V	3.0
Angle of Repose, º	37
Leaching Cycle, d	120
Tonnes Under Leach, Mt	7.4
Active Leach Area, m^2	402,000
Solution Application Method	Buried Drip Emitters
Solution Application Rate, Nominal, L/h/m^2	10
Heap Irrigation Rate, Nominal, m^3/h	4,020
Heap Leach Material Moisture Retention, % of Total Material Weight	8.3

17.4 Solution Application and Storage

Process solution storage for the Project will include a pregnant, barren, and event/overflow pond. The event pond will be maintained empty or at low levels whenever possible. Solution diverted to the event pond will be returned to the system as make-up water as soon as practical with every effort made to avoid storing excess solution over a long period of time.

Crushed material will be leached in a single stage using a barren solution consisting of a dilute sodium cyanide solution; additional residual leaching of material will occur as the leach solution from higher lifts percolates downward. Barren solution will be pumped from the barren solution pond to the active leach site using a dedicated set of pumps and applied to the heap via a system of drip emitters. The barren pond will be equipped with submersible high flow pumps connected to horizontal booster pumps at the bank of the pond. The barren solution piping design considers insulated and heat-traced pipe to reduce the risk of freezing during winter operations. Buried drip emitters will be used for solution application and buried at least 0.3 m below the heap surface during winter. A barren solution will be applied to the heap at an average rate of 10 L/h/m^2. Based on metallurgical test work results, a leach cycle of 120 days has been estimated. Concentrated cyanide will be added to the barren solution pond by metering pumps to maintain the cyanide in solution at 200 ppm to 300 ppm NaCN. An antiscalant polymer will be continuously added to the leach solutions at an average rate of 6 ppm to reduce the potential for scaling in the irrigation system.

Pregnant leach solution containing gold values from the heap will drain by gravity to the pregnant solution pond. Pregnant leach solution leaving the heap will be transferred to the pregnant solution pond via pipe in a lined solution collection ditch. Pregnant leach solution will then be pumped to the carbon adsorption circuit by the pregnant solution pumps (four operating, one warehouse spare), where the gold and silver values will be adsorbed from the pregnant solution, and the resulting barren solution will then be returned to the barren solution pond.

During normal operations, the pregnant solution pond will be maintained in the mid-to-lower range of its working capacity, while the event pond will be kept empty or at low levels whenever possible. The pregnant, barren, and event ponds are designed to handle the flow from the LOM heap leach pad. The storm water storage capacity was evaluated under the following conditions:

- Pregnant solution storage for 12 hours at 4,020 m^3/h of solution
- A 24-hour heap drain down volume of leach solution (due to an event such as loss of power or pump), also at the solution application rate of 4,020 m^3/h
- Barren solution storage for 6 hours at 4,020 m^3/h of solution
- A 100-year, 24-hour storm event of 64 mm over the heap, pond, and solution collection ditch lined area
- Highest recorded monthly snowfall of 838 mm (83.8 mm water equivalent) over the entire lined area
- Dead storage volume assuming 1 m of slimes at the bottom for all ponds
- Freeboard of 0.5 m for all ponds

The resulting accumulation is 730,000 m^3 which can be accommodated in the event pond (444,250 m^3), the barren pond (78,600 m^3) and the pregnant pond (210,500 m^3).

The pregnant pond, barren pond, and event/overflow pond will each be equipped with a submersible high flow pump to return solution to the system. The submersible pumps will be mounted on pump slides on the pond side walls to facilitate their placement and extraction from the pond. An additional textured protective liner panel and conveyor belting will be installed on the pond sidewalls in the area where the pump slide is located to protect the pond liner. The ponds will be composed of the following composite liner systems from top to bottom:

- 2 mm smooth high density polyethylene (HDPE) primary liner
- Geonet
- 2 mm smooth HDPE secondary liner
- 0.6 m compacted soil liner with a permeability of $1x10^{-7}$ cm/s or geosynthetic clay liner (GCL)
- Prepared sub-grade

Leak detection pipes are provided beneath the primary pond liner to allow for monitoring and pumping of solutions from within the leak detection sumps.

The solution storage system will be designed so that the pregnant solution pond overflows to the barren solution pond, and the barren solution pond overflows to the event/overflow pond in the event of an emergency or a significant storm. The pond design considers normal working solution volumes entering the pregnant solution pond, ensuring that the event/overflow solution pond will be used very infrequently during operation.

17.5 Process Water Balance

17.5.1 Precipitation Data

Precipitation and evaporation data for the Project process water balance were obtained from the Battle Mountain Airport weather station and the Rye Patch Dam pan evaporation station, respectively, with adjustments made for elevation. Precipitation and evaporation data are presented in Table 17-3 in millimetres of water.

The 100-year, 24-hour storm event is estimated at 64 mm.

Table 17-3: Annual Precipitation and Evaporation Data

Month	Average Year Precipitation (mm)	Wet Year Precipitation (mm)	Dry Year Precipitation (mm)	Average Year Pan Evap. (mm)	Wet Year Pan Evap. (mm)	Dry Year Pan Evap. (mm)
January	26	46	12	27	23	31
February	19	32	9	49	41	55
March	22	39	10	95	80	108
April	28	49	13	128	107	146
May	29	51	13	179	149	203
June	13	23	6	229	191	260
July	4	8	2	272	227	309
August	4	7	2	250	209	283
September	11	19	5	179	149	203
October	19	33	9	133	111	151
November	21	36	10	52	43	58
December	26	46	12	22	18	25
Total	**222**	**389**	**103**	**1,615**	**1,348**	**1,832**

17.5.2 Water Balance

Based on the preceding precipitation and evaporation data, active water balances were calculated based on the requirement for processing 61,644 tpd of material. The model approximates the circulation of solutions within the heap leach and process facility, as well as precipitation and evaporation as functions of time. The results of the water balance model predict make-up water flow rates and operational control strategies necessary to achieve a zero-discharge system. The model is based on the ultimate leach area of the heap and normal operations.

The model uses monthly time steps, providing monthly average flow rates and volumes rather than peak daily or instantaneous rates. This approach may attenuate the peak rate by averaging volumes over a monthly period.

Water balance models were prepared based on average, wet, and dry precipitation years. Inputs for the water balance models are presented in Table 17-4. Pond evaporation is assumed to equal 60% of the pan evaporation over 50% of the pond area. Idle heap evapotranspiration is assumed to be 67% of the pan evaporation or rainfall, whichever is less, for the inactive heap area.

Table 17-4: Water Balance Model Inputs

Parameter	Unit	Input Values
Active Leach Area	m^2	402,000
Lined Pad/Ditch Collection Area (LOM)	m^2	3,394,327
Lined Pond Collection Area	m^2	77,496
Total Nominal Flow to Heap	m^3/h	4,020
Wet Season Material Moisture	%	3
Dry Season Material Moisture	%	3
Retained Moisture After Draindown	%	8.3
Average Annual Emitter Evaporation	%	2
Material Throughput per Year	Mt	22.5

For all modelled scenarios, the Converse process will have a net annual water deficit during production and make-up water will be required. Estimated annual make-up requirements range from 177.3 m^3/h to 218.9 m^3/h, with a maximum monthly demand of 305.2 m^3/h. Water treatment and discharge of heap process should not be required based on the models.

17.6 Recovery Plant

The recovery plant will be designed to recover gold values using an Adsorption-Desorption-Recovery (ADR) process. Pregnant leach solution from the heap leach will be pumped to the carbon in column (CIC) circuit and adsorbed onto activated carbon (adsorption). Loaded carbon from the CIC circuit will then be desorbed or stripped in a high temperature elution process coupled to an electrowinning circuit (desorption), followed by retorting to recover mercury and smelting the resulting sludge to produce doré (recovery). Prior to elution, each batch of carbon will be acid washed to remove any scale and other inorganic contaminants that might inhibit gold adsorption on carbon. All activated carbon will be thermally reactivated using a rotary kiln after each elution batch.

The recovery plant will be semi-automatic with local human machine interface (HMI) panels displaying unit functions and controlling primary flow streams. Non-primary or batch flow streams, such as acid washing, will be controlled manually. All local sensors will provide a signal to the main PLC/control station for monitoring.

The recovery plant and refinery will be inside a building, with the carbon adsorption circuit being outdoors and uncovered.

17.6.1 Adsorption

Adsorption of gold onto activated carbon will be accomplished in the carbon adsorption circuit comprised of four column trains of five cascade type, open-top adsorption columns. Each column will have capacity for 11 t of carbon. Pregnant solution from the pregnant solution pond will be pumped to the adsorption circuit at a total nominal rate of 4,020 m^3/h (1,005 m^3/h each train) and a design rate of 4824 m^3/h (1,206 m^3/h each train). Barren solution exiting the last carbon adsorption column in each train will pass through a static carbon safety screen to separate any floating carbon from the solution, then flow by gravity into the barren pond.

Antiscalant will be added at the pregnant solution pond to prevent scaling of the carbon which reduces the carbon loading ability. Magnetic flowmeters equipped with totalizers will measure solution flow to the adsorption circuit. Pregnant solution will flow by gravity through each of the five columns in series in each column train, exiting the lowest column as barren solution. Continuous samplers of the pregnant and barren solutions will be installed at the feed and discharge ends of the carbon column trains. Solution samples will be used to measure gold concentrations in the pregnant and barren solutions, and to monitor the carbon adsorption efficiency.

The process of gold adsorption from the pregnant leach solution is continuous. Once the carbon in the lead column achieves the desired precious metal loading, it will be transferred to the carbon acid wash circuit using a screw impeller carbon transfer pump (one per column train). Carbon in the remaining columns will then be advanced, counter current to the solution flow, to each preceding column in series. New or acid washed/regenerated carbon will be added to the final column in the train.

17.6.2 Acid Washing

Acid washing consists of circulating a dilute acid solution through a bed of activated carbon to dissolve and remove scale and other inorganic contaminants. Acid washing of the Converse carbon will be performed on a batch basis before each desorption cycle.

Loaded carbon from the adsorption circuit will be transferred to the acid wash vessel. The acid wash vessel is designed for a total capacity of 11 tonnes of loaded activated carbon. After the carbon is transferred to the acid wash vessel, but before any acid is introduced, fresh water will be circulated through the carbon bed to remove any entrained alkaline cyanide solution. After rinsing, a dilute nitric acid solution will be prepared in the acid mix tank and cycled through the acid wash vessel and the acid mix tank using the acid wash circulation pump. Concentrated acid will be injected into the recycle stream to achieve and maintain a pH ranging from 1.0 to 2.0. Completion of the acid wash cycle is indicated when the pH stabilizes between 1.0 and 2.0 without acid addition for a minimum of one hour of circulation.

After acid washing is complete, the spent acid solution will be drained from the acid wash vessel into the acid mix tank. The spent acid solution can either be retained for reuse or neutralized to a pH ranging between 7.0 and 8.0 by adding caustic before being pumped to the barren pond.

To remove any residual acid in the acid washed carbon, carbon will be rinsed with fresh water. After the rinsing, the acid washed carbon will be pumped to the elution vessel using the acid wash carbon transfer pump. Total time required for acid washing a 11-tonne batch of carbon is approximately 6 hours.

17.6.3 Desorption

A Zadra pressure elution, hot caustic desorption circuit has been selected for the Project. This type of circuit requires less than 24 hours to complete a cycle and is sized for 11-tonne batches of carbon. During the desorption process, gold will be eluted, or “stripped,” from the batch of carbon into pregnant eluate solution. The gold is then extracted by electrowinning from the pregnant eluate produced by the desorption circuit. A complete desorption cycle will require approximately 18 hours.

After a batch of carbon has been transferred to the elution vessel, a barren strip solution (eluant) containing sodium hydroxide and sodium cyanide will be pumped through a recovery heat exchanger and a solution heating system, which will include an electric hot water boiler, a hot water circulation pump, and a primary heat exchanger. Hot water from the boiler will be pumped through the primary heat exchanger to heat the strip solution to the strip temperature of 135°C, before being introduced to the elution vessel with a nominal operating pressure of approximately 450 kPa. The final gold content of the stripped carbon will typically be less than 170 g/t of carbon.

The elution vessel will contain internal stainless-steel inlet screens to retain carbon within the vessel and distribute the incoming stripping solution evenly. Pregnant eluant solution leaving the elution vessel will pass through external stainless-steel strip solution discharge screens, before passing through the recovery heat exchanger and tertiary heat exchanger to reduce the eluate temperature to 90°C or less (to prevent boiling). The cooled pregnant eluate solution will then be discharged to the electrowinning cells.

After desorption is complete, the stripped carbon will be pumped to either the kiln feed dewatering screen to dewater the carbon and remove fines before thermal regeneration, or to a carbon storage tank for addition back to the circuit.

17.6.4 Electrowinning and Refining

The electrowinning circuit will be operated in series with the elution circuit. Cooled pregnant eluate from the elution circuit will pass through the electrowinning cells, with gold values being recovered from the pregnant eluant solution as the solution passes through the cells. Barren eluate solution leaving the electrowinning cells will flow by gravity to the barren eluant return tank, where it will then be pumped by the barren eluant return pump back to the eluant storage tank.

Gold will be recovered from the eluant in the electrowinning cells using stainless steel cathodes at a current density of approximately 5 A/ft^2 on the anode surface. Caustic soda (sodium hydroxide) in the eluate will act as an electrolyte, facilitating the free flow of electrons and promoting the recovery of precious metals from the solution. To keep the solution's electrical resistance low during the electrowinning cycle, make-up caustic soda will be added to the eluant storage tank as needed.

Periodically, all or part of the barren eluant will be dumped into the barren pond, and a new solution will be added to the eluant storage tank. Typically, approximately one-third of the barren eluant will be discarded after each elution or strip cycle. Sodium hydroxide and sodium cyanide will be added as required from the reagent handling systems to the eluant storage tank during fresh solution make-up.

The precious metal-laden cathodes in the electrowinning cells will be removed periodically and processed to produce the final doré product. The loaded cathodes will be transferred to the cathode wash box using a cathode hoist. Precipitated precious metals will then be removed from the cathodes with a high pressure sprayer. The resulting sludge will be pumped using a sludge filter feed pump to a plate-and-frame sludge filter press to remove water. The filter cake will then be loaded into pans and sent to the refinery for treatment in the mercury retort furnace. To volatilize the mercury, the sludge filter cake will be placed into pans and heated in the retort for up to 48 hours at approximately 482°C.

A vacuum pump system will continuously remove mercury vapor from the retort oven and pass the vapor through the water-cooled retort primary condenser. Condensed mercury will be trapped in the mercury collector and then transferred and stored in flasks. Cooled exhaust leaving the mercury collector will pass through the retort scrubbing system to remove any residual mercury. The retort scrubbing system will comprise three units connected in series: the mercury after-cooler condenser, retort air filter, and retort carbon scrubber filled with sulphur-impregnated activated carbon.

After mercury removal, the dried cathode filter cake will be mixed with fluxes and fed to a tilting crucible induction furnace. After melting, slag will be poured off into cast iron molds until the remaining molten furnace charge is mostly molten metal (doré). Doré will then be poured off into bar molds, cooled, cleaned, and stored in a vault pending shipment to a third-party refiner. The doré poured from the furnace represents the final product of the processing circuit.

Periodically, slag produced during the smelting operation will be re-smelted in batches to recover residual metal values. Reprocessed slag will be crushed and placed on the heap leach pad.

Furnace fumes will be pulled through the furnace fume hood by the furnace exhaust fan. Collected fumes will pass through the refinery bag house to remove particulates, then through the furnace carbon bed scrubber as a final exhaust cleaning step. The system will be designed to remove over 99.5% of the particulates present in the exhaust fumes.

17.6.5 Carbon Handling and Regeneration

The carbon handling and regeneration circuit will include all equipment required to regenerate, store, prepare, and transfer carbon. The carbon regeneration system will include a kiln dewatering screen, kiln feed hopper with screw feeder, carbon regeneration kiln, carbon quench tank, and carbon quench pump. The carbon preparation and storage system will include a carbon sizing screen, an 11-tonne carbon storage tank, a carbon conditioning tank with an agitated mixer, a carbon fines tank, a carbon fines filter press, and various carbon transfer pumps.

Thermal regeneration will consist of thoroughly drying the carbon and heating it to approximately 760°C for 10 minutes to maintain carbon activity levels.

Carbon from the desorption circuit to be thermally reactivated will first be dewatered using a static kiln dewatering screen, then transferred to the kiln feed hopper and fed to the carbon regeneration kiln by the kiln screw feeder. The kiln dewatering screen undersize will discharge to the carbon fines tank. Carbon fines collected in the carbon fines tank will be periodically pumped through the carbon fines filter press; carbon fines from the filter press are stored in bulk bags for removal from the system.

Hot, regenerated carbon leaving the kiln will pass into a water-filled quench tank for cooling before being transferred to the carbon sizing screen by the carbon quench pump. New carbon being added to the circuit will first be processed in the carbon conditioning tank and then transferred to the carbon sizing screen. The undersize from the sizing screen will discharge into the carbon fines tank, and the oversize will discharge into the carbon storage tank. The new and regenerated carbon stored in the carbon storage tank will be returned to the CIC circuit.

17.7 Process Reagents and Consumables

The reagent handling system will include equipment for mixing and/or storing all reagents required for the Converse process. Reagent mixing and storage will be at ambient temperature and pressure.

Average estimated annual reagent and consumable consumption quantities for the processing area are shown in Table 17-5.

Table 17-5: Projected Annual Reagents and Consumables

Item	Form	Average Annual Usage
Sodium Cyanide	Liquid @ 30% wt.	14,440 t
Cement	Bulk Delivery Trucks	135,000 t
Activated Carbon	500 kg Supersacks	120 t
Sodium Hydroxide	Liquid Delivery Trucks	88 t (wet)
Antiscalant	Liquid Bulk	237 m^3
Nitric Acid	Liquid Bulk	628 m^3
Fluxes	Dry Solid Sacks	62 t

17.7.1 Sodium Cyanide

Cyanide is used to dissolve gold during the leaching process. Cyanide solution will be provided to the site by a tanker truck. Each truck will deliver approximately 25 m^3 of 30% solution. The solution will be transferred to a 225 m^3 storage tank. The tank will store approximately 2 days of cyanide inventory for the plant.

Cyanide is primarily consumed during the leaching process at an average rate of 0.64 kg/t processed. A small amount of cyanide will also be added to the elution circuit.

17.7.2 Cement

Cement will be used to treat the crushed material prior to cyanide leaching to maintain the alkaline pH, heap stability, and proper percolation. Cement will be consumed at an estimated 6 kg/t of material. Cement will be delivered in bulk by 18-tonne trucks, which will be offloaded pneumatically into the storage silos. Cement will be metered onto the Agglomeration Drum feed conveyors with a rotary feeder and screw feeder from the three 362-tonne storage silos. The silo inventory is equivalent to approximately 3 days of cement.

17.7.3 Activated Carbon

Activated carbon will be used to adsorb precious metals from the leach solution in the adsorption columns. Make-up carbon will be 6 x 12 mesh and will be delivered in 500 kg supersacks. It is estimated that approximately 3% of the carbon stripped will have to be replaced due to carbon fines losses. Carbon consumption has been estimated at 120 tpa.

17.7.4 Sodium Hydroxide (Caustic)

Sodium hydroxide (caustic) solution will be delivered to the site as a 50% liquid concentrate. The delivered high-concentration caustic solution will be diluted to yield a 20% by weight sodium hydroxide solution for use in the process. Distribution of the caustic solution will be by the caustic transfer pump to points of use. Sodium hydroxide will primarily be used in the elution strip solution and will be consumed at an estimated rate of 120 kg per strip.

17.7.5 Nitric Acid

Nitric acid (HNO_3) is used in the acid wash section of the elution circuit. Nitric acid (67% by weight) will be delivered to the site in a tanker truck and stored in a 23 m^3 tank. HNO_3 consumption is estimated at 1,720 L per strip.

17.7.6 Antiscalant

Antiscalant is used to prevent the build-up of scale in process solutions and heap irrigation lines. Antiscalant will be delivered to site in liquid form in bulk trucks. Antiscalant will be added directly to the pregnant solution pond pump inlet, barren tanks, and the elution vessel feed line using variable speed, chemical-metering pumps.

Antiscalant consumption will vary depending on the concentration of scale-forming species in each treated process stream. On average, antiscalant consumption is expected to be approximately 6 ppm for the pregnant and barren leach solutions.

17.7.7 Fluxes

Various fluxes are used in the smelting process to remove impurities from the bullion in the form of a glass slag. Dry fluxes will be delivered in 23 kg bags.

The normal flux components will be a mix of silica sand, borax, and sodium carbonate (soda ash). The flux mix composition is variable and will be adjusted to meet individual project smelting needs: fluorspar and/or potassium nitrate (niter) may also be added to the mix. Average consumption of the mixed fluxes is estimated to be 1.5 kg of flux per kg of gold and silver produced.

17.7.8 Pulp Agglomeration (Future Opportunity)

Test work for Converse shows potential for improved recoveries by grinding and cyanide leaching of the material, with milled recoveries averaging 93% for gold, compared to 70.9% by heap leaching alone. Although the overall tonnage and grade of the current project do not support a standalone milling case, there is the potential to improve the metal production and overall project economics by milling select portions of high-grade material in parallel with the heap leach operation. Pulp agglomeration is a potential option that requires less capital investment than a conventional mill and can be easily incorporated into the existing flowsheet during later operational years. The first known operation with pulp agglomeration is the historical Castle Mountain Mine, operated in the 1990s by Viceroy. Two other operations are Ruby Hill and Dolores. KCA was involved as the EPCM for the Dolores mine in Chihuahua, Mexico, with construction completed in 2016.

