Rowan Project

NI 43-101 Technical Report and Preliminary Economic Assessment

Ontario, Canada

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Important Notice

Abbreviations and Units of Measure

Abbreviation	Definition
Actlabs	Activation Laboratories Ltd.
Au	gold
BWi	Bond Work Index
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
CN_{WAD}	weak acid dissociable cyanide
Cu	copper
CV	coefficient of variation
DDH	diamond drill hole
EA	environmental assessment
ELOS	equivalent overbreak slough
FA-AAS	fire assay with atomic absorption spectroscopy
Fuse Advisors	Fuse Advisors Inc.
HDPE	high-density polyethylene
ICP	inductively coupled plasma
IDF	Inflow Design Flood
Integrated Sustainability	Integrated Sustainability Consultants Ltd.
IP	induced polarization
IRR	Internal Rate of Return
ISO	International Organization for Standardization
ISRM	International Society for Rock Mechanics and Rock Engineering
KP	Knight Piésold Ltd.
LHD	load-haul-dump
LiDAR	light detection and ranging
LIMS	Laboratory Information Management System
LOM	life of mine
MC	master composite
MDMER	Metal and Diamond Mining Effluent Regulations
MECP	Ministry of Environment, Conservation and Parks (Ontario)
MINES	Ministry of Mines (Ontario)
MNR	Ministry of Natural Resources (Ontario)
MNRF	Ministry of Natural Resources and Forestry (Ontario)
MOL	Ministry of Labour, Training and Skills Development (Ontario)
MSO	minable shape optimizer
MTO	Ministry of Transportation (Ontario)
NPV	Net Present Value
NSR	Net Smelter Return
NT	Newman-Todd
Pb	lead
PBDZ	Pipestone Bay-St Paul Deformation Zone
PEA	preliminary economic assessment
PFS	pre-feasibility study
PMP	Probable Maximum Precipitation

Abbreviation	Definition
Project	Rowan Project
QA/QC	quality assurance and quality control
QEMSCAN	quantitative evaluation of minerals by scanning electron microscopy
QMS	Quality Management System
QP	Qualified Person
RCM	Rowan Consolidated Mines
RLG	West Red Lake Gold Mines Inc.
RMR	Rock Mass Rating
ROM	Run-of-Mine
SAG	semi-autonomous grinding
SARA	*Species at Risk Act*
SD	standard deviation
SG	specific gravity
SGS	SGS Natural Resources
SMC	SAG mill comminution
SO_2	sulphur dioxide
SR	SIMS Resources LLC
SRK	SRK Consulting (Canada) Inc.
UG	underground
WBM	water balance model
WBS	work breakdown structure
WRLG	West Red Lake Gold Mine Ltd.
WRS	Waste Rock Stockpile
WTP	Water Treatment Plant
Zn	zinc

Unit	Definition
%	percent
<	less than
≤	less than or equal to
>	more than
°	degree
°C	degrees Celsius
µm	micron
C$/t	Canadian dollars per tonne
C$	Canadian dollar
C$M	million Canadian dollars
cm	centimetre
ft	foot or feet
g/cm^3	grams per cubic centimetre
g/L	grams per litre
g/t	grams per tonne
g	gram
h	hour

Unit	Definition
ha	hectare
in.	inch
kg	kilogram
kg/t	kilograms per tonne
km^2	square kilometres
km	kilometre
km/h	kilometres per hour
koz	thousand ounces
kt	kilotonne
kWh	kilowatt-hour
kWh/m^3	kilowatt-hours per cubic metre
kWh/t	kilowatt-hours per tonne
m	metre
m^2	square metres
m^3	cubic metres
m^3/s	cubic metres per second
masl	meters above sea level
mbgs	metres below ground surface
mg/L	milligrams per litre
mL	millilitre
mm	millimetre
Moz	million ounces
oz/t	ounces per tonne
oz	ounce
ppb	parts per billion
ppm	part per million
t	tonnes
t/m^3	tonnes per cubic metre
t/d	tonnes per day
t/h	tonnes per hour
US$/oz	United States dollars per ounce
USD/CAD	United States Dollar to Canadian Dollar

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1 SUMMARY

1.1 Introduction

West Red Lake Gold Ltd. (WRLG) commissioned Fuse Advisors Inc. (Fuse Advisors) to compile a preliminary economic assessment (PEA or Technical Report) of the Rowan Project (Project). This PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101F1.

The Project will include an underground mine and surface facilities to support the extraction of gold ore from the Rowan deposit. The Project will span a seven-year period from the beginning of Construction, through Operations, to the end of Closure (Table 1-1). Construction activities will include site preparation, infrastructure construction, and capital development in the underground mine. Operations will include underground mining and the associated waste rock management, water management, and water treatment. During Operations, ore will be trucked from the Project site to an off-site toll milling facility for processing. Closure will include reclamation and closure activities.

Table 1-1: Phases and Durations for the Rowan Project

Project	Duration	Project Years
Construction	1 year	Year -1
Operations	5 years	Year 1 to Year 5
Closure	1 year	Year 6
Project lifespan	**7 years**	**-**

Notes:
- = not applicable.

The responsibilities of the engineering consultants and firms that were contracted by WRLG to prepare the Technical Report are as follows:

- Fuse Advisors Inc. (Fuse Advisors) managed and coordinated the work related to the Technical Report and developed the PEA-level design and cost estimates for mine operation, mine production schedule, mine capital costs, mine operating costs, toll milling costs, processing and metallurgy, general site infrastructure, ore haulage, review of environmental studies and permitting, and economic analysis.
- SIMS Resources LLC (SR) developed the mineral resource estimate for the Project and completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, sample preparation and analysis, and data verification.
- Knight Piésold Ltd. (KP) completed work related to waste rock and water management, site-wide water balance, and the associated cost estimate.
- PHC Inc. completed work related to underground geotechnical design.
- Integrated Sustainability Consultants Ltd. (Integrated Sustainability) developed the water treatment design and the associated cost estimate.

1.2 Property Description and Ownership

The Rowan Property is located in the Todd, Hammell Lake, and Fairlie townships, Red Lake Mining Division, District of Kenora (Patricia Portion), northwestern Ontario, Canada. The Rowan Property is 100% owned by WRLG (Ontario), a wholly owned subsidiary of WRLG, and consists of 146 contiguous patented, leased, and staked mining claims located 16 km west-northwest of the town of Red Lake, Ontario and 25 km due west of Evolution Mining's Red Lake Mine located in Balmertown, Ontario. These 146 mining claims cover a total area of 3,100 ha and include 58 patented claims, 20 leased claims, 65 staked crown claims, and 3 under licence-of-occupation claims.

1.3 History

Several companies have worked the Rowan Property claim group since the 1928 discovery of gold on "Discovery Hill" by the Rowan Hall Syndicate. Surface trenching completed on Discovery Hill in the early 1930s identified the Rowan Vein System. From 1936 to 1939, an adit was driven along the Rowan vein from the base of Discovery Hill, followed by sinking of a 425-foot (ft) shaft and three levels of development that were supported by previous surface and underground drilling results. From 1945 to 1947, drilling was completed near Rowan Lake. Underground work recommenced in 1953, with further development of the third level to the east. In 1958, additional drilling was completed to extend the Rowan Vein System over the strike length, however, work was discontinued after 1958. From 1981 to 2005, a number of companies completed geological prospecting, geophysical surveys, and drilling.

Gold in the Mount Jamie Mine area was discovered in the area of Shaft No. 1 in 1920. Eleven claims were patented in 1928. No information prior to 1934 is available regarding ownership or work history of the claims. Since 1934, various companies have owned and operated the Mount Jamie Mine, with work generally completed on two of the three veins known in the mine area. The work included surface and underground drilling, development of two shafts (i.e., Shaft No. 1 and Shaft No. 2), and construction of a mill. The historical records of actual mining (i.e., ore hoisted to surface) are limited due to poor record keeping. Approximately 2,000 tons to 3,000 tons of material were mined, with some material treated at an on-site 100-ton/day mill and the rest of the material stockpiled.

Gold in the Red Summit Mine area was discovered in the early 1930s. From 1935 to 1938, a shaft was sunk, and a 5-ton/day mill was installed and operated to treat high-grade ore from surface and to test some underground vein material.

Between 2005 to 2009, Hy Lake Gold Inc. (Hy Lake) entered into a number of option agreements to acquire claims that constitute the present Rowan Property. In 2012, Hy Lake changed its name to West Red Lake Gold Mines Inc. (RLG).

Between 2007 and 2021, Hy Lake/RLG completed approximately 40,000 m of diamond drilling at the Rowan Property and Newman-Todd (NT) Zone (also known as the NT Horizon), a total of approximately 11,000 m of diamond drilling at the Mount Jamie Mine, and approximately 4,400 m of diamond drilling at the Red Summit Mine.

1.4 Geological Setting and Mineralization

The Rowan Property is situated at the west end of the Red Lake Greenstone Belt. This belt is comprised of a relatively narrow series of six metavolcanic/metasedimentary supracrustal assemblages intruded by several bodies of variable size, form, and composition. These six assemblages have undergone several phases of deformation and metamorphism. The rocks that comprise the Rowan deposit are Mesoarchean and Neoarchean age and form part of the larger Uchi Subprovince of the Archean Superior Province of the Canadian Shield.

The Rowan Property is centred on a regional antiform that plunges moderately to the east, and straddles the intersection of two regional gold corridors (i.e., Pipestone Bay-St Paul Deformation Zone [PBDZ] and Golden Arm Fault). The Rowan Property mineralization contains typical Archean lode-style gold zones hosted within a sequence of hydrothermally altered mafic volcanics, with intercalated felsic volcanics and porphyries as well as ultramafics. The gold mineralization is associated with quartz veining and increased iron sulphide mineralization.

Currently, three principal gold occurrences known on the Rowan Property include the three past-producing mines (i.e., Rowan Mine, Mount Jamie Mine, and Red Summit Mine), the NT Zone, as well as numerous gold prospects. In general, gold mineralization occurs as visible millimetre-scale blebs in quartz veins, veinlets, and stockworks.

1.5 Exploration

WRLG's exploration concept has been to explore the 12 km section of the regional deformation zone and the 2 km section of the NT Zone situated on the Rowan Property with the purpose of identifying areas that have the potential to become a mineral resource. Three historical mines (i.e., Rowan Mine, Mount Jamie Mine, and Red Summit Mine) are situated on the east-west trending regional deformation zone on the Rowan Property.

The Rowan Property is currently at the exploration stage; the previous owner, RLG, had completed numerous exploration diamond drill programs on the Rowan Property since 2007. The majority of the exploration, which was completed by RLG between 2016 and 2022 and by WRLG in 2023, was completed on the historical Rowan Mine target area.

1.6 Drilling and Sampling

WRLG completed 77 diamond drill holes in 2022 and 2023 for a total length of approximately 25,000 m with the purpose of infilling and expanding the Rowan deposit as well as targeting additional zones of mineralization on the Rowan Property. This 2022 and 2023 drilling brings the total drill holes for the Rowan Property to 622 drill holes for a total length of approximately 128,000 m since 1934.

The drilling in 2022 and 2023 was completed with fully oriented NQ size core by Forage Lamontagne Fortier Inc., based out of Rouyn-Noranda, Quebec. Collar locations were captured using a high-precision (i.e., less than 1 m accuracy) Trimble R2 GPS. Drill hole orientation was recorded down hole at approximately 30 m intervals using an Imdex-Devico DeviGyro tool. Orientation marks were made on the drill core by trained drilling personnel using a Reflex Act III orientation tool.

A total of 24,147 primary samples were collected and submitted for gold assay from the 2022 and 2023 drilling campaigns. All samples were analysed by fire assay with atomic absorption spectroscopy (FA-AAS); samples that returned gold values greater than 10 grams per tonne (g/t) of gold (Au) were reanalyzed by fire assay with a gravimetric finish, and samples with visible gold were also analyzed by metallic screen analysis. All 2022 and 2023 drill hole sample results were incorporated into the 2025 mineral resource estimate.

1.7 Mineral Processing and Metallurgical Testwork

The metallurgical testwork program for the Rowan deposit was completed by Base Met Labs US Ltd. (Base Met Labs) in late 2023 on four master composite samples representing Vein 101, Vein 102, Vein 103, and Vein 104. This testwork program included chemical and mineralogical characterization, comminution testing, extended gravity recoverable gold testing, gravity leach testing, and cyanide destruction testing.

The ore from the Rowan deposit is characterized by high gravity-recoverable gold, low sulphur content, and no deleterious elements that would preclude successful gold extraction from the material. Gravity recoveries ranged from 28% to 81%, and overall gold extraction ranged from 94.3% to 99.5% after 24 hours. The leaching process was fast, with the majority of gold extracted (i.e., up to 99.3%) within the first 6 hours. High gold recoveries were also maintained at a coarser grind size of 125 microns (µm), indicating the ore is not highly sensitive to grind size within the range tested (i.e., 75 µm to 125 µm). Comminution testing confirmed that the material is hard, with a Bond Work Index (BWi) ranging from 16.2 kilowatt-hours per tonne (kWh/t) to 18.2 kWh/t.

Cyanide detoxification testing using the sulphur dioxide (SO_2)/air method confirmed that all four master composite samples could meet the target weak acid dissociable cyanide (CN_{WAD}) concentration, with cyanide detoxification optimization showing potential for reduced reagent consumption and retention time.

A design gold recovery of 97% was selected for the Project to account for minor expected losses in a commercial operating environment. The metallurgical testwork results support the use of a conventional gravity-leach-carbon-in-pulp (CIP) flowsheet for processing of ore from the Project.

1.8 Mineral Resource Estimate

Mineral resources for the Rowan deposit were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM 2014). The modelling and estimation of the mineral resources was completed between 1 January 2024 and 1 March 2024, by or under the supervision of John Sims, President of Sims Resources LLC (SR) and the Qualified Person (QP) for this mineral resource estimate. In June 2025, some minor updates were completed to the resource model that were not material but increased the overall accuracy of the mineral resource estimate. As of an effective date of 30 June 2025, indicated mineral resources are estimated to total approximately 478,707 tonnes grading 12.78 g/t Au and containing approximately 196,700 ounces of gold. In addition, inferred mineral resources are estimated to be approximately 421,181 tonnes grading 8.73 g/t Au and containing approximately 118,200 ounces of gold (Table 1-2).

For each area, domains representing gold mineralization were defined in Leapfrog Geo version 2024.1 software, and sub-block model estimates were completed in Leapfrog Edge software using 2.0 m capped composites and a single-pass inverse distance cubed (ID3) interpolation approach. Blocks were classified considering local drill hole spacing. Class groupings were based on criteria developed using continuity models (i.e., variograms) and modified to reflect geological understanding and to verify cohesive classification shapes.

Wireframe and block model validation procedures were completed for all zones, including wireframe to block volume confirmation, statistical comparisons of composite gold grades versus ID3, and nearest neighbour estimates using swath plots, visual reviews in three-dimensional (3D), as well as longitudinal, cross-section, 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 2025 mineral resource estimate.

Table 1-2: 2025 Mineral Resource Estimate Summary for the Rowan Deposit as of 30 June 2025

Category	Tonnage [t]	Average Grade [g/t Au]	Contained Metal [oz Au]
Indicated	478,707	12.78	196,747
Inferred	421,181	8.73	118,155

Notes:
- CIM (2014) definitions were followed for mineral resources.
- Mineral resources were estimated at a gold cut-off grade of 3.80 g/t using a long-term gold price of US$1,800 per ounce.
- There are no mineral reserves currently estimated at the Rowan deposit.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Mineral resources are reported within vein wireframes at the stated cut-off grade of 3.80 g/t Au.
- Density of 2.8 g/cm^3.
- Numbers may not add due to rounding.

t = tonne; g/t = grams per tonne; Au = gold; oz = ounce; g/cm^3 = grams per cubic centimetre.

1.9 Mining Methods

The Rowan deposit will be developed as an underground mine using conventional longhole retreat open stoping. Due to the narrow nature of the mineral deposit, a maximum of 475 tonnes per day (t/d) of ore is planned to be produced; this maximum is comprised of 400 t/d produced from underground stopes and 75 t/d of ore extracted by sublevel development along the veins of the Rowan deposit. During Operations, a total of approximately 705 kilotonnes (kt) of ore will be mined at an average gold grade of 8.01 g/t and is estimated to contain approximately 182 thousand ounces (koz) of gold.

The majority of excavated open stopes will be backfilled with unconsolidated rock fill; consolidated (e.g., cemented) rock fill will be used as backfill in the three sill levels where additional stopes will be mined below the backfilled area. Ore from the underground will be temporarily stockpiled on surface for crushing and sampling until the ore is hauled off site for processing. Waste rock will be preferentially used as backfill underground in excavated open stopes or will be temporarily stockpiled on surface. Geotechnically, the Rowan deposit geometry, rock mass conditions, and deposit depth are conducive to longhole open stoping with delayed backfill. A minimum stope width of 2.0 m with 15% added dilution was applied in the underground mine design.

A summary of the underground mine production schedule is provided in Table 1-3. The underground mine was designed at a cut-off grade of 3.0 g/t Au with a marginal cut-off grade of 1.5 g/t Au.

This PEA is preliminary in nature and includes 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, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Table 1-3: Mine Plan Production Summary for the Rowan Project

Mine Production Metric	Unit	Life of Mine Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5
Lateral Development								
Sublevel development	m	3,637	-	1,369	1,328	820	120	-
Capital development	m	6,487	1,736	1,364	1,527	1,858	2	-
Total lateral development	**m**	**10,124**	**1,736**	**2,733**	**2,855**	**2,678**	**122**	**-**
Total lateral development waste rock	**t**	**470,200**	**115,900**	**105,600**	**114,600**	**134,000**	**56**	**-**
Vertical Development								
Ventilation raises (raise bore)	m	197	-	197	-	-	-	-
Ventilation raises (drop raise)	m	210	-	-	88	122	-	-
Total vertical development	**m**	**407**	**-**	**197**	**88**	**122**	**-**	**-**
Total vertical development waste rock	**t**	**17,135**	**-**	**6,871**	**4,286**	**5,979**	**-**	**-**
Ore Production								
Development ore	t	91,574	-	34,400	32,400	21,100	3,800	-
Ore grade	g/t	9.43	-	8.91	8.64	11.24	10.88	-
Contained gold	oz Au	27,770	-	9,850	8,990	7,610	1,320	-
Stope ore	t	613,600	-	75,500	93,000	148,900	162,600	133,600
Ore grade	g/t	7.75	-	11.1	5.99	6.84	6.42	9.94
Contained gold	oz Au	153,830	-	26,940	17,910	32,730	33,570	42,690
Total Ore Tonnes	**t**	**705,185**	**-**	**109,900**	**125,300**	**167,000**	**166,400**	**133,600**
Total ore grade	**g/t**	**8.01**	**-**	**10.41**	**6.67**	**7.38**	**6.52**	**9.94**
Total contained gold mined	**oz Au**	**181,600**	**-**	**36,800**	**26,900**	**40,300**	**34,900**	**42,700**
Backfill Schedule								
Consolidated rock fill	t	85,200	-	15,200	24,400	27,500	18,000	-
Unconsolidated rock fill	t	278,500	-	29,500	30,700	60,700	78,400	79,200

Notes:
Numbers may not add due to rounding.
m = metre; t = tonne; g/t = grams per tonne; oz = ounce; Au = gold; - = not applicable.

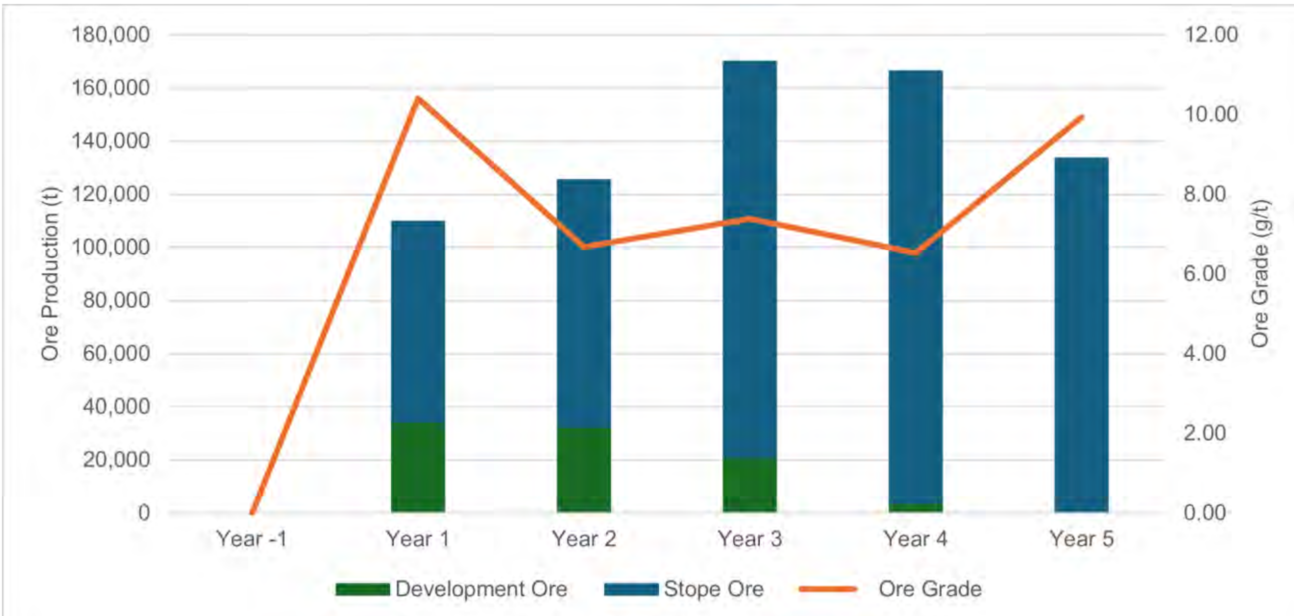
Stope shapes were generated using Deswik Stope Optimizer with the minimum 2 m stope thickness and a cut-off grade of 1.5 g/t to allow for inclusion of marginal grade stopes. Stopes were reviewed and removed manually to confirm profitability. Areas of material identified as above cut-off grade, but far enough away from the main capital development to be uneconomic, were removed from the mine design. All stope shapes were cut by sublevel development and reviewed against the provided block model.

Mining recovery for stopes was assumed to be 95% to account for ore loss during mining. An external dilution factor (i.e., 15% additional tonnes) was included above the minimum 2 m designed stope thickness. Dilution was given a grade of 0 g/t in the resource model, as mineralized material does not occur in significant quantities outside of the vein boundaries.

For horizontal pillars between sublevels on the same elevation, a minimum pillar width of 10 m will be maintained between stopes in cases where multiple veins exist on a single sublevel.

Approximately six months of capital development is required to access the first underground stoping faces. Ore production from the underground will average 385 t/d and will reach a peak production of approximately 465 t/d to 475 t/d in Year 3 when underground equipment transitions from capital development to production. The total Project ore production is shown on Figure 1-1.

Figure 1-1: Total Ore Production by Year for the Rowan Project



Notes:
t = tonnes; g/t = grams per tonne.

The Project will operate 24 hours per day and 365 days per year during the 5-year period of Operations. The diesel-powered underground mobile equipment fleet will include:

- Two-boom jumbo drills for capital development drilling;
- Top hammer longhole drill for production drilling;
- 7 t load-haul-dump (LHD) machines for narrow sublevel development and stope production areas;
- 10 t LHD for capital development drifts;
- 30 t haul trucks to transport ore to surface, waste rock to surface, and waste rock to excavated underground areas requiring backfill; and
- ancillary mobile equipment to support the mine operations.

The 7 t LHDs will include remote-operation capability to remove material from the open stopes to limit equipment operator exposure to unsupported ground. Consolidated rock fill will be mixed on surface with a temporary cement plant and two of the 30 t haul trucks will include ejector boxes to support underground backfilling of consolidated rock fill in the three sill levels.

Mine dewatering systems will include sumps constructed on each underground mine level to collect groundwater inflows and used service water generated during drilling. Collected water will be pumped to water management ponds on surface for management and treatment. An underground dewatering rate of 500 cubic metres per day (m^3/day) was assumed during Operations.

1.10 Recovery Methods

A Crusher and Sample Tower at the Project site will prepare ore prior to shipment to an off-site toll milling facility in the Red Lake region. This equipment will include a mobile primary jaw crusher, and a fully integrated sample tower designed to operate 12 hours per day and 365 days per year on day shift only, and is sized to support the Project maximum throughput of 475 t/d.

There are several existing ore processing facilities located within the Red Lake region of northwestern Ontario that could, in principle, be utilized as the off-site toll milling facility for ore from the Rowan deposit. These existing facilities currently operate conventional gravity-leach-CIP flowsheets and have extra milling capacity to receive ore from the Project. For the purposes of the Technical Report, one facility was selected as the basis for capital and operating cost estimates.

The selected toll milling facility is currently capable of treating ore from the Rowan deposit with little to no modifications; however, targeted upgrades are recommended as part of the Project to achieve the gold recoveries demonstrated in metallurgical testwork at the increased throughput. Key upgrades to the off-site toll milling facility would include:

- expansion of the existing crushing circuit with an additional secondary crusher (i.e., pre-crusher) to reduce ore feed size prior to grinding;
- expansion of the existing gravity circuit with an additional gravity concentrator and step deck vibrating screen;
- expansion of the existing pre-leaching thickening capacity with an additional pre-leach thickener; and
- expansion of the existing leaching, CIP, and cyanide detoxification circuits with additional pre-oxidation, leach, CIP, and cyanide detoxification tanks to maintain residence time under the higher throughput conditions.

The gravity concentrate would be processed in the existing intensive leach circuit, and no significant changes would be required to the existing grinding, refining, or gold recovery circuits at the selected off-site toll milling facility. Additional power and reagent use at the off-site toll milling facility would be required to support the Project, however, no modifications would be required to the power, water, and reagent systems at the off-site toll milling facility. The tailings management facility at the off-site toll milling facility would have sufficient capacity to accommodate the additional tailings generated from the Project and would require the next scheduled dam raise of this tailings management facility to be advanced by approximately one year. All proposed upgrades to the selected off-site toll milling facility were incorporated into the capital and operating cost estimates for the Project.

1.11 Project Infrastructure

The Project includes development and production of the underground mine as well as construction of civil infrastructure, ore and waste rock stockpiles, ore crushing and sampling facilities, water management and water treatment facilities, and ancillary infrastructure. There are no processing or tailings management facilities located at the Project site. Project infrastructure will include the following key facilities:

- underground mine, including the Portal at surface;

- ore storage facilities, including the Run-of-Mine (ROM) Ore Stockpile and Sampled Ore Stockpile;
- Crusher and Sample Tower;
- waste rock management facilities, including the Portal Pad and Waste Rock Stockpile (WRS);
- water management infrastructure, including ponds, channels, pipelines, and Water Treatment Plant (WTP);
- ancillary infrastructure, including:
  - Mine Dry, Administration Offices, and maintenance and warehouse facilities;
  - fuel storage facilities and power generation facilities;
  - exploration facilities;
  - Personnel Camp;
  - laydown facilities; and
  - site roads.

The Portal Pad will be constructed on surface using local borrow source material or waste rock from the underground mine; this pad will contain the ROM Ore Stockpile, Sampled Ore Stockpile, Crusher and Sample Tower, WRS, and water management infrastructure.

Ore from the underground will be crushed, sampled, and temporarily stored in the Sampled Ore Stockpile until this material is transported to an off-site toll milling facility for processing. Waste rock from development underground mining will be stored in the WRS, and the majority of this material will be used as backfill in the excavated underground workings.

Two water management ponds adjacent to the Portal Pad will manage site run-off and treated water from the WTP. Contact water, primarily from underground dewatering, will be treated year-round in the Water Treatment Plant. Treated water will be annually discharged from the Treated Water Pond for approximately eight months of the year and stored for approximately four months of the year. Surface water management infrastructure will include diversion and collection channels to separate non-contact and contact water. Seepage management infrastructure will include foundation drains and return sumps to collect seepage and groundwater, which will be managed as contact water.

1.12 Market Studies and Contracts

It was assumed in this PEA that the Project will produce gold in the form of doré bars. The market for doré is well-established and accessible to new producers. The doré bars will be refined in a certified North American refinery and the gold will be sold on the spot market. Gold is a freely traded commodity on the world market and there is a steady demand from multiple buyers.

No market studies were conducted regarding the sale of gold doré from the Rowan Project as part of the Technical Report. Existing terms and conditions from previous sales by WRLG were used as the basis for the economic modelling. Ore refining and ore transportation costs total to C$1.50/oz of gold produced.

1.13 Environmental, Permitting, and Social Considerations

WRLG is continuing scientific and engineering studies at the Project site; First Nation, community, and regulatory consultations; monitoring programs; and design planning to progress the Project. WRLG has focused its efforts since acquisition on reducing the uncertainty and risk associated with any new mining development and is actively designing operations to minimize water use, improve water quality, and bring overall benefit to First Nations and local communities.

The Project will require a Class environmental assessment (EA) because it is considered a Resource Stewardship and Facility Development Project in accordance with the Ministry of Natural Resources (MNR 2003). There is no requirement for a federal EA nor a provincial individual EA. WRLG will engage with all appropriate regulatory authorities to complete the appropriate submissions of any additional federal, provincial, and municipal permits and approvals that will be required for development of the Project.

WRLG has committed to engagement and consultation with local First Nations; municipal, provincial, and federal governments; other stakeholders; and the public throughout all stages of the Project. The intent of this engagement and consultation is to provide all interested parties with opportunities to learn about WRLG, identify concerns, and provide input with the goal of positively enhancing Project planning and development. WRLG recognizes the importance of timely, full, and open discussion of concerns and options associated with the Project and the related concerns those individuals or communities may have regarding Project activities. WRLG will maintain open and honest communications with local communities and individual stakeholders throughout all stages of the Project. WRLG will establish operational practices, both now and into the future, that reflect the values, expectations, and needs of the community in which WRLG operates, and that are based on continued mutually respectful consultation with all First Nations and other stakeholders.

1.14 Capital and Operating Cost Estimates

The preliminary economic evaluation of the Rowan Project was based on the capital and operating cost estimates outlined in the Technical Report. These cost estimates reflect an underground mining operation, and account for the development of both underground and surface infrastructure, including underground mine development and production, ore storage facilities, crushing and sampling facilities, waste rock management facilities, water management facilities, WTP, and ancillary infrastructure, as well as Owner's costs and appropriate provisions.

The capital and operating cost estimates conform to a PEA-level estimate with a ±30% accuracy range which is within the boundaries of the Association for the Advancement of Cost Engineering International Class 5 guidelines (AACE International 2012).

The capital and operating cost estimates were developed in Canadian dollars (C$) in the second quarter of 2025 (Q2 2025), with no escalation or exchange rate variations factored into these estimates. These estimates were developed using first-principles engineering combined with industry benchmarking against similar underground mining operations in Ontario.

1.14.1 Capital Cost Estimate

The initial capital cost for the Project is estimated to be C$70.4 million and the life of mine (LOM) sustaining capital cost is estimated to be C$102.6 million. A summary of capital costs for the Project is provided in Table 1-4. In addition to these capital costs, a closure cost of C$3.2 million was applied in Year 6 for reclamation and remediation activities.

Table 1-4: Capital Cost Estimate Summary for the Rowan Project

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Capital [C$M]
Mine Development	1000	13.4	58.6	72.1
Infrastructure	2000	14.2	3.2	17.4
Equipment	3000	9.1[(a)]	17.1	26.2
Crushing and Sampling	4000	2.2	-	2.2
Water Treatment Plant	5000	7.4	-	7.4
Waste Rock and Water Management	6000	4.9	-	4.9
Total Direct Costs		**51.3**	**78.9**	**130.2**
Project Indirect Costs	7000	1.4	-	1.4
Owner's Costs	8000	4.6	-	4.6
Contingency Costs	9000	13.2	23.7	36.9
Total Indirect Costs		**19.2**	**23.7**	**42.8**
Total Capital Costs		**70.4**	**102.6**	**173.0**

Notes:

Numbers may not add due to rounding.

a) There is an additional cost of C$15.3 million required for mining equipment associated with the assumed leasing agreement, which is included in the operating cost estimate for the Project.

WBS = work breakdown structure; C$M = million Canadian dollars.

1.14.2 Operating Cost Estimate

The operating costs for the Project include expenditures related to on-site underground mining, waste rock and water management, water treatment, crushing and sampling, and the associated maintenance, power supply, and general and administrative costs, as well as off-site toll milling and trucking of ore.

The total operating costs over the LOM are estimated to be C$213.6 million, with average annual operating costs of C$42.7 million. The total operating unit cost for the LOM is estimated to be C$302.8/t processed, which includes an on-site operating unit cost of C$207.1/t processed, and an off-site operating unit cost of C$95.7/t processed. A summary of the operating cost estimate for the Project is presented in Table 1-5.

Table 1-5: Operating Cost Estimate Summary for the Rowan Project

Cost Centre	LOM Cost [C$M]	Annual Average Cost [C$M]	LOM Cost [C$/t Processed]	Operating Cost [%]
On-Site Costs	**146.0**	**29.2**	**207.1**	**68%**
Mining	134.0	26.8	190.0	63%
Waste Rock and Water Management	0.4	0.1	0.5	0%
Water Treatment	3.0	0.6	4.3	1%
Crushing and Sampling	8.7	1.7	12.3	4%
Off-Site Costs	**67.5**	**13.5**	**95.7**	**32%**
Toll Milling	47.6	9.5	67.4	22%
Trucking to Toll Mill	10.6	2.1	15.0	5%
General and Administrative	9.4	1.9	13.3	4%
Total Operating Costs	**213.6**	**42.7**	**302.8**	**100%**

Notes:
Numbers may not add due to rounding.
LOM = life of mine; C$M = million Canadian dollars; C$/t = Canadian dollars per tonne; % = percent.

1.15 Economic Analysis

The economic analysis was completed using an annual discounted cash flow analysis and a discount rate of 5% was applied, which is typical for gold projects in Canada. Cash flows were discounted to the start of Construction, assuming that the Project execution decision will be made, and major financing for the Project will be carried out at this time.

For the economic analysis of the Project, the gold price was assumed at US$2,500/oz and a USD/CAD exchange rate of 1.35 was used. This gold price was determined to be appropriate as it reflects a reasonable scenario to test Project sensitivity for potential investor returns while remaining within the range of recent gold price trends.

The pre-tax Net Present Value (NPV) discounted at 5% is C$153.8 million, the Internal Rate of Return (IRR) is 47.2%, and the payback period is 2.4 years. The post-tax NPV discounted at 5% is C$125.3 million, the IRR is 41.9%, and the payback period is 2.4 years. Tax calculations were based on the tax law in place as of the date of the Technical Report, which included a 26.5% combined federal and provincial tax rate.

A sensitivity analysis was completed for the Project on both the base case pre-tax NPV and IRR and the base case post-tax NPV and IRR. The results of the sensitivity analysis indicate that NPV for the Project is most sensitive to gold price and gold mill head grade. It is noted that if a gold price of US$3,250/oz was assumed instead of the US$2,500/oz used in the economic analysis, the post-tax NPV discounted at 5% would be C$238.8 million, the IRR would be 81.7%, and the payback period would be 1.4 years.

This PEA is preliminary in nature and includes 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, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

This Technical Report was based on the 2025 mineral resource estimate for the Project, which is comprised of 63% indicated resources and 37% inferred resources by tonnes.

A summary of the Project economic analysis is provided in Table 1-6 and the cumulative post-tax discounted free cash flow for the Project is shown on Figure 1-2.

Table 1-6: Economic Analysis Summary for the Rowan Project

Description	Unit	Life of Mine Total or Life of Mine Average
General		
Gold Price	US$/oz	2,500
Exchange Rate	USD/CAD	1.35
Life of Mine	years	5
Total Ore Mined	kt	705
Total Toll Mill Feed	kt	705
Production		
Mill Head Grade	g/t	8.01
Head Grade Contained Gold	koz	181.6
Mill Recovery Rate	%	97
Total Gold Recovered	koz	176.2
Total Average Annual Gold Production	koz	35.2
Operating Costs		
On-Site Costs	**C$/t milled**	**207.1**
Rowan Mining Cost	C$/t milled	190.0
Waste Rock and Water Management	C$/t milled	0.5
Water Treatment	C$/t milled	4.3
Crushing and Sampling	C$/t milled	12.3
Off-Site Costs	**C$/t milled**	**95.7**
Toll Milling	C$/t milled	67.4
Trucking to Toll Mill	C$/t milled	15.0
General and Administrative	C$/t milled	13.3
Ore Refining and Ore Transportation	C$/t oz	1.5
Royalty	%	2.5
Cash Costs[(a)]	US$/oz Au	962.5
All-In Sustaining Cost[(b)]	US$/oz Au	1,407.8
Capital Costs		
Initial Capital	C$M	70.4
Sustaining Capital	C$M	102.6
Closure Costs	C$M	3.2
Total Capital Costs	C$M	176.2
Economic Model Results		
Pre-tax NPV @ 5%	C$M	153.8
Pre-tax IRR	%	47.2
Pre-tax Payback	years	2.4
Post-tax NPV @ 5%	C$M	125.3
Post-tax IRR	%	41.9
Post-tax Payback	years	2.4

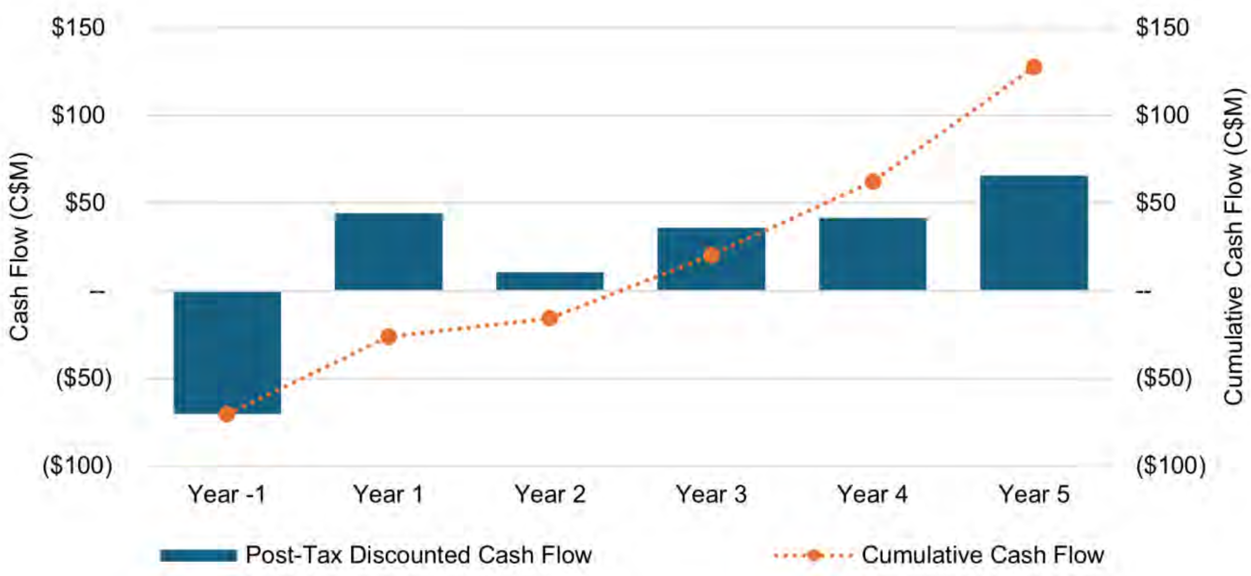
Notes:

a) Cash costs consist of mining costs, processing costs, mine-level G&A, ore refining charges, and royalties.

b) All-In Sustaining Cost includes cash costs plus sustaining capital and closure cost value.

US$/oz = United States dollars per ounce; USD/CAD = United States Dollar to Canadian Dollar; kt = kilotonne; g/t = grams per tonne; koz = thousand ounces; C$/t = Canadian dollars per tonne; oz = ounce; Au = gold; C$M = million Canadian dollars; NPV = Net Present Value; IRR = Internal Rate of Return; % = percent.

Figure 1-2: Post-tax Discounted Free Cash Flow for the Rowan Project



Notes:
C$M = million Canadian dollars.

1.16 Conclusions and Recommendations

This Technical Report was based on the 2025 mineral resource estimate for the Project, which is comprised of 63% indicated resources and 37% inferred resources by tonnes.

The proposed underground mine plan for the Project shows the potential for a small tonnage, high-grade open stoping mine that will produce approximately 705 kt of ore at an average gold grade of 8.01 g/t that contains approximately 182 koz of gold.

Based on the assumptions and parameters presented in the Technical Report, the Project shows positive economics with a post-tax NPV discounted at 5% of C$125.3 million, an IRR of 41.9%, and a payback period of 2.4 years.

It is recommended the Project is progressed through subsequent study phases. Additional studies and activities are recommended in the following areas to support Project advancement through the pre-feasibility study (PFS) stage:

- additional drilling to support an update to the mineral resource estimate;
- mineral processing and metallurgical testwork;
- geotechnical studies to support underground mine engineering;
- underground mine engineering;
- recovery methods;
- waste rock and water management; and
- environmental studies and permitting.

2 INTRODUCTION

West Red Lake Gold Mines Ltd. (WRLG) commissioned Fuse Advisors Inc. (Fuse Advisors) to compile a preliminary economic assessment (PEA or Technical Report) of the Rowan Project (Project). This PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101F1.

The responsibilities of the engineering consultants and firms that were contracted by WRLG to prepare the Technical Report are as follows:

- Fuse Advisors Inc. (Fuse Advisors) managed and coordinated the work related to the Technical Report and developed the PEA-level design and cost estimates for mine operation, mine production schedule, mine capital costs, mine operating costs, toll milling costs, processing and metallurgy, general site infrastructure, ore haulage, review of environmental studies and permitting, and economic analysis.
- SIMS Resources LLC (SR) developed the mineral resource estimate for the Project and completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, sample preparation and analysis, and data verification.
- Knight Piésold Ltd. (KP) completed work related to waste rock and water management, site-wide water balance, and the associated cost estimate.
- PHC Inc. completed work related to underground geotechnical design.
- Integrated Sustainability Consultants Ltd. (Integrated Sustainability) developed the water treatment design and the associated cost estimate.

This PEA is preliminary in nature and includes 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, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

2.1 Terms of References

The Technical Report has been prepared in support of disclosures by WRLG in a news release dated 8 July 2025, entitled, *West Red Lake Gold Announces Positive Preliminary Economic Assessment for the Rowan Project, Including Over 35,000 oz. Average Annual Production and 42% After-Tax IRR*. All measurement units used in the Technical Report are metric unless otherwise noted. Currency is expressed in Canadian dollars (C$) unless otherwise noted. The Technical Report uses Canadian English. Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines dated 19 November 2019 (CIM 2019).

2.2 Qualified Persons Responsibilities

The following individuals, by virtue of their education, experience, and professional association, are considered Qualified Persons (QPs) as defined in the NI 43-101 requirements and are members in good standing of appropriate professional institutions.

The QPs have contributed to the writing of the Technical Report and have provided QP certificates, which are included in Section 28, Qualified Person Certificates. The information contained in these certificates outlines the sections in the Technical Report for which each QP is responsible. The QPs have also contributed figures, tables, and portions of Section 1, Summary; Section 2, Introduction; Section 3, Reliance on Other Experts; Section 21, Capital and Operating Cost Estimates; Section 25, Interpretation and Conclusions; Section 26, Recommendations; and Section 27, References. The QP professional designations, employer, and section responsibilities are summarized in Table 2-1.

Table 2-1: Qualified Persons for the National Instrument 43-101 Technical Report

Name	Professional Designation	Title	Employer	Independent of WRLG	Section Responsibilities
Grant Carlson	P.Eng.	Director, Mining	Fuse Advisors Inc.	Yes	1.1, 1.9, 1.11, 1.12, 1.13, 1.14, 1.15, 1.16, 2, 3, 15, 16 (excluding 16.7), 18 (excluding 18.3 and 18.4), 19, 20 (excluding 20.5), 20.5.4, 21, 22, 24, 25.3, 25.5 (excluding 25.5.2), 25.6, 25.7, 25.8, 25.9, 25.10, 25.11.1.3, 25.11.1.4, 25.11.1.5, 25.11.1.6, 25.11.1.7, 25.11.2.2, 25.11.2.4, 26.4, 26.7, 26.8, and 27
John Sims	C.P.G.	President	SIMS Resources LLC	Yes	1.2, 1.3, 1.4, 1.5, 1.6, 1.8, 1.16, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.11.1.1, 25.11.2.1, 26.1, 26.8, and 27
Travis O'Farrell	P.Eng., MBA	Director, Processing and Metallurgy	Fuse Advisors Inc.	Yes	1.7, 1.10, 13, 17, 18.3, 25.1, 25.4, 25.11.1.2, 25.11.2.3, 26.2, 26.5, 26.8, and 27
Paul Hughes	P.Eng., Ph.D.	Geotechnical Engineer	PHC Inc.	Yes	1.9, 16.7, 25.11.1.3, 25.11.2.2, 26.3, 26.8, and 27
Daniel Ruane	P.Eng.	Specialist Engineer	Knight Piésold Ltd.	Yes	1.9, 1.11, 18.4 (excluding 18.4.10), 20.5 (excluding 20.5.4), 25.5.2, 25.11.1.4, 26.6, 26.8, and 27
AJ MacDonald	P.Eng.	Vice President	Integrated Sustainability Consultants Ltd.	Yes	1.11, 18.4.10, 25.11.1.4, 26.6, and 27

2.3 Site Visits

2.3.1 Grant Carlson, P.Eng.

Grant Carlson, P.Eng., did not complete a visit to the Project site.

2.3.2 John Sims, C.P.G.

John Sims, C.P.G., visited the Project site on 20 February 2024. During the site visit, Mr. Sims toured the site, inspected drill core, reviewed the geological interpretation information, and discussed various aspects of mineral resources with the Project site technical team.

Discussions were held with WRLG personnel and its consultants:

- Will Robinson, P.Geo., Vice President, Exploration; and
- Chris Lee, P.Geo., Consultant Geologist, Touchstone Geosciences.

2.3.3 Travis O'Farrell, P.Eng., MBA

Travis O'Farrell, P.Eng., MBA, did not complete a visit to the Project site.

2.3.4 Paul Hughes, P.Eng., Ph.D.

Paul Hughes, P.Eng., Ph.D., visited the Project site from 24 June 2023 to 27 June 2023. The site visit was coordinated by Mr. Ron Felton, WRLG Project Manager for the Rowan deposit exploration program. The purpose of the visit was to assess rock mass characteristics and structural geology in support of the underground mine planning and design. During the site visit, Mr. Hughes conducted a site induction, developed geotechnical logging tools and procedures, trained site personnel, and reviewed and validated geotechnical data collection practices. Mr. Paul Hughes also conducted a review of select 2022 drill core to support the geotechnical characterization of the Rowan deposit.

2.3.5 Daniel Ruane, P.Eng.

Daniel Ruane, P.Eng., did not complete a visit to the Project site.

2.3.6 A.J. MacDonald, P.Eng.

A.J. MacDonald, P.Eng., did not complete a visit to the Project site.

2.4 Sources of Information

Capital and operating cost estimates for the Project was primarily developed using a combination of first-principles engineering and benchmarking against best available information from nearby mine sites. The best available information from nearby mine sites served as the foundational reference document for labour rates, equipment pricing, infrastructure development costs, and processing assumptions; this information was then adjusted to reflect site-specific mine plans, throughputs, and logistics for the Project.

Operating costs, including mining, labour, fuel, power, toll milling, and ore transportation, were developed utilizing vendor quotations and industry-standard benchmarks from best available information from nearby mine sites.

2.5 Effective Date

The Technical Report has the following significant dates:

- Mineral resource statement: 30 June 2025.
- Economic analysis: 30 June 2025.

The effective date of the Technical Report is based on the date of the economic analysis, which is 30 June 2025.

3 RELIANCE ON OTHER EXPERTS

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

- information available to the QPs at the time of preparation of the Technical Report; and
- assumptions, conditions, and qualifications as set forth in the Technical Report.

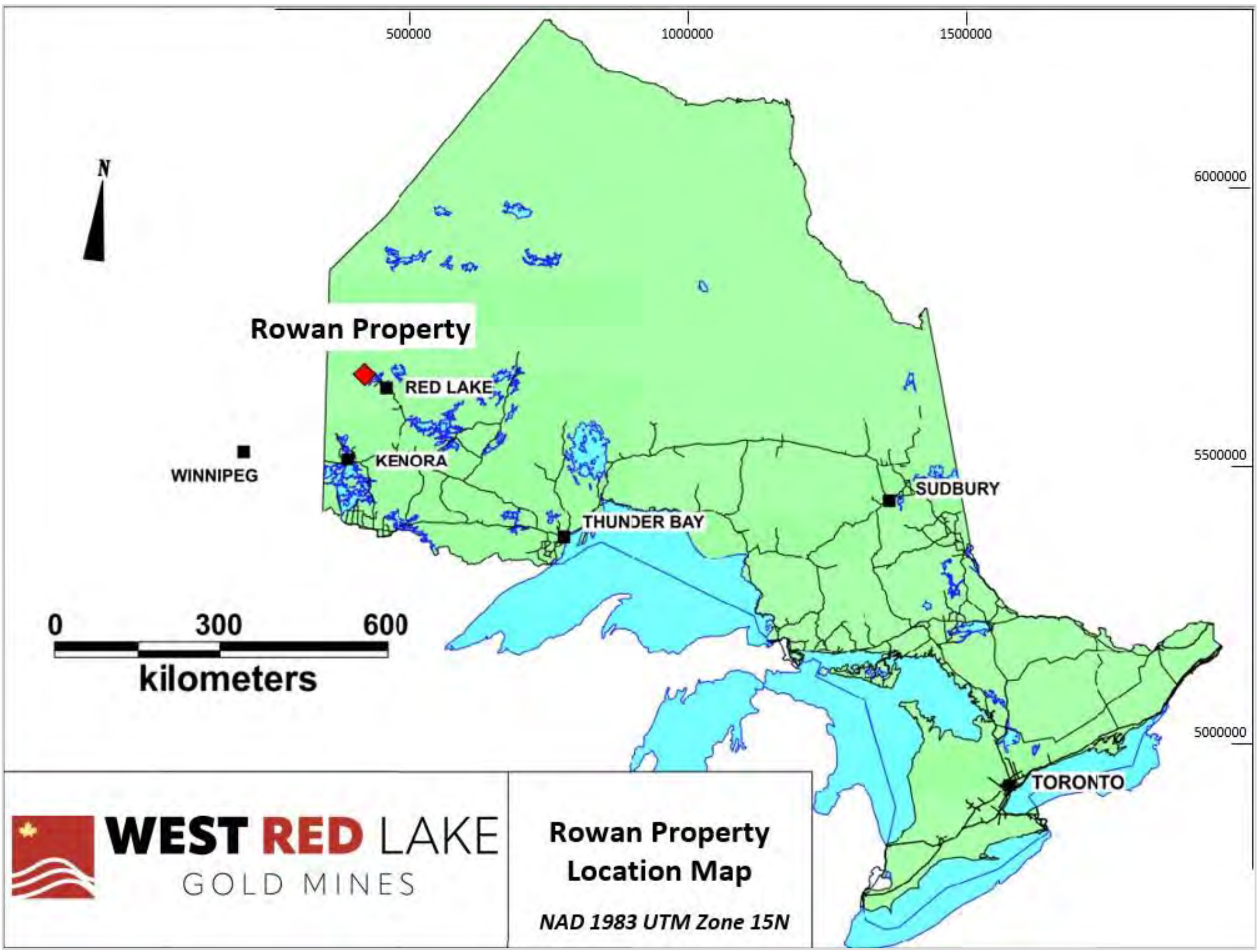
For the purpose of the Technical Report, the QPs have relied on ownership information provided by WRLG. This ownership information was provided in emails and data transfers received in February 2024, and from April 2025 to June 2025. WRLG manages the filing of required assessment work, fee payments required to maintain the unpatented mining claims, and tax payments for the patented claims; as such, the QP believes it is reasonable to rely on WRLG. The QP has not researched property title or mineral rights for the Rowan Property and expresses no opinion as to the ownership status of the Rowan Property.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

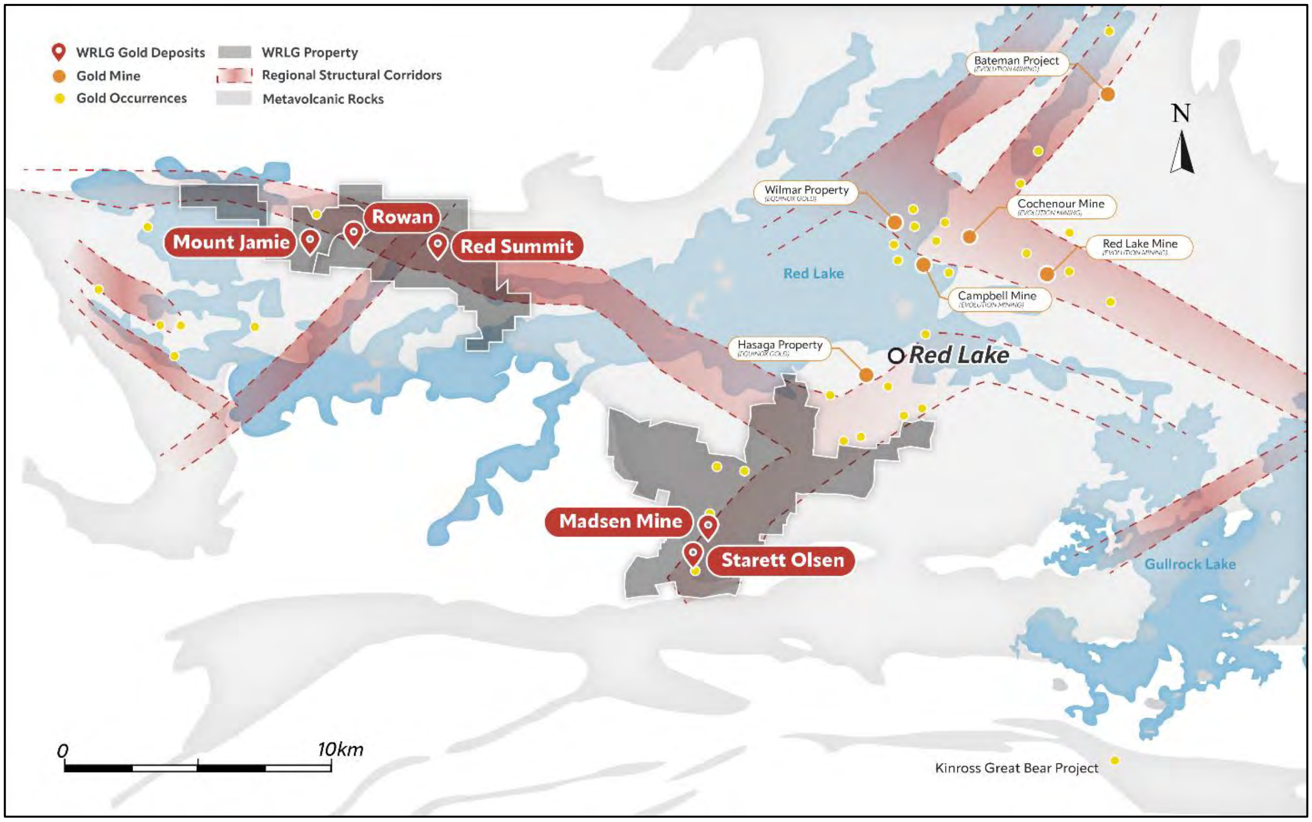
The Rowan Property is located in the Todd, Hammell Lake, and Fairlie townships, Red Lake Mining Division, District of Kenora (Patricia Portion), northwestern Ontario, Canada (Figure 4-1). The Red Lake Gold District is located 16 km northwest of the town of Red Lake, Ontario, 250 km northeast of Winnipeg, Manitoba, and 430 km northwest of Thunder Bay, Ontario. The Rowan Property hosts three past--producing gold mines: the Rowan Mine, Mount Jamie Mine, and Red Summit Mine (Figure 4-2).

Figure 4-1: Rowan Property Location Map



Source: SR 2025.

Figure 4-2: Geological Map of the Red Lake Greenstone Belt Highlighting Gold Producers, Prospects, and Showings



Source: Modified from Durochers et al. 1987.

4.2 Mineral Tenure

The Rowan Property is comprised of 146 mining claims with a total area of approximately 3,100 ha. These 146 claims include 58 patented claims, 20 leased claims, 65 staked crown claims, and 3 licence-of-occupation claims (Table 4-1; Figure 4-3).

WRLG (Ontario), a wholly owned subsidiary of WRLG, owns 100% of all mining leases, patents, and unpatented claims comprising the Rowan Property.

Unpatented mining cell claims confer title to hard-rock mineral tenure only, and claims must be converted to leases before mining can take place. Annual assessment work is completed to maintain unpatented mining claims in good standing. The Rowan Property benefits from exploration credits carried over from previous work; all claims remain in good standing through 3 February 2027.

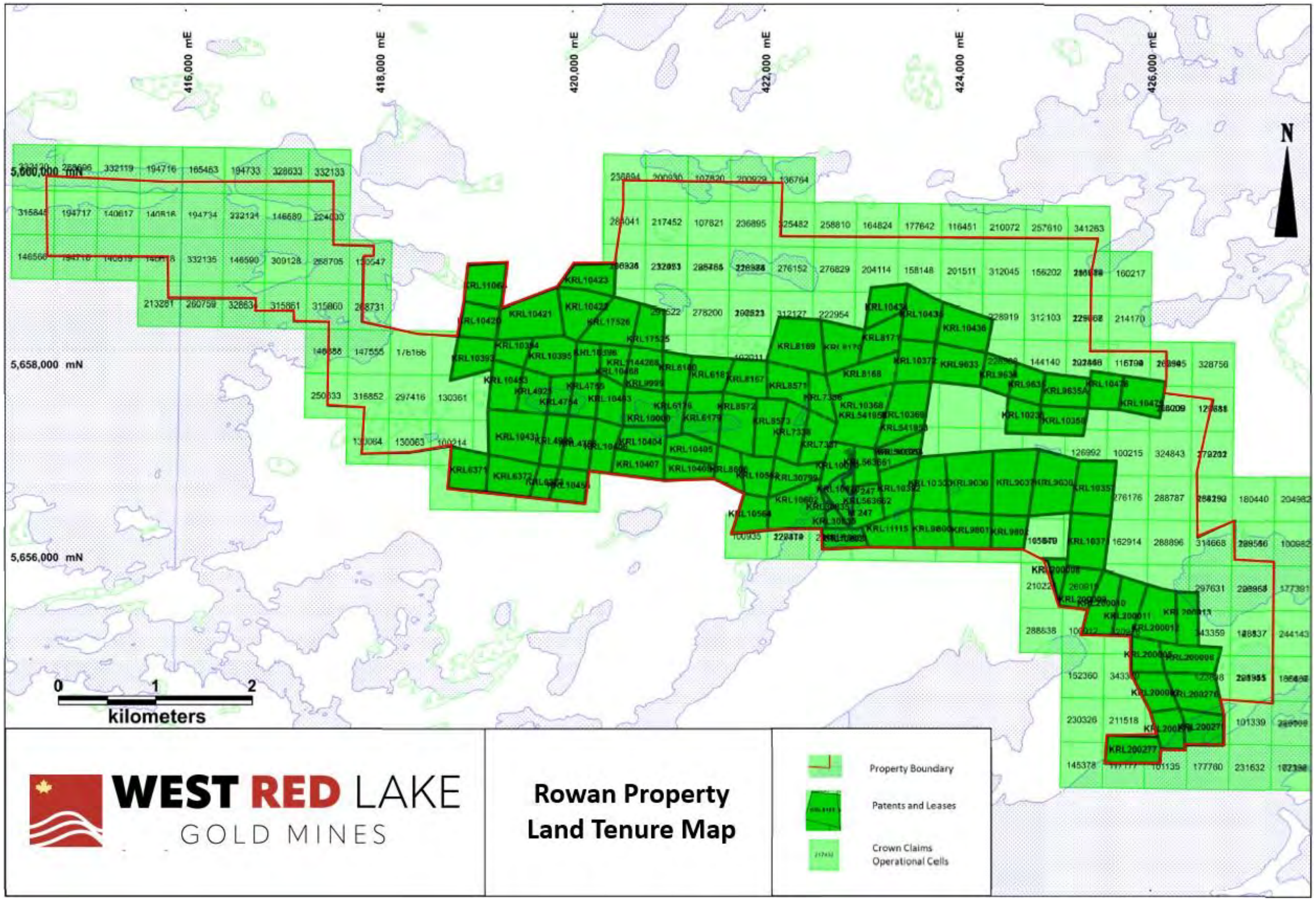
Patented mining claims (i.e., patents) confer fee-simple rights to hard-rock mineral tenure and allow extraction and sale of minerals. Most of the WRLG patents also include the surface rights above the mineral tenure; some easements for municipal services have been granted and a few claims have other surface owners. Patents do not require assessment work but are subject to an annual mining land tax.

Unpatented mining claims can be converted to mining leases, which grant the right to extract and sell minerals for a renewable period of 21 years. Surface rights can be granted with the mining lease if these rights were previously held by the Crown; if surface rights were not previously held of the Crown, an agreement with the surface rights owner must be completed as part of the leasing process. Boundaries of mining leases are defined by legal surveys completed at the time of lease conversion. Leases do not require assessment work but are subject to annual rent.

Table 4-1: Rowan Property Mineral Tenure Claim Summary

Claim No.	No. of Claims	Type	Expiry Date
"KRL-" 6178-6181, 7336-7338, 8167-8171, 8571-8573, 8606, 9633-9638, 9999-10000, 10357, 10371-10372, 10392, 10403-10408, 10434-10435, 10553, 10563, 10564, 10070, 11115, 9800 (27554), 9801 (27555), 9802 (27556), 10603, 30799	47	Patented	Does Not Expire
PT KRL 10070 w/ KRL (10000), KRL 10603 (27553) w/ 10564, KRL 30835 w/ 30799	3	Licence of occupation	Does Not Expire; Payment Every 2 Years
Lease# 109017 -- KRL 541952-541954, KRL 563661-563662; Lease # 107258 -- KRL 200005-200013, KRL 200276-200279	18	Leased	28 February 2033
541924-541951, 563036, 563666-563669, 563946-563950, 623493, 1144316, 1184146, 1184861-1184863, 1218922, 1218923, 1234138, 1234139, 1234151	49	Crown - Staked	3 February 2027
KRL 10235, KRL 10358	2	Patented	Does Not Expire
"KRL-" 10393-10396, 10420-10423, 11064	9	Patented	Does Not Expire
1184167, 1144269, 1184115, 1144277	4	Crown - Staked	31 December 2028
Lease #107316 -- KRL10468, 1144268	2	Leased	31 July 2042
1234187-1234192	6	Crown - Staked	27 September 2028
1234519, 1234522, 1234524, 1234534	4	Crown - Staked	9 November 2028
3017000, 3017001	2	Crown - Staked	31 December 2028
Total	**146**		

Figure 4-3: Rowan Property Claim Map



Source: SR 2025.

4.3 Royalties

A summary of royalty agreements, including mineral claims and royalty holders, is provided in Table 4-2 and shown on Figure 4-4; the are no other known royalties, back-in rights, payments, or other agreements and encumbrances related to the Rowan Property.

On 3 March 2023, WRLG and its wholly owned subsidiary WRLG (Ontario) completed the purchase of Evolution Mining's remaining interest in certain claims on the Rowan Property increasing WRLG ownership of those claims to 100% (i.e., the Purchase Agreement). WRLG paid $250,000, issued 3,645,000 shares and WRLG (Ontario) granted a 2.5% Net Smelter Return (NSR) royalty to Evolution Mining Gold Operations Ltd., a subsidiary of Evolution Mining, on certain claims on the Rowan Property. WRLG also issued an aggregate of 182,250 success fee shares to certain third parties in connection with the Purchase Agreement.

The Rowan Property also includes two patented Red Summit Mine claims in east central Todd Township that are surrounded by the Rowan Property and contain both mineral and surface rights. The prior owner West Red Lake Gold Mines Inc. (RLG), formerly known as Hy Lake Gold Inc. (Hy Lake), acquired a 100% ownership in the claims in 2009 pursuant to an option agreement with Claude Resources Inc. (Claude) dated 27 February 2008 after a cash payment of $25,000 and $100,000 of exploration expenditures. These two patented claims are subject to a 3% NSR, of which 1% is buyable by WRLG (Ontario) for $500,000. The claims are not subject to any back-in rights.

On 12 December 2005, RLG entered into an option agreement to acquire a 75% interest in nine patented mining claims containing mineral and surface rights from Jamie Frontier Resources Inc. (Jamie Frontier) for $80,000 in cash, 550,000 common shares of RLG, exploration work totalling $1 million, and a 3% NSR. On 11 April 2007, RLG completed the acquisition of the remaining 25% interest in these nine patented claims from Gsont Holdings Limited for 2,000,000 common shares of RLG, and RLG became a 100% owner of mineral and surface rights for these nine patented mining claims. Jamie Frontier has a 3% NSR on these nine claims.

On 5 March 2007, RLG entered into an option agreement with Martin Bobinski and Antony Maciejewski to earn a 100% interest in four staked claims and two leased claims containing mineral rights only, which are contiguous to the east of the nine patented mining claims discussed above. Total consideration for these six claims consisted of cash payments of $70,000, the issuance of 200,000 common shares of RLG, and a commitment to carry out exploration work totalling $140,000, or cash/shares in lieu thereof, over a four-year period. In February 2012, having met all of the requirements under the option agreement, RLG exercised its option and became a 100% owner of these six mining claims. These six claims are subject to a 3% NSR (2% of which can be repurchased for $1 million per 1%), an annual advance royalty in the amount of $10,000, and a one-time payment of $500,000 that was due on WRLG completing a bankable feasibility study.

On 11 October 2007, RLG entered into an option agreement with Martin Bobinski and Antony Maciejewski to acquire a 100% interest in six staked claims containing mineral rights only, which are contiguous to the west with the nine patented claims discussed above. RLG issued 150,000 common shares as consideration and became a 100% owner of the claims, which are subject to a 3% NSR (2% of which can be repurchased for $1 million per 1%).

On 20 February 2008, RLG entered into an option agreement with Rubicon Minerals Corporation (now known as Evolution Mining Gold Operations Ltd., a subsidiary of Evolution Mining) to earn a 100% interest in four staked claims containing minerals rights only, which are contiguous to the south of the above listed claims. The aggregate purchase price consisted of cash payments of $50,000 and the issuance of 75,000 common shares. As of the effective date of this Technical Report, WRLG holds a 100% interest in the four claims, which are subject to a 2% NSR (1% of which can be repurchased for 1% and WRLG has a right of first refusal on the other 1%).

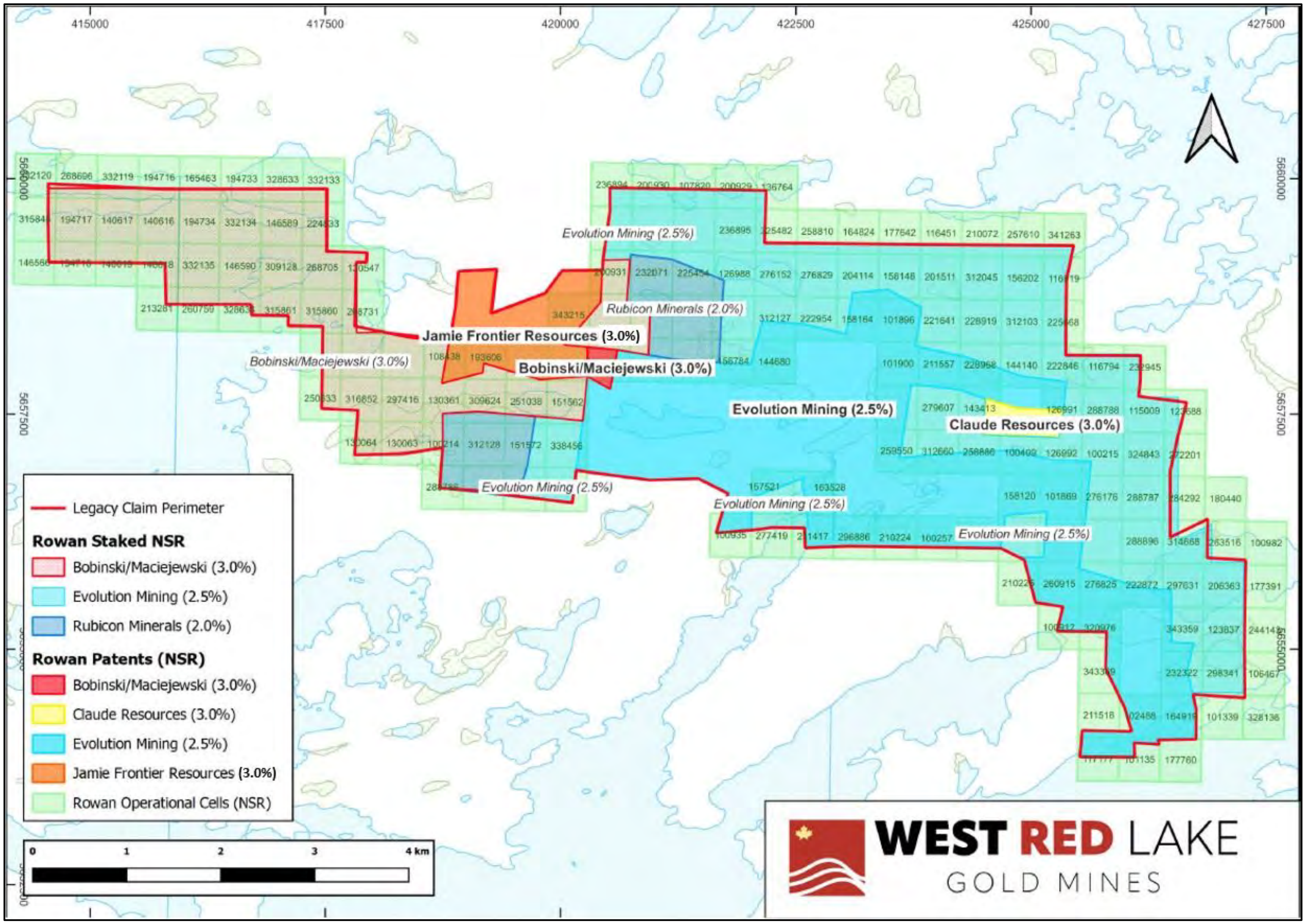
On 24 November 2010, RLG entered into an option agreement with Perry English on behalf of Rubicon Minerals Corporation to earn a 100% interest in two staked claims containing mineral rights only, which are contiguous to the east of the above-mentioned claims. The aggregate purchase price for these two staked claims consisted of cash payments of $125,000 and the issuance of 100,000 common shares of RLG over a four-year period. On 24 November 2014, after the payment of $85,000 cash and 100,000 common shares, the parties amended the option agreement to change the remaining cash commitment of $40,000 for the year ending 30 September 2015 into two payments with each consisting of $11,000 and 250,000 common shares on 31 December 2014 and 31 December 2015, respectively. WRLG (Ontario) currently holds a 100% interest in these two staked claims, which are subject to a 2% NSR (1% of which can be repurchased for $1 million).

Table 4-2: Royalty Agreements Summary on the Rowan Property

Claim No.	No. of Claims	Type	Royalty Holder	Royalty
"KRL-" 6178-6181, 7336-7338, 8167-8171, 8571-8573, 8606, 9633-9638, 9999-10000, 10357, 10371-10372, 10392, 10403-10408, 10434-10435, 10553, 10563, 10564, 10070, 11115, 9800 (27554), 9801 (27555), 9802 (27556), 10603, 30799	47	Patented	Evolution Mining Gold Operations Ltd.	2.5% NSR
PT KRL 10070 w/ KRL (10000), KRL 10603 (27553) w/ 10564, KRL 30835 w/ 30799	3	Licence of occupation		
Lease# 109017 -- KRL 541952-541954, KRL 563661-563662; Lease # 107258 -- KRL 200005-200013, KRL 200276-200279	18	Leased		
541924-541951, 563036, 563666-563669, 563946-563950, 623493, 1144316, 1184146, 1184861-1184863, 1218922, 1218923, 1234138, 1234139, 1234151	49	Crown - Staked		
KRL 10235, KRL 10358	2	Patented	Claude Resources Inc.	3% NSR, 1% purchasable for $500,000
"KRL-" 10393-10396, 10420-10423, 11064	9	Patented	Jamie Frontier Resources Inc.	3% NSR
1184167, 1144269, 1184115, 1144277	4	Crown - Staked	Bobinski & Maciejewski	3% NSR, 2% purchasable for $1M each, annual pre-production royalty of $10,000
Lease# 107316 -- KRL10468, 1144268	2	Leased		
1234187-1234192	6	Crown - Staked		
1234519, 1234522, 1234524, 1234534	4	Crown - Staked	Evolution Mining Gold Operations Ltd.	2% NSR, 1% purchasable for C$1M
3017000, 3017001	2	Crown - Staked		

Notes:
NSR = Net Smelter Return; C$M = million Canadian dollars.

Figure 4-4: Rowan Property Royalty Map



Source: SR 2025.

Note: Evolution Mining refers to its subsidiary Evolution Mining Gold Operations Ltd.

4.4 Surface Rights and Other Rights

The surface rights ownership for the Rowan Property claims, patents, and leases are provided in Table 4-3. Where WRLG does not hold surface rights, these claims are predominantly held by the Crown, as administered by the Province of Ontario. Timber rights are reserved to the Crown, and water rights are held for the public use. The QP is not aware of other conferred rights on the Rowan Property.

Table 4-3: Surface Rights Ownership for the Rowan Property

Claim No.	No. of Claims	Type	Surface Rights Owner
"KRL-" 6178-6181, 7336-7338, 8167-8171, 8571-8573, 8606, 9633-9638, 9999-10000, 10357, 10371-10372, 10392, 10403-10408, 10434-10435, 10553, 10563, 10564, 10070, 11115, 9800 (27554), 9801 (27555), 9802 (27556), 10603, 30799	47	Patented	WRLG
PT KRL 10070 w/ KRL (10000), KRL 10603 (27553) w/ 10564, KRL 30835 w/ 30799	3	Licence of occupation	WRLG
Lease# 109017 -- KRL 541952-541954, KRL 563661-563662; Lease # 107258 -- KRL 200005-200013, KRL 200276-200279	18	Leased	WRLG
KRL 10235, KRL 10358 (RED SUMMIT)	2	Patented	WRLG
"KRL-" 10393-10396, 10420-10423, 11064	9	Patented	WRLG
Lease# 107316 -- KRL10468, 1144268	2	Leased	WRLG
KRL10436	1	Patented	STEPHENS, CALVIN PIZANO, LINDA
KRL10478; KRL10479	2	Patented	CARLSON, GENE KENT
KRL10468; KRL17525; KRL17526	3	Patented	BOBINSKI, MARTIN JOHN MACIEJEWSKI, ANTONY JAMES
KRL4755	1	Patented	BOBINSKI, MARTIN JOHN MACIEJEWSKI, ANTONY JAMES
KRL10368; KRL10369; KRL10370	3	Patented	STEPHENS, CALVIN
KRL4921A TODD (RECORDED AS KRL10453); KRL4921; KRL6371; KRL6372; KRL10431	5	Patented	KEATING, GERALD FRANCIS KEATING, JULIE ANNE
KRL4754; KRL4756 (RECORDED AS KRL10454); KRL4919 (RECORDED AS KRL10456); KRL4920 (RECORDED AS KRL10455); KRL6373	5	Patented	KEATING, GERALD FRANCIS KEATING, JULIE ANNE

Notes:
WRLG = West Red Lake Gold Mines Ltd.

4.5 Environmental Liabilities, Permitting, and Other Factors and Risks

The QP is not aware of any environmental liabilities on the Rowan Property. WRLG (Ontario) holds all required permits to perform work on the Rowan Property. The QP is not aware of any other significant factors or risks that may affect access, title, or the right or ability to perform work on the Rowan Property.

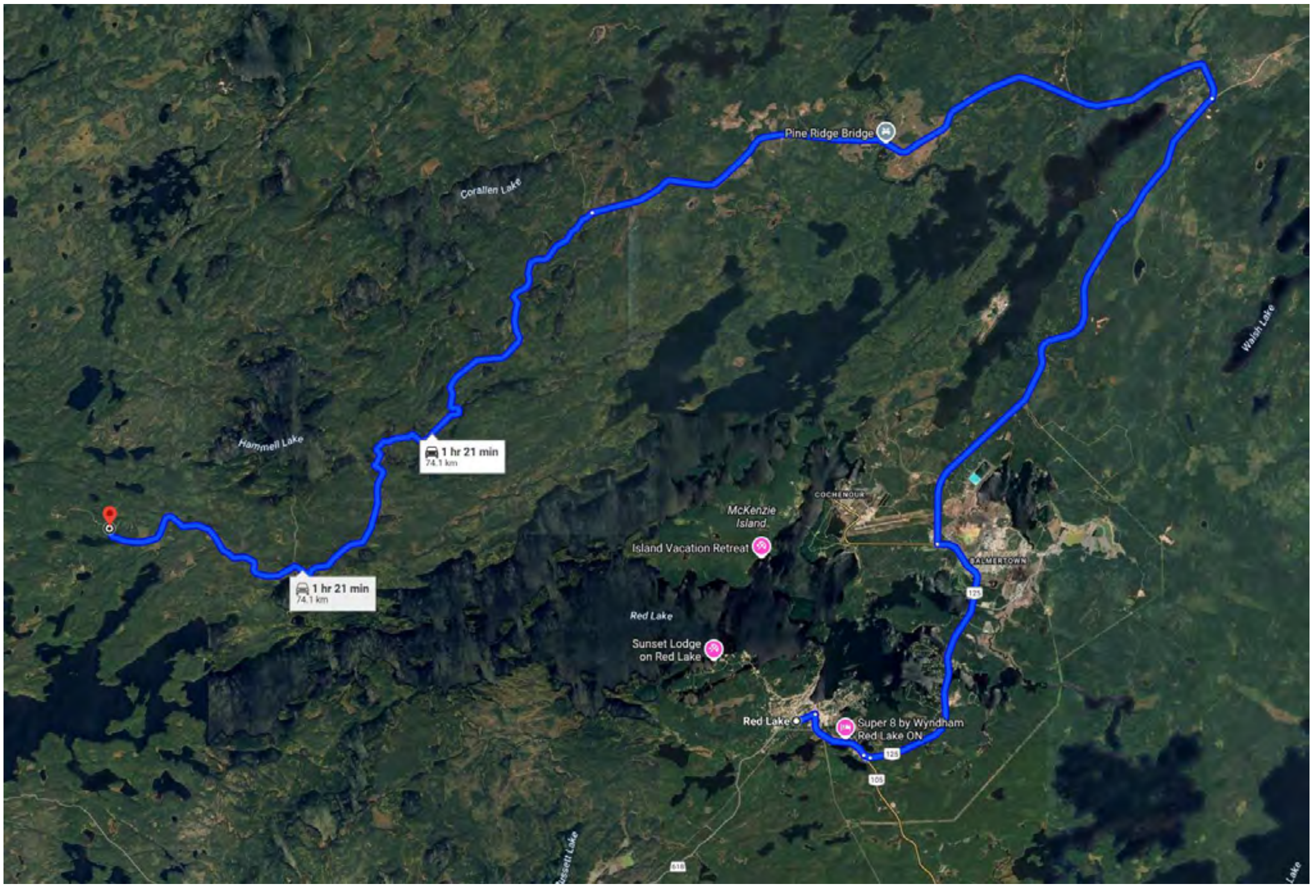
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located in northwestern Ontario, approximately 16 km northwest of the town of Red Lake. Access to the Project will be provided by existing highway and all-season gravel roads that connect to Balmertown, Ontario. Vehicles will travel north from Balmertown on Nungessor Road for 16 km, west on Pine Ridge Forest Access Road for 22 km, and then south on Mount Jamie Mine Road for 27 km (Figure 5-1). These existing roads are in good condition and are currently being used for exploration. While no power lines or telecommunications are installed along this route, these existing roads are is suitable for Project use, including the transportation of personnel, equipment, consumables, and ore.

The southern portion of the Rowan Property can also be accessed by water from Red Lake, Ontario by either via Martin Bay or by travelling north up the Golden Arm of Red Lake. Alternatively, the Rowan Property can be accessed from a dock on the eastern shore of Pipestone Bay along the Mount Jamie Mine Road.

Figure 5-1: Road Access to the Rowan Project from Red Lake, Ontario



5.2 Climate

The closest climate station to the Project site is located in Red Lake, Ontario. Average temperatures in the area of the Project normally range from a low in February of -19.6˚C to a high in July of between 18.1˚C and 23.3˚C. The average annual precipitation is 640.2 mm, with the expected minimum precipitation of 18.6 mm in February, and the expected maximum of 97.7 mm in June. At the Red Lake climate station, monthly wind speeds for the area are generally stable throughout the year, averaging approximately 9.4 km/h to 12.8 km/h.

5.3 Local Resources

The Municipality of Red Lake includes a population of 4,094 (Statistics Canada 2021 Census) and is comprised of six communities: Red Lake, Balmertown, Cochenour, Madsen, McKenzie Island, and Starratt-Olsen. Mining and mineral exploration are the primary industries in the area of the Project. Other industries include logging and tourism. The Municipality of Red Lake offers a full range of services and supplies for mineral exploration and mining, including both skilled and unskilled labour, bulk fuels, freight, heavy equipment, groceries, hardware, and mining supplies. The majority of the Rowan Property support staff live in the surrounding communities and out of town employees stay in local accommodations in Red Lake.

5.4 Infrastructure

Existing infrastructure at the Rowan Property includes an all-season trailer camp located in the Mount Jamie shaft area, which is currently used to support exploration and environmental baseline activities.

The historical Rowan Mine shaft is located on a hill at approximately 400 metres above seal level (masl) at coordinates 421602 E, 5657811 N (North American Datum [NAD] 83 UTM Zone 15N) and is located approximately 115 m west of the Portal for the Project. The historical headframe has been dismantled and capped with a concrete block. The entrance to the historical Rowan Mine adit, which is currently barricaded with broken muck, is located approximately 35 m southwest of the Portal for the Project.

Water is available for use in industrial quantities from either Rowan Lake, which is centrally located on the Rowan Property, or from Red Lake, which is located in the western portion of the Rowan Property.

5.5 Physical Geography

The Rowan Property is located at an average elevation of 370 masl and is part of the Lac Seul Upland, which extends eastward from Lake Winnipeg in Manitoba to the Albany River in northwestern Ontario. Topography at the Rowan Property is mostly small rolling hills or higher outcrop ridges, with high points extending approximately 45 m above their surroundings. Lower elevation areas of the Rowan Property are lakes and swamps, and the southeast portion of the Rowan Property is covered by relatively flat sand plain.

The physical geography for the Project includes a diverse range of environments from mature mixed forest to alder swamps that are typical within Archean terrains. The dominant land cover is coniferous forest with some limited areas of mixed forest. Characteristic vegetation includes white spruce, balsam fir, and black spruce with some trembling aspen and balsam poplar, although jack pine and black spruce are more common on moderately well to poorly drained areas. Poorly drained areas are covered by fens and bogs and are dominated by black spruce.

6 HISTORY

The information in this section has been modified from Kita (2022). Historical work at the Rowan Property has focused on the three main areas surrounding the past-producing Rowan Mine, Mount Jamie Mine, and Red Summit Mine.

6.1 Rowan Mine

6.1.1 Rowan Mine Ownership, Exploration, and Development History

Several companies have worked the Rowan Mine area claim group since the 1928 discovery of gold on “Discovery Hill” by the Rowan Hall Syndicate.

Surface trenching completed on Discovery Hill in the early 1930s identified the Rowan Vein System. From 1936 to 1939, an adit was driven along the Rowan vein from the base of Discovery Hill, followed by the sinking of a 425-foot (ft) shaft and three levels of development that were supported by previous surface and underground drilling results. From 1945 to 1947, drilling was completed near Rowan Lake. Underground work recommenced in 1953, with further development of the third level to the east. In 1958, additional drilling was completed to extend the Rowan Vein System over the strike length, however, work was discontinued after 1958. Development muck was stockpiled and later custom milled by Dickenson Mines Limited (Dickenson Mines) in the 1980s.

Goldquest Exploration Inc. (Goldquest), which is part of the Dickenson Group of Companies, completed systematic grassroots exploration over the Rowan Mine area from 1981 to 1988 as well as a bulk mining test of the Rowan Vein System above the adit level. A test shrinkage-stoping operation mined 2,600 tons of ore and 2,482 tons of this ore was milled at Dickenson Mines, which returned 610 ounces of gold (0.25 ounce per short ton [oz/ton] Au over 2.9 ft). Based on this work, Dickenson Mines conducted a feasibility study on the Rowan Vein System; the project was considered marginally profitable at that time.

Chevron Minerals Ltd. (Chevron), in a joint venture agreement with Goldquest, completed drilling at 100 m spacing over the Rowan Vein System attempting to expand zones of mineralization. Holes were also drilled at Martin Bay and along the Rowan Creek zone. No significant results were obtained from this drill program, and the option was terminated.

Goldquest, and later Goldcorp Inc. (Goldcorp) after their amalgamation in 1994, completed assessment drilling to test a major fold structure east of Rowan Lake in 1993 and 1997. In 2001, Goldcorp conducted infill drilling between the previous Chevron drill holes. In 2002, Goldcorp completed an induced polarization (IP) survey near Martin Bay over the areas reported by previous operators to contain wide zones with volcanogenic massive sulphide potential.

King's Bay Gold Corporation (King’s Bay) optioned the Rowan Property from Goldcorp in 2006 and completed a drilling program to test geological and geophysical anomalies in the Rowan Mine shaft area and Porphyry Hill area, with the best results obtained from the northeast shaft area. A total of 23 holes with an aggregate length of 4,846 m were drilled.

Over 2005 to 2009, Hy Lake Gold Inc. (Hy Lake) entered into a number of option agreements to acquire claims that constitute the present claims covering the Rowan Mine area. In 2012, Hy Lake changed its name to RLG.

Between 2007 and 2021, Hy Lake/RLG completed approximately 40,000 m of diamond drilling at the Rowan Mine area and Newman-Todd (NT) Zone.

The location of the Rowan Mine is shown on Figure 4-2, and additional details of the exploration and mining history of the Rowan Mine area are provided in Table 6-1.

Table 6-1: Exploration and Mining History of Rowan Mine Area

Phase/Company	Year	Activity
Discovery of gold	1928	Gold was discovered on "Discovery Hill" (near shaft) by the Rowan-Hall Syndicate. Several narrow gold-bearing quartz veins were exposed and identified as veins A-D at surface.
		Ownership dispute and litigation until 1934.
Paulore Gold Mines Ltd. (Paulore)	1934	Paulore conducted prospecting, trenching, and drilling of six holes in the Martin Bay area. A significant east-west surface shear zone was discovered. Test pits reported a 4 ft to 7 ft wide zone in sheared diorite.
		ODM Vol. XLIV pt 6 reported quartz veins with arsenopyrite and abundant visible gold.
Lake Rowan Gold Mines Ltd. (Lake Rowan Mines)	1936	Lake Rowan Mines drilled S-series holes 37-1 to 9, 17, and 18 in the Discovery Hill area. The locations for holes S-10 to 16 are uncertain and not plotted on any maps. In 1937, the adit was started followed by shaft sinking and development on three levels. Underground holes 37-19 to 37-31 (416.4 m).
		Financial problems. Mine grid was established using the shaft as 5000E, 5000N.
World War II	1939	World War II results in a work disruption.
		Forest fire destroys headframe and surface installations.
West Red Lake Gold Mines (West Red Lake)	1940	West Red Lake - McKenzie Option (West Red Lake Zone). Trenching, sampling, mapping, and drilling of M-series holes 1-18 (927 m). Groups 2, 3, and 4 in the current Rowan Mine area.
Rugged Red Lake Mines (Rugged Red Lake) and Rowan Consolidated Mines Ltd. (RCM)	1945	Rugged Red Lake. Mapping, trenching, and 25 drill holes (total length 4,746 m). Scheelite found in trenches in the Martin Bay area.
		Lake Rowan (1945) Mines. Mapping, 56 surface drill holes RW-46-1 to RW-47-56 and discovery of the Shaft Extension, Creek, and 10000 zones. Mineralization was found in iron formation on Porphyry Hill. Mine grid re-established using Post # 3 of KRL 10000 as 5000E, 5000N, and a 5,000 ft elevation.
	1950	RCM established; site rehabilitation. From 1953, an underground program continues drifting to the east on level 3 of the Rowan Vein System to test drill intersections obtained in 1946.
		Additional U-series underground drilling occurred while drifting.
		Intermittent work because of financial difficulties.
		Drilled eight surface x-ray holes due south in 1950 but locations are not certain.
	1952	Rugged Red Lake. Grades up to 12.8% Zn, 2.48% Pb, 1.15% Cu, 0.08 oz/ton Au, and 14.3 oz/ton Ag reported from surface showings near Martin Bay. Unsubstantiated. OFR 5958.
	1958	RCM resumed work. Seven drill holes RW 58 100-106 (total length 1,340.5 m) to test the eastern and western extensions of the Rowan main vein.
Cochenour Exploration Ltd. (Cochenour)	1969	Work on the "Rugged Group" near Martin Bay. Mapping, soil geochemistry, magnetics, horizontal loop electromagnetics (HLEM). Follow-up with eight drill holes (total length 597 m) to test west-southwest to east-northeast EM conductors. Drilling intersected dominantly mafic flows with intercalated cherts, magnetite bearing iron-formation, scattered pyrrhotite and chalcopyrite in holes MB 69 1-8 over claims KRL 63669 and 63670. All assays trace gold except in MB 69 4 returning 0.06 oz/ton Au in volcanics with <1% sphalerite, arsenopyrite, pyrite, and chalcopyrite.
	1971	Ontario Geological Survey (OGS) mapping of Todd and Fairlie townships by R.A. Riley. Maps 2406 and 2407. Cochenour completes magnetics and HLEM surveys near Martin Bay. EM-17 conductors K, Q, and R targeted for drilling. The area may have base metal potential.
Goldquest Exploration Inc. (Goldquest), part of the Dickenson Group of Companies	1981	Goldquest acquires a large land package around the A.W White and Campbell Red Lake mines that includes the Rowan Mine area. Additional claim staking of Block 10B. Transport 17,817.6 tons of Rowan stockpiled material to the A.W. White Mine at a cost of $14/ton. P.J. Vamos evaluation report recommending follow-up on 1) Shaft zone, 2) Creek zone and 3) Forgotten zone?
	1982	Goldquest – HLEM and magnetics on a cut grid.

Table 6-1: Exploration and Mining History of Rowan Mine Area (continued)

Phase/Company	Year	Activity
	1983	Goldquest conducts geological mapping (1:2500), radiometrics, and lithogeochemistry. Dozer stripping of the DLS Carbonate, Main Vein (1:100), and Headache zones (1:100).
	1984	Winter drill program (total length 3,622.76 m), 16 holes RW 84-57-66, 68-73. Dozer stripping and sampling at Martin Bay. Bulk mining test of a quartz vein above the adit level, with 2,482 tons later milled (in 1988) to recover 610 oz of gold. Mine sealed and flooded below the adit level. A portion of Rowan 1946-drill core was salvaged and stored on the Rowan Property.
	1985	Drill program (total length 4,539.45 m) consisting of 51 holes - RW 85 67, 74-91, 91A, 92-99, 107-123, 127-132 (Titley Lake unconformity test). Stripping, pumping, detailed mapping, and sampling at Martin Bay completed by July.
	1986	Milling of 10,541 tons Rowan Consolidated material producing 688 oz of gold (0.07 oz/ton Au). Not clear what proportion of the material was ore-grade. Forest fire in May and June over portions of the Rowan Property. Strathcona Mineral Services review of the Rowan Project.
	1987	Goldquest drills eight holes (total length 1,822.1 m) - RW 87 124-126, 133-137. Dickenson Mines Limited evaluation of the Rowan Prospect by Frank Godfrey. Road access to the Rowan Property from the Pine Ridge Forest Access Road completed.
	1988	Report on the Rowan property for United Reef Petroleum Limited by J. Siriunas. Milling of Rowan stockpile at DML FB-MR. Net to Goldquest 562.184 oz from 2,431.75 tons, with 35 tons remaining according to DML memo.
Chevron Minerals Ltd. (Chevron)	1989	Chevron JV with Goldquest. Compilation of drill data, drilling of holes RW-89 138-144 plus one deepened hole (RW-84-59), dozer stripping, reconnaissance mapping, and lithogeochemistry. Work tested the Rowan Vein System, Porphyry Hill, and Martin Bay areas. Relogging of various drill holes, including RW 58 102-106, and a photo mosaic study of the Rowan Property (Much of this work was not found in the Toronto office). Bruce Wilson did a structural study as presumably a government report. Goldquest Project Evaluation and Development Strategy by H. H. Wober.
	1990	Additional drilling by Chevron of holes RW-90 145-151. Chevron drops options because of corporate decision to abandon mineral exploration. Mineral inventory for the Rowan Vein System was estimated by Fumerton (1990) to be 160,000 tonnes at a grade of 14 g/t Au.
Goldquest	1993	Goldquest assessment drilling – three holes RW 93 152-154. Testing the fold closure east of the Rowan Mine shaft.
Goldcorp, Inc. (Goldcorp)	1994	Goldquest amalgamates with Goldcorp.
	1997	Goldcorp assessment drilling of two holes RW 97 155-156 (total length 995.26 m). Test fold closure. Fold closure interpreted by D.L. Sannes.
	2000	Goldcorp helicopter magnetics, EM, very low frequency (VLF), and radiometrics.
	2001	Goldcorp drills eight holes RW-01 157-164 (total length 1,974 m) to test the Martin Bay area. Follow-up of previous drilling, geophysics, and surface work. Goldcorp completes drilling on the QP zone near the Rowan Mine shaft with four holes RW 01 165-168 (total length 1,699 m). A total of 1,738 mobile metal ion (MMI) samples were taken over block 10A, B, and K. New north-south grid was re-cut over these areas. Geological mapping (scale = 1:2,500) over claim 1234151 (block 10M).
	2002	Goldcorp cuts a new grid near Martin Bay over the work area conducted by Cochenour in 1969. An IP gradient survey was completed testing the area's base metal potential.
Kings Bay	2006	Kings Bay drilled 23 holes, RW-06-101 to 129 (total length 4,856 m) from June to October 2006. The option was dropped. J. Archibald summarized the work performed in a report entitled, Diamond Drilling Report on the Rowan Lake Property for Kings Bay Corporation Ltd., dated 22 November 2006.
Hy Lake Gold Inc. (Hy Lake)	2007-2012	Entered into a number of option and purchase agreements, including with Kings Bay Red Lake Gold Mines (RLGM; a partnership of Goldcorp Inc. and Goldcorp Canada Ltd.), Martin Bobinski and Antony Maciejewski, and Rubicon Minerals Corporation (Rubicon). Completes over 8,000 m of diamond drilling at the Rowan Mine and approximately 5,000 m of diamond drilling at the NT Zone.
West Red Lake Gold Mines Inc. (RLG)	2012	Hy Lake changes its name to West Red Lake Gold Mines Inc. (RLG).
	2013 to 2021	Completes approximately 26,500 m of diamond drilling at the Rowan Mine and NT Zone.
	2020	A 100 line km AeroVision Canada drone magnetometer program over an area covering the 2 km long northeast striking NT Zone. The drone magnetometer program covered 4.52 km^2 and consisted of 68 lines spaced 50 m apart with readings recorded at 1.2 m intervals along each line.

Table 6-1: Exploration and Mining History of Rowan Mine Area (continued)

Phase/Company	Year	Activity
	2021 to 2022	Channel sampling program over a 200 m strike length at the Rowan Mine area along the east-west strike to investigate the potential for a surface bulk sample. A total of 97 samples were collected in 2021, and 182 additional samples were collected in 2022.
	2022	RLG acquired by WRLG.
West Red Lake Gold Mines Ltd. (WRLG)	2023	RLG changed name to West Red Lake Gold Mines Ltd. and completed 62 holes for 20,211.4 m of drilling at the Rowan Mine target.

Source: Archibald et al. 2016.
Notes:
ft = foot; m = metre; Zn = zinc; Pb = lead; Au = gold; Ag = silver; oz = ounce; < = less than; % = percent; km = kilometre; km^2 = square kilometres.

6.1.2 Rowan Mine Deposit Historical Resources

A number of historical resource and reserve estimates have been prepared for the Rowan Mine deposit in the past. All historical and previous estimates are superseded by the mineral resource estimate provided in Section 14, Mineral Resource Estimates.

6.2 Mount Jamie Mine

6.2.1 Mount Jamie Mine Ownership, Exploration, and Development History

It is reported that the discovery of gold on the Mount Jamie Mine in the area of Shaft No. 1 dates back to 1920. Eleven claims were patented in 1928. The completion of any substantial work in the mine area would have required those claims to be filed with the Ontario Bureau of Mines, however, no information regarding ownership or work history of the claims prior to 1934 is available.

Since 1934, various companies have owned and operated the Mount Jamie Mine, with work generally completed on two of the three veins known in the mine area. The work included surface and underground drilling, development of two shafts (i.e., Shaft No. 1 and Shaft No. 2), and construction of a mill. The historical records of actual mining (i.e., ore hoisted to surface) are limited due to poor record keeping. The two shafts were sunk on each of the two veins between 1935 and 1942. Shaft No. 1 reached a depth of 772 ft, with 3,200 ft of lateral development and 630 ft of raising on four levels. Shaft No. 2 was sunk to a depth of 559 ft, with some lateral development on the first level. Approximately 2,000 tons to 3,000 tons of material were mined, with some material treated at an on-site 100-ton/day mill and the rest of the material stockpiled.

In 2005, the Mount Jamie Mine claims were acquired by Hy Lake (renamed RLG in 2012), which between 2011 and 2017 completed a total of approximately 11,000 m of diamond drilling at the Mount Jamie Mine.

The location of the Mount Jamie Mine is shown on Figure 4-2, and additional details of the ownership, exploration, and development history are included in Table 6-2.

Table 6-2: Exploration and Mining History of Mount Jamie Mine Area

Phase/Company	Year	Activity
Discovery of gold	1920s	Gold discovered in 1920. Eleven claims patented in 1928. No record of work history.
Frontier Red Lake Gold Mines Ltd.	1934	Acquired the claims and completed trenching on Vein 1 that reportedly assayed 0.42 oz/ton Au over a width of 50 in., for a length of 120 ft. Subsequently drilled 24 holes for a total length of 6,545 ft.
	1936	Sank a shaft (i.e., Shaft No. 1) to a depth of 244 ft. It had stations at 130 ft and 230 ft, with approximately 155 ft of drifting at the top level and 50 ft of drifting at the 230 ft level. In December 1936, operations halted.
Gold Frontier Mines Ltd.	1939	Was incorporated and took over the mine.
	1940 to 1942	Dewatered Shaft No. 1 and resumed underground work. The shaft was deepened to 500 ft and increased to three compartments (this work was completed by 1942). The lateral development amounted to 2,881 ft, in addition to 630 ft of raising on 130 ft, 230 ft, 350 ft, and 475 ft levels. Work was then halted in Shaft No. 1 in favour of sinking a second shaft (i.e., Shaft No. 2) on a vein that had been discovered in 1941 (referred to at that time as the North Vein).
	1942	Shaft No. 2 was located approximately 2,550 ft northwest of Shaft No. 1 and was sunk to a depth of 559 ft. Some lateral development was completed at the 100 ft elevation. In August 1942, a government mandate terminated all work at non-productive gold mines, bringing the activity at the mine to a halt.
Bayview Red Lake Gold Mines Ltd.	1944 to 1947	Acquired the mine and deepened the Shaft No. 1 to 772 ft, with stations developed at the 625 ft and 750 ft elevations. In 1947, Shaft No. 1 was developed as a two-compartment shaft to the 230 ft level. From that depth, it was widened to three compartments all the way to the shaft bottom (772 ft). As of 1947, the total lateral development in the shaft amounted to 3,225 ft of drifting and crosscutting on the 130 ft, 230 ft, 350 ft, and 475 ft levels. In addition, a surface diamond drilling program was completed totaling 15,000 ft. The work was terminated due to financial difficulties.
Red Poplar Gold Mines Ltd.	1951	Acquired the mine and reportedly commenced dewatering, followed by sampling of the underground workings. No records of this work are currently available.
	1961 to 1971	Reorganization of the company, first as Consolidated Red Poplar Mines, and second as New Dimension Resources in 1971.
Mount Jamie Mines (Quebec) Ltd. (Mount Jamine Mines)	1975	Optioned a 75% interest in the mine from New Dimension Resources.
	1976	Dewatered and rehabilitated the mine to the 230 ft level. Three stopes were developed, and 1,224 tons of material were hoisted from these stopes (i.e., Stopes B, C-1, and C-2). Mount Jamie Mines also constructed an open-air gravity mill, capable of treating 100 tons per day. Remnants of this mill are still at the mine. The mill was in operation in 1976, at which time 550 tons of material was treated with a recovery of 78%.
	1980	Processed 420 tons remaining from the stockpile of 1976 and an additional 300 tons of low-grade material. Only the grade of the 1976 material was known (as 0.5 oz Au/ton). The concentrates of both processed materials were sent to a smelter. The weight of the concentrate shipped was 1.5 tons and it contained 175 oz of gold and 58 oz of silver.
	1981	Completed the metallurgical testing of a tailings sample from the 1980 milling, in addition to surface exploration. None of the reports on the metallurgical testing (completed by Lakefield Research) are available.
Oneiro–Alfa Ltd.	1982	Acquired a 52.5% interest in the mine and initiated a surface diamond drilling program consisting of 5,400 ft of drilling. Nineteen holes were drilled. Sixteen of these drill holes tested the main zone (Shaft No. 1), while three holes were completed at Shaft No. 2. Some geological mapping was reported around Shaft No. 1. The geological consulting firm, Derry Michener Booth & Wahl, prepared a set of compilation maps, plans, and a record of that work in December 1982.
Keeley Frontier Resources Ltd.	1983	Took over Oneiro-Alfa's interest in the mine. Dewatered Shaft No. 1 to below the 475 ft elevation for the purpose of implementing some of the recommendations made by Derry Michener Booth & Wahl. Reportedly, the work completed consisted of underground and surface diamond drilling with overburden stripping, sampling, and mapping. Surface diamond drilling included 22 holes in the vicinity of Shaft No. 1 and 2 holes near Shaft No. 2, for a combined total length of 8,400 ft. According to a report by John Reddick dated December 1983, 28 underground holes were drilled on the 130 ft level, 9 holes on the 230 ft level, and 2 holes on the 475 ft level for a combined total of 5,004 ft. Reddick mentions that the drifts had to be slashed at the drill stations and the muck was cleared out of the stations. The muck was left at the entrances to the drifts on either side of the stations, and the rails were blasted in several locations.

Table 6-2: Exploration and Mining History of Mount Jamie Mine Area (continued)

Phase/Company	Year	Activity
Jamie Frontier Resources Inc. (Jamie Frontier)	1984	Acquired the mine, which at that time consisted of 11 patented claims and 4 staked claims. Expanded the surface facilities, including upgrading the kitchen/dining area, refurbishing the living quarters, constructing a washhouse, and installing a septic tank/field and sewer system. Brought the camp up to accepted standards of the time. The plant was refurbished, with diesel-operated power generators and backup generators installed and an assaying facility built on the site. The mill was winterized and some of the mill equipment replaced while upgrading other facilities. Due to funding difficulties, this work was not completed.
	1985	Dewatering and refurbishing of the shaft were completed during the winter of 1985. Rehabilitation of the levels was delayed due to the poor condition of the stations, where development muck had been left at the entrances and 5,000 ft of rails blasted. Serious discrepancies in the underground surveying of the mine workings and drill hole locations were discovered and corrected.
	1985 to 1986	Completed underground sampling, and surface and underground drilling in the Shaft No. 1, Shaft No. 2, and North Vein areas.
Hy Lake Gold Inc. (Hy Lake)	2005	Entered into an option agreement to acquire a 75% interest in the Mount Jamie property (nine claims) from Jamie Frontier.
	2007	Completed acquisition of the remaining 25% interest in the nine claims from Gsont Holdings Limited.
	2011 to 2012	Completed diamond drilling in the mine area for a total of approximately 8,500 m.
West Red Lake Gold Mines Inc. (RLG)	2012	Hy Lake changes its name to West Red Lake Gold Mines Inc. (RLG).
	2017	Drilling of 15 holes for a total length of 2,544 m.
	2022	Acquired by WRLG.
West Red Lake Gold Mines Ltd. (WRLG)	2023	RLG changed name to West Red Lake Gold Mines Ltd.

Notes:
oz = ounce; in. = inches; ft = foot; Au = gold; m = metre.

6.2.2 Mount Jamie Mine Deposit Historical Resources

A number of historical resource and reserve estimates have been prepared for the Mount Jamie Mine in the past. The resource estimates summarized below are historical in nature and should not be relied upon, however, these estimates provide indication of mineralization in the Mount Jamie Mine area. Further drilling is required prior to preparing a current mineral resource estimate. A QP has not completed sufficient work to classify the historical estimate as a current mineral resource or mineral reserve, and WRLG is not treating the historical estimates as current mineral resources or mineral reserves.

The historical resource estimates by the various underground operators are listed in Table 6-3.

Table 6-3: Mount Jamie Mine Historical Resource Estimates

Author	Year	Tons	Grade [oz/ton Au]	Based on
A.H. Honsberger	1941	50,000	0.5	Channel sampling U/G + DDH
P.O. Broadhurst	1979	40,000	0.5	Underground sampling + DDH
G.R. Clark	1981	40,000	0.5	Proven + Probable – largely U/G sampling and DDH
D.E. Smith	1984	19,000	0.415	Recoverable gold content U/G (x 80% recovery)
J.B. Gordon	1988	44,535	0.437	U/G sampling + DDH Shaft No. 1 only
P.J. Vamos	1988	44,535	0.437	As above – Shaft No. 1
P.J. Vamos	1988	16,928	0.355	Surface and U/G sampling, DDH Shaft No. 2 and North Vein Zone
P.A. Bevan	2010	7,250 (measured) 27,502 (indicated) 20,989 (inferred) 1,269 (stockpile)	13.2 14.07 12.72 6.86	Cut-off grade 0.10 oz/ton Au, no minimum width. Measured based on surface sampling or underground drift and raise sampling. Indicated based on diamond drill holes spaced 60 ft (18 m) apart. Inferred include single isolated blocks or based on holes in a different zone as opposed to the majority of holes in the main section.

Source: Kita 2022.
Notes:
oz = ounce; Au = gold; U/G = underground; DDH = diamond drill hole; ft = foot; m = metre.

6.3 Red Summit Mine

The Red Summit Mine area consists of two patented claims located approximately 3 km east-southeast of the Rowan Mine deposit (Figure 4-2). The Red Summit occurrence was discovered in the early 1930s. From 1935 to 1938, a shaft was sunk, and a 5-five-ton/day mill was installed; the mill was operated to treat high-grade ore from surface and to test some underground vein material. The ownership, exploration, and mining history for the Red Summit Mine area is summarized in Table 6-4.

Table 6-4: Exploration and Mining History of Red Summit Mine Area

Company/Phase	Year	Activity
Rowan Discovery Syndicate	1930	Surface work.
Coniagas Mines Limited	1931	Optioned the mine. Eleven diamond drill holes totalling 611 m.
Red Crest Gold Mines Limited	1934	Eight diamond drill holes totalling 649 m.
	1935 to 1938	Installed 5-ton/day mill; three compartment shafts to 180 m levels at 45 m, 82.5 m, 127.5 m.
	1936	The mill was operated to treat high-grade ore from surface and to test some underground vein material. Apparently 277 oz Au and 65 oz Ag were produced from 591 tons milled (Ferguson et al. 1971).
Northgate	1981	Surface examination by Northgate.
Claude Resources Inc. (Claude)	2008	No work was completed.
Hy Lake Gold Inc. (Hy Lake)	2008 to 2009	Entered into an option agreement to acquire a 100% interest in two contiguous patented mining claims totalling 26 ha, which contain the former producing Red Summit mine from Claude.
	2008 to 2011	Completed two diamond drilling programs in the mine area in 2008 and 2011.
West Red Lake Gold Mines Inc. (RLG)	2012	Hy Lake changed its name to West Red Lake Gold Mines Inc. (RLG).
	2022	Acquired by WRLG.
West Red Lake Gold Mines Ltd (WRLG)	2023	RLG changed name to West Red Lake Gold Mines Ltd.

Notes:
oz = ounce; Au = gold; Ag = silver.

6.4 Past Production

The historical records of actual mining (i.e., ore hoisted to surface) are limited due to poor record keeping. At the Rowan Mine, the ore mined by previous operators was not systematically hoisted to surface and there was no mill on site. At Mount Jamie Mine, approximately 2,000 tons to 3,000 tons of material is reported to have been mined, with some material treated at an on-site 100-ton/day mill and the rest of the material stockpiled. At Red Summit Mine, a 5-ton/day mill was installed that reportedly treated 591 tons of material in 1936.

7 GEOLOGICAL SETTING AND MINERALIZATION

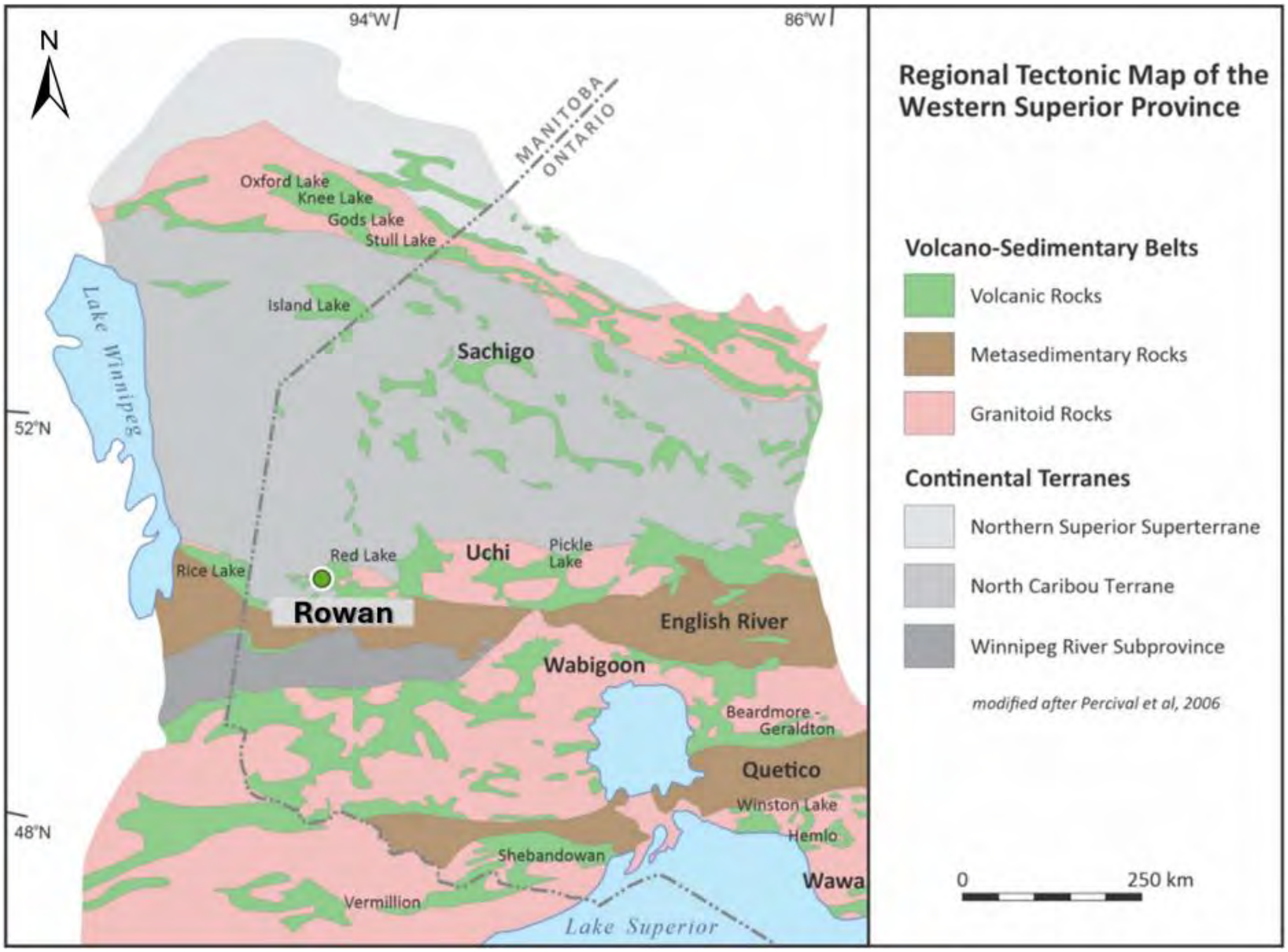
The information in this section is largely based on, or has been modified from, Kita (2022).

7.1 Regional Geology

The Rowan Property is located within the western portion of the Archean Superior Province of the Canadian Shield (Figure 7-1). The Rowan Property occupies part of the Uchi domain, which forms the southern margin of the North Caribou terrane, along its boundary with the English River Belt (Percival et al. 2012). The Uchi domain comprises a series of plutonic rocks discontinuously surrounded by arcuate belts of supracrustal volcano-sedimentary rocks (i.e., greenstone belts). These greenstone belts record more than 300 million years of tectonostratigraphic evolution, including: rifting and arc volcanism, plutonism, deformation, metamorphism, uplift, erosion, and gold mineralization. Most Uchi domain greenstone belts have some recorded historical gold production, however, only the Red Lake Greenstone Belt, where the Rowan deposit is located, is an important gold district, reported to have produced up to 30 million ounces (Moz) of gold to the end of 2022 (Malegus et al. 2023).

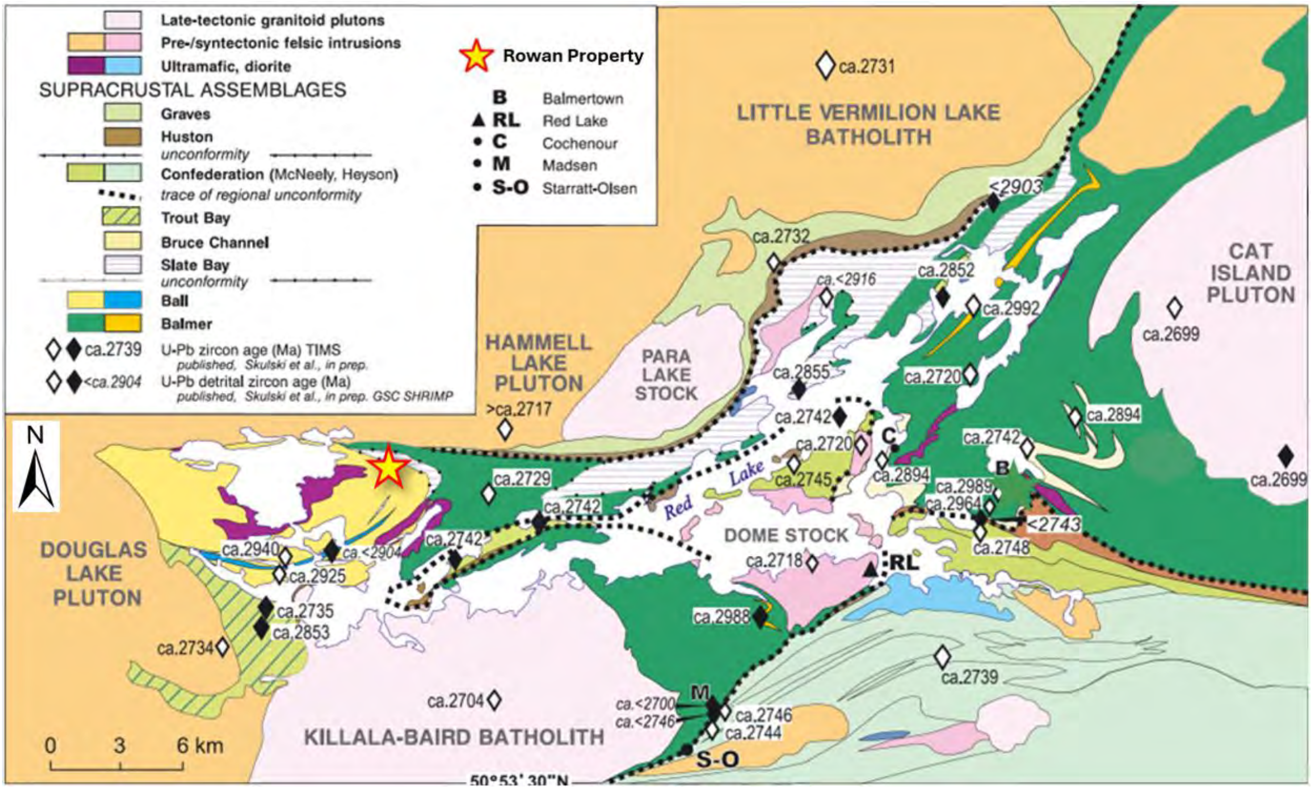
The Red Lake Greenstone Belt is approximately 50 km by 40 km and comprises a series of circa (ca.) 2.99 billion years (Ga) to 2.7 Ga supracrustal rocks intervening between three main granitoid batholiths ranging from 7 km to 20 km across (Figure 7-2). The supracrustals are dominated by the Mesoarchean Balmer assemblage (ca. 2.99 Ga to 2.96 Ga), which consists of mostly massive to pillowed tholeiitic sequences separated by distinctive felsic and ultramafic rocks and minor metasedimentary rocks (Sanborn-Barrie et al. 2004b). In the western part of the Red Lake Greenstone Belt, Balmer rocks are overlain by the Ball assemblage, which hosts the Rowan Property, and consists of a Mesoarchean (ca. 2.94 Ga to 2.93 Ga) sequence of mafic to felsic calc-alkaline metavolcanic and metasedimentary units that have been intruded by varying sizes of ultramafic to felsic intrusives. The relationship between the Balmer assemblage and Ball assemblage is uncertain as the Balmer-Ball contact is obscured by the Slate Bay assemblage (ca. 2.90 Ga to 2.85 Ga), which is a molasse sequence with a basal conglomerate, quartz arenites, siltstones and mudstones, with rare occurrences of chert, marble, and iron formations. Preservation of these sediments in a thin wedge at the Balmer-Ball contact suggests some type of structural juxtaposition between these two assemblages, at least 25 million years after their initial deposition.

Figure 7-1: Geology of the Western Portion of the Archean Superior Province



Source: after Percival et al. 2006.

Figure 7-2: Simplified Geological Map of the Red Lake Greenstone Belt



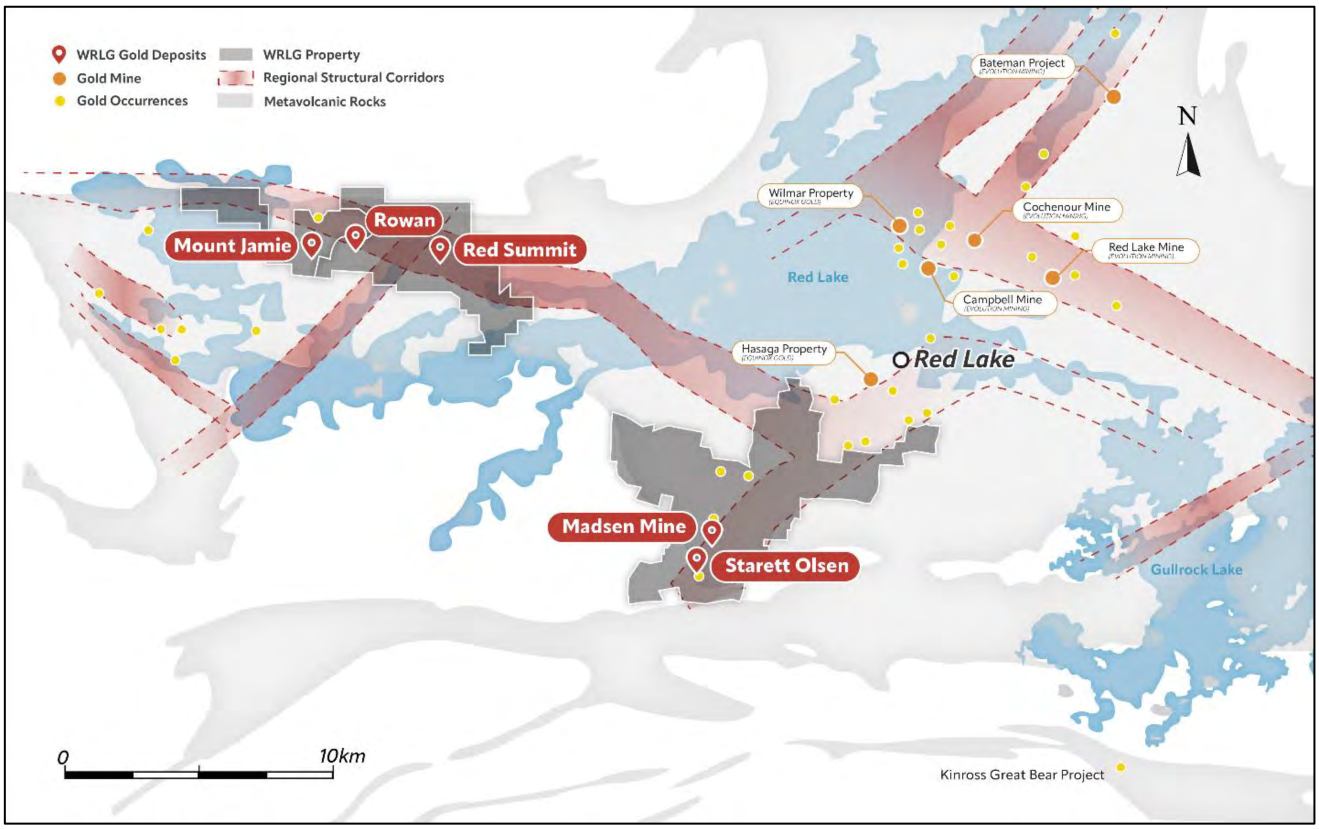
Source: Sanborn-Barrie et al. 2004a.

The northern and southern portions of the Rowan Property straddle two other important regional unconformities. In the south, parts of the Confederation assemblage (ca. 2.75 Ga to 2.74 Ga) and the Huston assemblage (ca. less than 2.74 Ga to greater than 2.73 Ga) overlay the Balmer assemblage along an angular unconformity that can be traced across the entire Red Lake Greenstone Belt (Figure 7-2). In the northern part of the Rowan Property, a different extension of this same unconformity is marked by the Huston assemblage only; Confederation assemblage rocks appear to be absent in this area.

This regional unconformity records a period of significant uplift (ca. 2.74 Ga) that closely precedes the timing of gold mineralization that occurred ca. 2.72 Ga, and subsequent deformation and metamorphism that peaked ca. 2.70 Ga. This period of gold mineralization is spatially associated with the major gold deposits in the area of the Rowan Property, including the Red Lake, Cochenour, and Madsen mines, as well as most of the known gold showings and prospects. The spatio-temporal association of gold with this style of early- to syn-orogenic unconformity is commonly observed in other gold belts around the world, including Archean-aged greenstone belts (e.g., Abitibi [Ontario and Quebec], Yellowknife [Northwest Territories], Yilgarn [Western Australia]), Proterozoic-aged greenstone belts (e.g., La Ronge [Saskatchewan], Birimian [Burkina Faso], Ashanti [Ghana]), and Mesozoic-aged greenstone belts (e.g., Golden Triangle [British Columbia]). These unconformities and their associated molasse-type sedimentation (commonly with alkaline magmatism) appear to signal the onset of reactivation of crustal-scale structures, magmatic-hydrothermal activity, deformation, and metamorphism; all of which have contributed to the formation of an environment favourable for gold mineralization.

The structure of the Red Lake Greenstone Belt records at least four periods of deformation (Sanborn-Barrie et al. 2004b), beginning with a regional tilting and uplift event (D0) that produced the angular unconformity upon which the Confederation assemblage was emplaced. This D0 event was followed by a belt-wide, east-west shortening event (D1) that produced northerly trending F1 folds after 2.74 Ga. The style of folding and deformation that occurred during this time is unclear, but it may have been accompanied by some thrusting in certain parts of the Red Lake Greenstone Belt. Superimposed on these structures is a regionally developed penetrative D2 foliation (S2) and tight, upright folds (F2) that are variably oriented throughout this belt. The D2 strain (ca. 2.72 Ga) is concentrated in altered and mineralized deformation corridors, forming rectilinear boundaries around domains of relatively lower strain and variably oriented D2 structures. These domain boundaries are well-known to be the most prospective regions in the Red Lake Greenstone Belt (Figure 7-3) and likely represent the surface expression of crustal-scale structures that acted as zones of weakness and fluid migration during orogenesis and mineralization. The northwest trending St. Paul's Bay - Pipestone Bay Deformation Zone, which hosts the Rowan deposit, and the Golden Arm Fault, are two such regional structures.

Figure 7-3: Geological Map of the Red Lake Greenstone Belt Highlighting Gold Producers, Prospects, and Showings

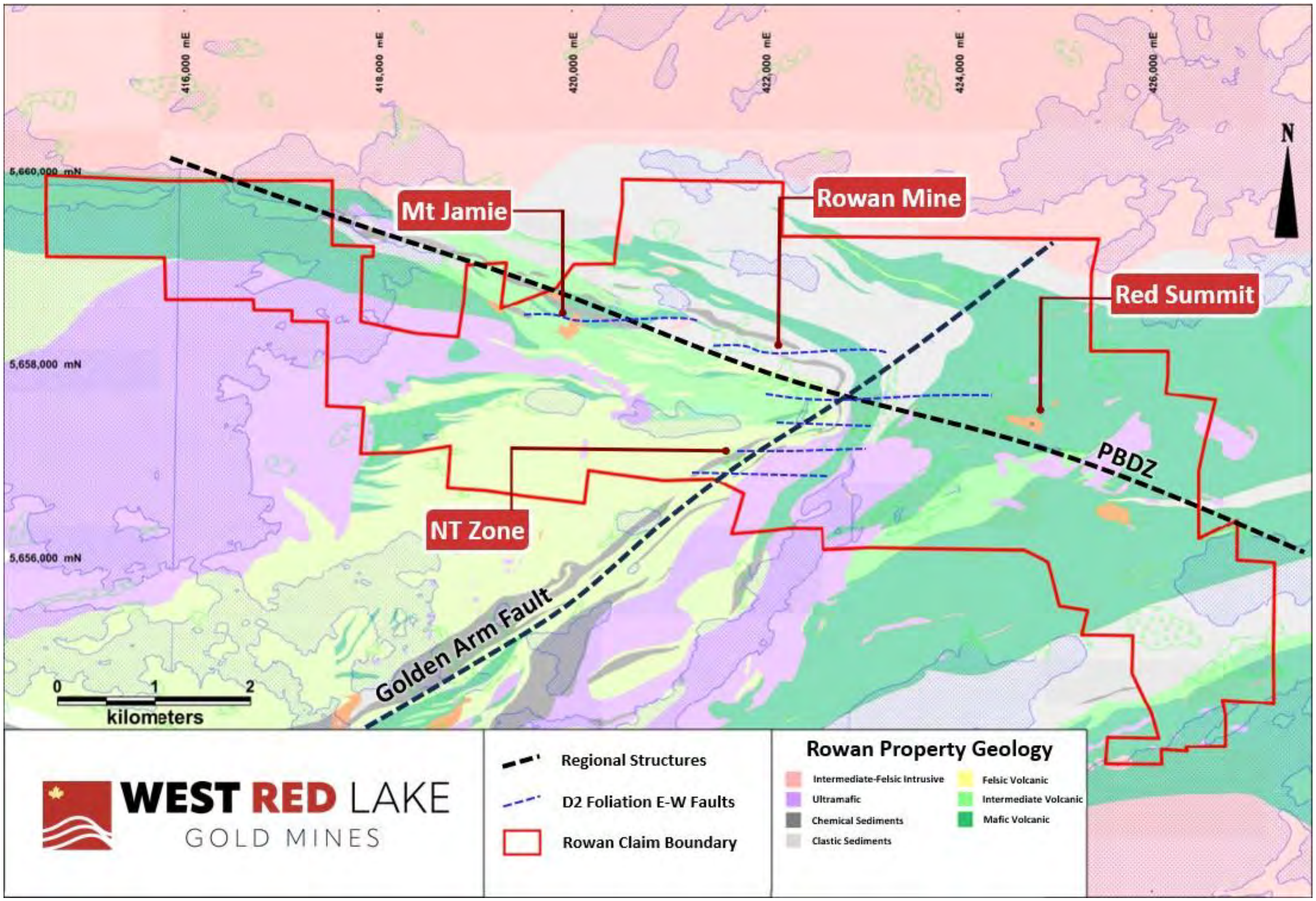


Source: Modified from Durochers et al. 1987.

7.2 Local and Property Geology

The Rowan Property is centred on a regional antiform that plunges moderately to the east (Figure 7-4) and straddles the intersection of two regional gold corridors (i.e., Pipestone Bay-St Paul Deformation Zone [PBDZ] and Golden Arm Fault) (Figure 7-3). The axial plane of the fold is roughly aligned with the regionally penetrative S2 foliation, and the antiform appears to be a simple F2 fold. However, local measurements of minor F1 folds with variably plunging hinge lines (Sanborn-Barrie et al. 2004a) and potential Type II interference fold shapes in map patterns raise the possibility of significant D1 structures in the region that remain undefined.

Figure 7-4: Rowan Property Geology Map



Source: SR 2025.

7.3 Mineralization

Currently, three principal gold occurrences known on the Rowan Property include the three past-producing mines (i.e., Rowan Mine, Mount Jamie Mine, and Red Summit Mine), the NT Zone, as well as numerous gold prospects. In general, gold mineralization occurs as visible millimetre-scale blebs in quartz veins, veinlets, and stockworks, which is true for many of the occurrences on the Rowan Property. There appears to be a bias towards folded/sheared lithological contacts often involving felsic porphyries and/or iron formations. When units of differing competencies are deformed, voids can be created at or near their contacts and gold bearing silica can later fill and seal these openings. The wall rock adjacent to the quartz veins is generally barren.

All of the vein systems on the Rowan Property are open along strike and down dip due to the limited exploration. Most of the systems strike in a general east-west direction and are steeply dipping.

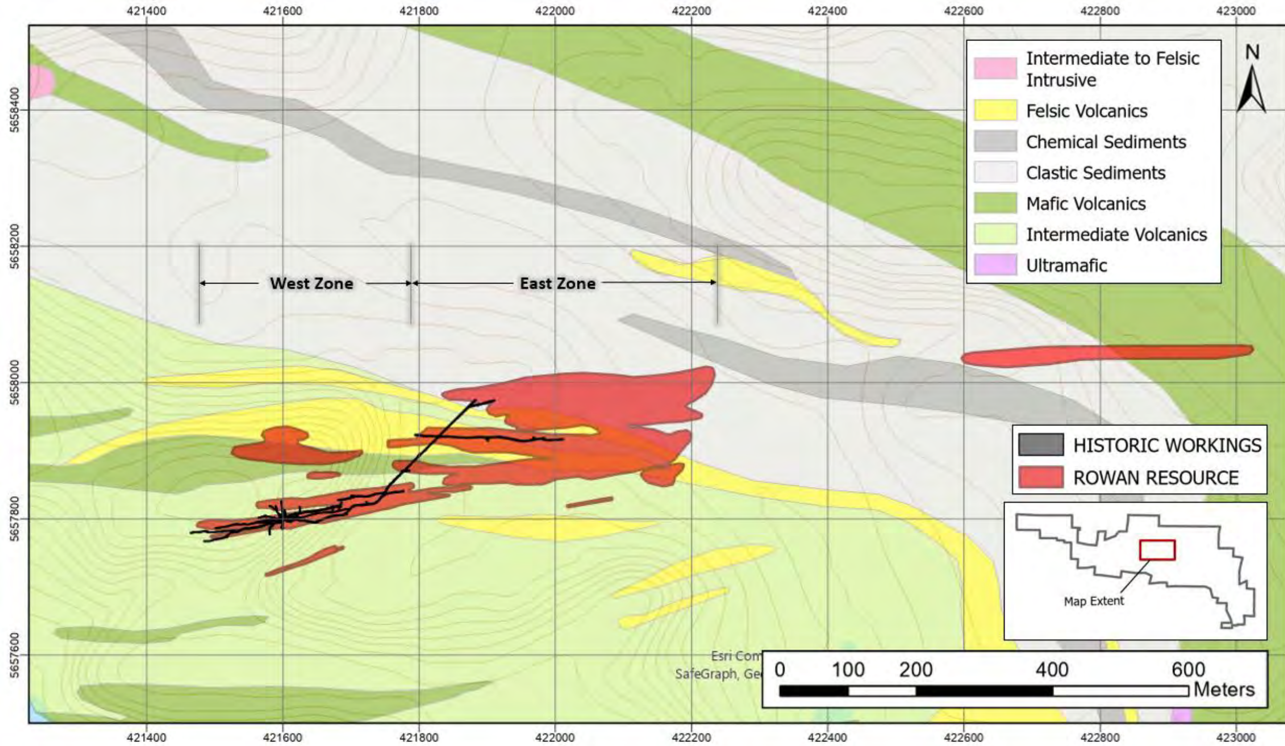
7.3.1 Rowan Vein System

The Rowan Vein System has been the focus of most of the exploration on the Rowan Property since the initial discovery of four sub-parallel narrow veins on surface at Discovery Hill in 1928. Since that time, this vein system has been drifted from underground on three levels and extensively drilled, including 62 drill holes in 2023.

The best gold grades in the Rowan Vein System often occur when coarse and visible native gold is present, which occurs within distinct bluish to grey, glassy quartz veins/stringer zones ranging in thickness of 10 cm to 30 cm and up to 1 m. These zones rarely exceed 60 cm wide, and broad zones of diffuse silicification have generally not been found. Trace amounts, and up to 1%, of pyrite and pyrrhotite is common within these veins/stringers. Less common, but a better positive indicator of gold grade, is the occurrence of sphalerite, galena, arsenopyrite, and chalcopyrite. Generally total sulphides make up less than 2%. Metallurgical testing completed to date indicates favourable recovery characteristics (Section 13, Mineral Process and Metallurgical Testing).

The overall Rowan deposit consists of numerous, narrow, high-grade quartz veins that define an east-northeast trending corridor, approximately 150 m wide (Figure 7-5). This corridor mainly transects the lower mafic to intermediate metavolcanic units of the Ball assemblage in the hinge of the Rowan Property-scale antiform and appears to dissipate once this corridor intersects the unconformity with the metasedimentary Slate Bay assemblage to the east.

Figure 7-5: Plan View of Rowan Deposit Projected to Surface, with a Transparent Geology Overlay



Source: SR 2025.

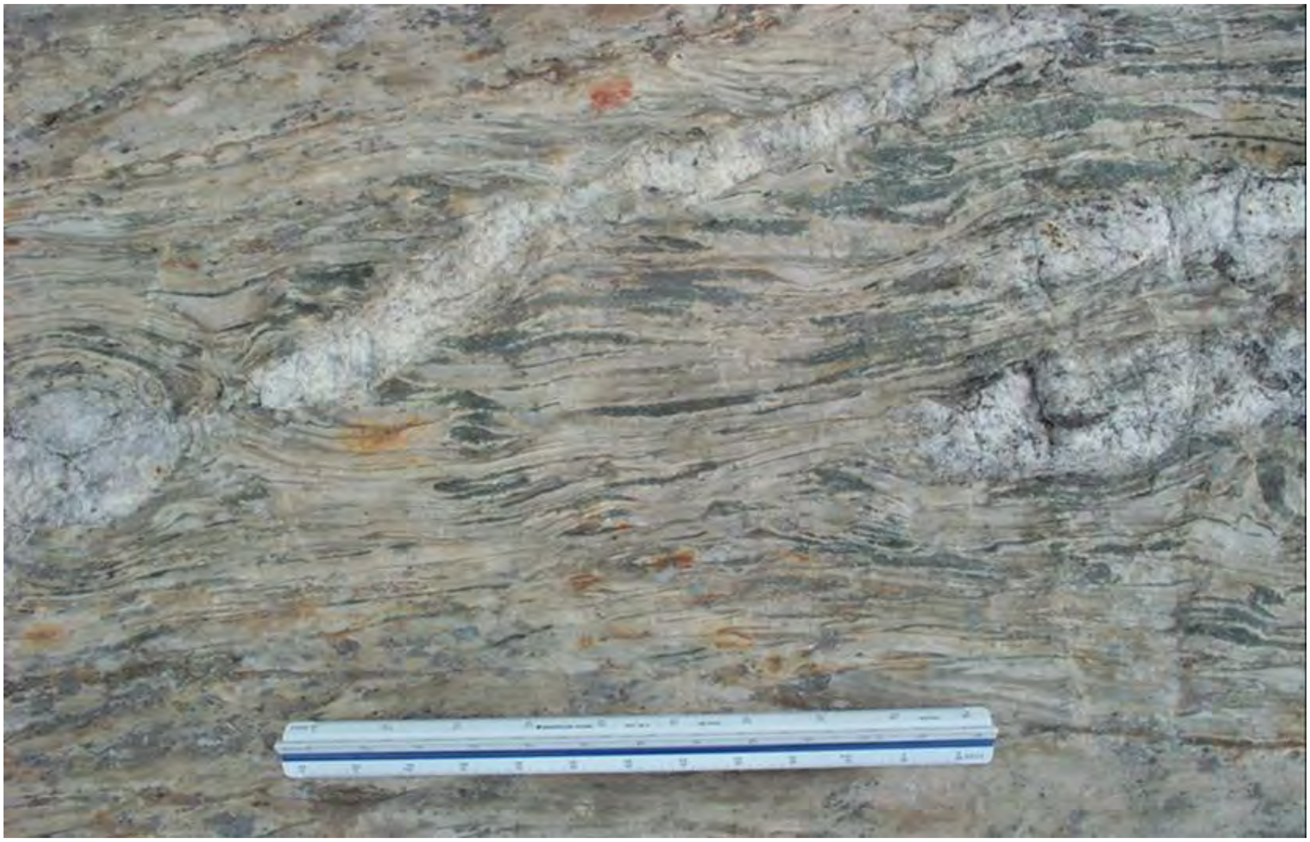
Within the main corridor, the veins are organized into two principal directions of continuity: an east-northeast direction, parallel to the main trend, which appears to dominate; and a subordinate east-west component, parallel to regionally penetrative S2 foliation (Figure 7-5). However, oriented core measurements collected at the Rowan Property and their analysis indicate that the opposite is true: the gold-bearing veins are predominantly oriented east-west, parallel to the S2 foliation, with a less pronounced group of east-northeast trending veins. A spread of orientations along a great circle between these two groups suggests the two directions could each represent limbs of a fold, or folds. The sugary, recrystallized texture of the veins, dominant parallelism between the veins and the S2 foliation, and local recognition of folded veins in drill core all support vein emplacement prior to intense D2 deformation.

The apparent contradiction of east-northeast trending continuity with predominantly east-west trending veins is reconciled if the veins are transposed or rotated into an en échelon arrangement of minor folds, parallel to S2 foliation, within a broader enveloping surface following the east-northeast trend. Since the deposit-scale east-northeast trend does not itself appear to be folded, the Rowan Vein System must have been emplaced post-F2 folding, but prior to the development of the intense, penetrative S2 foliation. This sequence of events, with an early- to syn-D2 deformation timing of structurally controlled gold mineralization is also seen in other major deposits of the Red Lake Greenstone Belt, including in the deposits of the Madsen and Red Lake mines.

Transposed vein systems can present unique challenges for modelling using drill core due to their irregular geometry. Continuity may be difficult to recognize in the short range due to segmentation and rotation during transposition, but long-range continuity may still be well developed and delineated by using the enveloping surface of the transposed veins.

During transposition, continuous planar features and veins become segmented and rotated or folded into and overprinted by a penetrative foliation such that their original continuity is difficult to delineate. This effect is particularly acute in the hinges of folds where the original plane of continuity may now pass through the foliation rather than along it, even though the planar features themselves are aligned with the foliation as shown in (Figure 7-6). In this figure, the dark green tuff layer provides an example of how a once continuous layer can be segmented and rotated during transposition. Local continuity follows the foliation, whereas, deposit-scale continuity is defined by tracing an enveloping surface around the broader fold hinge, perpendicular to the foliation.

Figure 7-6: Photo of Transposed Fold near Barrick's Hemlo Operation



Source: SR 2025.

Modelling complex geological shapes such as transposed veins systems (e.g., Figure 7-6) can be achieved using implicit methods in Leapfrog provided sufficient closely spaced data are present to capture the variability of the deformed vein geometries. These implicit methods are being used successfully at the Madsen mine where drill spacing is predominantly 6 m or less. While the drill spacing in the Rowan deposit is greater than 6 m, the spacing is sufficient to produce geologically plausible representations of the transposed vein geometries (Figure 7-7). For areas with wider drill spacing, the implicit shapes did not control abnormally wide or high-grade intercepts propagating across an unrealistic span of the vein; for this reason, and to minimize dilution in the Rowan deposit model, the implicit shapes were used only as a guide for manually constructed vein models using the vein modelling tool in Leapfrog. This approach allowed the construction of more tightly controlled shapes that capture the structural style of the Rowan Vein System and remain optimized for use in mineral resource estimation (Section 14, Mineral Resource Estimates).

On Figure 7-7, the red-shaded zones were generated using implicit modelling controlled by manually placed structural discs guided by 2 m composites of drill hole data. The green outlined areas were generated using the Leapfrog vein modelling tool controlled by manual interval selections of the same 2 m composites. Implicit modelling captures geological shapes with a similar style to those of transposed vein systems, which can be difficult to control in areas of lower data density. The vein modelling tool in Leapfrog allows for greater control on the thickness and continuity of the veins and production of more reasonable volumes for mineral resource estimation.

The result of the implicit methods is a more realistic model of the Rowan Vein System that agrees with the structural observations at the deposit-scale and drill core-scale, and that aligns with the structural setting and controls seen at other deposits within the Red Lake Greenstone Belt, including in the deposits of the Madsen and Red Lake mines.

Figure 7-7: Comparison of Different Modelling Styles for the Rowan Deposit

Source: SR 2025.

7.3.2 Mount Jamie Vein System

Most of the descriptions of the mineralized bodies at Mount Jamie vein system are from the Main Zone and the Shaft No. 2 Zone.

The Main Zone strikes 60 degrees (°) north of west (i.e., N60W) and has a dip ranging from 45° to 85° to the south, based on surface observation. The vein splits and branches, but in general, is confined to a width of 1.2 m and occupies a fracture zone in altered greenstone close to and along tongues of quartz porphyry.

The North Vein Zone is not a simple vein structure but a larger linear structure controlling a system of quartz veins and lenses. This zone extends between the Shaft No. 2 area and the southeast, and also through the gold occurrences on the north shore of Rowan Property. The width of the North Vein Zone structure varies from several metres to tens of metres and the individual veins range between 0.6 m and 0.9 mm, which was confirmed by the underground work completed by Jamie Frontier in the mid-1980s.

The mineralogical description states "traces of pyrite, pyrrhotite, sphalerite, chalcopyrite and galena" and visible gold was noted as rare. Certain quantitative relationships between gold and other minerals exist. A relationship was found between the enrichment in gold and the amounts of chalcopyrite, and the same relationship was found to exist between gold and galena, which could be an important factor in designing a working hypothesis and logistics for exploration.

A second very different and important type of gold mineralization was observed in the underground diamond drill core, and later by a crosscut, sub-drift, and several lifts on the same horizon. Characteristics of this second deposit, the North C Vein, are as follows:

- massive, almost homogeneous smoky quartz vein;
- very fine grain size, almost glassy;
- vertical to steeply north dipping;
- highly stressed, mechanically unstable rock;
- virtually no sulphides;
- very fine-grained free gold content, resulting in fairly significant assays; and
- requires special sampling and assaying efforts.

The North C Zone was a distinct en échelon unit, approximately 30 m north of the Main Zone. Because this zone was so different from the usual targets, there were difficulties in drill core evaluation and the North C Zone showed clear evidence of being mechanically unstable rock. This duality of mineralization will be accounted for during the planning of any upcoming exploration programs.

The Mount Jamie vein system strikes N60W and has a dip ranging from 60° to 85° to the south, based on surface observation. The vein splits and branches, but in general, is confined to a width of 1.2 m and the vein occupies a fracture zone in altered volcanics close to and along tongues of quartz porphyry.

The Mount Jamie vein system is a gold-bearing shear zone averaging 1.2 m in width. This vein zone has been traced for 165 m on the 38 m level and was found to be discontinuous at deeper levels. This vein zone strikes S65E and dips 85° south where exposed on surface. Six mineralized zones have been outlined along this vein.

Three of these mineralized zones are located along the intermediate volcanic rock-felsic breccia contact. This zone is accessed via Shaft No. 1 and lateral workings on the 38 m, 69 m, and 145 m levels.

In conclusion, the gold mineralization in the Mount Jamie Mine area is hosted by a shear-controlled linear feature striking approximately N30W. The mineralized zones appear to fall into two distinct groups:

- Veins and lenses of gold-bearing quartz in association with a variety of sulphide minerals including pyrite, chalcopyrite, pyrrhotite, sphalerite, galena, and rare flakes of native gold.
- Smoky quartz veins, massive with stress lines and random distribution of fine flakes of gold.

7.3.3 Red Summit Vein System

The Red Summit vein system was described by Horwood (1940) as follows:

- The claims are underlain by Keewatin lava flows of andesitic and basaltic composition, a small stock of quartz diorite, and later fine-grained diorite dikes. The lava flows, generally termed greenstones, have been deformed and range from slightly schisted rocks to chloritic schists. A zone of fracturing and shearing with quartz veins as much as 6 feet in width was discovered and opened up in a series of surface trenches. The zone occurs along or close to a contact between a small stock of quartz diorite on the north and Keewatin greenstones on the south.

Horwood (1940) describes the veining and mineralization as follows:

- The quartz veins occur in a zone of shearing and fracturing close to or along the south side of the quartz diorite stock and dip north with the contact at angles from 60 [degrees] to 70 degrees. The strike of the zone is at a slight angle to the contact; to the east the shearing goes into the greenstones, whereas to the west it occurs on the quartz diorite or along contacts between this rock and the later fine-grained diorite.
- Two types of quartz veins occur. The earlier type, which makes up the bulk of the vein quartz, is a barren, white quartz. The later type, a banded, bluish-grey quartz, which carries most of the mineralization, occurs in places along the walls of the barren veins but more often obliquely across them or as separate veins in the diorite stock. Later quartz-carbonate veins, which do not contain any gold, also occur.
- Values in gold are associated with a coarse bronzy pyrite, which generally occurs in the bluish-grey quartz veins or in the shattered walls along the margins of these veins. Although some bronzy pyrite occurs scattered along the zone, the best concentration has been found in the section close to the junction of the zone and the diorite-greenstone contact. This section appears to have been more favorable for the development of open spaces for vein-filling. More fracturing took place here, and there is a greater development of the later bluish-grey type of quartz. Consequently, the possible ore shoots are in this section. Both to the northwest extending into the diorite and to the southeast extending into the greenstone, the zone is narrower and there is less quartz of both types and less bronzy pyrite.
- A pale, whitish pyrite, which occurs widely disseminated through the diorite and in places in appreciable quantities in stringers in and about the sheared walls of the veins, contains very little gold.
- Visible gold is rare in the veins in the underground workings and was noted in only a few places associated with a grey mineral of unknown composition.

7.3.4 Newman-Todd Zone

The Newman-Todd (NT) Zone, also known as the NT Horizon, is a northeast trending stratigraphic horizon located in the south-central portion of the Rowan Property. The mineralized zone is typically 50 m to 100 m wide and consists of mixed chemical to clastic sediments, including stromatolitic marbles with discontinuous felsic volcanic layers. The NT Zone has been traced along strike on the Rowan Property for approximately 2 km. Clear evidence of shearing is typically absent within this zone, but it represents a corridor of intense quartz-ankerite alteration and brecciation with varying degrees of gold mineralization.

Gold within the NT Zone occurs as free gold in quartz veins including within massive sulphide units of pyrite and pyrrhotite, associated with sulphide-magnetite zones within the breccia, and in vein arrays within quartz porphyry intrusions internal to the NT Zone. Associated sulphides include pyrite, pyrrhotite, galena, sphalerite, and lesser arsenopyrite and chalcopyrite.

8 DEPOSIT TYPES

Gold mineralization on the Rowan Property belongs to the Archean lode gold class of deposits (Roberts 1986) or, using the current term, "orogenic gold deposit" class (Groves et al. 1998, Kerrich et al. 2000). Structurally controlled, low-sulphide, lode gold vein systems in metamorphic terrains from around the world possess many characteristics in common, spatially and through time; these systems constitute a single class of mesothermal precious metal deposits, formed during accretionary tectonics.

The Superior Province is the largest exposed Archean craton in the world and has accounted for more gold production than any other Archean craton, with the 25 largest-known deposits having produced more than 1 Moz of gold, which equals 30 t of gold.

Most lode gold deposits form close to regional terrane-boundary structures that act as vertically extensive hydrothermal plumbing systems. Major mining operations tend to be sited near deflections, strike slip, or dilational jogs on the major structures. Most lode gold deposits are situated in second or third order splays, or fault intersections, that define domains of low mean stress and correspondingly high fluid fluxes. Accordingly, mineralization and associated alteration is most intense in these flanking domains. The largest lode gold mining operations are in terrains that possess greenschist facies hydrothermal alteration assemblages developed in cyclic ductile to brittle deformation. Fewer deposits are known in amphibolite to granulite facies terranes characterized by amphibolite to granulite facies alteration assemblages, ductile shear zones, and ductilely deformed veins (McCuaig and Kerrich 1998).

Characteristically, the largest gold deposits of the Red Lake Gold District are spatially associated with, but not hosted in, porphyries like those exposed at the Dome mine, in Timmins, Ontario. This association has led to considerable speculation regarding the genetic relationship of felsic porphyry emplacement to gold mineralization. Magmatism provides an attractive source of heat and fluids to transport and focus gold mineralization, but the fluid chemistry typical of most orogenic gold deposits (i.e., low salinity, cardon dioxide-rich) can be generated by any mix of magmatic-hydrothermal, metamorphogenic, or even mantle-derived sources. It may be that the spatial association of porphyries and gold deposits is more of a reflection of their mutually favourable environment, rather than any causative link between the two.

Another commonly recognized feature of this class of orogenic gold deposits is their proximity in space and time to regional, angular unconformities of similar age, with their contemporary molasse-type sediments and alkaline magmatism. These features record periods of rapid uplift and denudation, and sediment accumulations are preferably preserved adjacent to crustal-scale structures that were actively accommodating this uplift. Alkaline magmatism points to a mantle source, which further implicates deep crustal-scale structures, and their spatial proximity to gold implies common structural pathways for the gold-bearing fluids. The contemporary relationship between these features and gold mineralization suggests that there is something unique about this relatively punctuated period of tectonism, within a more protracted orogeny, that promotes gold transport and emplacement. The coincidence of mantle involvement and rapid uplift within an evolving orogen can be explained by sudden interruptions in the lower crust and mantle lithosphere, such as slab break-off or delamination.

Gold deposits in the Red Lake Greenstone Belt differ from the typical orogenic gold deposits in that the principal timing of gold emplacement predates the development of the regionally penetrative foliation and amphibolite-grade metamorphism. Minor events followed, but none at the scale of the main deposits themselves.

The intense transposition of the original vein systems has obscured many of the key structural relationships that controlled mineralization, which has led to ongoing debates about their origins.

Mineralization at the Red Lake Mine takes the form of gold-bearing, sulphide-bearing, quartz-carbonate veins hosted by mafic to ultramafic volcanic rocks. At the nearby Madsen mine, gold mineralization is associated with intensely altered and deformed zones characterized by strongly foliated diopside-amphibole-biotite-quartz-carbonate veins and alteration. Recent work at the Madsen mine has shown that the vein emplacement was originally controlled by a more typical brittle-ductile deformation zone that was then modified into its current form during D2 deformation and the associated amphibolite-grade metamorphism, which led to irregular geometries and recrystallized alteration assemblages atypical of this class of orogenic gold deposits. Pre-D2 emplacement and subsequent ductile transposition play a fundamental role in controlling the shape and distribution of mineral deposits at a stope-scale in both the Red Lake and Madsen mines.

Other mineralization styles in the Red Lake Gold District include gold-bearing quartz veins hosted by iron formation (e.g., McFinley deposits), sulphide-rich quartz lenses, veins and stringers in a porphyry dyke (e.g., Hasaga mine), and siliceous shears within granitic stocks (e.g., McKenzie mine).

9 EXPLORATION

WRLG completed exploration programs including diamond drilling, a property-wide light detection and ranging (LiDAR) survey, and a regional soil sampling orientation survey after acquiring the Rowan Property in December 2022.

The majority of the exploration work completed on the Rowan Property by RLG, the previous owner, between 2007 and 2021 consisted of numerous diamond drilling programs, an airborne drone magnetic survey, and a channel sampling program near the surface exposure of the historical Rowan Mine in the area, now being termed as the "West Zone". Historically, the Rowan Property has seen limited underground development and therefore historical underground drilling data is only available in limited detail.