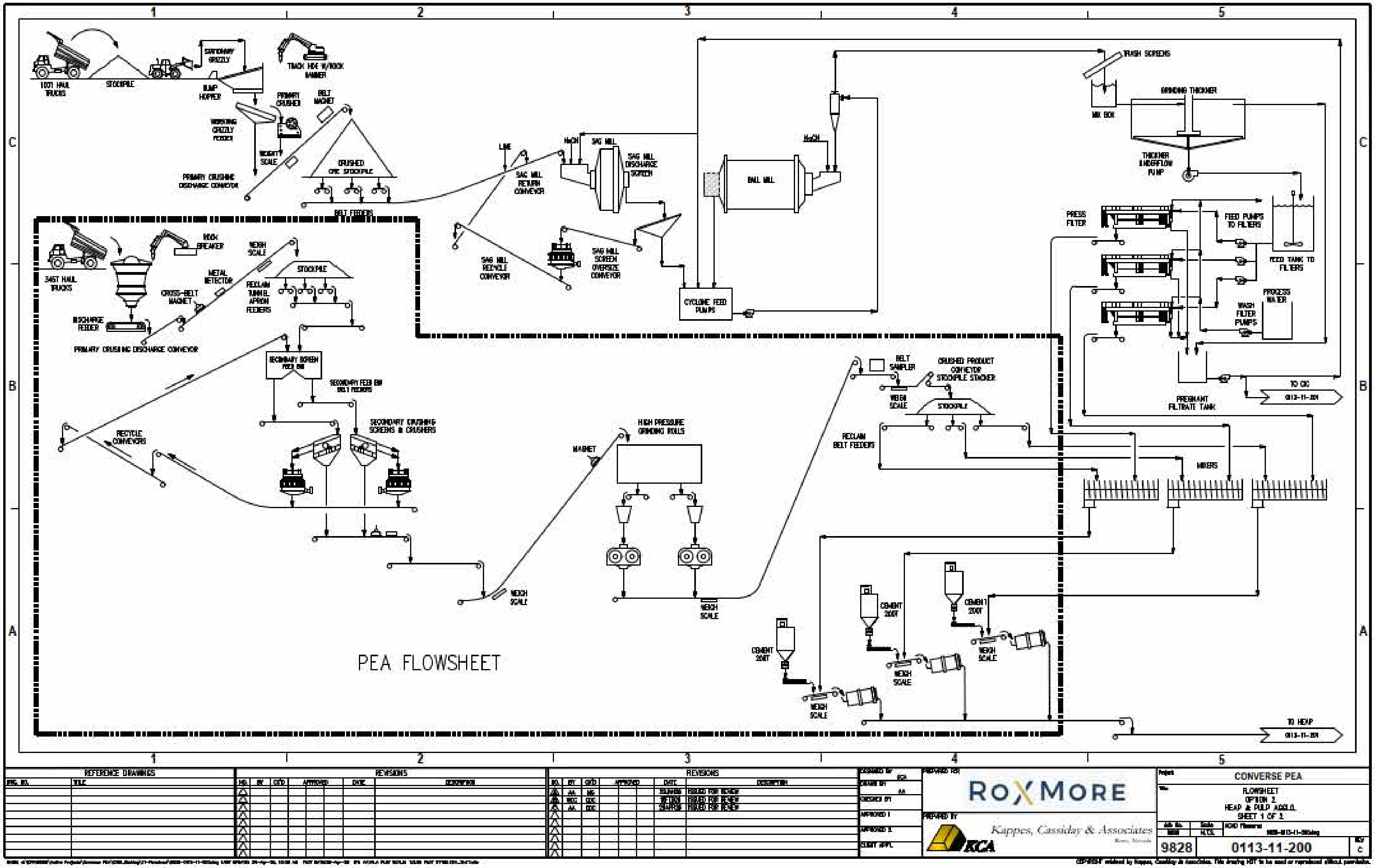
Caleb Cook and Timothy Scott of KCA presented a paper, “Pulp Agglomeration Process and Applications for Gold and Silver Heap Leach Operations”, at the Heap Leach Solutions 2022 conference in Sparks, Nevada.

The pulp agglomeration process consists of blending a ground product slurry with a crushed product and binding them with cement to form stable agglomerates for heap leaching. For Converse, the HPGR crushed low-grade material (existing flowsheet) would be blended with milled high-grade material in mixers to ensure an even distribution of fines before being drum agglomerated and stacked on the heap leach pad. The milled product would be dewatered using plate and frame filters similar to what would be used for dry stacked tailings. The flowsheet for pulp agglomeration inclusion is presented in Figure 17-3

Cyanide would be present in the milling circuit, allowing for leaching and recovery of a significant portion of the recoverable metal values from the high-grade material before final processing by heap leaching. The resulting pregnant leach solution from the mill would be collected as filtrate from the filtration system and sent to the recovery plant. A conceptual flowsheet of the pulp agglomeration circuit, illustrative of how it would integrate with the existing flowsheet, is presented in Figure 17-3.

Assuming a 5 Mtpa production rate and cement addition of 12 kg/t, the estimated operating cost for a pulp agglomeration circuit would be approximately $19/t.

Figure 17-3 Pulp Agglomeration Inclusion with PEA Flowsheet

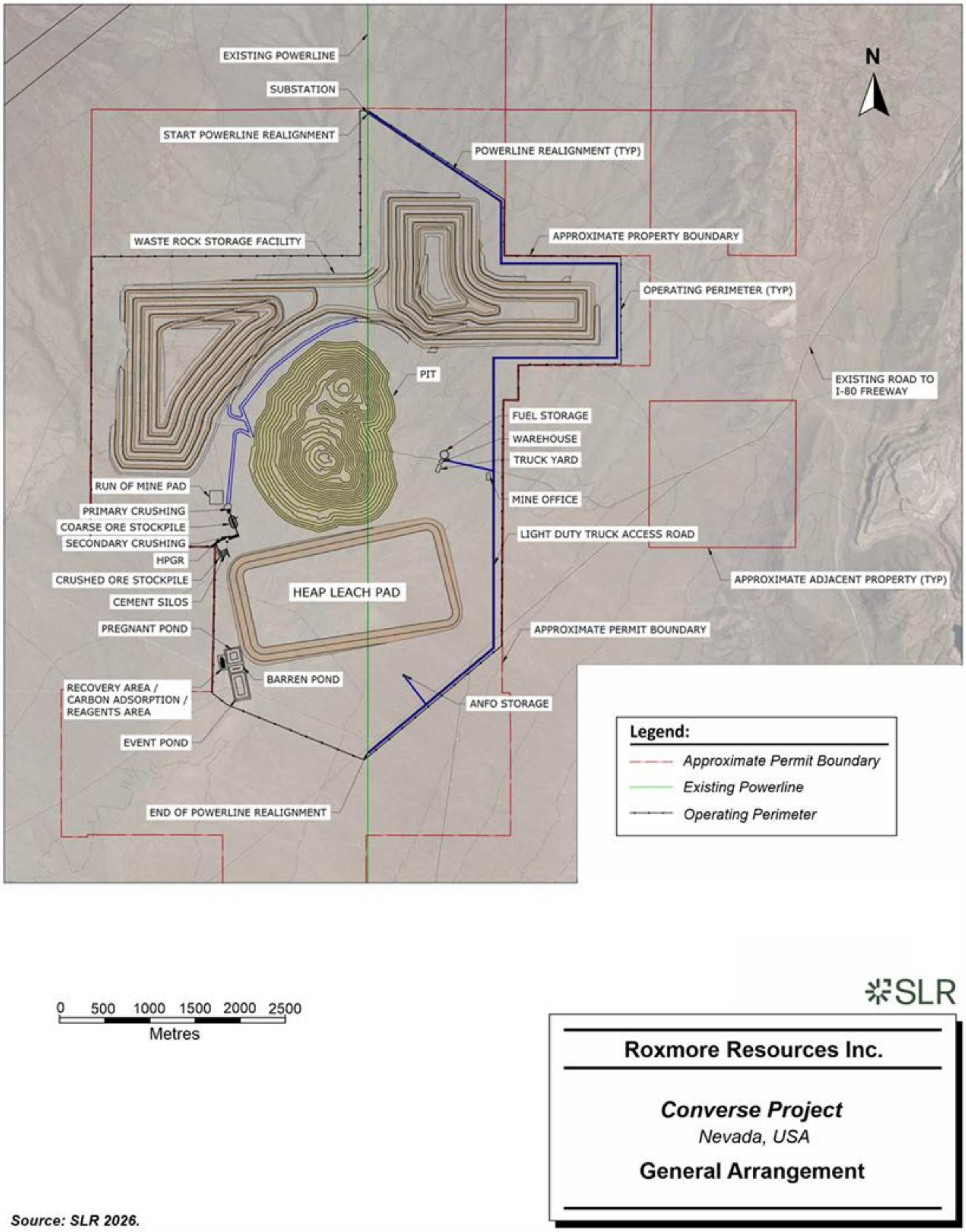


Source: KCA.2026.

18.0 Project Infrastructure

The Project infrastructure has been designed to support a large-scale open pit mining and heap leach operation utilizing conventional, well established technologies commonly employed in northern Nevada. Major infrastructure components include site access and internal mine roads, WRSFs, ROM and crushed stockpiles, a single heap leach pad with associated solution management and storm event ponds, an ADR plant for precious metals recovery, electrical power supply and distribution systems, and process and raw water pipelines. The Project benefits from its location within an established mining district with access to existing regional infrastructure, experienced contractors, grid power, and services. Infrastructure development is planned to be staged in alignment with the mine production schedule and designed to meet regulatory, environmental, and operational requirements. The overall site layout is illustrated in Figure 18-1.

Figure 18-1: Site General Arrangement



Source: SLR 2026.

18.1 Access Roads

Access to the Project site from Interstate 80 (I-80) will be along Buffalo Valley Road from the Valmy exit. Existing dirt roads that traverse from Buffalo Valley Road to the west to access the Project site will need to be widened and resurfaced. The estimated length of access road that will need to be widened and resurfaced is approximately 1.75 miles.

18.2 Power

18.2.1 Process Electrical Power

The estimated electrical power demand for the Project is presented in Table 18-1.

Table 18-1: Converse Project Power Demand

Area	Attached Power (kW)	Demand (kW)	Peak Demand (kW)	Power Consumption (kWh/t)
Area 113 - Crushing	12,699	7,144	8,945	2.775
Area 115 - Agglomeration & Stacking	7,241	3,698	5,114	1.439
Area 122 - Heap Leach Pad & Ponds	4,216	2,872	3,054	1.118
Area 128 - Carbon Adsorption & Handling	834	550	644	0.055
Area 129 - Desorption	3,625	2,041	2,082	0.391
Area 131 - Refinery	434	285	326	0.084
Area 134 - Reagents	29	9	178	0.002
Area 338 - Laboratory	450	253	338	0.099
Area 360 - Power	0	0	0	0.000
Area 362 - Water	186	63	112	0.033
Area 366 - Facilities	30	11	23	0.004
Total	**29,744**	**16,924**	**20,814**	**6.001**

18.2.1.1 Emergency Power

In the event of a power failure or interruption, a diesel-fired backup generator will provide emergency power to ensure project safety and security.

To maintain critical solution balances in the solution handling systems during power outages, two 3.4 MW generators are required for the recovery area serving the critical pumps. This emergency generator will be located next to the recovery plant. A fuel tank will be provided for the generator to maintain a 24 hour fuel supply. The fuel storage system will also include a concrete containment area sized to accommodate 110% of the tank's capacity.

18.3 Water

Make-up water for mine operations will be used for dust suppression, agglomerations, crusher and screening spray systems, leach solution makeup, heap irrigation, pregnant leach solution conveyance, barren solution makeup, pipeline flushing and maintenance, ADR plant processes, operational pond balance, equipment wash-down, fire suppression, and laboratory operations management. It will be sourced from production wells developed on site in the early years of the mine life and will rely on water resultant from pit dewatering activities in the later years as the pit limits extend beyond the groundwater table.

In 2019, CRL purchased 2,560 acre-ft of irrigation water rights from the New Nevada Lands, LLC, which was subsequently converted to mining and milling use. The acquired water rights will support the construction and operation of a future mine at the Property, though a detailed site water balance has not been completed.

18.4 Heap Leach Pad

A synthetically lined heap leach pad of approximately 3.3 Mm^2 in size will be constructed near the mine pit to accommodate approximately 310 Mt of mineralized material. The agglomerated mineralized material will be placed on the heap leach pad and leached with a cyanide solution. The pregnant leach solution will be recovered from the pad via a system of collection pipelines above the synthetic liner and routed to the external Pregnant Solution Pond (Preg Pond). The heap leach pad, pregnant and barren solution ponds are synthetically lined so that the solution is contained within a closed system, with the only net solution loss being to evaporation.

18.5 Project Buildings

18.5.1 Office Buildings

Office buildings will include a process office trailer and a separate mine office. The process office trailer, adjacent to the processing infrastructure, will be a single-wide, 3.7 m x 18.3 m prefabricated trailer with two office areas, a washroom, and a central common area. The mine office, on the east side of the pit, will be a pre-engineered building supporting mine operations and salaried staff with approximate dimensions of 75 m x 45 m. Attached and adjacent to the mine office will be a Dry facility for operations personnel with approximate dimensions of 36 m x 26 m, including washrooms, showers, lockers, and laundry facilities.

18.5.2 Laboratory

The laboratory facility will process samples from the mine and process plant. The laboratory will be an insulated, pre-engineered steel building with atomic adsorption and fire assay capabilities, as well as a full wet laboratory and metallurgical laboratory for monthly column and bottle roll tests on production composite samples. The laboratory will have the capacity to process 150 samples per day and includes all necessary eye wash/safety showers and fire water distribution to meet safety requirements.

18.5.3 Truck Shop

The truck shop is a central maintenance facility designed to support the reliable, safe, and continuous operation of the Project’s mobile equipment fleet. Its primary function is to provide preventive maintenance, repairs, inspections, and component change-outs for large off-highway haul trucks and other heavy mining equipment. The truck shop will be a pre-engineered steel structure with approximate plan dimensions of 84 m x 40 m.

18.5.4 Warehouse

The onsite warehouse serves as the primary logistics, storage, and distribution hub for materials, consumables, spare parts, and supplies required to support continuous mining, crushing, heap leaching, and maintenance operations. The warehouse will also be a pre-engineered steel structure with approximate plan dimensions of 40 m x 36 m.

18.5.1 Ammonium Nitrate / Fuel Oil (ANFO)

The ANFO storage building is a dedicated, purpose-built facility used for the safe, compliant, and controlled storage of blasting agent components required to support routine mining operations. The ANFO storage building will be an engineered reinforced masonry structure located remotely on site, away from regular operational activities.

18.6 Accommodation Camp

An accommodation camp is not required for the Project due to its proximity to the established community of Winnemucca, located approximately 50 miles from the site. Winnemucca provides sufficient existing infrastructure to support the workforce, including housing, hotels, dining, medical services, and commercial amenities. The Project is accessible via I-80 and maintained access roads, enabling reliable daily commuting for employees.

The Project is expected to utilize a local and regional workforce on a drive-in/drive-out basis, consistent with standard practice for mining operations in north-central Nevada. This approach reduces capital and operating costs associated with constructing and maintaining a camp, while also minimizing environmental disturbance and permitting requirements.

19.0 Market Studies and Contracts

19.1 Markets

The principal commodity for the Converse Project is gold, which is freely traded at prices that are widely known and is considered a highly liquid commodity. As such, prospects for the sale of Converse gold production are virtually assured.

The gold prices used in this report have been provided to SLR by Roxmore and are aligned with the long-term views of several reputable market analysts and with SLR's internal pricing guidelines.

SLR notes that the pit optimization analysis and the Mineral Resource estimation are based on a long-term gold price of US$3,000/oz.

SLR notes that for the economic analysis, the gold prices used are based on analyst market consensus as of March 31, 2026. The SLR QP considers the selected metal prices acceptable for the economic analysis, given the current mine life. The prices used for the economic analysis are shown in Table 19-1.

Table 19-1: Metal Price Assumptions for Economic Analysis

Metal Prices	Year -1	Year 1	Year 2	Year 3	Long-term
Gold (US$/oz)	4,000	4,000	3,600	3,600	3,600

19.2 Contracts

Contracts for the Project are expected to include design and construction contracts for the facilities and mine development; however, none of these contracts are in place at this time. Contracts for the transportation, refining, and commercialization of the gold produced will also be required.

Contracts will be negotiated and executed as necessary.

20.0 Environmental Studies, Permitting, and Social or Community Impact

The Project is currently approaching the PEA level of development; however, minimal environmental studies and project permitting efforts have been performed to date. This section provides a summary of the readily available information on environmental, permitting, and social or community factors related to the Project. Based on the review summarized herein, and on analogous projects in Northern Nevada, there do not appear to be any red flags or insurmountable hurdles to developing the Project.

20.1 Environmental Studies

Environmental conditions and studies described in this section are summarized based on information documented in previous EAs (e.g., EMA 1998). No recent or ongoing baseline studies or EAs specific to the Project have been reviewed for this report as none are known to have been performed.

The Project exists on private public lands administered by the BLM. As a result, the majority of environmental studies related to mining activities will need to be conducted under the BLM authority as part of the *National Environmental Policy Act* (NEPA) regulations, which require various degrees of environmental impact analyses dictated by the scope of the proposed action. Table 20-1 summarizes the major environmental baseline studies and assessments that will likely be required.

Table 20-1: Baseline Studies and Environmental Assessments

Study Media	Baseline Studies and Environmental Data
Hydrology/Water Quality/Geochemistry	Hydrogeologic Assessment, Geochemical Testing, Water Balance Study, Seeps & Springs
Air Quality	Air Quality Assessment, PSD Baseline (minimum 12-months)
Flora/Fauna	Golden Eagles Surveys, Sage Grouse Surveys
Socio-Economic	Social Baseline Assessment
Cultural Resources	Site-Wide Cultural Resource Survey

20.1.1 Physiography

The Project is located in northern Nevada in the Great Basin region of the Basin and Range physiographic province. The project area is situated in a valley on a gently southwest sloping alluvial plain at an approximate elevation of 5,000 ft above mean sea level. The site is surrounded by Buffalo Mountain to the northwest, the Havallah Hills to the north, and the Battle Mountain range to the east. The Buffalo Valley slopes to the south towards Alkali Lake, a typically dry playa.

20.1.2 Geology

The Basin and Range physiographic province is characterized by a series of generally north-trending, fault-bounded mountain ranges separated by broad alluvium or lake sediment filled valleys. The Project is at the north end of the Buffalo Valley, which is a typical basin and range, fault-bounded valley. The surficial geology at the Project site is alluvial gravels, mapped as

Quaternary alluvium, with Pennsylvanian Havallah sequence cherts, basalt and sandstone cropping out in the vicinity (Ferguson et. al. 1951). The area is underlain by rocks of the Golconda allochthon, which was thrust eastwards over older sedimentary rocks during the Sonoma Orogeny, approximately 250 million years ago.

The mineralized material forms a low-grade, gold-copper intrusion-related skarn system hosted within calcareous sandstone of the Havallah sequence. Two main skarn zones have been identified (North Redline and South Redline). Mineralized material consists of liberated gold, iron oxides, chalcopyrite, and pyrrhotite. Skarn systems form when high temperature, mineral rich magmatic fluids (in this case the Redline intrusive stock) interact with carbonate-rich rocks. The carbonaceous nature of the host rock has implications for acid generation and neutralization potential during and after mining.

20.1.3 Acid Rock Drainage/Metal Leaching Potential

A comprehensive geochemical testing program has not been completed; however, in 2012 SRK Consulting conducted a scoping level geochemical characterization study to assess the acid rock drainage and metal leaching (ARD/ML) potential of the deposits at Converse. The study was focused on the potential for waste rock to generate acid and the ability to predict future water quality from precipitation contacting the material (Srivastava et. al. 2012). SRK had the following conclusions:

> Preliminary acid-base accounting (ABA) results indicate that waste rock from the Converse deposit will have a low potential for acid generation. Because some of the samples fall within the uncertain zone, in particular the supergene altered samples as well as the tuffaceous sandstone, kinetic testing may be required for confirmation. However, the risk that these samples that fall in the uncertain zone would generate acid is considered low based on non-detect sulphide sulphur concentrations and Neutralization Potential Ratio (NPR) values greater than 3.

Metallurgical work by KCA indicates relatively low sulphide content (0.5% to 5%). Acid rock drainage (ARD) could be a problem depending on available neutralization potential. The presence of carbonate-rich country rocks would likely provide sufficient neutralization potential.

The SLR QP recommends detailed geochemical assessment composed of static acid-base accounting and metals leaching testing to be developed to characterize the various rock types within the context of the mining and mine waste management plans. Follow-up analysis may include kinetic testing such as humidity and field cells.

20.1.4 Atmospheric Environment

The climate at Converse is arid, characterized by warm, dry summers and cold, dry winters. Site specific data has not been collected; however historical data from the nearby Lone Tree Mine collected in the 1990s indicated a temperature range of –23°C to 38°C with 150 mm to 200 mm of average annual precipitation.

The air quality in the region of the Project is generally good, due to the limited population and industrial activity. Several mines in the vicinity have the potential to contribute to particulate emissions and industrial pollutants in the Buffalo Valley. The nearest currently active mine is the Marigold open pit gold mine, located approximately 8 km to the east.

The SLR QP recommends that baseline meteorological data collection should commence as soon as possible and include a 10 m meteorological station with 12 to16 months of monitoring for Prevention of Significant Deterioration (PSD) compliance.

20.1.5 Acoustic Environment

Converse is in a remote area with limited human activity, and no substantive anthropogenic noise sources within 5 km. The Project site is located approximately 15 km southwest of I-80 and between 16 km and 21 km from two major railways. The nearby Marigold Mine is another source of intermittent anthropogenic noise. All of the major anthropogenic noise sources are on the opposite side of mountains and hills, so the background site noise level is expected to be relatively low.

20.1.6 Groundwater and Surface Water

The Project area is located within the Buffalo Valley groundwater basin. No permanent water sources are located within the Project area and ephemeral drainages flow southward into the closed basin at Alkali Lake. No seeps and springs have been identified for the Project area.

Exploration drilling indicates groundwater depths of approximately 350 ft below ground surface. Groundwater is inferred to flow to the south following topographic contours. No information on baseline water quality was available for this report.

The Project proponents have maintained two water rights (as detailed below). The water rights have a total allocation of 1,984 acre-feet per annum (AFA). It is currently unclear if this allowance will be sufficient for operations (see Section 17.5.2). It is also not known if the effective yield of the aquifer can provide the allocated volume.

20.1.7 Wildlife

A desktop study of wildlife in the area was performed in 1998 as part of the EA for exploration work (EMA 1998). The study summarized species typical of the area, which include mule deer, antelope, coyote, bobcat, mountain lion, multiple rabbit, rodent and shrew species, upland game birds, and some reptiles. Raptors reportedly observed in the vicinity include golden eagle, Swainson’s hawk, ferruginous hawk, and American kestrel. Bald eagle and various owl species are also likely to be present.

Sage grouse and chukars are present in the Buffalo Mountains. The BLM has identified Greater Sage Grouse habitat management areas in the Buffalo Valley both to the northwest and southeast of the Project area. The nearest General Habitat Management Area (GHMA), which may support Greater Sage Grouse populations at a reduced density, is less than 2 km southeast of the Project. An approximately 20,000 ha Priority Habitat Management Area (PHMA), which is considered high-quality seasonal habitat essential for maintaining population connectivity, has been mapped for the Battle Mountain range, located less than 10 km southeast of Converse.

No threatened or endangered species are known to exist within the Project area. The area may provide habitat for Western burrowing owl and pygmy rabbit, which are both designated as BLM Sensitive Species.

20.1.8 Cultural Resources

Federal lands managed under the *National Historic Preservation Act* (NHPA) and BLM guidelines require cultural resource surveys prior to approval of a ground-disturbing action. Class II and Class III cultural resource inventories were performed for the EA developed as part of the 1998 exploration work. A total of eight archaeological sites were found and recorded. None of the recorded sites meet the criteria for nomination to the National Register of Historic Places (NRHP) or warrant further research. As such, it is unlikely that significant cultural resources will be identified as the Project develops; however, additional surveys will most likely be required. BLM typically considers a previous survey valid for no more than 5 to 10 years.

20.2 Environmental Approvals

Various federal, state, and local permits and authorizations will be required for Project construction and operations. Currently, a limited set of approvals have been established, primarily for the purposes of exploration and maintenance of water rights. A list of currently maintained permits and obligations is presented in Table 20-2.

Table 20-2: Converse Project Obligation Register

Permit	Authorizes	Permit No.	Agency	Status
Reclamation Permit	Exploration	0122	BMRR	Current
Environmental Assessment	Exploration	N20-98-001P	BLM	Current
Plan of Operations	Exploration	NVN-65461	BLM	Current
Water Rights	Water Right	71715	NDWR	Current
Water Rights	Water Right	71716	NDWR	Current

Note: BMRR is the Nevada Division of Environmental Protection (NDEP) Bureau of Mining Regulation and Reclamation; NDWR is the Neveda Division of Water Resources.

20.2.1 Federal Permitting

Because much of the Project area is situation on BLM land, a Plan of Operations (PoO) will be required. The PoO is a comprehensive document describing in detail how the operation will be constructed, operated, monitored, closed and reclaimed. PFS level planning and at least one year of baseline studies will be required prior to submittal of a draft PoO to BLM. The PoO must include detailed reclamation plans, a detailed cost estimate, and a bond sufficient for full reclamation.

BLM will require some level of environmental characterization as part of the approval process, also referred to as the Record of Decision (ROD). If BLM determines that the Project will have no significant impact on human health or the environment, a Finding of No Significant Impact (FONSI) is issued and a detailed Environmental Impact Statement (EIS) performed under the NEPA will not be required. Due to the scope of the Project, it is very unlikely a FONSI would be issued. It should, therefore, be anticipated that a full NEPA-compliant EIS will be required prior to approval.

Other federal approvals may include a Section 404 *Clean Water Act* permit from the US Army Corps of Engineers. Though this is unlikely if no Waters of the United States (WOTUS) are present within the Project area, as appears to be the case. Air permitting (such as Title V Clear Air Act) is administered by the State of Nevada.

20.2.2 State Permitting

The majority of Nevada approvals would be issued through the NDEP and its various bureaus and divisions. A Water Pollution Control Permit (WPCP) is issued for the management or disposal of water in ways that could infiltrate to groundwater. The WPCP will include a groundwater and surface water monitoring program.

A Class I (Title V, major source) air permit through the NDEP Bureau of Air Pollution Control will most likely be required depending on the emissions created by the processing facility and/or fugitive dust emissions from mining operations.

A Reclamation Permit will be required through the NDEP Bureau of Mining Regulation and Reclamation (BMRR). Similar to the requirements of BLM, the Reclamation Permit will require detailed closure and reclamation plans, a Financial Assurance Cost Estimate developed using the Nevada Standardized Reclamation Cost Estimator (SRCE 2.0) tool, and appropriate bonding.

The Nevada Division of Water Resources may require additional permits to appropriate water and a permit to construct a dam; however, based on the SLR QP's current understanding of the mine plan, neither a tailings facility nor surface water impoundment is planned.

The Nevada Sagebrush Ecosystem Program (SEP) is a state-led, collaborative initiative established to conserve greater sage grouse habitat and prevent Greater Sage Grouse from being listed under the *Endangered Species Act*. The program focuses on protecting PHMAs, none of which would be directly impacted by the Project) using a conservation credit system where developers (such as the Project proponent) purchase credits from landowners who protect or restore habitat to ensure a net benefit to the sage grouse.

20.2.3 Local Permitting

Humboldt County will require a combination of land use, construction, and safety permits. A Conditional Use Permit (CUP) will need to be approved by the regional planning committee. Humboldt County's Mining Operations ordinance (Title III, Division 9) requires a Use Permit for mining operations within the county. Although reclamation is primarily regulated by NDEP and BLM, Humboldt County incorporates reclamation requirements into its mining ordinance and may require consistency with county standards. Humboldt County's Building Department issues permits for the various structures and related infrastructure (grading, electrical and mechanical systems, water supply, wastewater, HVAC, and solar – if any).

20.2.4 Mine Waste Management and Monitoring Requirements

The approvals described in this section include robust mine waste management and monitoring requirements under Nevada and Federal law. Many of the required baseline studies (such as geochemical and hydrogeologic modeling) will be used to adequately inform the design of waste containment systems. Both tailings storage facilities (TSF) and waste rock/overburden disposal facilities (WRDF) must be designed to prevent seepage of mine-impacted water into soil and groundwater. The specific design will be dependent, in part, on whether the material is geochemically reactive and capable of potential acid generation or metals leaching. The containment system may require both engineered liners and covers. Leak detection systems and monitoring wells will be required around TSFs and will probably be required around WRDFs. Seepage must be managed through leachate collection and recovery systems (LCRS) and/or pumpback wells.

The WPCP, administered under NAC 4445A by NDEP, requires that all mine system designs prevent degradation of waters of the state. Groundwater monitoring wells and (if applicable) surface water monitoring points will require periodic (likely quarterly) monitoring and sampling. Analytical parameters will be established as part of the permitting process, but the most likely list is referred to as "Profile I" and includes major ions and cations, metals, and water quality field parameters such as pH, temperature, and turbidity. Process water and stormwater controls must also be implemented to prevent discharge of liquid mine waste or contact water.

Stormwater diversion structures and other Best Management Practices (BMPs) are often required to achieve this.

Post-closure monitoring will continue for up to 30 years, albeit in a reduced state which will depend on the effectiveness of closure and reclamation. Post-closure monitoring requirements will ultimately be determined by NDEP.

20.3 Social or Community Requirements

There are currently no known outstanding negotiations or social obligations related to the Project; however, stakeholder engagement has not yet begun in earnest to develop the Project. The NEPA process and any related large-scale state permitting will involve public comment periods as well as public meeting and outreach to stakeholders. The SLR QP recommends that the Project proponent begin stakeholder engagement as soon as possible. This might include commissioning a Stakeholder Engagement Study and/or reaching out to the Western Shoshone tribe, planning officials at Humboldt County, and non-governmental organizations (NGOs) such as Great Basin Resource Watch and Western Watersheds.

20.4 Mine Closure Requirements

All activities associated with the Project will require an approved reclamation plan that includes a Reclamation Cost Estimate (RCE) for all permitted facilities and activities. State regulatory requirements mandate a formal closure plan be filed two years before the facility initiates closure. Both the BLM and State require a tentative closure plan as part of normal NEPA permitting and operating permit requirements.

The mine closure and reclamation plans should include grading, backfilling, and recontouring to stable slopes with re-vegetation of all disturbed areas. Additionally, the plans should include discussions on removal of most infrastructure, monitoring, and heap leach facility closure, including long-term draindown solution management. BLM (under 43 CFR 3809) and NDEP (under NAC 519A) will require reclamation bonding based on 100% of the RCE. The financial assurance instrument is typically an irrevocable letter of credit or surety bond, though other options are available.

21.0 Capital and Operating Costs

The capital and operating costs presented in this section were estimated in Q1 2026 US dollars, and include the costs required for mining and processing mineralized material from the Converse Project.

21.1 Capital Costs

21.1.1 Basis of Estimate

The Converse capital costs estimate meets the requirements of an Association of the Advancement of Cost Engineering International (AACE) Class 5 estimate with an accuracy range of -20% to -50% and +30% to +100%. The cost estimates are appropriate for a PEA level of study, based within the range of 0% to 2% project definition per the AACE classification system.

Exclusions from the capital cost estimate include:

- Project financing and interest charges.
- Permits, fees, and process royalties.
- Environmental impact studies.
- Escalation beyond the date of inputs to the estimate.
- Import duties and custom fees, including tariffs.
- Inflation due to oil price impacts of the current supply restriction.

21.1.2 Pre-Production Capital Costs

The Project's pre-production capital costs are estimated to be $829 million, as summarized in Table 21-1, and the corresponding LOM sustaining capital costs are estimated at approximately $514 million.

Table 21-1: Pre-Production Capital Cost Summary

Cost Category	Initial Capital Cost (US$ million)
Mine Pre-Production Development	116
Mine Equipment Purchase (down payment)	39
Mine Facilities (Including truck shop)	38
Process - Heap Leach Pad & Process Plant	312
Site Powerlines & Substation	16
Administration Building	8
Access Road	3
Sub-total Direct Cost	**531**
Engineering, Procurement, and Construction Management (EPCM) / Indirect Costs / Spares & First Fills	79

Cost Category	Initial Capital Cost (US$ million)
Owner's Costs	53
Sub-total Indirects & Owners Cost	**132**
Total Excluding Contingency	**663**
Contingency	166
Total Capital Costs	**829**

Pre-production capital costs cover the mining activities executed during the 9 month development period. During this phase, a total of 75 Mt of material is required to establish the initial pit geometry and provide sufficient access to the deposit for sustained operations. This volume primarily consists of approximately 67 Mt of alluvial waste, characterized by high diggability and requiring only minimal blasting or free-digging, contributing to an average unit cost of $1.54/t.

The total expenditure for mine pre-production development is estimated at $116 million (Table 21-1). This includes the extraction of 2.63 Mt of mineralized material, scheduled for direct delivery to the leach pad during commissioning in the final quarter of the pre-production period. Mining operations utilize the primary fleet acquired under the 20% down payment leasing structure (Section 21.1.3). This strategic development phase is essential to "daylight" the deposit and establish the necessary working room to ensure a consistent and reliable production rate of 5.625 Mt per quarter (22.5 Mtpa) starting from Quarter 1 of Year 2.

21.1.2.1 Mining Capital Cost Estimate

The mining capital cost estimate was developed using the Costmine Intelligence SHERPA estimating tool (CostMine, 2025), utilizing the 2025 Dataset to determine costs for all initial mine equipment and supporting facilities. The estimate is directly driven by the Mine Production Schedule (Table 16-8) and the Mine Equipment Fleet requirements detailed in Section 16.9.

Pre-production Mine Equipment Capital

The pre-production mining equipment fleet was estimated to US$193 million. To optimize initial capital intensity, the primary mining fleet was modelled using a leasing structure. This financing strategy assumes a 20% down payment of US$39 million (detailed in Table 21-1), with the remaining 80% of the purchase price amortized over a 6 year lease period at an annual interest rate of 6%. Under this model, full ownership of the equipment is transferred to the Project upon completion of the lease term.

Mine Supporting Facilities Capital

The capital cost for mine facilities is estimated at $38 million (Table 21-1), based on the SHERPA 2025 Dataset. This estimate covers the essential infrastructure required to support the large-scale mining operation, including the construction of surface haul roads for both mineralized material and waste, a comprehensive truck shop and maintenance facility, and a mine warehouse.

Additional infrastructure captured within this estimate includes administrative and personnel support spaces such as employee dry rooms (change rooms), as well as operational support facilities including a dedicated mine yard, fuel storage and dispensing stations, and explosives storage. The estimate also accounts for site preparation requirements such as clearing, fencing, and the installation of necessary utilities including sewage treatment and water management. By utilizing benchmarked industry data, the estimate ensures that all ancillary mining infrastructure is accounted for in the initial capital outlay without the need for exhaustive civil engineering quantities at this stage of the Project.

21.1.2.2 Process and Infrastructure Capital Cost Estimate

Process and Infrastructure Capital Cost Basis

All equipment and material requirements are based on the design information described in previous sections of this report. Budgetary capital cost estimates were developed based on recent quotes from similar projects in KCA's files for all major and most minor equipment. Where recent quotes were unavailable, reasonable cost estimates or allowances were provided based on cost guide data. All capital cost estimates were based on the purchase of equipment quoted new from the manufacturer or to be fabricated new. Construction costs by area are presented in Table 21-2.

For each area listed in Table 21-2, the following disciplines were calculated individually, as applicable:

- Major earthworks and liner;
- Civil (concrete);
- Structural steel;
- Platework;
- Mechanical equipment;
- Piping;
- Electrical;
- Instrumentation; and
- Infrastructure and Buildings.

Process and infrastructure costs by discipline are presented in Table 21-3.

Table 21-2: Process Capital Cost Summary by Area

Area	Total Supply Cost (US$ 000)	Install (US$ 000)	Grand Total (US$ 000)
Area 113 - Crushing	$93,801	$23,625	$117,426
Area 115 - Agglomeration & Stacking	$42,164	$10,904	$53,067
Area 122 - Heap Leach Pad & Ponds	$57,649	$14,271	$71,919
Area 128 - Carbon Adsorption & Handling	$16,557	$6,179	$22,736
Area 129 - Desorption	$18,316	$11,086	$29,402
Area 131 - Refinery	Incl.	Incl.	Incl.
Area 134 - Reagents	$773	$343	$1,116
Area 338 - Laboratory	$2,557	$788	$3,345

Area	Total Supply Cost (US$ 000)	Install (US$ 000)	Grand Total (US$ 000)
Area 360 - Power	$7,808	$2,155	$9,963
Area 362 - Water	$2,677	$648	$3,325
Area 366 - Facilities	$208	Incl.	$208
Area 008 - Plant Mobile Equipment	$9,792	$0	$9,792
Plant Direct Capex (excluding spare parts & contingency)			**$327,673**
Sales Tax			(10,585)
Plant Total Direct Costs Before Contingency			**$311,716**

Table 21-3: Process and Infrastructure Pre-Production Capital Costs by Discipline

Discipline	Total Supply Cost (US$ 000)	Install (US$ 000)	Grand Total (US$ 000)
Major Earthworks & Liner	$61,243	$500	$61,743
Civils	$10,517	Incl.	$10,517
Structural Steelwork	$7,020	$6,649	$13,669
Mechanical Equipment & Platework	$134,309	$40,614	$174,923
Piping	$7,318	$7,351	$14,669
Electrical	$29,084	$13,866	$42,950
Instrumentation	$2,602	$1,020	$3,622
Infrastructure	$208	Incl.	$208
Plant Direct Capex (excluding spare parts & contingency)			**$327,673**
Sales Tax			(10,585)
Plant Total Direct Costs Before Contingency			**$311,716**

Sales tax was removed for the final capital estimate.

Freight, sales tax, and installation costs were also considered for each discipline. Freight costs were based on loads as bulk freight and were estimated at 10% of the equipment cost.

Sales tax for Humboldt County in Nevada is 6.85% and was applied to the supply cost of all equipment and materials.

Installation estimates were based on the equipment type and included all installation labour, tools, and equipment usage at an average hourly installation rate of $125.00 based on KCA's experience from recent projects.

Major Earthworks and Liner

Earthworks and liner quantities for the Project were estimated by KCA for all Project areas. Earthworks and liner supply, and installation were assumed to be performed by contractors. Cost estimates for these activities were developed based on recent contractor quotes. The earthworks and liner discipline also includes the cost for materials to construct the crushing retaining wall.

Civils

Civils include detailed earthworks and concrete. Concrete quantities were estimated by KCA based on layouts and similar equipment installations for other projects. Unit costs for concrete supply, which include production (supply of aggregates, water, and cement; batching; and mixing), and delivery of concrete and concrete installation, which include all excavations, formwork, rebar, placement, and curing, were based on recent contractor quotes in KCA's files.

Structural Steel

Costs for structural steel, including steel grating and handrails, were estimated based on equipment sizes and recent contractor quotes for similar projects. The costs assume the fabrication and installation of the structural steel.

Mechanical Equipment and Platework

Costs for mechanical equipment were based on a detailed equipment list for all major process equipment. Costs for all major and most minor equipment items were based on recent supplier budgetary quotes. Where recent supplier quotes were not available, reasonable allowances were made based on cost guide data and information from KCA's files. All costs assume equipment purchased new from the manufacturer or to be fabricated new.

Platework includes the costs of supplying and installing steel tanks, bins, and chutes. Platework costs were estimated based on preliminary weights and recent supplier cost information or included as part of complete equipment supply packages.

Installation costs for mechanical equipment and platework were based on estimated installation hours and hourly contractor rates from KCA's experience on recent similar projects.

Piping

Major piping, including heap irrigation and gravity solution collection pipes, was based on recent estimates from similar sized projects in the United States. An allowance of $1 million was included for water delivery and distribution for the Converse site. Additional ancillary piping, fittings, and valve costs were estimated on a percentage basis of the mechanical equipment supply costs by area, ranging from 0% to 25%.

Installation costs for piping equipment were based on estimated installation hours and hourly contractor rates from KCA's experience on recent similar projects.

Electrical

Electrical costs were estimated as percentages of the mechanical equipment supply cost for each process area and range between 0% and 30% based on benchmarked costs from recent similar projects.

Installation of electrical equipment and ancillary electrical items was estimated based on estimated installation hours and hourly contractor rates from KCA's experience on recent projects.

Instrumentation

Instrumentation costs were primarily estimated as percentages of the mechanical equipment supply cost for each process area and ranged between 0% and 2.5%. An allowance of $400,000 was included for the PLC control system. Laboratory costs are included in the mechanical equipment discipline.

Infrastructure and Buildings

New buildings for the Project will include a process office trailer, recovery plant and refinery buildings. An allowance of $75,000 was included for the septic system. Costs for the process office trailer were based on a recent budgetary quote from suppliers and reasonable allowances based on KCA experience. Costs for the recovery plant and refinery building were included in the vendor supply packages and are accounted for in the Mechanical discipline.

Process Mobile Equipment

Process mobile equipment includes a 2-ton forklift, 5-ton boom truck, 10-ton telehandler, mechanic service truck, flatbed truck, backhoe loader (CAT 430E or equivalent), skid steer, heap leach pad dozer (CAT D6 or equivalent), heap leach piping dozer (CAT D4 or equivalent), crusher area loader (CAT 992 or equivalent), and fourteen ¾ st pickup trucks. Costs for mobile equipment are included in the mechanical equipment discipline.

21.1.2.3 Indirect Costs

The Project indirect costs includes costs not directly attributable to the completion of the facilities and considers the costs that do not become a final part of the installation, but which are required for the orderly completion of the installation. This includes, but is not limited to items such as field administration, direct supervision, capital tools, startup costs, contractors' fees, insurance, taxes, etc.

The estimated indirect costs for the Project include:

- Engineering, Procurement, and Construction Management (EPCM) costs are estimated at 12.5% of the Project direct cost
- Construction Indirects estimated at 5% of the Project's direct costs. Construction Indirects includes consumables, equipment rental, shipping, contractor mobilization and demobilization, duties, tariffs, and vendors' assistance.
- Spares and first fills are estimated at 5% of the Project's direct costs. The initial fills consist of consumable items stored on site at the outset of operations, which include sodium cyanide (NaCN), cement, activated carbon, nitric acid (HNO_3), caustic soda (NaOH), antiscalants, and fluxes.
- Owners' costs estimated at 10% of the Project's direct costs. Consists of Owners' Team salaries, insurance, legal, consultants, safety, environmental, and security costs during the construction period.

21.1.2.4 Contingency

AACE International defines a contingency as "An amount added to an estimate to allow for items, conditions or events for which the state, occurrence and/or effect is uncertain and that experience shows will likely result, in the aggregate, in an additional cost". Contingency is expected to be spent during the execution of the work.

Contingency for the Project, based on the estimator's experience and the level of design, was estimated at 25% of the direct costs.

Contingency for sustaining capital for processes and infrastructure was estimated at 25% of the sustaining costs. Sustaining capital contingency is included in the sustaining cost estimate.

21.1.3 Sustaining Capital Costs

The sustaining capital cost estimate for the Project was developed on the same basis as the initial capital cost. KCA developed the processing sustaining capital and SLR estimated the mining sustaining capital. SLR found the sustaining capital costs to be an acceptable representation of the Project.

The total sustaining capital costs are estimated to be $514 million over the LOM, as summarized in the Table 21-4.

Table 21-4: Sustaining Capital Cost Summary

Sustaining Capital Costs	US$ million
Mine Equipment Lease	188
Mine Equipment Sustaining	177
Processing Sustaining	148
Total LOM Sustaining Capital Costs	**514**

21.1.3.1 Mining Sustaining Capital

Mining sustaining capital comprises two primary components: the settlement of initial equipment lease obligations and the ongoing procurement of replacement and expansion fleet as defined in Section16.9.7.

Following the initial 20% down payment described in Section 21.1.2.1, the remaining 80% of the initial fleet value is serviced through a leasing schedule. This financing obligation, which includes principal and the 6% annual interest rate, amounts to $188 million over the first 6 years of operation (Table 21-4). Upon completion of these payments, full ownership of the initial fleet is transferred to the Project.

Independent of the leasing schedule, $177 million is allocated for sustaining capital throughout the remainder of the LOM. This expenditure accounts for the following costs:

- Periodic Replacement: Replacing primary and ancillary units as they reach the end of their effective service life (e.g., engine hours and frame-life limits).
- Fleet Expansion: The acquisition of additional haulage units required to maintain production as the pit deepens and cycle times increase.