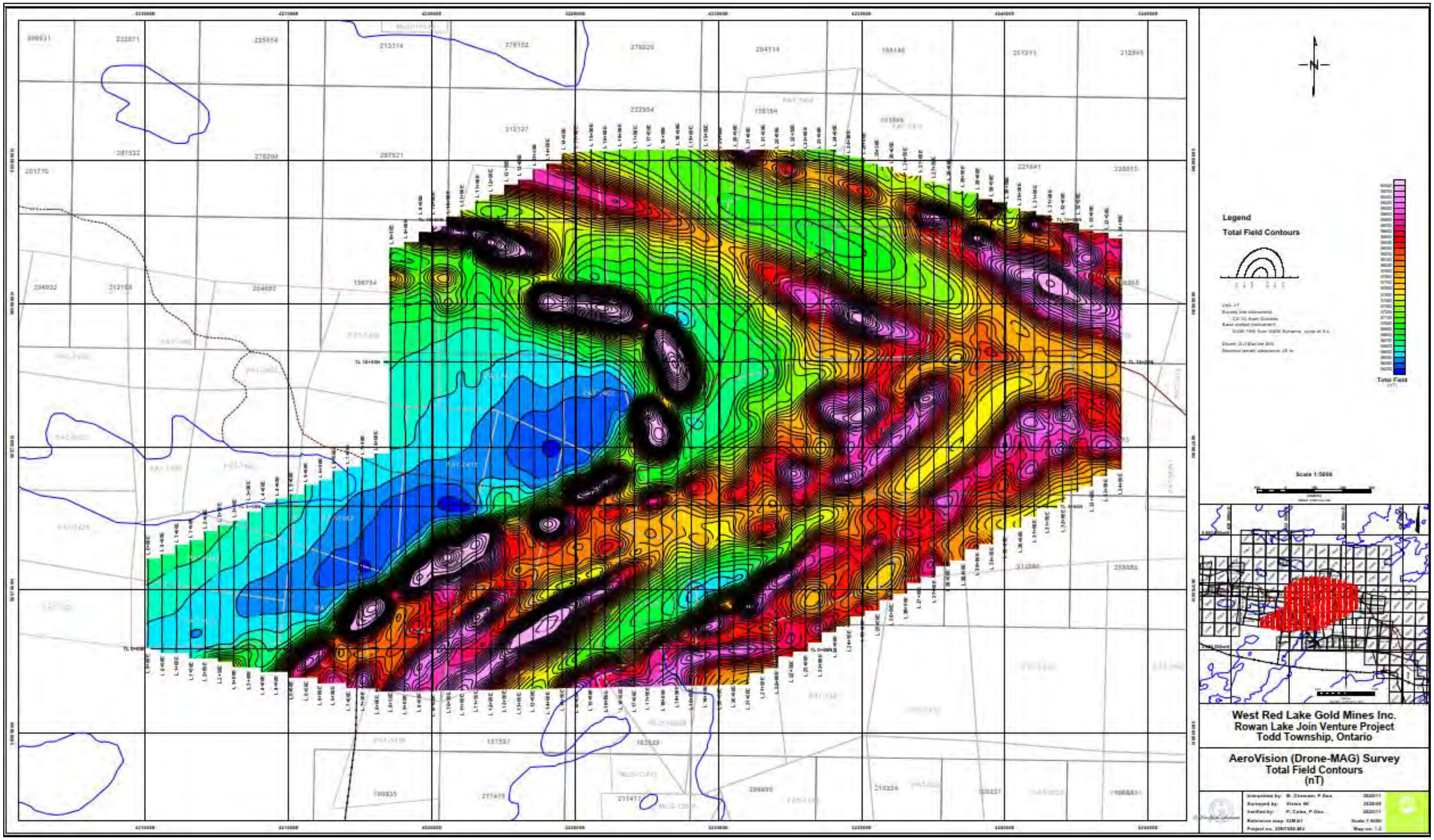
Additional information on exploration programs is summarized in Section 9.1 to Section 9.4. Additional information on diamond drilling is provided in Section 10, Drilling, and additional details on historical exploration at the Rowan Property are provided in Section 6, History.

9.1 2020 Airborne Drone Magnetic Survey

In 2020, Abitibi Geophysics Inc. completed a 100 line km AeroVision Canada drone magnetometer survey on behalf of RLG (Figure 9-1). This survey was conducted over an area covering the 2 km long northeast striking NT Zone from the south edge of the Rowan Property boundary to where the NT Zone folds to the west and proximal with the PBDZ regional geological structure which hosts the Rowan Mine gold zones. The drone magnetometer program covered 4.52 square kilometres (km^2) and consisted of 68 lines spaced 50 m apart with readings recorded at 1.2 m intervals along each line. This survey represents a significant increase in resolution over the previous survey conducted over the Rowan Property, which included 100 m spaced lines with readings taken at 3.5 m intervals and was completed by Sial Geosciences Inc. for Goldcorp Inc. in 2000.

The 2020 drone magnetometer survey was effective in mapping lithologic units with moderate- to high-magnetic susceptibility and clearly delineates the east-plunging regional-scale antiform. This survey also appears to have delineated a number of areas that may have been subjected to structural offset along the east-west trending faults that run sub-parallel to the trend of D2 foliation. The QP recommends that follow-up fieldwork should be completed to ground-truth these apparent structural features and help refine the geologic interpretation over the Rowan Property.

Figure 9-1: Plan View of 2020 Airborne Drone Magnetic Survey at the Rowan Property



Source: Abitibi Geophysics 2020.

9.2 2021 and 2022 Channel Sampling Program

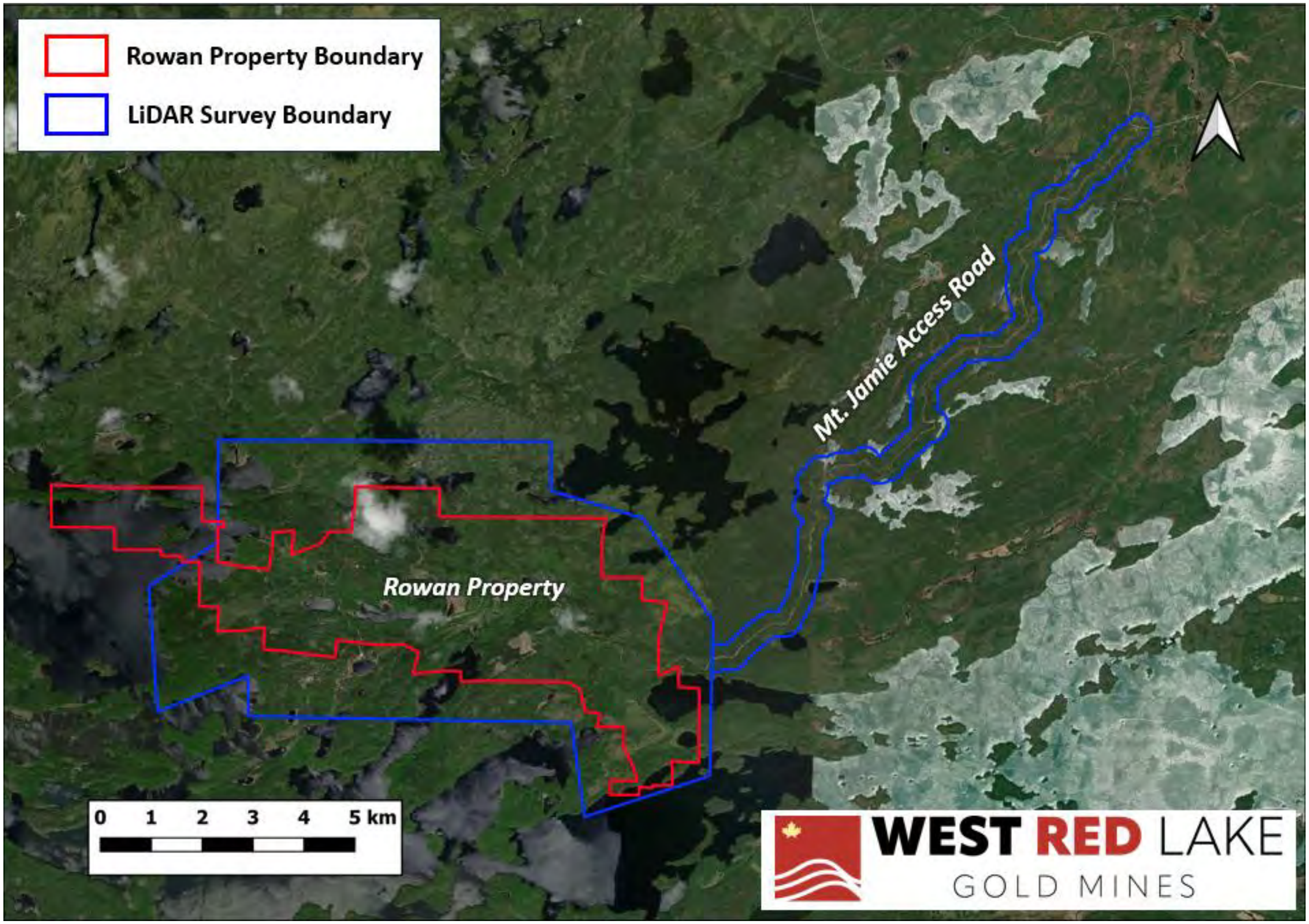
In 2021, RLG completed a preliminary surface channel sampling program over a 200 m strike length at the Rowan Mine area along the east-west strike to investigate the potential for a surface bulk sample, which is now termed the 'West Zone' by WRLG. The 2021 program included 97 samples along discontinuous lines with approximately 5 m line spacing and with up to 7 contiguous 1 m samples along each line segment, oriented perpendicular to stratigraphy. A follow-up sampling program in 2022 included 182 additional channel and grab samples collected to test gold distribution along the veins identified in the 2021 sampling program. The samples from both programs were sent for assay under similar protocol as used for drill core assaying by RLG. The results of the 2021 and 2022 sampling programs were encouraging and indicate that gold mineralization does persist to surface, however, a surface bulk sample is not being considered by WRLG at this time.

9.3 2023 Light Detection and Ranging Survey

In March 2023, Eagle Mapping Ltd., completed a property-wide LiDAR survey over the Rowan Property and Mount Jamie Mine Road on behalf of WRLG (Figure 9-2). The survey extents totalled 66.8 km^2, which included a 500 m wide corridor flown over the entire Mount Jamie Mine Road. The LiDAR data was collected at 10 pulses/m^2 and included aerial photography at 10 cm ground sampling distance for creation of a 10 cm resolution orthophoto to complement the digital surface model (DSM). Deliverables provided as part of the survey included:

- Point cloud information greater than or equal to 10 points/m^2 with classified ground and non-ground (e.g., water).
- Classified bare earth point file in LAS digital format.
- Digital elevation model (DEM) at 0.5 m and a DSM.
- 0.5 m hill shade DEM.
- Contours that are aesthetically accurate for the ground surface.
- 10 cm resolution orthophoto.

Figure 9-2: Plan View of 2023 Light Detection and Ranging Survey at the Rowan Property

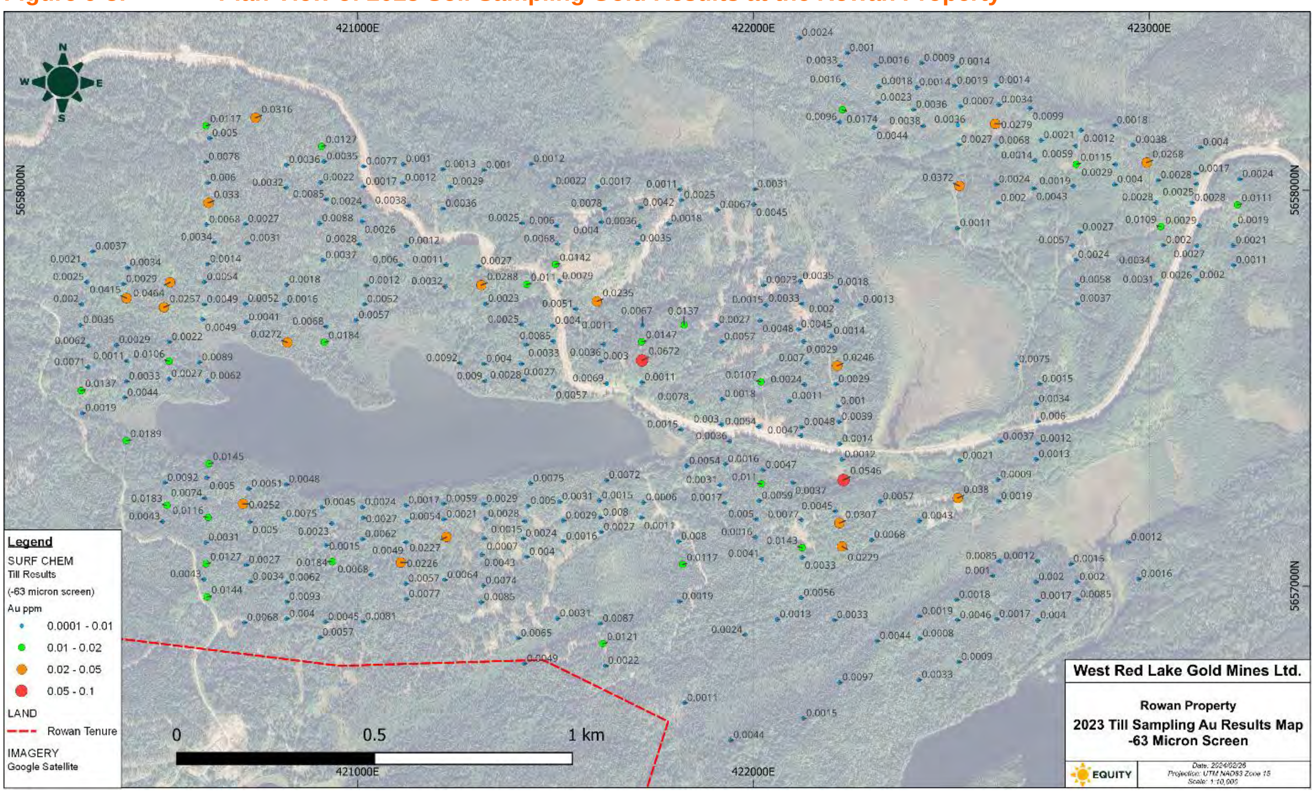


Source: SR 2025.

9.4 2023 Orientation Soil Sampling Survey

In 2023, Equity Exploration Consultants Ltd. (Equity) completed a soil sampling survey on behalf of WRLG (Ontario) that included a proposed 660 till sample stations over a 3.5 km^2 sampling grid. Sample spacing was generated using 50 m spacing on north-south lines and 100 m spacing on east-west lines. A total of 334 successful till samples were collected using Dutch-style hand augers, which have a maximum sampling depth of 2 m (Figure 9-3). A certified reference material of Canadian glacial basal till (i.e., OREAS 46) and duplicate field samples were inserted into the sampling sequence at a 5% rate for each quality assurance / quality control (QA/QC) sample type in the field. After field collection, samples were transported to ALS Global in Vancouver for preparation and analysis. All samples were dried to 60°C to reduce loss of volatile mercury and arsenic, sieved to generate two fraction datasets (i.e., -180 +63 micron (µm), and -63 µm) before undergoing argon inductively coupled plasma mass spectrometry (ICP-MS) digestion and analysis. Review of assay data shows both datasets returned high quality, usable data, with the -63 µm dataset returning a preferential return in quality based on cumulative frequency probability plots and Tukey plotting of gold and pathfinders in parts per million (ppm) values. Exploratory data analysis using principal component analysis (PCA) was completed on both datasets and the components display both geochemical associations that reflect local geology and also zones of exploration potential. Glacial drift is estimated to range from 100 m to 300 m towards south by southwest across the Rowan Property based on the geological associations shown in the PCA compared to mapped lithologies.

Figure 9-3: Plan View of 2023 Soil Sampling Gold Results at the Rowan Property



Source: SR 2025.

10 DRILLING

10.1 Summary

Since 1934, 622 diamond drill holes have been completed at the Rowan Property for a total length of approximately 128,000 m, with 77 diamond drill holes completed by WRLG in 2022 and 2023 totalling approximately 25,000 m (Table 10-1). A drill hole compilation map of the Rowan Property is shown on Figure 10-1.

Table 10-1: Diamond Drilling Summary for the Rowan Property

Company	Year	Drill Hole Series in Database	Target	Metres	No. of Drill Holes	Type	Drill Core Size
Paulore Gold Mines	1934	Unknown	Rowan Mine	Unknown	6	Unknown	Unknown
Lake Rowan Gold Mines	1937 to 1938	RWS	Rowan Mine	1,094.0	11	DDH	Unknown
		RWU	Rowan Mine	415.9	13	DDH-UG	Unknown
West Red Lake Gold Mines	1940	M	Rowan Mine	927.0	18	DDH	Unknown
Golden Frontier	1940	GU	Mount Jamie	120.0	17	DDH-UG	Unknown
	1941	GF	Mount Jamie	123.0	7	DDH	Unknown
		GU	Mount Jamie	1,265.0	106	DDH-UG	Unknown
	1942	GF	Mount Jamie	102.0	4	DDH	Unknown
		GU	Mount Jamie	153.0	9	DDH	Unknown
Bayview Red Lake	1945	BW	Mount Jamie	4,593.0	34	DDH	Unknown
Rugged Red Lake Mines	1945	Unknown	Rowan Mine	4,746.0	25	DDH	Unknown
Lake Rowan Mines	1946	RW-46	Rowan Mine	9,845.0	56	DDH	Unknown
	1950	Unknown	Rowan Mine	Unknown	8	X-Ray	Unknown
Rowan Consolidated Mines Ltd.	1953	RWU-53	Rowan Mine	1,845.0	38	DDH-UG	Unknown
	1958	RW-58	Rowan Mine	1,340.5	7	DDH	Unknown
Cochenour Exploration Ltd.	1969	MB-69	Rowan Mine	597.0	8	DDH	Unknown
Byng Red Lake	1977 to 1978	HL	Mount Jamie	132.0	5	DDH	EXT
Oneiro-Alfa	1982	Unknown	Mount Jamie	1,646.0	19	DDH	Unknown
Keeley Frontier	1983	KF	Mount Jamie	2,564.0	28	DDH	BQ
		KU	Mount Jamie	1,583.0	38	DDH-UG	AQ
Robert Gibson	1984	RG-84	Mount Jamie	313.0	10	DDH	XRT
Goldquest Exploration Inc.	1984	RW-84	Rowan Mine	3,622.8	16	DDH	BQ
	1985	RW-85	Rowan Mine	4,539.5	51	DDH	BQ
Jamie Frontier	1985	JF	Mount Jamie	721.0	13	DDH	BQ
		JU	Mount Jamie	5,110.0	108	DDH-UG	BQ
Robert Gibson	1985	Unknown	Mount Jamie	62.0	2	DDH	XRT
	1986	RG-86	Mount Jamie	288.0	9	DDH	XRT
Goldquest Exploration Inc.	1987	RW-87	Rowan Mine	1,822.1	8	DDH	BQ
Robert Gibson	1987	RG-87	Mount Jamie	385.0	11	DDH	EXT
Jamie Frontier	1987	JU	Mount Jamie	524.0	3	DDH	BQ
Byron Bay	1987	BB87	Mount Jamie	375.0	1	DDH	BQ
Robert Gibson	1988	RG-88	Mount Jamie	113.0	3	DDH	EXT
Pezgold	1989	P	Mount Jamie	3,683.0	39	DDH	NQ
Chevron / Goldquest	1989	RW-89	Rowan Mine	2,713.0	7	DDH	BQ
	1990	RW-90	Rowan Mine	3,131.0	7	DDH	BQ

Table 10-1: Diamond Drilling Summary for the Rowan Property (continued)

Company	Year	Drill Hole Series in Database	Target	Metres	No. of Drill Holes	Type	Drill Core Size
	1993	RW-93	Rowan Mine	995.0	3	DDH	BQ
Goldcorp Inc.	1997	RW-97	Rowan Mine	904.0	2	DDH	BQ
	2001	RW-01	Rowan Mine	3,673.0	12	DDH	BQ
Zenda / Vedron	2003	JF-03	Mount Jamie	900.0	6	DDH	NQ
Kings Bay Gold Corp. Ltd.	2006	RW-06	Rowan Mine Porphyry Hill	4,846.0	23	DDH	BQ
Hy Lake Gold Inc.	2007 to 2008	HYR-07, HYR-08	Rowan Mine	8,317.0	15	DDH	NQ
			Red Summit	2,259.0	8	DDH	NQ
		HY-07	Mount Jamie	7,687.0	38	DDH	NQ
	2010	HYR-10	NT Zone	1,147.0	5	DDH	NQ
	2011	HY-11	Mount Jamie	3,489.0	31	DDH	NQ
			NT Zone	3,880.0	17	DDH	NQ
			Red Summit	2,153.0	9	DDH	NQ
	2012	HY-12	Mount Jamie	5,133.0	31	DDH	NQ
			Various	5,212.0	32	DDH	NQ
West Red Lake Gold Mines Inc. (RLG)	2013	RLG-13	Rowan Mine	3,283.0	8	DDH	NQ
	2014	RLG-14	Rowan Mine	1,416.0	10	DDH	NQ
	2015	RLG-15	Rowan Mine	1,767.0	6	DDH	NQ
	2016	RLG-16	Rowan Mine	5,176.0	16	DDH	NQ
	2017	RLG-17	Rowan Mine	5,415.5	14	DDH	NQ
			Mount Jamie	2,544.0	15	DDH	NQ
	2018	RLG-18	Rowan Mine	1,272.0	1	DDH	NQ
			NT Zone	1,443.0	8	DDH	NQ
	2019	RLG-19	NT Zone	3,060.0	12	DDH	NQ
	2022	RLG-20	NT Zone	6,373.5	20	DDH	NQ
	2021	RLG-21	Rowan Mine	3,033.0	19	DDH	NQ
			NT Zone	636.0	1	DDH	NQ
	2022	RLG-22	NT Zone	1,657.0	5	DDH	NQ
			Rowan Mine	1,428.0	4	DDH	NQ
			Porphyry Hill	1,104.0	4	DDH	NQ
West Red Lake Gold Mines Ltd. (WRLG)	2023	RLG-23	Rowan Mine	20,211.4	62	DDH	NQ
			Red Summit	780.0	2	DDH	NQ

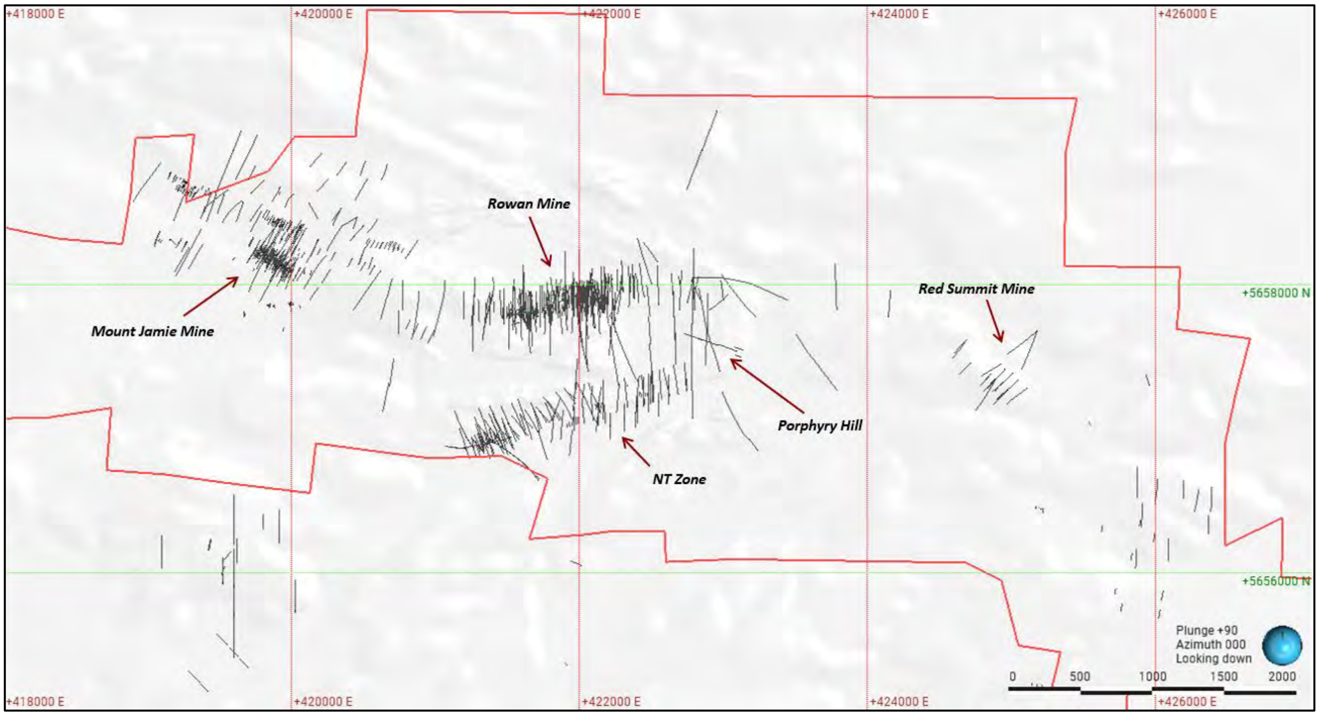
Notes:

The above information represents a compilation of information gathered from the Geology Ontario database of annual assessment report filings, as well as work summaries from authors on previous Technical Reports for the Rowan Property. The QP verified the drill hole information used for the mineral resource estimate as described in Section 12, Data Verification.

DDH-UG refers to diamond drill holes completed from historical underground workings at the Rowan Mine.

DDH = diamond drill hole; UG = underground; NT Zone = Newman-Todd Zone.

Figure 10-1: Plan View of Diamond Drilling Compilation Map for the Rowan Property



Source: SR 2025.

10.2 Diamond Drilling from 2007 to 2021

RLG, and the previous owner (i.e., Hy Lake), completed significant surface diamond drilling on the West Red Lake Project from 2007 to 2021. This drilling was completed in both the Rowan Mine area and the Mount Jamie Mine area. Diamond drilling by RLG from 2007 to 2021 included 226 drill holes for a total length of 55,423 m.

A 12 km section of the east-west striking PBDZ and the 2 km section of the NT Zone were the two main areas of the Rowan Property where diamond drilling was undertaken by RLG. The primary exploration focus was on the Rowan Mine area, which is situated near the centre of the Rowan Property within the PBDZ. A secondary exploration focus was early-stage drilling on the PBDZ that is outside of the Rowan Mine area. Additional diamond drilling was completed on the southern portion of the NT Zone associated with the northeast trending Golden Arm Fault, which crosses onto the Rowan Property from the neighbouring property located adjacent and to the south.

All 226 holes were drilled with NQ size (i.e., 47.6 mm) core, with Ontario-based Chibougamau Drilling of Red Lake contracted for most of the drilling. Collars were surveyed by a handheld global positioning system (GPS) instrument that is accurate to within two to three metres, and downhole surveys were completed using a Reflex Early Shot instrument with readings taken every 50 m.

Core was collected twice per day by RLG core technicians and taken to the core shack located at the Mount Jamie Mine site. The core technicians then measured the drill core and stapled a metal tag to each of the core boxes and recorded the hole number, box number, and footage on the tag. The technicians also recorded measurements from the drill core, including rock quality designation (RQD), core recovery, and orientation of any structures, contacts, and veins.

Ninety-nine percent (99%) of the drill core achieved 100% core recovery. Core was logged by geologists, with altered and mineralized sections marked for sampling, and core photographs were taken and stored digitally.

The GPS coordinates for each drill hole collar were determined in the field using a handheld GPS instrument, and collar locations were recorded in UTM coordinates, NAD 83 Zone 15N.

Sections of drill core to be assayed were identified by geologists during core logging. These drill core sections were split using a diamond-blade rock saw. Half of each core sample was sealed in a plastic sample bag with a sample identification tag, and the remaining half of each sample was replaced in the core box as a permanent record. This drill core is stored at the Mount Jamie Mine.

All drill holes were logged and sampled at the Mount Jamie Mine field camp. Certified gold reference standards, blanks, and field duplicates were routinely inserted into the sample stream as part of the RLG QA/QC program. Assaying was completed by either Activation Laboratories Ltd. (Actlabs) or SGS Natural Resources (SGS) at their laboratories in Ontario. Gold analyses were performed by fire assay, however, higher-grade (i.e., greater than 5 grams per tonne [g/t] of gold [Au]) samples were analyzed with a gravimetric finish. Visible gold samples, when noted, were assayed by a pulp metallic method.

Drill hole intersection lengths are not true widths; the relationship to true widths depends on the dip of the drill hole and the dip of the mineralized zone. The dips of the various mineralized zones at the Rowan Property differ but are predominately in the range 80° south to 80° north.

10.2.1 Rowan Mine Area Drilling

Between June 2007 and September 2008, RLG completed a comprehensive two-year drill program that included 15 drill holes for a total length of 8,317 m. This program mainly focused on the Rowan Mine area and extensions, with the primary purpose of testing the depth and strike extensions of vein mineralization.

In 2009, work focused on additional infill sampling of previously drilled core and data compilation.

In 2010, work included resource assessment and data reorganization as well as drilling in the Rowan Mine area. At the Rowan Mine zones, examination of the longitudinal sections for the 3-8, 3-6, 3-5, 3-2, and SXZ zones have identified the stronger gold trends, and the 2010 program focused on expanding these zones. Diamond drilling in 2010 attempted to expand the mineralization down dip and between historical drill holes, RW-85-61 and RW-85-62.

In 2013, RLG completed a drill program that included 8 drill holes for a total length of 3,283 m in the Rowan Mine area.

In 2014, RLG completed a drill program that included 10 drill holes for a total length of 1,416 m in the Rowan Mine area. This program was designed to test for depth and strike extensions of known mineralized Rowan Mine zones as well as other known gold mineralized zones. The drill holes were following up on the positive results of the 2013 drill program. Every drill hole in the 2024 program intercepted multiple zones and mineralization with anomalous- to high-grade gold assays. The high-grade intercepts correspond to historical high-grade results and are a confirmation of the continuity and extensions of the mineralized zones to depth and along strike.

In 2015 and 2016, RLG completed drill programs that included 6 drill holes for a total length of 1,767 m and 16 drill holes for a total length of 5,176 m, respectively. These programs explored the regional geological structure extending east from the Rowan Mine area along strike for a distance of 1 km to where the PBSZ, which hosts the Rowan Mine gold zones, intersects with the northeast trending NT Zone.

In 2017, RLG completed a drill program that included 9 drill holes for a total length of 3,013.5 m in the Rowan Mine area; this program included 2 holes drilled to expand gold mineralized zones to depth in the area, 4 holes drilled on the western side of the area, and 3 holes drilled adjacent to the east of the area. An additional drill program of 5 drill holes totalling 2,402 m was completed in 2017; this program included 3 holes drilled in the Rowan Mine area to test geological targets, and 2 holes drilled 1 km east of the Rowan Mine area to test targets at the intersection of the northeast trending NT Zone with the easterly extension of the PBSZ.

In 2018, RLG drilled a 1,272 m deep hole below the Rowan Mine area mineralization to test for depth extension of mineralization and the drill hole intercepted 4.39 g/t Au over 1.5 m approximately 1,050 m below surface.

In 2021, RLG completed a drill program that included 19 drill holes for a total length of 3,033 m program in the Rowan Mine area; this program included 16 near-surface drill holes to test the potential for a surface bulk sample together with surface channel sampling, and 3 deeper holes for infill drilling.

In 2022, RLG completed a drill program of 4 drill holes for a total length 1,428 m in the Rowan Mine area for the purpose of was infill and expansion of the Rowan deposit.

10.2.2 Mount Jamie Mine Area Drilling

The Golden Tree Zones and North Vein Zone are on strike and to the west of the Rowan Mine and are situated within the PBSZ located on the Mount Jamie Mine portion of the Rowan Property. Early-stage exploration drilling primarily focused on tracing the Golden Tree Zones and North Vein Zone by following the west by northwest trend of the mineralized regional structure crossing the Mount Jamie Mine with the purpose of establishing strike continuity of mineralization from the Rowan Mine area onto and across the Mount Jamie Mine area. A 31 drill hole program totalling 3,489 m was completed in 2011, a 31 drill hole program totalling 5,133 m was completed in 2012, and a 15 drill hole program 2,544 m was completed in 2017.

Gold mineralization in the Mount Jamie Mine area is generally hosted by thin quartz veins and veinlets associated with zones of carbonate and sericite-chlorite alteration and sulphide mineralization.

10.2.3 Red Summit Mine Area Drilling

The Red Summit Mine is located east and on strike with the Rowan Mine area and has very similar geology. RLG completed an 8 drill hole program totalling 2,259 m in 2008, and a 9 drill hole program totalling 2,153 m in 2011. The purpose of these two programs was to test the depth and strike extension of the mineralized zones in the vicinity of the historical underground workings at the Red Summit Mine.

These two drill programs indicated the potential for high-grade mineralization in the vicinity of the Red Summit underground workings. High-grade mineralization was intersected on a 100 m step out from the historical underground workings, which indicates that the mineralized zones extend beyond the workings. Despite the vertical to sub-vertical nature of vein sets noted historically, examination of the drill results suggests that mineralized envelopes containing the vein sets lie mainly within a shallow southwest plunging zone situated on a lithological contact between mafic intrusive (e.g., quartz diorite) and mafic volcanic.

10.2.4 Newman-Todd Zone Area Drilling

The NT Zone is the northeast extension of a large geological structure discovered on the Newman-Todd property south of the Rowan Mine. The northeast trending Newman-Todd Structural Zone hosts high-grade gold zones over a 2 km strike length to a depth of over 300 m. RLG traced this gold system for 1 km on to the Rowan Mine where iron formations continue to the northeast, towards the Rowan Creek Zone, in close proximity to the Golden Arm ultramafic structure, which is a primary control for gold mineralization in the Red Lake Gold District.

Early-stage exploration drilling was completed on the southern portion of the NT Zone from the south property boundary along a 1 km strike length and towards the northeast with a 5 drill hole program totalling 1,147 m completed in 2010, and a 17 drill hole program totalling 3,880 m completed in 2011.

An 8 drill hole program totalling 1,443 m was completed in the NT Zone area in 2018, followed by a 12 drill hole program totalling 3,060 m in 2019 in the same area. During 2020, a 10 drill hole program totalling 3,178.5 m was completed, followed by a second 10 drill hole program totalling 3,195 m in the area of the exploration drilling. These four drill programs from 2018 to 2020 were completed on the regional-scale NT Zone from the south property boundary over a 1 km distance along strike to the northeast. Several near-parallel gold zones trending along strike were intercepted from surface to a depth of approximately 200 m.

In 2021, RLG drilled a 636 m deep hole into the northeastern area of the NT Zone.

In 2022, RLG completed a 5 drill hole program totalling 1,657 m at the NT Zone to test previously intercepted areas of high-grade mineralization. An additional 4 drill hole program totalling 1,104 m was completed at the Porphyry Hill target, which sits along the overall northeast trend of NT Zone mineralization.

10.3 Diamond Drilling from 2022 to 2023

The following information summarizes drilling completed by WRLG (Ontario) following acquisition of the Rowan Property in 2022.

In 2022, WRLG completed a total of 13 drill holes at multiple targets, including a 4 drill hole program totalling 1,428 m in the Rowan Mine area, with a purpose of infilling and expanding the Rowan deposit. A 5 drill hole program totalling 1,657 m was also completed at the NT Zone to test previously intercepted areas of high-grade mineralization. An additional 4 drill holes totalling 1,104 m were completed at the Porphyry Hill target, which sits along the overall northeast trend of NT Zone mineralization.

During 2023, a total of 64 drill holes for a total length of 21,191.4 m of infill and expansion drilling was completed across the Rowan Property; this drilling was primarily focused on the Rowan Mine deposit and Red Summit northeast (NE) target, with 62 of these drill holes completed at the Rowan Mine target for a total length 20,211.4. The purpose of this drilling was to validate and increase confidence in the December 2022 Rowan mineral resource estimate, as well as test the down-dip continuity of high-grade mineralized shoots within the Rowan Vein System. Drilling was primarily focused within two mineralized shoots called the East Zone and West Zone. Assay results received from the 2023 program confirmed that quartz veining and gold mineralization continue at depth and along strike, with grades consistent with, or higher than, those outlined in the December 2022 mineral resource estimate. The Rowan deposit remains open for expansion in all directions and will continue to be a primary target for future drilling programs.

An additional 2 drill holes for a total length of 780 m were completed at the Red Summit NE target, which is located approximately 250 m northeast of the past-producing Red Summit Mine within a flexure of the PBDZ. In the Red Summit vein system, gold mineralization tends to be localized within quartz-carbonate veins hosted along the margin of a porphyritic felsic intrusive. The contact between the felsic intrusive and surrounding mafic volcanic rocks provides a favourable rheologic setting for dilation and emplacement of quartz veining and gold mineralization. The intrusive at Red Summit NE is approximately three times the size of the intrusive adjacent to the historical Red Summit Mine, which would suggest the potential for a much larger target at Red Summit NE. The drilling completed at Red Summit NE confirmed the geologic thesis, which is where a large felsic intrusive body was intercepted with a concentration (i.e., tenor) of gold mineralization increasing near the contact between the intrusive and surrounding metavolcanic rocks. The results of this initial work warrant additional drilling in this area.

All drill holes from the 2022 and 2023 programs were drilled with fully oriented NQ size core and Forage Lamontagne Fortier Inc., based out of Rouyn-Noranda, Quebec, was the contractor for all the drilling. Collars were captured using a handheld GPS instrument accurate to within 2 m to 3 m. The drill rig was aligned at the proper azimuth and declination on each planned drill hole using the Imdex-Devico DeviAligner tool. Downhole survey information was collected using an Imdex-Devico DeviGyro tool. Downhole survey tests were completed approximately every 30 m during drilling to monitor the downhole deviation of each drill hole. On completion of a drill hole, continuous in and out downhole surveys were collected. After completion and abandonment of a drill hole, collars were re-surveyed using a high-precision (i.e., less than 1 m accuracy) Trimble R2 GPS unit. Orientation marks were made on the drill core by trained drilling personnel utilizing a Reflex Act III orientation tool.

All drill holes from the 2022 and 2023 programs were systematically logged, photographed, and sampled by trained geologists at WRLG's Mount Jamie core processing facility. The minimum allowable sample length was 0.5 m, and the maximum allowable sample length was 1.5 m. Standard reference materials and blanks were inserted at a targeted 5% insertion rate. The drill core was then cut lengthwise utilizing a diamond blade core saw along a line pre-selected by the geologist. To reduce sampling bias, the same side of drill core was sampled consistently utilizing the orientation line as a reference. For samples containing visible gold, a trained geologist supervised the cutting and bagging of these samples and verified the core saw blade was 'cleaned' with a dressing stone following each visible sample interval. Bagged samples were then sealed with zip ties and transported by WRLG personnel directly to the SGS laboratory in Ontario for assay. The archived and halved core is stored at the Mount Jamie camp.

The QP is not aware of any drilling, sampling, or recovery factors that could materially affect the accuracy and reliability of the results.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Sampling procedures and methods have evolved over the long history of exploration and mining on the Rowan Property and specific procedures also varied among operators. Sample preparation, analyses, and security are separately described below based on the time period and/or project operator.

The QP is of the opinion that, based on review of the of historical information available, the sampling, sample preparation, security, and analytical procedures were generally in line with best practices for their time and the sampling, sample preparation, security, and analytical procedures currently undertaken by WRLG meet or exceed modern best practices. The historical procedures and those undertaken by WRLG are adequate for modern targeting, modelling, and resource estimation.

11.1 Rowan Mine Area

Since 1937, there have been 30 diamond drill programs completed in the Rowan Mine area by 11 different companies. Between 1937 and 2008, no company QA/QC programs were in place. A summary drill collars, total metres drilled, QA/QC programs, and laboratories used for the 1937 to 2021 drill programs is provided in Table 11-1. Information on drilling completed at the Rowan Mine area from 2022 and 2023 is provided in Section 11.1.6, West Red Lake Gold Mines Ltd., 2023.

Table 11-1: Historical Drill Hole Summary of Quality Assurance / Quality Control Programs at the Rowan Mine Area (1937 to 2021)

Year	Company	Drill Hole Series in Database	No. of Collars	Laboratory Certificates	Assay Numbers	Detection Limit [g/t Au]	Comment
1937	Lake Rowan Gold Mines	RWS-37-**	12	Red Crest / Bell White	151	0.34	-
1937	Lake Rowan Gold Mines	RWU-37-**	1	Red Crest	19	0.34	-
1938	Lake Rowan Gold Mines	RWU-38-**	11	Red Crest / ALS Chemex	105	0.34	-
1946	Rowan Consolidated Mines	RW-46-**	14	Dickenson / Bell White	257	0.34	-
1953	Rowan Consolidated Mines	RWU-53-**	38	Dickenson	884	0.34	-
1958	Rowan Consolidated Mines	RW-58-**	7	Dickenson	120	0.34	-
1983	Pipestone Bay Resources	P-83-**	2	Bourlanmac	299	0.34	-
1984	Goldquest	RW-84-**	14	Cochenour P Okanski	943	0.34	-
1985	Goldquest	RW-85-**	45	Cochenour P Okanski	699	0.34	-
1987	Goldquest	RW-87-**	6	-	301	0.01	-
1989	Chevron	RW-89-**	4	-	1,122	-	-
1990	Chevron	RW-90-**	6	ALS Chemex	1334	-	-
1993	Goldquest	RW-93-**	3	-	116	-	-
1997	Goldcorp	RW-97-**	2	-	261	-	-
2001	Goldcorp	RW-01-**	4	ALS Chemex	219	-	-
2006	Kings Bay	RW-06-**	8	SGS	434	0.01	-

Table 11-1: Historical Drill Hole Summary of Quality Assurance / Quality Control Programs at the Rowan Mine Area (1937 to 2021) (continued)

Year	Company	Drill Hole Series in Database	No. of Collars	Laboratory Certificates	Assay Numbers	Detection Limit [g/t Au]	Comment
2007	Hy Lake	HYR-07-**	8	ALS Chemex	1,050	0.001	-
2008	Hy Lake	HYR-08-**	3	SGS	796	0.01	-
2010	Hy Lake	HY-10-**	4	-	1,508	-	Company standards and duplicates
2011	Hy Lake	HY-11-**	4	ActLabs	1,633	0.01	Company standards and duplicates
2013	RLG	RLG-13-**	8	ActLabs	3,172	0.01	Company standards and duplicates
2014	RLG	RLG-14-**	10	ActLabs	395	0.01	Company standards and duplicates
2015	RLG	RLG-15-**	6	SGS	368	0.005	Company standards, blanks 1/4 core duplicates, laboratory QA/QC
2016	RLG	RLG-16-**	15	SGS	1,579	0.005	Company standards, blanks 1/4 core duplicates, laboratory QA/QC
2017	RLG	RLG-17-**	7	SGS	1,272	0.005	Company standards, blanks 1/4 core duplicates, laboratory QA/QC
2018	RLG	RLG-18-**	2	SGS	678	0.005	Company standards, blanks 1/4 core duplicates, laboratory QA/QC
2021	RLG	RLG-21-**	20	SGS	2,083	0.005	Company standards, blanks 1/4 core duplicates, laboratory QA/QC

Notes:
g/t = grams per tonne; Au = gold; - = unknown; RLG = West Red Lake Gold Mines Inc.; SGS = SGS Natural Resources; QA/QC = quality assurance and quality control.

11.1.1 Lake Rowan Gold Mines, 1937 to 1938

Lake Rowan Gold Mines conducted surface and underground drilling at the Rowan Mine in 1937 and 1938. The assay information was listed in the database and 75% of the assays were confirmed by assay certificates.

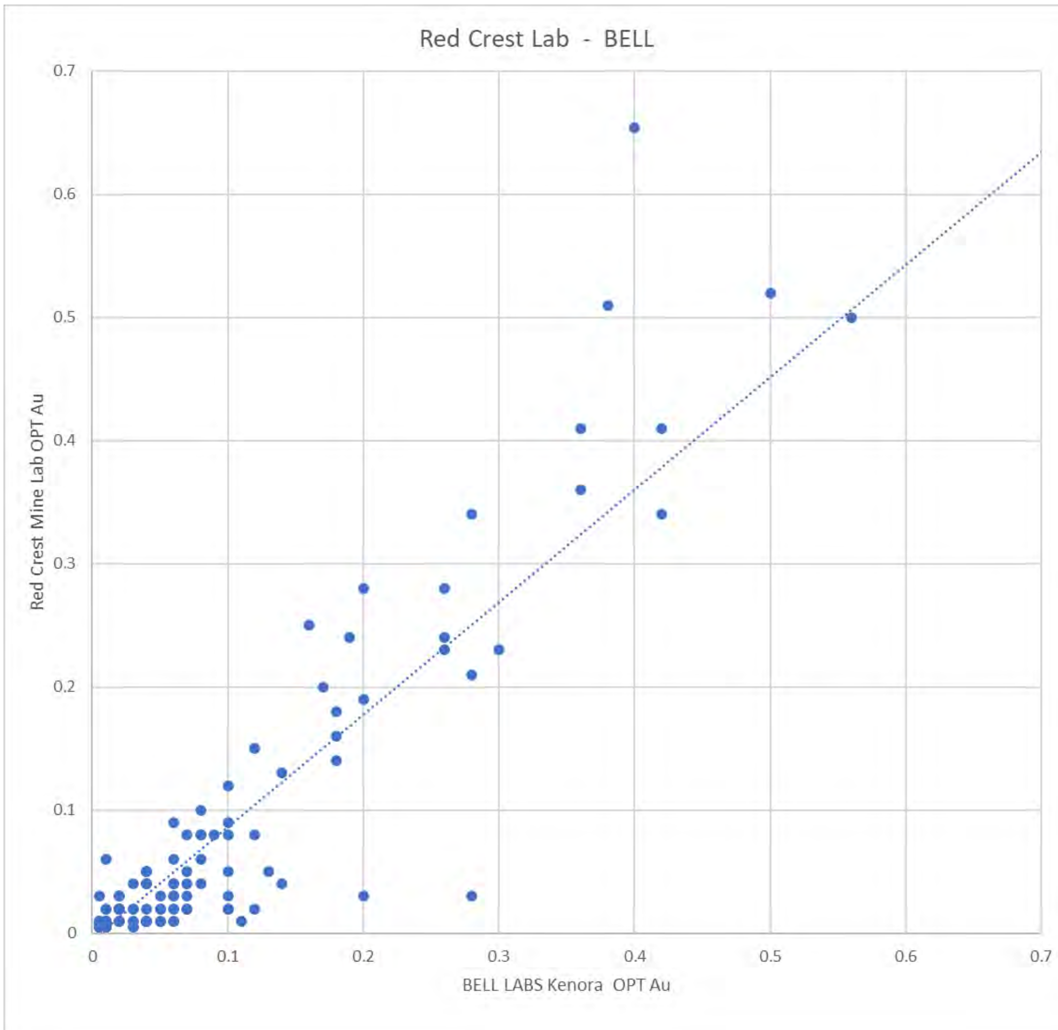
There are no records available for the company and laboratory QA/QC programs and procedures for diamond drilling and assaying. However, it appears the company may have limited the sample length to obtain a representative sample. The surface and underground drilling programs have an average sample length of 0.63 m, with a maximum length of 12.25 m and a minimum length of 0.07 m. The length of samples grading above 6.0 g/t Au averaged 0.29 m.

Lake Rowan Gold Mines did send reject material from 140 samples comparing assay results from the Red Crest Assay Lab to J.W.N Bell Assay Lab in Kenora, Ontario (Figure 11-1).

The laboratory QA/QC program is unknown. Based on the assay certificates, fire assay with gravimetric finish had a detection limit of 0.34 g/t Au. Based on this detection value, trace and nil values were recorded as 0.001 g/t rather than using 0.17 g/t (i.e., equal to half the detection limit), which would be anomalous using current measuring technology.

Based on the lack of recorded QA/QC procedures and assay certificates, the QP considers the assay data from this program suitable to either limit mineralized zones or be used with assay results from more recent drilling. If a zone is solely defined by these drill holes, the zone should be considered inferred until confirmed with more recent results.

Figure 11-1: Assay Results Comparison between Laboratories – Red Crest versus J.W.N Bell



Source: SR 2025.
Notes: OPT = ounces per tonne; Au = gold.

11.1.2 Rowan Consolidated Mines, 1946 to 1958

Rowan Consolidated Mines (RCM) conducted surface and underground drilling at the Rowan Mine between 1946 and 1958. The assay information was listed in the database and 60% of the assays were confirmed by assay certificates.

There are no records available for company and laboratory QA/QC programs and procedures for diamond drilling and assaying. To mitigate bias in assaying, RCM limited the sample length to obtain a representative sample. The surface and underground drilling programs have an average sample length of 0.41 m, with a maximum length of 1.95 m and a minimum length of 0.03 m. Sample length of samples grading above 6.0 g/t Au averaged 0.22 m.

The laboratory QA/QC program is unknown. Based on the assay certificates, fire assay with gravimetric finish had a detection limit of 0.34 g/t Au. Based on this detection value, trace and nil values were recorded as 0.001 g/t rather than using 0.17 g/t (i.e., equal to half the detection limit), which would be anomalous using current measuring technology.

Based on the lack of recorded QA/QC procedures, the QP considers the assay data from this program suitable to either limit mineralized zones or be used with assay results from more recent drilling. If a zone is solely defined by these drill holes, the zone should be considered inferred until confirmed with more recent results.

11.1.3 Various Companies, 1983 to 2006

A number of companies conducted surface diamond drilling programs in the area of the Rowan Mine shaft (Table 11-1). RCM conducted surface and underground drilling at the Rowan Mine between 1946 and 1958. The assay information was listed in the database and 25% of the assays were confirmed by assay certificates.

There are no records available for laboratory QA/QC programs and company QA/QC procedures for diamond drilling and sampling.

During Goldcorp's 1985 diamond drill program, the company sent 64 pulp samples from the Paul Okanski Cochenour Lab to X-Ray Assay Labs in Toronto, Ontario. There was a slight positive bias in the data above 1.102 ounce per tonne (oz/t) Au.

Based on the lack of recorded QA/QC procedures, the QP considers the assay data from this program suitable to either limit mineralized zones or be used with assay results from more recent drilling. If a zone is solely defined by these drill holes, the zone should be considered inferred until confirmed with more recent results.

11.1.4 Hy Lake, 2007 to 2012

Hy Lake recorded QA/QC, sample preparation, analyses, and security procedures for drilling completed at the Rowan Mine from in 2007, 2011, and 2012 as described in Guy (2015).

All drill holes were logged and sampled at the Mount Jamie Mine field camp. Assaying was completed by either independent ActLabs or SGS based in Ontario. Samples were transported directly to the laboratories by company core technicians for sample preparation and analyses. Gold analyses were performed by fire assay, and higher-grade (i.e., greater than 5 g/t Au) samples were analyzed with a gravimetric finish.

After the 2009 drill program, Hy Lake maintained its own QA/QC program. Certified gold reference standards, blanks, and field duplicates were routinely inserted into the sample stream as part of Hy Lake QA/QC program.

Both ActLabs and SGS developed a Quality Management System (QMS) designed to confirm the production of consistently reliable data; these QMSs were implemented at each laboratory location. The QMSs cover all laboratory activities and consider the requirements of the International Organization for Standardization (ISO) standards.

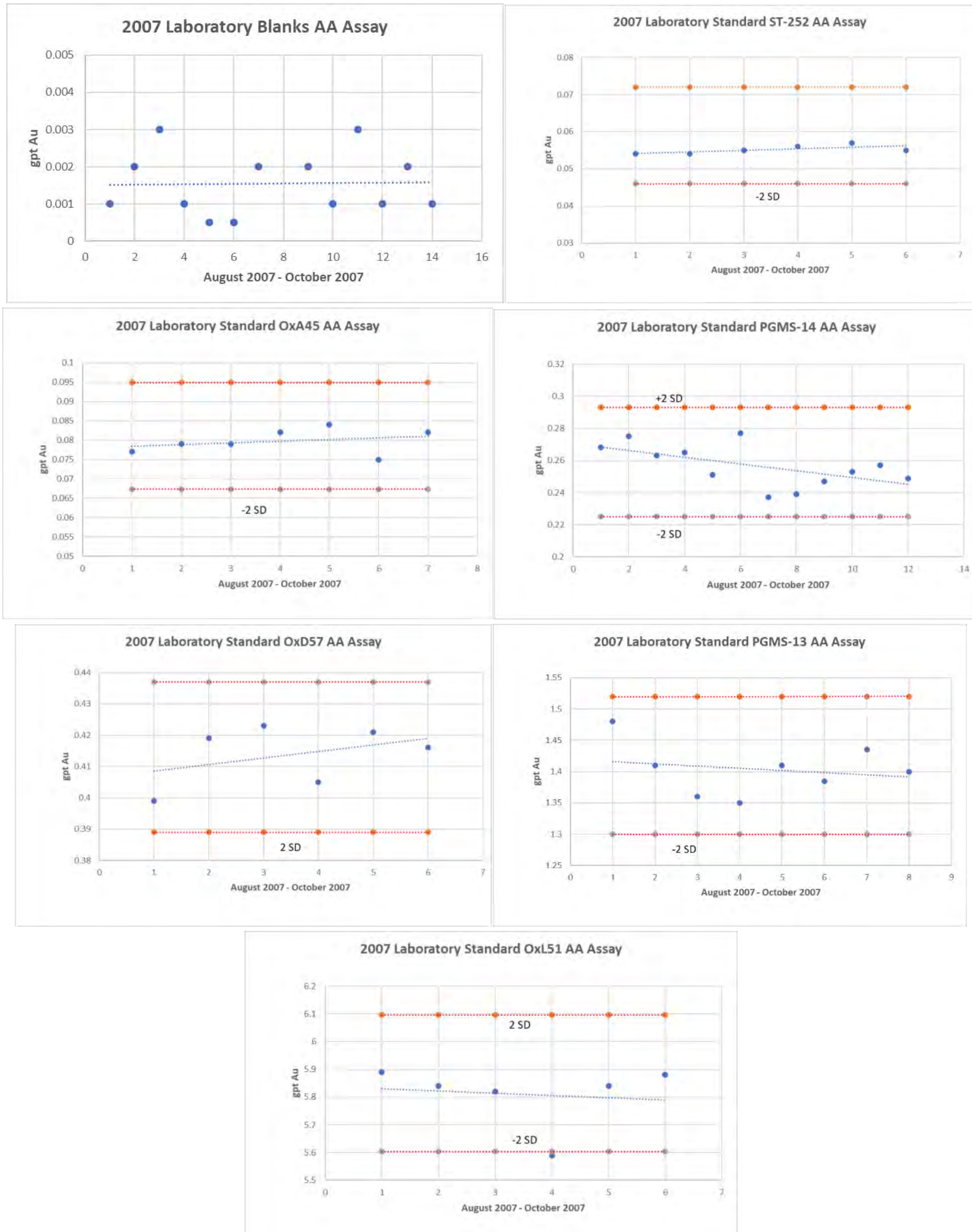
During 2007 to 2012, the laboratories maintained ISO registrations and accreditations, and were registered or pending registration to ISO 9001:2008.

The samples were dried and crushed to 70% passing minus ten (-10) mesh. A Jones riffle splitter was used to take a 250 g subsample for pulverizing, and the reject portion was bagged and stored. After reducing the 250 g sample to 85% passing -200 mesh, the sample was thoroughly blended, and a 50 g charge was assayed for gold by standard fire assay with an inductively coupled plasma (ICP) finish. Gold values in excess of 10 parts per million (ppm) were re-analyzed by fire assay with gravimetric finish for greater accuracy.

Total metallics analysis was completed out on samples with visible gold at the request of the geologist in charge. Core samples were crushed and ground completely so that there was no reject. The sample was screened through a 150 mesh screen, and the plus and minus fractions were weighted. A representative 50 g weight of each fraction was submitted to fire assay for fusion and cupellation followed by gravimetric determination. The total gold content is calculated by weighting the plus and minus fractions and converting this value to oz/t, as described on the SGS fact sheet.

Figure 11-2 presents QA/QC graphs for ALS Chemex Labs from August 2007 to October 2007. The graphs show no issues with assay results for blanks and certified standards. There is one failure on standard OxL51 on assay certificate TB07091584 dated 11 September 2007. The certificate contained 126 samples with three samples having grades between 1.2 g/t Au and 2.1 g/t Au. The other nine standards assayed on the certificate passed.

Figure 11-2: 2007 Laboratory Quality Assurance / Quality Control Results



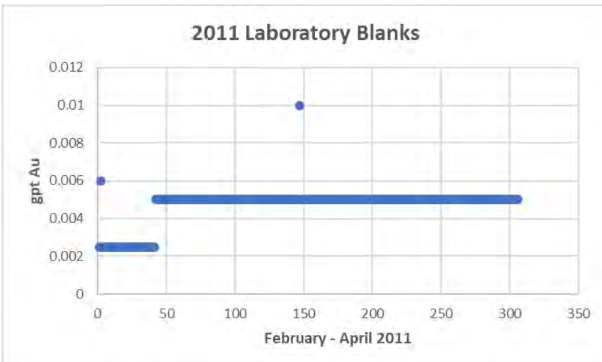
Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; SD = standard deviation.

The following QA/QC graphs from ActLabs cover the period from February 2011 to April 2011 (Figure 11-3). The graphs show no issues with assay results for blanks and certified standards. There was an error on certificate A11-1553 with standards CDN-GS-3H and CDN-GS-P7B being switched. The assay certificates returned on 10 March 2011 and 11 March 2011 had 8 warnings and 2 failures of 17 assays for standard CDN-GS-P3A. The laboratory may have been having issues with this standard. After 11 March 2011, the laboratory was using standards CDN-GS-P2 and CDN-GS-P3A for the low-end standard, with no issues.

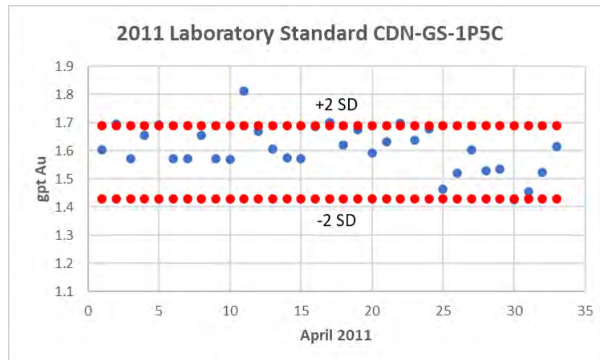
The QP's opinion is that the data supplied by the laboratory is suitable for a resource study.

Figure 11-3: 2011 Laboratory Quality Assurance / Quality Control Results

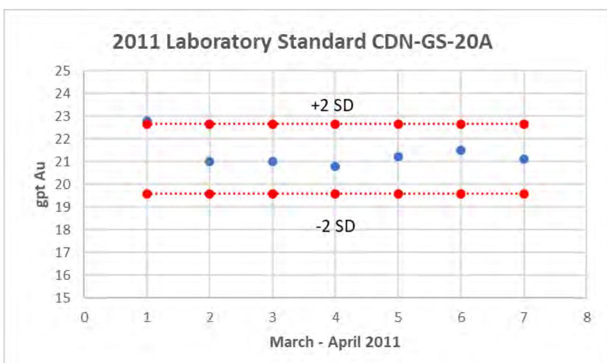
Laboratory Blanks



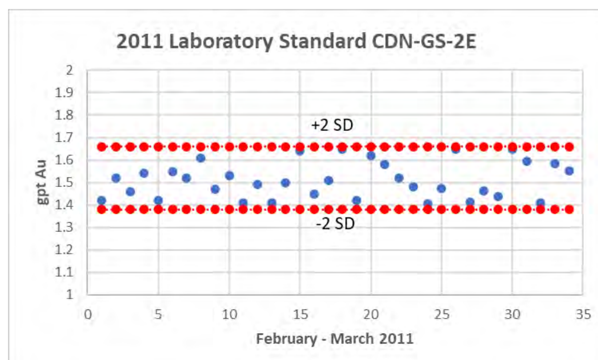
Lab Standard CDN-GS-1P5C



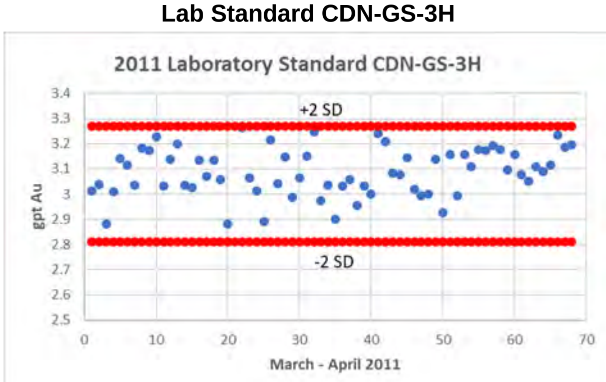
Lab Standard CDN-GS-20A



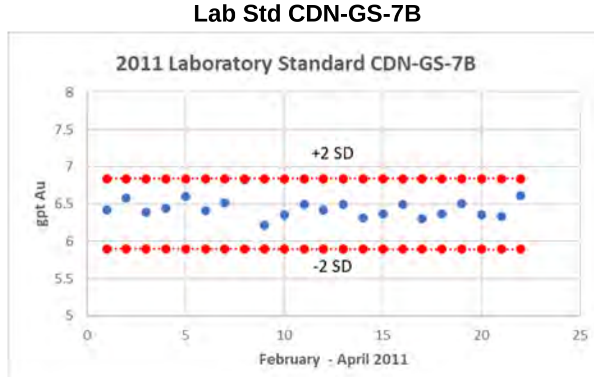
Lab Standard CDN-GS-2E



Lab Standard CDN-GS-3H



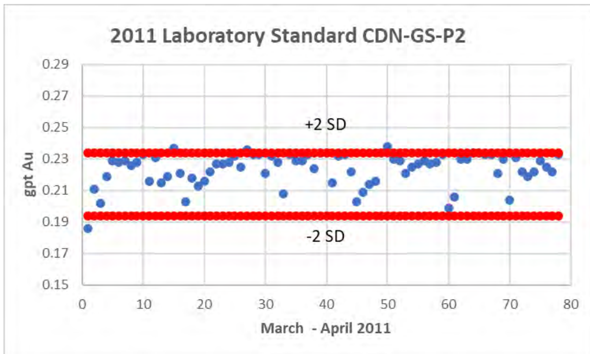
Lab Std CDN-GS-7B



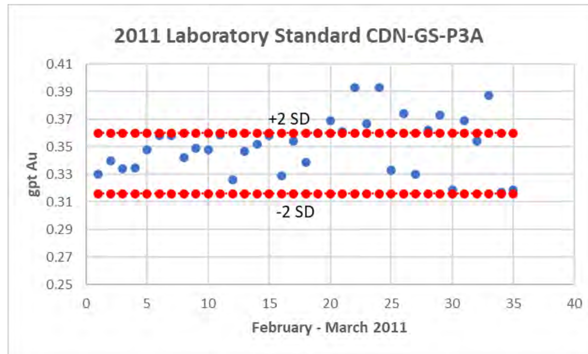
Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; SD = standard deviation.

Figure 11-3: 2011 Laboratory Quality Assurance / Quality Control Results (continued)

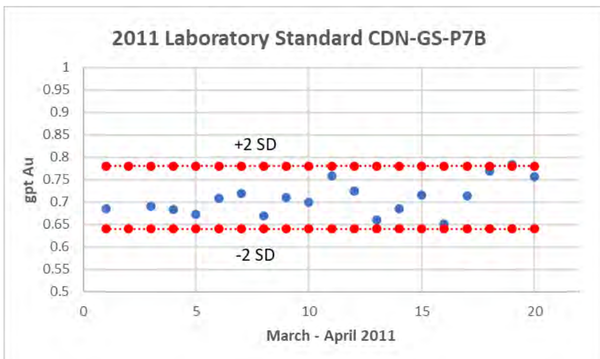
Lab Std CDN-GS-P2



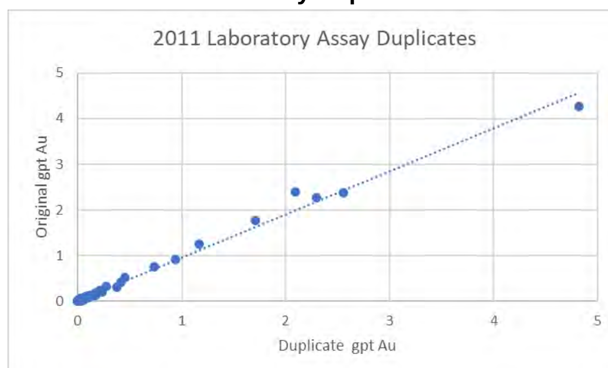
Lab Std CDN-GS-P3A



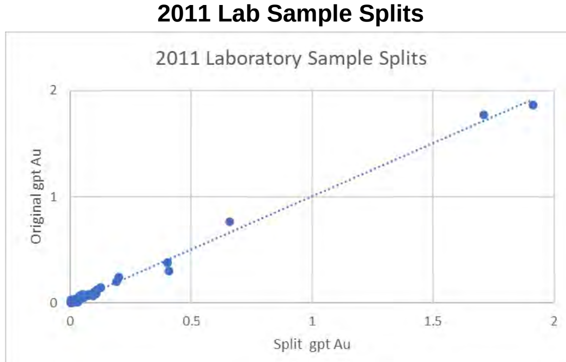
Lab Std CDN-GS-P7B



Lab Assay Duplicates



2011 Lab Sample Splits



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; SD = standard deviation.

During the 2011 diamond drill program, each drill hole was entirely split and sampled. A sample length of 1 m was used. Duplicate samples were systematically selected by the company. The following is a summary of the company QA/QC program for the surface diamond drilling program:

- Insert duplicate sample every 49 samples (including blanks and standards);
- Insert standard every 49 samples (including duplicates and blanks); and
- Insert blank every 49 samples (including duplicates and standards).

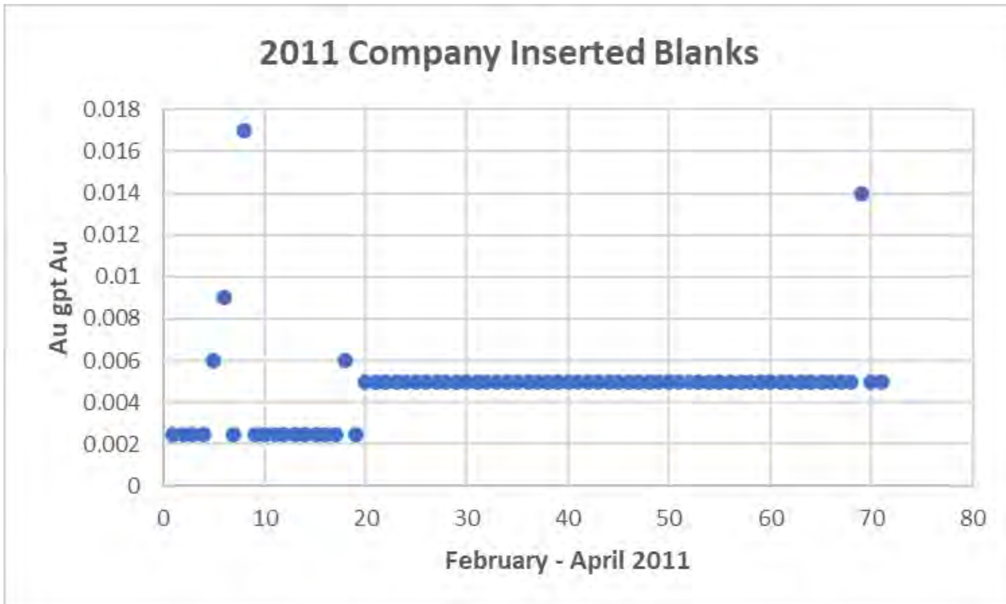
Figure 11-4 illustrates the results of the company QA/QC program. Inserted blanks indicate no issues with contamination during crushing, grinding, fire assay, or measurements.

The certified standards used are unknown. For the Hi Standard, the assayed data averaged 8.06 g/t Au with a double standard deviation of 0.40 g/t Au. The standard deviation was comparable to certified standards where the average gold value was between 6.3 g/t Au and 7.4 g/t Au. There are two warnings on the low side in holes HY-11-02 and HY-11-04; the remaining standard, duplicates, and blanks showed no issue. Laboratory QA/QC showed no issues with the assay certificates. The holes had values lower than 1.0 g/t Au and therefore would not be used in a resource study.

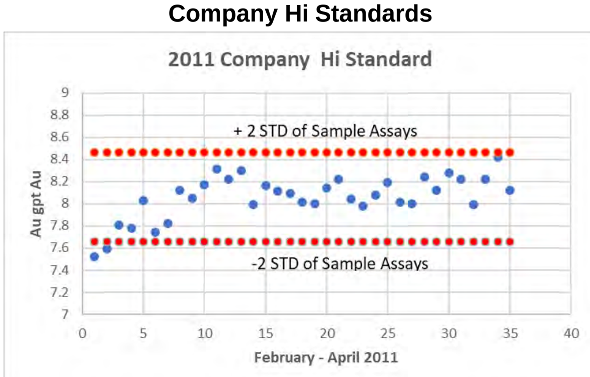
For the Low Standard, the assayed data averaged 2.43 g/t Au with a double standard deviation of 0.32 g/t Au. One outlier value (i.e., 0.018 g/t Au) was not used for the calculation. The standard deviation was slightly higher compared to certified standards where the average gold value was between 1.9 g/t Au and 3.5 g/t Au. There were two warnings on the low side in holes HY-11-09 and HY-11-16; the remaining standard, duplicates, and blanks showed no issue. For HY-11-09, a blank may have been submitted instead of a standard. The company QA/QC showed no issues with the assay certificates.

There are QA/QC procedures recorded and assay certificates available for these surface drill programs. The QP considers the assay data from the Hy Lake programs to be suitable for use in a resource study.

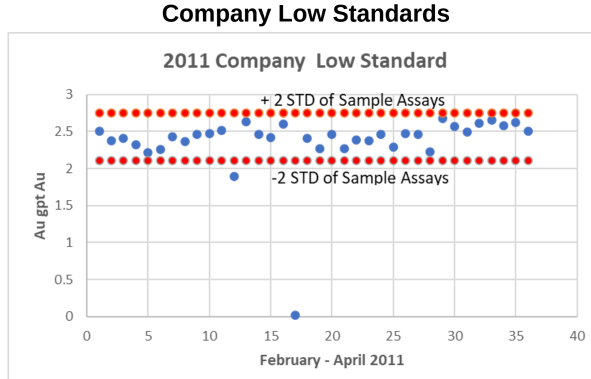
Figure 11-4: 2011 Hy Lake Quality Assurance / Quality Control Results



Company Hi Standards



Company Low Standards



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; STD = standard.

11.1.5 West Red Lake Gold Mines Inc., 2013 to 2022

RLG recorded QA/QC, sample preparation, analyses, and security procedures for drilling completed at the Rowan Mine from 2013 to 2022, as described in Kita, 2022.

During this period, RLG used similar drilling protocols to Hy Lake (Section 11.1.4, Hy Lake, 2007 to 2012). All drill holes were assayed from top to bottom with predominately 1.0 m sample lengths, and 0.5 m sample lengths used on the small vein widths.

RLG maintained its own QA/QC program for the drilling completed in the Rowan Mine area. Certified gold reference standards, blanks, and field duplicates were routinely inserted into the sample stream as part of RLG QA/QC program. Samples were transported directly to the laboratories by company core technicians for sample preparation and analyses. Assaying was completed by either ActLabs or SGS laboratories in Ontario. Both laboratories were independent of RLG. Gold was analyzed by fire assay with atomic absorption spectroscopy (FA-AAS) methods, with a gravimetric assay used for reassays. The laboratories maintained ISO registrations and accreditations, and were registered to International Organization for Standardization and International Electrotechnical Commission (ISO/IEC) 17043:2010.

Sample preparation procedures and total metallics analysis were similar to those used by Hy Lake (Section 11.1.4, Hy Lake, 2007 to 2012).

The company QA/QC was monitored during the assay import into the Geotic software system. Any anomalies were addressed and, if required, reruns were requested by the company geologist.

The QP has reviewed all available QA/QC plots for the various standards and blanks used during the 2013 to 2021 drill campaigns and finds the results satisfactory for use in modern targeting, modelling, and resource estimation.

11.1.6 West Red Lake Gold Mines Ltd., 2023

Samples from drilling completed at the Rowan Property in 2022 and 2023 were transported by WRLG personnel directly to SGS laboratory in Ontario for assay. A total of 24,147 primary samples were collected and submitted for gold assay from the 2022 and 2023 drilling campaigns; all sample results were incorporated into the 2025 mineral resource estimate.

Sample preparation by SGS consisted of drying at 105°C and crushing to 75% passing 2 mm. A Jones riffle splitter was then utilized to produce a 500 g course reject for archive. The remainder of the sample was then pulverized to 85% passing 75 µm from which 50 g was analyzed by FA-AAS. Samples returning gold values greater than 10 g/t Au were reanalyzed by fire assay with a gravimetric finish on a 50 g sample. Samples with visible gold were also analyzed by metallic screen analysis (SGS code: GO_FAS50M). For multi-element analysis, samples were sent to SGS's facility in Burnaby, British Columbia and analyzed via four-acid digest with an atomic emission spectroscopy (ICP-AES) finish for 33-element analysis on 0.25 g sample pulps (SGS code: GE_ICP40Q12). SGS Natural Resources analytical laboratories operate under a QMS that complies with ISO/IEC 17025.

The company QA/QC was monitored during the assay import into the Geotic software system. Any anomalies were addressed and, if required, reruns were requested by the WRLG geologist.

A summary of the QA/QC samples used for the 2022 and 2023 programs is provided in Table 11-2 and Table 11-3, respectively. A summary the QA/QC results from the 2022 and 2023 programs is presented on Figure 11-5 and Figure 11-6, respectively.

It is the QP's opinion that the sampling methods, security, and analytical procedures used were adequate to provide sufficient geotechnical and geological information for the resource study.

Table 11-2: 2022 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Sample Summary

Company	No. of Samples	Standard
WRLG	31	1/4 Duplicate
WRLG	37	Blank
WRLG	22	CDN-CM-40
WRLG	3	CDN-GS-12B
WRLG	3	CDN-GS-7M
WRLG	31	OREAS 18c
WRLG	11	OREAS 204

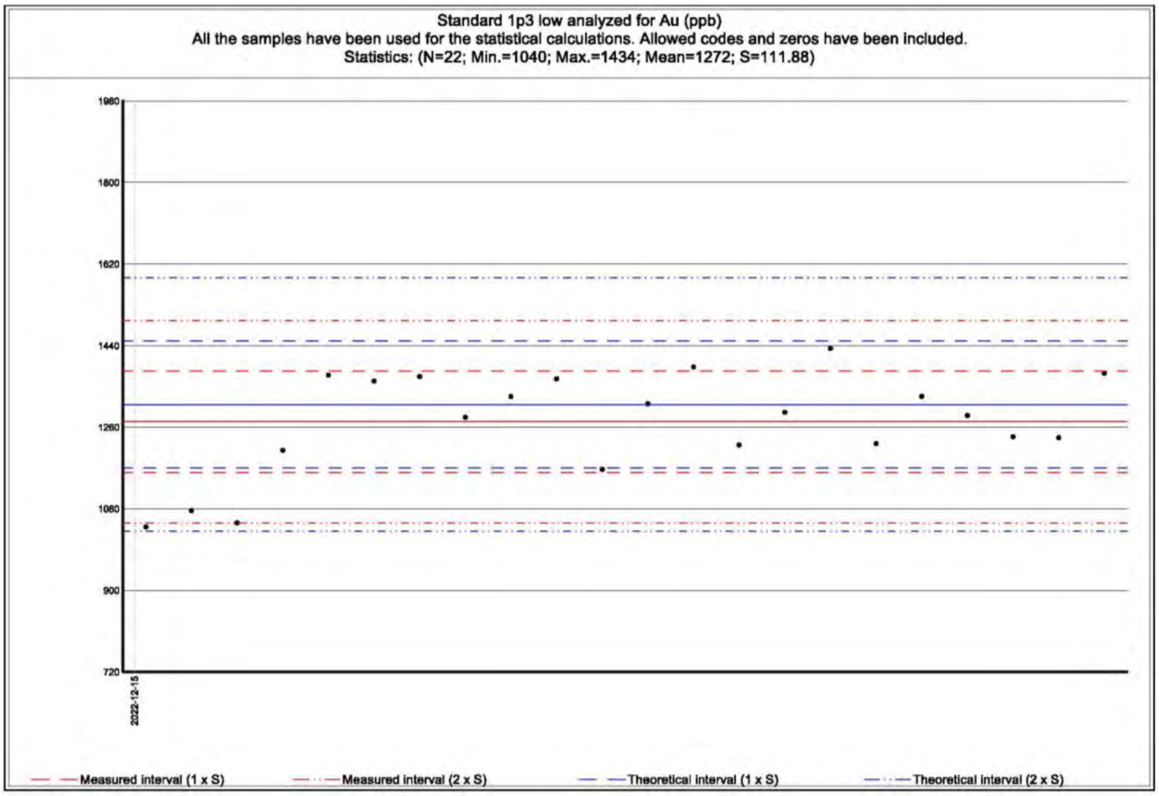
Notes:
WRLG = West Red Lake Gold Mines Ltd.