- Outright Procurement: Unlike the initial fleet, these subsequent requirements are staggered and are planned as outright purchases to simplify the Project's long-term capital structure.

All replacement timings and life-cycle costs are scheduled according to mechanical availability benchmarks and life-cycle hour standards provided by the 2025 SHERPA dataset.

21.1.3.2 Process Sustaining Capital

Sustaining capital for process and infrastructure includes the costs for constructing Phase 2 and Phase 3 of the leach pad in future operations including the expansion of the overland conveyor and barren piping. Contingency, construction indirect costs, and EPCM were included in the sustaining capital estimates as a percentage of the direct costs. Total process and infrastructure sustaining capital was estimated at $148.0 million including $103.0 million in direct costs, $25.7 million for EPCM and construction indirect costs, and $19.3 million for contingency.

21.1.4 Closure and Reclamation Capital

Closure and reclamation costs have been estimated at a conceptual level for the PEA. Closure costs are assumed to be incurred over a two-year period following cessation of operations, reflecting a conceptual closure schedule without detailed sequencing or progressive reclamation assumptions. The estimate ($100M) is based on high-level disturbance assumptions and benchmarking against similar Nevada heap leach operations. The estimate has not been developed using the Nevada Standardized Reclamation Cost Estimator (SRCE), consistent with the current level of study, and is subject to significant uncertainty (±50% or greater). Advancement to the next level of study should include the development of SRCE-level inputs and a site-specific closure plan to further refine this estimate and timeline, as part of permitting and financial assurance requirements in Nevada.

21.2 Operating Costs

The LOM mine operating costs total $4,757 million during the production period, and are estimated to be $16.01/t processed, as summarized in Table 21-5. The costs were estimated in Q1 2026 US dollars.

Table 21-5: Operating Cost Summary

Operating Costs	LOM Costs (US$ million)	LOM Average (US$ million/year)	Unit Costs (US$/t processed)
Open Pit Mining Costs	2,601	186	8.75
Processing Costs	1,772	127	5.96
G&A and Support Costs	384	27	1.29
Total Operating Costs	**4,757**	**340**	**16.01**

Notes:
1. Table values may not sum due to rounding.
2. Considers operating costs during the 14 years of the production period.

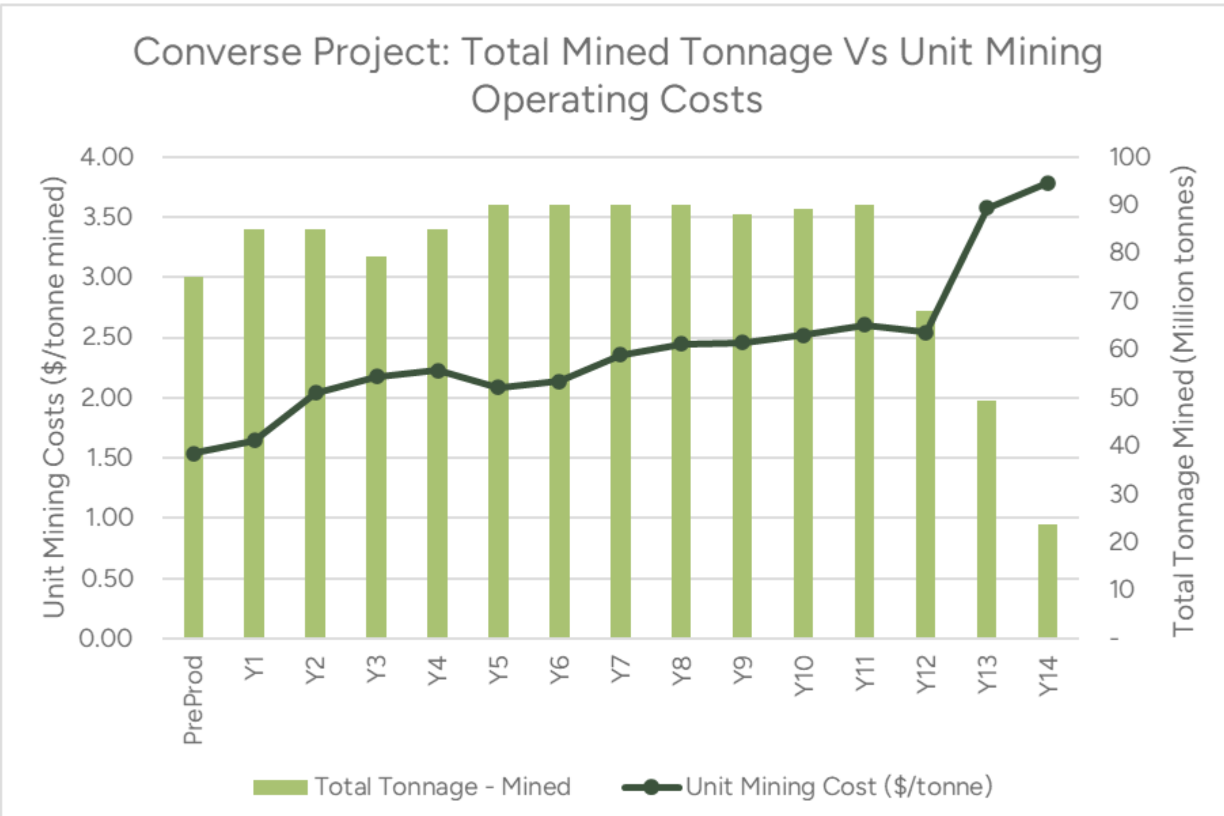
21.2.1 Mining Operating Costs

The LOM mine operating costs were calculated using the 2025 SHERPA dataset, incorporating site-specific parameters including material densities, equipment Key Performance Indicators

(KPIs), and evolving haulage distances and pit depths over the LOM. The operating cost model assumes an owner-operator framework with a diesel fuel price of $0.90/L and an electrical power cost of $0.08/kWh for the cable shovel fleet. Operational cost factors were further refined to account for the varying proportions of alluvial waste and bedrock material encountered as the pit progresses.

The total LOM mine operating cost is estimated at $2,601 million (Table 21-5), resulting in an average unit rate of $2.36/t mined or $8.75/t processed. This estimate specifically excludes the pre-production period and its associated capitalized development costs. As illustrated in the unit cost profile in Figure 21-1, the costs vary throughout the mine life in response to shifting pit geometries, increasing haulage cycle times, and the transition into deeper bedrock units within the final 620 m deep pit.

Figure 21-1: Converse Project: Unit Mining Operating Costs



To ensure regional accuracy, the results from the SHERPA 2025 Dataset were reconciled against current operating benchmarks and recent PFS/FS-level studies for similar large-scale open pit projects within the region.

21.2.2 Process Operating Costs

Average Annual process and support services operating costs for the Converse deposits are presented in Table 21-6.

Table 21-6: Process and Support Services Annual Operating Costs

Area	Unit	Quantity	Unit Cost (US$)	Annual Cost (US$)	US$/t Processed
Labour - All Process Areas					
Process Labor	persons	132		$17,595,548	$0.782
Sub-total				$17,595,548	$0.782
Area 113 - Crushing					
Power	kWh/t	2.775	$0.09	$5,669,941	$0.252
992 Loader	h/yr	2,920	$190.73	$556,917	$0.025
Gyratory Crusher Liners	kg/yr	155,567	$4.60	$715,125	$0.032
Cone Crusher Liners	kg/yr	393,441	$3.02	$1,188,223	$0.053
HPGR Tires	set/yr	1.5	$1,498,164	$2,247,245	$0.100
Screen Panels	set/yr	6	$70,000	$420,000	$0.019
Operating Supplies	$/yr			$1,125,000	$0.050
Maintenance Supplies	$/yr			$2,250,000	$0.100
Sub-total				$14,172,107	$0.630
Area 115 - Agglomeration & Stacking					
Power	kWh/t	1.439	$0.09	$2,939,662	$0.131
Cement	kg/t	6	$0.33	$44,550,000	$1.980
Operating Supplies	$/yr			$1,125,000	$0.050
Maintenance Supplies	$/yr			$2,250,000	$0.100
Sub-total				$50,864,662	$2.261
Area 122 - Heap Leach Pad & Ponds					
Power	kWh/t	1.118	$0.09	$2,283,513	$0.101
Heap Dozer (D6 or equivalent)	h/yr	4,380	$35	$151,357	$0.007
Heap Dozer (D4 or equivalent)	h/yr	832	$55	$45,779	$0.002
Piping	$/yr			$225,000	$0.010
Drip tube	m/d	14,000	$0.40	$2,044,000	$0.091
Operating Supplies	$/yr			$1,125,000	$0.050
Maintenance Supplies	$/yr			$1,125,000	$0.050
Sub-total				6,999,649	$0.311
Area 128 - Carbon Adsorption & Handling					
Power	kWh/t	0.055	$0.09	$112,981	$0.005
Carbon Consumption	kg/t	0.005	$3.28	$394,186	$0.018

Area	Unit	Quantity	Unit Cost (US$)	Annual Cost (US$)	US$/t Processed
Operating Supplies	$/yr			$112,500	$0.005
Maintenance Supplies	$/yr			$337,500	$0.015
Sub-total				$957,167	$0.043
Area 129 - Desorption					
Power	kWh/t	0.249	$0.09	$508,480	$0.023
Cyanide, strip	kg/t	0.001	$2.30	$40,547	$0.002
Caustic, strip	kg/t	0.004	$0.71	$62,373	$0.003
Nitric Acid	kg/t	0.013	$1.10	$328,403	$0.015
Misc. Operating Supplies	$/yr			$562,500	$0.025
Maintenance Supplies	$/yr			$562,500	$0.025
Sub-total				$2,064,803	$0.092
Area 131 - Refinery					
Power	kWh/t	0.084	$0.09	$172,371	$0.008
Fluxes	kg/t	0.002	$0.22	$11,256	$0.001
Misc. Operating Supplies	$/yr			$225,000	$0.010
Maintenance Supplies	$/yr			$112,500	$0.005
Sub-total				521,127	$0.023
Area 134 - Reagents					
Power	kWh/t	0.002	$0.09	$4,060	$0.000
Cyanide, leach	kg/t	0.641	$2.30	$33,226,958	$1.477
Antiscalant	kg/t	0.012	$4.63	$1,209,164	$0.054
Maintenance Supplies	$/yr			$1,125,000	$0.050
Sub-total				$35,565,182	$1.581
Area 338 - Laboratory					
Power	kWh/t	0.099	$0.09	$201,338	$0.009
Fire Assays, Solids	#/day	150	20	$1,095,000	$0.049
CN Soluble Assays, Solids	#/day	150	15	$821,250	$0.037
Assays, Solutions	#/day	350	10	$1,277,500	$0.057
Misc. Operating Supplies	$/yr			$562,500	$0.025
Maintenance Supplies	$/yr			$225,000	$0.010
Sub-total				$4,182,588	$0.186
Area 360 - Power					
Diesel	L/yr	9,084	$0.76	$6,911	$0.000

Area	Unit	Quantity	Unit Cost (US$)	Annual Cost (US$)	US$/t Processed
Maintenance Supplies	$/yr			$112,500	$0.005
Sub-total				$119,411	$0.005
Area 362 - Water					
Power	kWh/t	0.033	$0.09	$66,748	$0.003
Maintenance Supplies	$/yr			$225,000	$0.010
Sub-total				$291,748	$0.013
Area 366 - Facilities					
Power	kWh/t	0.004	$0.09	$8,948	$0.000
Sub-total				$8,948	$0.000
Area 008 - Plant Mobile Equipment					
Clean Up Loader	h/mo	122	$37.13	$54,212	$0.002
Fork Lift	h/mo	122	$10.35	$15,113	$0.001
Boom Truck	h/mo	17	$67.61	$14,063	$0.001
Crane, 85 ton	h/mo	17	$83.14	$17,293	$0.001
Mechanic Service Truck	h/mo	122	$21.83	$31,866	$0.001
Backhoe/Loader	h/mo	122	$30.42	$44,407	$0.002
Flatbed Truck	h/mo	122	$35.27	$51,490	$0.002
Excavator	h/mo	122	$30.19	$44,076	$0.002
Pickup Truck	h/mo	1703	$26.46	$540,857	$0.024
Sub-total				$813,377	$0.036
TOTAL				**$134,156,663**	**$5.963**

21.2.2.1 Personnel and Staffing

KCA estimated staffing requirements for the process based on experience with similar sized operations. The total number of process personnel was estimated at 132, including 21 laboratory workers.

21.2.2.2 Power

Power usage for the process and process-related infrastructure was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage and cost. The Project's power requirements are presented in Section 18.0.

Power will be supplied by an existing transmission line. The approximate power cost is estimated at $0.09/kWh and is based on pricing obtained from a recent project in close proximity to the site.

21.2.2.3 Consumable Items

Operating supplies were estimated based upon unit costs and consumption rates predicted by metallurgical tests and have been broken down by area. Freight costs were included in all operating supply and reagent estimates. Reagent consumption was estimated from test work and design criteria. Other consumable items were estimated by KCA based on experience with other similar operations.

Operating costs for consumable items were distributed based on tonnage and gold production or smelting batches, as appropriate.

21.2.2.4 Laboratory

Fire assaying and solution assaying of samples will be conducted in the onsite laboratory. It was estimated that approximately 150 solid assays and solution assays will need to be performed each day.

21.2.2.5 Miscellaneous Operating and Maintenance Supplies

Overhaul and maintenance of equipment, along with miscellaneous operating supplies for each area were estimated as allowances based on tonnes processed. The allowances for each area were developed based on published data as well as KCA's experience with similar operations.

21.2.2.6 Mobile Support Equipment

Mobile and support equipment will be required for the process and include a 2-ton forklift, 5-ton boom truck, 10-ton telehandler, mechanic service truck, flatbed truck, backhoe loader (CAT 430E or equivalent), skid steer, heap leach pad dozer (CAT D6 or equivalent), crusher area loader (CAT 992 or equivalent), personnel van and six each ¾ ton pickup trucks. The costs to operate and maintain each piece of equipment were estimated primarily using published information and project specific fuel costs. Where published information was not available, allowances were made based on KCA's experience from similar operations.

21.2.3 General and Administrative (G&A)

SLR estimates an annual G&A cost of $27.9 million, equivalent to $1.24/t. The cost items included in G&A include the following:

- Vehicle maintenance and materials
- Road maintenance and materials
- General Manager salary
- Recruiting
- Training
- Site administration labour
- Security
- Health and safety
- Environmental sampling and analysis
- IT and telecommunications

- Contract services such as legal, consulting, audits, and insurance
- Cyanide code annual fees
- Cost exclusions are camps and employee transport

22.0 Economic Analysis

The economic analysis in this Technical Report was prepared in accordance with paragraphs 2.3(1)(b) and (c), subsections 2.3(3) and (4), and paragraph 3.4(e), of the Instrument, including any required cautionary language, which is stated with equal prominence.

The economic analysis contained in this Technical Report is based, in part, on Inferred Mineral Resources and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this Preliminary Economic Assessment is based will be realized.

SLR notes that the economic analysis presented in this section is based on revenue from gold only. An after-tax Cash Flow Projection has been generated from the LOM production schedule and capital and operating cost estimates, and is summarized in Table 22-1.

A summary of the key parameters and assumptions for the economic analysis is provided below.

22.1 Economic Criteria

22.1.1 Revenue

- Mine life: Starts with 9 months of mine pre-production development, followed by 13.75 years of production.
- Peak mining rate of 90 Mtpa per year (mineralized material and waste).
- Total LOM mineralized material feed to process: 299,795 kt, with an average of 59,000 tpd processed (considering full production years) at LOM average grade of 0.51 g/t Au.
- Contained gold: 4,910 koz Au.
- Average LOM process gold recovery: 70.9%.
- Recovered gold over the LOM totals 3,482 koz Au, with an average gold production of 246 koz/yr (years of full production).
- Gold payable is assumed at 99.5% based on the benchmark from nearby projects.
- Annual average gold sales of 245 koz Au payable per year (full production years).
- Gold price is based on analyst consensus price forecasts from March 2026. For the economic analysis was assumed:
    - Y-1 and Y1: US$4,000/oz Au
    - Y2 to Y14: US$3,600/oz Au
- Transportation, logistic and refining charges assumed at US$1.14/oz Au, based on benchmark from nearby projects.
- NSR royalty of 6% payable to Royalty Consolidation Company, LLC (RCC Royalty).
- Revenue is recognized at the time of gold production. Gold production starts in the pre-production period.

- LOM net revenue is US$11,803 million (after royalty, and transportation and refining charges).

22.1.2 Costs

- Pre-production capital costs were estimated at $829 million, including a 25% contingency. See Table 21-1 for the pre-production capital costs breakdown.
- Mine life sustaining capital costs were estimated at $514 million.
- Mine equipment lease. Roxmore plans to use a mine equipment lease strategy to obtain the initial equipment for the pre-production period. This strategy assumes a 20% downpayment and lease payments at a 6% annual rate over 6 years. Annual lease payment was estimated at US$31.4 million.
- Mine equipment additions and replacement over the LOM were estimated by SLR at US$177 million.
- Process sustaining estimated by KCA at US$148 million over the LOM.
- Capitalized operating costs for processing and administrative activities during pre-production.
- Final reclamation costs assumed at US$100 million at the end of the LOM.
- Average LOM operating cost is US$16.01/t processed.
- Open pit operating costs of US$2.36/t mined ($8.75/t processed).
- Processing operating costs of US$5.96/t processed.
- Site services and general and administrative (G&A) costs of US$27.9 million/yr (LOM average of $1.29/t processed).
- Total All-in Sustaining Cost (AISC) is approximately US$1,769/oz Au.

22.1.3 Taxation and Royalties

- The Project is subject to a state income tax rate of 5.0% and a federal income tax rate of 21.0%.
- LOM taxes total approximately US$974 million.
- The entire Property is subject to an NSR production royalty payable to RCC Royalty on the sale of any minerals from the Property. The RCC Royalty rate is 6%, except as to those portions of the Property that were subject, as of the date of the RCC Royalty grant, to existing royalty obligations, in which case the RCC Royalty rate is the difference between 6% and the rate of the existing royalty obligations. Effectively, the RCC Royalty means that the Property is subject to a blanket 6% NSR royalty on the production of all minerals.

22.2 Cash Flow Analysis

SLR developed, for the Converse Project, an unlevered after-tax cash flow model that consolidates physicals, costs, and revenue.

The inputs for the cash flow model for Converse were developed by the following considerations:

- Production schedule by SLR.
- Metallurgy assumptions by KCA.
- Revenue is based on analyst consensus price forecasts from March 2026.
- Capital and operating costs were developed by KCA (processing) and SLR (others).

All costs are in Q1 2026 US$ dollars with no allowance for inflation.

The Converse unlevered after-tax cash flow model excludes the following components:

- Escalation beyond the date of our source inputs.
- Effect of tariffs on non-US products or supplies.
- Financing costs.
- Insurance.
- Overhead cost for a corporate office.

The annual after-tax cash flow summary is presented in Table 22-1

Table 22-1: After-Tax Cash Flow Summary

Economic Model Annual Summary

SLR

Company: Roxmore Resources Inc.
Project Name: Converse OP Project
Scenario Name: Base Case
Analysis Type: PEA & NI-43-101

Calendar Year				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Project Stage				Pre-Prod.	Pre-Prod.	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Ops	Closure	Closure	Closure
Time Until Closure In Years		US$ & Metric Units	LoM Avg / Total	16	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2	-3
Market Prices																						
Gold, Forecast	/	US$/oz	$3,626	-	4,000	4,000	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800	3,800
Physicals																						
Total Mineralized Material Mined		kt	399,795	-	2,023	13,949	22,500	23,500	31,602	22,500	22,500	23,103	22,500	22,500	22,500	18,434	22,500	22,500	16,505	-	-	-
Total Waste Mined		kt	877,578	-	73,377	71,051	62,500	55,745	63,425	67,500	67,500	66,897	67,500	65,500	66,613	71,566	45,390	26,765	7,246	-	-	-
Total Material Mined		kt	1,177,373	-	75,000	85,000	85,000	79,245	95,027	90,000	90,000	90,000	90,000	88,000	89,113	90,000	67,890	49,265	23,751	-	-	-
Stripping Ratio		W:O	2.95	-	37.59	5.09	2.78	2.37	2.94	3.00	3.00	2.90	3.00	2.91	2.96	3.88	2.02	1.19	0.44	-	-	-
Total Mineralized Material Processed		kt	399,795	-	2,023	13,949	22,500	23,500	31,602	22,500	22,500	23,103	22,500	22,500	22,500	18,434	22,500	22,500	16,505	-	-	-
Gold Grade, Processed		g/t	0.51	-	1.01	0.62	0.57	0.59	0.35	0.51	0.52	0.55	0.53	0.47	0.31	0.34	0.41	0.59	0.61	-	-	-
Contained Gold, Processed		koz	4,910	-	66	278	411	446	352	367	376	407	385	338	226	202	297	430	326	-	-	-
Average Recovery, Gold		%	70.9%	-	61.7%	64.5%	68.7%	64.2%	69.5%	76.8%	76.5%	75.6%	74.9%	72.8%	71.1%	68.6%	71.1%	72.4%	71.8%	-	-	-
Recovered Gold, Processed		koz	3,482	-	42	178	282	288	244	282	287	306	288	246	161	138	212	311	234	-	-	-
Finished Goods - Gold		koz	3,482	-	42	178	282	288	244	282	287	306	288	246	161	138	212	311	234	-	-	-
Payable Gold, Total		koz	3,465	-	42	177	280	288	243	281	286	306	287	245	160	138	211	310	233	-	-	-
Cash Flow																						
Gold Gross Revenue	100.00%	$000s	12,580,665	-	166,025	707,996	937,754	1,031,297	875,260	1,010,302	1,029,381	1,102,234	1,033,117	882,088	578,666	495,459	756,078	1,114,924	839,241	-	-	-
Total Gross Revenue		$000s	12,580,665	-	166,025	707,996	937,754	1,031,297	875,260	1,010,302	1,029,381	1,102,234	1,033,117	882,088	578,666	495,459	756,078	1,114,924	839,241	-	-	-
Mining Cost		$000s	(2,801,419)	-	-	(139,869)	(173,536)	(173,382)	(169,265)	(167,850)	(192,390)	(212,130)	(220,231)	(216,196)	(224,372)	(234,487)	(172,572)	(178,145)	(90,193)	-	-	-
Process Cost		$000s	(1,772,036)	-	-	(83,176)	(134,168)	(140,131)	(128,611)	(134,168)	(134,167)	(137,762)	(134,168)	(134,167)	(134,168)	(109,920)	(134,168)	(134,168)	(96,696)	-	-	-
G&A Cost		$000s	(383,625)	-	-	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(27,900)	(20,925)	-	-	-
Refining and Freight Cost		$000s	(3,963)	-	(183)	(245)	(277)	(287)	(270)	(285)	(287)	(294)	(287)	(271)	(238)	(230)	(258)	(296)	(266)	-	-	-
Royalties		$000s	(753,401)	-	(9,996)	(42,485)	(56,249)	(61,081)	(52,501)	(60,601)	(61,748)	(66,116)	(61,970)	(52,909)	(34,507)	(29,714)	(45,469)	(66,878)	(50,336)	-	-	-
Subtotal Cash Costs Before By-Product Credits		$000s	(5,514,464)	-	(10,191)	(293,475)	(392,131)	(402,560)	(398,724)	(410,783)	(416,489)	(444,202)	(444,555)	(431,443)	(421,265)	(402,251)	(378,366)	(405,385)	(259,618)	-	-	-
By-Product Credits		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Cash Costs After By-Product Credits		$000s	(5,514,464)	-	(10,191)	(293,475)	(392,131)	(402,560)	(398,724)	(410,783)	(416,489)	(444,202)	(444,555)	(431,443)	(421,265)	(402,251)	(378,366)	(405,385)	(259,618)	-	-	-
Operating Margin	56%	$000s	7,066,201	-	155,834	414,521	545,622	628,737	476,536	599,519	612,892	658,032	588,562	450,644	156,421	93,208	377,712	709,539	579,623	-	-	-
Other Admin Expenses		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EBITDA		$000s	7,066,201	-	155,834	414,521	545,622	628,737	476,536	599,519	612,892	658,032	588,562	450,644	156,421	93,208	377,712	709,539	579,623	-	-	-
Depreciation Allowance		$000s	(1,342,588)	-	(9,979)	(46,665)	(74,945)	(85,435)	(78,704)	(96,558)	(108,834)	(119,320)	(117,469)	(103,794)	(70,077)	(63,609)	(102,522)	(150,781)	(113,496)	-	-	-
Earnings Before Taxes		$000s	5,703,613	-	145,856	367,856	470,677	543,302	397,832	502,961	504,058	538,712	471,093	346,851	86,344	29,600	275,190	558,757	466,126	-	-	-
Federal & State Income Tax		$000s	(974,062)	-	(104)	(66,573)	(64,444)	(99,272)	(68,467)	(89,946)	(89,599)	(95,632)	(81,193)	(65,507)	(10,572)	(3,693)	(41,991)	(100,137)	(86,524)	-	-	-
Net Income		$000s	4,729,551	-	145,471	300,984	406,233	444,030	329,365	413,015	414,459	443,080	389,900	281,343	74,175	25,907	233,200	458,571	379,601	-	-	-
Non-Cash Add Back - Depreciation		$000s	1,342,588	-	9,979	46,665	74,945	85,435	78,704	96,558	108,834	119,320	117,469	103,794	70,077	63,609	102,522	150,781	113,496	-	-	-
Working Capital		$000s	(0)	(8,101)	(15,743)	(17,118)	(6,294)	(4,340)	4,065	(6,662)	(2,949)	(2,496)	1,923	5,496	12,307	1,172	(11,730)	(14,525)	4,556	61,437	-	-
Operating Cash Flow		$000s	6,072,118	(8,101)	139,707	330,530	474,885	525,124	412,134	502,910	520,344	559,904	509,292	400,833	157,166	90,688	323,991	594,827	496,657	61,437	-	-
Initial Capital		$000s	(826,986)	(270,045)	(556,941)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalized Opex		$000s	(29,593)	-	(29,593)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$000s	(513,601)	-	-	(83,460)	(70,130)	(31,404)	(90,756)	(49,999)	(75,001)	(16,500)	(31,066)	(16,500)	(16,500)	(16,690)	(10,596)	-	-	-	-	-
Closure/Reclamation/Monitoring Costs		$000s	(100,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(40,000)	(60,000)	-
Total Capital		$000s	(1,472,180)	(270,045)	(586,534)	(83,460)	(70,130)	(31,404)	(90,756)	(49,999)	(75,001)	(16,500)	(31,066)	(16,500)	(16,500)	(16,690)	(10,596)	-	-	(40,000)	(60,000)	-
Cash Flow Adj./Reimbursements		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LoM Metrics																						
Economic Metrics																						
Discount Rate	BOP	5%		0.9524	0.9070	0.8638	0.8227	0.7835	0.7462	0.7107	0.6768	0.6446	0.6139	0.5847	0.5568	0.5303	0.5051	0.4810	0.4581	0.4363	0.4155	0.3957
a) Pre-Tax																						
Free Cash Flow		$000s	5,574,020	(278,146)	(448,643)	313,943	461,190	592,993	411,044	542,857	531,942	637,036	558,410	437,940	148,229	75,690	347,387	695,014	583,101	21,437	(60,000)	-
Cumulative Free Cash Flow		$000s		(278,146)	(726,789)	(412,846)	48,352	641,344	1,053,188	1,596,046	2,127,988	2,765,024	3,324,443	3,762,083	3,911,311	3,987,001	4,334,388	5,029,402	5,612,503	5,634,020	5,574,020	5,574,020
NPV @ 5%		$000s	3,391,119	(264,901)	(406,932)	271,196	379,429	464,626	307,324	385,789	360,039	410,639	343,434	256,079	83,066	40,140	175,484	334,314	267,162	9,353	(24,931)	-
Cumulative NPV @ 5%		$000s		(264,901)	(406,932)	6,294	385,723	850,349	1,157,673	1,543,471	1,903,511	2,314,150	2,657,584	2,913,464	2,996,530	3,036,700	3,212,154	3,546,467	3,813,629	3,822,983	3,798,051	3,798,051
IRR		%	60.3%																			
Payback Period		years	1.9			1.0	0.9															
b) After-Tax																						
Free Cash Flow		$000s	4,599,936	(278,146)	(448,027)	267,075	376,754	493,720	343,376	452,911	442,543	541,405	478,226	392,133	158,667	71,887	306,397	594,827	496,657	21,437	(60,000)	-
Cumulative Free Cash Flow		$000s		(278,146)	(725,974)	(479,903)	(103,149)	390,571	733,949	1,186,860	1,629,203	2,170,608	2,648,834	3,030,967	3,189,623	3,241,821	3,547,017	4,141,844	4,638,501	4,659,938	4,599,938	4,599,936
NPV @ 5%		$000s	2,749,445	(264,901)	(407,100)	213,429	309,957	386,840	256,234	321,876	299,595	348,994	293,569	223,426	77,209	38,103	154,346	286,122	227,524	9,353	(24,931)	-
Cumulative NPV @ 5%		$000s		(264,901)	(672,001)	(458,572)	(148,616)	238,227	494,461	816,336	1,115,731	1,464,726	1,758,315	1,981,740	2,058,950	2,097,131	2,251,377	2,537,499	2,765,023	2,774,376	2,749,445	2,749,445
IRR		%	42.0%																			
Payback Period		years	2.2			1.0	1.0	0.2														
Operating Metrics																						
Mine Life		Years	14																			
Average Daily Mining Rate		t/d mined	241,666	-	208,402	242,857	242,857	226,413	242,934	257,143	257,143	257,143	257,143	251,429	254,607	257,143	193,976	140,758	91,604	-	-	-
Average Daily Processing Rate		t/d placed	59,000	-	9,539	38,216	61,644	64,384	59,103	61,644	61,644	63,295	61,644	61,644	61,644	50,503	61,644	61,644	60,309	-	-	-
Mining Cost		$ / t mined	$2.38	-	-	1.64	2.04	2.18	2.23	2.09	2.14	2.36	2.45	2.46	2.52	2.61	2.54	3.58	3.78	-	-	-
Mining Cost		$/t process	$6.75	-	-	10.01	7.71	7.34	8.76	6.35	8.55	9.18	9.79	9.61	9.97	12.72	7.67	7.83	5.44	-	-	-
Processing Cost		$/t process	$5.96	-	-	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	5.96	-	-	-
G&A Cost		$/t process	$1.29	-	-	2.00	1.24	1.19	1.29	1.24	1.24	1.21	1.24	1.24	1.24	1.51	1.24	1.24	1.26	-	-	-
Subtotal Direct Operating Costs		$/t process	$16.01	-	-	17.98	14.92	14.49	16.02	15.55	15.75	16.35	16.99	16.81	17.18	20.20	14.87	15.03	12.66	-	-	-
Refining and Freight Cost		$/t process	$0.01	-	0.07	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02	-	-	-
NSR Royalty		$/t process	$2.54	-	3.01	3.04	2.50	2.60	2.40	2.69	2.74	2.86	2.75	2.35	1.54	1.61	2.03	2.97	3.04	-	-	-
Total Operating Cost		$/t process	$18.56	-	3.08	21.04	17.43	17.13	18.46	18.26	18.51	19.23	19.76	19.19	18.72	21.82	16.91	18.02	15.71	-	-	-
Sales Metrics																						
Au Sales		koz	3,465	-	42	177	280	288	243	281	286	306	287	245	160	138	211	310	233	-	-	-
Total Cash Cost (LOM)		$ / oz Au	1,592																			
Total AISC (LOM)		$ / oz Au	1,766			2,130	1,605	1,515	1,923	1,642	1,739	1,511	1,657	1,838	2,745	3,059	1,695	1,309	1,118	-	-	-
Avg. LOM Annual Au Sales (excl. rinsing phase)		koz/yr	248																			

SLR prepared a LOM unlevered after-tax cash flow model to confirm the economics of the Converse Project over the LOM. Economics have been evaluated using the discounted cash flow method, considering annual processed tonnages and the associated gold grades. The process gold recovery, gold price forecast, operating costs, refining and transportation charges, royalties, and initial and sustaining capital expenditures were also considered.

The economic analysis demonstrates that the Project's Mineral Resources have reasonable prospects for economic extraction at the LOM net average realized prices of $3,625/oz Au, with a long-term price of $3,600/oz Au, and that further advancement of Project studies is warranted.

The base discount rate assumed in this Technical Report is 5% as per standard industry practice for evaluating precious-metal projects in North America. Discounted present values of annual cash flows are summed to arrive at the Project's Base Case NPV.

The Project's pre-tax NPV at a 5% discount rate is approximately US$3,391 million, and the pre-tax internal rate of return (IRR) is approximately 50.3%. The Project's after-tax NPV at a 5% discount is approximately US$2,749 million, the after-tax IRR is approximately 42.8%, and the payback period is approximately 2.2 years from the start of production.

The Project's undiscounted pre-tax cash flow is approximately US$5,574 million, and the undiscounted after-tax cash flow is approximately US$4,600 million.

SLR has also run a stand-alone economic analysis for the Project using flat resource metal prices of US$3,000/oz for gold, and the analysis demonstrates that the Project's Mineral Resources also have reasonable prospects for economic extraction at these prices.

The World Gold Council Adjusted Operating Cost (AOC) is US$1,592/oz Au. The mine life sustaining capital unit cost is US$177/oz Au, for an AISC of US$1,769/oz Au.

The Project's average annual gold sales during the LOM are approximately 245 koz Au per year, considering years of full production.

22.3 Sensitivity Analysis

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Gold price
- Gold recovery
- Gold head grade
- Operating costs
- Capital costs (Initial, Sustaining, and Closure costs)

Where possible, the after-tax NPV 5% and IRR sensitivities relative to the base case have been calculated for -20% to +20% variations in head grade and recovery, and -30% to 40% in gold price. Operating and capital cost sensitivities have been calculated at -15% to +35% variations. The sensitivities are shown in Table 22-2 and Figure 22-1 and Figure 22-2.

Table 22-2: After-Tax Sensitivity Analyses

Variance	Head Grade (g/t Au)	NPV at 5% (US$ million)	IRR (%)
80%	0.41	1,534	28.3%
90%	0.46	2,143	35.7%
100%	**0.51**	**2,749**	**42.8%**
110%	0.56	3,358	49.5%
120%	0.61	3,967	56.1%
Variance	**Recovery (% Au)**	**NPV at 5% (US$ million)**	**IRR (%)**
80%	57%	1,534	28.3%
90%	64%	2,143	35.7%
100%	**71%**	**2,749**	**42.8%**
110%	78%	3,358	49.5%
120%	85%	3,967	56.1%
Variance	**Metal Prices (US$/oz Au)**	**NPV at 5% (US$ million)**	**IRR (%)**
69%	$2,500	884	19.6%
85%	$3,060	1,840	32.1%
100%	**$3,600**	**2,749**	**42.8%**
120%	$4,320	3,967	56.1%
139%	$5,000	5,111	68.1%
Variance	**Operating Costs (US$/t)**	**NPV at 5% (US$ million)**	**IRR (%)**
85%	$13.61	3,226	48.3%
93%	$14.81	2,988	45.5%
100%	**$16.01**	**2,749**	**42.8%**
118%	$18.81	2,193	36.2%
135%	$21.61	1,637	29.3%
Variance	**Capital Costs (US$000)**	**NPV at 5% (US$ million)**	**IRR (%)**
85%	$1,251	2,934	51.4%
93%	$1,362	2,842	46.7%
100%	**$1,472**	**2,749**	**42.8%**
118%	$1,730	2,534	35.3%
135%	$1,987	2,319	29.7%

Figure 22-1: After-Tax NPV 5% Sensitivity Analysis

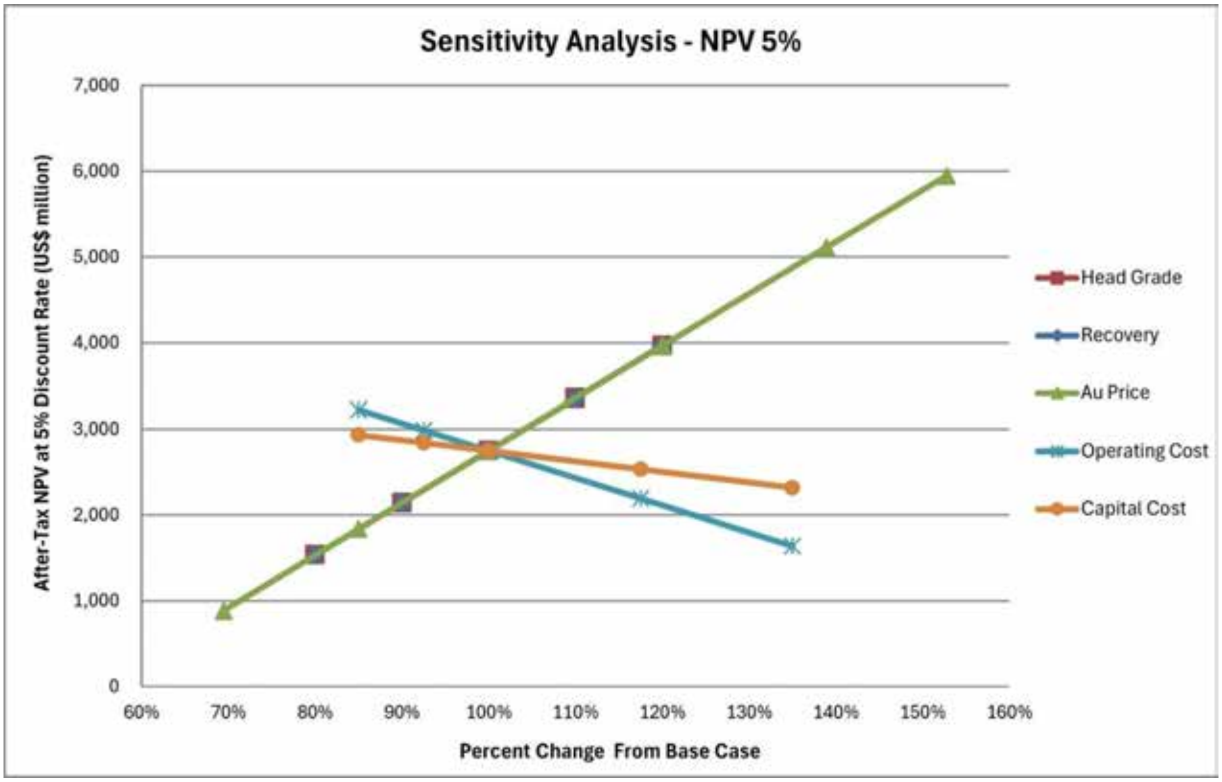
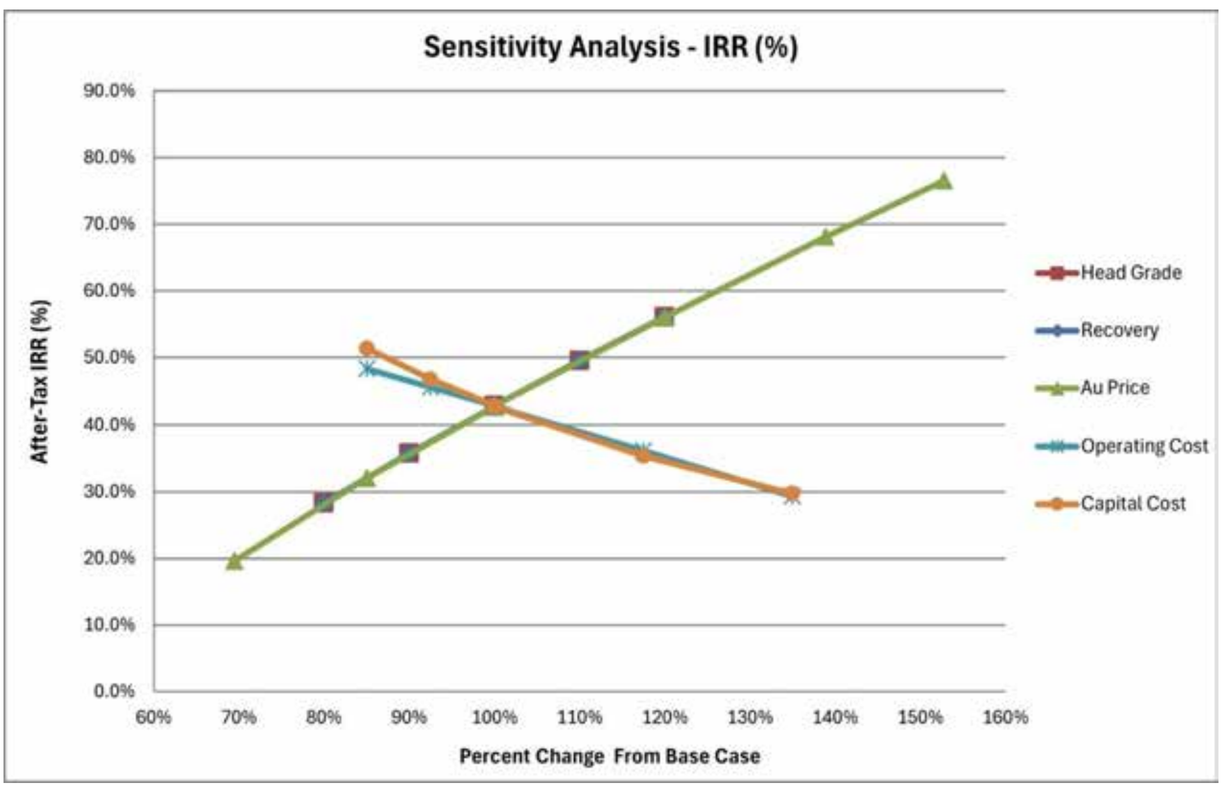


Figure 22-2: After-Tax IRR Sensitivity Analysis

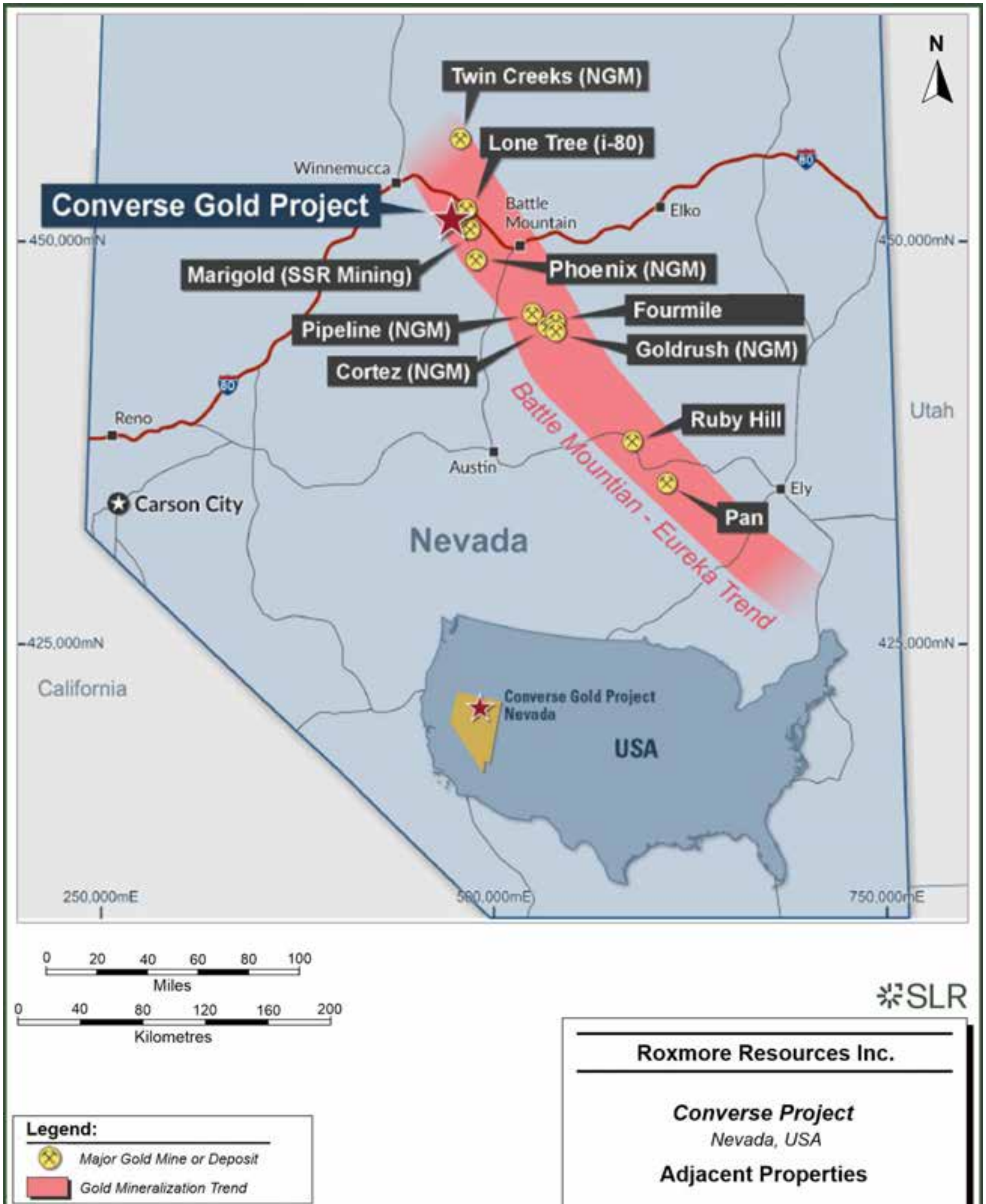


23.0 Adjacent Properties

The Converse Property is located along the western margin of the Battle Mountain Gold Belt, a northwest-trending mineralized corridor extending from the Twin Creeks deposits in the north to the Pan deposit in the south (Figure 23-1). The past-producing Lone Tree Mine, now owned by i-80 Gold, lies approximately 13.7 km (8.5 mi) north-northeast of the Converse Property and has reported historical production of approximately 4.6 million ounces of gold (GeoGlobal 2025). SSR Mining's Marigold Mine is situated roughly 8 km (5 mi) to the east. According to a 2024 technical report (SLR 2024), Marigold has produced approximately 4.8 million ounces of gold as of September 2023.