Table 11-3: 2023 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Sample Summary

Company	No. of Samples	Standard
WRLG	78	CDN-GS-12B
WRLG	51	CDN-CM-40
WRLG	143	CDN-GS-7M
WRLG	226	CDN-GS-1P5W
WRLG	31	OREAS 18c
WRLG	11	OREAS 204

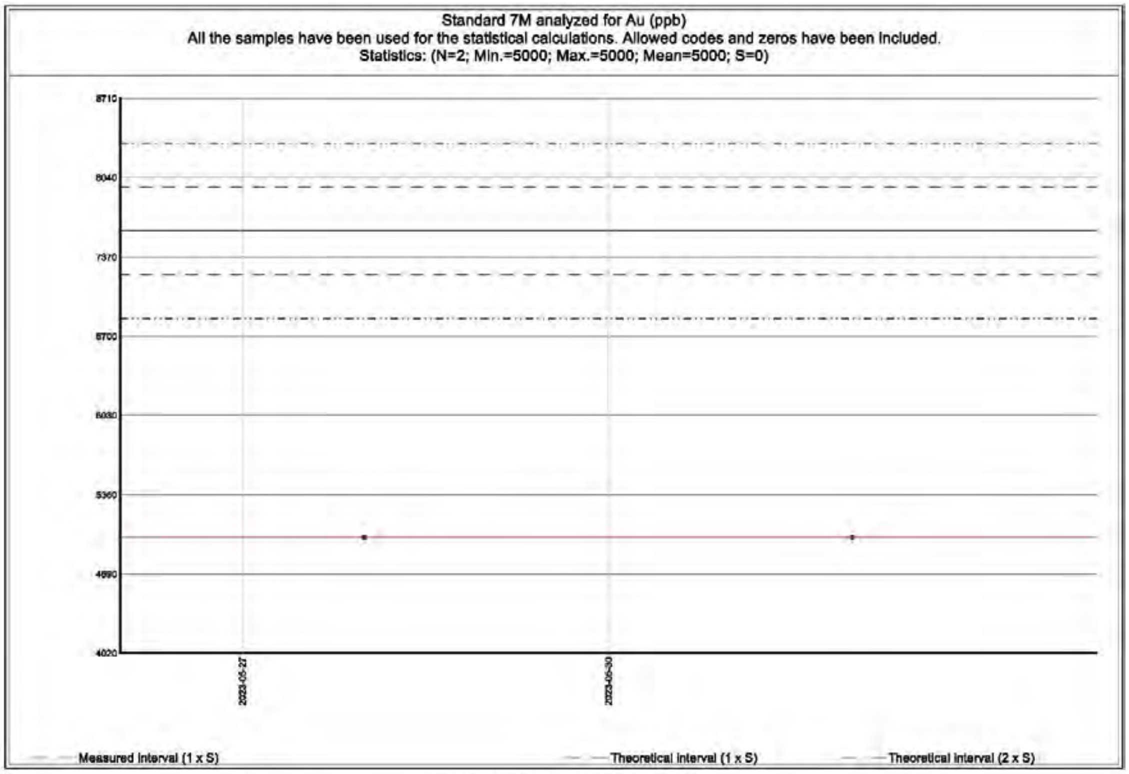
Notes:
WRLG = West Red Lake Gold Mines Ltd.

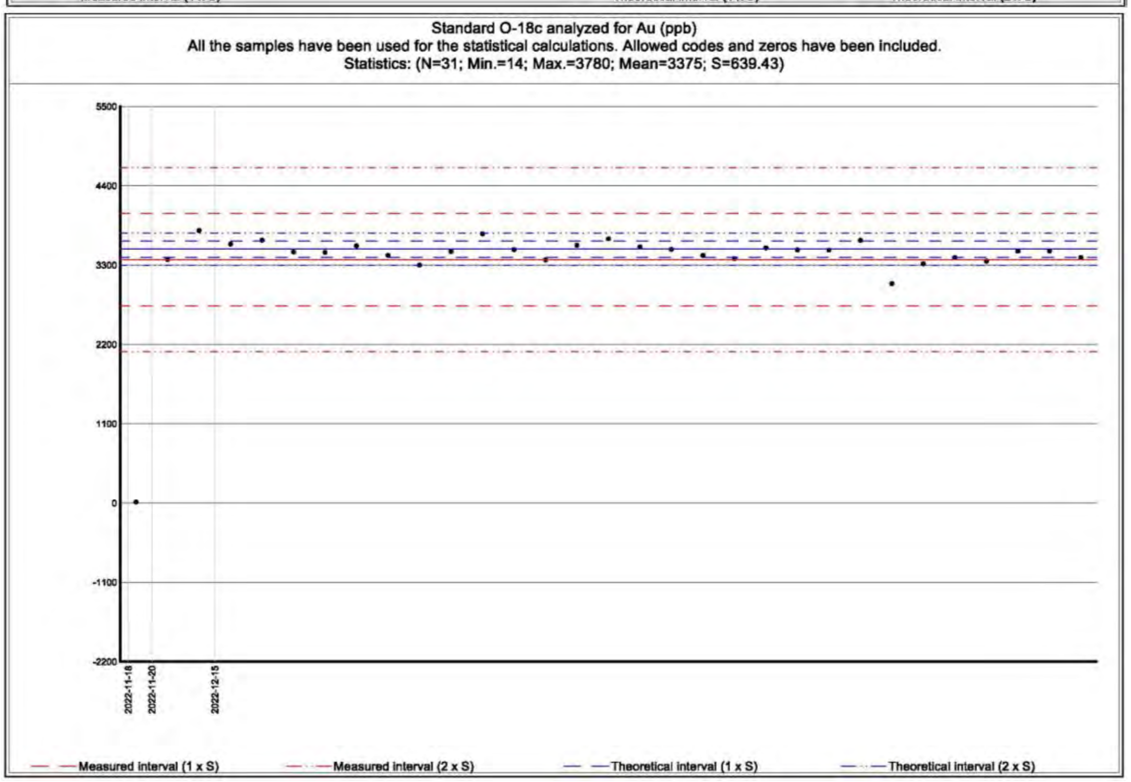
Figure 11-5: 2022 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results



Source: SR 2025.
Notes: Au = gold; ppb = parts per billion.

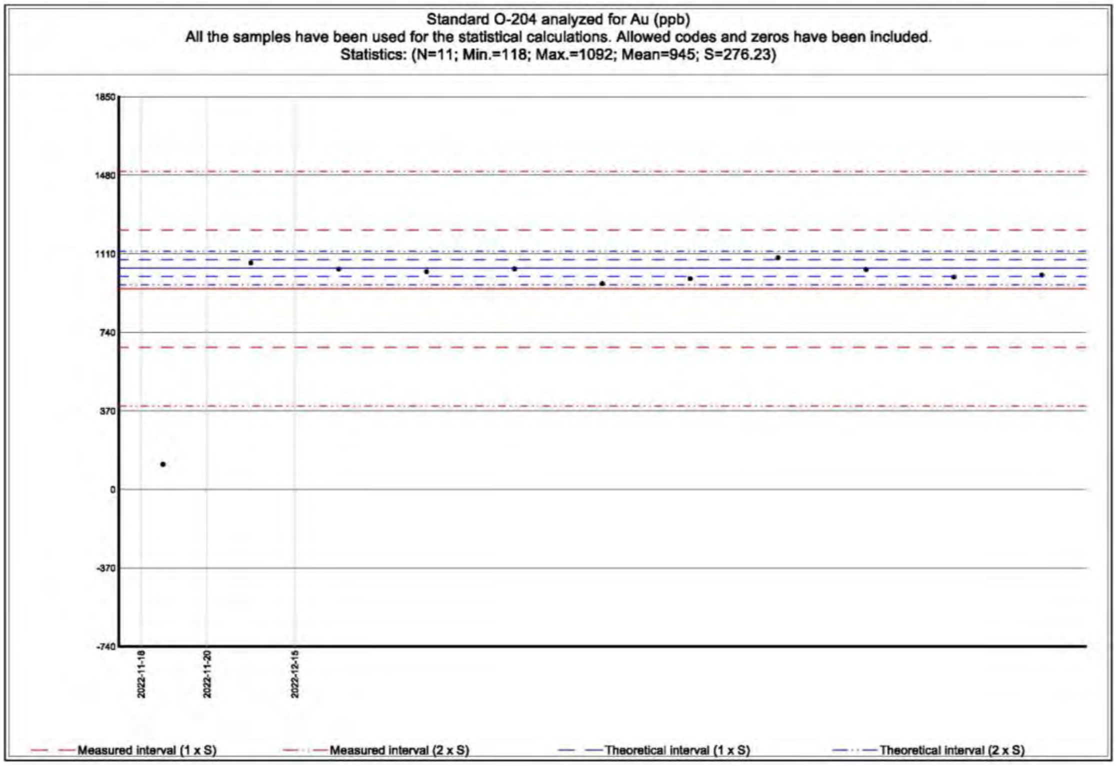
Figure 11-5: 2022 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results (continued)

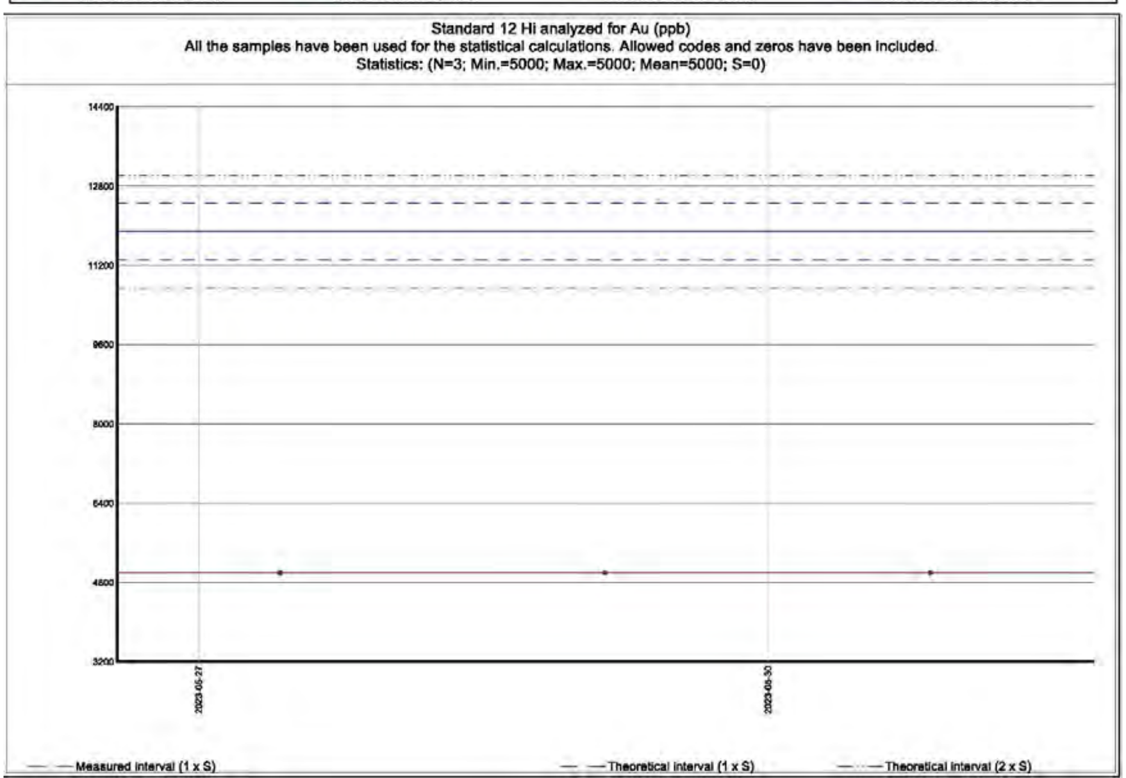




Source: SR 2025.
Notes: Au = gold; ppb = parts per billion.

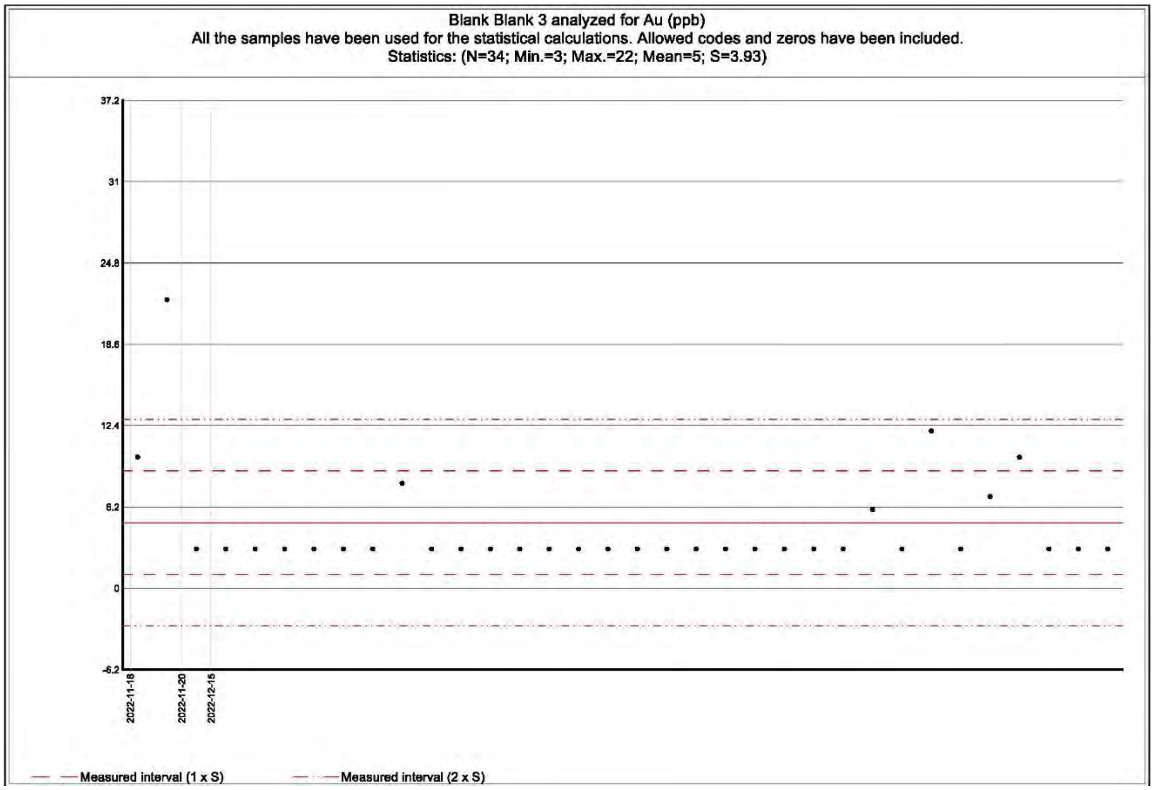
Figure 11-5: 2022 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results (continued)





Source: SR 2025.
Notes: Au = gold; ppb = parts per billion.

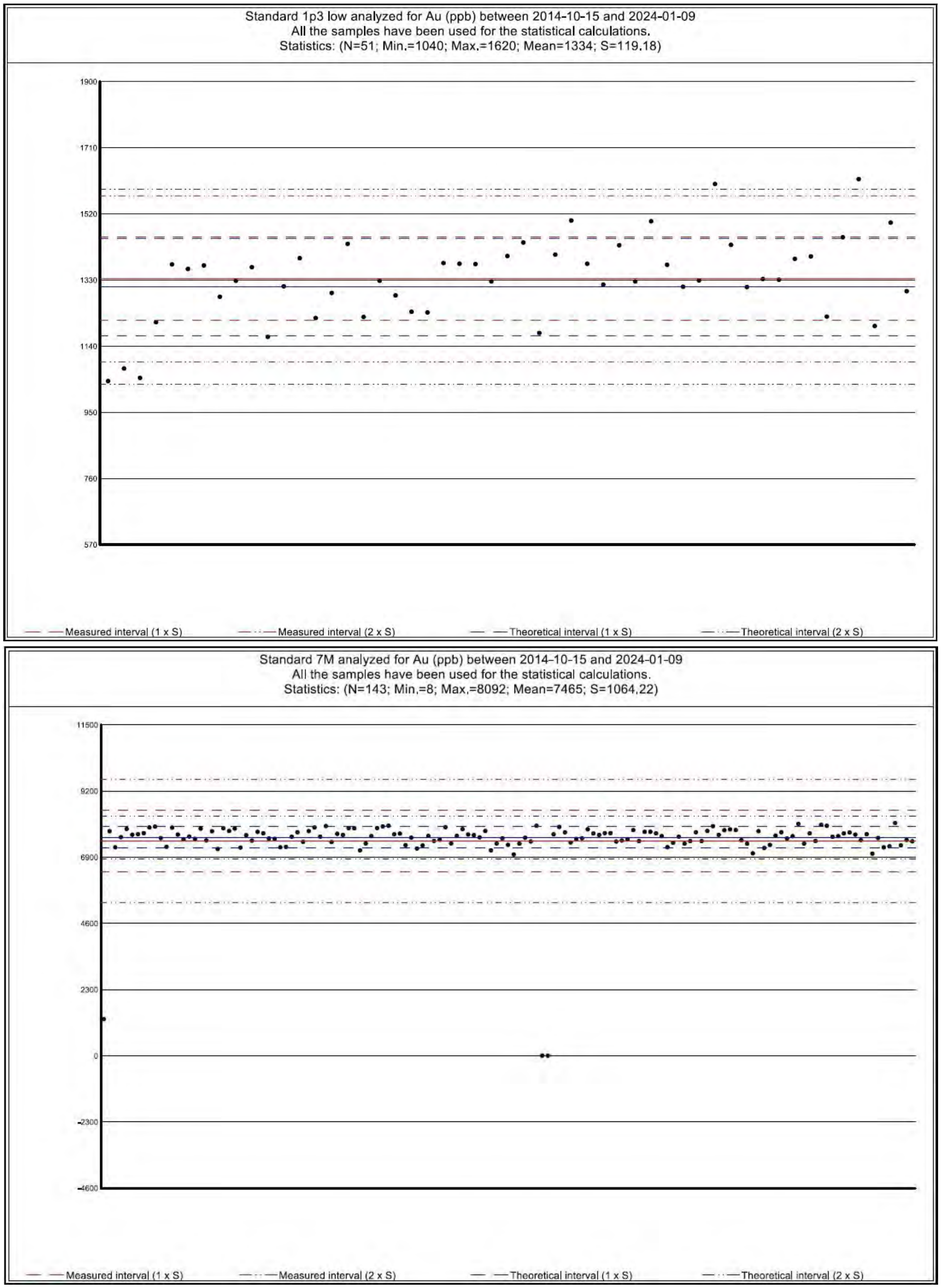
Figure 11-5: 2022 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results (continued)



Source: SR 2025.

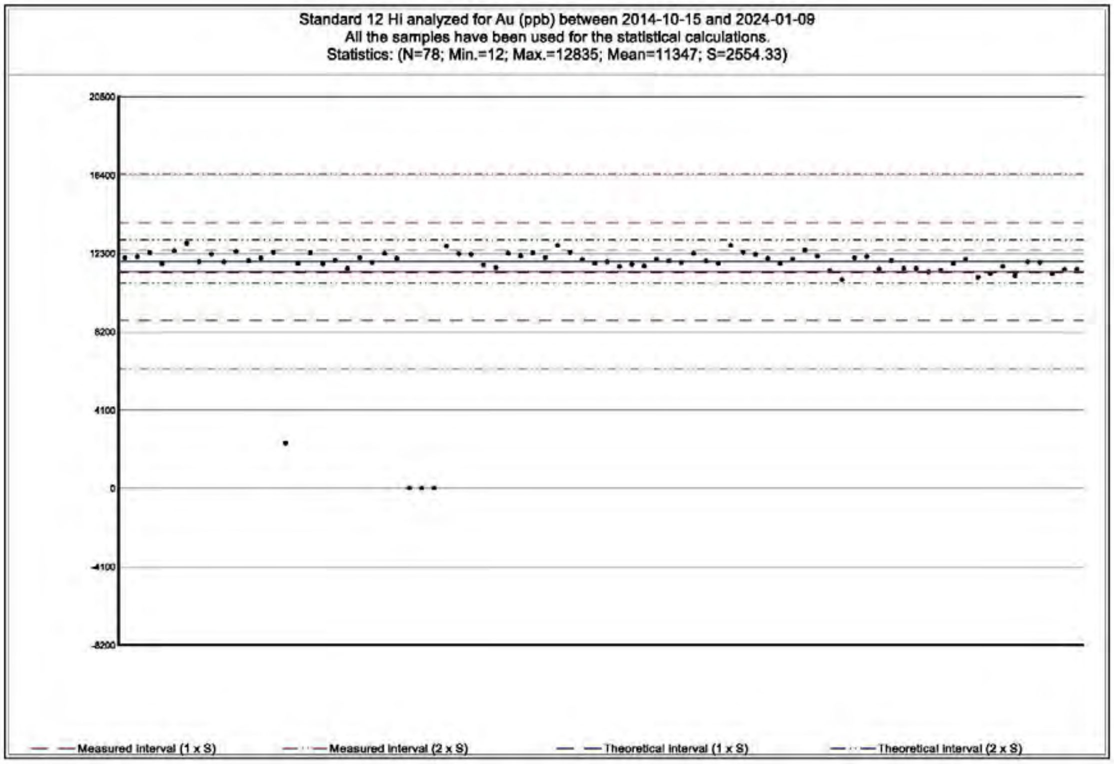
Notes: Au = gold; ppb = parts per billion.

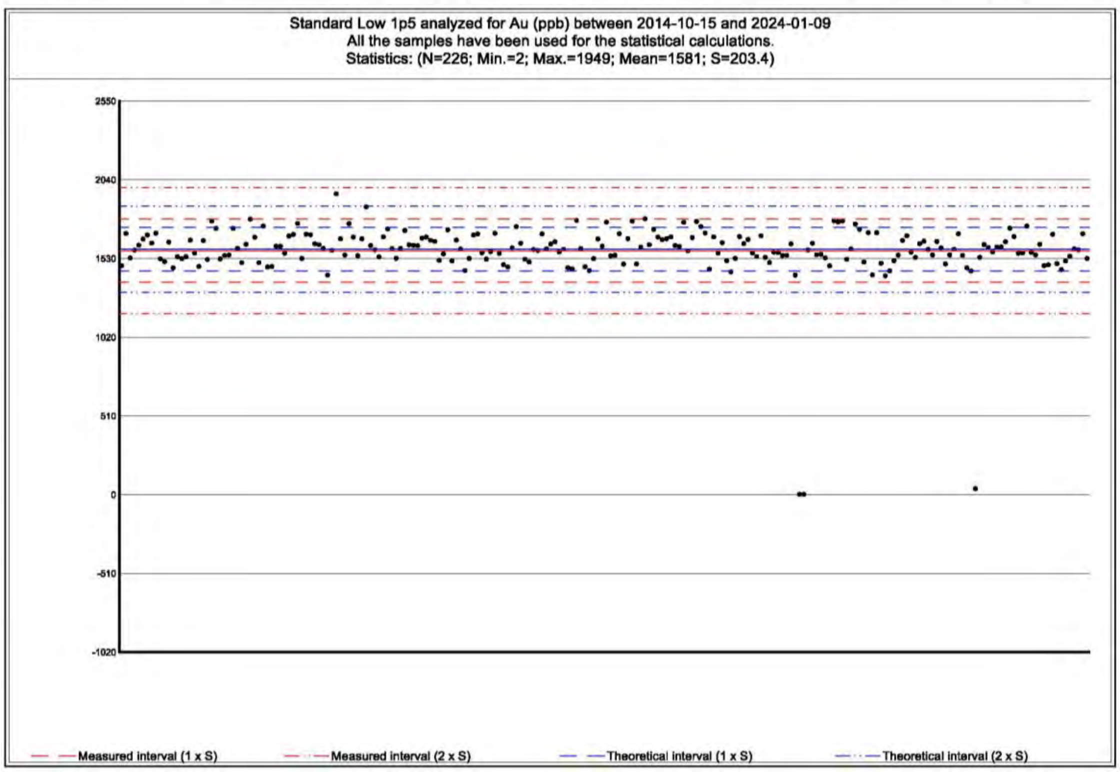
Figure 11-6: 2023 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results



Source: SR 2025.
Notes: Au = gold; ppb = parts per billion.

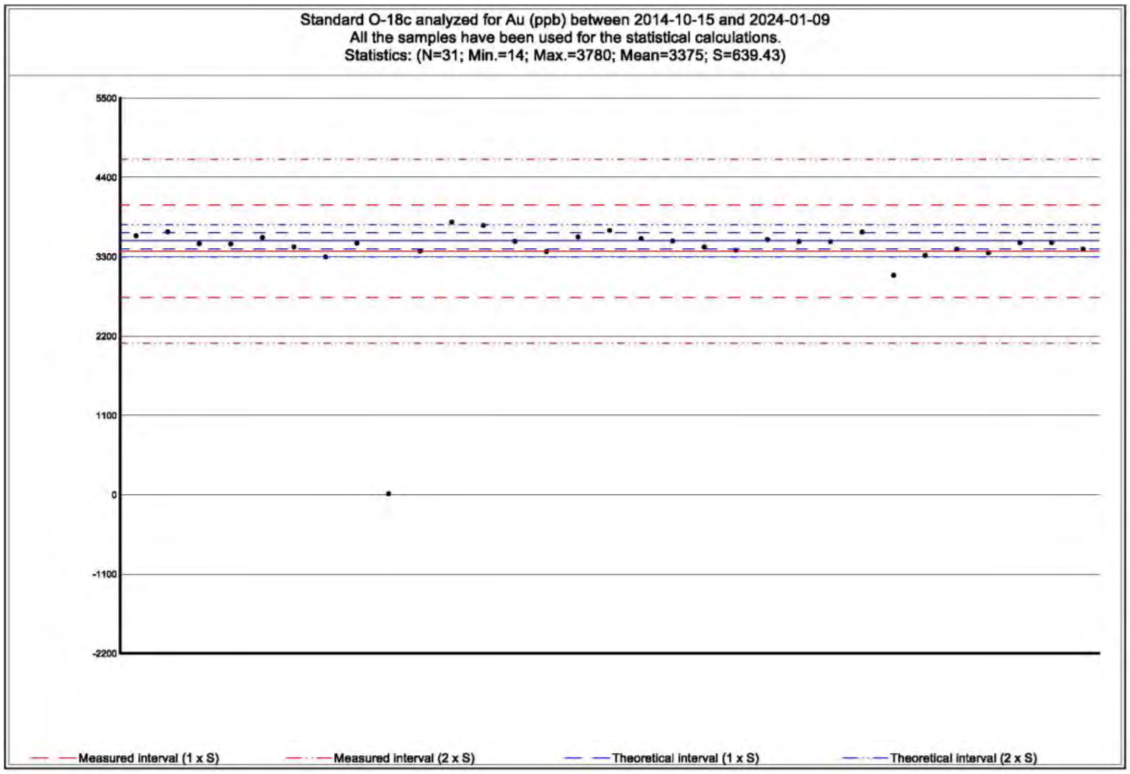
Figure 11-6: 2023 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results (continued)

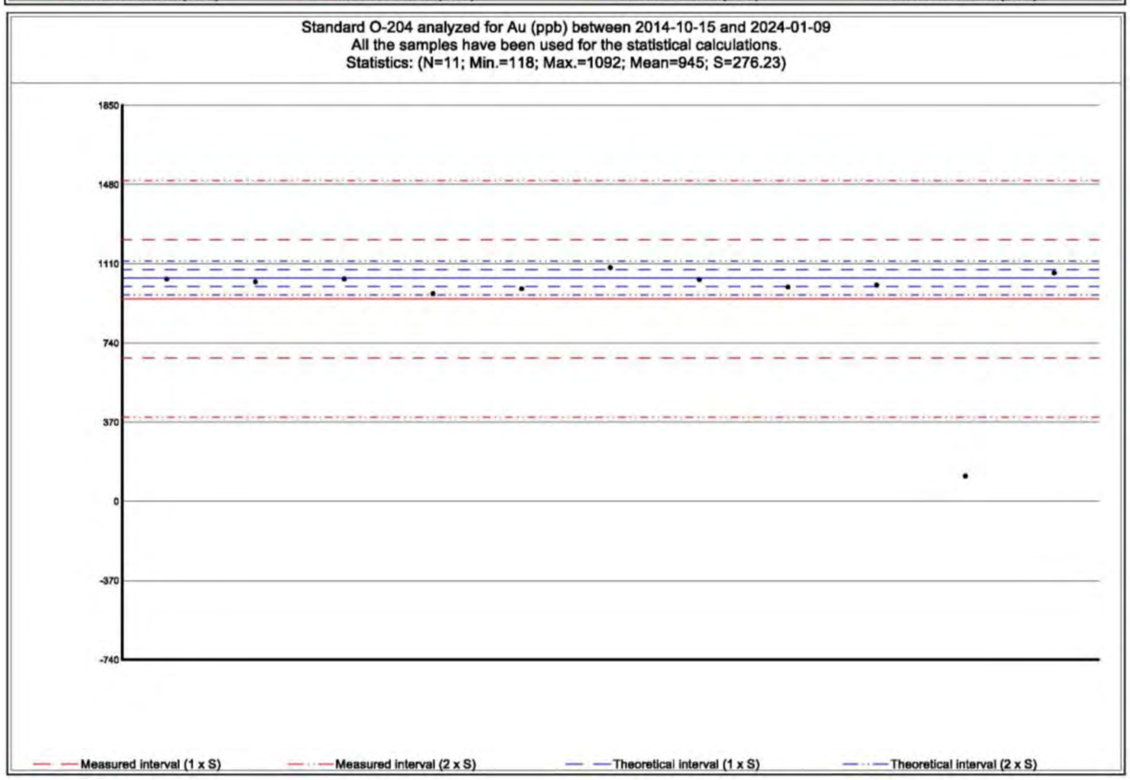




Source: SR 2025.
Notes: Au = gold; ppb = parts per billion.

Figure 11-6: 2023 West Red Lake Gold Mines Ltd. Quality Assurance / Quality Control Results (continued)





Source: SR 2025.
Notes: Au = gold; ppb = parts per billion.

11.2 Mount Jamie Mine Area

This section summarizes historical QA/QC work for Mount Jamie Mine area and is presented here for reference for future work programs.

During the past 82 years, there have been 15 major diamond drilling campaigns conducted at the Mount Jamie Mine. The first records are from 1940 and the most recent are from 2017. Since acquiring the mine, WRLG has not completed any drilling as of the date of this report.

A summary of the number of drill holes, total metres drilled, original units of measurements, assay detection limits, QA/QC programs, and laboratories used is provided in Table 11-4.

The QP has reviewed all available QA/QC plots for the various standards and blanks used during the 1983 to 2017 drill campaigns at Mount Jamie and finds the results satisfactory for use in modern targeting, modelling, and resource estimation.

Table 11-4: Historical Summary of Quality Assurance / Quality Control in Mount Jamie Mine Area (1940 to 2017)

Year	Company	Drill Hole Series in Database	Collar Location	No. of Holes	Total Metres	Core Size	Original Units	Quality Assurance / Quality Control	Assay Detection Limit [g/t Au]	Laboratory
1940	Golden Frontier	GU-1-***	Shaft No. 1 Underground First Level	14	81	-	feet; $ per ton	-	0.69	-
		GU-2-***	Shaft No. 1 Underground Second Level	3	39					
1941	Golden Frontier	GF-***	Shaft No. 2 Surface	7	123	-	feet; $ per ton	-	0.34	-
		GU-1-***	Shaft No. 1 Underground First Level	39	489					
		GU-2-***	Shaft No. 1 Underground Second Level	38	333					
		GU-3-***	Shaft No. 1 Underground Third Level	16	252					
		GU-4-***	Shaft No. 1 Underground Forth Level	13	191					
1942	Golden Frontier	GF-***	North Vein Surface	4	102	-	feet; $ per ton	-	0.34	-
		GF-1-2**	Shaft No. 1 Underground First Level	7	107					
		GU-2-233	Shaft No. 1 Underground Second Level	1	23					
		GU-4-230	Shaft No. 1 Underground Forth Level	1	23					
1945	Bayview Red Lake	BW-**	Shaft No. 1 Surface	20	2,476	-	feet; oz/t	-	0.34	ALS Chemex
			Shaft No. 2 Surface	5	393					
			Other	9	1,724					
1977	Byng Red Lake	HL-77-01,02,03	South of Shaft No. 2 north of Dupont Lake	3	97	EXT	feet	No assays.	-	-
1978	Byng Red Lake	HL-78-04,05	South of Shaft No. 2 north of Dupont Lake	2	35				-	-
1982	Oneiro-Alfa	Not in database	Shaft No. 1 Shaft No. 2	19	1,646	-	-	-	-	-
1983	Keeley Frontier	KF-83-**	Shaft No. 2 Surface	6	396	BQ	metric; oz/t	50% of samples. Samples greater then 3.0 g/t had multiple reassays; 2% of samples reassayed.	0.34	Cochenour Fire Assay
		KF-83-**	Shaft No. 1 Surface	22	2,168	BQ	metric; oz/t			
		KU-1**	Shaft No. 1 Underground First Level	28	1,050	AQ	feet; oz/t	-		
		KU-2**	Shaft No. 1 Underground Second Level	8	365	AQ	feet; oz/t			
		KU-40*	Shaft No. 1 Underground Forth Level	2	168	AQ	feet; oz/t			
1984	Robert Gibson	RG-84-**	South of Shaft No. 2 north of Dupont Lake	10	313	XRT	feet	No assays.	-	-
1985	Jamie Frontier	JF-31 - 43	Shaft No. 1 Surface	13	721	BQ	feet; oz/t	5% of samples greater then 3 g/t Au had screened metallics assays; 96% of samples. Samples greater then 1.0 g/t Au had multiple reassays 16% of samples reassayed.	0.068	Swastika Lab
1985	Jamie Frontier	JU-110 - 141	Shaft No. 1 Underground First Level	32	1,041	BQ	feet; oz/t	66% of samples greater then 1.5 g/t Au had screened metallics assays; 88% of samples greater then 1.5 g/t Au had multiple reassays; 19% of samples reassayed.	0.068	Cochenour Fire Assay / Swastika
		JU-210 - 240	Shaft No. 1 Underground Second Level	31	1,189					
		JU-310 - 330	Shaft No. 1 Underground Third Level	21	1,086					
		JU-410 - 434	Shaft No. 1 Underground Fourth Level	24	1,794					
1985	Robert Gibson	Not in database	South of Shaft No. 2 north of Dupont Lake	2	62	XRT	feet	-	-	-
1986		RG-86-**	South of Shaft No. 2 north of Dupont Lake	9	288					
1986	Jamie Frontier	JF-44 - 92	Shaft No. 2 Surface, North Vein, Other	50	2,752	BQ	feet; oz/t	83% of samples greater then 4.5 g/pt had screened metallics assays; samples greater then 1.0 g/t had 74% multiple reassays and 83% of samples >4.5 g/t analyzed with screened metallics; 10% of total samples duplicated.	0.068	Cochenour Fire Assay / ALS Chemex

Table 11-4: Historical Summary of Quality Assurance / Quality Control in Mount Jamie Mine Area (1940 to 2017) (continued)

Year	Company	Drill Hole Series in Database	Collar Location	No. of Holes	Total Metres	Core Size	Original Units	Quality Assurance / Quality Control	Assay Detection Limit [g/t Au]	Laboratory
1986	Jamie Frontier	JU-142 - 149	Shaft No. 1 Underground First Level	8	572	BQ	feet; oz/t	65% of samples greater then 3 g/t had screened metallics assays; 97% of samples. Samples greater then 3.0 g/t Au had multiple reassays; 19% of samples reassayed.	0.068	Cochenour Fire Assay / Swastika
	Jamie Frontier	JU-241 - 246	Shaft No. 1 Underground Second Level	6	804					
	Jamie Frontier	JU-416,435,436	Shaft No. 1 Underground Fourth Level	3	715					
1987	Robert Gibson	RG-87-**	South of Shaft No. 2 north of Dupont Lake	11	385	EXT	feet	No assays.	-	-
1987	Jamie Frontier	JU-437,438,439	Shaft No. 1 Underground Fourth Level	3	524	BQ	feet; oz/t	7% of samples duplicated	0.069	Cochenour Fire Assay
1987	Byron Bay	BB87-1	North West of Shaft No. 2	1	375	BQ	feet; ppb	-	0.005	-
1988	Robert Gibson	RG-88-**	South of Shaft No. 2 north of Dupont Lake	3	113	EXT	feet	No assays.	-	-
1989	Pezgold	P, PSE, PSW, PW	North Vein and East of North Vein	39	3,683	NQ	feet; oz/t	4% of samples duplicated	0.034	-
2003	Zenda/ Vedron	JF-03-**	Shaft No. 2 Area and west	6	900	NQ	metric; g/t	50% of samples greater then 10 g/t had screened metallics assays; 67% of samples greater then 1.0 g/t had multiple reassays; 18% of samples reassayed.	0.01	ALS Chemex
2007	Hy Lake	HY-07-**	Shaft No. 2 Area, east west along Strike	38	7,687	NQ	metric; oz/t/g/t	Lab standards and duplicates. 25% of samples greater then 0.75 g/t had screened metallics assays; 5% of samples reassayed.	0.01/.001	SGS / ALS Chemex
2011	Hy Lake	HY-11-**	Outside of Shaft No. 1 Area	31	3,490	NQ	metric; oz/t/g/t	Company and lab QA/QC Program in place, 100% core sampled cupellation followed by FA-AAS (for >5 g/t Au samples rerun with gravimetric).	0.005/0.01	Actlabs
2012	Hy Lake	HY-12-**	Property Wide	32	5,212	NQ	metric; oz/t/g/t	Company QA/QC program in place 100% core sampled, FA-AAS finish used, for >5 g/t Au samples screened metallics or gravimetric finish.	0.01	Actlabs
2017	RLG	MJ-17-**	Shaft No. 1 Area and North Vein	15	1,893	NQ	metric; g/t	Company QA/QC program in place, insertion of blanks and standards.	0.005	SGS

Notes:
oz/t = ounces per tonne; g/t = grams per tonne; ppb = parts per billion; Au = gold; - = unknown; SGS = SGS Natural Resources; > = greater than; FA-AAS = fire assay with atomic absorption spectroscopy; QA/QC = quality assurance and quality control; RLG = West Red Lake Gold Mines Inc.

11.2.1 Golden Frontier, 1940 to 1942

Golden Frontier conducted surface and underground drilling at the Mount Jamie Mine from 1940 to 1942. The drilling explored Shaft No. 1, Shaft No. 2, and the North Vein, with the majority of the drilling completed on Shaft No. 1. The assay information was recovered from the existing diamond drill logs. Distance measurements on the logs were in feet and assays recorded in dollars per ton. The assay values were converted to ounces per ton by multiplying the dollar value by 0.02853. There were no recorded duplicates listed on the drill logs.

The company and laboratory QA/QC programs and procedures for diamond drilling and assaying are not recorded. However, it appears they may have limited the sample length to obtain a representative sample. The surface drilling had an average sample length of 0.71 m, with a maximum length of 1.22 m and a minimum length of 0.15 m. The underground drilling program had an average sample length of 0.77 m, with a maximum length of 2.29 m and a minimum length of 0.15 m. All underground drilling samples grading above 3 g/t Au averaged 0.66 m in length.

The company QA/QC program and procedures for underground sampling are not recorded. Sample locations and assay results were recorded using historical plans, sections, and longitudinal views.

The laboratory QA/QC program is unknown. Assays may have been completed in an on-site assay office, or the samples may have been shipped out to an independent laboratory. Based on the assay results, the fire assay with gravimetric finish had a detection limit of 0.34 g/t Au. Based on this detection value, trace and nil values were recorded as 0.001 g/t rather than using 0.17 g/t (i.e., equal to half the detection limit), which would be anomalous using current measuring technology.

Based on the lack of recorded QA/QC procedures and assay certificates, the QP considers the assay data from this program suitable to either limit mineralized zones or be used with assay results from more recent drilling. If a zone is solely defined by these drill holes, the zone should be considered inferred until confirmed with more recent results.

11.2.2 Bayview Red Lake, 1944 to 1945

Bayview Red Lake conducted surface drilling at the Mount Jamie Mine from 1944 to 1945. The drilling explored Shaft No. 1, Shaft No. 2, North Vein, and other areas, with the majority of drilling completed on Shaft No. 1. The assay information was recovered from the existing diamond drill logs. Distance measurements on the logs were in feet and assays recorded in ounces per ton. There were no recorded duplicates listed on the logs.

The company QA/QC program and procedures for diamond drilling are not recorded. The surface drilling had an average sample length of 1.07 m, with a maximum length of 8.84 m and a minimum length of 0.04 m. Samples grading above 3 g/t Au averaged 0.49 m in length.

The laboratory QA/QC program is unknown. Drill hole number BW-33 listed ALS Chemex as the laboratory used for assaying the core samples. Based on the assay results, fire assay with gravimetric finish had a detection limit of 0.34 g/t Au. All trace and nil values were recorded as 0.001 g/t rather than using 0.17 g/t (i.e., equal to half the detection limit), which would be anomalous using current measuring technology.

Based on the lack of recorded QA/QC programs and assay certificates, the QP considers the assay data from this program suitable to either limit mineralized zones or be used with assay results from more recent drilling. If a zone is solely defined by these drill holes, the zone should be considered inferred until confirmed with more recent results.

11.2.3 Byng Red Lake, 1977 to 1978

Byng Red Lake conducted surface drilling 400 m south of Shaft No. 1 and north of Dupont Lake. No drill core samples were recorded.

The claims were held by Byng Red Lake and have since been acquired by WRLG.

11.2.4 Oneiro-Alfa Red Lake, 1982

Oneiro-Alfa conducted surface drilling at the Mount Jamie Mine in 1982. Nineteen drill holes totalling 1,646 m were completed. These holes are not in the database, and no records are available.

11.2.5 Keeley Frontier, 1983

Keely Frontier conducted surface and underground drilling at the Mount Jamie Mine in 1983. The surface drilling explored the Shaft No. 1 and Shaft No. 2 areas. Underground drilling took place at the Shaft No. 1 area. The assay information was recovered from the existing diamond drill logs. The gold assaying was performed at Cochenour Fire Assay in Cochenour, Ontario. Based on the assay results, fire assay with gravimetric finish had a detection limit of 0.34 g/t Au. All trace and nil values were recorded as 0.001 g/t rather than using 0.17 g/t (i.e., equal to half the detection limit), which would be anomalous using current measuring technology.

The company QA/QC program and procedures for underground sampling are not recorded. Sample locations and assay results were recorded using historical plans, sections, and longitudinal views, and the data was transferred into the database. Assay information was transferred from existing drill logs. Original measurement units on the drill logs were imperial lengths and assay results were reported as ounces per ton of gold. Sample lengths were limited to improve the accuracy of the assay by reducing the nugget effect within the sample. The underground drilling had an average sample length of 0.31 m, with a maximum length of 0.49 m and a minimum length of 0.06 m. A historical report (Vamos 2003) states that the underground sampling by Keeley Frontier confirmed the earlier (i.e., 1940 to 1942) sampling; there are no recorded comparisons or data available to support this statement.

The company and laboratory QA/QC programs and procedures for the surface diamond drilling program are not recorded. Assay information was transferred from existing drill logs. Original measurement units on the drill logs were metric lengths and assay results reported as ounces per ton of gold. Sample lengths were limited to improve the accuracy of the assay by reducing the nugget effect within the sample. The surface drilling had an average sample length of 0.29 m, with a maximum length of 1.30 m and a minimum length of 0.15 m. Duplicate assays were completed through instructions of the company or as part of the laboratory QA/QC and recorded on the drill logs. A total of 2% of all samples were duplicated, and 50% of the original samples grading greater than 3.0 g/t Au were duplicated. The duplicate assay results indicated no bias or reproducibility issues with the original assay results.

11.2.6 Robert Gibson, 1984 to 1988

Robert Gibson drilled a series of surface diamond drill holes south of Shaft No. 2 in the Mount Jamie Mine area on claims not held by the company. There are no recorded assays. WRLG has since acquired these claims.

11.2.7 Jamie Frontier, 1985 to 1987

Jamie Frontier conducted surface drilling, underground drilling, and underground sampling at the Mount Jamie Mine from 1985 to 1987.

The surface and underground drilling explored the Shaft No. 1, Shaft No. 2, and North Vein areas. The drill hole assay information was recovered from the existing diamond drill logs. Based on historical descriptions (Vamos 2003), core recovery was excellent (i.e., 90% or better) in most cases, therefore the accuracy as well as the reliability of the results was considered high. Drill core sampling included all mineralized zones with additional material taken from the wall rock on either side of the mineralization. The core samples were split using a regular core splitter. Half of the core became the sample, and the other half of the core was retained in labelled core boxes for future reference.

Assay information was transferred from existing drill logs. Original measurement units on the drill logs were imperial lengths and assay results reported as ounces per ton of gold. Sample lengths were limited to improve the accuracy of the assay by reducing the nugget effect within the sample. A summary of sample lengths used in surface and underground drilling in the 1985 to 1987 programs is provided in Table 11-5.

Table 11-5: Drill Sample Lengths Used in Jamie Frontier Drill Programs in the Mount Jamie Mine Area

Program	Average [m]	Minimum [m]	Maximum [m]
Surface			
1985	0.51	0.30	0.92
1986	0.32 (0.30 for samples >0.35 g/t Au)	0.23	0.61 (0.31 for samples >0.35 g/t Au)
Underground			
1985	0.42 (0.36 for samples >1.0 g/t Au)	0.30	0.61
1986 to 1987	0.33	0.16	0.91

Notes:
m = metre; > = greater than; g/t = grams per tonne; Au = gold.

During the 1986 to 1987 drilling programs of the new North C Zone, it was suspected that the assays were not up to the expected grade based on visual observations, especially in those locations where fine granular gold was seen. To overcome the potential error, the entire core sample was crushed and pulverized by the assayer and fine screened before assaying. Any granular gold found by this process was reported and the assay was completed accordingly.

For the underground sampling, sample locations and assay results were recorded on plans, sections, and longitudinal views. Sample locations were measured, and the assays were recorded into the database. The underground workings were resampled except for areas of the underground workings that were considered unsafe. The purpose of this underground sampling program was to verify the values and widths of the gold-bearing zones. The method used was chip sampling across the backs of the drifts. The wall rock was separated from the vein samples on both sides of the vein, resulting in a minimum of three individual samples at each location. Samples of the drift backs, face, and walls of the new drives were collected at each round taken, and mapping of backs and faces was completed simultaneously with the sampling. The individual samples were typically less than 1 m in length. Several hundred samples were taken on each level, and the results closely resembled the original sampling completed by A. H. Honsberger in 1941 (Table 6-3). No tables or comparisons of actual data are available.

The samples were bagged, tagged, and packed in cardboard boxes that were taped shut using packing tape. These samples were either shipped by the company to the Cochenour Assay Laboratory, in Cochenour, Ontario, or were shipped via Bus Express to Swastika Assay Laboratory (Swastika); both laboratories were certified by the Canadian Testing Association at the time.

Gold assaying was completed at Cochenour Assay Laboratory for the drill cutting samples and Swastika for the core samples. Based on the assay results, fire assay with gravimetric finish had a detection limit of 0.068 g/t Au. All trace and nil values were recorded as 0.001 g/t rather than using 0.017 g/t Au (i.e., equal to half the detection limit).

The company and laboratory QA/QC programs and procedures are recorded for the surface and underground diamond drilling programs in Vamos (2003).

Duplicate assays were completed through instructions of the company or as part of the laboratory QA/QC and recorded on the drill logs. Samples with visible gold or suspected mineralized zones were assayed using screened metallics of the plus and minus fractions through a 100 mesh screen. The amount of material used for screening is unknown; all samples with screened metallic assay results also included multiple regular assay results.

There are no company inserted blanks or standards; at the time of this work, a company’s use of blanks and certified standards was not commonplace. The company did mitigate the risk of inaccurate assays by limiting sample length, duplicating samples, and using the screened metallics assay method.

1985 Quality Assurance / Quality Control

The following is a summary of the QA/QC program for the 1985 surface diamond drilling program:

- 19% of all samples duplicated.
- 65% of samples assaying greater than 3 g/t Au also had a screened metallic assay recorded.
- 96% of samples assaying greater than 3 g/t Au had multiple duplicate results recorded.

The following is a summary of the QA/QC program for the 1985 underground diamond drilling program:

- 19% of all samples were duplicated.
- 3% of all samples had screened metallic analysis.
- 66% of samples assaying greater than 3.4 g/t Au had a screened metallic assay recorded.
- 88% of samples assaying greater than 1.50 g/t Au had multiple duplicate results recorded.

1986 to 1987 Quality Assurance / Quality Control

The following is a summary of the QA/QC program for the 1986 to 1987 surface diamond drilling program:

- 10% of all samples were either duplicated or had screened metallic analysis.
- 83% of samples assaying greater than 4.5 g/t Au were analyzed using screened metallic assay method.
- 74% of samples assaying greater than 1.0 g/t Au have multiple duplicate results recorded.

There are no recorded QA/QC procedures or assay certificates available for the 1986 to 1987 surface drill program.

The following is a summary of the QA/QC program for the 1986 to 1987 underground diamond drilling program:

- 9% of all samples duplicated.
- 2 samples had screened metallic analysis.
- 92% of samples assaying greater than 1.0 g/t Au have duplicate assay recorded.

11.2.8 Pezgold, 1989

Pezgold conducted surface drilling at the Mount Jamie Mine in 1989. The surface drilling explored the North Vein area and a zone 500 m east of the North Vein. The assay information was recovered from the existing diamond drill logs.

The company and laboratory QA/QC programs and procedures for the surface diamond drilling program are not recorded. Assay information was transferred from existing drill logs. Original measurement units on the drill logs were imperial lengths and assay results reported as ounces per ton of gold. Based on the assay results, fire assay with gravimetric finish had a detection limit of 0.034 g/t Au. All trace and nil values were recorded as 0.001 g/t rather than using 0.17 g/t (i.e., equal to half the detection limit).

Sample lengths were significantly longer than for previous drill programs. The 1989 drilling program was for exploration and not for resource development. The surface drilling had an average sample length of 0.69 m, with a maximum length of 1.95 m and a minimum length of 0.21 m. Samples with assays greater than 1.0 g/t Au had an average sample length of 0.63 m, with a maximum length of 1.53 m and a minimum length of 0.30 m. Duplicate assays were completed through instructions of the company or as part of the laboratory QA/QC and recorded on the drill logs. The following is a summary of the QA/QC program for the surface diamond drilling program:

- 4% of all samples are duplicated.
- 67% of samples assaying greater than 0.48 g/t Au have multiple duplicate results recorded.

There are no recorded QA/QC procedures or assay certificates available for the 1989 surface drill program. At the time of the work, the use of company inserted blanks and certified standards was not commonplace. The company did attempt to mitigate the risk of inaccurate assays by using duplicate samples. The program results indicated a problem with possibly the laboratory or the character of the mineralization. The QP considers the assay data from this program of poor quality and would limit the use of this data for inferred resources unless higher quality data is available.

11.2.9 Zenda/Vedron, 2003

Zenda/Vedron conducted surface drilling at the Mount Jamie Mine in 2003. Six drill holes were completed, with five holes drilled in the Shaft No. 1 area and the one hole drilled to target a geophysical anomaly west of Shaft No. 1. The assay information was recovered from the existing diamond drill logs.

The company and laboratory QA/QC programs and procedures for the surface diamond drilling program are not recorded. Assay information was transferred from existing drill logs. Original measurement units on the drill logs are metric lengths and assay results reported as grams per tonne of gold. All values recorded as less than 0.01 g/t Au were recorded as 0.005 g/t Au (i.e., equal to half the detection limit).

The surface drilling had an average sample length of 0.44 m, with a maximum length of 1.00 m and a minimum length of 0.30 m. Samples with assays greater than 1.0 g/t Au have an average sample length of 0.46 m, with a maximum length of 0.70 m and a minimum length of 0.30 m. Duplicate assays were completed through instructions of the company or as part of the laboratory QA/QC and recorded on the drill logs. The following is a summary of the QA/QC program for the surface diamond drilling program:

- 18% of all samples are duplicated.
- 67% of samples assaying greater than 1.0 g/t Au have duplicate results recorded.
- 50% of samples assaying greater than 10.0 g/t Au have screened metallic results recorded.

There are no recorded QA/QC procedures or assay certificates available for the 2003 surface drill program. At the time of the work, the use of company inserted blanks and certified standards was not commonplace. The company did mitigate the risk of inaccurate assays by using duplicate samples and using the screened metallics assay method. The QP considers the assay data from this program suitable for use in a resource study.

11.2.10 Hy Lake, 2007 to 2012

The sample preparation, analyses, and security procedures for drilling completed at the Mount Jamie Mine in 2007, 2011, and 2012 are described Guy (2015). Summaries of the QAQC for each year at the mine is provided below, and additional information on QAQC procedures is described in Section 11.1.4, Hy Lake, 2007 to 2012.

2007 Quality Assurance / Quality Control Summary

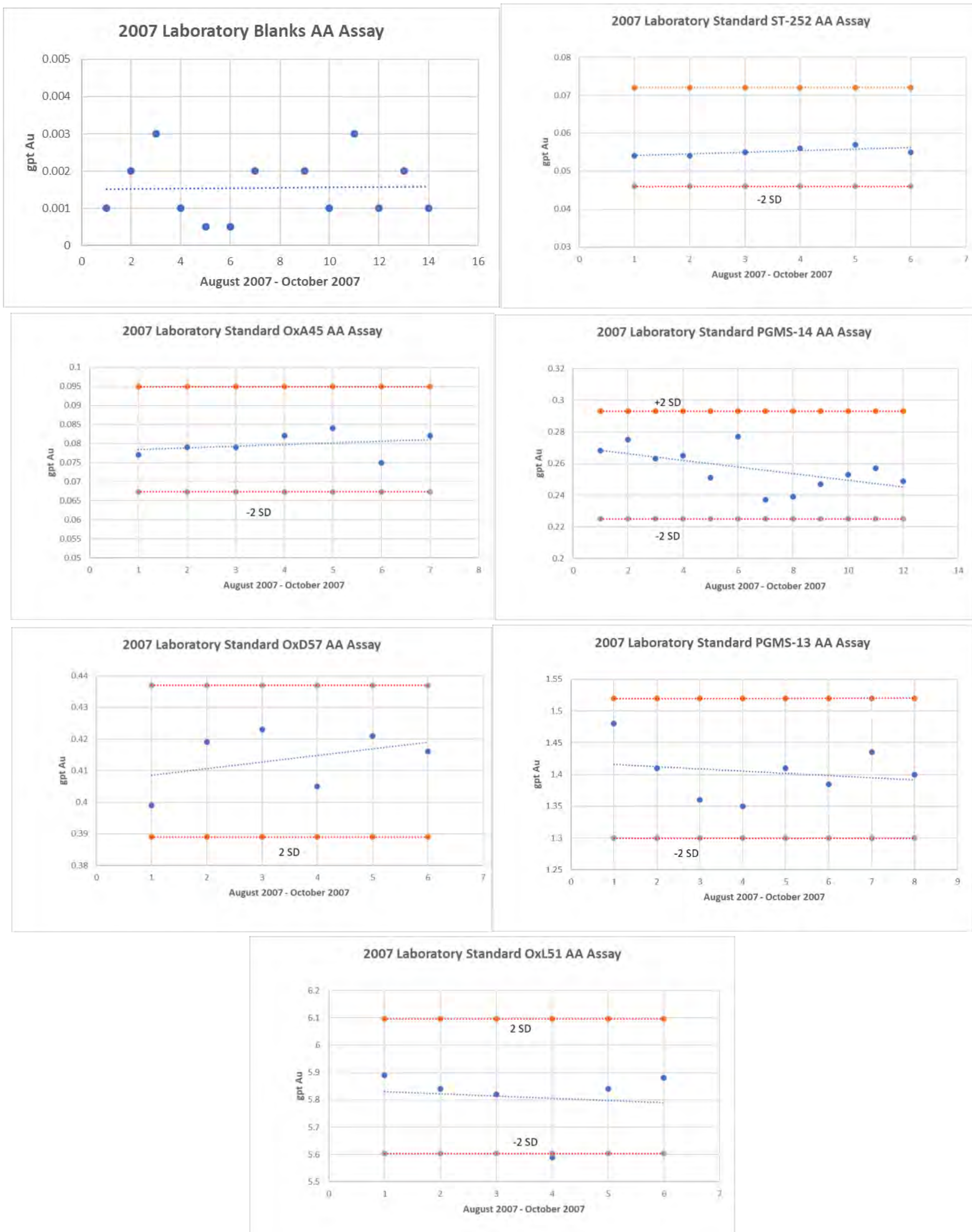
Hy Lake conducted surface drilling at the Mount Jamie Mine in 2007. Exploration drilling was completed in the Shaft No. 1 and Shaft No. 2 areas and east-west along strike. The assay information was provided by existing diamond drill logs and assay certificates.

Original measurement units on the drill logs were metric lengths and assay results were reported as grams per tonne of gold. Values less than 0.01 g/t Au were recorded as 0.005 g/t Au (i.e., equal to half the detection limit), and values less than 0.001 g/t Au were recorded as 0.001 g/t Au.

Laboratory Quality Assurance / Quality Control – ALS Chemex

Figure 11-7 presents QA/QC graphs for ALS Chemex laboratory covering the period from August 2007 to October 2007. The graphs show no issues with assay results for blanks and certified standards. There is one failure on standard OxL51 on assay certificate TB07091584 dated 11 September 2007. The certificate contained 126 samples, with 3 samples having grades between 1.2 g/t Au and 2.1 g/t Au. The other nine standards assayed on the certificate passed.

Figure 11-7: 2007 Laboratory Quality Assurance / Quality Control Results



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; SD = standard deviation.

Company Quality Assurance / Quality Control

Selected core was sampled with an average sample length of 0.44 m, with a maximum length of 2.50 m and a minimum length of 0.20 m. Samples with assays greater than 1.0 g/t Au had an average sample length of 0.55 m, with a maximum length of 1.4 m and a minimum length of 0.20 m. Duplicate assays were completed through instructions of the company and as part of the laboratory QA/QC and recorded on the drill logs.

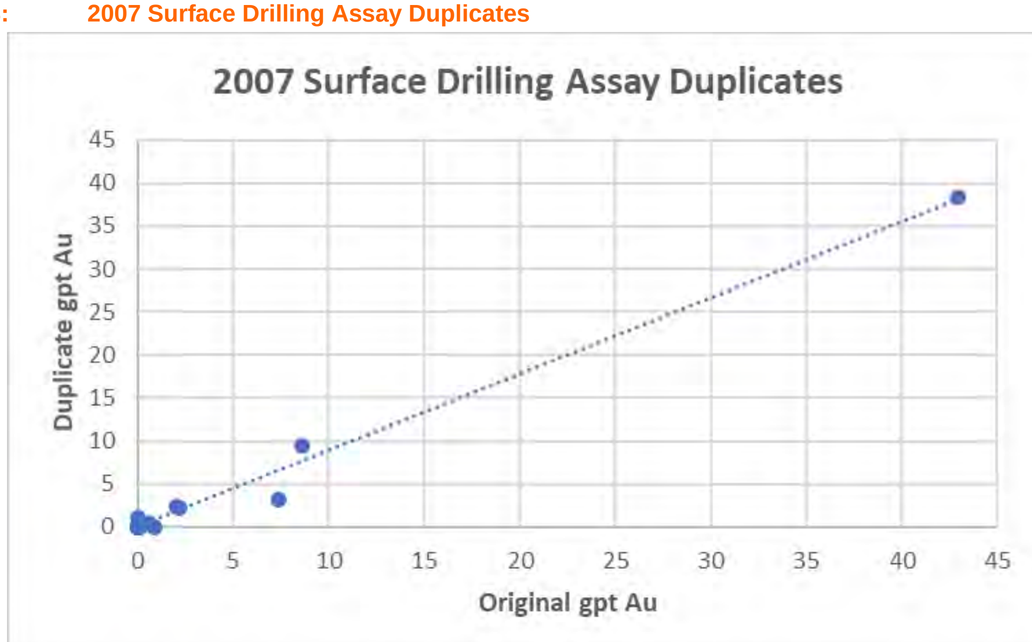
The following is a summary of the Hy Lake QA/QC program for the 2007 surface diamond drilling program:

- 5% of all samples were duplicated.
- 25% of samples assaying greater than 0.75 g/t Au have screened metallic results recorded.

Figure 11-8 and Figure 11-9 compare the results of the duplicate sampling program. The duplicate assay has good reproducibility of the original assay and shows no bias between the original and duplicate samples. There are no indications of a major concern regarding a nugget effect in the sampling.

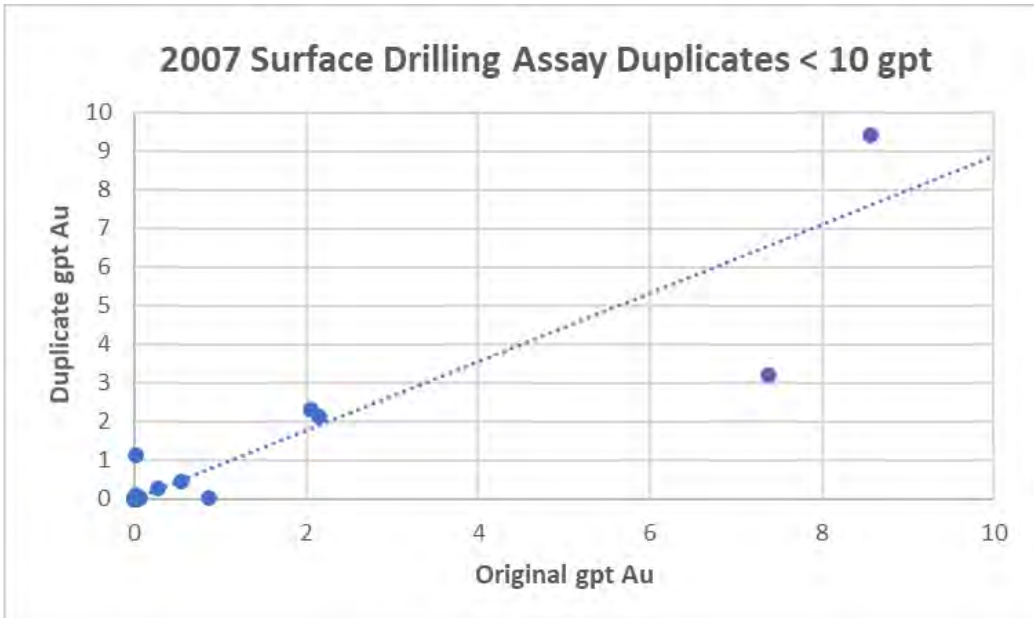
There are recorded QA/QC procedures and assay certificates available for the 2007 surface drill program. At the time of the work, the use of company inserted blanks and certified standards was not commonplace. The company did mitigate the risk of inaccurate assays by using duplicate samples and using the screened metallics assay method. The QP considers the assay data from this program suitable for use in a resource study.

Figure 11-8: 2007 Surface Drilling Assay Duplicates



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold.

Figure 11-9: 2007 Surface Drilling Assay Duplicates with Less than 10 grams per tonne of Gold



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; < = less than.

2011 Quality Assurance / Quality Control Summary

Exploration drilling was completed in the Shaft No. 2 area, North Vein, and east-west along strike. The assay information was provided by existing diamond drill logs and confirmed with assay certificates.

Original measurement units on the drill logs are metric lengths and assay results reported as grams per tonne of gold. Values less than 0.01 g/t Au were recorded as 0.005 g/t Au (i.e., equal to half the detection limit).

Laboratory Quality Assurance / Quality Control – ActLabs

The ActLabs QA/QC program included:

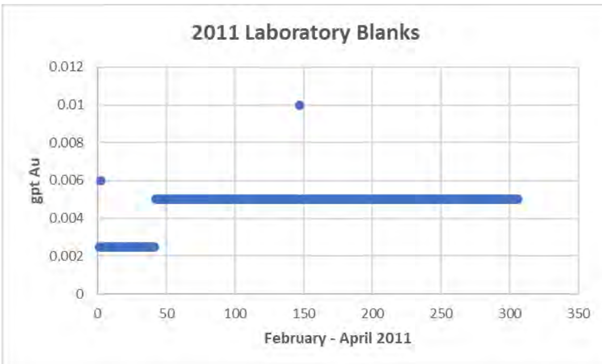
- Duplicated 14% of the samples.
- Split 7% of the samples.
- 9% added certified standards.
- 9% added blanks.

The following QA/QC graphs from Actlabs cover the period from February 2011 to April 2011 (Figure 11-10). The graphs show no issues with assay results for blanks and certified standards. There was an error on certificate A11-1553 with standards CDN-GS-3H and CDN-GS-P7B being switched. The assay certificates returned on 10 March 2011 and 11 March 2011 had 8 warnings and 2 failures of 17 assays for standard CDN-GS-P3A. The laboratory may have been having issues with this standard. After 11 March 2011, the laboratory was using standards CDN-GS-P2 and CDN-GS-P3A for the low end standard, with no issues.

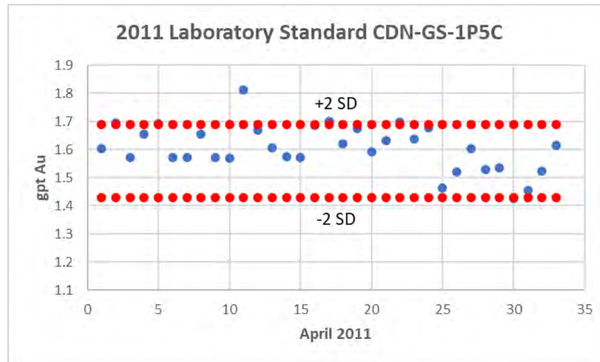
The QP's opinion is that the data supplied by the laboratory is suitable for a resource study.

Figure 11-10: 2011 Laboratory Quality Assurance / Quality Control Results

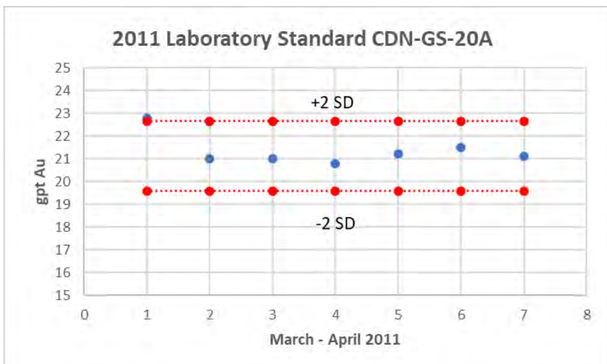
Laboratory Blanks.



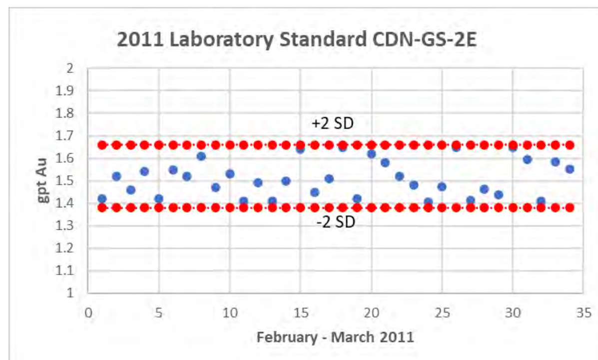
Lab Standard CDN-GS-1P5C.



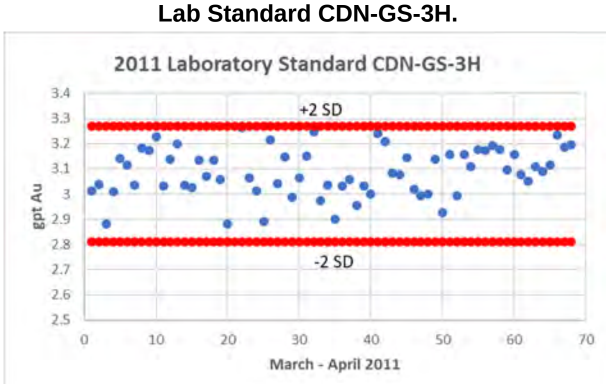
Lab Standard CDN-GS-20A.



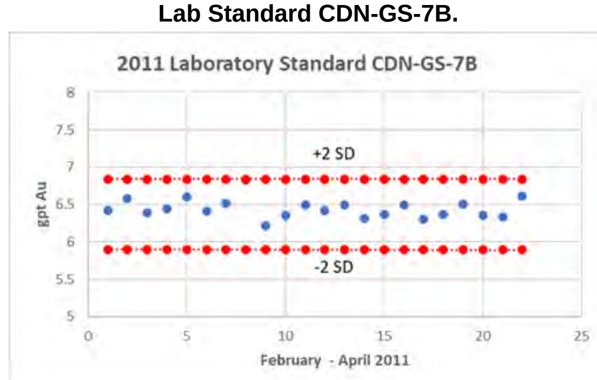
Lab Standard CDN-GS-2E.



Lab Standard CDN-GS-3H.



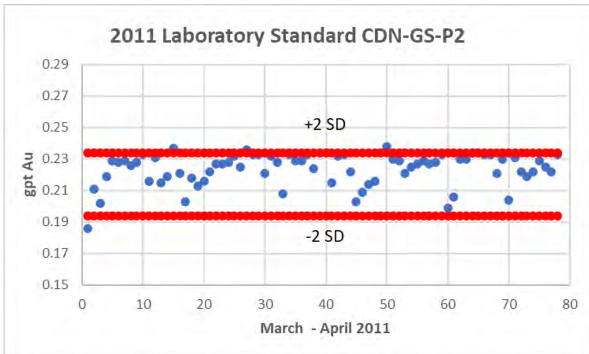
Lab Standard CDN-GS-7B.



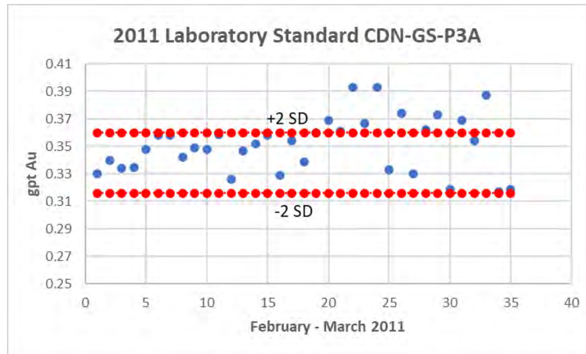
Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; SD = standard deviation.

Figure 11-10: 2011 Laboratory Quality Assurance / Quality Control Results (continued)

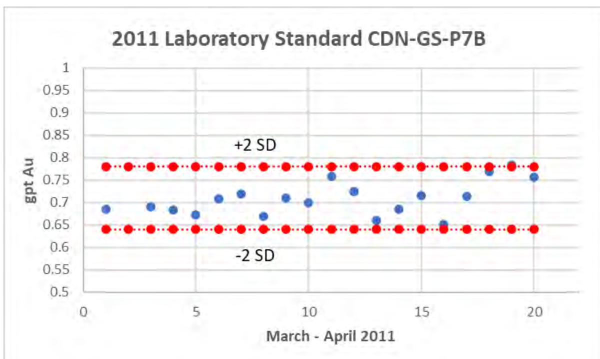
Lab Standard CDN-GS-P2.



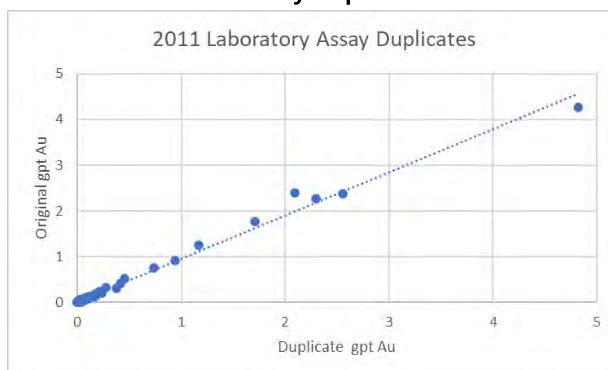
Lab Standard CDN-GS-P3A.



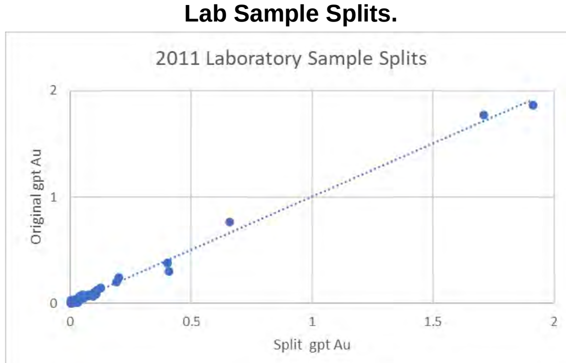
Lab Standard CDN-GS-P7B.



Lab Assay Duplicates.



Lab Sample Splits.



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; SD standard deviation.

Company Quality Assurance / Quality Control

During the 2011 diamond drill program, each hole was entirely split and sampled. A sample length of 1 m was used. Duplicate samples were systematically selected by the company. The following is a summary of the company QA/QC program for the surface diamond drilling program:

- Duplicate sample every 49 samples (including blanks and standards);
- Insert standard every 49 samples (including duplicates and blanks); and
- Insert blank every 49 samples (including duplicates and standards).

Figure 11-11 presents the results of the company QA/QC program. Inserted blanks indicate no issues with contamination during crushing, grinding, fire assay, or measurements.

The sample duplicates requested by the company had one major issue on certificate A11-2123. The original sample graded 0.016 g/t Au, while the duplicate graded 4.231 g/t Au. It is not known what actions the company took regarding this result. The remainder of the results indicate no issues with reproducibility of samples from duplicates.

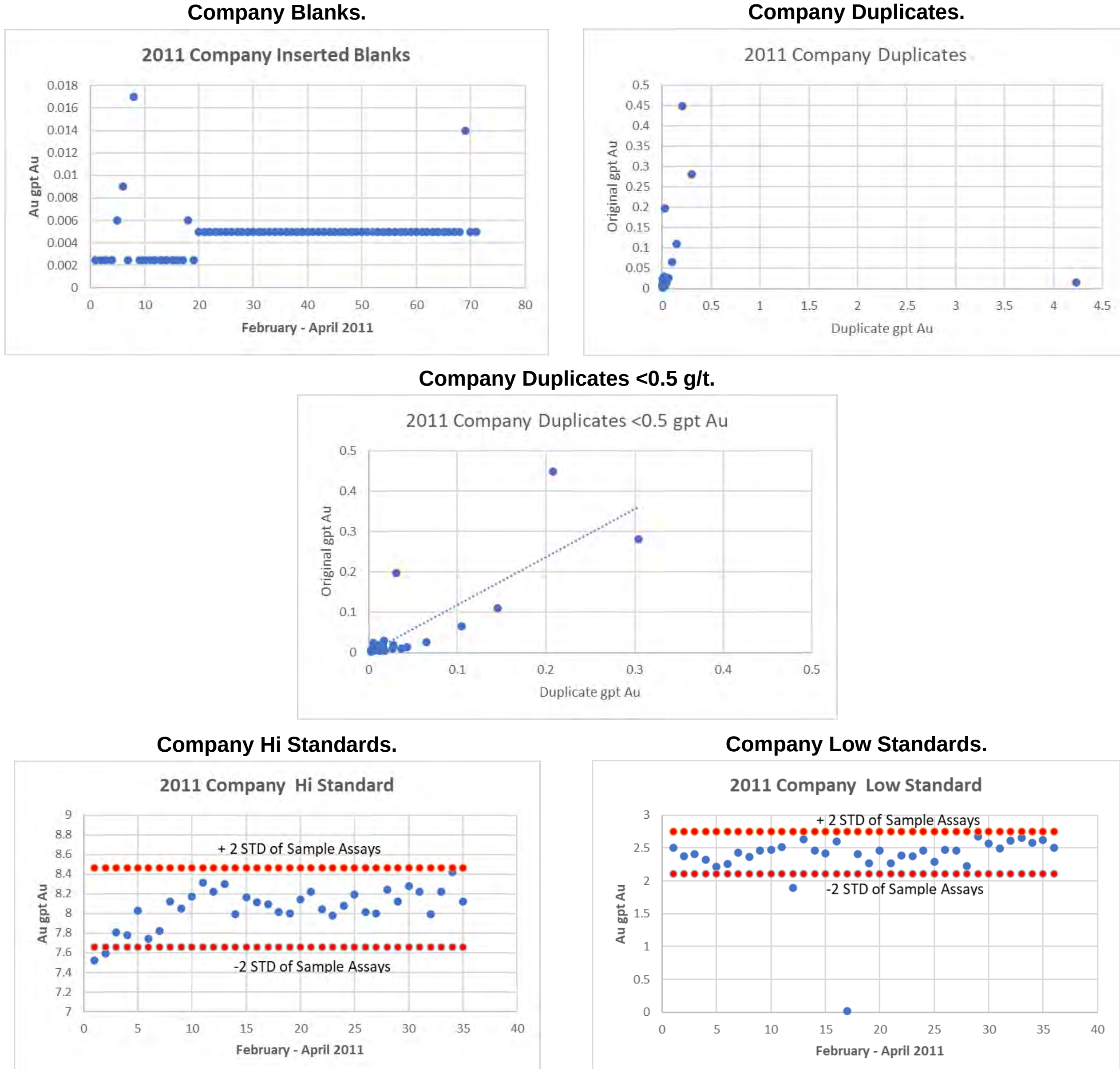
The certified standards used are unknown. For the Hi Standard, the assayed data averaged 8.06 g/t Au with a double standard deviation of 0.40 g/t Au. The standard deviation was comparable to certified standards where the average gold value was between 6.3 g/t Au and 7.4 g/t Au. There were two warnings on the low side in holes HY-11-02 and HY-11-04; the remaining standard, duplicates, and blanks showed no issue. Laboratory QA/QC showed no issues with the assay certificates. The holes had values lower than 1.0 g/t Au and therefore would not be used in a resource study.