The QP has not independently verified this information, and this information is not necessarily indicative of the mineralization at the Converse Property.

Figure 23-1: Adjacent Properties



Source: Roxmore 2026.

24.0 Other Relevant Data and Information

24.1 Silver

The Roxmore database contains approximately 17,600 historical silver assays. Of these, 7,400 samples were analyzed using a four-acid digestion, roughly 200 samples were analyzed by fire assay, and 9,900 samples were analyzed using a two-acid ICP-MS method. An additional 100 samples have unknown analytical methods.

There are also approximately 16,000 intervals that contain a gold assay with no corresponding silver assay. To classify silver to the same confidence category as gold in the current Mineral Resource estimate, Roxmore will need to retrieve historical pulps and re-analyze a substantial portion of the two-acid silver assays, as this digestion method is not considered sufficiently accurate or complete for resource estimation purposes.

SLR reviewed the silver database and noted many intervals report appreciable silver grades. However, historical operators used different analytical methods, with some samples analyzed by two-acid aqua-regia and others by four-acid digestion. Because aqua regia provides only a partial digestion, it is not recommended for estimating silver Mineral Resources, and the distribution of four-acid silver assays is insufficient to support a silver resource at this time.

25.0 Interpretation and Conclusions

The QPs conclude that the PEA demonstrates robustly positive economics at gold prices lower than current forecasts:

- after-tax NPV at a 5% discount rate of US$2,749 million
- after-tax IRR of 42.8%
- payback period of approximately 2.2 years from the start of production

The PEA is preliminary in nature, and is based, in part, on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Further conclusions by area are as follows:

25.1 Geology and Mineral Resources

- Effective March 31, 2026, within an optimized pit shell and above cut-off grades ranging from 0.18 g/t to 0.20 g/t Au, the Converse Project Mineral Resource is estimated to total:
    - Indicated: 103 Mt at an average grade of 0.65 g/t Au, totalling 2,162 koz; and
    - Inferred: 218 Mt at an average grade of 0.43 g/t Au, totalling 3,035 koz.
- The deposit remains open at depth and along key structural trends, indicating potential for additional discoveries and resource growth.
- The Converse Project is characterized as a reduced gold skarn deposit.
- The Project area exhibits a well-constrained geological framework, characterized by Havallah sequence sedimentary units that are intruded by a central porphyritic stock. Gold mineralization is primarily localized parallel to bedding planes and further focused by a series of southwest- and southeast-trending structural corridors that acted as conduits for hydrothermal fluid flow.
- Protocols for drilling, sample preparation and analysis, data verification, and security meet industry standard practices and are appropriate for use in a Mineral Resource estimate.
- Mineral Resource classification was based primarily on drill hole spacing, applied to designate contiguous zones of similar classification.

25.2 Mining Methods

- The mine plan demonstrates the technical viability of exploiting a 1.18 billion tonne deposit to deliver 299.8 Mt of mineralized material to the leach pad over a 14.5-year LOM.
- Strategic scheduling is designed to maintain a consistent steady-state production rate of 22.5 Mtpa from Year 2 through Year 14, supported by a peak annual mining capacity of 90 Mtpa.

- NPV is optimized through a phased mining approach that front loads higher grade material, maintaining gold grades above the 0.51 g/t LOM average through Year 8.
- Economic resilience is supported using a $3,000/oz gold price and the adoption of conservative reporting cut-off grades between 0.18 g/t and 0.20 g/t Au, providing a significant buffer above the marginal breakeven limit.
- The conversion of approximately 93% of the optimized pit shell resource into the final 299.8 Mt production schedule demonstrates high design efficiency while accounting for necessary operational constraints such as ramp geometries and geotechnical berms.
- The utilization of an owner-operator model with 240-tonne trucks and large-scale shovels provides the necessary economies of scale to manage the 2.93:1 LOM strip ratio with high operational efficiency.
- The increase in the LOM strip ratio to 2.93:1—representing a 12.5% variance from the theoretical optimization—ensures that the Project’s cost model and 14.5 year schedule are grounded in a realistic, mineable geometry. Crucially, following the pre-production phase, the active mining schedule achieves a more efficient production strip ratio of 2.71:1.
- Geotechnical and logistical stability is maintained through engineered designs featuring OSA of 38° in alluvium and 45° in bedrock, alongside optimized haulage gradients of 10% in-pit and 8% ex-pit.
- All labour requirements and operating costs are referenced against the SHERPA 2025 Dataset, ensuring that the Project's economic model reflects current industry-standard benchmarks.
- Clear paths for further project optimization and de-risking have been established for the pre-feasibility stage, specifically regarding expanded geotechnical characterization of the Redline porphyry and deep pit hydrogeology.

25.3 Mineral Processing

- The distribution of metallurgical samples collected to date adequately covers the oxide, transition, and sulphide domains. In addition, the test work includes a balanced representation of both high-grade and low-grade mineralized zones. The QP notes, however, that there is insufficient sampling at the lower portion of the deposit and additional sampling is required to ensure comprehensive characterization of the deposit.
- Material from the Converse deposits is amenable to cyanide leaching for recovery of gold and silver values, providing overall good recoveries with moderate reagent requirements at a HPGR crushed product size of 80% passing 6.3 mm. Based on the current minable resource, the overall average gold recovery is estimated at 70.9%.
- Mineralized material will be heap leached at a rate of 22.5 Mtpa on a multi-lift heap that will be developed in three phases. Crushed material will be drum-agglomerated with cement to provide stability and pH buffering, then stacked and leached with a low-concentration cyanide solution. Pregnant solution from the heap leach will be processed in an ADR recovery plant to produce doré bars.

25.4 Infrastructure

- Services are readily available in nearby towns including Lovelock, Winnemucca and Battle Mountain where skilled labour, suppliers, and experienced contractors can also be sourced.
- Grid electrical power is available on site via an existing transmission line that bisects the property. A substation drop will be placed at the north side of the mine and the existing powerline infrastructure will be rerouted around the mine perimeter.
- In 2019, CRL purchased 2,560 acre-ft of irrigation water rights from the New Nevada Lands, LLC, which was subsequently converted to mining and milling use. The acquired water rights will support the construction and operation of a future mine at the Property.

25.5 Environment

- The Converse Project is in the very early stages of environmental and social permitting. With the exception of water rights and exploration permits, no environmental approvals have been secured.
- Because much of the Project area is situated on BLM land, a PoO will be required. PFS level planning and at least one year of baseline studies will be required prior to submittal of a draft PoO to BLM.
- Due to the scope of the Project, a detailed EIS performed under the NEPA will likely be required for environmental project approval.
- Previous studies indicate a low potential for acid generating waste. Metallurgical work by KCA indicates relatively low sulphide content (0.5% to 5%).
- The BLM identifies mining as one of the primary threats contributing to sagebrush habitat loss for Greater Sage-Grouse in Nevada; however, the Project generally falls outside currently mapped Greater Sage-Grouse habitat, thus the need for habitat loss mitigation may be low or nonexistent.
- The existing water rights and aquifer yield may not be sufficient for operations and hydrological characterization work will form an important part of the next phase of work. As of the date of this writing, the water right is not fully vested under Nevada law because Proof of Completion and Proof of Beneficial Use filings have not been completed.

25.6 Capital and Operating Costs

- Mining capital and operating estimates are validated by the SHERPA 2025 Dataset and reconciled against regional operating benchmarks and recent PFS- or Feasibility Study (FS)-level studies to ensure cost competitiveness.
- Initial capital intensity is strategically managed through a 20% equipment down payment ($39 million) and a $116 million capitalized stripping program, providing the necessary pit access to reach the 22.5 Mtpa target by Year 2.
- Mining-specific infrastructure is budgeted at $38 million, covering truck shops, mine site haul roads, and fuel/explosive storage; this excludes general project infrastructure such as the main access road, site power substation, and administration buildings.

- The Project maintains a resilient $2.36/t LOM unit mining cost, which accounts for the transition from low cost alluvial stripping to the extended haulage cycles and increased fuel consumption associated with the 620 m deep bedrock pit.
- Long-term capital stability is supported by a 6 year lease amortization of the initial fleet ($193 million), followed by $177 million in staggered sustaining capital for replacements and expansions.
- Process plant capital costs are based on recent budgetary quotes from similar projects in KCA's files for all major and most minor equipment. Where recent quotes were unavailable, reasonable cost estimates or allowances were provided based on cost guide data. All capital cost estimates were based on the purchase of equipment quoted new from the manufacturer or to be fabricated new.
- Process mechanical equipment installation estimates were based on the equipment type and included all installation labour, tools, and equipment usage at an average hourly installation rate of $125.00, based on KCA's experience from recent projects.
- Process operating costs were developed by first principles and test work results based on KCA's experience in this jurisdiction.

25.7 Risks

- Metallurgical recoveries
  - Confirm metallurgical recoveries with more variability test work for all mining areas.
  - Confirm HPGR heap leach recoveries versus conventional crushing.
- Copper in pregnant leach solution may require removal with a SART plant to recover cyanide; conduct more testing to understand copper implications, if any.
- Water use and water quality protection will be key environmental issues under scrutiny by BLM and NDEP. It is currently unclear whether the existing water rights and effective aquifer yield will be sufficient for operational demand.
- Proof of Completion and Proof of Beneficial Use for the existing water rights may not have been completed, putting the rights at risk of being forfeit or cancelled.

25.8 Opportunities

- Geotechnical Slope Refinement: Utilize high-density data in future PFS/FS studies to safely steepen inter-ramp pit slopes, reducing the LOM strip ratio and lowering waste mining costs.
- Optimized Phase Scheduling: Refine the current 3-phase pit sequencing during subsequent PFS/FS stages to further accelerate high-grade material extraction, optimize waste deferral, and maximize project NPV ahead of reserve declaration.
- Evaluation of pulp agglomeration processing to explore the potential to improve metal recovery and overall project economics.
- Silver as a by-product:
  - The current economic model does not include silver, so it is not treated as contributing to project revenues.

    - While silver is less valuable than gold, recovering it as a by-product could improve the Project's economics, and its processing would likely not require additional infrastructure.

- Early Stakeholder Engagement: Begin engaging with stakeholders (NDEP, BMRR, BLM, Humboldt County officials, NGOs, and interested indigenous organizations). A Stakeholder Engagement Study may be a beneficial approach at this stage.

- Exploration Permitting: Notice level exploration permitting with BLM for areas of potential interest beyond the existing permitted exploration limits.

26.0 Recommendations

The Project should be advanced to the next stage of engineering study.

26.1 Geology and Mineral Resources

1 Continue exploration with emphasis on infill drilling inside the resource pit shell to support the conversion of Inferred to Indicated Mineral Resources.

2 Continue to refine the redox model by integrating relogged and newly collected geology data, cyanide solubility assays, and additional geochemical indicators.

3 Relogging efforts have further differentiated the prograde and retrograde skarn assemblages. Incorporate these updated classifications into a comprehensive skarn model to evaluate the spatial and temporal interactions between prograde and retrograde mineralizing processes and gold recovery within these domains.

4 Undertake careful logging of the alluvium with attention to the competency, hardness, and a generalized composition of the matrix and the cobbles, particularly if sulphides are present.

5 Continue collecting density measurements, with attention to the oxide zone and other areas of the deposit where sample coverage is limited. In intervals characterized by highly broken or porous rock, the use of a wax emulsion coating is recommended to ensure accurate volume determination.

6 Evaluate the database to identify samples lacking silver assays and those analyzed only by two-acid digestion. Using statistical and spatial criteria, select a representative subset for re-analysis using a four-acid digestion to generate complete silver values. Once a sufficiently robust silver dataset has been established, undertake a Mineral Resource estimate for silver.

26.2 Mining Methods

1 Execute an expanded geotechnical drilling program to characterize the Redline Porphyry and deep bedrock units at the 620 m ultimate pit floor. While the current pit design applies conservative engineering parameters to manage the limited five-hole dataset, this expanded program is required to optimize slope angles and reduce risk for a project of this billion-tonne scale.

2 Build a 3D structural domain model using oriented core data to determine if the dominant west-dipping structural fabric persists or changes orientation across the wider pit footprint.

3 Conduct specialized stability modelling and detailed mapping for the sharp "nose" geometry located between the North and South Redline deposits to mitigate identified risks of tensile failure.

4 Prioritize a comprehensive hydrogeological investigation, including the installation of piezometers, to establish the phreatic surface, quantify the impact of pore water pressure on slope stability at depth, inform the dewatering requirements, and inform the site water balance.

5 Implement real-time in situ mapping and additional geotechnical testing of weak clay units during the early phases of alluvium excavation to refine conceptual slope stability and trafficability assumptions.

6 Quantify mining dilution and mineralized material loss through a trade-off study evaluating the relationship between the 29.8 m^3 hydraulic shovel selectivity, bench geometries, and contact mineralization to move beyond the 0% PEA assumption.

7 Perform advanced scheduling and phasing trade-off studies to balance vertical advance rates and optimize the sequence of pushbacks, specifically addressing the Year 11 feed shortfall and narrowing pit floor constraints.

8 Execute dedicated blast fragmentation and metallurgical trade-off studies to optimize hole diameters, confirm drill rig configurations, and refine the efficiency of wall-control drilling for final highwalls.

9 Conduct additional bulk density testing across all alluvial and bedrock material types to validate the 25% swell factor and the spatial capacity of the 507 Mm^3 WRSF.

10 Refine the mining fleet requirements by transitioning from SHERPA benchmarks to vendor-specific performance data and site-specific haulage simulations tailored to the 10% in-pit and 8% ex-pit gradients.

26.3 Mineral Processing

1 Complete additional metallurgical testing in the deeper areas of the deposit to ensure its comprehensive characterization.

2 Complete additional metallurgical test work on HPGR and conventionally crushed material.

3 Conduct variability testing to better characterize redox units.

4 Compacted permeability testing and SART tests should be included in the test program.

5 Perform pulp agglomeration test work with high-grade and low-grade material to evaluate a potential hybrid flow sheet.

26.4 Infrastructure

1 Evaluate site-wide water balance and discharge requirements.

2 Confirm power requirements and supply availability.

3 Develop site-specific design criteria for the heap leach pad and perform field investigations and laboratory testing to support advancing the heap leach pad design.

4 Develop a PFS-level heap leach pad design that incorporates site specific data into the engineering calculations and analysis for such items as grading plan, slope stability analysis, water balance, settlement, and closure.

26.5 Environment

1 Commence baseline studies as soon as practical. Of urgency are hydrogeologic, surface water (if any), wildlife, and air quality studies, as these require at least one year of field data collection.

2 Conduct additional geochemical characterization using the updated mine plan and geologic block model.

3 Begin stakeholder engagement activities. Include federal, state and local regulatory agencies, tribal groups and NGOs. Consider commissioning a stakeholder engagement study.

4 Perform an updated cultural resource survey of the entire Project area of influence.

5 Identification of a significant wildlife or cultural resource that requires mitigation. Early commencement of baseline and other studies can reduce this risk.

6 Finalize Proof of Completion and Proof of Beneficial Use for existing water rights.

26.6 Capital and Operating Costs

1 Transition from benchmarked estimates to a formal RFP with OEMs to secure binding quotes and guaranteed maintenance terms for the primary 240 t haulage and shovel fleet.

2 Conduct a formal RFP for contractor mining services as an alternative to the owner mining model.

3 Conduct site-specific geotechnical testing on the alluvium to maximize "free-dig" zones, potentially reducing the initial drilling and blasting capital and operating requirements.

4 Perform localized studies to confirm that the assumed diesel ($0.90/L), power ($0.08/kWh), and hourly labour rates align with current regional market conditions to de-risk the LOM unit cost.

5 Perform a trade-off study on AHS to determine if the long-term operating cost savings justify the higher initial capital outlay, given the Project's scale and 14.5 year life.

6 Advancement to the next level of study should include the development of SRCE-level inputs and a site-specific closure plan to further refine this estimate and timeline, as part of permitting and financial assurance requirements in Nevada.

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28.0 Date and Signature Date

This report titled "NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA" with an effective date of May 2, 2026 was prepared and signed by the following authors:

	(Signed & Sealed) ***April Barrios***
Dated at Vancouver, BC, Canada May 29, 2026	April Barrios, P.Geo.
	(Signed & Sealed) ***Balaji Subrahmanyan***
Dated at Lakewood, CO, USA May 29, 2026	Balaji Subrahmanyan, SME(RM)
	(Signed & Sealed) ***Caleb Cook***
Dated at Reno, NV, USA May 29, 2026	Caleb Cook, P.E.
	(Signed & Sealed) ***Mark Trevor***
Dated at Folsom, California, USA May 29, 2026	Mark Trevor, PG, CPG, SME(RM)
	(Signed & Sealed) ***Matthew Behling***
Dated at Reno, NV, USA May 29, 2026	Matthew Behling, P.E.
	(Signed & Sealed) ***Jason J. Cox***
Dated at Toronto, ON, Canada May 29, 2026	Jason J. Cox, P.Eng

29.0 Certificate of Qualified Person

29.1 April Barrios

I, April Barrios, P.Geo., as an author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA" with an effective date of May 2, 2026 prepared for Roxmore Resources Inc., do hereby certify that:

1. I am Senior Resource Geologist with SLR Consulting (Canada) Ltd, of 887 Great Northern Way, Vancouver BC V5T4T5.
2. I am a graduate of the University of Victoria in 2004 with a Bachelor of Science Earth and Ocean Science.
3. I am registered as a Professional Geologist in the Province of British Columbia (Reg.# 35736). I have worked as a professional geologist for over 20 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Mineral Resource estimation and preparation of NI 43-101 Technical Report for the Black Pine gold deposit, Idaho.
    - Mineral Resource estimation and preparation of NI 43-101 Technical Report for the Goldstrike gold deposit, Utah.
    - More than 10 years of professional experience in the Great Basin USA (Nevada, Utah, Idaho), including geological mapping, 3D geological and resource modelling, RC and core logging, and development, implementation, and monitoring of QA/QC programs.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Converse Project on November 4, 2025.
6. I am responsible for Sections 1.1, 1.1.1.1, 1.1.2.1, 1.3.1 to 1.3.6, 2 to 12, 14, 23, 24, 25.1, 26.1, 30, and related disclosure in Section 27 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2026

(Signed & Sealed) ***April Barrios***

April Barrios, P.Geo.

29.2 Balaji Subrahmanyan

I Balaji Subrahmanyan, SME(RM), as an author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA" with an effective date of May 2, 2026 prepared for Roxmore Resources Inc., do hereby certify that:

1. I am Principal Mining Engineer with SLR International Corporation, of Suite 100, 1658 Cole Boulevard, Lakewood, CO, USA 80401.
2. I am a graduate of College of Engineering, Chennai, India in 1993 with a Bachelor of Science degree in Mining Engineering.
3. I am a Registered Member of the Society for Mining, Metallurgy & Exploration (SME RM# 04224038). I have worked as a mining engineer for a total of 32 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Open pit operational experience in Asia, Australia, Africa and abroad.
    - Review and report as a consultant on open pit mining projects and operations around the world for studies, audits, due diligence, and regulatory requirements.
    - Open pit mine planning and cost estimation.
    - Project cash flow modelling and economic analysis.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Converse Project on November 4, 2025.
6. I am responsible for Sections 1.1.1.2, 1.1.2.2, 1.3.7, 1.3.10, 15, 16, 19, 21 (except 21.1.2.2, 21.1.3.2, and 21.2.2), 25.2, 26.2, and related disclosure in Section 27 of the Technical Report.
7. I am independent of the Issuer, applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2026

(Signed & Sealed) ***Balaji Subrahmanyan***

Balaji Subrahmanyan, SME(RM)

29.3 Caleb Cook

I, Caleb Cook, P.E., as an author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA" with an effective date of May 2, 2026 prepared for Roxmore Resources Inc., do hereby certify that:

1. I am Project Engineer and Engineering Manager with Kappes, Cassiday & Associates located at 7950 Security Circle, Reno, Nevada, USA 89506.
2. I am a graduate of Project Engineer and Engineering Manager with Kappes, Cassiday & Associates located at 7950 Security Circle, Reno, Nevada, USA 89506.
3. I am registered as a Professional Engineer in the State of Nevada (No. 025803). I have worked as a process engineer and metallurgist for a total of 15 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Process design and development for heap leach projects in Nevada, Utah, Mexico, Argentina, Turkey and others.
    - Capital and operating cost development for process and infrastructure requirements for various mining properties.
    - Review and interpretation of metallurgical test work data as a consultant as part of engineering studies, audits and due diligence reviews.
    - Most of my professional experience has focused on the development of gold and silver heap leaching projects.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Converse Project site. In my opinion, a site visit was not required to support my contribution to this Technical Report, and its absence does not materially affect my conclusions.
6. I am responsible for Sections 1.1.1.3, 1.1.2.3, 1.3.8, 13, 17, 21.1.2.2, 21.1.3.2, 21.2.2, 25.3, 26.3, and related disclosure in Section 27 of the Technical Report.
7. I am independent of the Issuer and the Converse Project, applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2026

(Signed & Sealed) ***Caleb Cook***

Caleb D. Cook, P.E.

29.4 Mark Trevor

I, Mark Trevor, PG, CPG, SME(RM) as an author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA" with an effective date of May 2, 2026 prepared for Roxmore Resources Inc., do hereby certify that:

1. I am a Senior Principal Geologist with SLR International Corporation of 1024 Iron Point Rd. in Folsom, California.
2. I am a graduate of Humboldt State University in 1998 with a Bachelor of Science in Geology.
3. I am a Registered Member of the Society for Mining, Metallurgy & Exploration (SME RM# 4293241). I am registered as a Certified Professional Geologist (CPG) by the American Institute of Professional Geologist (12313) and am a registered Professional Geologist in Alaska, California, Oregon, and Arizona. I have worked as an environmental geologist in mining for a total of 20 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Mine environmental permitting and compliance experience in Nevada. I have supported environmental permitting and compliance at the Argenta Mine near Battle Mountain, the Comstock Mine near Virginia City, the Rhyolite Ridge Lithium-Boron Project in Esmeralda County and the Aurora Lithium-Uranium Project near McDermitt.
    - Mining environmental compliance experience in Nevada and various other US states including Alaska, California, Arizona and New York. Experience includes baseline studies, permitting, permitting roadmap development, mine waste management, closure, reclamation, remediation and site assessments, and post-closure monitoring and reporting.
    - Experience as a Qualified Person and peer reviewer consulting on environmental and social due diligence for mining projects in North America, including NI 43-101 Technical Reports and S-K 1300 Initial Assessments in British Columbia, Newfoundland, Nevada, Colorado, California, Virginia, Tennessee and New York.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Converse Project site. In my opinion, a site visit was not required to support my contribution to this Technical Report, and its absence does not materially affect my conclusions.
6. I am responsible for Sections 1.1.1.5, 1.1.2.5, 1.3.11, 20, 25.5, 26.5, and related disclosure in Section 27 of the Technical Report.
7. I am independent of the Issuer and the Converse Project, applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific

and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2026

(Signed & Sealed) *Mark Trevor*

Mark S. Trevor, PG, CPG, SME(RM)

29.5 Matthew Behling

I, Matthew A. Behling, P.E., as an author of this report entitled "NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA" with an effective date of May 2, 2026 prepared for Roxmore Resources Inc., do hereby certify that:

1. I am a Principal Engineer with SLR International Corporation, 6100 Plumas Street, Suite 203, Reno, NV 89519.
2. I am a graduate of the University of Nevada, Reno in 2005 with a Bachelor of Science in Civil Engineering
3. I am registered as a Professional Engineer in the State of Nevada (Reg.#19867). I have worked as an engineer for a total of 21 years since my graduation, predominantly supporting mine infrastructure development in the Western United States. My relevant experience for the purpose of the Technical Report is:
    - Design of Heap Leach Facilities
    - Design of Civil Infrastructure
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Converse Project site. In my opinion, a site visit was not required to support my contribution to this Technical Report, and its absence does not materially affect my conclusions.
6. I am responsible for Sections 1.1.1.4, 1.1.2.4, 1.3.9, 18, 25.4, 26.4, and related disclosure in Section 27 of the Technical Report.
7. I am independent of the Issuer and the Converse Project, applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2026

(Signed & Sealed) *Matthew Behling*

Matthew A. Behling, PE

29.6 Jason J. Cox

I, Jason J. Cox, P.Eng., as an author of this report entitled “NI 43-101 Technical Report on the Preliminary Economic Assessment for the Converse Project, Nevada, USA” with an effective date of May 2, 2026 prepared for Roxmore Resources Inc., do hereby certify that:

1. I am Global Technical Director – Canada Mining Advisory with SLR Consulting (Canada) Ltd, of Suite 501, 55 University Ave., Toronto, ON M5J 2H7.
2. I am a graduate of the Queen’s University, Kingston, Ontario, Canada, in 1996 with a Bachelor of Science degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 90487158). I have worked as a mining engineer for a total of 25 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and reporting as a consultant on many mining operations and projects around the world for due diligence and regulatory requirements.
   - Engineering study work (PEA, PFS, and FS) on many mining projects around the world, including commodities such as precious metals, base metals, bulk commodities, industrial minerals, and rare earths.
   - Operational experience as Planning Engineer and Senior Mine Engineer at three North American mines.
   - Contract Coordinator for underground construction at an American mine.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Converse Project site. In my opinion, a site visit was not required to support my contribution to this Technical Report, and its absence does not materially affect my conclusions.
6. I am responsible for Sections 1.1.1, 1.1.1.6, 1.1.2.6, 1.1.3, 1.1.4, 1.2, 1.3.12, 22, 25.0, 25.6, 25.7, 25.8, 26.6, and related disclosure in Section 27 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Section Nos. in the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2026

(Signed & Sealed) ***Jason J. Cox***

Jason J. Cox, P.Eng.

30.0 Appendix 1

30.1 Unpatented Lode and Placer Mining Claims

Table 30-1: Converse Project - Unpatented Placer Mining Claims

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV101354428	BV 1	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354429	BV 2	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354430	BV 3	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354431	BV 4	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354432	BV 5	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354433	BV 6	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354434	BV 7	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354435	BV 8	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354436	BV 9	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354437	BV 10	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354438	BV 11	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354439	BV 12	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354440	BV 13	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101354441	BV 14	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355422	BV 15	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355423	BV 16	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355424	BV 17	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355425	BV 18	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355426	BV 19	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355427	BV 20	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355428	BV 21	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355429	BV 22	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355430	BV 23	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355431	BV 24	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355432	BV 25	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18
NV101355433	BV 26	13/10/2006	Active	NNR	Humboldt	Placer	20.22	8.18

Source: Roxmore 2026.

Note:

- NNR = Nevada North Resources (USA) Inc.

Table 30-2: Converse Project - Unpatented Lode Mining Claims

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779739	BV 001	22/11/2025	Filed	CRL	Humboldt	Lode	14.47	5.86
NV106779740	BV 002	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779741	BV 003	22/11/2025	Filed	CRL	Humboldt	Lode	14.35	5.81
NV106779742	BV 004	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779743	BV 005	22/11/2025	Filed	CRL	Humboldt	Lode	14.28	5.78
NV106779744	BV 006	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779745	BV 007	22/11/2025	Filed	CRL	Humboldt	Lode	14.22	5.75
NV106779746	BV 008	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779747	BV 009	22/11/2025	Filed	CRL	Humboldt	Lode	14.16	5.73
NV106779748	BV 010	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779749	BV 011	22/11/2025	Filed	CRL	Humboldt	Lode	14.10	5.70
NV106779750	BV 012	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779751	BV 013	22/11/2025	Filed	CRL	Humboldt	Lode	13.34	5.40
NV106779752	BV 014	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779753	BV 015	22/11/2025	Filed	CRL	Humboldt	Lode	8.25	3.34
NV106779754	BV 016	22/11/2025	Filed	CRL	Humboldt	Lode	17.29	7.00
NV106779755	BV 017	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779756	BV 018	22/11/2025	Filed	CRL	Humboldt	Lode	18.34	7.42
NV106779757	BV 019	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779758	BV 020	22/11/2025	Filed	CRL	Humboldt	Lode	18.20	7.37
NV106779759	BV 021	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779760	BV 022	22/11/2025	Filed	CRL	Humboldt	Lode	18.21	7.37
NV106779761	BV 023	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779762	BV 024	22/11/2025	Filed	CRL	Humboldt	Lode	18.23	7.38
NV106779763	BV 025	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779764	BV 026	22/11/2025	Filed	CRL	Humboldt	Lode	18.24	7.38
NV106779765	BV 027	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779766	BV 028	22/11/2025	Filed	CRL	Humboldt	Lode	18.25	7.39
NV106779767	BV 029	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779768	BV 030	22/11/2025	Filed	CRL	Humboldt	Lode	17.82	7.21
NV106779769	BV 031	22/11/2025	Filed	CRL	Humboldt	Lode	17.16	6.94
NV106779770	BV 032	22/11/2025	Filed	CRL	Humboldt	Lode	14.30	5.79

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779771	BV 033	21/11/2025	Filed	CRL	Humboldt	Lode	12.61	5.10
NV106779772	BV 034	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779773	BV 035	21/11/2025	Filed	CRL	Humboldt	Lode	11.91	4.82
NV106779774	BV 036	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779775	BV 037	21/11/2025	Filed	CRL	Humboldt	Lode	11.95	4.84
NV106779776	BV 038	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779777	BV 039	21/11/2025	Filed	CRL	Humboldt	Lode	11.99	4.85
NV106779778	BV 040	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779779	BV 041	21/11/2025	Filed	CRL	Humboldt	Lode	12.03	4.87
NV106779780	BV 042	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779781	BV 043	21/11/2025	Filed	CRL	Humboldt	Lode	12.07	4.89
NV106779782	BV 044	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779783	BV 045	21/11/2025	Filed	CRL	Humboldt	Lode	12.04	4.87
NV106779784	BV 046	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779785	BV 047	21/11/2025	Filed	CRL	Humboldt	Lode	11.01	4.45
NV106779786	BV 048	21/11/2025	Filed	CRL	Humboldt	Lode	18.61	7.53
NV106779787	BV 049	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779788	BV 050	21/11/2025	Filed	CRL	Humboldt	Lode	20.57	8.33
NV106779789	BV 051	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779790	BV 052	21/11/2025	Filed	CRL	Humboldt	Lode	20.55	8.32
NV106779791	BV 053	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779792	BV 054	21/11/2025	Filed	CRL	Humboldt	Lode	20.55	8.32
NV106779793	BV 055	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779794	BV 056	21/11/2025	Filed	CRL	Humboldt	Lode	20.54	8.31
NV106779795	BV 057	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779796	BV 058	21/11/2025	Filed	CRL	Humboldt	Lode	20.53	8.31
NV106779797	BV 059	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779798	BV 060	21/11/2025	Filed	CRL	Humboldt	Lode	20.52	8.31
NV106779799	BV 061	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779800	BV 062	21/11/2025	Filed	CRL	Humboldt	Lode	20.52	8.30
NV106779801	BV 063	21/11/2025	Filed	CRL	Humboldt	Lode	17.95	7.26
NV106779802	BV 064	21/11/2025	Filed	CRL	Humboldt	Lode	16.92	6.85
NV106779803	BV 065	20/11/2025	Filed	CRL	Humboldt	Lode	20.07	8.12

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779804	BV 066	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779805	BV 067	20/11/2025	Filed	CRL	Humboldt	Lode	20.27	8.20
NV106779806	BV 068	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779807	BV 069	20/11/2025	Filed	CRL	Humboldt	Lode	20.23	8.19
NV106779808	BV 070	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779809	BV 071	20/11/2025	Filed	CRL	Humboldt	Lode	20.20	8.17
NV106779810	BV 072	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779811	BV 073	20/11/2025	Filed	CRL	Humboldt	Lode	20.16	8.16
NV106779812	BV 074	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779813	BV 075	20/11/2025	Filed	CRL	Humboldt	Lode	20.12	8.14
NV106779814	BV 076	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779815	BV 077	20/11/2025	Filed	CRL	Humboldt	Lode	20.09	8.13
NV106779816	BV 078	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779817	BV 079	20/11/2025	Filed	CRL	Humboldt	Lode	20.05	8.11
NV106779818	BV 080	20/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779819	BV 081	20/11/2025	Filed	CRL	Humboldt	Lode	18.06	7.31
NV106779820	BV 082	20/11/2025	Filed	CRL	Humboldt	Lode	18.46	7.47
NV106779821	BV 083	20/11/2025	Filed	CRL	Humboldt	Lode	19.97	8.08
NV106779822	BV 084	21/11/2025	Filed	CRL	Humboldt	Lode	10.00	4.05
NV106779839	BV 101	21/11/2025	Filed	CRL	Humboldt	Lode	10.65	4.31
NV106779840	BV 102	21/11/2025	Filed	CRL	Humboldt	Lode	18.56	7.51
NV106779841	BV 103	21/11/2025	Filed	CRL	Humboldt	Lode	11.95	4.83
NV106779842	BV 104	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779843	BV 105	21/11/2025	Filed	CRL	Humboldt	Lode	11.94	4.83
NV106779844	BV 106	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779845	BV 107	21/11/2025	Filed	CRL	Humboldt	Lode	11.93	4.83
NV106779846	BV 108	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779847	BV 109	21/11/2025	Filed	CRL	Humboldt	Lode	11.92	4.82
NV106779848	BV 110	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779849	BV 111	21/11/2025	Filed	CRL	Humboldt	Lode	11.91	4.82
NV106779850	BV 112	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779851	BV 113	21/11/2025	Filed	CRL	Humboldt	Lode	11.90	4.82
NV106779852	BV 114	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779853	BV 115	21/11/2025	Filed	CRL	Humboldt	Lode	11.89	4.81
NV106779854	BV 116	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779855	BV 117	21/11/2025	Filed	CRL	Humboldt	Lode	11.73	4.75
NV106779856	BV 118	21/11/2025	Filed	CRL	Humboldt	Lode	20.34	8.23
NV106779857	BV 119	21/11/2025	Filed	CRL	Humboldt	Lode	18.78	7.60
NV106779858	BV 120	21/11/2025	Filed	CRL	Humboldt	Lode	11.33	4.58
NV106779859	BV 121	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779860	BV 122	21/11/2025	Filed	CRL	Humboldt	Lode	12.35	5.00
NV106779861	BV 123	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779862	BV 124	21/11/2025	Filed	CRL	Humboldt	Lode	12.35	5.00
NV106779863	BV 125	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779864	BV 126	21/11/2025	Filed	CRL	Humboldt	Lode	12.35	5.00
NV106779865	BV 127	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779866	BV 128	21/11/2025	Filed	CRL	Humboldt	Lode	13.58	5.50
NV106779867	BV 129	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779868	BV 130	21/11/2025	Filed	CRL	Humboldt	Lode	20.03	8.11
NV106779869	BV 131	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779870	BV 132	21/11/2025	Filed	CRL	Humboldt	Lode	20.01	8.10
NV106779871	BV 133	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779872	BV 134	21/11/2025	Filed	CRL	Humboldt	Lode	20.00	8.09
NV106779873	BV 135	21/11/2025	Filed	CRL	Humboldt	Lode	20.27	8.20
NV106779874	BV 136	21/11/2025	Filed	CRL	Humboldt	Lode	19.20	7.77
NV106779875	BV 137	22/11/2025	Filed	CRL	Humboldt	Lode	18.72	7.58
NV106779876	BV 138	22/11/2025	Filed	CRL	Humboldt	Lode	20.44	8.27
NV106779877	BV 139	22/11/2025	Filed	CRL	Humboldt	Lode	19.14	7.75
NV106779878	BV 140	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779879	BV 141	21/11/2025	Filed	CRL	Humboldt	Lode	19.19	7.77
NV106779880	BV 142	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779881	BV 143	21/11/2025	Filed	CRL	Humboldt	Lode	19.24	7.79
NV106779882	BV 144	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779883	BV 145	21/11/2025	Filed	CRL	Humboldt	Lode	19.29	7.81
NV106779884	BV 146	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779885	BV 147	21/11/2025	Filed	CRL	Humboldt	Lode	19.34	7.83

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779886	BV 148	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779887	BV 149	21/11/2025	Filed	CRL	Humboldt	Lode	19.39	7.84
NV106779888	BV 150	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779889	BV 151	22/11/2025	Filed	CRL	Humboldt	Lode	19.43	7.86
NV106779890	BV 152	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779891	BV 153	22/11/2025	Filed	CRL	Humboldt	Lode	18.48	7.48
NV106779892	BV 154	22/11/2025	Filed	CRL	Humboldt	Lode	19.62	7.94
NV106779893	BV 155	21/11/2025	Filed	CRL	Humboldt	Lode	20.63	8.35
NV106779894	BV 156	21/11/2025	Filed	CRL	Humboldt	Lode	13.56	5.49
NV106779895	BV 157	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779896	BV 158	21/11/2025	Filed	CRL	Humboldt	Lode	13.48	5.46
NV106779897	BV 159	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779898	BV 160	21/11/2025	Filed	CRL	Humboldt	Lode	13.49	5.46
NV106779899	BV 161	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779900	BV 162	21/11/2025	Filed	CRL	Humboldt	Lode	13.50	5.46
NV106779901	BV 163	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779902	BV 164	21/11/2025	Filed	CRL	Humboldt	Lode	13.50	5.46
NV106779903	BV 165	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779904	BV 166	21/11/2025	Filed	CRL	Humboldt	Lode	13.51	5.47
NV106779905	BV 167	21/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779906	BV 168	21/11/2025	Filed	CRL	Humboldt	Lode	13.51	5.47
NV106779907	BV 169	22/11/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779908	BV 170	22/11/2025	Filed	CRL	Humboldt	Lode	13.52	5.47
NV106779909	BV 171	22/11/2025	Filed	CRL	Humboldt	Lode	19.47	7.88
NV106779910	BV 172	22/11/2025	Filed	CRL	Humboldt	Lode	12.66	5.13
BLM Serial Numbers Pending	BVR 085	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00
BLM Serial Numbers Pending	BVR 087	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00
BLM Serial Numbers Pending	BVR 089	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
BLM Serial Numbers Pending	BVR 091	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00
BLM Serial Numbers Pending	BVR 093	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00
BLM Serial Numbers Pending	BVR 095	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00
BLM Serial Numbers Pending	BVR 097	28/04/2026	Filed	CRL	Humboldt	Lode	14.82	6.00
BLM Serial Numbers Pending	BVR 099	28/04/2026	Filed	CRL	Humboldt	Lode	13.13	5.31
NV106779251	CV 005	23/10/2025	Filed	CRL	Humboldt	Lode	19.36	7.84
NV106779252	CV 006	23/10/2025	Filed	CRL	Humboldt	Lode	19.29	7.81
NV106779253	CV 007	23/10/2025	Filed	CRL	Humboldt	Lode	20.64	8.35
NV106779254	CV 008	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779255	CV 009	23/10/2025	Filed	CRL	Humboldt	Lode	20.59	8.33
NV106779256	CV 010	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779257	CV 011	23/10/2025	Filed	CRL	Humboldt	Lode	20.54	8.31
NV106779258	CV 012	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779259	CV 013	23/10/2025	Filed	CRL	Humboldt	Lode	20.49	8.29
NV106779260	CV 014	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779261	CV 015	23/10/2025	Filed	CRL	Humboldt	Lode	20.44	8.27
NV106779262	CV 016	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779263	CV 017	23/10/2025	Filed	CRL	Humboldt	Lode	20.39	8.25
NV106779264	CV 018	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779265	CV 019	23/10/2025	Filed	CRL	Humboldt	Lode	20.22	8.18
NV106779266	CV 020	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779267	CV 021	23/10/2025	Filed	CRL	Humboldt	Lode	17.61	7.13
NV106779268	CV 022	23/10/2025	Filed	CRL	Humboldt	Lode	18.27	7.39
NV106779269	CV 023	23/10/2025	Filed	CRL	Humboldt	Lode	19.27	7.80
NV106779270	CV 024	23/10/2025	Filed	CRL	Humboldt	Lode	9.50	3.84
NV106779271	CV 025	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779272	CV 026	23/10/2025	Filed	CRL	Humboldt	Lode	10.21	4.13

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779273	CV 027	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779274	CV 028	23/10/2025	Filed	CRL	Humboldt	Lode	10.23	4.14
NV106779275	CV 029	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779276	CV 030	23/10/2025	Filed	CRL	Humboldt	Lode	10.26	4.15
NV106779277	CV 031	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779278	CV 032	23/10/2025	Filed	CRL	Humboldt	Lode	10.28	4.16
NV106779279	CV 033	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779280	CV 034	23/10/2025	Filed	CRL	Humboldt	Lode	10.30	4.17
NV106779281	CV 035	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779282	CV 036	23/10/2025	Filed	CRL	Humboldt	Lode	10.33	4.18
NV106779283	CV 037	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779284	CV 038	23/10/2025	Filed	CRL	Humboldt	Lode	10.35	4.19
NV106779285	CV 039	23/10/2025	Filed	CRL	Humboldt	Lode	18.26	7.39
NV106779286	CV 040	23/10/2025	Filed	CRL	Humboldt	Lode	9.16	3.71
NV106779287	CV 041	23/10/2025	Filed	CRL	Humboldt	Lode	15.34	6.21
NV106779288	CV 042	23/10/2025	Filed	CRL	Humboldt	Lode	18.46	7.47
NV106779289	CV 043	23/10/2025	Filed	CRL	Humboldt	Lode	17.19	6.96
NV106779290	CV 044	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779291	CV 045	23/10/2025	Filed	CRL	Humboldt	Lode	17.17	6.95
NV106779292	CV 046	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779293	CV 047	23/10/2025	Filed	CRL	Humboldt	Lode	17.15	6.94
NV106779294	CV 048	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779295	CV 049	23/10/2025	Filed	CRL	Humboldt	Lode	17.14	6.94
NV106779296	CV 050	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779297	CV 051	23/10/2025	Filed	CRL	Humboldt	Lode	17.12	6.93
NV106779298	CV 052	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779299	CV 053	23/10/2025	Filed	CRL	Humboldt	Lode	17.11	6.92
NV106779300	CV 054	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779301	CV 055	23/10/2025	Filed	CRL	Humboldt	Lode	17.09	6.92
NV106779302	CV 056	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779303	CV 057	23/10/2025	Filed	CRL	Humboldt	Lode	15.29	6.19
NV106779304	CV 058	23/10/2025	Filed	CRL	Humboldt	Lode	18.54	7.50
NV106779305	CV 059	23/10/2025	Filed	CRL	Humboldt	Lode	18.50	7.49