For the Low Standard, the assayed data averaged 2.43 g/t Au with a doubled standard deviation of 0.32 g/t Au. One outlier value (i.e., 0.018 g/t Au) was not used for the calculation. The standard deviation was slightly higher compared to certified standards where the average gold value was between 1.9 g/t Au and 3.5 g/t Au. There were two warnings on the low side in holes HY-11-09 and HY-11-16; the remaining standards, duplicates, and blanks showed no issue. For HY-11-09, a blank may have been submitted instead of a standard. The company QA/QC showed no issues with the assay certificates.

There are QA/QC procedures recorded and assay certificates available for the 2011 surface drill program.

The QP considers the assay data from this program suitable for use in a resource study.

Figure 11-11: 2011 Hy Lake Quality Assurance / Quality Control Results



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; < = less than; g/t = grams per tonne; STD = standard.

2012 Quality Assurance / Quality Control Summary

The 2012 drilling program included drilling targets east along strike from the Shaft No. 1 and Shaft No. 2, and north of Shaft No. 1. The assay information was provided by existing diamond drill logs and confirmed with assay certificates.

Original measurement units on the drill logs are metric lengths and assay results reported as grams per tonne of gold. Values less than 0.01 (FA-AAS) g/t Au were recorded as 0.005 g/t Au (i.e., equal to half the detection limit), and values less than 0.03 (gravimetric) g/t Au were recorded as 0.015 g/t Au (i.e., equal to half the detection limit).

Laboratory Quality Assurance / Quality Control – ActLabs

The laboratory QA/QC program included:

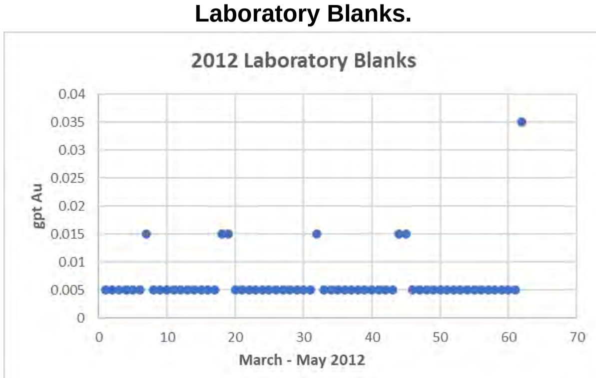
- Duplicated 9% of the samples.
- Split 5% of the samples.
- 7% added certified standards.
- 1% added blanks (some certificates did not record blanks).

The following QA/QC graphs from ActLabs cover the period from March 2012 to May 2012 (Figure 11-12). The graphs show no issues with assay results for blanks. There are no indications of cross contamination from the fire assay or analysis process. The samples grading 0.015 g/t Au were recorded using the gravimetric measurement. There is an outlier of 0.035 g/t Au on assay certificate A12-03200 that may be due to a recording error by the laboratory. The sample is listed as a screen metallic total with no other support data. The graphs for the standard assays show no major issues. There is an assay failure on assay certificate A12-03200 with an assay result above the acceptable range on standard CDN-GS-2C. This assay certificate covers the results for drill hole HY-12-25; the remaining fourteen standards assayed are within value limits. There are sample warnings (within two to three standard deviations or less than 10% of the certified average value) from OxJ80 (four assays on the low side), and standard OxK79 (two assays on the high side). The laboratory duplicates and splits show good repeatability.

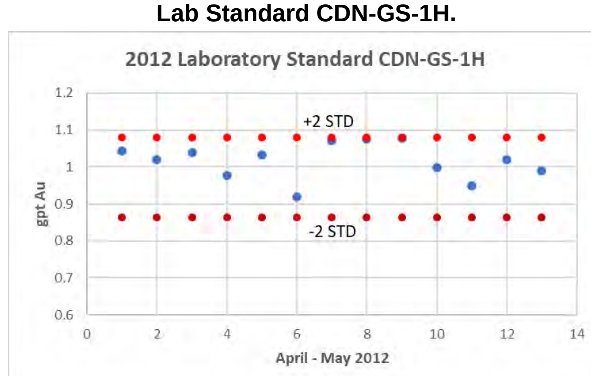
The QP’s opinion is that the data supplied by the laboratory is suitable for a resource study.

Figure 11-12: 2012 Laboratory Quality Assurance / Quality Control Results

Laboratory Blanks.



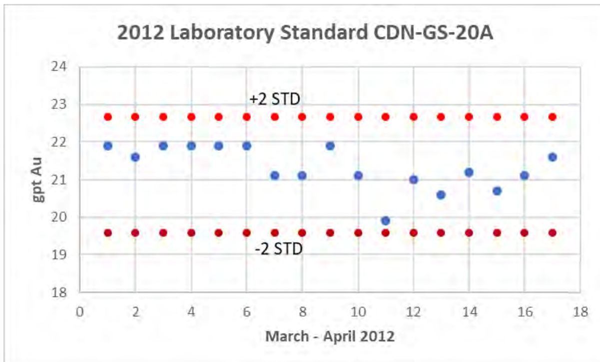
Lab Standard CDN-GS-1H.



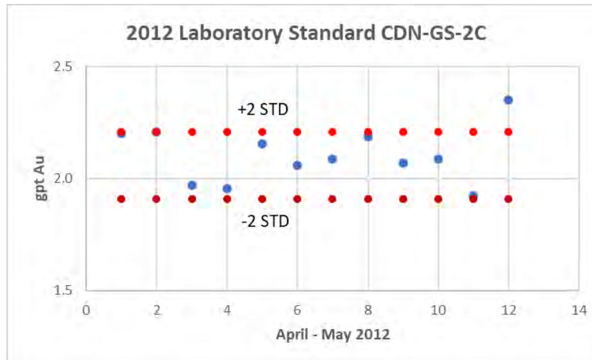
Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; STD = standard.

Figure 11-12: 2012 Laboratory Quality Assurance / Quality Control Results (continued)

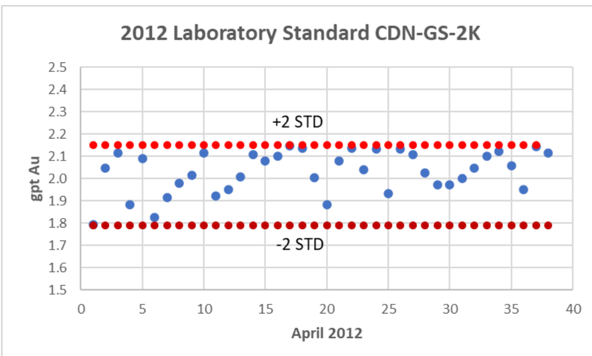
Lab Standard CDN-GS-20A.



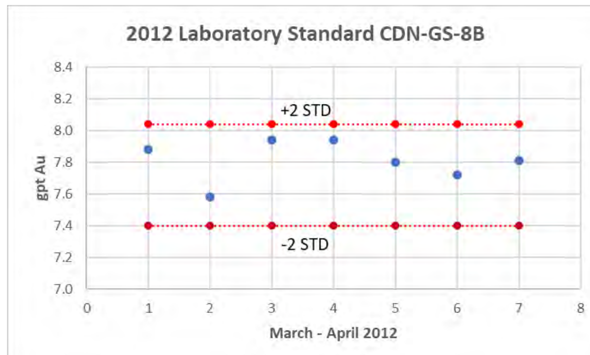
Lab Standard CDN-GS-2C.



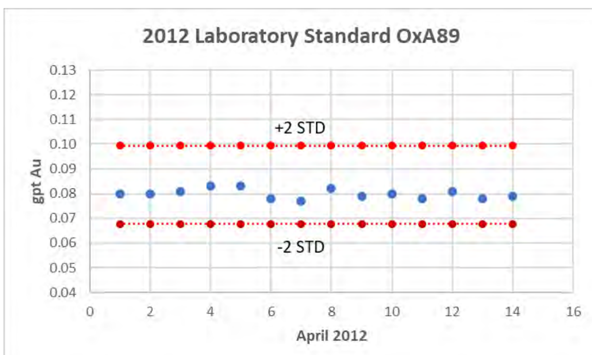
Lab Standard CDN-GS-2K.



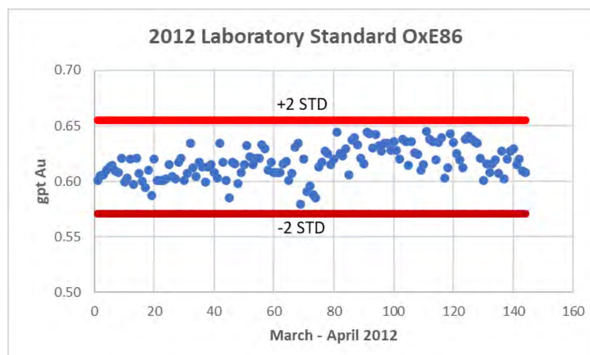
Lab Standard CDN-GS-8B.



Lab Standard OxA89.



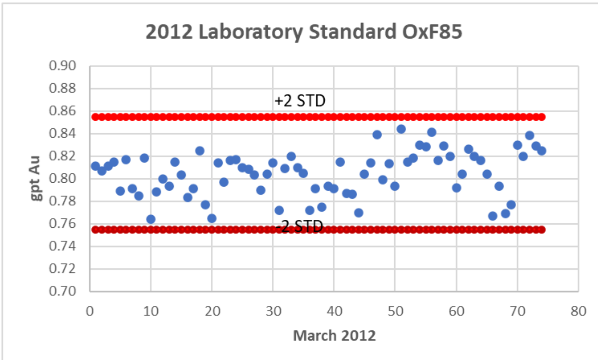
Lab Standard OxE86.



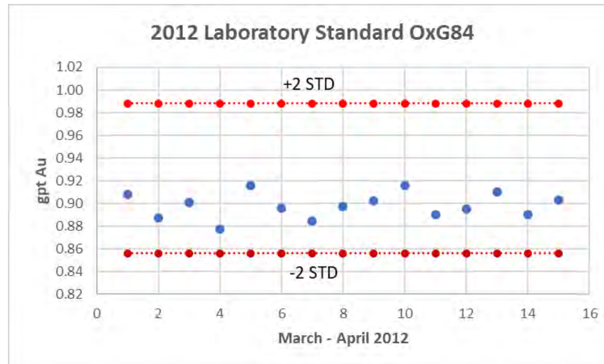
Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; STD = standard.

Figure 11-12: 2012 Laboratory Quality Assurance / Quality Control Results (continued)

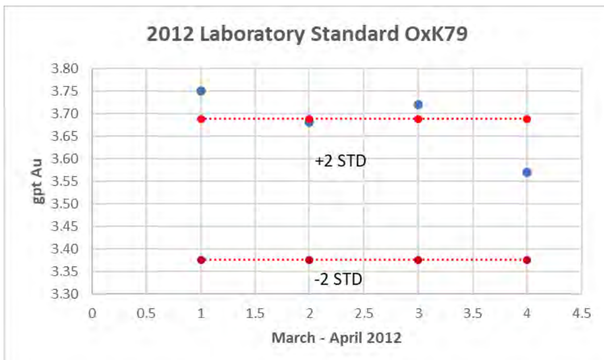
Lab Standard OxF85.



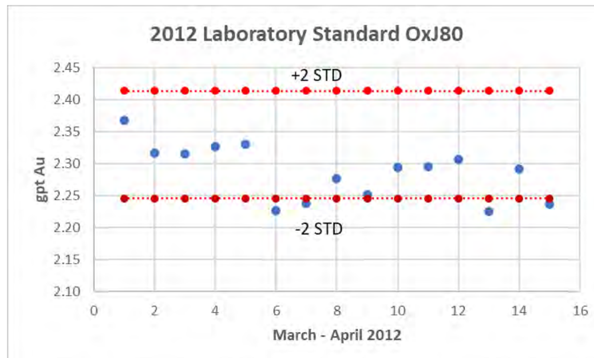
Lab Standard OxG84.



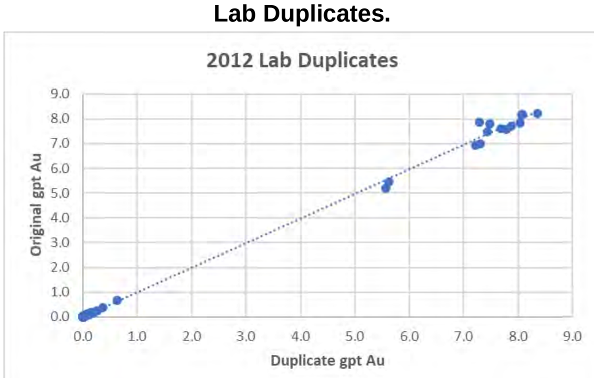
2012 Lab Standard OxK79.



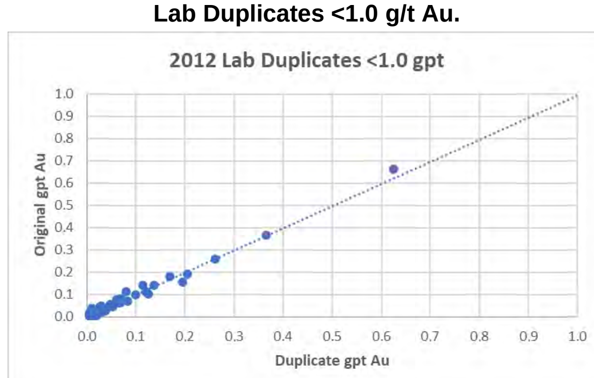
2012 Lab Standard OxJ80.



Lab Duplicates.



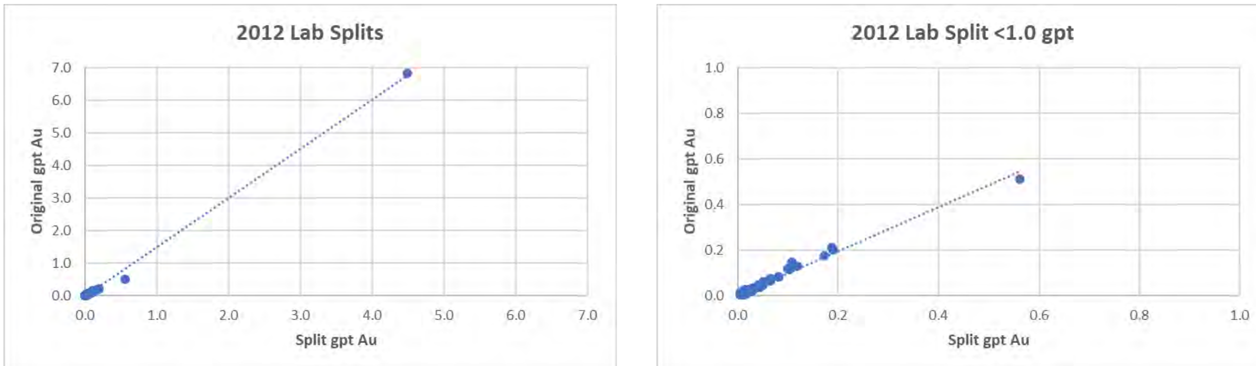
Lab Duplicates <1.0 g/t Au.



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; < = less than; g/t = grams per tonne; STD = standard.

Figure 11-12: 2012 Laboratory Quality Assurance / Quality Control Results (continued)

Lab Sample Splits. **Lab Sample Splits <1.0 g/t Au.**



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; < = less than.

Company Quality Assurance / Quality Control

For the 2012 diamond drill program, each hole was entirely split and sampled. A sample length of 1 m was used. Duplicate samples were systematically selected by the company. The following is a summary of the company QA/QC program for the surface diamond drilling program:

- Duplicate sample every 49 samples (including blanks and standards);
- Insert a standard every 49 samples (including duplicates and blanks); and
- Insert a blank every 49 samples (including duplicates and standards).

Figure 11-13 presents the results of the company QA/QC program. Inserted blanks indicate no issues with contamination during crushing, grinding, fire assay, or measurements.

The sample duplicates requested by the company had one issue on certificate A12-02833. The original sample graded 2.56 g/t Au while the duplicate graded 0.048 g/t Au. This inconsistency may be a reflection of the character of the gold mineralization rather than a sampling error. The database drill hole lithology does not indicate this unit to be a mineralized zone. It is not known what actions the company took regarding this result. The remainder of the QA/QC on this certificate from the laboratory and company results indicate no issues. The remaining duplicate samples show no major issues.

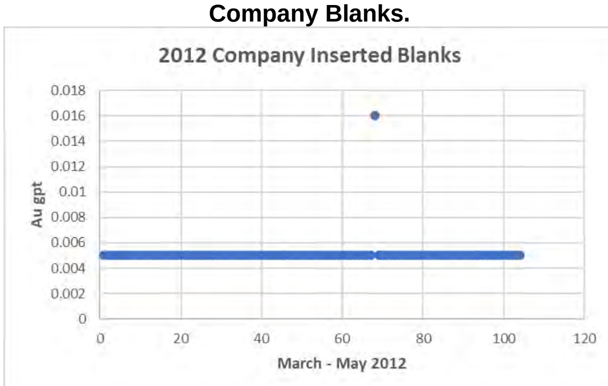
The certified standards used are unknown. For the Hi Standard, the assayed data averaged 7.69 g/t Au (compared to 8.06 g/t Au in 2011) with a doubled standard deviation of 0.76 g/t Au (compared to 0.40 g/t Au in 2011). There is no indication that the standards used in 2012 were changed from 2011. The average gold value is 4% lower than the 2011 average of 8.06 g/t Au, and there is a significant increase in the standard deviation, from 0.20 g/t in 2011 to 0.38 g/t in 2012. The source and storage of the standard is unknown. If the standard was supplied in a bulk jar or lined paper rather then a mylar packet, environmental factors during storage may have had a negative effect on the standard material. Using the greater then ten times the average value, there are two failures in the data set. One failure occurs in drill hole HY-12-05, which had one value of 1.39 g/t Au and the remainder of the values are below 0.30 g/t Au; the other failure occurs in drill hole HY-12-16, which returned values under 0.60 g/t Au. These two failures do not negatively impact the drill hole assay data.

The Low Standard assayed data averaged 2.247 g/t Au (2.43 g/t in 2011) with a standard deviation of 0.13 g/t Au (0.16 g/t in 2011). There is no indication that the standards used in 2012 were changed from 2011. The average value is 8% lower than the 2011 average of 2.43 g/t Au, and the standard deviation is comparable between the 2011 and 2012 programs. There is one warning.

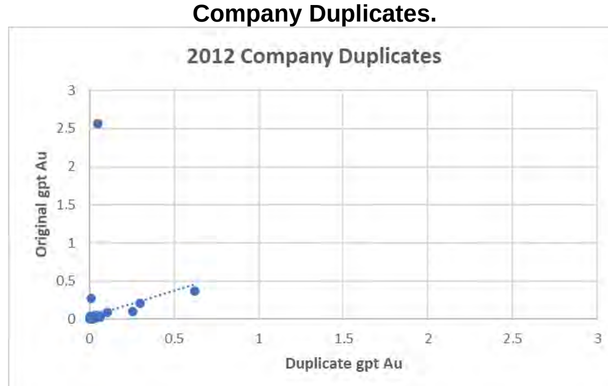
There are QA/QC procedures recorded and assay certificates available for the 2012 surface drill program. The company and laboratory inserted blanks and certified standards as part of their QA/QC programs. The results indicated no major issues with the sampling and assaying. The QP considers the assay data from this program suitable for use in a resource study.

Figure 11-13: 2012 Hy Lake Quality Assurance / Quality Control Result

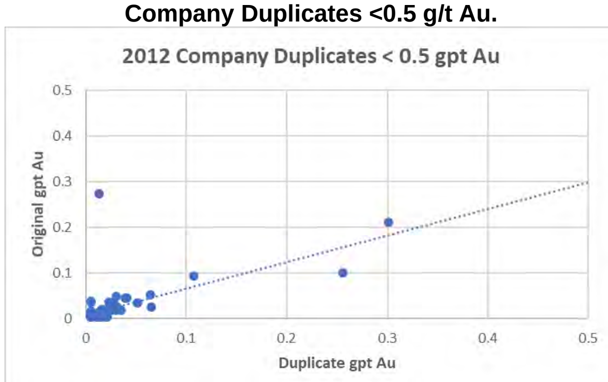
Company Blanks.



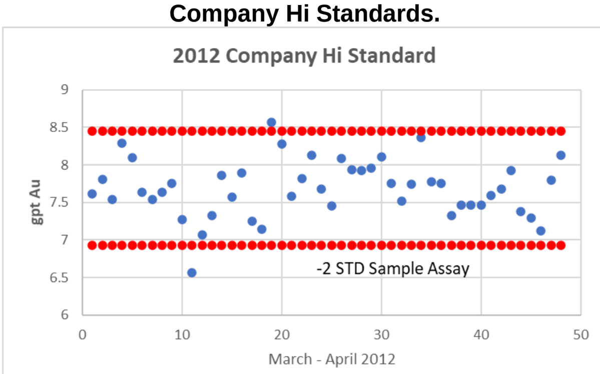
Company Duplicates.



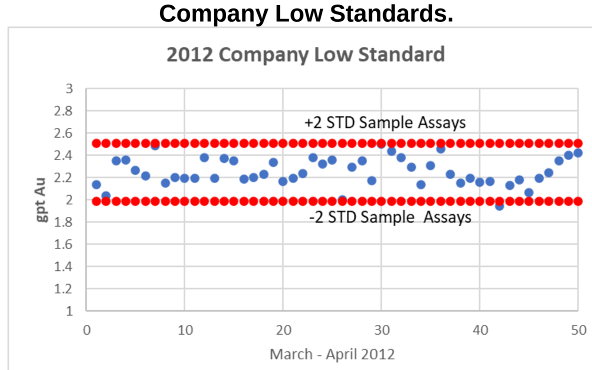
Company Duplicates <0.5 g/t Au.



Company Hi Standards.



Company Low Standards.



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold; STD = standard.

11.2.11 West Red Lake Gold Mines Inc., 2017

The 2017 drilling program included drilling targets in the Shaft No. 1 area and the North Vein. The holes in the Shaft No. 1 area were also designed to confirm the rock conditions around the historical underground workings. The assay information was provided by existing diamond drill logs and confirmed with assay certificates.

Original measurement units on the drill logs are metric lengths and assay results reported as grams per tonne of gold. Values less than 0.005 (FA-AAS) g/t Au were recorded as 0.003 g/t Au (i.e., equal to half the detection limit).

Analytical work for RLG was conducted by SGS laboratory in Ontario. Gold was analyzed by FA-AAS methods, with a gravimetric assay performed on those samples assaying greater than 10 g/t Au. Screened metallic assays were also completed on samples selected by the company geologist. SGS is independent of RLG. The SGS laboratory in Ontario is accredited to ISO/IEC 17025:2017 (general requirements for the competence of testing and calibration laboratories).

The samples were dried and crushed to 75% passing 2 mm. A Jones riffle splitter was used to take a 500 g subsample for pulverizing, and the reject portion was bagged and stored. After reducing the 500 g sample to 85% passing minus 75 µm, the sample was thoroughly blended and a 50 g charge was assayed for gold by standard FA-ICP finish. Gold values in excess of 10 ppm were re-analyzed by fire assay with gravimetric finish for greater accuracy.

Total metallics were completed on samples with visible gold at the request of the geologist in charge. Core samples were crushed to 75% passing 2 mm. A Jones riffle splitter was used to take a 1,000 g subsample for pulverizing, and the reject portion was bagged and stored. This 1,000 g sample was screened through a 150 mesh screen, and the plus and minus fractions were weighed. The entire plus fraction was assayed and the undersize was assayed in duplicate using a 50 g charge. Each fraction was submitted to fire assay for fusion and cupellation followed by gravimetric determination. The total gold content was calculated by weighing the plus and minus fractions and converting these values to g/t of gold.

Laboratory Quality Assurance / Quality Control – SGS

SGS has developed a Laboratory Information Management System (LIMS) designed to confirm the production of consistently reliable data and implemented this system at each of its locations. The LIMS covers all laboratory activities and takes into consideration the requirements of ISO standards.

The laboratory QA/QC program includes an insertion frequency of 14% that includes sample reductions, blanks and duplicates, method blanks, weighed pulp replicates, and reference materials.

The LIMS automatically flags whenever a QC material fails to meet established statistical rules preset in the system. The LIMS QC module is based on the Thompson and Howarth precision curve, which sets tolerance requirements that are associated with the detection limit and expected precision of the analyte within the method. These rules are based on rigorous method validation requirements established by SGS methodology.

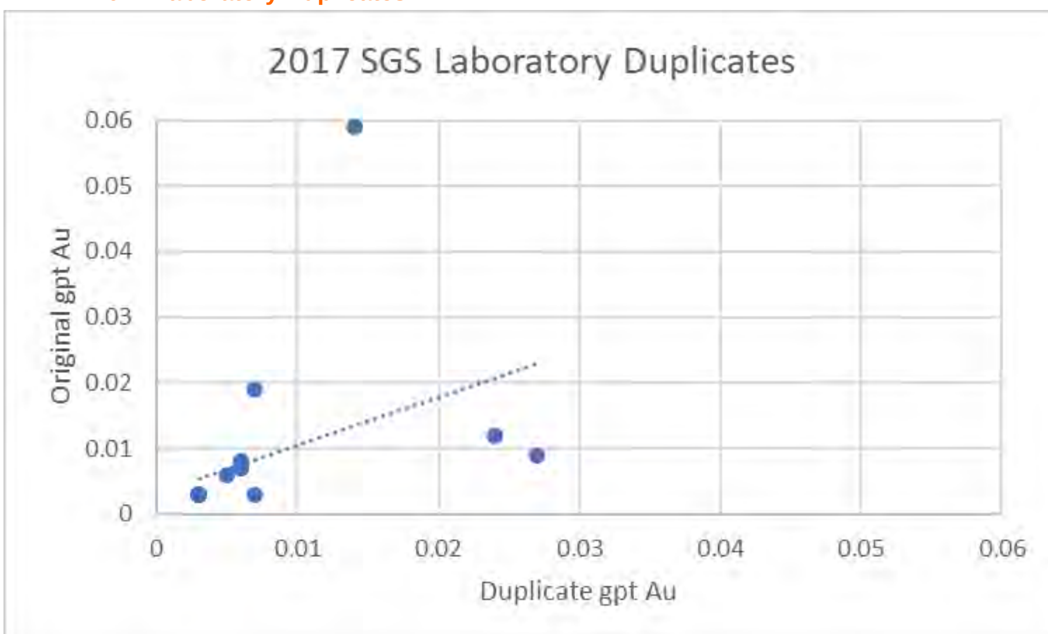
Sample reduction blanks, method blanks, and reagent blanks are used to assess) responses other than those inherent to the blank material and correct these responses when appropriate. If failure occurs that cannot be accounted for based on set rules for exemption, a minimum of 25% of the samples, including the failed blank, is repeated. Repeated sample failure results in investigation and repetition the entire batch.

When inputting the acceptable values and associated tolerances for duplicates, replicates, and reference materials, consideration is given to the fitness for purpose of the method as well as the certification process of reference material when compared to the method for which the material will be used. Repeats are performed based on a percentage of reference material, duplicate, and replicate failures to the total number of these materials inserted in a batch and repetitions range from 25% to 100%.

Figure 11-14 shows SGS duplicate results for the period June to August 2017. The graphs show no issues with assay results for duplicates. There are no indications of cross contamination from the fire assay or analysis process.

The QP's opinion is that the data supplied by the laboratory is suitable for a resource study.

Figure 11-14: 2017 Laboratory Duplicates



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold.

Company Quality Assurance / Quality Control

RLG maintained its own QA/QC program for the drilling completed at the Mount Jamie Mine. Sections of drill core to be assayed were identified by the geologist during core logging. Sample lengths averaged 1.49 m, with a minimum length of 0.5 m and maximum 2.5 m. These sections were split, using a diamond blade rock saw. Half of each sample was sealed in a plastic sample bag along with a sample identification tag. The remaining half of each sample was replaced in the core box and a copy of the sample identification tag stapled in place as a permanent record. Drill core was stored at the Mount Jamie Mine. Approximately five plastic sample bags were placed into each labelled rice bags for transport to the laboratory. Samples were transported from the Mount Jamie core processing facility to SGS laboratory in Ontario by the camp manager and core technician using a company vehicle.

Certified gold reference standards and blanks were systematically inserted into the sample stream as part of RLG QA/QC program.

The following is a summary of the company QA/QC program for the surface diamond drilling program in 2017:

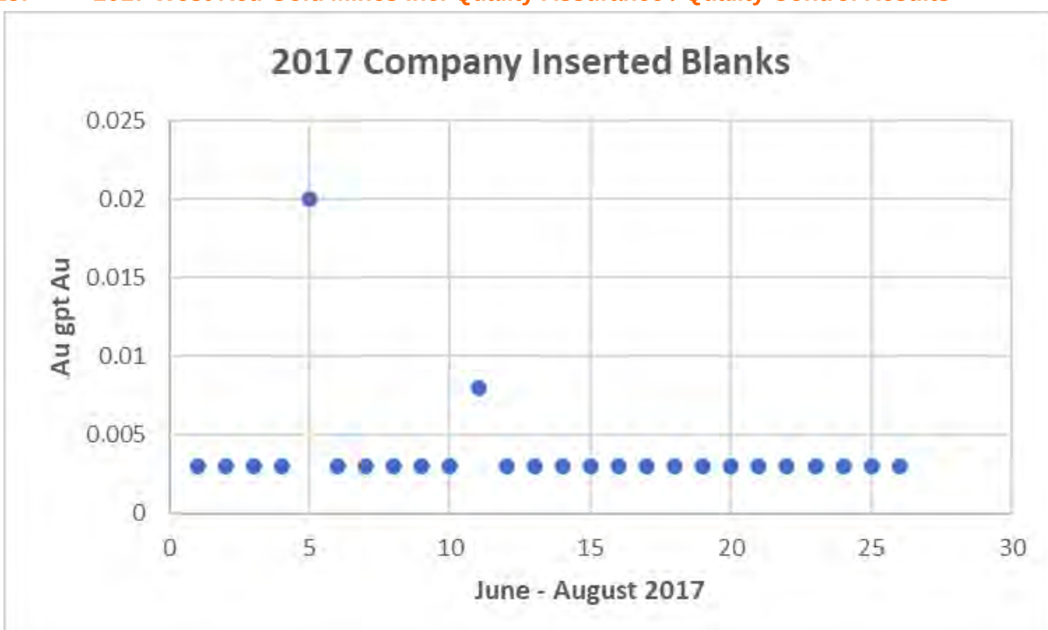
- Insert a standard every 49 samples (including blanks); and
- Insert a blank every 49 samples (including standards).

Figure 11-15 presents the results of the company QA/QC program. Inserted blanks indicate no issues with contamination during crushing, grinding, fire assay, or measurements.

The inserted certified standards generated three warnings on standard CDN-GS-7E and two warnings on standard CDN-GS-2K. The largest warning was from sample 8417 (8.459 g/t Au). This certified sample was inserted in drill hole MJ-17-08 which had no assay results greater than 1.0 g/t Au.

There are recorded QA/QC procedures and assay certificates available for the 2017 surface drill program. The company and laboratory inserted blanks and certified standards as part of their QA/QC programs. The results indicated no major issues with the sampling and assaying. The QP considers the assay data from this program suitable for use in a resource study.

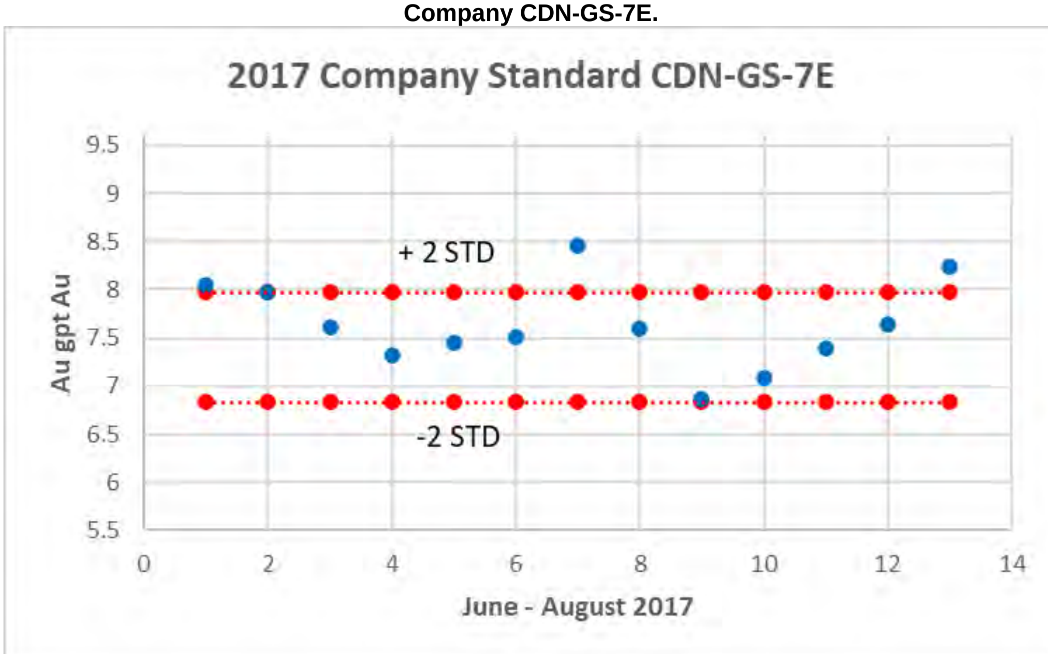
Figure 11-15: 2017 West Red Gold Mines Inc. Quality Assurance / Quality Control Results



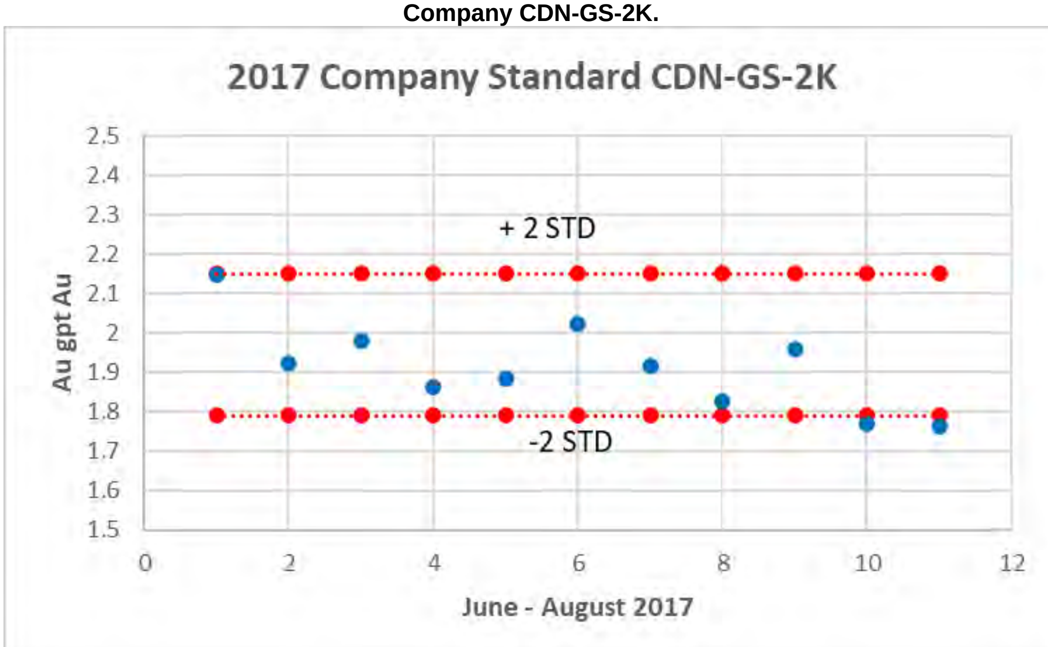
Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold.

Figure 11-15: 2017 West Red Gold Mines Inc. Quality Assurance / Quality Control Results (continued)

Company CDN-GS-7E.



Company CDN-GS-2K.



Source: SR 2025.
Notes: gpt = grams per tonne; Au = gold.

11.3 Qualified Persons Opinion

In the QP's opinion, the QA/QC program for the Rowan Mine deposit as designed is adequate and the database is suitable for use in a mineral resource estimate.

12 DATA VERIFICATION

John Sims, C.P.G., President of SIMS Resources LLC (SR) and the QP for the Rowan Mine deposit mineral resource estimate in this Technical Report, visited the Rowan Property on 20 February 2024. During the site visit, the QP toured the Project site, inspected the core, reviewed the geological interpretation and drill hole database, and discussed various aspects of mineral resources with the site technical team.

SRK Consulting (Canada) Inc. (SRK), an independent consulting firm, was retained by WRLG to complete a data verification process with a cut-off date of 15 December 2023. SRK compiled the following drill hole files that contained the detail related to the data verification:

- WRLG_Collars_15Dec_2023u.xlsx
- WRLG_Surveys_15Dec_2023.xlsx
- WRLG_Lithos_15Dec_2023.xlsx
- WRLG_Assays_15Dec_2023u.xlsx
- WRLG_Geochem_15Dec_2023.xlsx
- WRLG_Density_5Oct_2023.xlsx
- WRLG_Struct_27Nov_2023.xlsx
- WRLG_Geotech_18Dec_2023.xlsx
- WRLG_Veins_15Dec_2023.xlsx
- WRLG_Alteration_27Nov_2023.xlsx

In summary, SRK completed the following drill hole database validations and corrections:

1. During the review, SRK noted that two survey datums were used during the collar survey pick-up in the field, namely NAD 83 or WGS84 datum related to the UTM Zone 15N coordinate grid. All collar coordinate data was standardized by SRK to NAD83 UTM Zone 15N datum. The relevant drill holes were identified and corrected in ArcGIS. The file, WRLG_Collars_15Dec_2023_convert.xlsx, records this information in a summary format and all the detail is contained in WRLG_Collars_15Dec_2023u.xlsx.
2. A total of 81 drill holes were verified in the field. Two of these drill holes required a second round of verification (i.e., RLG 14-13 and RLG 23-148) as these holes did not plot in the correct position after the first round of results.
3. Drill hole collar elevation adjustments were implemented when a collar elevation position was significantly different from the recent LiDAR survey provided. Drill hole differences to surface topography ranged between -20 metres below ground surface (mbgs) and 40 mbgs. Actual values of recent drilling survey results were used to establish threshold beyond which a mean adjustment would be applied. If a surface drill hole collar elevation was greater than 0.9 m below the topography, the collar was adjusted to 0.36 m below the topography. If the collar elevation was greater than 1.4 m above the topography, the collar elevation was adjusted to 0.69 m above the topography. Values above topography are potentially related to drilling through winter ice. Adjustments were considered because the logged lithologies were typically recorded at the start of the drill hole indicating that this was not casing related to drilling through the winter snow and ice pack.

4. Underground drill hole positions could not be verified and differed between various sources. These drill holes should be treated with caution when related to resource estimation and classification. The QP did not use these data for the 2025 mineral resources estimate.
5. Drill hole data was compiled from all the relevant sources:
   - John Kita;
   - Geotic drill hole database (MS Access version);
   - Geotic extractions (different generations); and
   - Leapfrog Geo versions.
6. Approximately 43% of assay results were verified against all the assay certificates provided by WRLG.
7. Historical assays:
   - Compared with John Kita's archive of all the historical assay certificates provided as image PDFs and what was captured between the different data sources provided.
   - Gold assays unit conversions from pennyweight (dwt) and ounces per short ton (oz/ton) to grams per tonne (g/t) were checked. Discrepancies were noted and adjusted in the files provided to WRLG in Leapfrog Project and as Microsoft Excel spreadsheets.
   - Depth conversions from feet to metres were checked. A few depths required adjustments as the incorrect fractions were applied for respective inches.
   - Laboratory, certificates, and date details were added to each assay so information could be verified.
   - Approximately 24% (i.e., 1,910 of 8,080) of the historical assays could be verified directly with assay certificates.
   - Two drill holes that were previously used in resource estimates were identified as the source of sludge sample and removed (i.e., RWS 37-06 and RWS 37-07).
8. Certain drill holes were noted to have erratic downhole survey traces. These drill holes were reviewed, and invalid entries were removed.
9. Log depth errors were corrected based on the Geotic database, which is the most reliable drilling data source as this database was the primary data source for collation and capture.
10. Changes were communicated with the site staff so that corrections could be made in the Geotic database.
11. Errors were noted with drill hole naming conventions between the geotechnical data provided and the drill hole names used in Geotic. These errors, which were related to spaces in the naming conventions, were adjusted. Thirty-six depth errors were adjusted during the review.
12. Drill hole naming inconsistencies were also noted in the specific gravity data provided, and these inconsistencies were corrected.
13. A total of 67 depth issues were identified in the lithological data related to overlaps in entries, and these issues were corrected. Twenty-six entries were deleted as these entries were related to logging that was incomplete and continued at a later stage but not adjusted in Geotic.

The QP has reviewed the data adjustments and verification checks completed by SRK. The QP's validation of the data provided by SRK included:

- visual inspection of the drill hole database in Leapfrog 3D modelling software;
- collar checks;
- downhole deviation checks;
- survey checks; and
- review of drill holes removed by SRK due to insufficient supporting data, such as assay certificates and reliable collar surveys.

Based on the review of the geological data at site and validation of the work completed and assumptions made by SRK, the QP is of the opinion that the database is adequate for use in the 2025 mineral resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Testwork on the Rowan deposit was completed in November 2023 by Base Met Labs US Ltd. (Base Met Labs) in Kamloops, British Columbia and the results of the metallurgical test program (Base Met Labs 2024) are summarized below. The objective of this testwork program was to evaluate the chemical, mineral, and comminution characteristics of the Rowan deposit, as well as metallurgical gold extraction and cyanide destruction of four master composite samples from the Rowan deposit. Base Met Labs received material from 14 drill holes from the Rowan deposit representing Vein 101, Vein 102, Vein 103, and Vein 104 in September 2023 and October 2023. Four master composites were created from the 14 drill cores. Testing included chemical analysis, bulk mineral analysis, comminution testing, extended gravity recoverable gold testing, gravity leach testing, and cyanide destruction testing. This section includes a summary of the metallurgical test program results that were used as the basis for the process design and recovery methods for the Project (Section 17, Recovery Methods).

13.2 Metallurgical Samples

13.2.1 Sample Selection

Drill core from 14 drill holes representing the four main veins at Rowan deposit (i.e., Vein 101, Vein 102, Vein 103, and Vein 104) were provided for metallurgical testwork at Base Met Labs. The continuous intervals of drill core were used to create four master composites (i.e., MC-101, MC-102, MC-103, and MC-104). Vein 101, Vein 102, and Vein 103 represent approximately 75% of the resource to be mined for the Project. The locations of these 4 veins and 14 drill holes relative to the Rowan deposit are shown in plan view on Figure 13-1 and in profile view on Figure 13-2.

Figure 13-1: Plan View of Vein 101 and Vein 104 and Drill Hole Locations at the Rowan Deposit

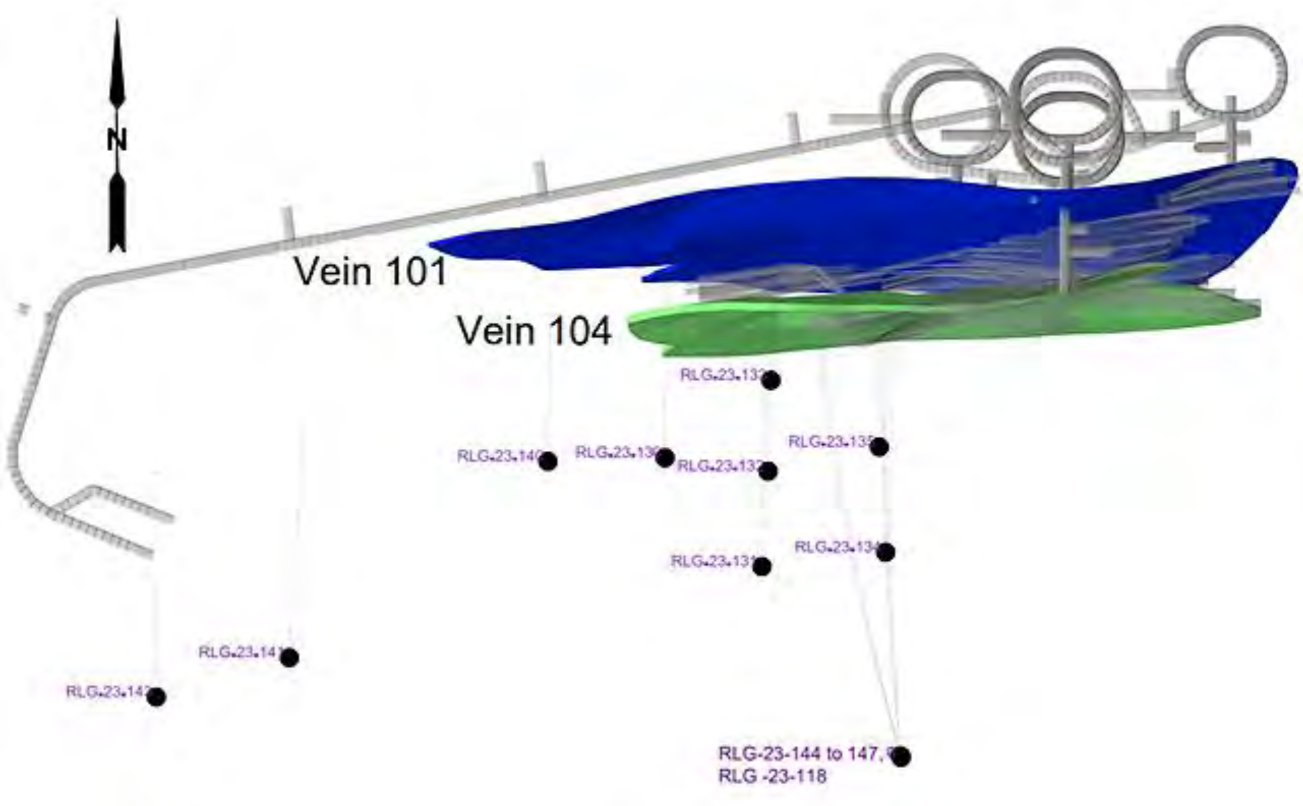
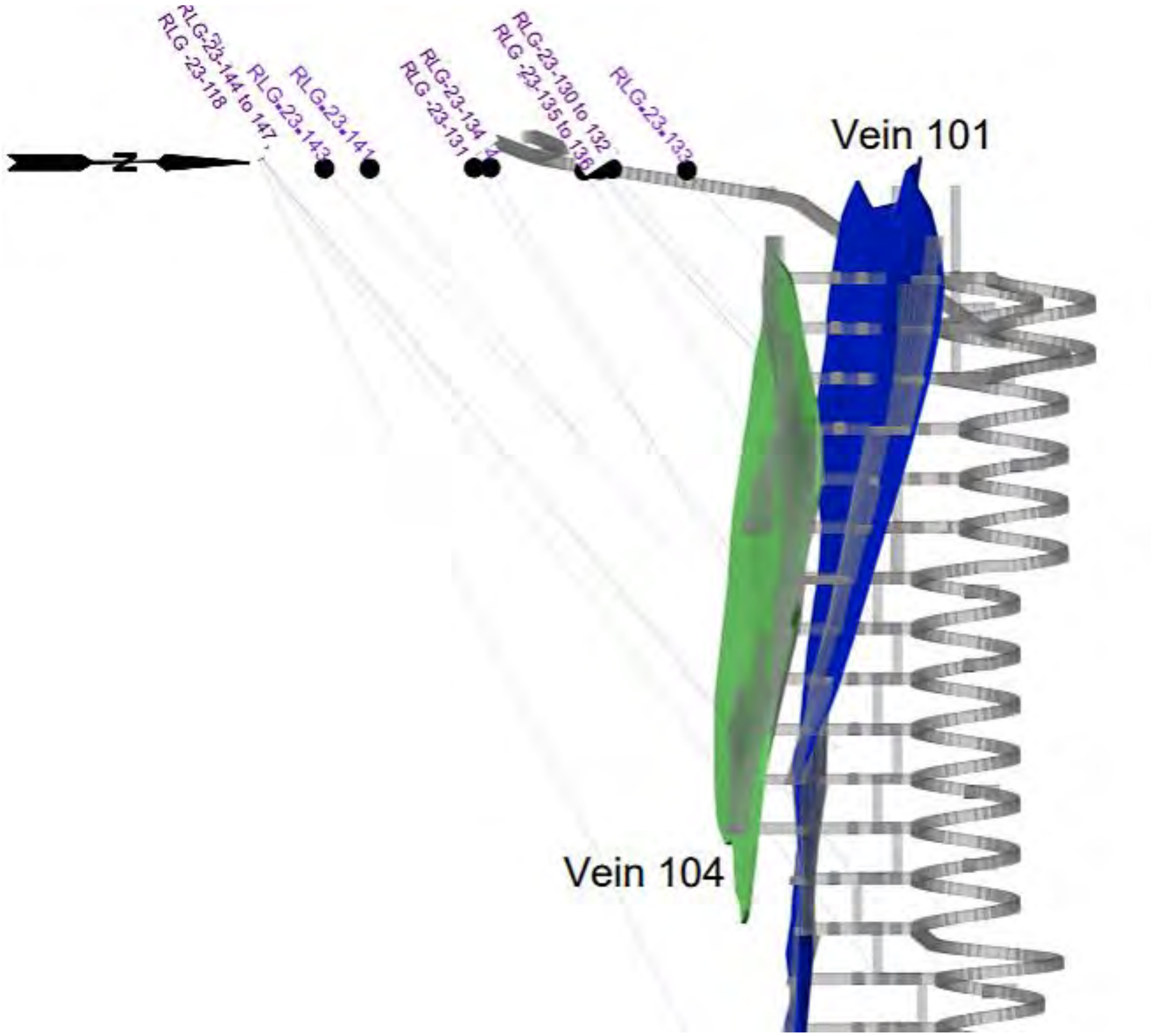


Figure 13-2: Profile View of Vein 101 and Vein 104 and Drill Hole Locations at the Rowan Deposit – Looking West



13.2.2 Head Assays

Duplicate head cuts (i.e., small, representative samples taken from each master composite to reflect its overall metal content) were removed from the four master composites and assayed for elements of interest. A minor-element inductively coupled plasma (ICP) analysis and a whole rock analysis were also performed on one head cut from each master composite. A summary of the master composite chemical content results is shown in Table 13-1.

The average gold content in MC-101, MC-102, MC-103, and MC-104 measured 5.94, 9.54, 3.31, and 3.09, respectively. Due to the range of gold content in the head assays for MC-102 resulting from coarse gold in the feed, a screen metallics gold analysis using fire assay was completed, which returned a gold assay of 11.8 g/t. The amount of coarse gravity gold in the sample material plays a significant role in the variability seen in the head grade assays.

Sulphur content in the master composite samples was low, measuring between 0.44% and 1.92%. Total organic carbon content measured 0.02% for sample MC-103 and was below the detection limit for all other master composites, which indicates carbon or carbon compounds in the deposit should not inhibit gold recovery (i.e., preg-robbing) during the leaching process.

Table 13-1: Master Composite Head Assay Results Summary for the Rowan Deposit

Master Composite Sample ID	Assay Results							
	Gold [g/t]	Silver [g/t]	Arsenic [g/t]	Copper [%]	Iron [%]	Sulphur [%]	Carbon [%]	Total Organic Carbon [%]
MC-101 Hd 1	5.56	2.0	96	0.008	2.06	0.5	0.33	<0.01
MC-101 Hd 2	6.32	2.1	104	0.008	2.08	0.52	0.34	<0.01
Average	**5.94**	**2.1**	**100**	**0.008**	**2.07**	**0.51**	**0.34**	**<0.01**
MC-102 Hd 1	11.1	1.5	412	0.004	1.62	0.41	0.46	<0.01
MC-102 Hd 2	8.01	2.6	324	0.004	1.58	0.47	0.46	<0.01
Average	**9.54**	**2.1**	**368**	**0.004**	**1.6**	**0.44**	**0.46**	**<0.01**
MC-103 Hd 1	3.22	2.8	1,754	0.018	5.44	1.95	0.61	0.02
MC-103 Hd 2	3.4	2.8	1,470	0.017	5.64	1.88	0.64	0.02
Average	**3.31**	**2.8**	**1,612**	**0.018**	**5.54**	**1.92**	**0.62**	**0.02**
MC-104 Hd 1	3.67	1.6	12	0.013	14.2	0.89	0.55	<0.01
MC-104 Hd 2	2.5	1.6	12	0.013	13.9	0.87	0.55	<0.01
Average	**3.09**	**1.6**	**12**	**0.013**	**14.1**	**0.88**	**0.55**	**<0.01**
Method	FA-AAS	ICP	ICP	FA-AAS	FA-AAS	CA[(a)]	CA[(a)]	CA[(a)]

Source: Base Met Labs 2024.
Notes:
a) Combustion analysis completed using a LECO-brand furnace.
g/t = grams per tonne; % = percent; < = less than; FA-AAS = fire assay with atomic absorption spectroscopy; ICP = inductively coupled plasma; CA = combustion analysis.

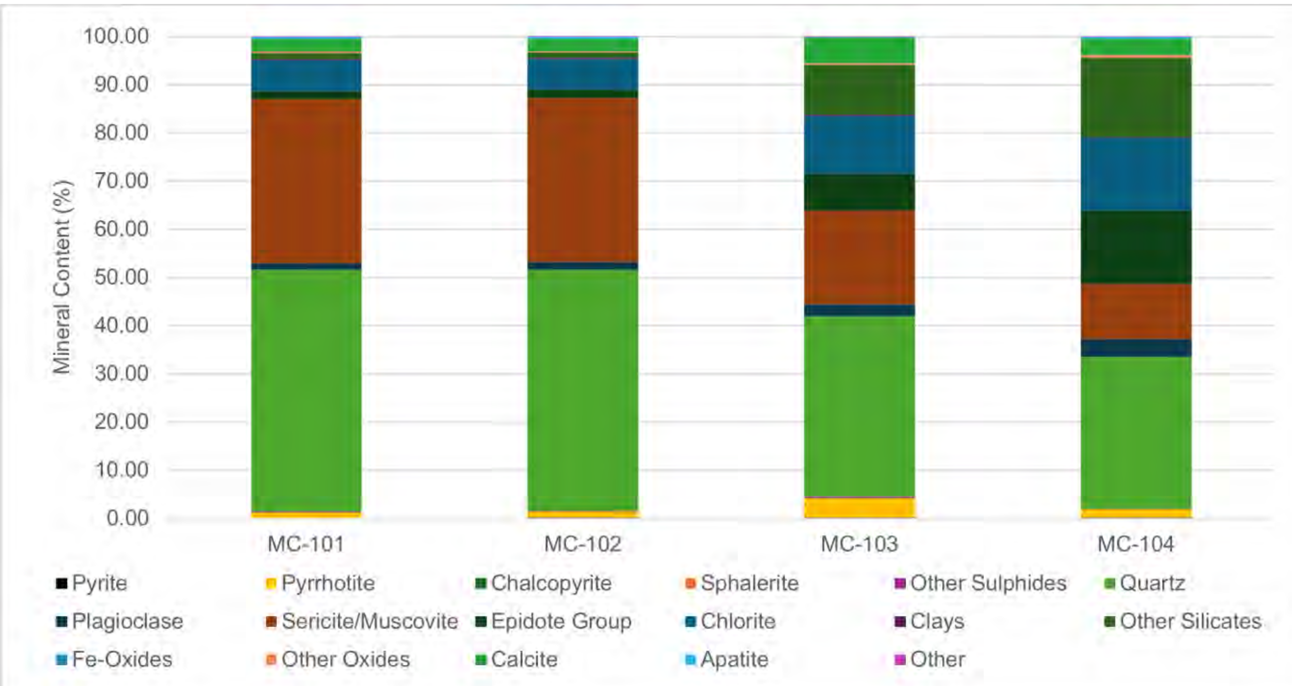
13.2.3 Mineralogy

A mineral analysis was conducted on the four master composites using the bulk mineral analysis by quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN). A summary of the master composite bulk mineral analysis results is presented on Figure 13-3 and Figure 13-4.

The sulphide content in the four master composite samples ranged from 1.5% to 4.6%, with sample MC-103 having the highest sulphur content. For all master composite samples, sulphides were dominated by pyrrhotite, followed by pyrite and sphalerite.

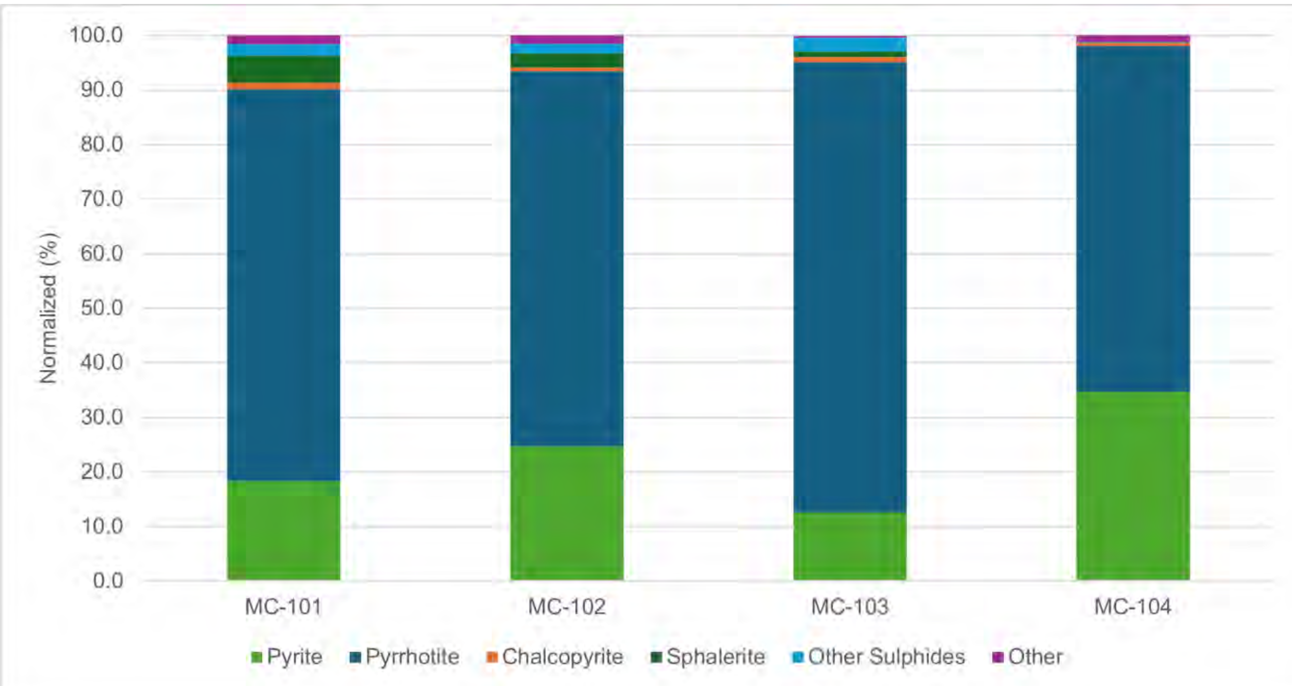
The non-sulphide minerals present in the master composite samples were primarily quartz, sericite, and muscovite.

Figure 13-3: Master Composite Mineral Content Summary for the Rowan Deposit



Source: Base Met Labs 2024.
Notes:
MC = master composite; % = percent.

Figure 13-4: Master Composite Sulphur Mineral Content Summary for the Rowan Deposit



Source: Base Met Labs 2024.
Notes:
MC = master composite; % = percent.

13.2.4 Comminution

The four master composites were subject to a semi-autonomous grinding (SAG) mill comminution (SMC) test and a Bond Work Index (BWi) test at a closing screen size of 106 microns (µm). The comminution test results for the master composites are presented in Table 13-2.

The comminution test results indicate the Rowan deposit material is hard with respect to SAG milling and ball milling, with the Axb ranging from 21.7 to 26.9, and the BWi ranging from 16.2 kilowatt-hours per tonne (kWh/t) to 18.2 kWh/t.

Table 13-2: Master Composite Comminution Test Results for the Rowan Deposit

Master Composite Sample ID	Size Fraction Tested [mm]	DWi	DWi	Mia	Mih	Mic	A	b	Axb	SG	t_a	SCSE [kWh/t]	BWi @ 106µm CSS			
		kWh/m³	%	kWh/t	kWh/t	kWh/t			SMC				F_{80} [µm]	P_{80} [µm]	Gpr	kWh/t
MC-101	22.4 - 19.0	10.4	90.0	27.1	21.8	11.3	89.7	0.30	**26.9**	2.76	0.25	12.2	2,493	74	1.02	**17.2**
MC-102	22.4 - 19.0	10.8	92.0	27.9	22.7	11.8	100.0	0.26	**26.0**	2.76	0.24	12.4	2,537	75	1.10	**16.2**
MC-103	22.4 - 19.0	12.5	98.0	29.8	25.0	12.9	83.5	0.28	**23.4**	2.91	0.21	13.7	2,284	76	1.08	**16.8**
MC-104	22.4 - 19.0	13.4	99.0	31.6	26.8	13.9	80.5	0.27	**21.7**	2.89	0.19	14.2	2,625	74	0.94	**18.2**

Source: Base Met Labs 2024.

Notes:

DWi = Drop Weight Index; Mia = comminution parameter representing the coarse particle component (i.e., greater than 750 µm) of the overall comminution energy; Mih = high pressure grinding roll parameter; Mic = crushing comminution parameter; A = JKSimMet rock breakage parameter that measures of how easily the ore breaks on impact; b = JKSimMet rock breakage parameters that reflects how the ore resists further breakage after the first impact; Axb = JKSimMet rock breakage parameter that measures resistance to impact breakage in a SAG mill; SMC = SAG mill comminution; SG = specific gravity; ta = tumbling index; SCSE = SAG circuit specific energy; SAG = semi-autonomous grinding; BWi = Bond Work Index; CSS = closed side setting; F_{80} = particle size at which 80% of the feed material mass passes through screen size; P_{80} = particle size at which 80% of the product material mass passes through screen size; Gpr = grams of product per revolution.

mm = millimetre; kWh/m^3 = kilowatt hours per cubic metre; % = percent; kWh/t = kilowatt-hours per tonne; µm = micron.

13.3 Metallurgical Testwork

13.3.1 Extended Gravity Recoverable Gold

The extended gravity recoverable gold tests were completed on samples MC-101, MC-102, and MC-103. There was insufficient sample material remaining to test MC-104 with this method. All three samples tested high for gravity recoverable gold, with overall gravity gold recoveries of 93%, 95%, and 76% for MC-101, MC-102, and MC-103, respectively. Testwork also indicated that 32% to 44% of the gravity recoverable gold is recoverable in the coarse size fraction (i.e., greater than or equal to 1,200 µm).

13.3.2 Gravity Leach

Gravity leach testing was performed on the four master composites at a primary grind size of 75 µm. Samples MC-102 and MC-103 were also tested at a coarser primary grind size of 125 µm. A bulk gravity leach test was then conducted on each master composite to generate tailings for cyanide detoxification testing. A summary of the master composite gravity leach test results is presented in Table 13-3.

Overall, all master composite samples performed very well under the conditions tested. Gravity leach testwork results demonstrate high gravity recovery to the gravity concentrate (versus the gravity tailings), ranging from 28% to 81%. The gravity leach results indicate gold extraction in the range of 94.3% to 99.5% in 24 hours, with the majority of the gold recovered in the gravity circuit concentrate. For most of the tests, the leach kinetics were fast, with most of the gold extraction (i.e., up to 99.3%) occurring in the first 6 hours. The two tests conducted at a coarser grind (i.e., 125 µm) yielded recoveries greater than 99%, which could be attributed to the high recovery to the gravity concentrate. The gold extraction recovery versus time is illustrated on Figure 13-5.

Table 13-3: Master Composite Gravity Leach Test Results Summary for the Rowan Deposit

Master Composite Sample ID	Sample Mass	Composite Test ID #	Gold Head Grade	Grind Size, P_{80}	Gravity Recovery	Leach Extraction Over Time Periods [%]			Cyanide Consumption over Time Period [kg/t]			Lime Consumption[b]
	kg		g/t	µm	%	2 h	6 h	24 h	48 h	24 h	48 h	kg/t
MC-101	2	MC-101-03	7.4	75	75.4	93.6	97.7	99.2	99.7	0.07	0.14	0.82
MC-101	10	MC-101-05[a]	6.4	75	80.1	95.4	98.0	98.8	99.6	0.07	0.07	0.51
MC-102	2	MC-102-04	15.3	75	67.1	77.5	83.0	94.3	99.7	0.06	0.08	0.79
MC-102	2	MC-102-10	13.1	125	79.2	94.3	98.2	99.3	99.8	0.10	0.10	0.55
MC-102	10	MC-102-06[a]	7.8	75	72.2	90.7	95.8	98.3	99.6	0.10	0.10	0.48
MC-103	1	MC-103-08	4.6	75	42.3	77.7	88.8	97.1	98.6	0.17	0.19	0.85
MC-103	4	MC-103-12[a]	3.4	75	27.7	76.8	92.8	97.2	98.4	0.16	0.16	0.73
MC-104	1	MC-104-09	4.6	75	81.2	97.4	99.3	99.4	99.3	0.18	0.23	0.88
MC-104	1	MC-104-11	5.1	125	68.9	91.6	97.8	99.5	99.5	0.15	0.22	0.68
MC-104	9	MC-104-13[a]	2.5	75	69.6	94.7	98.9	99.3	99.8	0.19	0.19	0.72

Source: Base Met Labs 2024.

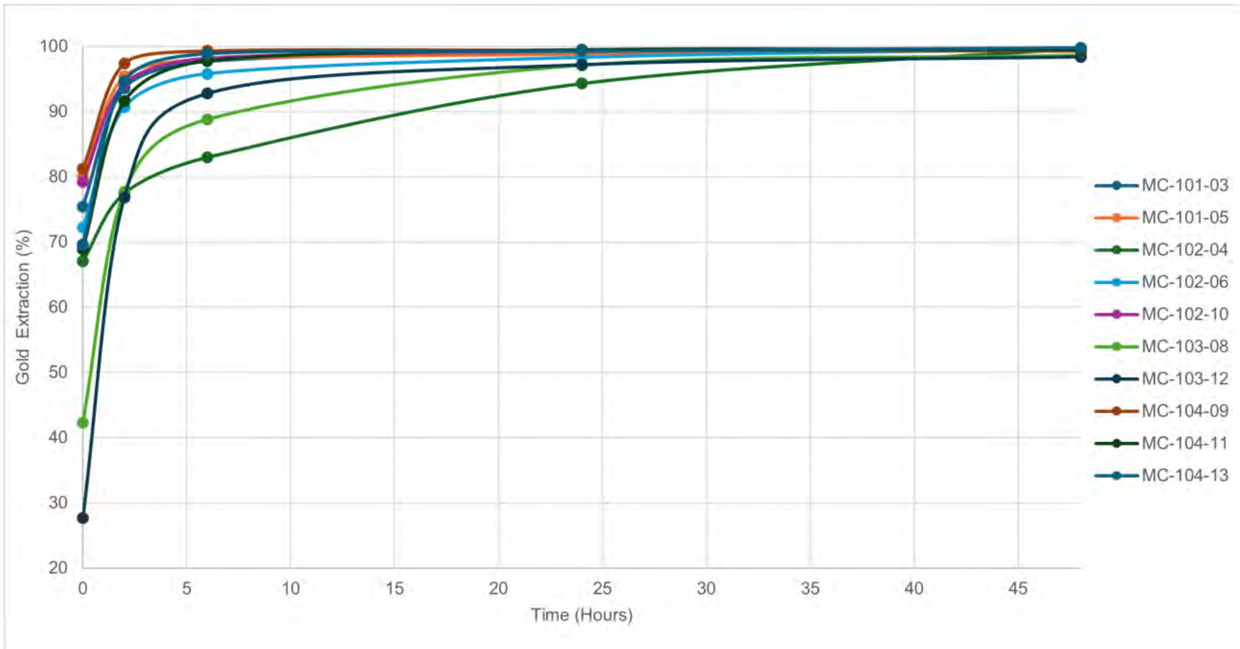
Notes:

a) Performed as bulk leach tests to generate tailings for downstream cyanide destruction testwork.

b) All lime added between 0 hours and 24 hours of leaching (i.e., 48-hour lime consumption is equal to the 24-hour lime consumption).

kg = kilogram; g/t = grams per tonne; P_{80} = particle size at which 80% of the product material mass passes through screen size; µm = micron; % = percent; h = hour; kg/t = kilograms per tonne.

Figure 13-5: Master Composite Gold Extraction Recovery by Time Period for the Rowan Deposit



Source: Base Met Labs 2024.
Notes:
MC = master composite; % = percent.

13.3.3 Cyanide Destruction

Bulk leach tests (i.e., 9 kg and 10 kg samples; Table 13-3) were performed on the four master composite samples, followed by cyanide destruction on the leach tailings using the sulphur dioxide (SO_2)/air method, a process for destroying cyanide by converting cyanide to cyanate through oxidation using SO_2 and air. Approximately 25 grams per litre (g/L) of carbon was included in the master composite samples for 6 hours after bulk leach termination and prior to cyanide destruction testing. The master composite cyanide destruction test results are summarized in Table 13-4.

The initial conditions tested consisted of a sulphur dioxide to weak acid dissociable cyanide (SO_2:CN_{WAD}) ratio of 5, and included the addition of 30 milligrams per litre (mg/L) of copper. The target CN_{WAD} value of less than 5 parts per million (ppm) was achieved for all master composite samples under these initial conditions.

For MC-101 and MC-102, optimization of the cyanide destruction showed that retention time could be reduced from 60 minutes to 45 minutes, SO_2:CN_{WAD} ratio could be reduced from 5 to 4, and copper addition could be reduced from 30 mg/L to 10 mg/L while still obtaining less than 1 ppm CN_{WAD}.

Fewer tests were performed on MC-103 due to the low sample mass; however, the target CN_{WAD} concentration (i.e., concentration limit required to discharge to tailings) was reached with an SO_2:CN_{WAD} ratio of 7 and a copper addition of 30 mg/L. Given more sample mass, further optimization tests could be completed on MC-103 that could potentially achieve results similar to MC-101 and MC-102.

For MC-104, optimization of the bulk leach testing resulted in an SO_2:CN_{WAD} ratio of 4 and no copper addition required to reach the target CN_{WAD} concentration.

The cyanide destruction test results confirm that the SO_2/air method is effective for all master composite samples tested, with future opportunities for reagent and retention time optimization depending on sample chemistry.

Table 13-4: Master Composite Cyanide Destruction Results Summary for the Rowan Deposit

Master Composite Sample	Sample ID	Detox Test	Test Parameters				Test Length		Feed/Detox Solution Assays [ppm]				
			pH	Retention Time [min]	Reagent Used		Time [min]	Number of Displacements	CN_{WAD}	Copper	Iron	Zinc	Nickel
					Sulphur Dioxide [g/g CN_{WAD}]	Copper [mg/L]							
MC-101	T05 CNTI	Feed	10.2	-	-	-	-	-	181	9.04	4	0.24	6.57
		C1	8.1	60	5.0	30	180	3	1.16	0.39	<1	0.12	0.03
		C2	8.1	60	5.0	15	180	3	0.64	0.23	<1	0.16	<0.01
		C3	8.1	60	4.0	10	180	3	0.61	0.16	<1	0.13	<0.01
		C4	8.1	45	4.0	10	135	3	0.50	0.10	<1	0.11	<0.01
		C5[(a)]	8.0	45	4.0	0	45	1	0.40	0.09	<1	0.10	0.01
MC-102	T06 CNTI	Feed	10.2	-	-	-	-	-	162	4.28	5	0.47	5.57
		C1	8.3	60	5.0	30	180	3	1.55	0.31	<1	0.34	0.01
		C2	8.1	60	5.0	15	180	3	0.87	0.33	<1	0.32	<0.01
		C3	8.0	60	4.0	10	180	3	0.81	0.43	<1	0.27	<0.01
		C4	8.1	45	4.0	10	135	3	0.44	0.13	<1	0.22	<0.01
		C5[(a)]	8.2	45	4.0	0	45	1	0.47	0.07	2	0.21	0.02
MC-103[(b)]	T12 CNTI	Feed	10.2	-	-	-	-	-	125	20.9	5	1.00	8.26
		C1	8.0	60	5.0	30	180	3	8.0	9.27	<1	0.22	<0.01
		C2	8.1	45	7.0	30	90	2	0.6	0.22	<1	0.10	<0.01
MC-104	T13 CNTI	Feed	10.3	-	-	-	-	-	194	31.1	2	3.36	2.88
		C1	8.1	60	5.0	30	180	3	2.54	2.72	<1	0.40	<0.01
		C2	8.0	45	4.0	10	90	3	5.62	7.93	<1	0.92	<0.01
		C3	8.2	60	4.0	10	180	3	2.20	3.12	<1	0.23	<0.01
		C4	8.1	60	4.0	0	180	3	1.40	1.14	<1	0.58	<0.01

Source: Base Met Labs 2024.

Notes:

a) Insufficient sample material available.

b) Due to low sample mass, 550 mL reactor was used for this sample; all other bulk leach and cyanide destruction tests were conducted in a 900 mL reactor.

T = test; CNTI = cyanidation tailings; - = feed solution assays; CN_{WAD} = weak acid dissociable cyanide; min = minutes; g/g CN_{WAD} = grams per gram of CN_{WAD}; mg/L = milligrams per litre; mL = millilitre; ppm = parts per million; < = less than.

13.4 Recovery Estimates

Gold extraction was evaluated using a combination of gravity concentration followed by cyanide leaching of the gravity tailings. Testwork was conducted on the four master composites, with overall gold extractions ranging from 94.3% to 99.5% after 24 hours, and 98.4% to 99.8% after 48 hours (Table 13-3). Most of the gold (i.e., up to 99.5%) was leached within the first 6 hours to 24 hours, and coarser grind tests (i.e., 125 µm) also showed high recoveries (i.e., up to 99.4%) after 24 hours, suggesting the material from the Rowan deposit is not highly sensitive to grind size within the tested range.

Gravity gold recovery ranged from 28% to 81%, indicating the tested material from the Rowan deposit is highly amenable to flowsheets incorporating gravity concentration.

For the Project, material from the Rowan deposit is assumed to be toll milled at an existing off-site processing facility equipped with a conventional gravity-leach-carbon-in-pulp (CIP) circuit. While laboratory results indicate very high gold extraction under ideal conditions (i.e., greater than 99%), a design gold recovery of 97% was selected to account for anticipated solution losses and other operational inefficiencies typical of commercial-scale ore processing.

The gold recovery estimates are based on testwork completed to date and are considered appropriate for the current level of study. Gold recovery assumptions may be refined as additional variability testwork is conducted on a broader set of samples from the Rowan deposit.

13.5 Deleterious Elements

Based on the current flowsheet (Section 17, Recovery Methods) and the results of the metallurgical test program (Base Met Labs 2024), no deleterious elements have been identified that would preclude successful gold extraction or materially affect processing performance or tailings characteristics at this stage of the Project.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

Mineral resources for the Rowan deposit were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM 2014). The modelling and estimation of the mineral resources was completed between 1 January 2024 and 1 March 2024 by or under the supervision of the John Sims, President of SR and the QP for this mineral resource estimate. In June 2025, some minor updates were completed to the resource model that were not material but increased the overall accuracy of the mineral resource estimate. The effective date of the mineral resource estimate is 30 June 2025. The mineral resource estimate presented here supersedes any previously stated mineral resource estimates for the Project.

For each area, domains representing gold mineralization were defined in Leapfrog Geo version 2024.1 software, and sub-block model estimates were completed in Leapfrog Edge software using 2.0 m capped composites and a single-pass inverse distance cubed (ID3) interpolation approach. Blocks were classified considering local drill hole spacing. Class groupings were based on criteria developed using continuity models (i.e., variograms) and modified to reflect geological understanding and to verify cohesive classification shapes.

Wireframe and block model validation procedures were completed for all zones, including wireframe to block volume confirmation, statistical comparisons of composite gold grades versus ID3, and nearest neighbour estimates using swath plots, visual reviews in 3D, as well as longitudinal, cross-section, and plan views.