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV106779306	CV 060	23/10/2025	Filed	CRL	Humboldt	Lode	13.40	5.42
NV106779307	CV 061	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779308	CV 062	23/10/2025	Filed	CRL	Humboldt	Lode	14.21	5.75
NV106779309	CV 063	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779310	CV 064	23/10/2025	Filed	CRL	Humboldt	Lode	14.23	5.76
NV106779311	CV 065	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779312	CV 066	23/10/2025	Filed	CRL	Humboldt	Lode	14.24	5.76
NV106779313	CV 067	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779314	CV 068	23/10/2025	Filed	CRL	Humboldt	Lode	14.26	5.77
NV106779315	CV 069	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779316	CV 070	23/10/2025	Filed	CRL	Humboldt	Lode	14.27	5.78
NV106779317	CV 071	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779318	CV 072	23/10/2025	Filed	CRL	Humboldt	Lode	14.29	5.78
NV106779319	CV 073	23/10/2025	Filed	CRL	Humboldt	Lode	20.66	8.36
NV106779320	CV 074	23/10/2025	Filed	CRL	Humboldt	Lode	14.30	5.79
NV106779321	CV 075	23/10/2025	Filed	CRL	Humboldt	Lode	18.58	7.52
NV106779322	CV 076	23/10/2025	Filed	CRL	Humboldt	Lode	13.52	5.47
NV106735092	CV001	14/04/2025	Filed	CRL	Humboldt	Lode	10.27	4.16
NV106735093	CV002	14/04/2025	Filed	CRL	Humboldt	Lode	18.11	7.33
NV106735094	CV003	14/04/2025	Filed	CRL	Humboldt	Lode	18.27	7.40
NV106735095	CV004	14/04/2025	Filed	CRL	Humboldt	Lode	18.16	7.35
NV101526474	NIKE # 1	10/02/1988	Active	NNR	Humboldt	Lode	20.00	8.09
NV101602386	NIKE # 2	10/02/1988	Active	NNR	Humboldt	Lode	20.39	8.25
NV101478997	NIKE # 3	10/02/1988	Active	NNR	Humboldt	Lode	20.51	8.30
NV101609637	NIKE # 4	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101731116	NIKE # 5	10/02/1988	Active	NNR	Humboldt	Lode	20.47	8.28
NV101459253	NIKE # 6	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101610145	NIKE # 7	10/02/1988	Active	NNR	Humboldt	Lode	20.43	8.27
NV101491454	NIKE # 8	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101478511	NIKE # 9	10/02/1988	Active	NNR	Humboldt	Lode	20.39	8.25
NV101526846	NIKE # 10	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101759264	NIKE # 11	10/02/1988	Active	NNR	Humboldt	Lode	20.36	8.24
NV101609615	NIKE # 12	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36

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NV101527149	NIKE # 13	10/02/1988	Active	NNR	Humboldt	Lode	20.33	8.23
NV101344422	NIKE # 14	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101301738	NIKE # 15	10/02/1988	Active	NNR	Humboldt	Lode	20.30	8.22
NV101460084	NIKE # 16	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101348060	NIKE # 17	10/02/1988	Active	NNR	Humboldt	Lode	20.28	8.21
NV101601049	NIKE # 18	10/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101304261	NIKE # 19	11/02/1988	Active	NNR	Humboldt	Lode	20.54	8.31
NV101602076	NIKE # 20	11/02/1988	Active	NNR	Humboldt	Lode	10.61	4.29
NV101303023	NIKE # 21	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101550243	NIKE # 22	11/02/1988	Active	NNR	Humboldt	Lode	10.77	4.36
NV101348087	NIKE # 23	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101752983	NIKE # 24	11/02/1988	Active	NNR	Humboldt	Lode	10.90	4.41
NV101349979	NIKE # 25	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101453959	NIKE # 26	11/02/1988	Active	NNR	Humboldt	Lode	11.03	4.47
NV101406006	NIKE # 27	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101752806	NIKE # 28	11/02/1988	Active	NNR	Humboldt	Lode	11.17	4.52
NV101494791	NIKE # 29	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101544932	NIKE # 30	11/02/1988	Active	NNR	Humboldt	Lode	11.28	4.56
NV101494578	NIKE # 31	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101477819	NIKE # 32	11/02/1988	Active	NNR	Humboldt	Lode	11.38	4.60
NV101408471	NIKE # 33	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101543233	NIKE # 34	11/02/1988	Active	NNR	Humboldt	Lode	11.48	4.65
NV101523416	NIKE # 35	11/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101607848	NIKE # 36	11/02/1988	Active	NNR	Humboldt	Lode	11.58	4.69
NV101523434	NIKE # 37	12/02/1988	Active	NNR	Humboldt	Lode	18.39	7.44
NV101780970	NIKE # 38	12/02/1988	Active	NNR	Humboldt	Lode	10.42	4.22
NV101497499	NIKE # 39	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101479650	NIKE # 40	12/02/1988	Active	NNR	Humboldt	Lode	11.64	4.71
NV101459956	NIKE # 41	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101455573	NIKE # 42	12/02/1988	Active	NNR	Humboldt	Lode	11.69	4.73
NV101499864	NIKE # 43	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101495699	NIKE # 44	12/02/1988	Active	NNR	Humboldt	Lode	11.74	4.75
NV101303470	NIKE # 45	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36

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NV101456241	NIKE # 46	12/02/1988	Active	NNR	Humboldt	Lode	11.79	4.77
NV101302318	NIKE # 47	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101457922	NIKE # 48	12/02/1988	Active	NNR	Humboldt	Lode	11.84	4.79
NV101300430	NIKE # 49	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101601062	NIKE # 50	12/02/1988	Active	NNR	Humboldt	Lode	11.89	4.81
NV101347403	NIKE # 51	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101751483	NIKE # 52	12/02/1988	Active	NNR	Humboldt	Lode	11.94	4.83
NV101347443	NIKE # 53	12/02/1988	Active	NNR	Humboldt	Lode	20.00	8.09
NV101751575	NIKE # 54	12/02/1988	Active	NNR	Humboldt	Lode	11.92	4.82
NV101300788	NIKE # 55	12/02/1988	Active	NNR	Humboldt	Lode	17.91	7.25
NV101542311	NIKE # 56	12/02/1988	Active	NNR	Humboldt	Lode	18.16	7.35
NV102520502	NIKE # 57	12/02/1988	Active	NNR	Humboldt	Lode	20.59	8.33
NV101754030	NIKE # 58	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101492369	NIKE # 59	12/02/1988	Active	NNR	Humboldt	Lode	20.54	8.31
NV101754142	NIKE # 60	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101494798	NIKE # 61	12/02/1988	Active	NNR	Humboldt	Lode	20.49	8.29
NV101479477	NIKE # 62	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101406434	NIKE # 63	12/02/1988	Active	NNR	Humboldt	Lode	20.44	8.27
NV101477827	NIKE # 64	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101498867	NIKE # 65	12/02/1988	Active	NNR	Humboldt	Lode	20.39	8.25
NV101731264	NIKE # 66	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101523424	NIKE # 67	12/02/1988	Active	NNR	Humboldt	Lode	20.34	8.23
NV101540782	NIKE # 68	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101496855	NIKE # 69	12/02/1988	Active	NNR	Humboldt	Lode	20.29	8.21
NV101780980	NIKE # 70	12/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101497507	NIKE # 71	12/02/1988	Active	NNR	Humboldt	Lode	20.11	8.14
NV101456277	NIKE # 72	12/02/1988	Active	NNR	Humboldt	Lode	20.26	8.20
NV101348346	NIKE #127	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101453636	NIKE #128	15/02/1988	Active	NNR	Humboldt	Lode	10.30	4.17
NV101730784	NIKE #129	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101451944	NIKE #130	15/02/1988	Active	NNR	Humboldt	Lode	12.04	4.87
NV101603139	NIKE #131	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101752875	NIKE #132	15/02/1988	Active	NNR	Humboldt	Lode	12.01	4.86

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NV101496696	NIKE #133	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101601152	NIKE #134	15/02/1988	Active	NNR	Humboldt	Lode	11.97	4.85
NV101496283	NIKE #135	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101478854	NIKE #136	15/02/1988	Active	NNR	Humboldt	Lode	11.94	4.83
NV101490740	NIKE #137	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101460291	NIKE #138	15/02/1988	Active	NNR	Humboldt	Lode	11.91	4.82
NV101603424	NIKE #139	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101459355	NIKE #140	15/02/1988	Active	NNR	Humboldt	Lode	11.87	4.81
NV101607723	NIKE #141	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101479282	NIKE #142	15/02/1988	Active	NNR	Humboldt	Lode	11.84	4.79
NV101300235	NIKE #145	15/02/1988	Active	NNR	Humboldt	Lode	20.02	8.10
NV101479203	NIKE #146	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101304466	NIKE #147	15/02/1988	Active	NNR	Humboldt	Lode	20.48	8.29
NV101345669	NIKE #148	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101347042	NIKE #149	15/02/1988	Active	NNR	Humboldt	Lode	20.45	8.28
NV101529552	NIKE #150	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101300851	NIKE #151	15/02/1988	Active	NNR	Humboldt	Lode	20.43	8.27
NV101451725	NIKE #152	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101301491	NIKE #153	15/02/1988	Active	NNR	Humboldt	Lode	20.41	8.26
NV101452915	NIKE #154	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101348070	NIKE #155	15/02/1988	Active	NNR	Humboldt	Lode	20.39	8.25
NV101453466	NIKE #156	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101348695	NIKE #157	15/02/1988	Active	NNR	Humboldt	Lode	20.36	8.24
NV101491626	NIKE #158	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101604994	NIKE #159	15/02/1988	Active	NNR	Humboldt	Lode	20.34	8.23
NV101454006	NIKE #160	15/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101494461	NIKE #163	03/05/1988	Active	NNR	Humboldt	Lode	18.15	7.34
NV101752884	NIKE #164	03/05/1988	Active	NNR	Humboldt	Lode	16.67	6.75
NV101492873	NIKE #165	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101780946	NIKE #166	03/05/1988	Active	NNR	Humboldt	Lode	19.32	7.82
NV101496290	NIKE #167	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101490866	NIKE #168	03/05/1988	Active	NNR	Humboldt	Lode	19.49	7.89
NV101731620	NIKE #169	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36

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NV101490412	NIKE #170	03/05/1988	Active	NNR	Humboldt	Lode	19.66	7.96
NV101730656	NIKE #171	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101480026	NIKE #172	03/05/1988	Active	NNR	Humboldt	Lode	19.83	8.03
NV101607329	NIKE #173	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101479289	NIKE #174	03/05/1988	Active	NNR	Humboldt	Lode	20.00	8.10
NV101345722	NIKE #175	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101480328	NIKE #176	03/05/1988	Active	NNR	Humboldt	Lode	20.17	8.16
NV101529663	NIKE #177	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101508221	NIKE #178	03/05/1988	Active	NNR	Humboldt	Lode	20.34	8.23
NV101405534	NIKE #179	03/05/1988	Active	NNR	Humboldt	Lode	20.28	8.21
NV101528428	NIKE #180	03/05/1988	Active	NNR	Humboldt	Lode	20.11	8.14
NV101408558	NIKE #181	03/05/1988	Active	NNR	Humboldt	Lode	11.47	4.64
NV101453541	NIKE #182	03/05/1988	Active	NNR	Humboldt	Lode	18.61	7.53
NV101601312	NIKE #183	03/05/1988	Active	NNR	Humboldt	Lode	11.70	4.73
NV101455869	NIKE #184	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101304821	NIKE #185	03/05/1988	Active	NNR	Humboldt	Lode	11.66	4.72
NV101454860	NIKE #186	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101350217	NIKE #187	03/05/1988	Active	NNR	Humboldt	Lode	11.62	4.70
NV101606131	NIKE #188	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101347294	NIKE #189	03/05/1988	Active	NNR	Humboldt	Lode	11.58	4.69
NV101452559	NIKE #190	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101350278	NIKE #191	03/05/1988	Active	NNR	Humboldt	Lode	11.54	4.67
NV101494306	NIKE #192	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV102520620	NIKE #193	03/05/1988	Active	NNR	Humboldt	Lode	11.50	4.66
NV101494363	NIKE #194	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101496736	NIKE #195	03/05/1988	Active	NNR	Humboldt	Lode	11.47	4.64
NV101477106	NIKE #196	03/05/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101459740	NIKE #197	03/05/1988	Active	NNR	Humboldt	Lode	11.41	4.62
NV101496448	NIKE #198	03/05/1988	Active	NNR	Humboldt	Lode	20.48	8.29
NV101606360	NIKE #235	17/02/1988	Active	NNR	Humboldt	Lode	18.90	7.65
NV101495931	NIKE #236	17/02/1988	Active	NNR	Humboldt	Lode	9.88	4.00
NV101603551	NIKE #237	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101608819	NIKE #238	17/02/1988	Active	NNR	Humboldt	Lode	9.78	3.96

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NV101348332	NIKE #239	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101607668	NIKE #240	17/02/1988	Active	NNR	Humboldt	Lode	9.86	3.99
NV101301353	NIKE #241	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101529622	NIKE #242	17/02/1988	Active	NNR	Humboldt	Lode	9.93	4.02
NV101457577	NIKE #243	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101502133	NIKE #244	17/02/1988	Active	NNR	Humboldt	Lode	10.01	4.05
NV101408746	NIKE #245	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101341914	NIKE #246	17/02/1988	Active	NNR	Humboldt	Lode	10.11	4.09
NV101402125	NIKE #247	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101451283	NIKE #248	17/02/1988	Active	NNR	Humboldt	Lode	10.21	4.13
NV101405357	NIKE #249	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101458746	NIKE #250	17/02/1988	Active	NNR	Humboldt	Lode	10.32	4.18
NV101406765	NIKE #251	17/02/1988	Active	NNR	Humboldt	Lode	19.53	7.90
NV101602388	NIKE #252	17/04/1988	Active	NNR	Humboldt	Lode	10.33	4.18
NV101401777	NIKE #253	17/02/1988	Active	NNR	Humboldt	Lode	19.58	7.92
NV101730443	NIKE #254	17/02/1988	Active	NNR	Humboldt	Lode	19.33	7.82
NV101304690	NIKE #255	17/02/1988	Active	NNR	Humboldt	Lode	20.36	8.24
NV101730574	NIKE #256	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101347843	NIKE #257	17/02/1988	Active	NNR	Humboldt	Lode	20.24	8.19
NV101604261	NIKE #258	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101525884	NIKE #259	17/02/1988	Active	NNR	Humboldt	Lode	20.13	8.14
NV101602952	NIKE #260	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101524670	NIKE #261	17/02/1988	Active	NNR	Humboldt	Lode	20.01	8.10
NV101492811	NIKE #262	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101604623	NIKE #263	17/02/1988	Active	NNR	Humboldt	Lode	19.90	8.05
NV101459833	NIKE #264	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101605480	NIKE #265	17/02/1988	Active	NNR	Humboldt	Lode	19.78	8.01
NV101496434	NIKE #266	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101606876	NIKE #267	17/02/1988	Active	NNR	Humboldt	Lode	19.67	7.96
NV101459573	NIKE #268	17/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101347465	NIKE #269	17/02/1988	Active	NNR	Humboldt	Lode	17.82	7.21
NV101609402	NIKE #270	17/02/1988	Active	NNR	Humboldt	Lode	19.11	7.73
NV101349282	NIKE #271	18/02/1988	Active	NNR	Humboldt	Lode	20.14	8.15

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NV101340637	NIKE #272	18/02/1988	Active	NNR	Humboldt	Lode	20.12	8.14
NV101454774	NIKE #273	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101341942	NIKE #274	18/02/1988	Active	NNR	Humboldt	Lode	20.46	8.28
NV101406075	NIKE #275	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101345739	NIKE #276	18/04/1988	Active	NNR	Humboldt	Lode	20.47	8.28
NV101408655	NIKE #277	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101454934	NIKE #278	18/02/1988	Active	NNR	Humboldt	Lode	20.48	8.29
NV101400864	NIKE #279	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101492750	NIKE #280	18/02/1988	Active	NNR	Humboldt	Lode	20.49	8.29
NV101405636	NIKE #281	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101730532	NIKE #282	18/02/1988	Active	NNR	Humboldt	Lode	20.50	8.29
NV101409366	NIKE #283	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101528242	NIKE #284	18/02/1988	Active	NNR	Humboldt	Lode	20.50	8.30
NV101477139	NIKE #285	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101501854	NIKE #286	18/02/1988	Active	NNR	Humboldt	Lode	20.51	8.30
NV101301209	NIKE #287	18/02/1988	Active	NNR	Humboldt	Lode	19.93	8.06
NV101610377	NIKE #288	18/02/1988	Active	NNR	Humboldt	Lode	18.90	7.65
NV101527095	NIKE #289	18/02/1988	Active	NNR	Humboldt	Lode	11.97	4.84
NV101609879	NIKE #290	18/02/1988	Active	NNR	Humboldt	Lode	19.85	8.03
NV101523475	NIKE #291	18/02/1988	Active	NNR	Humboldt	Lode	12.11	4.90
NV101602278	NIKE #292	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101303255	NIKE #293	18/02/1988	Active	NNR	Humboldt	Lode	12.07	4.88
NV101609173	NIKE #294	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101731145	NIKE #295	18/02/1988	Active	NNR	Humboldt	Lode	12.03	4.87
NV101491557	NIKE #296	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101732062	NIKE #297	18/02/1988	Active	NNR	Humboldt	Lode	11.99	4.85
NV101605614	NIKE #298	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101548830	NIKE #299	18/02/1988	Active	NNR	Humboldt	Lode	11.94	4.83
NV101610242	NIKE #300	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101781004	NIKE #301	18/02/1988	Active	NNR	Humboldt	Lode	11.90	4.82
NV101341922	NIKE #302	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36
NV101730735	NIKE #303	18/02/1988	Active	NNR	Humboldt	Lode	11.86	4.80
NV101344517	NIKE #304	18/02/1988	Active	NNR	Humboldt	Lode	20.66	8.36

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV101460149	NIKE #305	18/02/1988	Active	NNR	Humboldt	Lode	11.60	4.69
NV101342021	NIKE #306	18/02/1988	Active	NNR	Humboldt	Lode	20.12	8.14
NV101345618	NIKE 331	13/05/1996	Active	NNR	Humboldt	Lode	0.59	0.24
NV101523453	NIKE 332	13/05/1996	Active	NNR	Humboldt	Lode	1.04	0.42
NV101602431	NIKE 333	13/05/1996	Active	NNR	Humboldt	Lode	0.54	0.22
NV101730820	NIKE 337	13/05/1996	Active	NNR	Humboldt	Lode	0.09	0.04
NV101491579	NIKE 341	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101758163	NIKE 342	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101731246	NIKE 343	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101756924	NIKE 344	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101344535	NIKE 349	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101494522	NIKE 350	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101300192	NIKE 351	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV102521531	NIKE 352	16/05/1996	Active	NNR	Humboldt	Lode	0.00	0.00
NV101755296	PUMP #37	23/03/1996	Active	CRL	Humboldt	Lode	19.35	7.83
NV101605685	PUMP #38	23/03/1996	Active	CRL	Humboldt	Lode	18.32	7.41
NV101459467	PUMP #39	23/03/1996	Active	CRL	Pershing	Lode	20.66	8.36
NV101602934	PUMP #40	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101497172	PUMP #41	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101609363	PUMP #42	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101540691	PUMP #43	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101350073	PUMP #44	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101478734	PUMP #45	23/03/1996	Active	CRL	Pershing	Lode	20.66	8.36
NV101301473	PUMP #46	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101756639	PUMP #47	23/03/1996	Active	CRL	Pershing	Lode	20.66	8.36
NV101305259	PUMP #48	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101345751	PUMP #49	23/03/1996	Active	CRL	Pershing	Lode	20.66	8.36
NV101347099	PUMP #50	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101502162	PUMP #51	23/03/1996	Active	CRL	Pershing	Lode	20.66	8.36
NV101479755	PUMP #52	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101457062	PUMP #53	23/03/1996	Active	CRL	Pershing	Lode	19.75	7.99
NV101305083	PUMP #54	23/03/1996	Active	CRL	Humboldt	Lode	18.19	7.36
NV101459046	PUMP #55	23/03/1996	Active	CRL	Humboldt	Lode	18.20	7.37

Serial Number	Claim Name	Date of Acquisition	Status	Claimant	County	Claim Type	Acres	ha
NV101301311	PUMP #56	23/03/1996	Active	CRL	Humboldt	Lode	16.61	6.72
NV101458990	PUMP #57	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101602186	PUMP #58	23/03/1996	Active	CRL	Humboldt	Lode	19.32	7.82
NV101452152	PUMP #59	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101604426	PUMP #60	23/03/1996	Active	CRL	Humboldt	Lode	19.31	7.82
NV101453992	PUMP #61	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101495683	PUMP #62	23/03/1996	Active	CRL	Humboldt	Lode	19.31	7.81
NV101478063	PUMP #63	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101460308	PUMP #64	23/03/1996	Active	CRL	Humboldt	Lode	19.31	7.81
NV101478059	PUMP #65	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101459138	PUMP #66	23/03/1996	Active	CRL	Humboldt	Lode	19.30	7.81
NV101479392	PUMP #67	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101731685	PUMP #68	23/03/1996	Active	CRL	Humboldt	Lode	19.30	7.81
NV101459403	PUMP #69	23/03/1996	Active	CRL	Humboldt	Lode	20.66	8.36
NV101606424	PUMP #70	23/03/1996	Active	CRL	Humboldt	Lode	19.29	7.81
NV101460003	PUMP #71	23/03/1996	Active	CRL	Humboldt	Lode	18.27	7.39
NV101608320	PUMP #72	23/03/1996	Active	CRL	Humboldt	Lode	17.15	6.94

Source: Roxmore 2026.

Note:

- NNR = Nevada North Resources (USA) Inc, CRL – Converse Resources LLC.
- Claim NIKE 341 to NIKE 352 are active claims but have subsequently been over staked by more recent claims and as such the acres has been set to zero.

Making Sustainability Happen