The Rowan deposit mineral resource estimate is presented in Table 14-1. 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 2025 mineral resource estimate.

Table 14-1: 2025 Mineral Resource Estimate Summary for the Rowan Deposit as of 30 June 2025

Category	Tonnage [t]	Average Grade [g/t Au]	Contained Metal [oz Au]
Indicated	478,707	12.78	196,747
Inferred	421,181	8.73	118,155

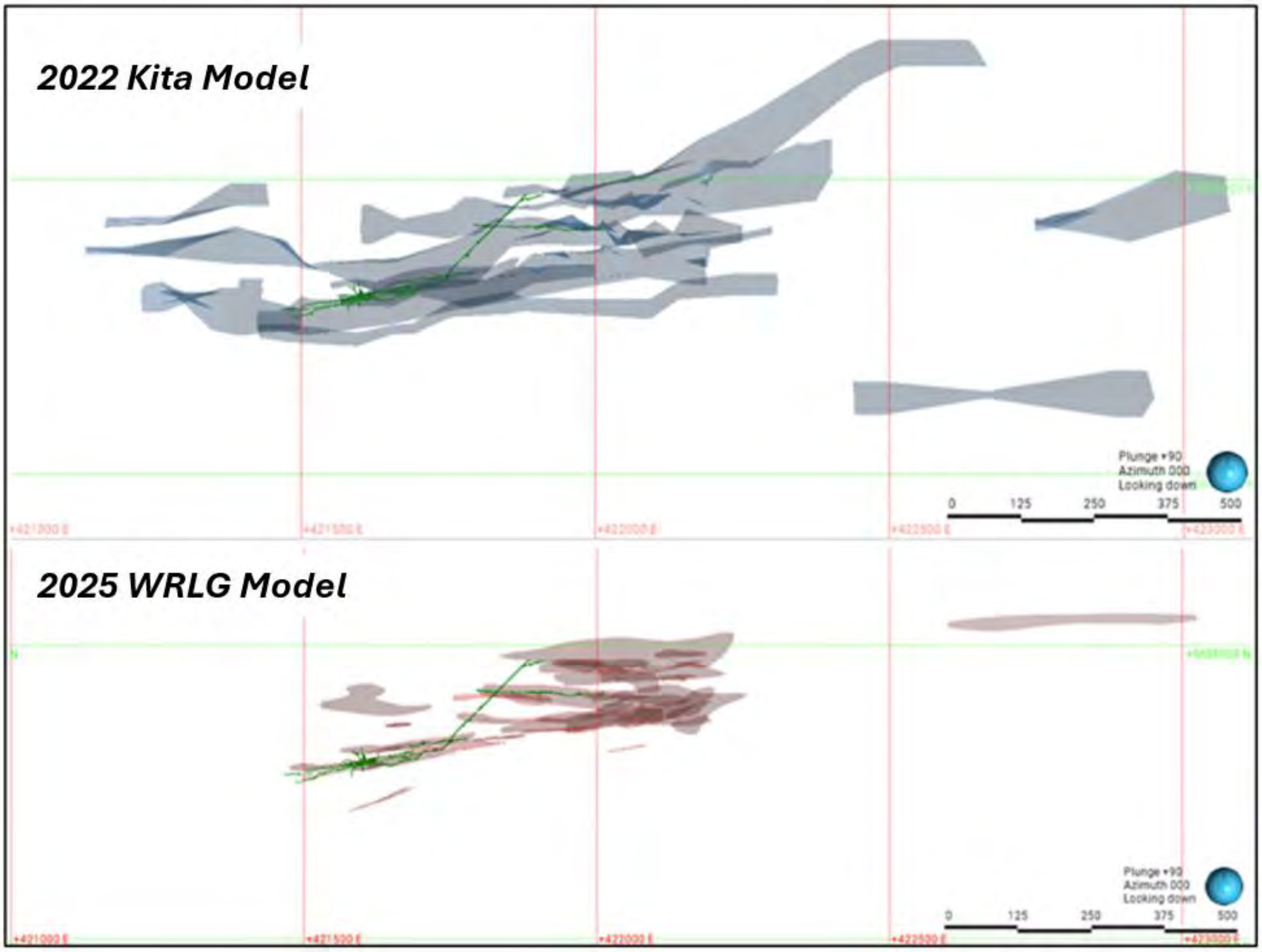
Notes:
- CIM (2014) definitions were followed for mineral resources.
- Mineral resources were estimated at a gold cut-off grade of 3.80 g/t using a long-term gold price of US$1,800 per ounce.
- There are no mineral reserves currently estimated at the Rowan deposit.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Mineral resources are reported within vein wireframes at the stated cut-off grade of 3.80 g/t Au.
- Density of 2.8 g/cm^3.
- Numbers may not add due to rounding.

t = tonne; g/t = grams per tonne; Au = gold; oz = ounce; g/cm^3 = grams per cubic centimetre.

14.2 Comparison to Previous Mineral Resource Estimates

The previous interpretation of the Rowan Vein System (Kita 2022) was that the veins were emplaced after D2 foliation and were essentially undeformed and highly continuous over hundreds of metres. Such an interpretation was supported by the consistency of multiple narrow vein occurrences in drill holes spanning long strike lengths. With the benefit of a tighter drill spacing and oriented core following the 2023 WRLG drill programs (Section 10.3, Diamond Drilling from 2022 to 2023), this interpretation is no longer tenable. The new interpretation on veining and structural controls has resulted in a more compact and higher-grade resource at the Rowan deposit, with opportunity for expansion and growth on the existing model through additional drilling. Figure 14-1 provides a visual comparison of model domains between the previous model (Kita 2022) and the current 2025 model completed by WRLG.

Figure 14-1: Plan View Image Comparison of 2022 Kita Model Domains to 2025 West Red Lake Gold Mines Ltd. Model Domains for the Rowan Deposit

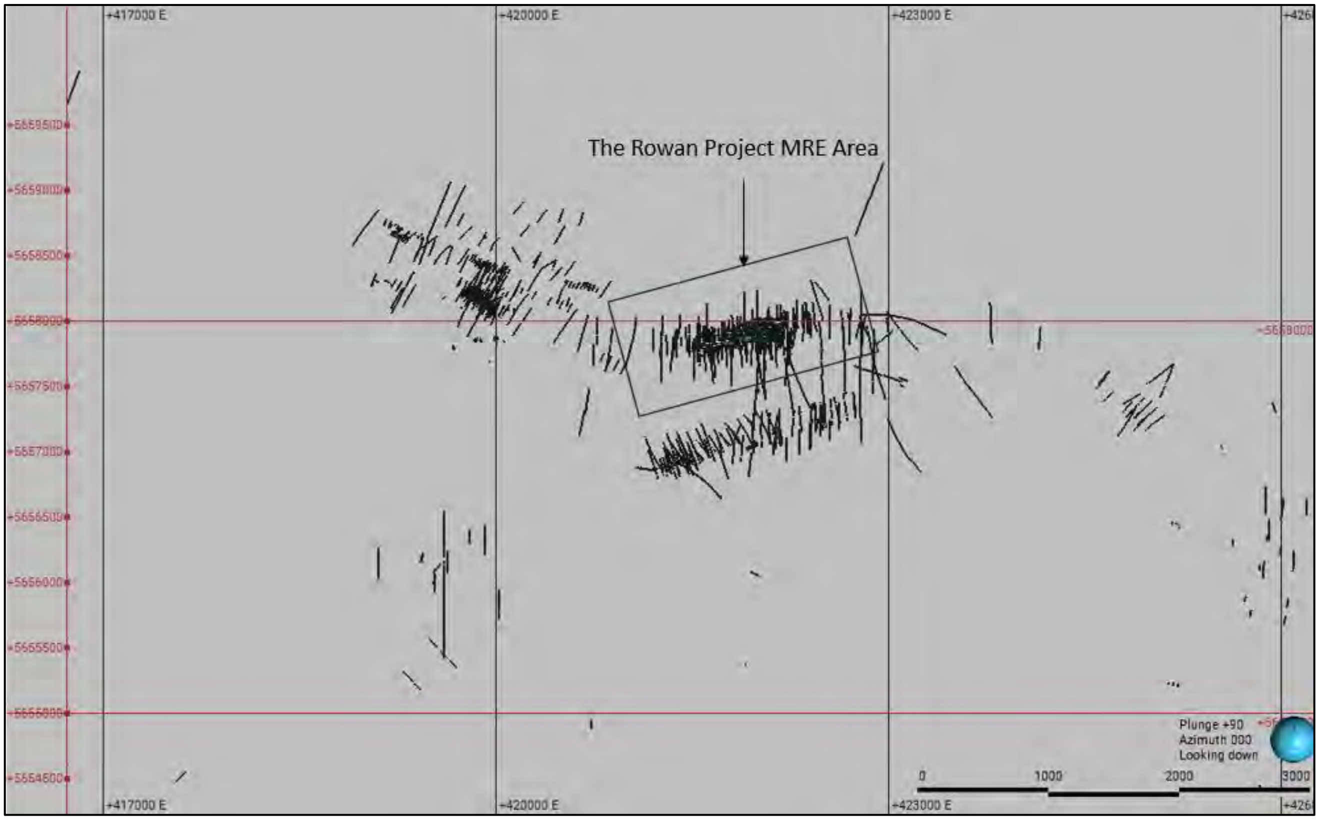


Source: SR 2025.
Notes: WRLG = West Red Lake Gold Mines Ltd.

14.3 Database

The database was supplied by WRLG and is currently managed in Geotic database software. WRLG transitioned to a DataShed (Maxgeo) database management system in 2024. WRLG provided a Leapfrog Project file (Rowan_MRE_2_23_2024) that the QP reviewed and used as the basis for the 2025 mineral resource estimate. The Rowan Leapfrog Project data used for the 2025 mineral resource estimate contains 304 diamond drill holes, with 46,148 raw assay records from the Rowan mineral resource estimate area (Figure 14-2). The QP maintained all standard tables, including coded lithology and intercepted zones within the Leapfrog Project used for the mineral resource estimate, and is of the opinion that the Rowan Project data is suitable for resource estimation.

Figure 14-2: Plan View of Drill Holes in Leapfrog Software for the Rowan Deposit



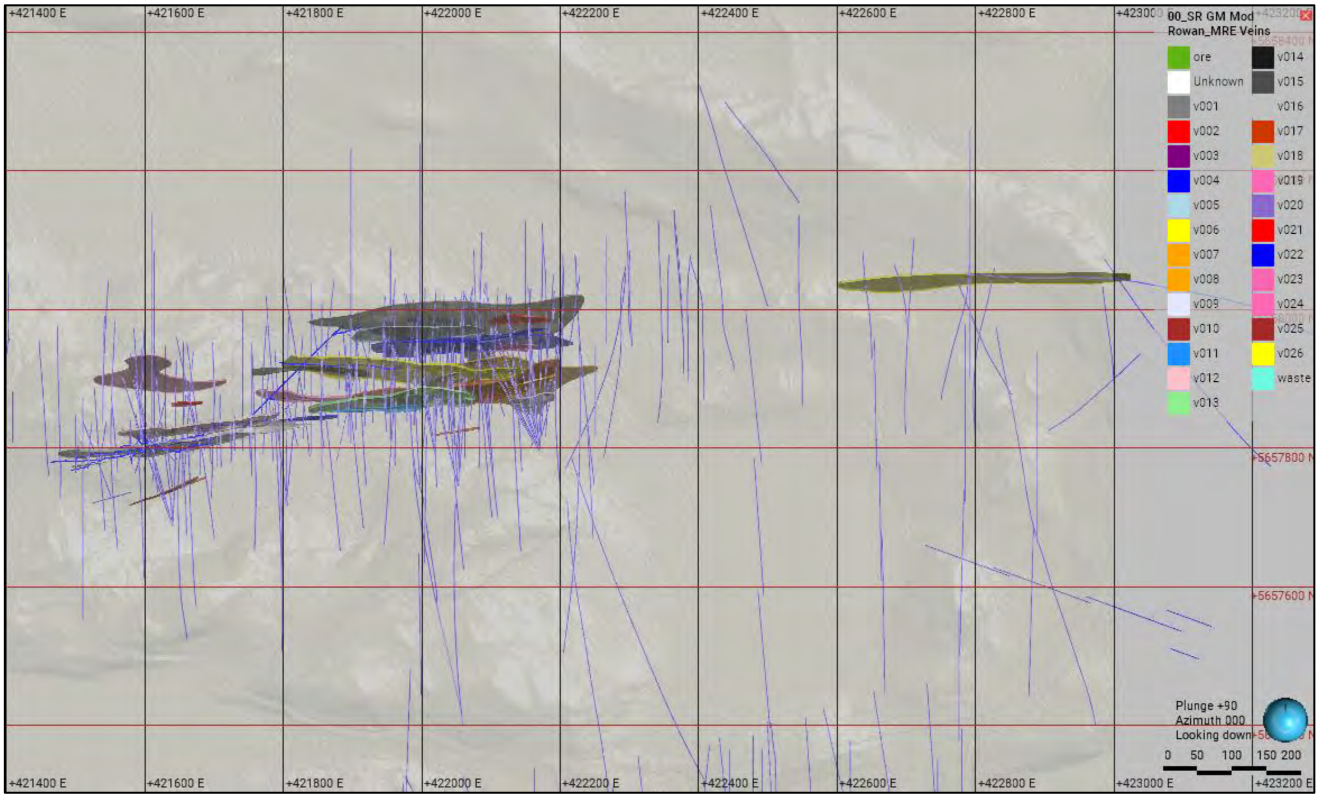
Source: SR 2025.
Notes: MRE = Mineral Resource Estimate.

14.4 Domain Interpretation

WRLG geologists completed a first-pass wireframe model of the Rowan gold mineralization in Leapfrog using both implicit and vein modelling tools. The mineral domains were interpreted to consider the drill hole assay data within the context of the Rowan deposit lithological and structural modelling. Receptive contacts between veins and lithologies were the primary controls on the modelling of the gold domains. For the Rowan mineral resource estimate domains, 2.0 m width criteria was used, which adds dilution to the narrow zones. The QP reviewed and completed small revisions on the resultant WRLG wireframes and finalized the gold domains using the same method.

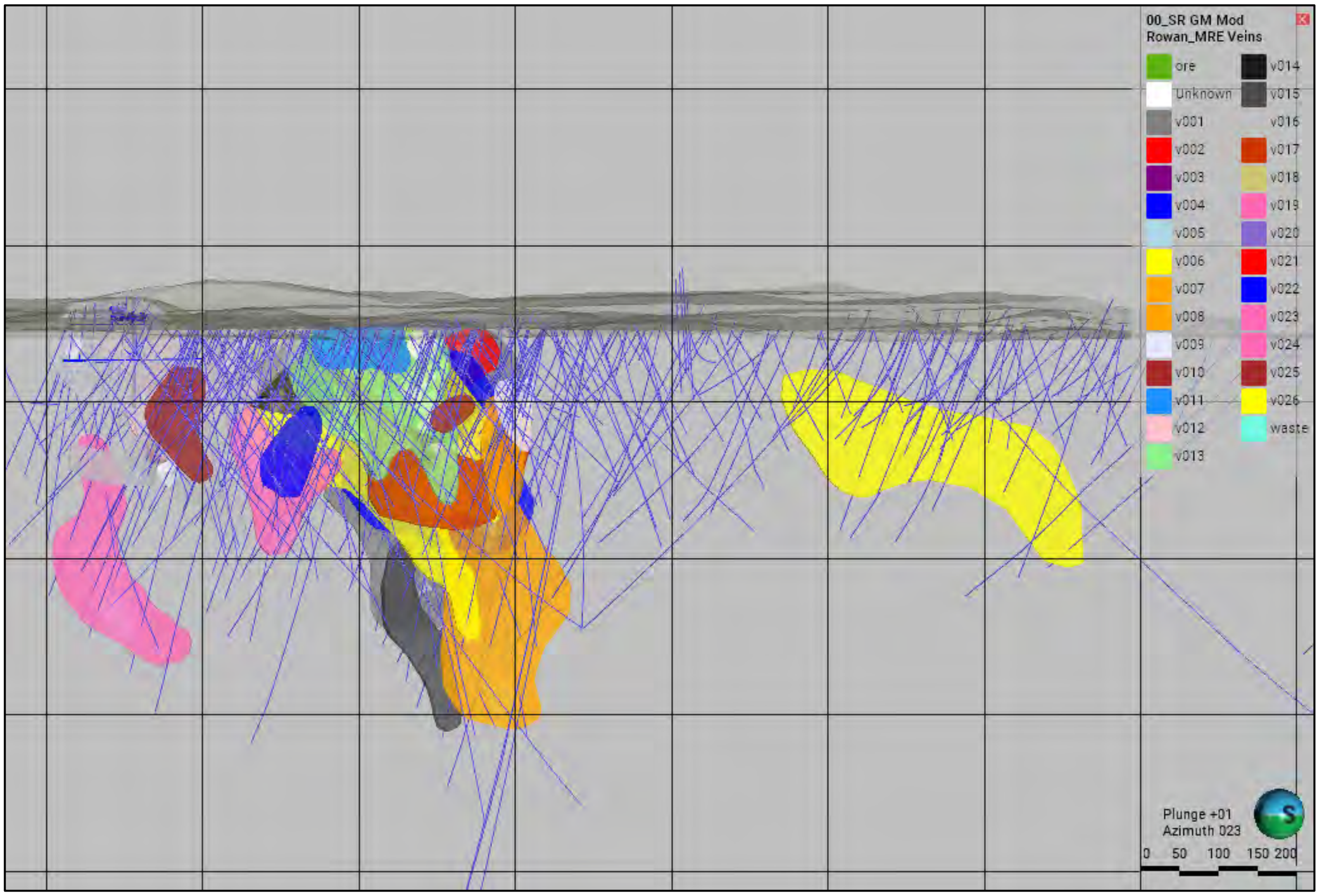
A total of 26 individual domain wireframes (i.e., v001 to v026) were created for the Rowan deposit and used for the 2025 mineral resource estimate completed by the QP. A plan view and longitudinal section view showing examples of the gold domains are shown on Figure 14-3 and Figure 14-4, respectively. The QP notes the historical underground workings have a very small volume when intersecting the Rowan veins; however, the QP recommends that additional work is completed in these historical areas for the next model update as there are likely mined-out areas that not reflected in the current underground wireframes.

Figure 14-3: Plan View of Mineralized Domains for the Rowan Deposit



Source: SR 2025.

Figure 14-4: Longitudinal-Section View of Vein Domains for the Rowan Deposit – Looking North

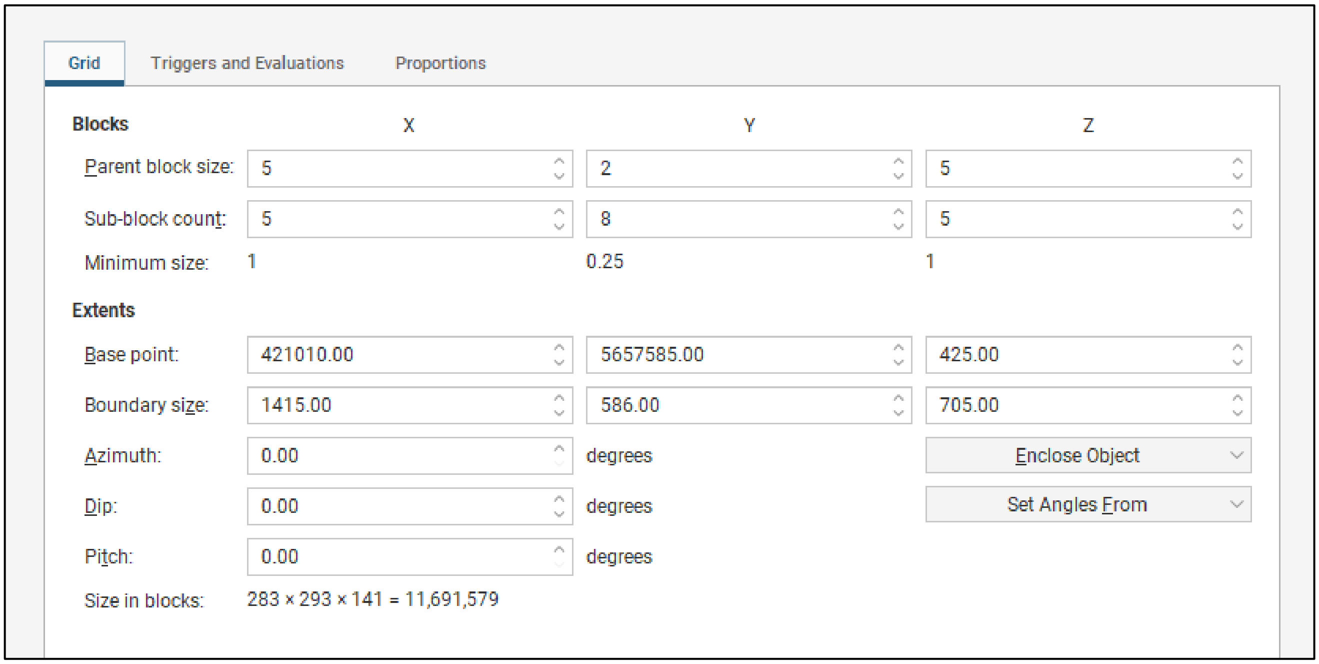


Source: SR 2025.

14.5 Block Model Geometry

Figure 14-5 defines the origin, block size, and orientation of the block model used by the QP for the 2025 mineral resource estimate.

Figure 14-5: Block Model Geometry Parameters for the 2025 Mineral Resource Estimate



Source: SR 2025.

14.6 Cut-off Grade Estimation

The calculated reported cut-off grade for the 2025 mineral resource estimate was based on the parameters outlined in Table 14-2. Costs were estimated based on recent technical reports for underground lode gold operations with milling rates ranging between 1,000 tonnes per day (t/d) and 1,500 t/d. The gold price is based on a five-year average. There are Net Smelter Return (NSR) royalties on certain claims held by RLG. The estimated resources are all contained within claims having a royalty of 3% to Jamie Frontier.

As additional metallurgical testing results become available, the cut-off grade may require adjusting to account for changes in processing costs and gold recovery. It is the QP's opinion that the calculated cut-off grade is reasonable for use in a resource estimation.

Table 14-2: Cut-off Grade Parameters for the 2025 Mineral Resource Estimate

Parameter	Unit	2025 Base Case (Long Hole) $1,600[a]
Mining Cost	C$/t	108.6
Milling	C$/t	73.3
General and Administrative	C$/t	54.1
Total	**C$/t**	**235.99**
Metal Price	US$/oz Au	1,600
Recovery	%	95
Cut-off Grade	**g/t Au**	**3.80**

Notes:
a) These operating costs were estimated assuming long-hole mining method and US$1,600/oz gold.
C$/t = Canadian dollars per tonne; US$/oz = United States dollars per ounce; g/t = grams per tonne; Au = gold.

14.7 Bulk Density Estimation

Bulk density (i.e., specific gravity) measurements were collected by WRLG geology staff during the 2023 drilling program. A total of 723 density measurements were collected across all lithologies and a number of mineralized zones. Taking an average of densities recorded across all 26 mineralized domains results in an average specific gravity of 2.85 grams per cubic centimetre (g/cm^3). A specific gravity of 2.8 g/cm^3 was used by the QP for the tonnage calculations on the current mineral resource. This specific gravity value is within the range for typical Archean age volcanic-hosted gold deposits and is suitable for use in a resource estimation.

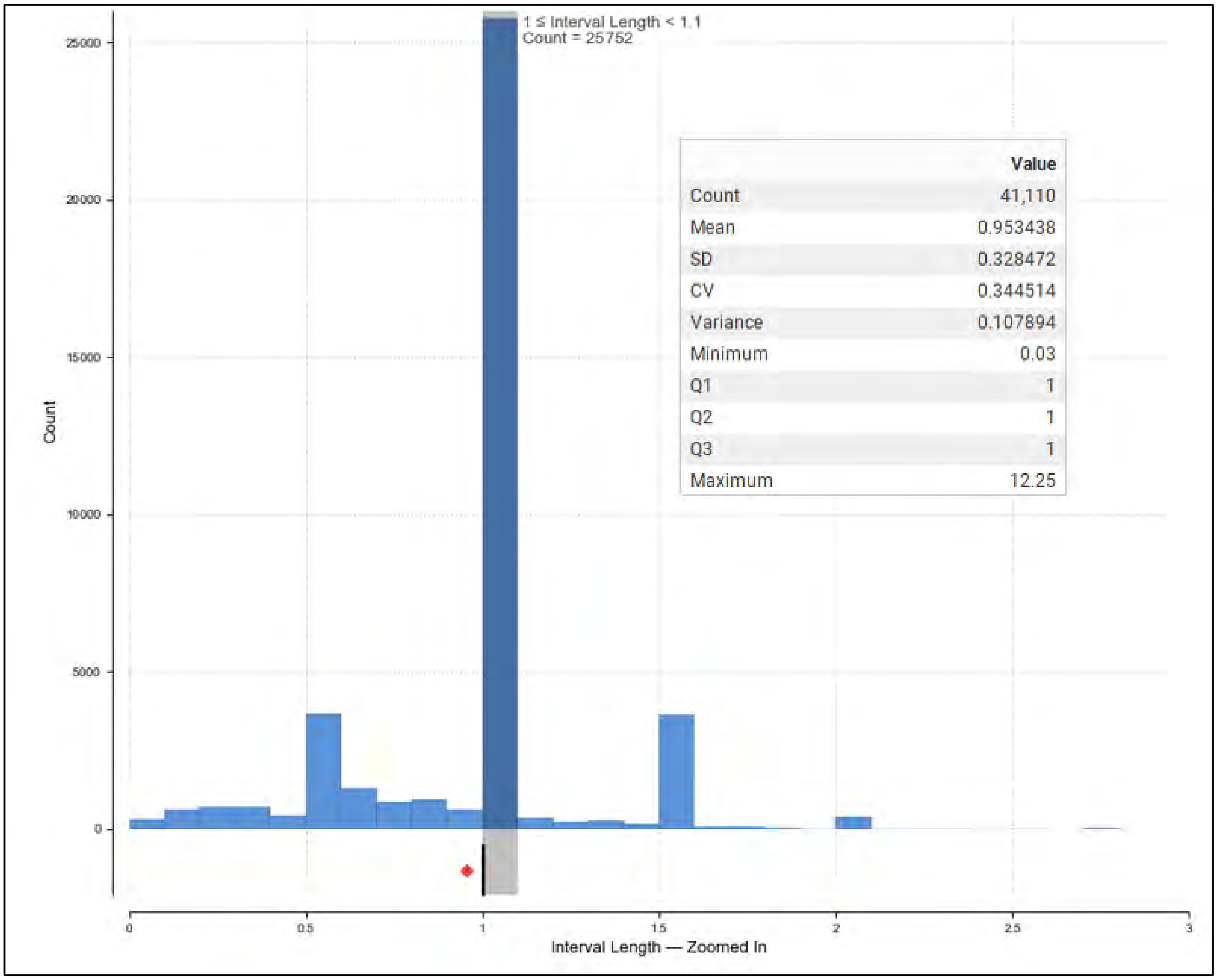
14.8 Assay Compositing

The assay data confirms the narrow width of the mineralized zones in the Rowan deposit, and the contact between mineralized and non-mineralized material is very sharp. For the Rowan mineral resource estimate domains, 2.0 m width criteria was used, which adds dilution to the narrow zones.

The QP reviewed the raw gold assay lengths in the mineralized domains using the drill hole table (i.e., Drillholes_JK22_WRLG23/WRLG_ASSAY_9) in Leapfrog and found that the mean length was approximately 1.0 m (Figure 14-6). The QP composited the raw gold assay data to 2.0 m composites within each domain. The compositing parameters from the Leapfrog output is provided in Table 14-3.

The drill hole table, Drillholes_JK22_WRLG23, used for the mineral resources estimates is a product of SRK and included drill collar corrections and insertions of half of the detection limit (i.e., 0.0025 g/t Au) for unsampled intervals. The QP believes that this practice is reasonable for unsampled intervals as using only high-grade sampled intervals would skew the exploratory data analysis and mineral resources estimate results. The QP also recommends that some areas are drilled where there are high-grade gold intervals without sampled intervals within the estimation domains; this additional drilling may assist in gold-grade distributions for the mineral resources estimates and mine planning.

Figure 14-6: Raw Gold Assay Lengths within Model Domains for the Rowan Deposit



Source: SR 2025.
Notes: ≤ = less than or equal to; < = less than; SD = standard deviation; CV = coefficient of variation.

Table 14-3: 2025 Model Domain Compositing Output Parameter File from Leapfrog for the Rowan Deposit

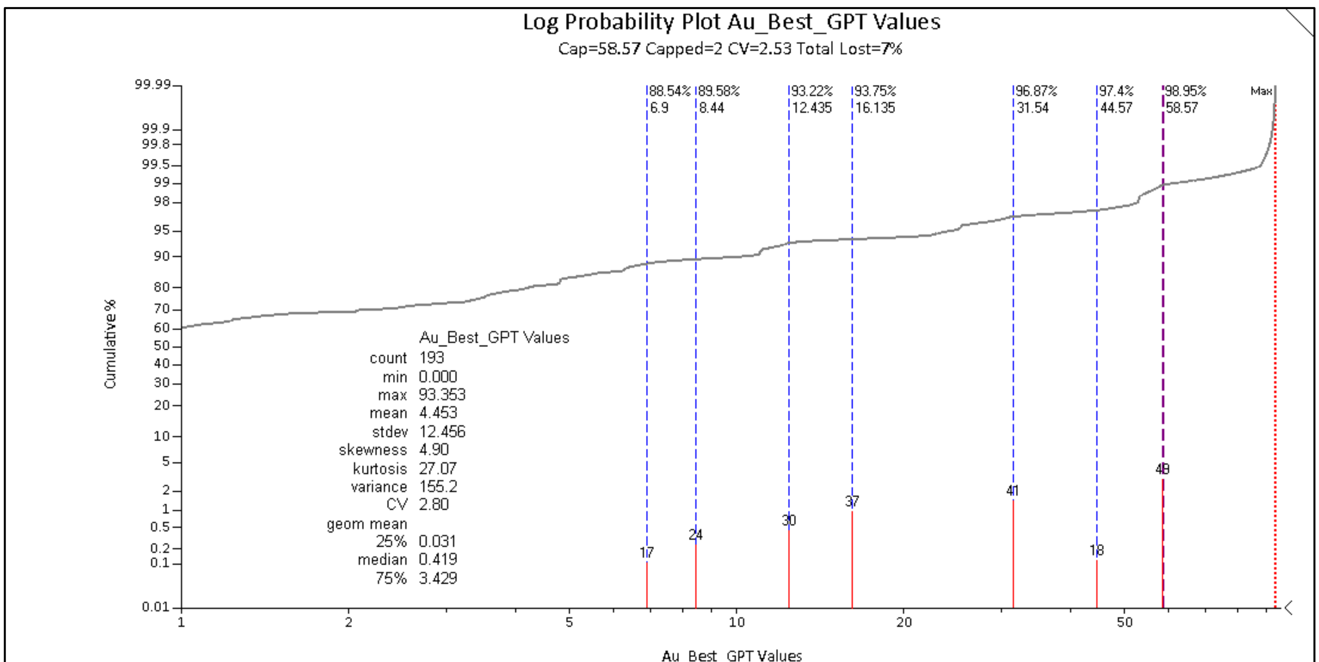
General		Boundary	Compositing				
Name	Source	Boundary Type	Composite in	Composite Length	Residual End Length	End Length Handling	Minimum Coverage
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v001	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v002	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v003	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v004	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v005	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v006b	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	1.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v007	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v008	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v009	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v010	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v011	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v012	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v013	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v014	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v015	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v016	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v017	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v018	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v019	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v020	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v021	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v022	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v023	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v024	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v025	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0
Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v026	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	Hard	Within boundary	2.0	0.5	Distributed equally	0

14.9 Capping Analyses

Erratic high-grade assay values can have a disproportionate effect on the average grade of a deposit when the assay distribution is skewed positively or approaches log-normal. One method of treating these outliers to reduce their influence on the average grade is to cap or restrict them at a specific grade level and/or search distance.

The QP is of the opinion that the influence of high-grade gold assays must be reduced or controlled by using a number of industry best practice methods, including capping of high-grade values at the composite level. The QP used disintegration analysis by domain on the 2.0 m composites to determine an appropriate gold capping grade. Figure 14-7 shows the disintegration analysis for model domain v006b; this analysis uses a 15% step function to find population breaks for the capping analysis. This method provides granularity on the cumulative distribution function and allows for step function changes to test capping levels for use in the mineral resource estimate. The capped and uncapped gold grade statistics applied to the 2025 mineral resource estimate for each domain is provided in Table 14-4.

Figure 14-7: Disintegration Analysis Used to Cap Composites in the Model Domains



Column	Filter	Cap	Capped	Percentile	Capped%	Lost Total%	Lost CV%	Count	Min	Max
Au_Best_GPT V...								193	0	93.35
Au_Best_GPT V...		58.57	2	98.95%	1%	7%	10%	193	0	58.57
Au_Best_GPT V...		44.57	5	97.4%	2.6%	14%	16%	193	0	44.57
Au_Best_GPT V...		31.54	6	96.87%	3.1%	23%	25%	193	0	31.54
Au_Best_GPT V...		16.135	12	93.75%	6.2%	41%	39%	193	0	16.14
Au_Best_GPT V...		12.435	13	93.22%	6.7%	47%	43%	193	0	12.44

Source: SR 2025.
Notes: Au = gold; GPT = grams per tonne; CV = coefficient of variation.

Table 14-4: Capping Levels for the 2025 Mineral Resource Estimate

Domain	Max Uncap	Cap	# Capped	Percentile	Capped %	Lost Total %	Lost CV%	Minimum	Mean Uncap	Mean Cap	CV Uncap	CV Cap
v001	280.69	280.69	Clamp D25%, 86 g/t	-	-	-	-	0.003	8.97	7.78	3.71	3.24
v002	3.95	2.74	1	87.5	11.1	14	8	0.003	0.99	0.85	1.44	1.33
v003	45.26	26.13	2	87.5	22.2	24	8	0.001	8.92	6.80	1.81	1.66
v004	106.68	96.80	1	98.92	1.1	2	2	0.001	5.42	5.31	2.99	2.93
v005	21.50	21.50	N/C	-	-	-	-	0.001	2.10	-	2.34	-
v006b	93.35	58.57	Clamp D35%, 58 g/t	-	-	-	-	0.001	4.45	3.54	2.80	2.14
v007	7.58	6.50	2	95.24	9.1	4	3	0.001	1.26	1.21	1.71	1.65
v008	9.81	7.06	2	94.28	5.6	12	4	0.001	1.13	0.99	2.32	2.21
v009	4.40	0.39	1	93.75	5.9	74	54	0.001	0.32	0.08	3.33	1.54
v010	5.67	0.05	1	75.00	20	97	71	0.002	1.16	0.03	2.18	0.63
v011	11.43	6.78	1	93.34	6.3	11	14	0.001	2.70	2.41	1.16	0.99
v012	15.77	8.74	3	96.47	3.5	11	14	0.003	1.82	1.62	1.70	1.47
v013	44.80	24.90	2	97.29	2.7	12	15	0.001	4.38	3.87	1.93	1.64
v014	12.34	7.49	2	94.44	5.4	14	11	0.001	1.71	1.47	1.84	1.64
v015	18.64	12.42	1	95.83	4.0	11	7	0.003	2.33	2.08	2.05	1.91
v016	48.76	16.61	2	98.94	1.1	16	27	0.003	1.34	1.12	3.35	2.44
v017	45.84	9.62	3	97.00	3.0	36	34	0.003	1.28	0.82	4.00	2.62
v018	68.56	16.55	1	92.86	6.7	41	36	0.003	8.52	5.05	2.07	1.33
v019	66.62	6.54	1	88.89	10.0	68	61	0.090	8.77	2.76	2.33	0.90
v020	24.60	7.29	2	75.00	40.0	44	32	0.200	7.81	4.35	1.28	0.87
v021	2.83	-	N/C	-	-	-	-	0.003	0.63	-	2.83	-
v022	71.21	37.65	1	83.34	14.3	28	12	0.009	17.17	12.37	1.59	1.41
v023	10.45	6.04	1	95.65	4.2	8	11	0.003	2.19	2.00	1.22	1.09
v024	24.70	10.84	1	93.34	6.3	22	23	0.003	3.89	3.02	1.65	1.26
v025	4.15	3.07	2	83.34	15.4	8	16	0.730	1.90	1.74	0.65	0.55
V026	37.78	17.47	1	83.34	14.3	32	9	0.007	7.73	5.25	1.75	1.59

Notes:
CV = coefficient of variation; g/t = grams per tonne; - = not applicable; N/C = no cap.

14.10 Grade Estimation

The software formats used by the QP for the Rowan exploratory data analysis and mineral resource estimate were X10 by Phinar Software and Leapfrog Geo version 2024.1.

A multiple 'hard' domain model was constructed using previously defined drill hole composite intervals. To avoid domains with a large percentage of low-grade waste blocks, the domain extents were also limited to projecting the shape approximately 25 m or halfway to the next low-grade or waste data point, whichever was less, along strike and dip. Each zone was estimated from composites tagged as being from that specific domain, which allowed composites selections from within a hard boundary. The 2025 mineral resource estimate is based on a 3D block model interpolated using an ID3 method for 2.0 m composite gold grades.

A single pass interpolation process was used to populate the block model with gold grades. This process limits the smearing of higher-grade values while concentrating the values closer to the data source. The estimation parameters and search parameters used by the QP for the Rowan 2025 interpolation process are provided Table 14-5 and Table 14-6, respectively. The QP used a high-grade transition (i.e., clamping) method for model domains v001 and v006b as these two domains contain high-grade composites that WRLG believes are continuous within the mineralized shoots. The QP recommends that these two areas are drill tested to confirm the use of a high-grade restriction instead of grade capping or additional domaining work.

Table 14-5: 2025 Mineral Resource Estimate Estimation Parameters for the Rowan Deposit

General		Value Clipping		Estimator	IDW Options
Estimator Name	Source	Lower Bound	Upper Bound	Estimate Type	Estimate Type
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v001 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	Clamp	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v001 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	No Clamp	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v002 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	2.74	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v002 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v003 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	26.13	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v003 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v004 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	96.8	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v004 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v005 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	21.5	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v005 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v006b Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	Clamp	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v006b Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	No Clamp	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v007 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	6.5	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v007 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v008 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	7.1	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v008 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v009 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	0.4	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v009 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v010 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	0.05	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v010 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v011 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	6.78	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v011 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v012 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	8.7	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v012 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v013 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	25	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v013 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v014 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	7.5	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v014 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v015 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	12.4	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v015 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v016 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	16.6	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v016 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v017 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	9.6	IDW	3

Table 14-5: 2025 Mineral Resource Estimate Estimation Parameters for the Rowan Deposit (continued)

General		Value Clipping		Estimator	IDW Options
Estimator Name	Source	Lower Bound	Upper Bound	Estimate Type	Estimate Type
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v017 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v018 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	16.5	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v018 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v019 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	6.5	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v019 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v020 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	7.3	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v020 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v021 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v021 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v022 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	37.6	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v022 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v023 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	6.04	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v023 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v024 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	10.84	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v024 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v025 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	3.07	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v025 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v026 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	17.7	IDW	3
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v026 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	IDW	3
NN, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v001	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	-	NN	3
NN, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v004	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	0.001	96.8	NN	3

Notes:
IDW = inverse distance; - = not applicable; NN = near neighbour.

Table 14-6: 2025 Mineral Resource Estimate Search Parameters for the Rowan Deposit

General		Ellipsoid Ranges			Ellipsoid Directions			Ellipsoid Orientation	No. of Samples		Outlier Restrictions			Drillhole Limit
Estimator Name	Source	Max	Inter	Min	Dip	Dip Azi.	Pitch	VO	Min	Max	Method	Distance	Threshold	Max Samp
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v001 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	70	30	-	-	-	VO	4	15	Clamp	25	86	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v001 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	70	30	-	-	-	VO	4	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v002 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	60	40	20	75	180	80	None	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v002 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	60	40	20	75	180	80	None	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v003 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	35	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v003 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	35	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v004 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	130	100	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v004 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	130	100	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v005 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	75	65	20	85	180	80	None	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v005 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	75	65	20	85	180	80	None	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v006b Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	120	35	-	-	-	VO	4	10	Clamp	35	58	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v006b Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	120	35	-	-	-	VO	4	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v007 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	80	60	20	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v007 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	80	60	20	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v008 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	60	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v008 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	60	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v009 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	110	90	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v009 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	110	90	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v010 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	60	40	20	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v010 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	80	60	20	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v011 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	110	90	25	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v011 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	25	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v012 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	60	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v012 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	80	60	20	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v013 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	120	80	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v013 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	120	80	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v014 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	120	90	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v014 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	120	90	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v015 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	140	120	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v015 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	140	120	35	-	-	-	VO	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v016 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	140	120	35	-	-	-	VO	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v016 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	140	120	35	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v017 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v017 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v018 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v018 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v019 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	3	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v019 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v020 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v020 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	100	80	30	-	-	-	VO	2	15	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v021 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	130	40	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v021 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	130	40	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v022 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	130	40	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v022 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	130	40	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v023 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	130	40	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v023 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	130	40	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v024 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	140	50	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v024 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	140	50	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v025 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	140	50	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v025 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	150	140	50	-	-	-	VO	2	10	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v026 Capped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	300	300	75	-	-	-	VO	2	5	None	-	-	2
ID, Au_Best_GPT in 00_SR GM Mod Rowan_MRE: v026 Uncapped	Drillholes_JK22_WRLG23: WRLG_ASSAY_9	300	300	75	-	-	-	VO	2	5	None	-	-	2

Notes:

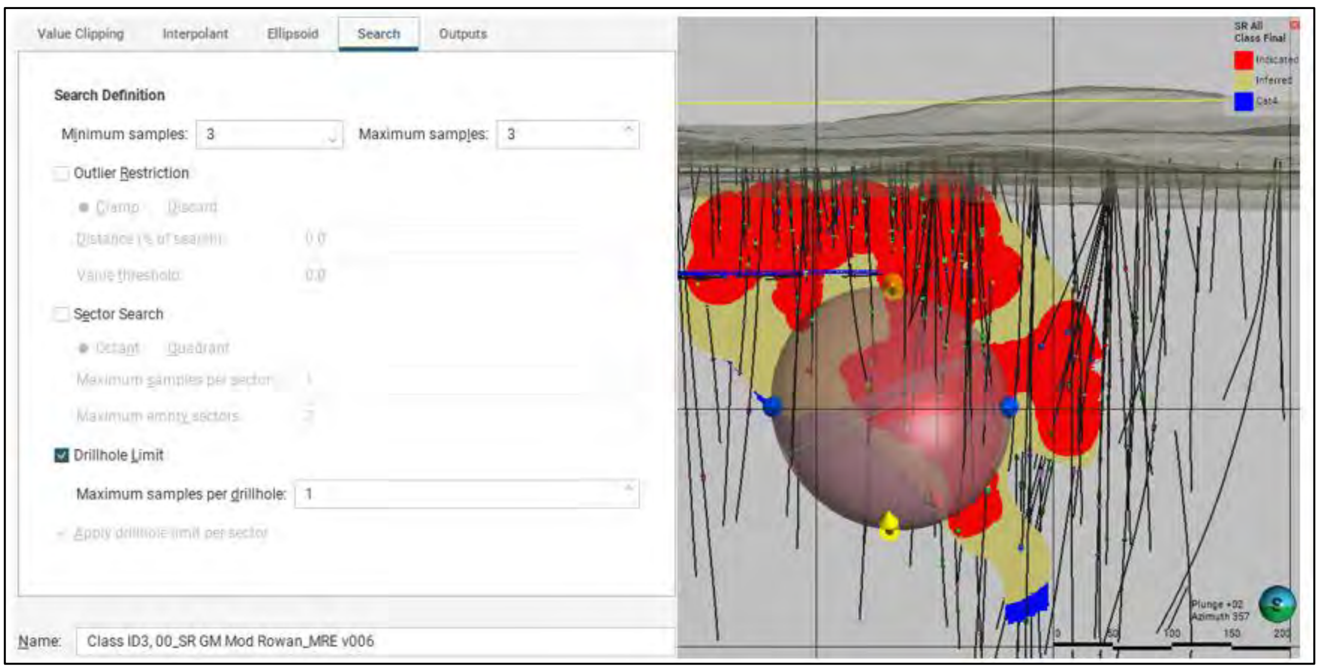
VO = variable orientation; - = not applicable.

14.11 Mineral Resource Estimate Classification

The mineral resource estimate classification used the 2.0 m composited data by model domain and was based on a Euclidean distance approach that uses one sample from three drill holes where the average distance is divided by 0.707 to solve the triangulated distances spatially. This approach provided a tool to test variable distances from the variography and help eliminate the 'spotted dog' effect. Figure 14-8 and Figure 14-9 show the QP's classification for model domain v006 based on gold interpolated distances and Euclidean calculations to produce a classified model for reporting the 2025 mineral resource estimate. Classification results used the following criteria:

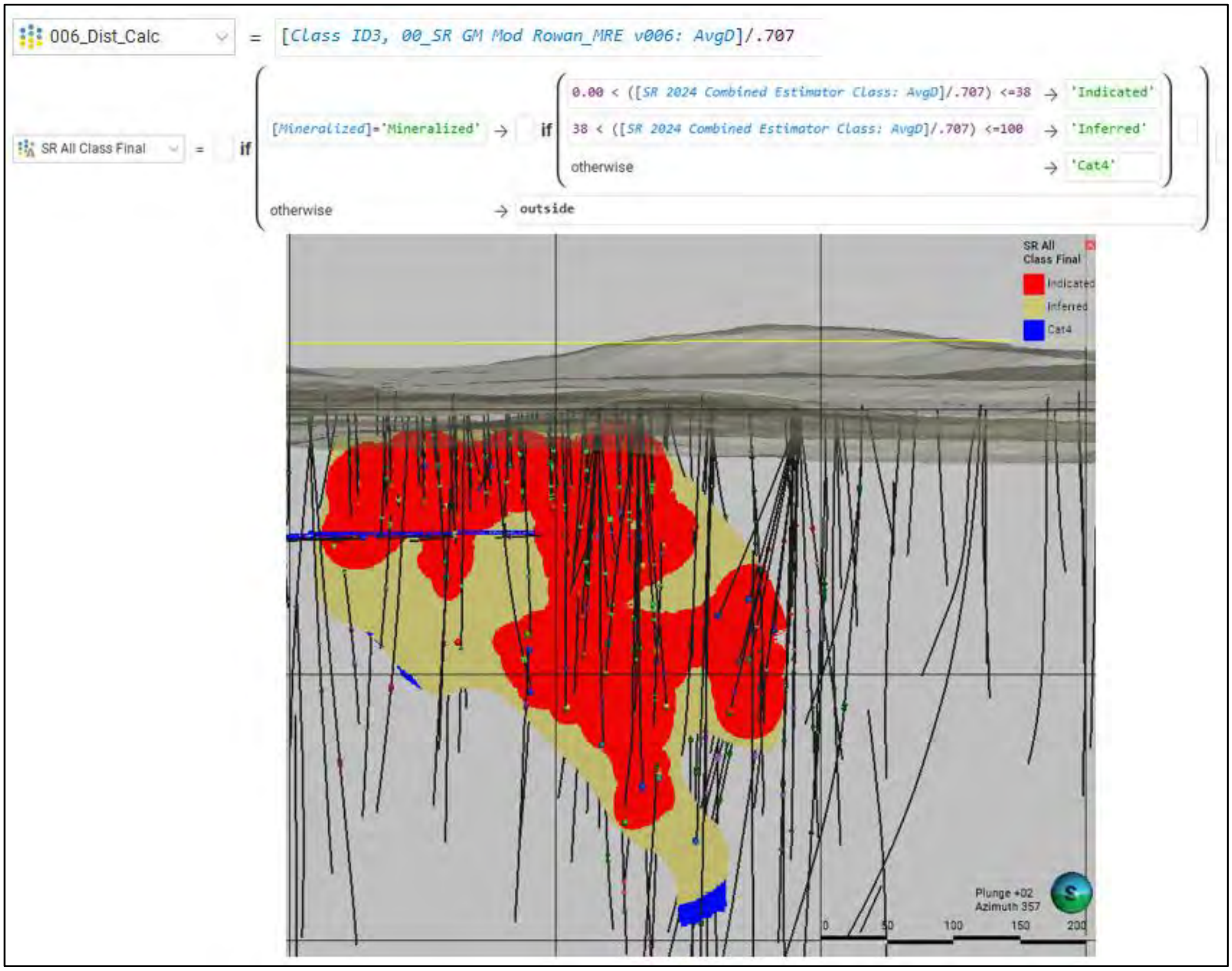
- Indicated: 0 m < and ≤ 38 m;
- Inferred: < 38 m and ≤ 100 m; and
- Category 4 (i.e., exploration potential): > 100 m.

Figure 14-8: Search and Sample Criteria Used for Classification for Model Domain v006b – Looking North



Source: SR 2025.

Figure 14-9: Euclidean Distance Calculations and Indicated and Inferred Blocks for Model Domain v006b – Looking North



Source: SR 2025.

14.12 Resource Estimates

A summary of the 2025 mineral resource estimate is presented in Table 14-7 and is prepared in accordance with CIM (2014) definitions. 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 2025 mineral resource estimate.

Table 14-7: 2025 Mineral Resource Estimate Summary for the Rowan Deposit as of 30 June 2025

Category	Tonnage [t]	Average Grade [g/t Au]	Contained Metal [oz Au]
Indicated	478,707	12.78	196,747
Inferred	421,181	8.73	118,155

Notes:
- CIM (2014) definitions were followed for mineral resources.
- Mineral resources were estimated at a gold cut-off grade of 3.80 g/t using a long-term gold price of US$1,800 per ounce.
- There are no mineral reserves currently estimated at the Rowan deposit.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Mineral resources are reported within vein wireframes at the stated cut-off grade of 3.80 g/t Au.
- Density of 2.8 g/cm^3.
- Numbers may not add due to rounding.

t = tonne; g/t = grams per tonne; Au = gold; oz = ounce; g/cm^3 = grams per cubic centimetre.

A summary of the Rowan 2025 mineral resource estimate by model domain is provided in Table 14-8. Some domains (i.e., v002, v009, v010, v014, v021, and v025) do not contribute to the mineral resource estimate statement at the 3.80 g/t gold cut-off grade but do contribute at lower gold cut-off grades.

Table 14-8: 2025 Mineral Resource Estimate by Model Domain at the Gold Cut-Off Grade (3.80 grams per tonne)

Model Domain	SR All Class Final	Mass [t]	Average Value - Gold Final Capped [g/t]	Material Content - Gold Final Capped [oz]
v001	Indicated	140,895	22.65	102,590
	Inferred	81,797	11.94	31,390
v003	Indicated	2,475	9.72	773
	Inferred	23,636	7.32	5,562
v004	Indicated	109,639	11.12	39,185
	Inferred	58,329	9.65	18,090
v005	Indicated	7,025	6.65	1,502
	Inferred	4,678	5.50	827
v006b	Indicated	103,304	7.39	24,541
	Inferred	26,046	7.05	5,903
v007	Indicated	2,561	4.48	369
	Inferred	2,998	4.49	432
v008	Indicated	2,141	4.47	308
v011	Indicated	725	5.06	118
v012	Indicated	17,777	5.11	2,922
	Inferred	1,495	4.03	194
v013	Indicated	28,840	6.33	5,871
	Inferred	8,425	5.83	1,580
v015	Indicated	130	10.05	42
	Inferred	60,701	5.51	10,760
v016	Indicated	14,458	8.68	4,033
	Inferred	28,636	7.24	6,661
v017	Indicated	9,776	4.97	1,563
	Inferred	5,797	4.47	834
v018	Indicated	12,800	8.81	3,625
	Inferred	367	6.01	71
v019	Indicated	706	4.57	104
	Inferred	5,353	4.52	778

Table 14-8: 2025 Mineral Resource Estimate by Model Domain at the Gold Cut-Off Grade (3.80 grams per tonne) (continued)

Model Domain	SR All Class Final	Mass [t]	Average Value - Gold Final Capped [g/t]	Material Content - Gold Final Capped [oz]
v020	Indicated	9,594	5.79	1,787
	Inferred	1,847	5.76	342
v022	Indicated	8,405	22.26	6,014
	Inferred	35,011	13.62	15,335
v023	Indicated	4,793	4.93	759
	Inferred	16,937	4.33	2,358
v024	Indicated	2,665	7.50	642
	Inferred	26,874	5.41	4,678
v026	Inferred	32,255	11.92	12,358
Total	**Indicated**	**478,707**	**12.78**	**196,747**
	Inferred	**421,181**	**8.73**	**118,155**

Notes:
- CIM (2014) definitions were followed for mineral resources.
- Mineral resources were estimated at a gold cut-off grade of 3.80 g/t using a long-term gold price of US$1,800 per ounce.
- There are no mineral reserves currently estimated at the Rowan deposit.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Mineral resources are reported within vein wireframes at the stated cut-off grade of 3.80 g/t Au.
- Density of 2.8 g/cm^3.
- Numbers may not add due to rounding.

SR = SIMS Resources LLC; t = tonne; g/t = grams per tonne; Au = gold; oz = ounce; g/cm^3 = grams per cubic centimetre.

14.13 Resource Block Model Validation

The block model was validated using several industry standard methods, including:

- Visual validation by comparing block estimates to composite gold values on cross-sections and plan view sections.
- Metal loss by comparison of capped and uncapped gold grades by model domain.
- Swath plot comparison of ID3 (capped and uncapped) and near neighbour estimations against 2.0 m capped gold composites.

The 2025 block model for the Rowan deposit was examined in section views and 3D views. Blocks were queried to confirm domain identification within the selected shapes. Composite domain identification was confirmed. The boundary conditions between blocks and domain solids were checked. Block grade values were visually compared to composite point gold values within each domain. In the QP's opinion, the block gold grades did not show significant estimation issues.

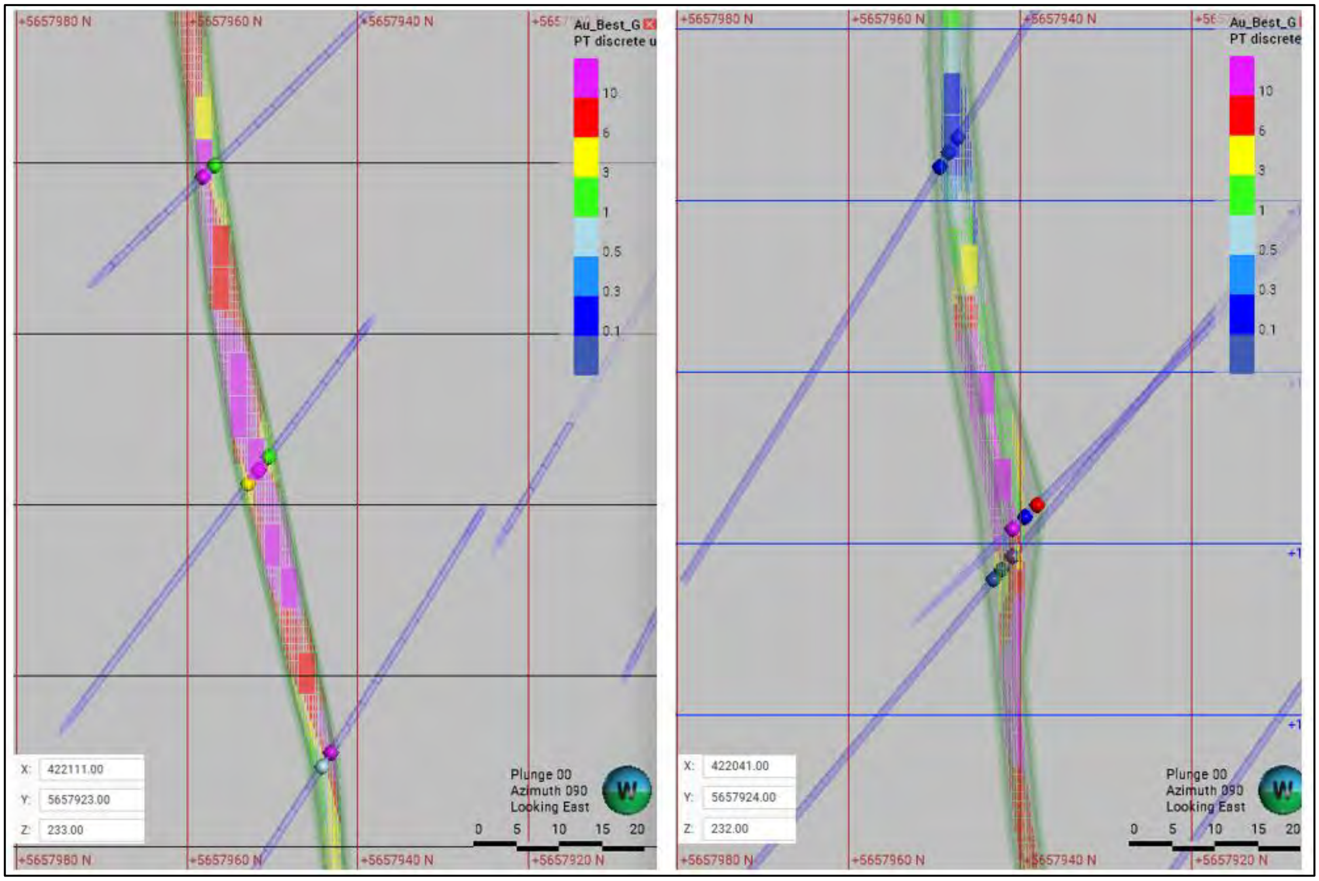
14.13.1 Mineral Resource and Mining Reconciliation

The available historical records are not adequate to conduct a comparison with the resource model.

14.13.2 Visual Section Validation

The capped block gold grades were visually compared to the capped 2.0 m composite gold grades within each model domain (Figure 14-10). The block gold grades did not show significant estimation issues, however, the QP recommends infill drilling to determine mineralized shoot geometries to provide more granularity on the gold distributions and validate future mineral resource estimation.

Figure 14-10: Vertical Section Views for Model Domain, v004

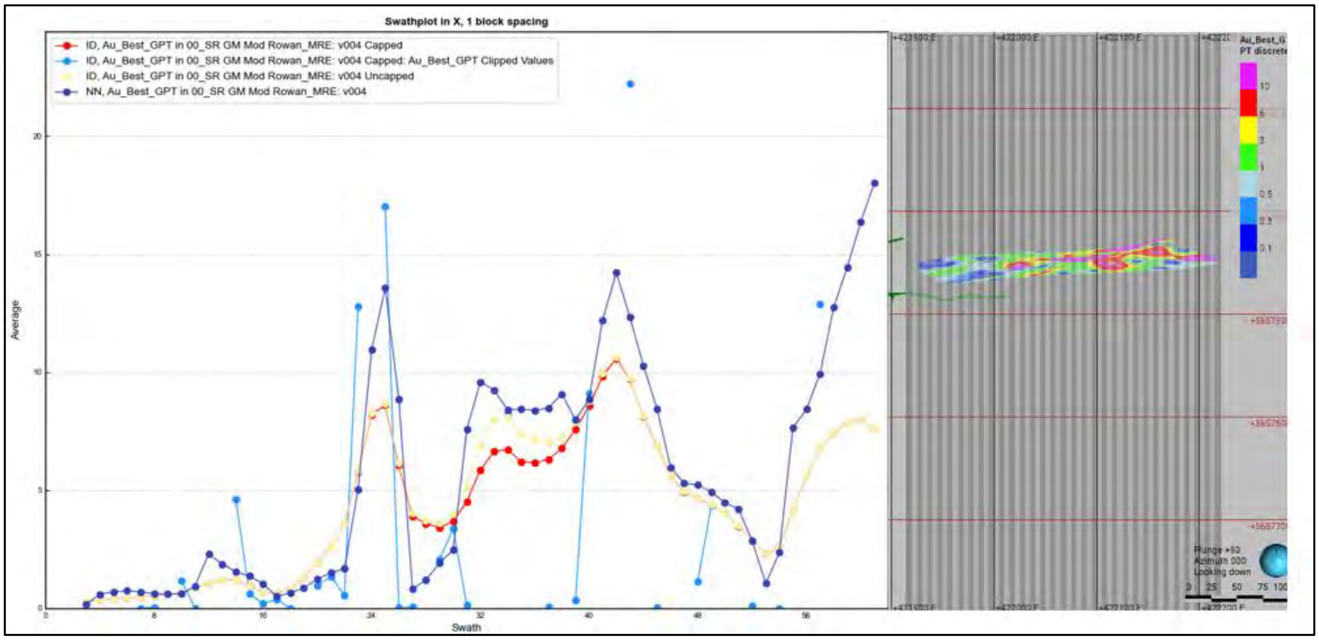


Source: SR 2025.

14.13.3 Swath Plot Validation

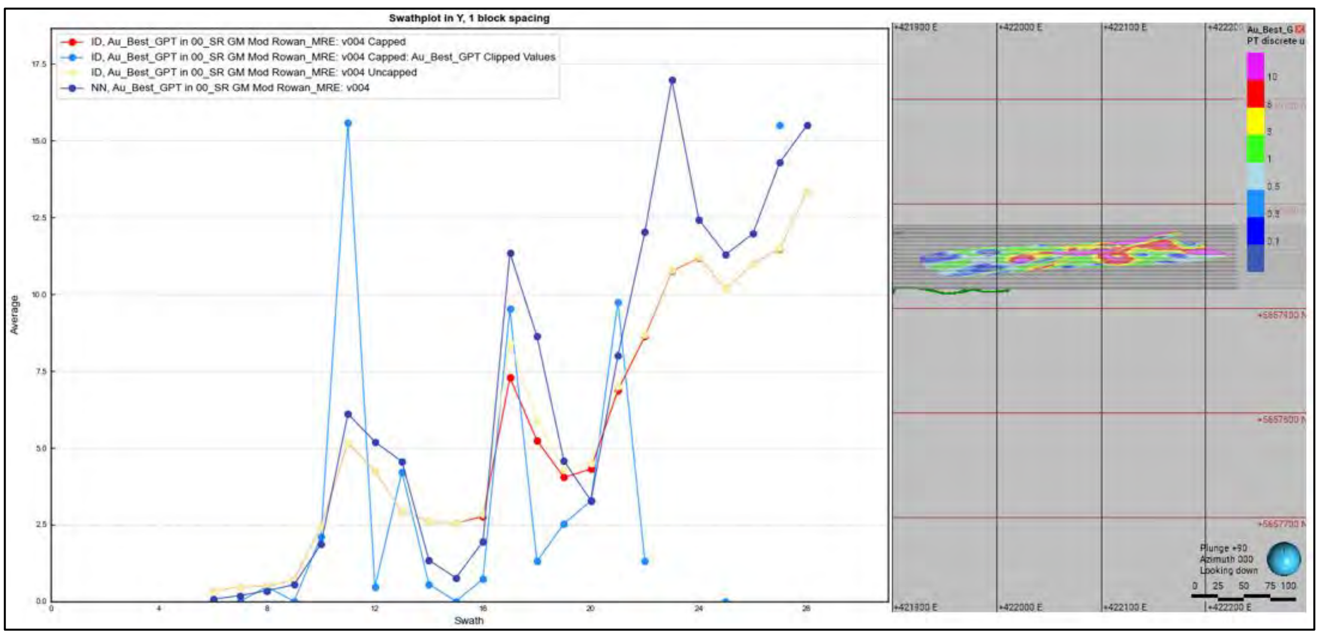
The 2025 Rowan block model was examined by domain using swath plots to compare the ID3 (capped and uncapped) and near neighbour estimations (Figure 14-11, Figure 14-12, and Figure 14-13). The QP believes that the models are comparable, and no estimation issues are apparent. The QP believes the composite gold grades in the swath plots are capped at the appropriate level; however, high-grade transitions should be explored by model domain in the next mineral resource estimate update.

Figure 14-11: Swath Plot for Model Domain v004 – X Direction



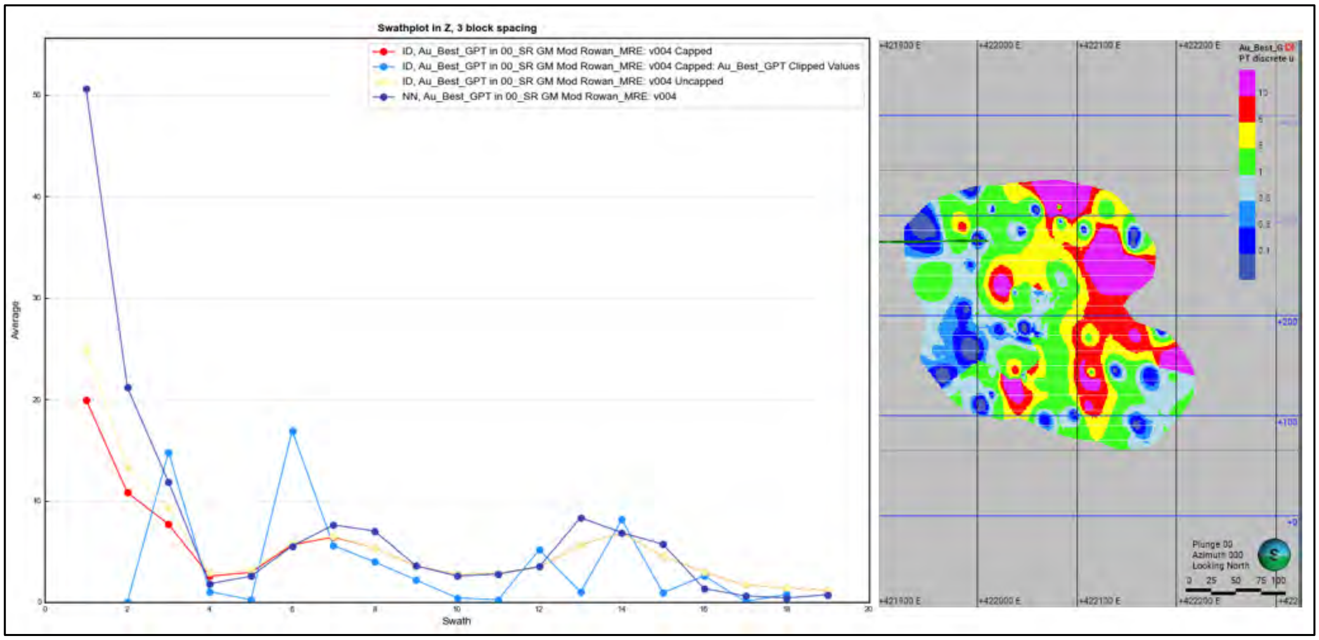
Source: SR 2025.

Figure 14-12: Swath Plot for Model Domain v004 – Y Direction



Source: SR 2025.

Figure 14-13: Swath Plot for Model Domain v004 – Z Direction

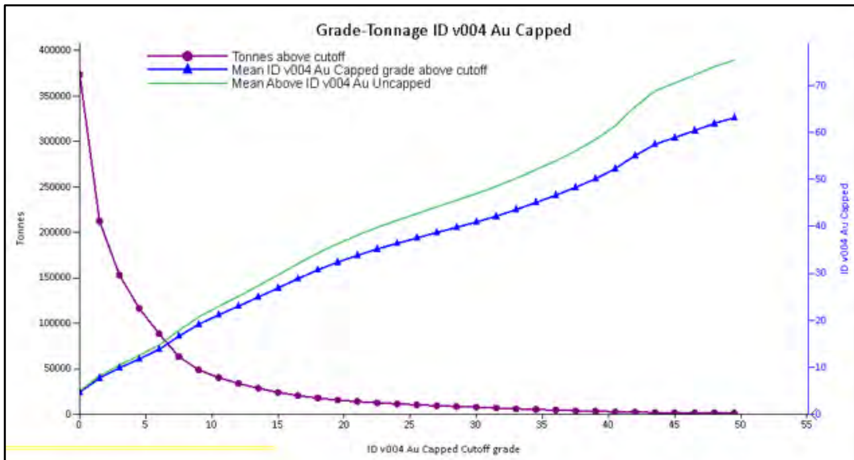


Source: SR 2025.

14.13.4 Changes To Block Cut-Off Grade

Figure 14-14 illustrates the effects of changes for both the capped and uncapped mineral resource estimates at various gold cut-off grades. In both cases, values show a relatively moderately sloped graph; results diverge significantly above a 13 g/t gold cut-off grade, which is expected.

Figure 14-14: Grade-Tonnage Curve for V001 Capped versus Uncapped Gold



ID v004 Au Capped	Tonnes above cutoff	Mean ID v004 Au Capped grade above cutoff	Mean ID v004 Au Uncapped grade above cutoff
0.0	373,639.50	4.67	4.93
1.5	212,330.70	7.70	8.15
3.0	153,097.42	9.84	10.45
4.5	116,471.25	11.75	12.51
6.0	88,582.27	13.80	14.71
7.5	63,273.15	16.63	17.86
9.0	48,756.60	19.15	20.70
10.5	40,250.25	21.14	22.97
12.0	33,874.87	23.01	25.12
13.5	28,541.02	24.93	27.37
15.0	24,133.27	26.89	29.68
16.5	20,514.60	28.87	32.06
18.0	17,668.12	30.74	34.34
19.5	15,545.92	32.38	36.34
21.0	13,890.15	33.83	38.12
22.5	12,475.35	35.21	39.81
24.0	11,346.07	36.40	41.29
25.5	10,330.20	37.54	42.72
27.0	9,363.60	38.71	44.18
28.5	8,502.97	39.81	45.58
30.0	7,679.47	40.95	47.01
31.5	6,854.62	42.17	48.52
33.0	5,995.35	43.60	50.23
34.5	5,198.17	45.11	52.08
36.0	4,496.85	46.65	53.99
37.5	3,843.45	48.33	56.09
39.0	3,259.57	50.14	58.50
40.5	2,691.22	52.33	61.46
42.0	2,153.25	55.09	65.53
43.5	1,808.32	57.48	68.84
45.0	1,628.77	58.94	70.57
46.5	1,462.72	60.43	72.28
48.0	1,312.20	61.95	74.06
49.5	1,198.80	63.20	75.48

Source: SR 2025.

14.13.5 Metal Loss

A comparison of the capped versus uncapped gold grades on the block model is provided in Table 14-9; this comparison demonstrates the true spatial impact of metal loss due to capping. The same search ellipses and number of composites were used in both cases. Only the composite gold grade was changed, from the uncapped to the capped gold grades. The results are controlled by the “SR Combined Estimator Au Capped” calculation variable “Au_Final Capped” at the 3.80 g/t gold cut-off grade so the tonnages are the same for the comparison. The metal loss due to capping is reasonable for mineral resource estimate classes as Indicated at 12% has a higher confidence than Inferred at 21%. The QP believes that the metal loss due to capping should ideally be at 10% or less and notes that the percentage difference for some model domains is greater than 10%. This higher percentage difference is primarily due to the small number of composites and volumes. This difference also identifies the requirement for additional drilling to provide more data for non-sampled intervals given the half detection limit used in the mineral resource estimates and the capping analysis, especially for model domain v001 and model domain v006 to support the use of a high-grade transition strategy instead of grade capping.

Table 14-9: Metal Loss on the 2025 Block Model due to Gold Capping

Model Domains	SR All Class Final	Mass [t]	Average Value		Material Content		Difference [%]
			Au Final Capped [g/t]	Au Final Uncap [g/t]	Au Final Capped [oz]	Au Final Uncap [oz]	
v001	Indicated	140,895	22.65	25.59	102,590	115,903	11%
	Inferred	81,797	11.94	16.35	31,390	43,004	27%
v003	Indicated	2,475	9.72	9.76	773	777	0%
	Inferred	23,636	7.32	7.35	5,562	5,585	0%
v004	Indicated	109,639	11.12	11.82	39,185	41,665	6%
	Inferred	58,329	9.65	10.18	18,090	19,086	5%
v005	Indicated	7,025	6.65	6.65	1,502	1,502	0%
	Inferred	4,678	5.50	5.50	827	827	0%
v006b	Indicated	103,304	7.39	7.39	24,541	24,541	0%
	Inferred	26,046	7.05	7.05	5,903	5,903	0%
v007	Indicated	2,561	4.48	5.97	369	492	25%
	Inferred	2,998	4.49	5.14	432	495	13%
v008	Indicated	2,141	4.47	4.50	308	310	1%
v011	Indicated	725	5.06	5.24	118	122	3%
v012	Indicated	17,777	5.11	6.49	2,922	3,707	21%
	Inferred	1,495	4.03	5.62	194	270	28%
v013	Indicated	28,840	6.33	6.88	5,871	6,381	8%
	Inferred	8,425	5.83	5.98	1,580	1,621	3%
v015	Indicated	130	10.05	15.04	42	63	33%
	Inferred	60,701	5.51	6.14	10,760	11,978	10%
v016	Indicated	14,458	8.68	11.52	4,033	5,356	25%
	Inferred	28,636	7.24	7.30	6,661	6,717	1%
v017	Indicated	9,776	4.97	9.97	1,563	3,134	50%
	Inferred	5,797	4.47	14.25	834	2,655	69%
v018	Indicated	12,800	8.81	13.11	3,625	5,393	33%
	Inferred	367	6.01	9.85	71	116	39%
v019	Indicated	706	4.57	40.82	104	926	89%
	Inferred	5,353	4.52	10.31	778	1,775	56%
v020	Indicated	9,594	5.79	11.49	1,787	3,545	50%
	Inferred	1,847	5.76	12.74	342	756	55%
v022	Indicated	8,405	22.26	31.31	6,014	8,460	29%
	Inferred	35,011	13.62	18.72	15,335	21,071	27%
v023	Indicated	4,793	4.93	4.93	759	760	0%
	Inferred	16,937	4.33	5.35	2,358	2,912	19%
v024	Indicated	2,665	7.50	13.60	642	1,165	45%
	Inferred	26,874	5.41	6.39	4,678	5,522	15%
v026	Inferred	32,255	11.92	17.77	12,358	18,424	33%
Total	**Indicated**	**478,707**	**12.78**	**14.57**	**196,747**	**224,201**	**12%**
	Inferred	**421,181**	**8.73**	**10.98**	**118,155**	**148,716**	**21%**

Notes:
- CIM (2014) definitions were followed for mineral resources.
- Mineral resources were estimated at a gold cut-off grade of 3.80 g/t using a long-term gold price of US$1,800 per ounce.
- There are no mineral reserves currently estimated at the Rowan deposit.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Mineral resources are reported within vein wireframes at the stated cut-off grade of 3.80 g/t Au.
- Density of 2.8 g/cm^3.
- Numbers may not add due to rounding.

t = tonne; g/t = grams per tonne; Au = gold; oz = ounce; g/cm^3 = grams per cubic centimetre.

15 MINERAL RESERVE ESTIMATES

There are no current mineral reserves estimated for the PEA.

16 MINING METHODS

16.1 Mine Design Overview

The underground mine design, production schedule, and capital and operating costs were developed for the Rowan deposit at a scoping level of engineering. The mineral resource estimate for the Project, dated 30 June 2025, forms the basis of the underground mine plan (Section 14, Mineral Resource Estimates).

The Project will include one year of Construction, five years of Operations, and one year of Closure. Due to the narrow nature of the mineral deposit, a maximum of 475 t/d of ore is planned to be produced; this maximum is comprised of 400 t/d produced from underground stopes and 75 t/d of ore extracted by sublevel development along the veins of the Rowan deposit. During Operations, a total of approximately 705 kilotonnes (kt) of ore will be mined at an average gold grade of 8.01 g/t and is estimated to contain approximately 182 thousand ounces (koz) of gold.

The Rowan deposit will be accessed from a single decline driven from the Portal on surface and is planned to be mined using longhole retreat open stoping (Section 16.3, Mining Method Details). A dedicated escapeway network will be progressively installed as the underground mine is advanced (Section 16.3.7.6, Escapeways and Refuge Stations). The majority of excavated open stopes will be backfilled with unconsolidated rock fill; consolidated (e.g., cemented) rock fill will be used as backfill in the three sill levels where additional stopes will be mined below the backfilled area.

Ore from the underground will be temporarily stockpiled on surface for crushing and sampling (Section 17, Recovery Methods) until the ore is hauled off site for processing (Section 18.7, Off-Site Ore Transportation). Waste rock will be preferentially used as backfill underground in excavated open stopes or will be temporarily stockpiled on surface (Section 18.4, Waste Rock and Water Management Facilities).

A summary of the underground mine production schedule is provided in Table 16-1, and an isometric view of the underground mine layout showing gold grades is shown on Figure 16-1. The underground mine was designed at a cut-off grade of 3.0 g/t Au with a marginal cut-off grade of 1.5 g/t Au. The underground mine will produce 1,960 ore tonnes per vertical metre and 61 stope tonnes per development metre.

This PEA is preliminary in nature and includes 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, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Table 16-1: Mine Plan Production Summary for the Rowan Project

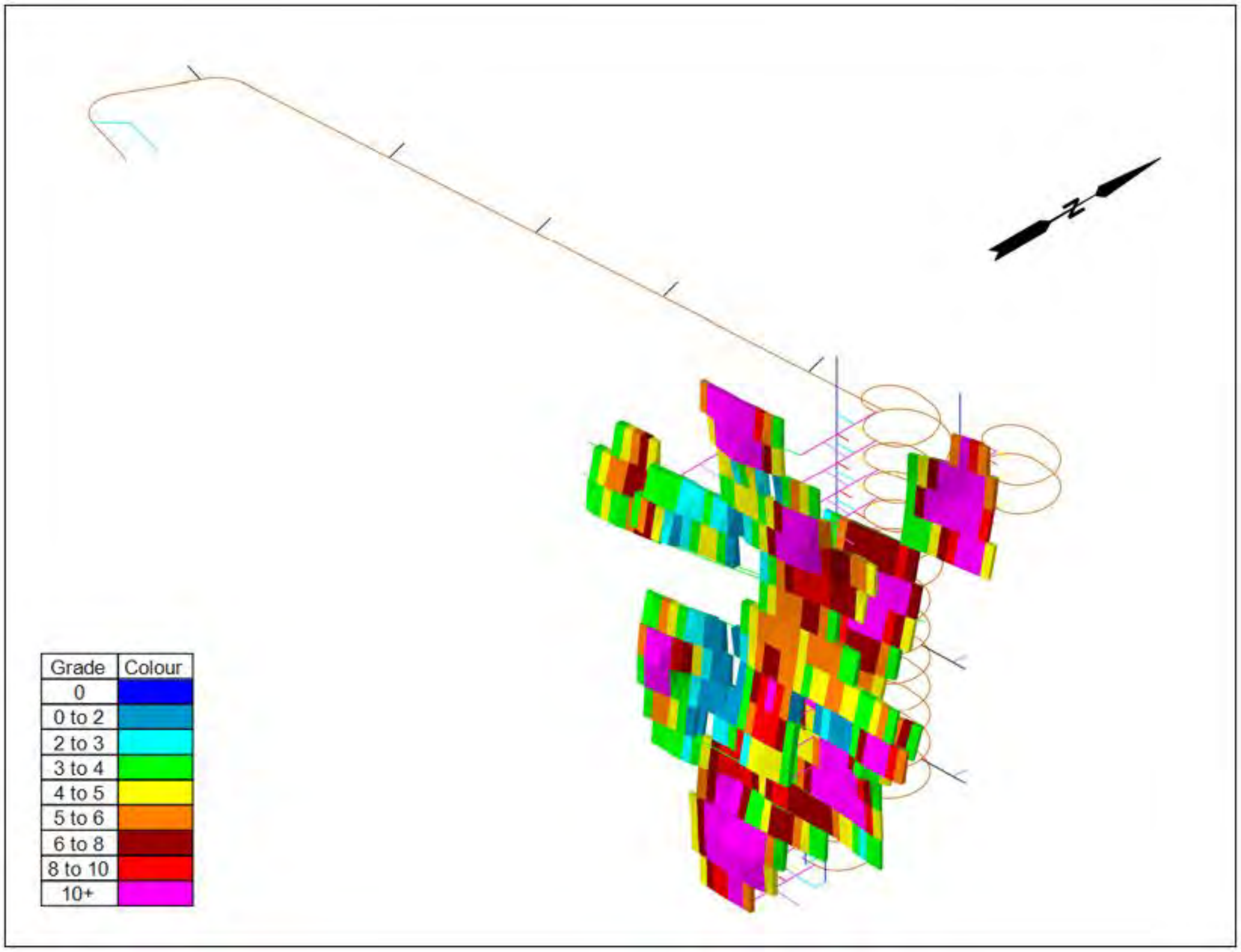
Mine Production Metric	Unit	Life of Mine Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5
Lateral Development								
Sublevel development	m	3,637	-	1,369	1,328	820	120	-
Capital development	m	6,487	1,736	1,364	1,527	1,858	2	-
Total lateral development	**m**	**10,124**	**1,736**	**2,733**	**2,855**	**2,678**	**122**	**-**
Total lateral development waste rock	**t**	**470,200**	**115,900**	**105,600**	**114,600**	**134,000**	**56**	**-**
Vertical Development								
Ventilation raises (raise bore)	m	197	-	197	-	-	-	-
Ventilation raises (drop raise)	m	210	-	-	88	122	-	-
Total vertical development	**m**	**407**	**-**	**197**	**88**	**122**	**-**	**-**
Total vertical development waste rock	**t**	**17,135**	**-**	**6,871**	**4,286**	**5,979**	**-**	**-**
Ore Production								
Development ore	t	91,574	-	34,400	32,400	21,100	3,800	-
Ore grade	g/t	9.43	-	8.91	8.64	11.24	10.88	-
Contained gold	oz Au	27,770	-	9,850	8,990	7,610	1,320	-
Stope ore	t	613,600	-	75,500	93,000	148,900	162,600	133,600
Ore grade	g/t	7.75	-	11.1	5.99	6.84	6.42	9.94
Contained gold	oz Au	153,830	-	26,940	17,910	32,730	33,570	42,690
Total Ore Tonnes	**t**	**705,185**	**-**	**109,900**	**125,300**	**167,000**	**166,400**	**133,600**
Total ore grade	**g/t**	**8.01**	**-**	**10.41**	**6.67**	**7.38**	**6.52**	**9.94**
Total contained gold mined	**oz Au**	**181,600**	**-**	**36,800**	**26,900**	**40,300**	**34,900**	**42,700**
Backfill Schedule								
Consolidated rock fill	t	85,200	-	15,200	24,400	27,500	18,000	0
Unconsolidated rock fill	t	278,500	-	29,500	30,700	60,700	78,400	79,200

Notes:

Numbers may not add due to rounding.

m = metre; t = tonne; g/t = grams per tonne; oz = ounce; Au = gold.

Figure 16-1: Isometric View of Underground Mine Layout and Gold Grades (grams per tonne) for the Rowan Project



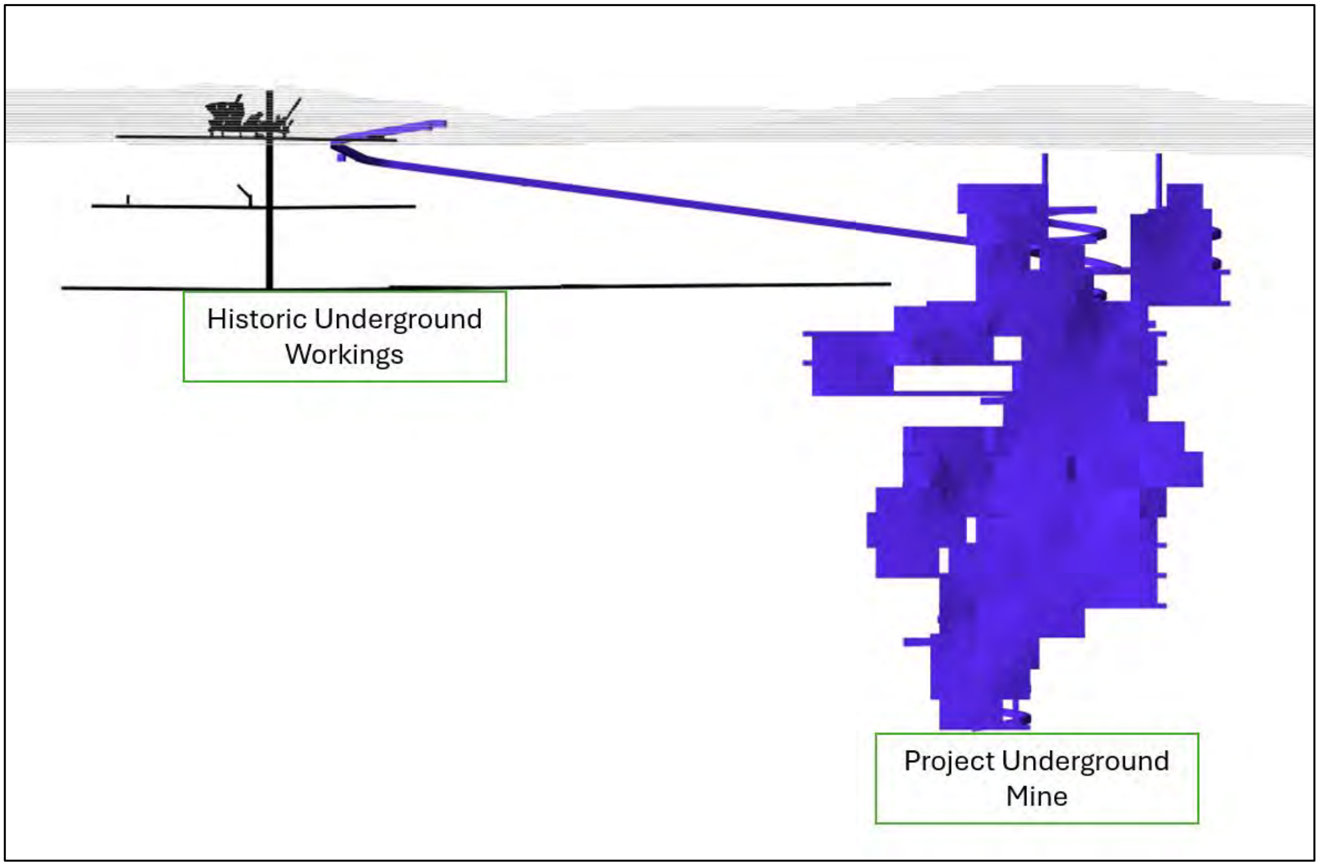
16.2 Key Assumptions and Inputs

Key assumptions and inputs in the underground mine design included:

- Topography was based on light detection and ranging (LiDAR) surveys provided by WRLG.
- No faulting has been identified in the area of the Rowan deposit that would materially affect the mine design.
- Resource model was provided as a sub-blocked DataMine file from WRLG and includes gold grades, ore and waste rock density, and resource classification (i.e., indicated, inferred).
- Capital development (i.e., preproduction) underground excavations were designed with a height of 5 m and a width of 5 m.
- Sublevel development (i.e., production) underground excavations were designed with a height of 3.5 m and a width of 3 m.

- Stope dimensions:
  - Stopes were designed to a height of 20 m; this height is equivalent to sublevel spacing height.
  - Stopes were designed to a minimum width of 2 m, with 15% external dilution added.
  - Individual minable shape optimizer (MSO) shapes were set to a length of 6 m.
- Existing models of historical underground workings are accurate and do not overlap with the underground mine for the Project (Figure 16-2).

Figure 16-2: Profile View of Underground Mine and Historical Underground Workings for the Rowan Project – Looking North



16.3 Mining Method Details

The Rowan deposit will be developed as an underground mine using conventional longhole retreat open stoping. Based on review of the block model and geology wireframes, this underground mining method demonstrates the best potentially viable option for mining of the Rowan deposit based on the narrowness of the veins, highly competent rock, and steep dips of the mineral deposit.

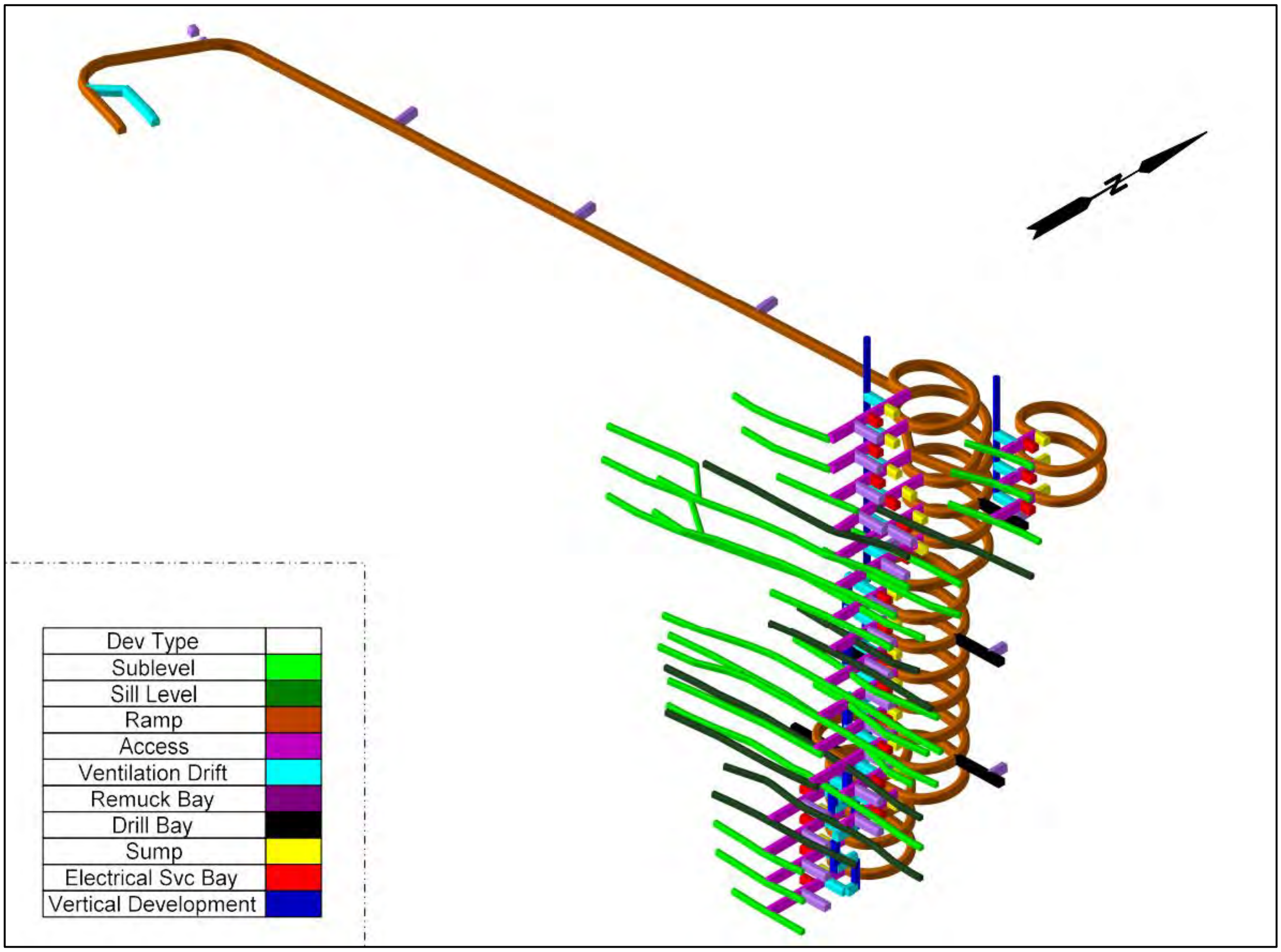
Surface mining as an option is not considered for the Rowan deposit due to the narrow nature of the veins and the high strip ratios that would be required for open pit extraction, even at shallow depths. As the geometry of the mineral deposit makes it amenable to open stoping, use of a more expensive underground excavation methods (e.g., cut and fill) were not assessed.

The underground mine design includes three sill levels that will be used as single access points and multiple sublevels in between the sill levels; underground stopes will be extracted by longitudinal retreat from one of these three sill levels. The underground mine design from the decline from the Portal includes a spiral ramp that provides single point access to the multiple mining levels (i.e., sublevels and the associated capital development on that level), which are vertically spaced at 20 m. Each mining level will have a capital access drift connecting the spiral ramp to the mineral deposit, along which remuck bays, ventilation drifts, sumps, and electrical service bays will be excavated. The capital access drift will end at the mineral deposit contact, where sublevel drifts will be developed at approximately 90-degree (°) angles from the access drift and follow the vein along strike.

In longhole retreat open stoping, the horizontal sublevel drifts along mineralized veins act as top access for drilling and blasting stopes, and bottom access for mucking out blasted material. Extraction of stopes occurs in a bottom-up method and will start from one of the three sublevels identified as sill levels in the mine plan. Excavated stopes will be backfilled with either unconsolidated waste rock or consolidated waste rock, this backfill material will then be used as a working platform to extract the stope above in the sequence. Consolidated rock fill will be used in the three sill levels as the mineralized material below will not yet have been extracted, and this rock fill type allows the stope below to be mined without leaving a remnant pillar, increasing the recovery of the mineral deposit.

An isometric view of the Rowan deposit showing underground capital and production development types is shown on Figure 16-3.

Figure 16-3: Isometric View of Underground Development Types for the Rowan Underground Mine



Notes:
Dev = Development; Svc = Service.

Alternative Underground Mining Methods

Alternative underground mining methods were considered for the Rowan deposit.

Mechanical cut-and-fill mining was not considered viable, as this method would incur unnecessary costs from lateral development rounds for ore extraction as opposed to bulk stoping blasts. Additionally, the widths needed for mechanized equipment would add dilution to cut-and-fill stopes.

Avoca-style mining was reviewed but not selected for the Rowan deposit. While the additional access points and continuous back filling would allow faster ore extraction, the small size of the overall mineral deposit did not justify the additional capital development. Use of this underground mining method should be reassessed should future drilling expand the ore zones of the Rowan deposit.

16.3.1 Cut-off Grade Determination

A gold cut-off grade of 3.0 g/t was used for the underground stoping design cut-off based on the expected operating costs from comparable projects. Ore down to a marginal gold cut-off grade of 1.5 g/t was included in the underground stoping design if this material required extraction to access upcoming stopes. Marginal grade ore was also included in the underground mine design when it located within a larger area of higher-grade material and required no additional development to extract.

The gold cut-off grade was estimated with a total operating cost of C$300 per mined tonne, gold price of US$2,500/oz, USD/CAD exchange rate of 1.35, and then rounded up to 3.00 g/t. This total operating cost was benchmarked against comparable underground mining operations in Canada.

16.3.2 Underground Design Parameters

16.3.2.1 Capital Development

Lateral capital development includes areas that will not be backfilled during Operations, including the main haulage ramp (i.e., decline), level accesses, remuck bays, sumps, electrical service bays, and dedicated ventilation drifts (Figure 16-3). All lateral capital development was designed with a height of 5 m and width of 5 m to accommodate the largest pieces of underground mining equipment (Section 16.4, Underground Equipment). An additional 10% tonnage was included in lateral capital development for overbreak.

Vertical capital development for underground ventilation will be excavated either by 3.5 m diameter raise bore with a raise-bore machine, or by conventional drill-and-blast methods with a top hammer longhole drill. Connection of the uppermost sill level to surface will be developed by raise bore, with additional levels of capital development connected to the ventilation network by drop raises. An additional 10% tonnage was included in drop raise development for overbreak.

A plan view of the typical underground level layout is shown on Figure 16-4. A summary of lateral and vertical capital development metres by capital development type is provided in Table 16-2.

Figure 16-4: Plan View of Typical Underground Level Layout for the Rowan Project

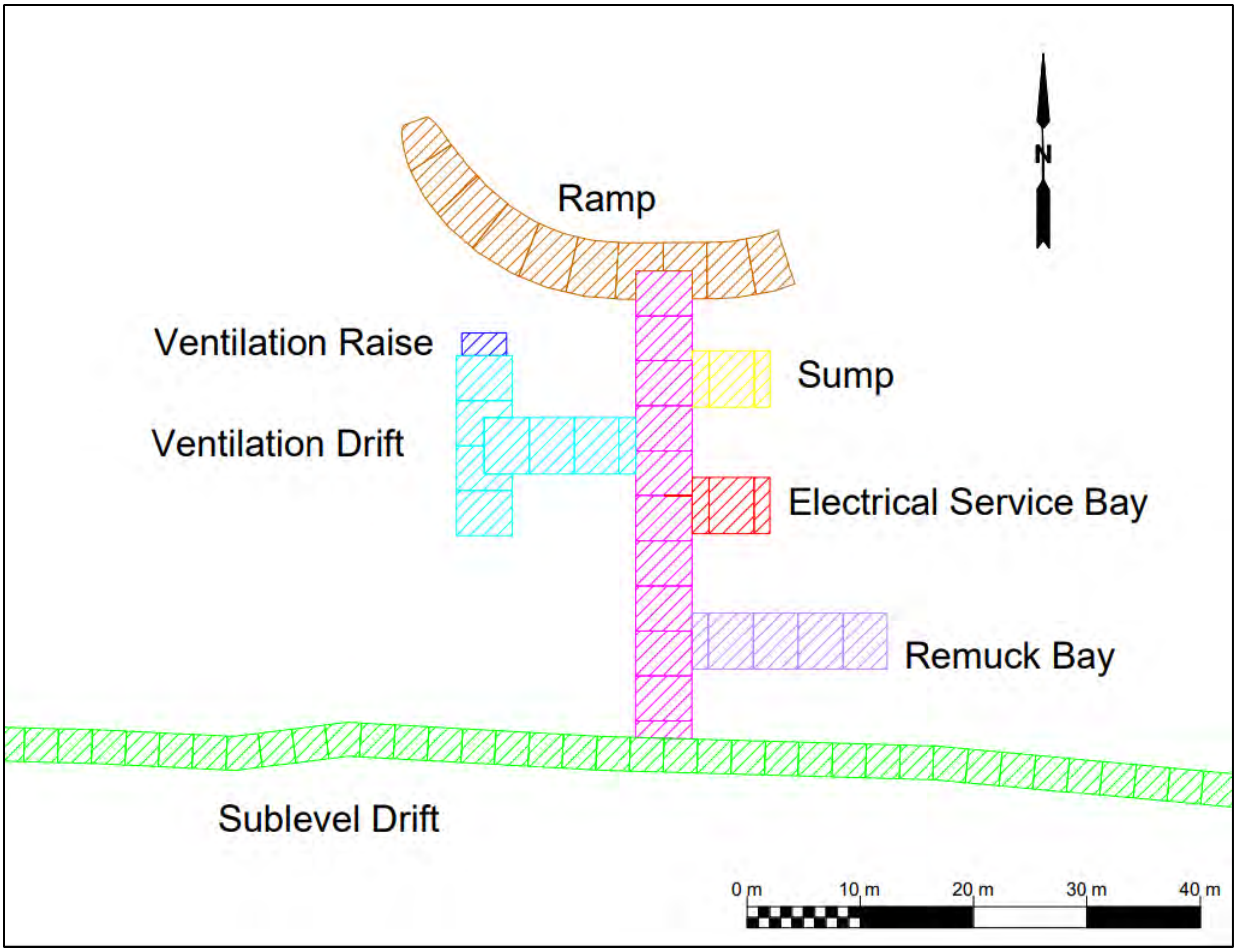


Table 16-2: Capital Development Metres Summary by Development Type for the Rowan Project

Capital Development Type	Total [m]
Lateral Development	
Decline	3,393
Level accesses	1,086
Remuck bays	372
Sumps	194
Electrical service bays	144
Ventilation drifts	512
Drill bay	196
Vertical Development	
Ventilation raises (raise bore)	210
Ventilation raises (drop raise)	197

Notes:
m = metre.

16.3.2.2 Operating Development

Operating development includes the lateral sublevel drifts that will be developed along the veins and will allow access to the top and bottom of stope blocks for drilling, blasting, loading, and hauling of material. Sublevels were designed with a height of 3.5 m and width of 3.0 m, and an additional 10% tonnage was included for overbreak.

16.3.2.3 Stope and Pillar Design

Stope shapes were generated using Deswik Stope Optimizer with the minimum 2 m stope thickness and a gold cut-off grade of 1.5 g/t to allow for inclusion of marginal grade stopes. Stopes were reviewed and removed manually to confirm profitability. Areas of material identified as above the cut-off grade, but far enough away from the main capital development were removed from the mine design. All stope shapes were cut by sublevel development and reviewed against the provided block model.

Mining recovery for stopes was assumed to be 95% to account for ore loss during mining. An external dilution factor (i.e., 15% additional tonnes) was included above the minimum 2 m designed stope thickness. Dilution was given a grade of 0 g/t in the resource model, as mineralized material does not occur in significant quantities outside of the vein boundaries.

For horizontal pillars between sublevels on the same elevation, a minimum pillar width of 10 m will be maintained between stopes in cases where multiple veins exist on a single sublevel.

16.3.3 Mine Scheduling

Mine scheduling was completed using Deswik Scheduler based on the completed underground mine design. The key scheduling parameters included:

- maximum of 200 t/d from a single stope;
- maximum single heading development of 6 metres per day (m/day);
- maximum total development of 9 m/d;
- two-week delay between level development to account for adequate backfilling time; and
- two-week delay between sublevel completion and stoping to account for initial production drilling.

16.3.4 Drill and Blast

Conventional drill-and-blast mining will be employed for lateral and vertical capital development using two-boom jumbo drills; lateral development will also be supported by a dedicated development emulsion loader.

Conventional drill-and-blast mining will be employed for production ore stoping using top hammer longhole drills.

Ore stope drilling will be initiated using small diameter blastholes (i.e., approximately 2 inches [in.] to 2.5 in.) that will be loaded with emulsion explosives by a dedicated stope emulsion loader. The initial slot blastholes will be reamed to larger diameters (i.e., 6 in. to 10 in.) by a top hammer drill to provide the appropriate void space for proper fracturing of the initial stope cuts.

Emulsion explosives for blasting will be stored on surface in a dedicated Emulsion Tank (Figure 18-1), and explosives will be resupplied from Red Lake, Ontario as required. Once underground development begins, a single blasting cap and detonator magazine will be installed underground off the decline. Blast consumables will be collected in emulsion loaders from surface at the beginning of each shift for use deeper in the underground mine at working stopes and faces.

16.3.5 Loading and Haulage

Loading and hauling of ore and waste rock will be completed by conventional underground diesel-powered, load-haul-dump (LHD) machines and haul trucks. Smaller 7 t LHDs will operate in the narrow sublevel development and stope production areas, and a larger 10 t LHD will operate in the capital development drifts.

Ore will be hauled out of the underground via the Portal to the Run-of-Mine (ROM) Ore Stockpile on surface (Figure 18-1). Waste rock will either be hauled to the Waste Rock Stockpile (WRS) on surface or preferably hauled directly to an excavated open stope within the underground for immediate use as backfill.

16.3.6 Backfill

The primary backfill method for the Project is unconsolidated rock fill using waste rock from active mining areas in the underground. Three sill levels within the mine design (Figure 16-3) will require consolidated (e.g., cemented) rock fill to allow for maximum recovery of the ore stopes located underneath these sill levels. The use of consolidated rock fill in these sill levels will remove the requirement for a solid rock pillar to be left behind during the mining of the sublevel below. Consolidated rock fill will be limited to the small quantity required for the three sill levels and will only be placed in short intervals during Operations. Consolidated rock fill will be mixed on surface with a temporary cement plant and transported underground by haul trucks equipped with ejector boxes.

16.3.7 Mine Services

The underground operation will be supported by standard mine services, including electrical distribution, service water and dewatering infrastructure, ventilation and heating, and communications. The required water distribution pipelines and communication cables will be installed along the spiral ramp and access drifts.

16.3.7.1 Electrical Distribution

Electrical power for the underground mine will be sourced from the Generator Farm on surface (Figure 18-1) and distributed underground at 4,160 volts (V) with cables entering the Portal; this voltage was considered appropriate for the size of the underground mine. Electrical substations will be installed underground at regular intervals throughout the mine to step down the voltage to 600 V for use by the electrical mine equipment (e.g., development drills, production drills). Each electrical substation will provide power for three underground levels, including the installation level, as well as one level directly above and below the installation level.

These substations will be used to power auxiliary ventilation fans, drilling and loading equipment, air compressors, pumps, and refuge stations.

16.3.7.2 Service Water

Service water for drilling will be sourced from the Treated Water Pond on surface (Figure 18-1) and pumped underground for use by drilling equipment at the working faces. Service water for surge capacity will be stored locally in repurposed remuck bays and will be recycled underground as much as practicable. Pressure-reducing valves will be installed along pipelines as necessary to keep pressures within operating ranges of the underground drills.

16.3.7.3 Mine Dewatering

Each underground mine level will be equipped with a sump constructed to collect groundwater inflows and used service water generated during drilling. Collected water will be pumped from these sumps using 40 kilowatt (kW) heavy-duty pumps located on each level that will operate in series to pump water out of the underground mine to the Contact Water Pond on surface (Figure 18-1). An underground dewatering rate of 500 cubic metres per day (m^3/day) was assumed during Operations (Section 18.4.9, Water Balance).

16.3.7.4 Ventilation and Heating

Ventilation underground will be provided by parallel fans operating at the Portal entrance that will push air down the decline. Air will exit the underground mine out of the three ventilation raises. Airlock doors, allowing equipment and personnel access, will be installed at the Portal entrance to confirm air moves down the ramp and does not short-circuit to surface.

The upper portion of these ventilation raises will be raise-bored from surface. Once the bored raises are connected, further vertical expansion of the ventilation network will be completed by conventional drill-and-blast drop raises (Section 16.3.2.1, Capital Development). The underground ventilation system was designed with a capacity of 180 cubic metres per second (m^3/s) at 930 pascals (Pa). This capacity, which included utilization factors, meets the requirement of 0.06 m^3/s of fresh ventilating air per kilowatt of operating underground diesel-powered equipment (Ontario Regulation [O. Reg.] 854: Mines and Mining Plants).

During the winter months, propane heating systems will maintain adequate working temperatures in the underground mine. Heaters, ducting, propane tanks, and control instrumentation will be installed alongside the primary ventilation fans at the Portal entrance.

16.3.7.5 Compressed Air

Mobile compressed air units for drilling equipment will be installed underground near working faces to provide the required air for the underground drill equipment.

16.3.7.6 Escapeways and Refuge Stations

A dedicated network of escapeways will be drilled using a 1.2 m diameter raise-bore machine, and durable polyethylene ladder tubing (e.g., SafeScape® or equivalent) will be installed with connection points on every sill level and sublevel. Mobile refuge stations will be installed periodically throughout the underground mine as development continues at depth to provide muster stations in close proximity to working areas in the event of an emergency. These refuge stations will be fully self contained, and include carbon dioxide scrubbing, provisions, and battery power, and there will be sufficient seating for all personnel in the vicinity.

Underground communication cables specially designed to transmit and receive radio signals underground (i.e., leaky feeders) will be installed and expanded throughout the mine to provide dependable radio communications at all active working areas. Underground vehicles will be equipped with permanently installed radios, and underground personnel will be equipped with handheld radios. A stench system will be installed in the ventilation network to be used in the case of an emergency, to alert all underground personnel that they are required to relocate to the nearest refuge until management has approved a return to work.

16.4 Underground Mobile Equipment

A summary of the estimated underground development and production mobile equipment as well as auxiliary mobile equipment for the Project is provided in Table 16-3. All mobile equipment will be equipped with a minimum of Tier 4 diesel engines, mobile fire suppression, and required roll-over and falling-object protection.

Table 16-3: Underground Mobile Equipment Summary for the Rowan Project

Mobile Equipment	Maximum Units
Development and Production Equipment	
Two-boom jumbo drill	2
Top hammer longhole drill	1
LHD (7 t)	2
LHD (10 t)	1
Development emulsion loader	1
Stope emulsion loader	1
Mechanized bolter	2
Haul truck (30 t), 2 with ejector boxes	4
Raise-bore machine	1 – Contracted
Auxiliary Equipment	
Scissor lift	1
Telehandler	1
Boom truck	1
Grader	1
Light vehicles	6
Water/fuel cassette truck	1
Maintenance truck	1
Total Units	**27**

Notes:
LHD = load-haul-dump.

16.4.1 Underground Development and Production Equipment

Capital development drilling will be completed using two-boom jumbo drills supported by mechanized bolters for installation of ground support. Production drilling will be completed using a top hammer longhole drill with a dedicated carrier and operator cab.

One 10 t LHD will be used in the capital development drifts, and two 7 t LHDs will operate in the narrow sublevel development and stope production areas. The 7 t LHDs will include remote-operation capability to remove material from the open stopes to limit equipment operator exposure to unsupported ground.

Four 30 t haul trucks will be used to transport ore and waste rock to surface, and two of these trucks will include ejector boxes to support underground backfilling of consolidated rock fill in the three sill levels.

Two emulsion loaders (i.e., one dedicated for development loading, and one dedicated for stope loading) will transport bulk explosives, detonators, and boosters from the surface Emulsion Tank and underground blasting cap and detonator magazine to working areas.

16.4.2 Underground Auxiliary Equipment

A suite of auxiliary mobile equipment will be included in the underground equipment fleet to support mining operations. This auxiliary equipment will be used by the underground crews for service installation, mine maintenance, and personnel transportation, and will include a scissor lift, telehandler, boom truck, grader, and several light vehicles. Additionally, a water/fuel cassette truck, and maintenance truck will support underground operations.

16.5 Mine Personnel

The Project will operate 24 hours per day and 365 days per year during the 5-year period of Operations. Mine personnel will operate on a two-week rotation (i.e., two weeks on, two weeks off) and will be flown into Red Lake, transported to the Project site by bus, and housed in the Personnel Camp for the duration of their shift.

A summary of the personnel requirements for the Project is provided in Table 16-4.

Table 16-4: Personnel Requirements Summary for the Rowan Project

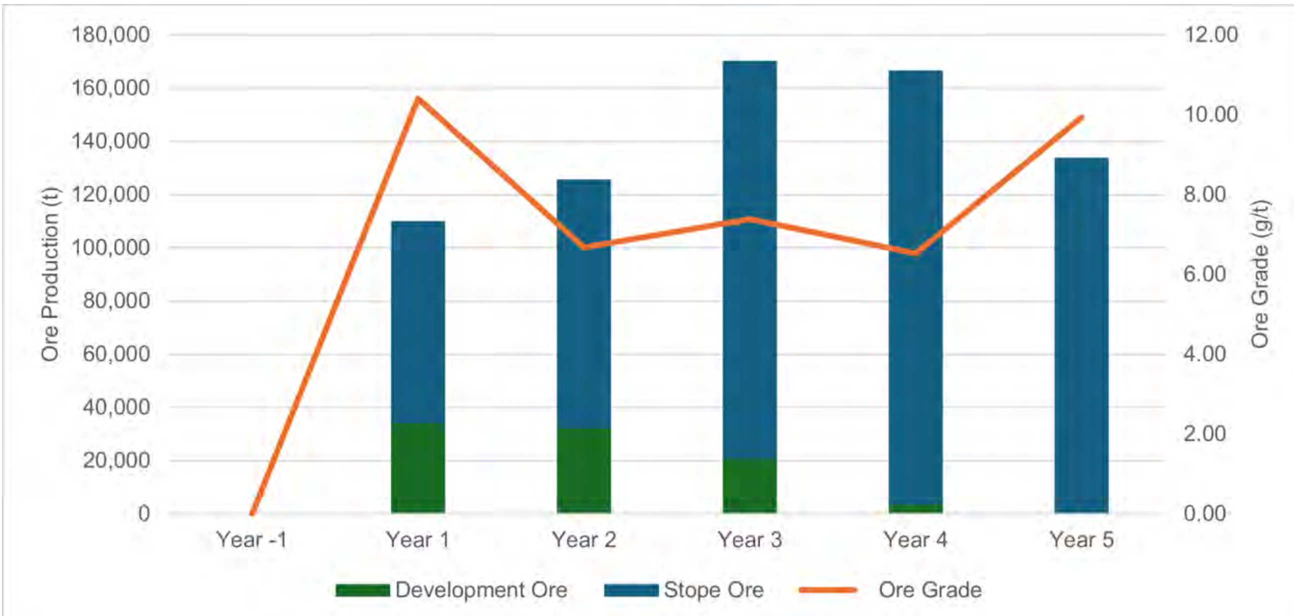
Personnel Position	Quantity
Underground	
Supervisor	4
Two-boom jumbo drill operator	4
Top hammer longhole drill operator	4
Bolter operator	4
LHD operator	8
Truck driver	16
Services and maintenance staff	16
Raise-bore machine operator	3 – Contracted
Surface	
Mechanic	12
Electrician	6
Surface equipment operator	8
Road maintenance operator	4 – Contracted
Bus driver	2 – Contracted
Haul truck driver	16 – Contracted
Technical Services and Management	
Mine manager	1
Senior engineer	1
Engineer	5
Senior geologist	1
Geologist	4
Surveyor	4
Safety	2
Environmental technician	4
Total Positions	**129**

Notes:
LHD = load-haul-dump.

16.6 Production Schedule

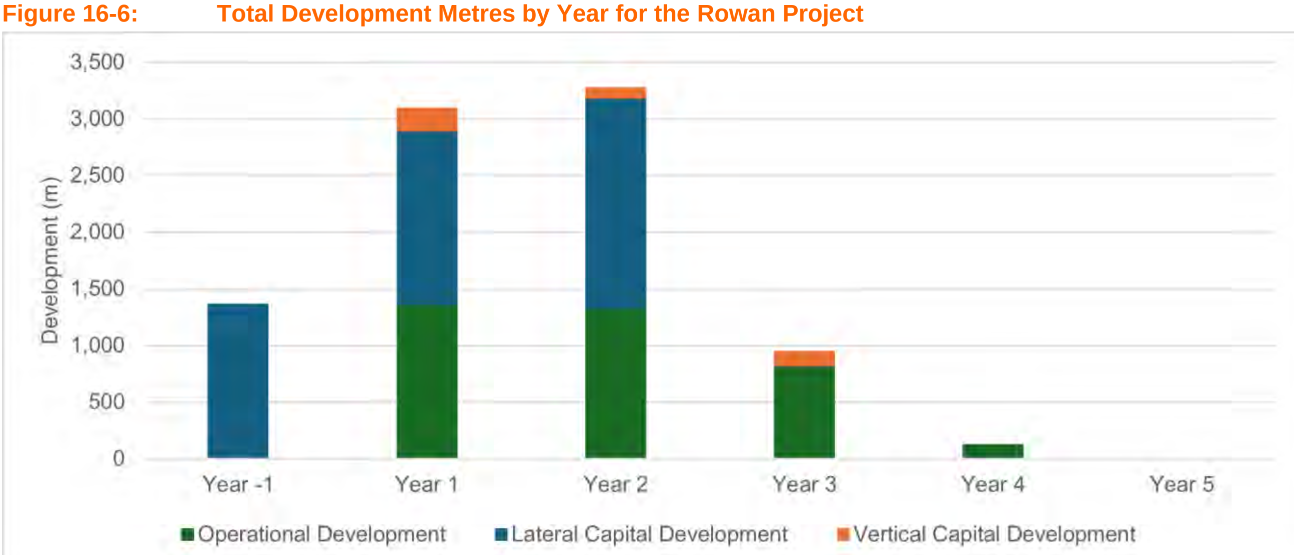
Approximately six months of capital development is required to access the first underground stoping faces. Ore production from the underground will average 385 t/d and will reach a peak production of approximately 465 t/d to 475 t/d in Year 3 when underground equipment transitions from capital development to production. The total Project ore production and development metres by year is shown on Figure 16-5 and Figure 16-6, respectively.

Figure 16-5: Total Ore Production by Year for the Rowan Project



Notes:
t = tonnes; g/t = grams per tonne.

Figure 16-6: Total Development Metres by Year for the Rowan Project



Notes:
m = metres.

16.7 Geotechnical Considerations

16.7.1 Geotechnical Model

The geotechnical model for the Rowan deposit was based on geotechnical core logging and the review of the Rowan structural model. The 2023 geotechnical program consisted of 8,227 m of logged drill core recorded over 2,748 intervals. The geotechnical results have undergone data entry validation. Field verification was not completed as only one site visit was performed at the beginning of the geotechnical program. A summary of the 2023 geotechnical logging intervals is provided in Table 16-5.

The Rowan deposit is located in the Red Lake Greenstone Belt, with Archean assemblages dating between 2.99 Ga and 2.72 Ga. Five known deformations have occurred regionally that create a pervasive second (S2) foliation across the deposit. Localized minor folds are known to be present, but the foliations and discontinuity orientations have yet to be geotechnically classified. Faulting and sheared gouge sections are discrete over short intervals where present within the Rowan deposit.

Table 16-5: 2023 Geotechnical Logging Intervals for the Rowan Deposit

Borehole	From [m]	To [m]	Length [m]
RLG-23-152	48	195	150
RLG-23-153	15	360	348
RLG-23-154	13.5	447	437
RLG-23-155B	18	378	363
RLG-23-156B	15	405	393
RLG-23-157	11.54	468	459
RLG-23-158	13.5	372	362
RLG-23-159	15	348	336
RLG-23-160	15	393	381
RLG-23-161	16	363	350
RLG-23-162	11	495	487
RLG-23-163B	12	498	489
RLG-23-164	9	297	291
RLG-23-165	5	408	406
RLG-23-166B	6	375	372
RLG-23-167	8.2	309	304
RLG-23-168	7.1	309	305
RLG-23-169	3	594	594
RLG-23-170	3.65	552	551
RLG-23-173	9	357	349
RLG-23-176C	6.78	504	500
Total			**8,227**

Notes:
m = metres.

Geotechnical logging was performed using the Rock Mass Rating (RMR89; Bieniawski 1989) system that combines the intact strength of the rock, and the degree and spacing of fractures. This system provides a semi-quantification of the shear strength characteristics of the fractures within the deposit. Each drill core run was assessed for these RMR89 parameters, and the rock was assessed on a scale of 0 (i.e., Very Poor Rock) to 100 (i.e., Very Good Rock). A summary of all RMRs for Rowan deposit lithologies are provided in Table 16-6 listed by percent of total core logged. Based on the average RMR, most deposit lithologies range from 61 to 80 (i.e., Good Rock), with one lithology (i.e., felsic intrusive) rating 55 (i.e., Fair Rock). The three most populated lithologies are basalt, quartz porphyry dyke, and intermediate volcanic. These lithologies account for the majority of meters logged (i.e., 82.5%) and are comparable using the RMR89 (Bieniawski 1989) system. Cumulative percentage RMR plots of the major lithologies at the Rowan deposit are shown on Figure 16-7. No discontinuity sets were considered in the geotechnical analysis as orientated core logging has not been performed at this stage of the Project. It is assumed that the primary discontinuity is parallel to the orebody.

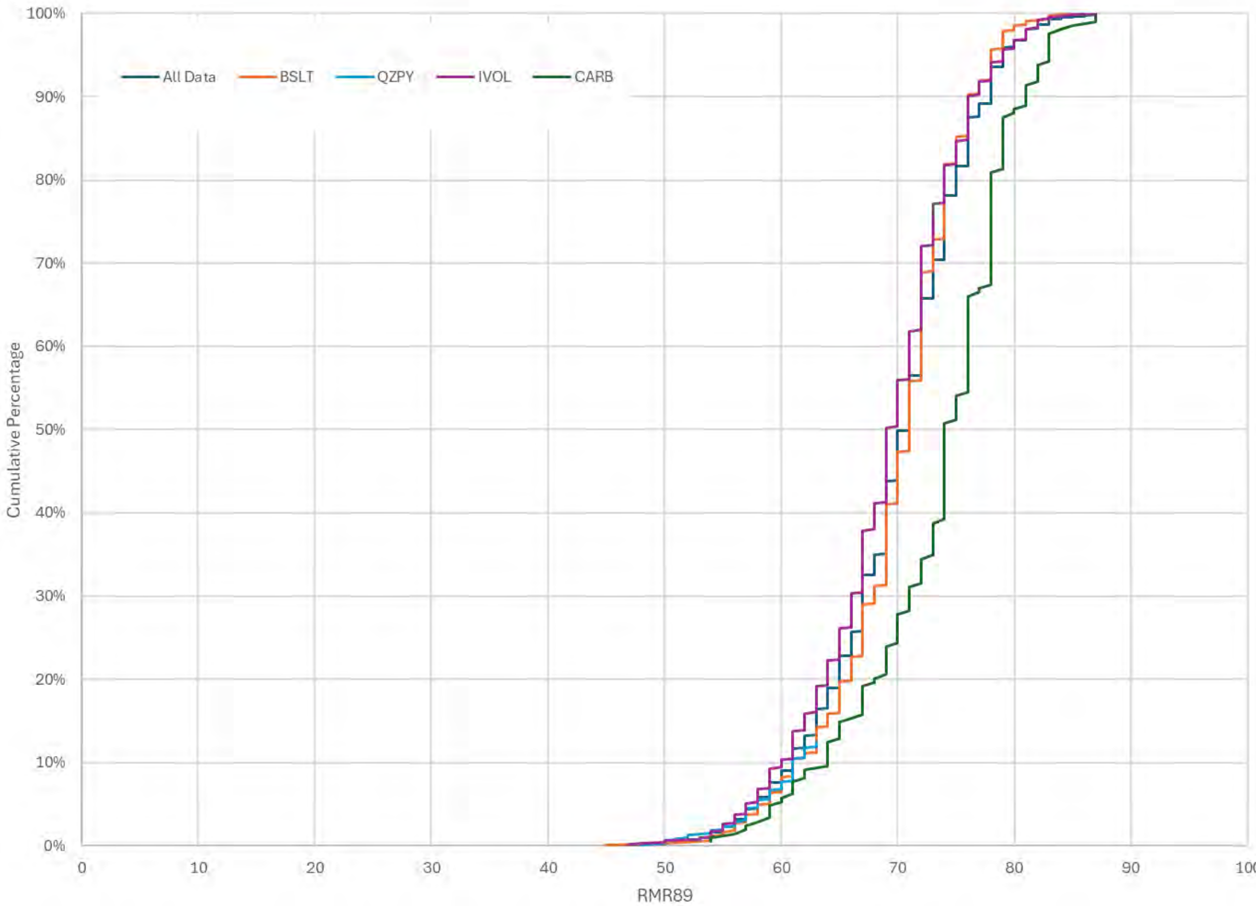
Rock strength was assessed with the International Society for Rock Mechanics and Rock Engineering (ISRM) Index Strength Test (ISRM 2014), and the majority of the rock has an R5 rating (i.e., Strong) or R6 rating (i.e., Very Strong). It is recommended that the reported strengths are verified with Unconfined Compressive Strength (UCS) Tests and subsequent geotechnical logging is performed with the more quantitative ISRM Point Load Test or equivalent.

Table 16-6: Rock Mass Ratings by Logged Lithology for the Rowan Deposit

Lithology Description	Lithology Code	Total Logged Metres	Percent of Total Core Logged	Rock Mass Rating (RMR89)[(a)]		
				Minimum	Average	Maximum
Basalt	BSLT	2,497	30.4%	45	70	87
Quartz Porphyry Dyke	QZPY	2,385	29.0%	48	70	86
Intermediate Volcanic	IVOL	1,934	23.5%	47	69	87
Metamorphosed Carbonate Units (marble)	CARB	627	7.6%	54	73	87
Post Tectonic Mafic Dyke	MINT	489	5.9%	52	67	81
Moderate Pervasive Alteration Zone	BSLA	117	1.4%	58	69	86
Undivided clastic sedimentary rocks	MTSD	84	1.0%	52	70	78
Strongly altered foliated zone	SAFZ	54	0.7%	61	69	80
No core	NCOR	21	0.3%	53	67	76
Iron Formation	IRFM	6	0.1%	70	75	79
Chert	CHRT	3	0.0%	71	71	71
Undivided chemical sedimentary rocks	CHSD	3	0.0%	72	72	72
Conglomerate	CONG	3	0.0%	73	73	73
Felsic Intrusive	FINT	3	0.0%	55	55	55
Total		**8,227**	**100.0%**			

Notes:
a) Bieniawski 1989.

Figure 16-7: Cumulative Percentage Rock Mass Rating Plot for Major Lithologies for the Rowan Project



Notes:
BSLT = basalt; QZPY = quartz porphyry dyke; IVOL = intermediate volcanic; CARB = metamorphosed carbonate units (marble).

16.7.2 Geotechnical Design

16.7.2.1 Ground Support

Ground support patterns for costing purposes were developed using the Norwegian Geotechnical Institute (NGI) Q-System Minimum Support Guidelines (NGI 2022) and reflect the anticipated rock mass conditions based on the geotechnical core logging (Table 16-2); the RMR89 values were transformed to Q-System values based on standard industry conversions. The Excavation Support Ratio (ESR) value used was 1.6, which applies to permanent mining openings (NGI 2022) and development profile widths were assumed to be 5 m wide.

Ground support estimates for drift development consisted of standard pattern bolting, and it is recommended that welded wire-mesh screen is used in the underground back and walls. Intersection support in the underground will include secondary deep support consisting of cables or super Swellex bolts. Allowances were made for shotcrete where discrete poor ground is encountered. Detailed ground support design that considers structure, rock mass, and stress conditions will be completed once the mine plan is finalized in future studies.

16.7.2.2 Stope Stability

The preliminary mine plan consists of 30 stopes based off a generation of 532 6-m long panel MSO shapes. The average planned stope width is 3.8 m, with a maximum stope width of 6.3 m. The maximum stope strike lengths were designed to be 100 m. Stope heights will be maintained at 20 m measured from floor to sill. An empirical analysis (Potvin 2014) was completed that considered the shape factor of the underground stope and geotechnical stability, and that was defined as the Modified Stability Number (N') using the following equation:

$$N' = Q' \times A \times B \times C$$

Where:

- N' = Modified Stability Number;
- Q' = modified Q Tunnelling Quality Index (after Barton et al. 1974);
- A = rock stress factor;
- B = joint orientation factor; and
- C = gravity adjustment factor.

For the Rowan deposit, design criteria were set where acceptable underground stope performance would include stopes that are empirically stable without support and have an estimated linear equivalent overbreak slough (ELOS) of 0.5 m or less as defined by the Clark method (1998). The ELOS and dilution estimates differ as the ELOS is independent of stope width and can materially impact dilution on narrow underground stopes.

From the mine plan provided, all planned underground stopes meet the design criteria for stable stopes given the narrow nature of the orebody and competent rock mass.

16.8 Hydrogeological Considerations

Hydrogeological modelling was not completed for the Technical Report, and groundwater conditions were conservatively estimated based on known geological characteristics of the rock mass. No major faults or significant jointing patterns were identified in the mining areas of the Rowan deposit that would be likely to conduct major water flows. An underground dewatering rate of 500 m^3/day was assumed during Operations (Section 18.4.9, Water Balance).

17 RECOVERY METHODS

There are several existing ore processing facilities located within the Red Lake region of northwestern Ontario that could, in principle, be utilized as the off-site toll milling facility for ore from the Rowan deposit. These existing facilities currently operate conventional gravity-leach-CIP flowsheets. For the purposes of the Technical Report, one facility was selected as the basis for capital and operating cost estimates.

The selected off-site toll milling facility has the ability to process ore from the Rowan deposit without significant modifications and capital expenditure; however, this approach would result in the reduction of overall gold recovery compared to the recovery results demonstrated in the metallurgical testwork program (Section 13, Mineral Processing and Metallurgical Testwork). To achieve optimal overall gold recovery and Project economics, a series of targeted processing modifications are proposed to the selected off-site toll milling facility. These modifications are specific to the selected toll milling facility and were designed to align the existing processing configuration with the metallurgical response of ore from the Rowan deposit, particularly its high gravity-recoverable gold content and leaching characteristics. Alternative processing modifications could be required if a different off-site toll milling facility was selected; however, these modifications could reasonably be expected to be in the same order of magnitude.

17.1 Proposed Flowsheet

The proposed processing flowsheet for Rowan deposit ore consists of conventional crushing and grinding, gravity concentration, pre-oxidation, leaching, CIP gold recovery, elution, electrowinning, refining, and cyanide detoxification. Ore from the Rowan deposit will be mined from the underground, stockpiled in the ROM Ore Stockpile, primary crushed and sampled, and stockpiled in the Sampled Ore Stockpile at the Project site (Figure 18-1). After sampling, the crushed ore will be transported by haul truck to the selected off-site toll milling facility for processing to produce gold doré bars.

17.1.1 On-Site Crushing and Sampling at the Rowan Project

A Crusher and Sampler Tower will be installed at the Project site to crush and sample ore prior to ore being transported to the off-site toll milling facility. Run-of-Mine ore from the underground mine will be feed into the Crusher, a mobile primary jaw crusher (i.e., Cedarapids CRJ3042), and reduced to 3 inch (in.) minus material. This crushed ore will be conveyed to the Sample Tower, a fully integrated crushing and sampling system (i.e., Westpro ST-MCP50), which is designed to operate at 50 tonnes per hour (t/h) or 1,200 t/d and deliver representative daily composite samples for assay and metallurgical accounting.

The Sampe Tower will include a linear cross-stream sampler, secondary and tertiary crushing modules (i.e., jaw crusher and roll crusher), a bucket elevator, surge bin, variable-speed belt feeder, and two Vezin samplers. This Sample Tower configuration will produce two 25 kilogram (kg) composite samples per day of 3 mm minus (i.e., 1/8 in. minus) ore. These composite samples will be collected in a sealed bin on a mobile cart for transfer to an off-site laboratory. The reject ore from the Sample Tower not collected for composite sampling will be conveyed to the Sampled Ore Stockpile for trucking to the off-site toll milling facility for processing.

The Crusher and Sampler Tower are designed to operate 12 hours per day and 365 days per year on day shift only. This equipment is appropriately sized to support the Project's required processing range of 300 t/d to 475 t/d over the five years of Operations, averaging approximately 385 t/d (Section 16.6, Production Schedule).

17.1.2 Off-Site Toll Milling Facility

The selected off-site toll milling facility currently operates a conventional gravity-leach-CIP flowsheet with downstream gold recovery. The major processing steps of this facility include:

- crushing and grinding;
- gravity concentration;
- pre-leach thickening;
- leaching in agitation tanks;
- carbon-in-pulp adsorption;
- gold recovery via elution and electrowinning;
- gold refining to produce doré bars; and
- cyanide destruction of the CIP tailings.

This processing configuration of the off-site toll milling facility is compatible with ore from the Rowan deposit and could be used with limited modifications, based on the metallurgical testwork completed for the Project (Section 13, Mineral Processing and Metallurgical Testwork). However, metallurgical testwork has indicated that gold recovery is maximized through enhanced gravity recovery and sufficient retention time in the leach and CIP circuits (i.e., 24 hours to 48 hours), which may not be available when processing higher throughputs under the current configuration at the selected off-site toll milling facility. To achieve the recoveries demonstrated in metallurgical testwork, a number of process upgrades are recommended (Section 17.2, Proposed Off-Site Toll Milling Facility Modifications).

17.2 Proposed Off-Site Toll Milling Facility Modifications

Targeted processing upgrades (i.e., modifications) are proposed to the selected off-site toll milling facility to achieve optimal overall gold recovery from the maximum 475 t/d of ore from the Rowan deposit (Section 16.6, Production Schedule). These modifications are based on the results from the metallurgical testwork (Section 13, Mineral Processing and Metallurgical Testwork), which indicate that optimal gold recovery requires increased gravity capacity and sufficient residence time in the leaching and adsorption circuits. While the off-site toll milling facility could technically process ore from the Rowan deposit without these modifications, this approach would likely reduce overall gold recovery and was therefore not considered in the Technical Report. The proposed modifications are summarized in this subsection.

17.2.1 Crushing

The off-site toll milling facility currently includes a primary crushing circuit capable of reducing ROM ore to 3 in. minus to feed the grinding circuit. The existing grinding circuit can achieve the target primary grind size of 75 microns (µm) at this feed size; however, the existing grinding circuit would not be able to achieve the target grind size with the additional ore from the Rowan deposit.

To support the Project, a new mobile secondary crusher (i.e., pre-crusher) would be installed at the toll mill facility to maintain the target grind size at the higher throughput without significant modifications to the existing grinding circuit or increasing the current power requirements for the facility. The secondary crusher (i.e., Cedarapids CRC380XHLS Cone Screen Plant) would reduce the 3 in. minus feed down to 3/4 in. minus so that the grinding circuit could maintain a target primary grind size of 75 µm. The additional crushing capacity would reduce the risk of overloading the SAG mill and ball mill or compromising grind size, which could negatively affect downstream gold recovery.

17.2.2 Grinding

The off-site toll milling facility currently includes a grinding circuit, which consists of a semi-autogenous grinding (SAG) mill followed by a ball mill that is capable of achieving the target primary grind size of 75 µm when fed with suitably sized material.

With the installation of the secondary crusher to reduce feed size to 3/4 in. minus material size (Section 17.2.1, Crushing), only minor modifications to the grinding circuit would be required to accommodate the additional ore from the Project, such as modifying the SAG trommel and SAG screen apertures. The pre-crushed material would enable the grind circuit to maintain grind efficiency and throughput without increasing SAG mill or ball mill power nor significantly altering the current grinding configuration.

17.2.3 Gravity Concentration

The off-site toll milling facility currently includes a gravity circuit comprised of two Knelson centrifugal concentrators (i.e., KC XD-20) and associated screens designed to handle the base throughput of the processing plant. While this gravity circuit is effective at recovering coarse gold from the plant's current feed, it lacks the capacity to efficiently process the additional ore from the Rowan deposit.

To support the Project, a new gravity concentrator (i.e., KC XD-20), step deck vibrating screen (i.e., Knelson sizetec SDS48-6-5), and associated pump would be installed at the toll mill facility to treat the incremental ore feed from the Rowan deposit. Metallurgical testwork has shown that ore from the Project contains a high proportion of gravity-recoverable gold; without this gravity circuit expansion, a significant portion of this gold may bypass gravity gold recovery and report to the leach circuit where it may not be fully leached. This additional equipment would increase the gravity circuit's capacity, reduce gold losses, and improve overall plant recovery.

The gravity concentrate would be treated in the existing facility's existing CS1000 intensive leach reactor; this unit is expected to have sufficient capacity to accommodate the additional concentrate generated from processing of ore from the Project, and no modifications would be required.

17.2.4 Pre-Leach Thickening

The off-site toll milling facility existing pre-leach thickener is sufficient for the current throughput but does not have capacity to handle the combined feed from the existing ore source and ore from the Rowan deposit.

To support the Project, a 6 m diameter supplementary pre-leach thickener would be installed to process the additional slurry volume. The addition of this supplementary thickener would be more cost effective than replacing the existing thickener with a larger unit and would allow the slurry to be thickened to 50% solids prior to leaching, which is currently achieved in the off-site toll milling facility.

17.2.5 Leaching and Carbon-in-Pulp

The off-site toll milling facility currently includes a leach circuit designed to provide approximately 30 hours of residence time. With the additional ore from the Rowan deposit, the residence time would be reduced, potentially leading to incomplete gold leaching and lower overall gold recovery.

To support the Project, an additional pre-oxidation tank (i.e., 5 m diameter by 6 m high) and additional leach tank (i.e., 7.3 m diameter by 8.2 m high) would be installed in the leach circuit at the toll mill facility to maintain residence time and leach efficiency. These additional tanks would be equipped with an agitator to provide adequate mixing and oxygen transfer.

The existing CIP circuit at the toll mill facility would also be expanded with an additional tank (i.e., 3.6 m diameter by 4.8 m high), with an interstage screen and agitator to confirm sufficient carbon contact time for gold adsorption.

Steel troughs (i.e., launders) and piping in both the leach and CIP circuits would be upgraded to accommodate increased flow rates.

These modifications would be required to maintain metallurgical performance and are consistent with recovery projections based on metallurgical testwork (Section 13, Mineral Processing and Metallurgical Testwork).

17.2.6 Cyanide Detoxification

The off-site toll milling facility currently includes a cyanide detoxification circuit comprised of one tank, agitator, and pump designed for the base throughput of the processing plant. At increased throughput rates, this circuit would not provide sufficient residence time to meet environmental discharge regulations.

To support the Project, an additional detoxification tank (i.e., 8.6 m diameter by 11.4 m high), agitator, and pump would be added to maintain compliance with cyanide destruction targets (i.e., 50 mg/L CN_{WAD}). The expanded cyanide detoxification circuit would confirm that tailings from the combined ore feed were treated effectively prior to discharge in accordance with environmental discharge regulations.

17.2.7 Gold Recovery and Refining

The off-site toll milling facility existing gold recovery and refining circuits, which include acid wash, elution, electrowinning, and carbon regeneration, are expected to have sufficient capacity to accommodate the processing of ore from the Rowan deposit without requiring any processing equipment modifications.

In the off-site toll milling facility, loaded carbon would be removed from the CIP circuit once daily, or twice daily if required, to manage the increased gold loading associated with ore from the Project. The carbon would be acid washed in an existing acid wash vessel and then stripped in the existing elution circuit. Stripped carbon would be regenerated in an electric kiln before being returned to the CIP circuit.

The gold-bearing (i.e., pregnant) solutions from both the CIP circuit and the intensive leach circuit in the off-site toll milling facility would be directed to dedicated electrowinning cells for gold recovery. Gold would plate on stainless steel cathodes, while the non-gold bearing (i.e., barren) solution would flow back to a barren solution tank. Barren solution would be bled off as required to prevent a build-up of impurities.

Gold-rich sludge from the gold recovery and intensive leach circuits would be periodically washed off the cathodes using high-pressure water. This sludge would be filtered, dried, mixed with fluxes, and smelted in an existing induction furnace to produce gold doré. Electrowinning and refining would take place in a secure area, and precious metal products would be stored in a vault for shipping off site.

17.2.8 Tailings Management

The off-site toll milling facility currently includes a tailings management facility that would have sufficient capacity to accommodate the additional tailings generated from the processing of Rowan deposit ore. However, the next planned dam raise of this tailings management facility would need to be advanced by approximately one year to allow for continued tailings deposition capacity to accommodate the additional tailings from the Project. This tailings management facility construction schedule adjustment was incorporated into the Project schedule and included the capital cost estimate (Section 21, Capital and Operating Cost Estimates).

17.3 Power, Water, and Reagent Requirements

17.3.1 Power Supply

The off-site toll milling facility existing power infrastructure includes sufficient capacity to accommodate the increase in throughput associated with ore from the Rowan deposit. No upgrades would be required to the existing power supply and distribution infrastructure at the off-site toll milling facility. The additional power demand associated with processing ore from the Rowan deposit was included in the operating cost estimate (Section 21, Capital and Operating Cost Estimates).

17.3.2 Water Supply and Water Treatment

The off-site toll milling facility process water requirement would increase in proportion to the additional ore feed generated by the Project. This increased requirement could be accommodated by the existing water supply and water treatment systems at the off-site facility, and no upgrades or modifications would be anticipated to support the Project.

17.3.3 Reagents

The off-site toll milling facility reagent consumption would increase as a result of the additional ore feed generated by the Project. The existing reagent storage, handling, and dosing systems at the off-site toll milling facility would be expected to accommodate the increased reagent usage without upgrades or modifications. The additional reagent consumption was included in the operating cost estimate (Section 21, Capital and Operating Cost Estimates).

18 PROJECT INFRASTRUCTURE

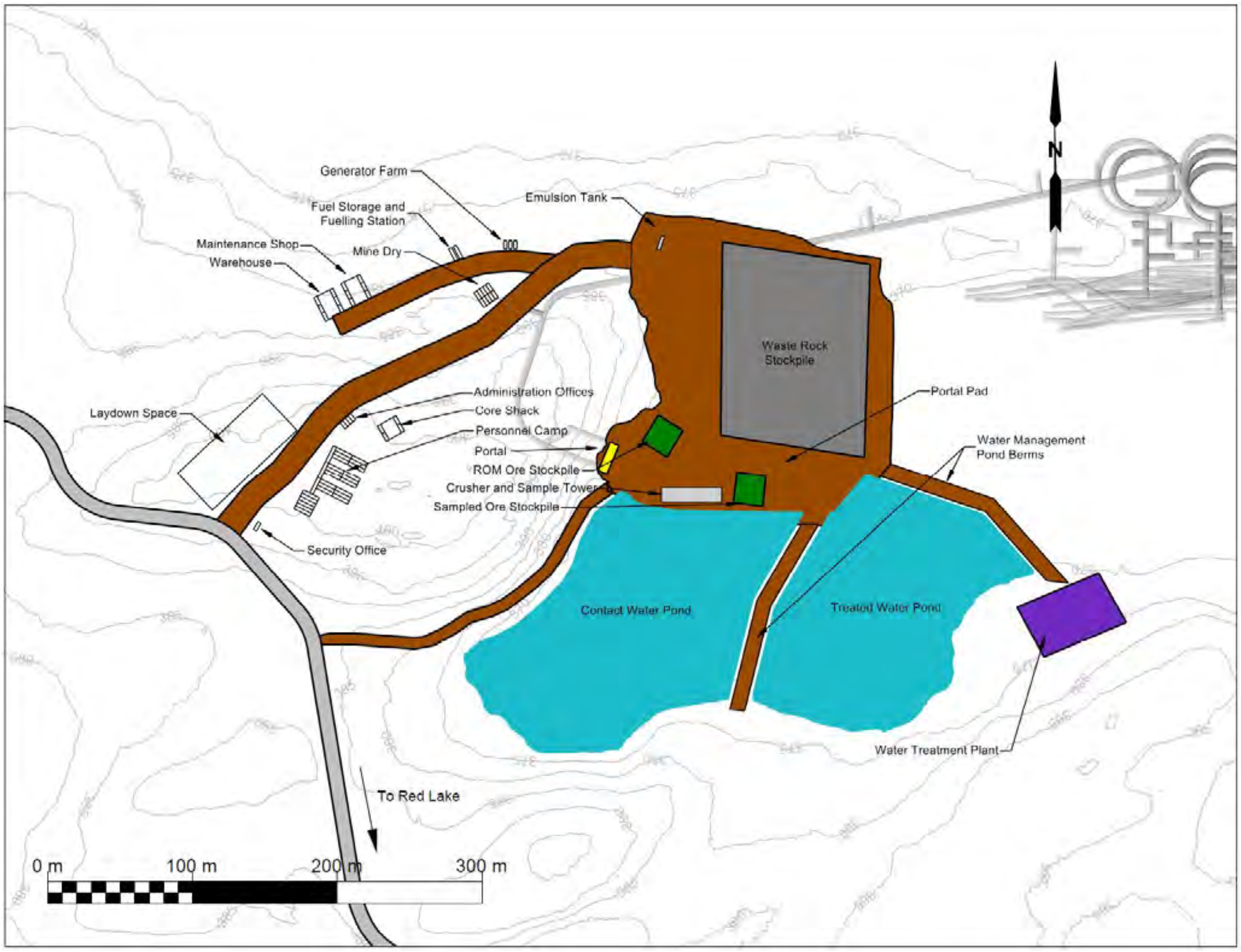
18.1 Introduction

The Project includes development and production of the underground mine as well as construction of civil infrastructure, ore and waste rock stockpiles, ore crushing and sampling facilities, water management and water treatment facilities, and ancillary infrastructure. There are no processing or tailings management facilities located at the Project site. Project infrastructure will include the following key facilities:

- underground mine development, including the Portal at surface;
- ore storage facilities, including the Run-of-Mine (ROM) Ore Stockpile and Sampled Ore Stockpile;
- Crusher and Sample Tower;
- waste rock management facilities, including the Portal Pad and Waste Rock Stockpile (WRS);
- water management infrastructure, including ponds, channels, pipelines, and Water Treatment Plant (WTP);
- ancillary infrastructure, including:
  - Mine Dry, Administration Offices, and maintenance and warehouse facilities;
  - fuel storage facilities and power generation facilities;
  - exploration facilities;
  - Personnel Camp;
  - laydown facilities; and
  - site roads.

Layout of infrastructure was completed with the goals of minimizing material handling, minimizing environmental effects, and consolidating infrastructure near the Portal. A layout of the Project site at the end of Operations is shown on Figure 18-1, and additional information on Project infrastructure is described in the following subsections.

Figure 18-1: Site General Arrangement for the Rowan Project



Notes:
ROM = Run-Of-Mine.

18.2 Ore Storage Facilities

Two ore storage facilities will be used on surface for the Project, the ROM Ore Stockpile and Sampled Ore Stockpile. The ROM Ore Stockpile will be located near the Portal entrance and will receive ore directly from the underground using haul trucks (Figure 18-1). The Sampled Ore Stockpile will be located adjacent to Sample Tower and will receive ore directly from the Sample Tower once it has been crushed and sampled (Section 18.3, Crusher and Sample Tower). Ore from the Sampled Ore Stockpile will be transported to an off-site toll milling facility for processing (Section 18.7, Off-Site Ore Transportation).

The ROM Ore Stockpile and Sampled Ore Stockpile were each designed with a maximum capacity of 2,000 t, which includes sufficient capacity to accommodate ore hauling from the underground and regular downtime associated with maintenance on the Crusher and Sample Tower. A surface loader will manage both ore stockpiles, load the Crusher, and load surface haul trucks for off-site ore transportation.

18.3 Crusher and Sample Tower

The Crusher and Sample Tower for the Project will crush and sample ore prior to ore being transported to the off-site toll milling facility. The Crusher, which will be a mobile crushing unit, will be fed ore from the ROM Ore Stockpile by a surface loader. All ore will pass through the Crusher and feed directly to the Sample Tower by conveyor. Crushed ore in the stationary Sample Tower will pass through a series of cutters and crushers to segregate out samples to identify gold grades in the ore prior to off-site processing. Two 25 kg samples will be produced daily from the Sample Tower, and all unsampled ore will feed directly to the Sampled Ore Stockpile by conveyor.

Additional information on on-site crushing and sampling is provided in Section 17.1.1, On-Site Crushing and Sampling at the Rowan Project.

18.4 Waste Rock and Water Management Facilities

18.4.1 Overview

Waste rock from underground mining will be managed on surface in the WRS located on the Portal Pad (Figure 18-1). The Portal Pad will be constructed immediately east of the Portal and will also contain two temporary ore stockpiles: the ROM Ore Stockpile and the Sampled Ore Stockpile. Two water management ponds will be constructed adjacent to the Portal Pad. The Contact Water Pond will manage contact water from underground dewatering and contact water runoff from the Portal Pad, and the Treated Water Pond will manage treated water discharged from the adjacent WTP. Ore will be crushed, temporarily stockpiled on the Portal Pad, and hauled off site for processing. Waste rock from underground mining development will be temporarily stored on surface in the WRS, and the majority of waste rock will be used as backfill in the excavated underground workings.

18.4.2 Design Criteria

The general high-level design parameters for the waste rock and water management facilities for the Project are summarized in Table 18-1. Tailings generated from the off-site processing of ore will be stored in the tailings management facility of the off-site toll facility.

Table 18-1: Waste Rock and Water Management Facilities High-Level Design Parameters for the Rowan Project

Design Parameter	Unit	Value
Operations length	years	5
Estimated mineral deposit	kt	705
Maximum off-site mill throughput	t/d	475
Peak WRS capacity	t	200,000
Waste rock dry density (assumed average)	t/m^3	2.0
Total combined ore storage volume in the ROM Ore Stockpile and Sampled Ore Stockpile	m^3	1,000
Geotechnical design – cut slopes	H:V	2:1
Geotechnical design – fill slopes	H:V	2:1

Notes:
WRS = Waste Rock Stockpile; kt = kilotonne; t/d = tonnes per day; t = tonne; t/m^3 = tonnes per cubic metre; H:V = horizontal to vertical.

The water management ponds will manage run-off from storm events up to the Inflow Design Flood (IDF) with flows exceeding the IDF conveyed from the ponds via an emergency discharge spillway. The Ontario *Lake and Rivers Improvement Act* Regulatory Flood, which is the estimated flood associated with the 1961 Timmins Storm transposed to the Project's watersheds (Timmins Flood; McMullen 1962, MNR 2004), was selected as the recommended IDF for the catchment area of the Project during Operations, at a value of 193 mm.

For the Project, treated water from the Treated Water Pond will be discharged to the environment when receiving waterways are ice-free, which is assumed to be approximately eight months of the year. During the remaining approximate four-month period in winter, treated water will be managed within the Treated Water Pond. The largest volume of the contact water to be managed and treated will be from underground mine dewatering.

18.4.3 Portal Pad

The Portal Pad will be constructed using local borrow source material with 2H:1V fill slopes. Waste rock from the underground mine will be used as construction material, where possible, to reduce borrow source material requirements. Foundation preparation for the Portal Pad will involve the clearing and grubbing of surface vegetation, followed by the removal of unsuitable foundation material, including topsoil and organic deposits. Cut slopes adjacent to the Portal Pad will be 2H:1V except at the Portal entrance, where the cut slopes will be 70° into intact bedrock.

18.4.4 Waste Rock Stockpile

Waste rock from underground mining will be managed on surface in the WRS located on the Portal Pad. The WRS layout provides capacity for approximately 200,000 t of material at an overall slope of 2H:1V. The WRS will be developed in stages depending on the mining sequence and is located in the contributing catchment area of the Portal Pad, with runoff conveyed towards the Contact Water Pond. It is expected that the majority of waste rock in the WRS will be used as part of backfilling the excavated underground workings with a small volume of waste rock remaining on surface at Closure.

18.4.5 Water Management Ponds

The Contact Water Pond and the Treated Water Pond will be constructed adjacent to the Portal Pad and will manage the following volumes while maintaining a minimum 1 m freeboard above the maximum IDF pond level to account for wave run-up protection, ice depth, and settlement. The total pond capacity of the Contract Water Pond is estimated at 70,000 m^3 and the total pond capacity of the Treated Water Pond is estimated at 80,000 m^3.

All site contact water will be managed within the Contact Water Pond. Contact water from the Contact Water Pond will be pumped to the WTP for treatment and then pumped to the Treated Water Pond prior to being discharged to the receiving environment. During the winter (i.e., approximately four months) when treated water cannot be discharged to the environment, treated water from the WTP will be temporarily stored in Treated Water Pond until this water can be discharged to the environment when waterways are ice-free.

Preparation of the water management pond basins will include the clearing and grubbing of surface vegetation, stripping of unsuitable materials, and smooth grading and shaping of the basins in preparation for liner placement. Both water management ponds will be continuously lined with a liner system comprised of a high-density polyethylene (HDPE) liner underlain by a protective non-woven geotextile. The liner system will prevent seepage from the water management ponds, and liners will be anchored in a trench on surrounding edge of each pond.

Two berms will be required to provide containment for the water management ponds. One berm will be located on the northeast side of the Treated Water Pond, and the other berm will be located between the Contact Water Pond and Treated Water Pond to fully separate the two ponds (Figure 18-1). Both berms will be constructed using local borrow source material with 2H:1V upstream and downstream slopes and a 10 m crest width. The West Perimeter Road will also act as a water management pond berm that will be located on the northwest side of the Contact Water Pond (Section 18.5, Ancillary Infrastructure). Foundation preparation for the berms will involve the removal of unsuitable foundation material, including topsoil and organic deposits. Waste rock from the underground mine will also be used as construction material, where possible, to reduce borrow source material requirements.

18.4.6 Surface Water Management

Surface water management structures around the perimeter of the Portal Pad will include a series of collection channels that will collect and direct contact water toward the Contact Water Pond, and diversion channels that will divert non-contact water around the Portal Pad and away from Project infrastructure. These channels, which will be lined with riprap to minimize erosion, will include:

- **West Perimeter Collection Channel:** To direct contact water from the Project infrastructure and the catchment area west of the Portal Pad toward the Contact Water Pond.
- **East Perimeter Collection Channel:** To direct contact water from the WRS and Portal Pad toward the Contact Water Pond and prevent this water from being released off the eastern edge of the Portal Pad.
- **North Perimeter Diversion Channel:** To divert non-contact water from the primary access road and catchment area north of the Portal Pad around Project infrastructure to discharge east, beyond the Portal Pad.

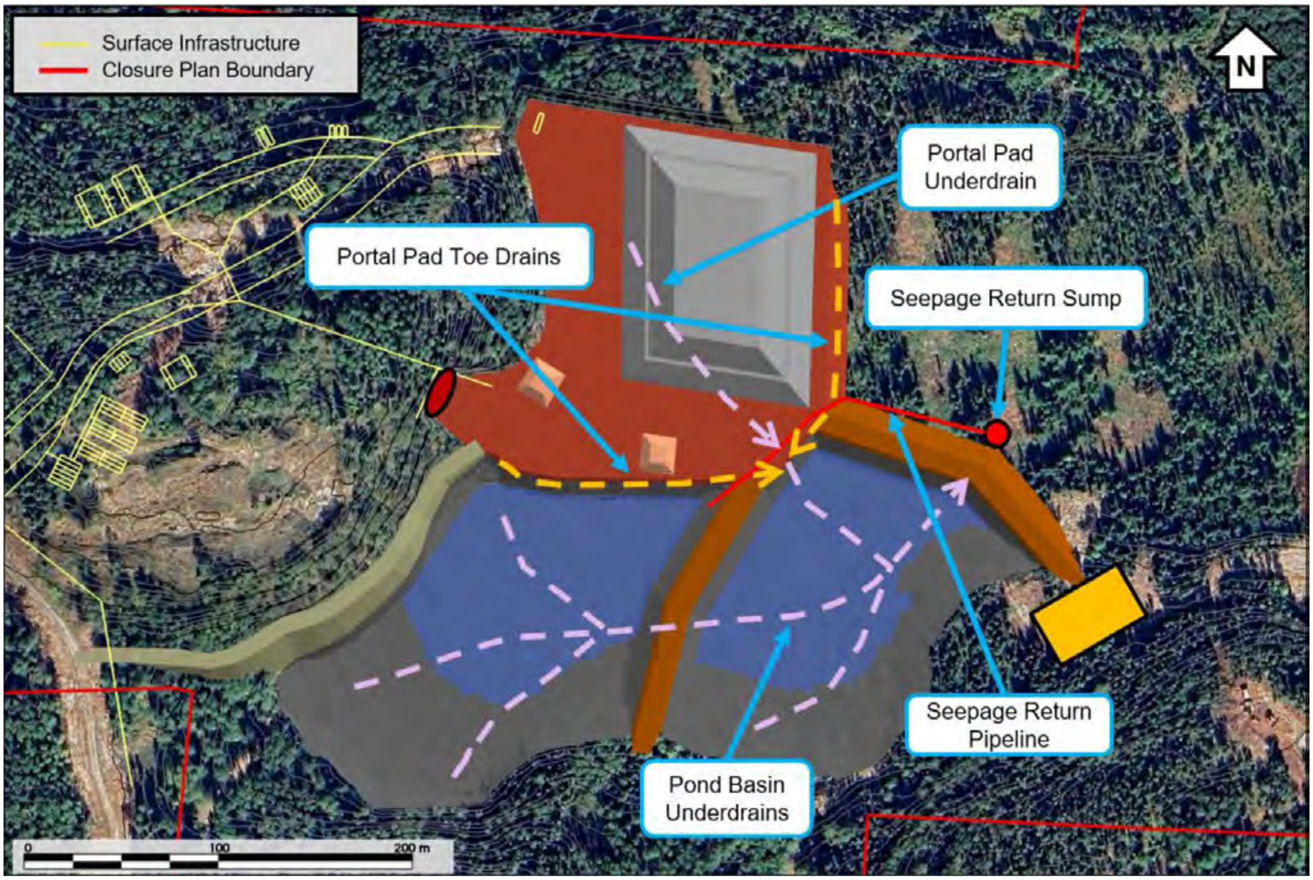
18.4.7 Seepage Management

Seepage management infrastructure will include a series of underdrains to facilitate the recovery of potential groundwater flows, liner leakage, and seepage from the Portal Pad. The underdrain system will include the following elements (Figure 18-2):

- **Portal Pad Underdrain:** Located under the main area of the Portal Pad and will extend under this pad and connect to the pond basin underdrains under the Treated Water Pond.
- **Portal Pad Toe Drains:** Located under and along the eastern and southern edges of the Portal Pad to collect potential seepage flowing downstream of this pad. The toe drains will connect to the pond basin underdrains under the Treated Water Pond.
- **Pond Basin Underdrains:** Located under both the Contact Water Pond and Treated Water Pond basins and will extend to the seepage return sump located downstream of the Treated Water Pond.

The underdrain system will direct all flows to the seepage return sump, where water will be pumped to the Contact Water Pond for treatment in the WTP. The underdrain system will be excavated into the natural ground surface following the foundation preparation works and will include drain gravel and perforated corrugated plastic tubing drainpipe.

Figure 18-2: Seepage Water Management System for the Rowan Project



Source: KP 2025.

18.4.8 Water Management Pipelines

Pipelines to support water management at the Project site will include:

- **WTP Intake Pipeline:** contact water pipeline from the Contact Water Pond to the WTP;
- **WTP to Treated Water Pond Pipeline:** non-contact water pipeline from the WTP to the Treated Water Pond;
- **Treated Water Discharge Pipeline:** treated water pipeline from the Treated Water Pond to the receiving environment for use during ice-free months; and
- **Seepage Return Pipeline:** contact water pipeline from the seepage return sump to the Contact Water Pond.

Pumps will be installed to move contact and non-contact water through these pipelines as required.

18.4.9 Water Balance

A high-level water balance model (WBM) was developed for the Project water management ponds, including the Contact Water Pond and Treated Water Pond. The WBM was used to evaluate the following components under average and wet climate conditions:

- Provide an understanding of major inflows and outflows of freshwater and contact water from Project infrastructure over the life of mine (LOM) to support mine infrastructure planning and design.
- Evaluate the contact water volumes for the Project to estimate annual water volumes that will require treatment and release to the downstream environment.
- Estimate the required operating storage capacity of the water management ponds and demonstrate that the storm runoff resulting from the IDF can be effectively managed within these ponds.

A large percentage of contact water treated in the WTP will be from underground mine dewatering. For WBM purposes, an underground dewatering rate of 500 m^3/day was assumed during Operations.

Contact water will be treated year-round in the WTP, with treated water only released during ice-free months. The WBM assumes that treated water will be discharged to the receiving environment from the Treated Water Pond for approximately eight months of the year; during the remaining approximately four months of the year, contact water will be treated and stored in the Treated Water Pond until discharge to the downstream environment resumes during ice-free months.

The WTP and treated water discharge systems were adequately sized to manage the contact water volume in an effective manner throughout the LOM to provide sufficient capacity in the water management ponds for the IDF and reduce the likelihood of an uncontrolled release of contact water to the environment during an extreme storm event.

18.4.10 Water Treatment Plant

Contact water will be generated from underground mine dewatering and surface water runoff from the Portal Pad, WRS, ore storage facilities, as well as from seepage water management systems for the Project (Section 18.4.7, Seepage Management). This contact water will be directed to the Contact Water Pond through a series of collection channels and pipelines and then pumped to the WTP for treatment. The WTP will operate year-round and discharge treated effluent to the Treated Water Pond. Treated water in the Treated Water Pond will be discharged to the receiving environment when ice-free via an outfall in Golden Arm of Red Lake.

The WTP was sized to treat 600 m^3/day based on the WBM for the Project. Treated effluent will comply with the Government of Canada Metal and Diamond Mining Effluent Regulations (MDMER) at the final discharge point and meet Canadian Water Quality Guidelines for the Protection of Aquatic Life (CCME 2025) requirements within the receiving environment. It is assumed that treatment may be required for ammonia, metals, and metalloids per the MDMER as no contact water chemistry data is available and pending development of site-specific treatment criteria. No treatment for cyanide is expected to be required as ore will not be processed at the Project site.

18.5 Ancillary Infrastructure

Ancillary infrastructure to support the construction and operation of the Project is listed in Table 18-2 and shown on Figure 18-1. This infrastructure is anticipated to include a range of prefabricated structures, including pre-engineered buildings (e.g., modular ATCO trailers) as well as facilities comprised of shipping containers for the walls and Quonset-style roofs.

Table 18-2: Ancillary Infrastructure Information for the Rowan Project

Ancillary Infrastructure	Infrastructure Description
Mine Dry	The Mine Dry will include a 26 m long by 13 m wide pre-engineered modular building with a compacted earth foundation. This facility will include approximately 60 mine-dry baskets, showers, and changerooms, as well as a space for personnel pre-shift meetings.
Administration Offices	The Administration Offices will include a 26 m long by 6.5 m wide pre-engineered modular building with a compacted earth foundation and will provide work spaces for the technical and supervision staff. This facility will include offices for the required technical services teams and will be connected to the wireless network at the Project site.
Maintenance Shop	The Maintenance Shop will include an 18 m long by 15 m wide structure comprised of shipping containers for the walls and a Quonset-style roof with a compacted earth foundation. This shop will support mining operations and provide a covered space for routine maintenance. It is assumed larger equipment maintenance projects (e.g., engine rebuilds) will be completed off site.
Warehouse	The Warehouse will include an 18 m long by 15 m wide structure comprised of shipping containers for the walls and a Quonset-style roof with a compacted earth foundation. This facility will store mechanical equipment, tools, and spare parts required to support mining operations and routine maintenance.
Fuel Storage and Fuelling Station	The Fuel Storage and Fuelling Station will include a 30 m long by 20 m wide compacted area for the placement of fuel tanks and level ground for vehicle fuelling. The fuel tanks will be located in a central area with a reinforced concrete containment and suitable spill control kits will be located at the facility. The diesel storage will include two 30,000 gal (115 m^3) tanks for a total diesel storage volume of 60,000 gal (230 m^3) and will provide fuel for all on-site heavy equipment and light vehicles. Bulk fuel will be delivered by truck to the Project site from local suppliers located near the Project (e.g., Red Lake).
Emulsion Tank	The Emulsion Tank will be a 20,000 gal (75 m^3) steel tank to provide bulk emulsion for the emulsion loaders for use in blasting. This tank will be located in a dedicated area on the northern end of the Portal Pad and will include appropriate setback distances from other Project infrastructure. Explosives will be resupplied as required by truck from Red Lake, Ontario. Once underground development begins, a single blasting cap and detonator magazine will be installed underground off the decline. Blast consumables will be collected in emulsion loaders from surface at the beginning of each shift for use deeper in the underground mine at working stopes and faces. All explosives handling, loading, management, and detonation will be conducted by authorized and trained personnel in accordance with the Ontario *Mining Act* and all other applicable regulations.
Generator Farm	The Generator Farm will have two primary 1.5 MW generators that will be housed in weatherproof enclosures with a compacted earth foundation that will provide power to all surface and underground infrastructure for the Project. A third 1.5 MW generator will provide backup power to the Project site during planned and unplanned downtime of either primary generator. Electricity for the Project will be stepped down to the required voltage levels and distributed across the Project site by buried distribution lines. Fuel for generators will be supplied from the Fuel Storage and Fuelling Station.
Exploration Facilities	The Exploration Facilities will include a 15 m long by 12 m wide structure comprised of shipping containers for the walls and a Quonset-style roof with a compacted earth foundation. These facilities will support ongoing exploration activities and provide space for core logging and core splitting.
Personnel Camp	The Personnel Camp will include pre-engineered modular buildings with a compacted earth foundation. This camp will include a capacity of 135 beds to house all personnel at the Project site, which includes a 15% contingency of beds for staff changeover, room unavailability, and additional contractors and guests at the Project site. The Personnel Camp will include kitchen and dining areas, recreational and fitness facilities, medical facilities, potable water and wastewater management, and will be connected to the wireless network at the Project site.
Security Office	The Security Office will include a 6.4 m long by 2.5 m wide pre-engineered modular building with a compacted earth foundation and will manage access to the Project site. The Project site will also include security fencing and gates as required to control access to the Project.
Laydown Space	The Laydown Space will include a 100 m long by 40 m wide area that will be cleared and levelled for laydown and staging space, equipment parking, shipment loading and unloading, and consumable storage to support all stages of the Project.

Table 18-2: Ancillary Infrastructure Information for the Rowan Project (continued)

Ancillary Infrastructure	Infrastructure Description
Site roads	Site roads will include primary and secondary roads to provide access between the Portal Pad and other Project infrastructure. Primary roads were designed as 16 m wide double-lane haul roads to accommodate on-site mobile equipment and will connect between the Portal Pad, Maintenance Shop and Warehouse facilities, and the existing access road to the Project site (Figure 18-1). The secondary road (i.e., West Perimeter Road) was designed as an 8 m wide single-lane haul road to accommodate off-site haul trucks for ore transportation and will connect from the Portal Pad to the existing access road to the Project site. Site roads will be constructed using local borrow source material with 2H:1V fill slopes and will include appropriate safety berms. Underground development waste rock will also be used as a construction material, where possible.

Notes:
m = metre; m^3 = cubic metres; gal = gallon; MW = megawatt; H:V = horizontal to vertical.

18.6 Surface Mobile Equipment

A summary of the estimated surface mobile equipment required for the Project is provided in Table 18-3.

Table 18-3: Surface Mobile Equipment Summary for the Rowan Project

Surface Mobile Equipment	Maximum Units
Loader (10 t)	2
Excavator	1
Grader	1
Forklift	2
Snow plow	1
Site shuttle	1
Light vehicles	4
Emergency response vehicle	1
Temporary cement plant for consolidated rock fill	1 – Contracted
Highway haul truck (20 t) for off-site ore transportation	4 – Contracted
Bus for off-site personnel transportation	2 – Contracted
Total Units	**20**

18.7 Off-Site Ore Transportation

Ore from the Sampled Ore Stockpile will be transported from the Project site to the off-site toll milling facility for processing. Ore transportation will be completed by contractor haul trucks suitable for highway transportation. While the specific make and model of highway haul trucks will be dependent on contractor availability at the time of Operations, it is estimated that a maximum of four trucks will be required to haul ore 24 hours per day and 365 days per year to the off-site toll milling facility, which was assumed to be a maximum of 100 km from the Project.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No market studies were conducted regarding the sale of gold doré from the Rowan Project as part of the Technical Report. Existing terms and conditions from previous sales by WRLG were used as the basis for the economic modelling (Section 22, Economic Analysis). Ore refining and ore transportation costs total to C$1.50/oz of gold produced.

19.2 Contracts

At this time, no contractual arrangements for shipping or ore refining exist, nor are there any contractual arrangements made for the sale of gold doré at this time. However, gold doré is a highly marketable product and it is not foreseen that there will be difficulties selling the gold doré produced from the Project.

19.3 Commodity Price Projections

For the economic analysis of the Project, the gold price was assumed at US$2,500/oz and a USD/CAD exchange rate of 1.35 was used. The gold price used is an analyst consensus long-term forecast price and as agreed with WRLG.

19.4 Royalties

The mine plan described in the Technical Report is only subject to a 2.5% NSR royalty held by Evolution Mining Gold Operations Ltd. (Evolution Mining). Additional information on royalties is provided in Section 22.3.4, Royalties.

19.5 Comments on Market Studies and Contracts

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cash flow analysis to support the Technical Report.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Summary

WRLG is continuing scientific and engineering studies at the Project site; First Nation, community, and regulatory consultations; monitoring programs; and design planning to progress the Project. WRLG has focused its efforts since acquisition on reducing the uncertainty and risk associated with any new mining development and is actively designing operations to minimize water use, improve water quality, and bring overall benefit to First Nations and local communities.

20.2 Permitting Requirements

20.2.1 Environmental Assessments

20.2.1.1 Federal Environmental Assessment

There is no requirement for a federal environmental assessment (EA).

The Project is not classed as a designated project under the *Impact Assessment Act* (IAA 2019) as it does not meet the thresholds defined in section 18 and section 19 of the Physical Activities Regulations, which are summarized as follows:

- 18(c) The construction, operation, decommissioning and abandonment of one a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5,000 t/day or more.
- 19(c) The expansion of an existing metal mine, other than a rare earth element mine, placer mine or uranium mine, if the expansion would result in an increase in the area of mining operations of 50% or more and the total ore production capacity of 5,000 t/day or more after the expansion.

20.2.1.2 Provincial Individual Environmental Assessment

There is no requirement for a provincial individual EA.

20.2.1.3 Class Environmental Assessment

The Project will require a Class EA because it is considered a Resource Stewardship and Facility Development Project in accordance with the Ministry of Natural Resources (MNR 2003). The Class EA process outlines how conservation authorities and regulators plan, design, evaluate, implement, and monitor projects while considering environmental effects (Conservation Ontario 2025). The Class EA process is mandated by Ontario's *Environmental Assessment Act* and ensures that environmental effects are assessed throughout the project lifecycle.

20.2.2 Federal and Provincial Permits

Additional federal, provincial, and municipal permits and approvals will be required for development of the Project; these permits and approvals can only be issued after the Class EA process is completed. WRLG will engage with all appropriate regulatory authorities (e.g., Fisheries and Ocean Canada, Transport Canada) to secure the required permit authorizations for the Project. Key permits and approvals for the Project will include:

- **Closure Plan:** A closure plan for the Project will be submitted in the first half of 2026. Closure plans are authorized by the Ontario Ministry of Mines (MINES) under the Ontario *Mining Act*.
- **Land Use Permit:** Allows for the construction of infrastructure on Crown land, which will be required for the construction of the treated water discharge pipeline in the receiving environment (i.e., Golden Arm of Red Lake); authorized by the Ministry of Natural Resources and Forestry (MNRF) under the *Public Lands Act*.
- **Forest Resource Licence:** Annual licence for clearing of any Crown merchantable timber at the Project site; authorized by the MNRF under the *Crown Forest Sustainability Act, 1994*.
- **Locations Approval, and Plans and Specifications Approval:** A two-stage approval process for the construction of dams or berms, including those associated with tailings facilities and/or new ponds and ditches; authorized by the MNRF under the *Lakes and Rivers Improvement Act*.
- **Permit to Take Water:** Allows for the drilling and installation of a water well for potable water supply for the Project; authorized by the Ministry of Environment, Conservation and Parks (MECP) under the *Ontario Water Resources Act*.
- **Permit to Take Water:** Allows for the taking of water from a water well for more than 50,000 litres per day (i.e., 50 m^3/d), which will be required for potable water withdrawal and dewatering of underground workings for the Project; authorized by the MECP under the *Ontario Water Resources Act*.
- **Environmental Compliance Approval for Air and Noise:** Required for air emissions that will be generated by mobile and stationary equipment operation at the Project site; authorized by the MECP under the Ontario *Environmental Protection Act*.
- **Environmental Compliance Approval for Sewage:** For approval to construct and operate a domestic sewage treatment system, water management ponds, WTP, and discharge pipeline for the Project; authorized by the MECP under the *Ontario Water Resources Act*.
- **Species at Risk Exemption and Benefit Program:** A species at risk exemption and benefit program for endangered species may be required in accordance with clause 17(2)c of the provincial *Endangered Species Act, 2007*. Identified endangered species in the area of the Project include, but are not limited to, bats, caribou, and wolverines. The *Endangered Species Act, 2007* is currently under review and will likely be amended; WRLG will work within the update regulations as required.

Other permits that may be required for the Project include:

- **Work Permit(s):** For development of water crossings less than 5 square kilometres (km^2), on private land or Crown-owned surface rights land.
- **Notice of Construction:** Notice is required before any contractor or construction activities take place at the Project site.

20.3 Environmental Studies

20.3.1 Environmental Baseline Studies

Environmental baseline studies and environmental modelling will be required to inform the Class EA application (Section 20.2.1, Environmental Assessment) as well as other federal, provincial, and municipal permit applications for the Project. Environmental baseline studies at the Project site began in 2023 and will continue as required to support the Project permitting requirements, including:

- air quality;
- surface water;
- groundwater and hydrogeology;
- geochemistry;
- aquatic plant and animal communities;
- terrestrial plant and animal communities; and
- archaeology.

Permit applications will be based on the environmental baseline studies and modelling outcomes. No harm to fish or biological systems in downstream waterbodies is currently expected near the Project site. WRLG will conduct species-specific studies to confirm the presence of species in the area of the Project, including species listed under the federal *Species at Risk Act* (SARA; Government of Canada 2023), federal Committee on the Status of Endangered Wildlife in Canada (COSEWIC 2022), and provincial *Endangered Species Act, 2007*. These species-specific studies may include:

- bat surveys (e.g., little brown bat; northern long-eared myotis, tri-colored myotis);
- migratory bird breeding surveys (e.g., bank swallow, barn swallow);
- winter track counts (e.g., woodland caribou, wolverine); and
- vegetation and plant community surveys.

20.3.2 Historical Activities and Infrastructure

Exploration activities have been completed in the area of Project since 1928, including the development of the Rowan Mine shaft. This historical shaft is located approximately 115 m west of the Project Portal at an approximate elevation of 400 metres above sea level (masl) (Section 5.4, Infrastructure). The historical headframe has been dismantled and capped with a concrete block. The entrance to the historical Rowan Mine adit, which is currently barricaded with broken muck, is located approximately 35 m southwest of the Project Portal.

Existing infrastructure at the Rowan Property includes an all-season trailer camp located in the Mount Jamie shaft area, which is currently used to support exploration and environmental baseline activities.

20.4 Social Considerations

WRLG has committed to engagement and consultation with local First Nations; municipal, provincial, and federal governments; other stakeholders; and the public throughout all stages of the Project. The intent of this engagement and consultation is to provide all interested parties with opportunities to learn about WRLG, identify concerns, and provide input with the goal of positively enhancing Project planning and development.

WRLG recognizes the importance of timely, full, and open discussion of concerns and options associated with the Project and the related concerns those individuals or communities may have regarding Project activities. WRLG will maintain open and honest communications with local communities and individual stakeholders throughout all stages of the Project. WRLG will establish operational practices, both now and into the future, that reflect the values, expectations, and needs of the community in which WRLG operates, and that are based on continued mutually respectful consultation with all First Nations and other stakeholders.

20.4.1 First Nations Considerations

The Rowan Property is located within the boundaries of Treaty # 3 (1873 and adhesions). Wabauskang First Nation and Lac Seul First Nation have identified the area of the Project as lying within their communities' traditional territory (Figure 20-1). MINES is still to advise WRLG whether the Wabauskang First Nation and Lac Seul First Nation represent the comprehensive list of First Nation communities to be engaged and consulted at this stage of the Project. MINES will also confirm that it will either delegate to WRLG or will undertake and fulfill the Crown's Duty to Consult where necessary. WRLG's primary role with First Nations at this time is to confirm that appropriate information sharing occurs, and WRLG has engaged with both the Wabauskang First Nation and Lac Seul First Nation.

When consultation requirements for the Project are confirmed by MINES and other regulatory agencies, WRLG will prepare an Indigenous Consultation Work Plan in accordance with the requirements of the *Mining Act*. WRLG will endeavour to prepare this work plan to meet the consultation requirements of all regulatory agencies that are involved with the Project permitting processes.

Figure 20-1: Area Map of Treaty #3 First Nations in Ontario, Canada

Source: WRLG 2025.

20.4.2 Community Considerations

Consultation with local and regional communities has commenced, including meeting with local and provincial governments as well as other parties. This consultation will continue as the Project progresses and will include meetings, Project updates, and other communications to confirm First Nations and other stakeholders are aware of WRLG's strategy for the Project.

Public information sessions were completed in 2024 as part of the Project permitting and Class EA processes in accordance with *Mining Act* requirements and guidance from MNRF. The overall engagement schedule is contingent on the Class EA category determination by MNRF. To reduce consultation fatigue with Indigenous communities and other stakeholders, WRLG intends to complete a single public information session and outreach campaign, which is consistent with recent precedent elsewhere in Ontario.

20.4.3 Regulator Considerations

WRLG is initiating early regulatory engagement prior to submission of the Class EA and other permitting authorizations; these regulatory authorities include MNRF, MECP, MINES, Ministry of Transportation (MTO), and the Ministry of Labour, Training and Skills Development (MOL). Consultation with these agencies, and other government agencies, is planned to confirm Class EA requirements, federal and provincial permit requirements, consultation requirements, and to obtain feedback so regulatory recommendations can be incorporated into the design and planning of the Project.

20.5 Mine Closure

20.5.1 Closure Objectives

The preliminary closure objectives for the Project include:

- **Design the Project site for Closure:** Identify the processes that will act upon Project components after Closure so that these components can be optimized in the design and development of the Project.
- **Achieve physical stability:** Project components that will remain post-Closure will be constructed or modified during Closure to be physically stable, and not erode, subside, or move from final locations due to physical forces to minimize risk to humans, wildlife, and the environment.
- **Achieve chemical stability:** All Project components that will remain on surface post-Closure will be chemically stable, and chemical constituents released from the Project site will not endanger humans, wildlife, or the environment.
- **Consider future use and aesthetics:** Consider compatibility of Project components that will remain on surface post-Closure with the surrounding lands after reclamation activities have been completed to support future land users and the creation of wildlife habitat.

20.5.2 Closure Activities

All closure activities will align with the O. Reg. 35/24 Rehabilitation of Lands and are expected to occur over a one-year period after the completion of Operations. Progressive reclamation activities will also occur during Operations, including the hauling waste rock from the WRS on surface to exhausted underground workings for permanent storage. It is expected that the majority of waste rock in the WRS will be used as part of underground backfilling, and a small volume of waste rock will remain on surface at Closure. There are no tailings management facilities at the Project site.

General aspects of the conceptual closure plan include:

- Underground workings will be backfilled with waste rock, the Portal will be plugged with waste rock, and underground mine will be passively flood to the natural water table, which will be located below the elevation of the Portal entrance.
- The small portion of waste rock in the WRS that will remain on surface after the underground workings are backfilled will be covered with locally available reclamation material to promote natural revegetation.
- Water management infrastructure will be reclaimed once no longer required as follows:
    - Diversion and collection channels will be graded to reestablish surface flow.

    - Ponds will be breached, liners will be removed, and pond berms and basins will be contoured to promote positive drainage and reestablish surface and groundwater flow recharge into the downstream area.
    - Pumps, pipelines, and the seepage underdrain system will be decommissioned and removed, and disturbed areas will be graded to promote positive drainage.
- All buildings and infrastructure will be dismantled and removed, and salvaged, recycled, or disposed of in accordance with application legislation. These areas will be recontoured to promote positive drainage.
- Equipment and materials will be salvaged where possible, and contractor-owned and leased items will be removed by the respective owners.
- Site roads, pads, laydown spaces, and borrow areas not required beyond Closure will be graded, contoured, and scarified to promote positive drainage.
- Exposed erodible materials will be reclaimed to provide long-term stabilization.

The WTP and associated water management infrastructure will remain operational until applicable water quality objectives are achieved. The WTP will then be decommissioned, and surface water from the Project site will be allowed to discharge naturally to the receiving environment.

20.5.3 Closure and Post-Closure Monitoring

Closure and post-Closure monitoring will be completed in the receiving environment, and results will be measured against applicable water quality objectives.

Closure monitoring is expected to include:

- construction-type monitoring across the Project site during decommissioning activities;
- water quality monitoring in Project water management ponds until applicable water quality guidelines are met; and
- regular inspections to confirm that closure and reclamation activities meet the closure objectives for the Project.

Post-closure monitoring will be required after completion of closure and reclamation activities. Post-Closure monitoring is expected to include:

- water quality sampling at the specified discharge location in accordance with applicable water quality objectives; and
- final environmental effects monitoring studies in accordance with applicable water quality objectives required to obtain status as a recognized closed mine from Environment Canada.

20.5.4 Closure Cost

WRLG inherited a mining legacy site with a history of almost a century of exploration and mining, including closure programs that were designed by previous operators. Closure costs associated with reclamation and remediation activities required for the Project is estimated at approximately C$3.2 million (Section 21.2.5.4, Closure Cost). The closure plan and related financial assurance for the Project will be updated with appropriate regulatory agencies during the Class EA permitting process.

21 CAPITAL AND OPERATING COST ESTIMATES

21.1 Introduction

The preliminary economic evaluation of the Rowan Project was based on the capital and operating cost estimates outlined in the Technical Report. These cost estimates reflect an underground mining operation, and account for the development of both underground and surface infrastructure, including underground mine development and production, ore storage facilities, crushing and sampling facilities, waste rock management facilities, water management facilities, WTP, and ancillary infrastructure (e.g., Mine Dry, fuel storage and power generation facilities, Personnel Camp) (Section 18, Project Infrastructure). Owner's costs and appropriate provisions were also included in the capital and operating cost estimates.

The capital and operating cost estimates are presented in Canadian dollars (C$), with no escalation or exchange rate variations factored into these estimates.

21.2 Capital Cost Estimate

21.2.1 Capital Cost Estimate Summary

Pre-production (i.e., initial) capital costs are estimated to be C$70.4 million and the LOM sustaining capital costs are estimated to be C$102.6 million. A summary of capital costs for the Project is provided in Table 21-1. In addition to these capital costs, closure costs were applied in Year 6 (i.e., Closure) for reclamation and remediation activities at a value of C$3.2 million.

Table 21-1: Capital Cost Estimate Summary for the Rowan Project

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Capital [C$M]
Mine Development	1000	13.4	58.6	72.1
Infrastructure	2000	14.2	3.2	17.4
Equipment	3000	9.1[(a)]	17.1	26.2
Crushing and Sampling	4000	2.2	-	2.2
Water Treatment Plant	5000	7.4	-	7.4
Waste Rock and Water Management	6000	4.9	-	4.9
Total Direct Costs		**51.3**	**78.9**	**130.2**
Project Indirect Costs	7000	1.4	-	1.4
Owner's Costs	8000	4.6	-	4.6
Contingency Costs	9000	13.2	23.7	36.9
Total Indirect Costs		**19.2**	**23.7**	**42.8**
Total Capital Costs		**70.4**	**102.6**	**173.0**

Notes:
Numbers may not add due to rounding.
a) There is an additional cost of C$15.3 million required for mining equipment associated with the assumed leasing agreement, which is included in the operating cost estimate for the Project (Section 21.3, Operating Cost Estimate).
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.2 Basis of Estimate

The capital cost estimate includes all expenditures required to develop and sustain the Project. The Project is designed as an underground mining operation with off-site toll milling and is supported by both on-site and off-site infrastructure.

The capital cost estimate conforms to a PEA-level estimate with a ±30% accuracy range which is within the boundaries of the Association for the Advancement of Cost Engineering International Class 5 guidelines (AACE International 2012).

The capital cost estimate was developed using a work breakdown structure (WBS) in Canadian dollars in the second quarter of 2025 (Q2 2025) and was based on a combination of industry benchmarks, data from best available information from nearby mine sites, vendor quotations, and relevant project experience.

The data for the capital cost estimate was derived from multiple sources, including:

- LOM mining schedule;
- conceptual engineering designs prepared by Fuse Advisors, Knight Piésold, and Integrated Sustainability;
- major mechanical equipment and infrastructure costs derived from vendor quotations;
- auxiliary equipment and infrastructure costs based on industry benchmarks; and
- topographical information for earthworks.

21.2.3 Direct Capital Costs (WBS 1000 to 6000)

21.2.3.1 Mine Development (WBS 1000)

The mine development capital cost estimate for the Project was estimated using a combination of first-principles engineering and benchmarking against comparable underground mining operations in Ontario, Canada. Labour and equipment rates were derived from benchmarks, and power consumption costs were estimated based on spot diesel prices and projected fuel usage in the Generator Farm.

Pre-production mine operating costs incurred prior to the commencement of commercial operations were capitalized as part of the development capital. These capitalized operating costs include costs associated with drilling, blasting, mucking, ground support, shotcrete, cable bolting, mine services, ventilation raise development, as well as a 15% general allowance. All Project site preparation and development activities required to initiate underground operations were fully capitalized.

The mine development capital costs for the Project are summarized in Table 21-2. It is the QP's opinion that these estimates are reasonable for the Project location and planned mine development for the Project and can be used for the Technical Report.

Table 21-2: Mine Development Capital Cost Estimate Summary for the Rowan Project (WBS 1000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Labour	1100	7.0	32.2	39.3
Equipment	1200	2.5	7.3	9.8
Development Consumables	1300	3.2	8.7	11.9
Drop Raise Development	1400	-	0.5	0.5
Raise Bore Development	1500	-	1.7	1.7
Power	1600	0.7	8.2	8.9
Total Cost		**13.4**	**58.6**	**72.1**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.3.2 Infrastructure (WBS 2000)

The infrastructure capital cost estimate for the Project includes both underground and surface components required to support underground mine development and operation. This cost includes the installation of underground mine services, such as ventilation, dewatering, electrical systems, site storage facilities (e.g., explosives magazine), as well as essential facilities including the Portal, escapeways, and maintenance areas.

Major infrastructure capital costs were developed based on vendor quotations, and auxiliary infrastructure capital costs were based on industry benchmark data.

A summary of the infrastructure capital costs for the Project is provided in Table 21-3.

Table 21-3: Infrastructure Capital Cost Estimate Summary for the Rowan Project (WBS 2000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Underground Infrastructure	**2100**	**3.6**	**3.2**	**6.8**
Safety and Emergency infrastructure	2110	0.1	1.1	1.2
Site Storage Facilities	2120	0.5	-	0.5
Utilities and Services	2130	1.2	0.5	1.7
Ventilation and Heating	2140	1.6	1.7	3.2
Portal Construction	2150	0.2	-	0.2
Surface Infrastructure	**2200**	**10.6**	**-**	**10.6**
Power Supply and Utilities	2210	3.1	-	3.1
Accommodation and Administration Facilities	2220	5.3	-	5.3
Security and Support Infrastructure	2230	0.3	-	0.3
Maintenance and Storage Facilities	2240	1.4	-	1.4
Exploration Geology Support	2250	0.2	-	0.2
Material Handling and Processing	2260	0.1	-	0.1
Surface Earthworks and Roads	2270	0.3	-	0.3
Total Cost		**14.2**	**3.2**	**17.4**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.3.3 Mining Equipment (WBS 3000)

The mining equipment capital cost estimate for the Project includes both underground and surface mobile equipment fleets required to support underground and surface activities during Construction, Operations, and Closure. All major equipment was assumed to be purchased through a financing agreement rather than lease or rental agreements.

Mobile equipment selection was based on the mine production schedule and expected equipment productivity. The underground mobile equipment fleet includes haul trucks, LHDs, underground drills, bolters, and support vehicles (Section 16.4, Underground Mobile Equipment). The surface mobile equipment fleet will support site operations and maintenance, site services, emergency response, and ore transportation (Section 18.6, Surface Mobile Equipment).

The mining equipment capital cost estimate was developed using vendor quotes, historical data, and benchmarking against similar mining projects. Multiple financing scenarios were evaluated to assess the effect of loan terms and interest rates. The selected financing scenario assumes a 48-month term with an interest rate of 8.5%, which results in total payments of approximately C$26.2 million, including C$3.5 million in interest.

A summary of the mining equipment capital costs for the Project is provided in Table 21-4. The initial capital cost for underground equipment of C$8.1 million is the upfront payment required for financing; additional equipment payments of C$15.3 million will be required during the assumed leasing agreement that are included in the operating cost estimate for the Project (Section 21.3, Operating Cost Estimate).

Table 21-4: Mining Equipment Capital Cost Estimate Summary for the Rowan Project (WBS 3000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Underground Equipment	**3100**	**8.1**[(a)]	**0**	**23.4**
Haulage & Loading Equipment	3110	4.3	8.0	12.3
Development & Production Drills	3120	2.0	3.7	5.7
Charging & Explosives Handling	3130	0.6	1.2	1.8
Underground Auxiliary Equipment	3140	1.3	2.4	3.7
Surface Equipment	**3200**	**1.0**	**1.8**	**2.8**
Total Cost		**9.1**	**17.1**	**26.2**

Notes:
Numbers may not add due to rounding.
a) There is an additional cost of C$15.3 million required for mining equipment associated with the assumed leasing agreement, which is included in the operating cost estimate for the Project (Section 21.3, Operating Cost Estimate).
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.3.4 Crushing and Sampling (WBS 4000)

Crushing and sampling capital cost estimate relates to on-site ore crushing and sampling prior to ore being transported to an off-site toll milling facility. The total daily throughput of ore that will be crushed and sampled at Project site will range from 300 t/d to 475 t/d over the five years of Operations, averaging approximately 385 t/d. There is no on-site processing infrastructure for the Project.

To support this approach, capital expenditures were allocated for the procurement, installation, and commissioning of the on-site infrastructure. The crushing and sampling capital cost estimate includes crushing and sampling equipment, electrical cabling, and the associated labour.

Crushing and sampling equipment costs were based on vendor quotations, and cabling and labour costs were benchmarked against similar operations.

A summary of the crushing and sampling capital costs for the Project is provided in Table 21-5.

Table 21-5: Crushing and Sampling Capital Cost Estimate Summary for the Rowan Project (WBS 4000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Crushing/Sampling Equipment	4100	2.2	-	2.2
Cabling	4200	0.1	-	0.1
Labour	4300	0.01	-	0.01
Total Cost		**2.2**	**-**	**2.2**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.3.5 Water Treatment Plant (WBS 5000)

The Water Treatment Plant capital cost estimate includes the modular, winterized WTP that will manage contact water at the Project, with a designed treatment capacity of approximately 600 m^3/day. The WTP will operate year-round, treating contact water from the Contact Water Pond and discharging to the Treated Water Pond.

The Water Treatment Plant capital cost estimate is a benchmark Class 5 estimate, developed using:

- Integrated Sustainability's internal database from similar Canadian mine water treatment projects; and
- published industry data, including information from projects located in British Columbia (e.g., Morrison Copper-Gold Project owned by Pacific Booker Minerals Inc., KSM Project owned by Seabridge Gold Inc.).

The Water Treatment Plant capital cost estimate includes:

- engineering and project management;
- procurement, including process / mechanical equipment, electrical equipment, instrumentation and controls, utilities, modules and prefabrication, and buildings;
- site labor and subcontracts, including earthworks, concrete and foundations, building installation, structural steel and supports, mechanical and piping installation, and electrical and instrumentation installation;
- field material costs;
- construction management; and
- commissioning and start-up.

Water Treatment Plant capital cost estimate excludes:

- mine water source controls;
- water collection systems, including mine dewatering, surface water collection, and seepage collection;
- infrastructure related to contact water and non-contact water segregation;
- infrastructure related to non-contact water management;
- untreated contact water storage;

- treated contact water storage;
- water management pipelines; and
- WTP site access and utilities supply.

Additionally, Owner's costs and other indirect costs (e.g., construction indirect costs) are excluded from this estimate. A summary of WTP capital cost estimate for the Project is provided in Table 21-6.

Table 21-6: Water Treatment Plant Capital Cost Estimate Summary for the Rowan Project (WBS 5000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Water Treatment Plant	5000	7.4	-	7.4
Total Cost		**7.4**	**-**	**7.4**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.3.6 Waste Rock and Water Management Cost (WBS 6000)

The waste rock and water management capital cost estimate for the Project includes key surface infrastructure for waste rock and water management to confirm compliance with environmental and operational requirements. These capital costs were allocated for the construction of water management ponds, systems, and pipelines, as well as supporting waste rock components such as construction of the Portal Pad, site roads, and general Project site grading.

The waste rock and water management capital cost estimate reflects the initial installed capital cost and excludes indirect costs, provisions, and contingency. A summary of waste rock and water management capital costs for the Project is provided in Table 21-7.

Table 21-7: Waste Rock and Water Management Capital Cost Estimate Summary for the Rowan Project (WBS 6000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Portal Pad Earthworks	6100	1.0	-	1.0
Water Management	6200	3.8	-	3.8
Water Management Ponds	6210	2.7	-	2.7
Collection Channels	6220	0.4	-	0.4
Water Management Pipelines	6230	0.7	-	0.7
Access Roads	6300	0.1	-	0.1
Contact Water Pond West Perimeter Road	6310	0.1	-	0.1
Total Cost		**4.9**	**-**	**4.9**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.4 Indirect Capital Costs (WBS 7000 to 9000)

The indirect capital cost estimate was calculated as a percentage of key direct cost components, including mine development, equipment, crushing and sampling, site infrastructure, WTP, and waste rock and water management systems. These indirect costs represent approximately 33% of the total direct capital costs over the LOM.

A summary of the indirect capital costs for the Project is provided in Table 21-8.

Table 21-8: Indirect Capital Cost Estimate Summary for the Rowan Project (WBS 7000 to 9000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Project Indirect Costs	7000	1.4	-	1.4
Owner's Costs	8000	4.6	-	4.6
Contingency Costs	9000	13.2	23.7	36.9
Total Cost		**19.2**	**23.7**	**42.8**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.4.1 Project Indirect Costs (WBS 7000)

The Project indirect cost estimate encompasses the delivery of construction materials and execution of construction activities. These costs include engineering, procurement, and construction management (EPCM) services associated with waste rock and water management, as well as construction indirect costs associated with the WTP.

The Project indirect cost estimate excludes indirect costs related to mine infrastructure, and no indirect costs were applied to sustaining capital.

The Project indirect cost estimate was based on historical data and project experience from Knight Piésold and Integrated Sustainability and represents approximately 11% of the total direct capital cost related to water treatment and water management infrastructure.

A summary of the Project indirect costs for the Project is provided in Table 21-9.

Table 21-9: Project Indirect Cost Estimate Summary for the Rowan Project (WBS 7000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Waste Rock and Water Management	**7100**	**0.6**	**-**	**0.6**
Engineering and Procurement	7110	0.3	-	0.3
Construction Management	7120	0.3	-	0.3
Freight and Duties	7130	0.1	-	0.1
Water Treatment Plant	**7200**	**0.8**	**-**	**0.8**
Construction Indirect Field Costs	7210	0.8	-	0.8
Total Cost		**1.4**	**-**	**1.4**

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.4.2 Owner's Costs (WBS 8000)

The Owner's indirect cost estimate for the Project was assumed as 10% of the total direct capital cost estimate, excluding mining equipment (WBS 8000). This indirect cost estimate was benchmarked against comparable recent projects.

A summary of the Owner's indirect costs for the Project is provided in Table 21-10.

Table 21-10: Owner's Indirect Cost Estimate Summary for the Rowan Project (WBS 8000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]
Owner's Costs	8000	4.6	-	4.6
Total Cost		**4.6**	**-**	**4.6**

Notes:
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.4.3 Contingency Costs (WBS 9000)

The contingency cost estimate for the Project was used to adjust for variations between estimated and actual costs for equipment and materials. These contingency costs will fluctuate according to the contract terms and the client's demands. The overall direct capital costs include a provision to offset the risks from these uncertainties.

The total contingency cost estimate for the Project capital was estimated to be C$36.9 million, which was calculated based on varying levels of cost uncertainty across key Project components. A 30% contingency was applied to the overall direct mining capital cost for a total of C$34.7 million (WBS 9100), which includes C$11.0 million in initial capital and C$23.7 million in sustaining capital. A 10% contingency was applied to the overall direct crushing and sampling capital cost for a total of C$0.2 million (WBS 9200), and a 40% contingency was applied to the overall waste rock and water management capital cost for a total of C$1.9 million (WBS 9300).

The contingency cost estimate does not account for:

- changes to market conditions affecting the cost of labour or materials;
- changes of scope within the general production and operating parameters;
- effects of industrial disruptions;
- abnormal weather conditions;
- financial modelling;
- technical engineering refinement; and
- estimate inaccuracy.

A summary of the contingency costs for the Project is provided in Table 21-11.

Table 21-11: Contingency Cost Estimate Summary for the Rowan Project (WBS 9000)

WBS Description	WBS	Initial Capital [C$M]	Sustaining Capital [C$M]	Total Cost [C$M]	Contingency [%]
Mining	9100	11.0	23.7	34.7	30%
Crushing and Sampling	9200	0.2	-	0.2	10%
Waste Rock and Water Management	9300	1.9	-	1.9	40%
Total Cost		**13.2**	**23.7**	**36.9**	

Notes:
Numbers may not add due to rounding.
WBS = work breakdown structure; C$M = million Canadian dollars; - = not applicable.

21.2.5 Sustaining Capital

The sustaining capital cost estimate for the Project was related to preserving the current assets' production capability and implementing the current production plan. The LOM project sustaining capital is C$102.6 million, which includes C$78.9 million in direct capital costs and C$23.7 million in indirect capital costs.

21.2.5.1 Mine Development

The sustaining capital costs for mine development includes continuous investments in the underground mobile equipment fleet, underground development consumables, labour, and power required to support operations throughout the LOM. The total LOM sustaining capital allocated to mine development is estimated to be C$58.6 million.

21.2.5.2 Infrastructure

The sustaining capital for underground infrastructure covers ongoing requirements to maintain safety systems, ventilation and heating, and to support continued access development to production levels. The total LOM sustaining capital allocated to on-site infrastructure is estimated to be C$3.2 million.

21.2.5.3 Equipment

All major mine equipment is planned to be purchased through financing from vendors. Equipment financing rates were estimated from best available information from nearby mine sites. Down payments and monthly lease payments were capitalized through the initial and sustaining periods of the Project. An initial payment of C$9.1 million is scheduled for Year 1, and the remaining C$17.1 million is allocated as sustaining capital over the LOM.

21.2.5.4 Closure Cost

The estimated total reclamation and closure costs for the Project is C$3.2 million, which excludes taxes and contingency. Closure costs were benchmarked against recent projects in similar jurisdictions. This estimate was based on a benchmark unit rate of C$4.50 per tonne of processed ore and is scheduled to be incurred in Year 6 of the LOM.

21.3 Operating Cost Estimate

21.3.1 Operating Cost Estimate Summary

The operating costs for the Project include expenditures related to on-site underground mining, waste rock and water management, water treatment, crushing and sampling, and the associated maintenance, power supply, and general and administrative costs, as well as off-site toll milling and trucking of ore.

Total operating costs over the LOM for the Project are estimated to be C$213.6 million, with average annual operating costs over the LOM of C$42.7 million. The total operating unit cost for the Project is estimated to be C$302.8/t processed, which includes an on-site operating unit cost of C$207.1/t processed, and an off-site operating unit cost of C$95.7/t processed.

A summary of the estimated operating costs for the Project is presented in Table 21-12.

Table 21-12: Operating Cost Estimate Summary for the Rowan Project

Cost Centre	LOM Cost [C$M]	Annual Average Cost [C$M]	LOM Cost [C$/t Processed]	Operating Cost [%]
On-Site Costs	**146.0**	**29.2**	**207.1**	**68%**
Mining	134.0	26.8	190.0	63%
Waste Rock and Water Management	0.4	0.1	0.5	0%
Water Treatment	3.0	0.6	4.3	1%
Crushing and Sampling	8.7	1.7	12.3	4%
Off-Site Costs	**67.5**	**13.5**	**95.7**	**32%**
Toll Milling	47.6	9.5	67.4	22%
Trucking to Toll Mill	10.6	2.1	15.0	5%
General and Administrative	9.4	1.9	13.3	4%
Total Operating Costs	**213.6**	**42.7**	**302.8**	**100%**

Notes:
Numbers may not add due to rounding.
LOM = life of mine; C$M = million Canadian dollars; C$/t = Canadian dollar per tonne.

21.3.2 Basis of Estimate

The operating cost estimate for the Project was developed using a combination of vendor quotations, cost database information, and benchmark data from comparable operations.

The operating cost estimate conforms to a PEA-level estimate with a ±30% accuracy range which is within the boundaries of the Association for the Advancement of Cost Engineering International Class 5 guidelines (AACE International 2012).

The data for the operating cost estimate was derived from multiple sources, including:

- mine operating costs developed from first principles and applied directly to the Project mine production schedule;
- processing (i.e., crushing and sampling) costs determined using vendor quotes and benchmarked values from similar crushing and material handling operations; and
- general and administrative costs estimated based on benchmarks from comparable projects of similar scale.

The operating cost estimate was developed in Q2 2025 Canadian dollars, with no provisions for inflation or exchange rate variation unless specifically noted. A USD/CAD exchange rate of 1.35 was used where applicable.

Key assumptions used to estimate the operating costs for the Project are listed in Table 21-13.

Table 21-13: Operating Cost Key Assumptions for the Rowan Project

Item	Unit	Value
Average Power Consumption	MW	2.3
Diesel Cost	$/litre	1.03
LOM Average Manpower	employees	200
Power Cost	$/kWh	0.25
General and Administrative	$/t	15
Trucking	cents per tonne km	20

Notes:
MW = megawatt; $ = dollar; LOM = life of mine; kWh = kilowatt-hour; t = tonne; km = kilometre.

21.3.3 On-Site Operating Costs

On-site operating costs are described below and are summarized in Table 21-12.

21.3.3.1 Mining

The mine operating cost estimate includes expenditures associated with all activities directly involved in the development and production of ore and waste rock, including drilling, blasting, mucking, loading, and hauling of material to the surface. These operating costs were developed from first principles and were aligned with the LOM production schedule.

The operating cost model incorporates the following major functional areas:

- Labour: Includes direct labour costs for production crews, support services, and surface operators. Labour costs also include camp services and safety personnel. Labour compensation was based on a combination costs from best available information from nearby mine sites and benchmarked salaries with associated burden.
- Equipment: Includes hourly operating costs of all mobile equipment (e.g., haul trucks, drills), which includes contractor-operated units.
- Development consumables: Includes unit costs for development activities applied to advance metres developed each year (e.g., drilling, blasting, ground support, and mine services). A 15% allowance is included for variability. Consolidated rock fill placement costs are also included in this functional area.
- Power: Power consumption for underground operations is estimated to be $0.25/kWh, with energy demand distributed across all Project years of active production. Power costs reflect generator fuel consumption based on tonnes handled and equipment use.
- Contingency: A 10% contingency is applied to mining operating costs to account for uncertainties in cost inputs and operational variability.

The LOM operating costs for the Project are summarized in Table 21-14. The mining operating cost estimate is approximately C$134.0 million, with an average of C$190.0/t processed over production years.

Table 21-14: Mining Operating Cost Estimate Summary for the Rowan Project

Cost Centre	LOM Cost [C$M]	LOM Cost [C$/t Processed]	Operating Cost [%]
Labour	67.0	95.1	50%
Equipment	29.4	41.6	22%
Development Consumables	8.5	12.1	6%
Power	16.9	24.0	13%
Contingency	12.2	17.3	9%
Total Cost	**134.0**	**190.0**	**100%**

Notes:
Numbers may not add due to rounding.
LOM = life of mine; C$M = million Canadian dollars; C$/t = Canadian dollar per tonne; % = percent.

21.3.3.2 Waste Rock and Water Management

The waste rock and water management operating cost estimate for the Project is approximately C$0.4 million, which includes of a 40% contingency. This estimate covers the operation and maintenance of water management infrastructure, including components supporting the WTP (e.g., pumps, pipelines).

21.3.3.3 Water Treatment

The water treatment operating cost estimate for the Project was benchmarked at approximately C$0.6 million per year, which was based on the WTP capital cost estimate and Integrated Sustainability's experience with similar mine water treatment systems.

Total water treatment operating cost over the LOM is estimated to be approximately C$3.0 million, and includes operating labor, water treatment chemicals, utilities (e.g., electricity, fuel), maintenance and repairs, and waste disposal.

21.3.3.4 Crushing and Sampling Tower

The crushing and sampling operating cost estimate for the Project is approximately C$8.7 million, based on a unit cost of C$12.30 per tonne processed. The Crusher and Sample Tower will prepare ore on site for processing prior to ore being transported to an off-site toll milling facility.

The unit cost estimate for crushing and sampling is derived from a combination of benchmark data and validated sources, including:

- labour: C$10.09/t, based on estimates from best available information from nearby mine sites;
- power: C$0.52/t, consistent with assumptions best available information from nearby mine sites;
- parts and consumables: C$0.63/t, estimated as 5% of capital cost;
- supplies and materials: C$0.63/t, estimated as 5% of capital cost; and
- services (e.g., assays): C$0.42/t, validated through consultation with WRLG on current assay laboratory costs.

21.3.4 Off-Site Operating Costs

Off-site operating costs are described below and are summarized in Table 21-12.

21.3.4.1 Toll Milling

The toll milling operating cost estimate for the Project is C$47.6 million, which is based on a unit cost of C$67.44 per tonne processed. This operating cost reflects the comprehensive fee that would be charged by an off-site toll milling facility and covers the off-site processing of ore from the Project.

The toll milling fee was based on the LOM production schedule, with the Project ore production ranging from 300 t/d to 475 t/d, and averaging 385 t/d (Section 16.6, Production Schedule).

The toll milling operating cost estimate was derived from detailed benchmarking and recent toll processing quotes, and was adjusted for inflation and operational scale. The unit off-site toll milling operating cost is estimate at C$67.44/t; this unit cost includes fixed costs, variable costs, and a 20% profit margin applied by the off-site toll milling facility as shown in Table 21-15.

Table 21-15: Unit Off-Site Toll Milling Operating Cost Breakdown for the Rowan Project

Description	Unit	Value
Fixed costs - capital infrastructure associated with ore processing	**C$/t**	**18.55**
Variable costs	**C$/t**	**35.40**
Power	C$/t	9.91
Reagents	C$/t	10.22
Parts and consumables	C$/t	8.00
Supplies and materials	C$/t	5.00
Services and maintenance	C$/t	0.06
Tailings handling and disposal	C$/t	0.61
Labour for secondary crushing	C$/t	1.60
Profit margin applied by off-site toll milling facility (20%)	**C$/t**	**13.49**
Total Unit toll milling operating cost	**C$/t**	**67.44**

Notes:
C$/t = Canadian dollar per tonne.

21.3.4.2 Trucking to Toll Mill

The trucking to toll mill operating cost estimate for the Project is C$10.6 million and was associated with trucking ore from the Project site to an off-site toll milling facility. The trucking unit rate used for this cost estimate was approximately $0.20/t per kilometre, which is a benchmarked value commonly applied in comparable mining projects in Ontario.

21.3.4.3 General and Administration

The general and administration operating cost estimate for the Project was benchmarked using a unit rate of C$15/t, based on comparable operations and scaled according to projected production rates. This estimate was scaled down to $13.3/t to account for the lower requirements of a toll mill operation. The general and administration operating cost estimate aligns with industry benchmarks and reflects cost assumptions used in similar operations and was adjusted for the scale and throughput of the Project.

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

The economic analysis presented in this Technical Report is based on forward-looking information and involves known and unknown risks, uncertainties, and assumptions that may cause actual results to differ materially.

Forward-looking information includes, but is not limited to:

- mineral resource estimates;
- assumed commodity prices and exchange rates;
- proposed mine plan and production schedule;
- projected recovery rates, mining dilution, and future production;
- operating, sustaining, and closure cost assumptions; and
- permitting, environmental, and social considerations.

Risks to these assumptions include potential changes in production costs, unforeseen environmental or reclamation considerations, variability in ore grade or recovery, equipment or process failures, as well as changes in permitting timelines, access, power supply, regulatory conditions, social license, interest rates, and tax regimes.

This PEA is preliminary in nature and includes 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, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

22.2 Methodology

The Project was evaluated using an annual discounted cash flow analysis and a discount rate of 5% was applied. Projected revenues from annual gold production were treated as cash inflows; cash outflows were comprised of pre-production capital expenditures, ongoing operating costs, royalties, and applicable taxes. Net annual cash flows were calculated by subtracting total outflows from inflows, and cash flows were assumed to occur at the midpoint of each operating year.

It is noted that tax estimates involve assumptions that are subject to change and can only be confirmed with certainty during Project Operations. As a result, actual after-tax financial outcomes may differ from the outcomes presented herein.

To assess the preliminary potential economic robustness of the Project, a sensitivity analysis was completed, evaluating the effect of variations in gold price, gold mill head grade, operating costs, and capital expenditures.

Capital and operating cost estimates used in this economic analysis are presented in Section 21, Capital and Operating Cost Estimates, and are stated in Q2 2025 Canadian dollars. The economic analysis model was prepared in constant (i.e., real) dollar terms and does not incorporate inflation.

22.3 Economic Model Parameters

22.3.1 Assumptions

The economic analysis was performed assuming a gold price of US$2,500/oz, which was based on consensus analyst estimates and assumptions in recent NI 43-101 technical reports. The forecasts are intended to reflect the average gold price expectations over the LOM. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis used the following key assumptions:

- Construction will occur over 12 months.
- Operations will occur over five years.
- Results were based on 100% ownership of the Project by WRLG.
- Nominal Q2 2025 Canadian dollars.
- No inflation or escalation was considered.
- Revenues were modeled to occur in the period when the material was processed.
- Costs and taxes were calculated for each period in which they occur.
- Costs from Operations incurred in the pre-production period were capitalized.
- Exclusion of sunk costs (e.g., exploration and resource definition costs, engineering fieldwork and study costs, environmental baseline study costs, construction costs); however, pre-development costs were utilized for tax deductions.
- All cash flows were discounted to the start of Construction using a mid-period discounting convention.
- Project revenue will be derived from the sale of gold doré bars.
- All gold doré will be sold in the same year gold is produced.
- No contractual arrangements currently exist for the processing of ore at an off-site toll milling facility.

22.3.2 Taxes

The Project was evaluated on an after-tax basis, but it should be noted that tax calculations can be complex and the taxation presented in the Technical Report is indicative only. A tax model was prepared by WRLG and implemented into the economic model by Fuse Advisors. The following tax rate assumptions were used in the economic analysis:

- Corporate (Combined Federal and Provincial) Tax Rate: 26.5%, which includes:
  - Ontario Provincial Tax Rate: 11.5%; and
  - Federal Tax Rate: 15%.

The taxes in the economic model were calculated at a high level to provide a general concept of the potential tax payments, which are anticipated to change as the economics of the economic model change. The total tax payments for the Project were estimated to be approximately C$34 million over the LOM at the assumed gold price.

22.3.3 Ore Refining and Ore Transportation Costs

Ore refining and ore transportation costs were included in the economic analysis based on a benchmark rate of C$1.50/oz of gold produced. These ore refining and ore transportation costs for the Project were estimated to be approximately C$0.3 million over the LOM.

22.3.4 Royalties

The Project is subject to a 2.5% NSR royalty that is held by Evolution Mining. No other royalties are applicable to the Project. This 2.5% NSR royalty was estimated to total approximately C$14.8 million over the LOM.

22.4 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax Net Present Value (NPV) discounted at 5% is C$153.8 million, the Internal Rate of Return (IRR) is 47.2%, and the payback period is 2.4 years. On a post-tax basis, the NPV discounted at 5% is C$125.3 million, the IRR is 41.9%, and the payback period is 2.4 years. For the economic analysis of the Project, the gold price was assumed at US$2,500/oz and a USD/CAD exchange rate of 1.35 was used. This gold price was determined to be appropriate as it reflects a reasonable scenario to test Project sensitivity for potential investor returns while remaining within the range of recent gold price trends. A summary of the Project economic analysis is provided in Table 22-1.

The cumulative pre-tax discounted free cash flow over the LOM totals C$189.1 million (Figure 22-1) and the cumulative post-tax discounted free cash flow over the LOM totals C$155.1 million (Figure 22-2). The final product produced by the Project will be gold doré; an annual gold production profile is shown on Figure 22-3. The potential economic cash flow projections for the Project are provided in Table 22-2.

Table 22-1: Economic Analysis Summary for the Rowan Project

Description	Unit	Life of Mine Total or Life of Mine Average
General		
Gold Price	US$/oz	2,500
Exchange Rate	USD/CAD	1.35
Life of Mine	years	5
Total Ore Mined	kt	705
Total Toll Mill Feed	kt	705
Production		
Mill Head Grade	g/t	8.01
Head Grade Contained Gold	koz	181.6
Mill Recovery Rate	%	97
Total Gold Recovered	koz	176.2
Total Average Annual Gold Production	koz	35.2
Operating Costs		
On-Site Costs	**C$/t milled**	**207.1**
Rowan Mining Cost	C$/t milled	190.0
Waste Rock and Water Management	C$/t milled	0.5
Water Treatment	C$/t milled	4.3
Crushing and Sampling	C$/t milled	12.3
Off-Site Costs	**C$/t milled**	**95.7**
Toll Milling	C$/t milled	67.4

Table 22-1: Economic Analysis Summary for the Rowan Project (continued)

Description	Unit	Life of Mine Total or Life of Mine Average
Trucking to Toll Mill	C$/t milled	15.0
General and Administrative	C$/t milled	13.3
Ore Refining and Ore Transportation	C$/t oz	1.5
Royalty	%	2.5
Cash Costs[(a)]	US$/oz Au	962.5
All-In Sustaining Cost[(b)]	US$/oz Au	1,407.8
Capital Costs		
Initial Capital	C$M	70.4
Sustaining Capital	C$M	102.6
Closure Costs	C$M	3.2
Total Capital Costs	C$M	176.2
Economic Model Results		
Pre-tax NPV @ 5%	C$M	153.8
Pre-tax IRR	%	47.2
Pre-tax Payback	years	2.4
Post-tax NPV @ 5%	C$M	125.3
Post-tax IRR	%	41.9
Post-tax Payback	years	2.4

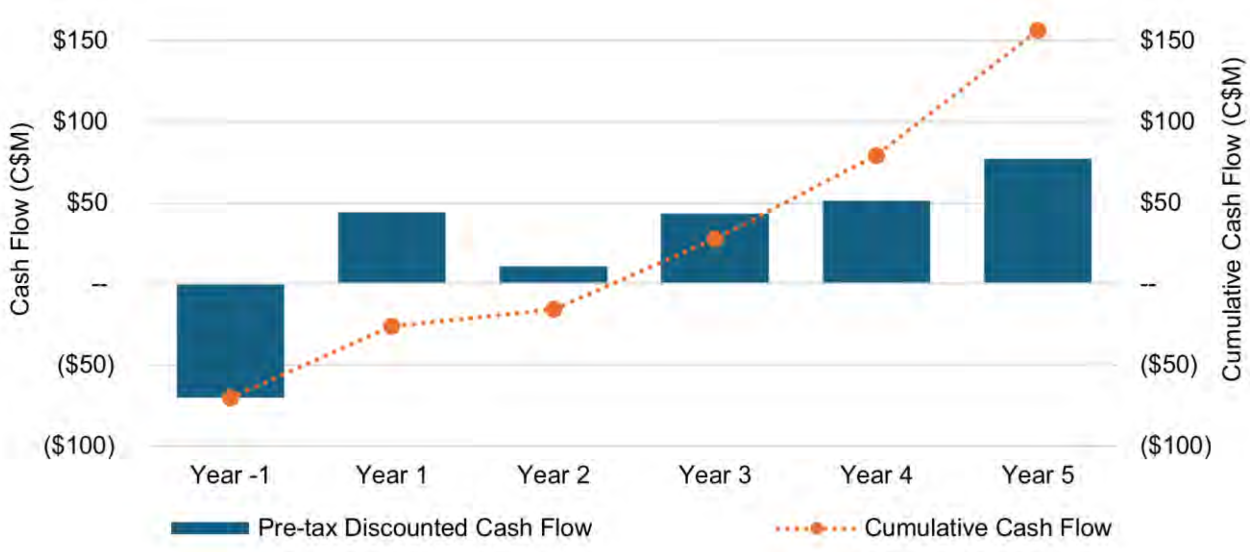
Notes:

a) Cash costs consist of mining costs, processing costs, mine-level G&A, ore refining charges and royalties.

b) All-In Sustaining Cost includes cash costs plus sustaining capital and closure cost value.

US$/oz = United States dollars per ounce; USD/CAD = United States Dollar to Canadian Dollar; kt = kilotonne; g/t = grams per tonne; koz = thousand ounces; C$/t = Canadian dollars per tonne; oz = ounce; Au = gold; C$M = million Canadian dollars; NPV = Net Present Value; IRR = Internal Rate of Return.

Figure 22-1: Pre-tax Discounted Free Cash Flow for the Rowan Project



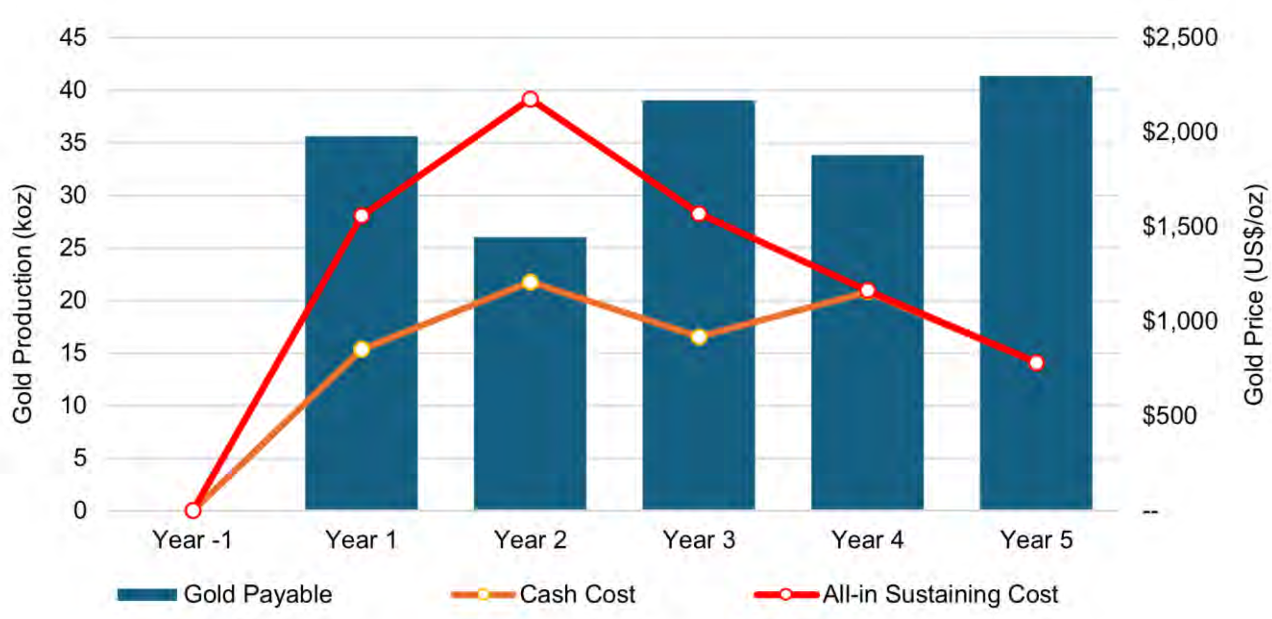
Notes:

C$M = million Canadian dollars.

Figure 22-2: Post-tax Discounted Free Cash Flow for the Rowan Project

Notes:
C$M = million Canadian dollars.

Figure 22-3: Gold Production Profile with Cash Cost and All-In Sustaining Cost for the Rowan Project



Notes:
koz = thousand ounces; US$/oz = United States dollars per ounce.

Table 22-2: Potential Cash Flow Details for the Rowan Project

Cost Centre	Units	LOM Total or LOM Average	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Macro Assumptions									
Gold Price	US$/oz	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500
Exchange Rate	USD/CAD	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35
Free Cash Flow Valuation									
Revenue	**C$M**	**$593.9**	**-**	**$120.3**	**$88.0**	**$131.9**	**$114.1**	**$139.6**	**-**
Ore Refining and Ore Transportation Costs	C$M	($0.3)	-	($0.1)	($0.0)	($0.1)	($0.1)	($0.1)	-
Net Smelter Return	**C$M**	**$593.7**	**-**	**$120.3**	**$87.9**	**$131.9**	**$114.1**	**$139.6**	**-**
Cash Operating Costs	C$M	($213.6)	-	($38.0)	($40.3)	($45.2)	($49.9)	($40.2)	-
Royalties	C$M	($14.8)	-	($3.0)	($2.2)	($3.3)	($2.9)	($3.5)	-
EBITDA	**C$M**	**$365.3**	**-**	**$79.3**	**$45.4**	**$83.4**	**$61.3**	**$95.9**	**-**
Capital Expenditures	C$M	($70.4)	($70.4)	-	-	-	-	-	-
Sustaining Capital Expenditures	C$M	($102.6)	-	($33.9)	($34.0)	($34.3)	($0.3)	-	-
Closure Capital Expenditures	C$M	($3.2)	-	-	-	-	-	-	($3.2)
Total Pre-tax Free Cash Flows	**C$M**	**$189.1**	**($70.4)**	**$45.3**	**$11.4**	**$49.1**	**$61.0**	**$95.9**	**($3.2)**
Pre-tax Discounted Free Cash Flow	**C$M**	**$153.8**	**($70.4)**	**$44.2**	**$10.6**	**$43.4**	**$51.4**	**$77.0**	**($2.4)**
Capital Expenditures	C$M	($176.2)	($70.4)	($33.9)	($34.0)	($34.3)	($0.3)	-	($3.2)
Federal Taxes	C$M	($34.0)	-	-	-	($8.5)	($11.4)	($14.1)	-
Total Post-tax Cash Flows	**C$M**	**$155.1**	**($70.4)**	**$45.3**	**$11.4**	**$40.6**	**$49.6**	**$81.7**	**($3.2)**
Post-tax Discounted Cash Flow	**C$M**	**$125.3**	**($70.4)**	**$44.2**	**$10.6**	**$35.9**	**$41.8**	**$65.6**	**($2.4)**
Production									
Rowan Project Ore Mined	kt	705	-	110	125	170	166	134	-
Rowan Project Gold Grade	g/t	8.01	-	10.41	6.67	7.38	6.52	9.94	-
Contained Gold Mined	koz	182	-	37	27	40	35	43	-
Trucking to Toll Mill									
Ore Trucked	kt	705	-	110	125	170	166	134	-
Ore Trucked Grade	g/t	8.01	-	10.41	6.67	7.38	6.52	9.94	-
Contained Gold - Trucked to Toll Mill	koz	182	-	37	27	40	35	43	-
Toll Mill Processing									
Ore Processed	kt	705	-	110	125	170	166	134	-
Head Grade	Au g/t	8.01	-	10.41	6.67	7.38	6.52	9.94	-
Head Grade Contained Gold	koz	182	-	37	27	40	35	43	-
Toll Mill Recovery	%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%

Table 22-2: Cash Flow Details for the Rowan Project (continued)

Cost Centre	Units	LOM Total or LOM Average	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Recovered Gold	koz	176	-	36	26	39	34	41	-
Gold Payability	%	99.9%	0.0%	99.9%	99.9%	99.9%	99.9%	99.9%	0.0%
Gold Payable	**koz**	**176**	**-**	**36**	**26**	**39**	**34**	**41**	**-**
Total Revenue	**C$M**	**$593.9**	**-**	**$120.3**	**$88.0**	**$131.9**	**$114.1**	**$139.6**	**-**
Operating Costs									
On-Site Costs									
Rowan Mining Cost	C$M	$134.0	-	$24.9	$25.8	$26.4	$31.5	$25.4	-
Waste Rock and Water Management	C$M	$0.4	-	$0.1	$0.1	$0.1	$0.1	$0.1	-
Water Treatment	C$M	$3.0	-	$0.6	$0.6	$0.6	$0.6	$0.6	-
Crushing and Sampling	C$M	$8.7	-	$1.4	$1.5	$2.1	$2.0	$1.6	-
Off-Site Costs									
Toll Milling	C$M	$47.6	-	$7.4	$8.5	$11.5	$11.2	$9.0	-
Trucking to Toll Mill	C$M	$10.6	-	$1.6	$1.9	$2.5	$2.5	$2.0	-
General and Administration Cost	C$M	$9.4	-	$2.0	$2.0	$2.0	$2.0	$1.5	-
Total Operating Costs	**C$M**	**$213.6**	**-**	**$38.0**	**$40.3**	**$45.2**	**$49.9**	**$40.2**	**-**
Ore Refining, Ore Transport, and Royalties									
Ore Refining and Ore Transport	C$M	$0.3	-	$0.1	$0.0	$0.1	$0.1	$0.1	-
NSR Royalty	C$M	$14.8	-	$3.0	$2.2	$3.3	$2.9	$3.5	-
Cash Costs									
Cash Cost	**US$/oz**	**$963**	**-**	**$853**	**$1,210**	**$919**	**$1,157**	**$784**	**-**
All-in Sustaining Cost (AISC)	**US$/oz**	**$1,408**	**-**	**$1,558**	**$2,176**	**$1,570**	**$1,164**	**$784**	**-**
Capital Costs (Includes Direct, Indirect, and Contingency Costs)									
Initial Capital	**C$M**	**$70.4**	**$70.4**	**-**	**-**	**-**	**-**	**-**	**-**
Mining	**C$M**	**$24.5**	**$24.5**	**-**	**-**	**-**	**-**	**-**	**-**
Infrastructure	**C$M**	**$14.2**	**$14.2**	**-**	**-**	**-**	**-**	**-**	**-**
Equipment	**C$M**	**$9.1**	**$9.1**	**-**	**-**	**-**	**-**	**-**	**-**
Crushing and Sampling	**C$M**	**$2.5**	**$2.5**	**-**	**-**	**-**	**-**	**-**	**-**
Water Treatment	**C$M**	**$8.2**	**$8.2**	**-**	**-**	**-**	**-**	**-**	**-**
Waste Rock and Water Management	**C$M**	**$7.4**	**$7.4**	**-**	**-**	**-**	**-**	**-**	**-**
Owner's Cost	**C$M**	**$4.6**	**$4.6**	**-**	**-**	**-**	**-**	**-**	**-**
Sustaining Capital Expenditures	**C$M**	**$102.6**	**-**	**$33.9**	**$34.0**	**$34.3**	**$0.3**	**-**	**-**
Mining	**C$M**	**$82.3**	**-**	**$28.2**	**$26.9**	**$27.1**	**$0.1**	**-**	**-**

Table 22-2: Cash Flow Details for the Rowan Project (continued)

Cost Centre	Units	LOM Total or LOM Average	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Infrastructure	C$M	$3.2	-	-	$1.4	$1.5	$0.3	-	-
Equipment	C$M	$17.1	-	$5.7	$5.7	$5.7	-	-	-
Total Closure Capital Expenditures	C$M	$3.2	-	-	-	-	-	-	$3.2
Total Capital Expenditures	C$M	$176.2	$70.4	$33.9	$34.0	$34.3	$0.3	-	$3.2

Notes:

Numbers may not add due to rounding.

US$/oz = United States dollars per ounce; USD/CAD = United States Dollar to Canadian Dollar; EBITDA = earnings before interest, taxes, depreciation, and amortization; kt = kilotonne; g/t = grams per tonne; koz = thousand ounces; C$/t = Canadian dollars per tonne; oz = ounce; Au = gold; C$M = million Canadian dollars; NPV = Net Present Value; IRR = Internal Rate of Return.

22.5 Sensitivity Analysis

A sensitivity analysis was completed for the Project on both the base case pre-tax NPV and IRR (i.e., C$153.8 million and 47.2%) and the base case post-tax NPV and IRR (i.e., C$125.3 million and 41.9%). This sensitivity analysis considered gold price, gold mill head grade, initial capital costs, sustaining capital costs, total capital costs, and operating costs. The results of the sensitivity analysis indicate that NPV for the Project is most sensitive to gold price and gold mill head grade. It is noted that if a gold price of US$3,250/oz was assumed instead of the US$2,500/oz used in the economic analysis, the post-tax NPV discounted at 5% would be C$238.8 million, the IRR would be 81.7%, and the payback period would be 1.4 years.

The pre-tax sensitivity analysis results for the Project are provided in Table 22-3 and shown on Figure 22-4. The post-tax sensitivity analysis results for the Project are provided in Table 22-4 and shown on Figure 22-5.

Table 22-3: Pre-tax Sensitivity Analysis Results on Net Present Value and Internal Rate of Return for the Rowan Project

PRE-TAX NPV @ 5%		Gold Price [%]				
		(30%)	(15%)	0%	+15%	+30%
Gold Mill Head Grade [%]	+30%	$108	$208	$307	$407	$506
	+15%	$54	$142	$230	$319	$407
	0%	$1	$77	$154	$230	$307
	(15%)	($53)	$12	$77	$142	$207
	(30%)	($107)	($53)	$0	$54	$108

PRE-TAX IRR @ 5%		Gold Price [%]				
		(30%)	(15%)	0%	+15%	+30%
Gold Mill Head Grade [%]	+30%	33.4%	63.2%	93.2%	123.3%	153.8%
	+15%	17.0%	43.7%	70.1%	96.6%	123.3%
	0%	0.2%	24.1%	47.2%	70.1%	93.1%
	(15%)	(18.0%)	3.9%	24.1%	43.7%	63.2%
	(30%)	(39.7%)	(18.0%)	0.2%	17.0%	33.3%

		Initial Capital [%]				
		(30%)	(15%)	0%	+15%	+30%
Sustaining Capital [%]	+30%	$146	$136	$125	$115	$104
	+15%	$161	$150	$140	$129	$118
	0%	$175	$164	$154	$143	$133
	(15%)	$189	$179	$168	$158	$147
	(30%)	$204	$193	$182	$172	$161

		Initial Capital [%]				
		(30%)	(15%)	0%	+15%	+30%
Sustaining Capital [%]	+30%	54.8%	44.1%	36.1%	29.9%	24.9%
	+15%	62.7%	50.5%	41.5%	34.6%	29.0%
	0%	71.1%	57.2%	47.2%	39.5%	33.3%
	(15%)	79.8%	64.2%	53.0%	44.5%	37.8%
	(30%)	88.9%	71.5%	59.1%	49.8%	42.4%

		Operating Costs [%]				
		(30%)	(15%)	0%	+15%	+30%
Total Capital Costs [%]	+30%	$210	$181	$153	$125	$96
	+15%	$210	$182	$153	$125	$97
	0%	$210	$182	$154	$125	$97
	(15%)	$211	$183	$154	$126	$98
	(30%)	$211	$183	$155	$126	$98

		Operating Costs [%]				
		(30%)	(15%)	0%	+15%	+30%
Total Capital Costs [%]	+30%	64.0%	55.6%	47.1%	38.6%	30.0%
	+15%	64.0%	55.6%	47.1%	38.6%	30.1%
	0%	64.0%	55.6%	47.2%	38.7%	30.1%
	(15%)	64.0%	55.6%	47.2%	38.7%	30.1%
	(30%)	64.1%	55.7%	47.2%	38.7%	30.2%

Notes:
NPV = Net Present Value; IRR = Internal Rate of Return; % = percent.

Table 22-4: Post-tax Sensitivity Analysis Results on Net Present Value and Internal Rate of Return for the Rowan Project

POST-TAX NPV @ 5%						
		Gold Price [%]				
		(30%)	(15%)	0%	+15%	+30%
Gold Mill Head Grade [%]	+30%	$91	$165	$239	$309	$377
	+15%	$54	$117	$182	$247	$309
	0%	$1	$72	$125	$182	$239
	(15%)	($53)	$12	$72	$117	$165
	(30%)	($107)	($53)	$0	$54	$91

POST-TAX IRR [%]						
		Gold Price [%]				
		(30%)	(15%)	0%	+15%	+30%
Gold Mill Head Grade [%]	+30%	30.2%	56.0%	81.7%	105.4%	126.9%
	+15%	17.0%	38.9%	62.0%	84.7%	105.4%
	0%	0.2%	23.0%	41.9%	62.0%	81.7%
	(15%)	(18.0%)	3.9%	23.0%	38.9%	56.0%
	(30%)	(39.7%)	(18.0%)	0.2%	17.0%	30.1%

		Initial Capital [%]				
		(30%)	(15%)	0%	+15%	+30%
Sustaining Capital [%]	+30%	$120	$112	$104	$96	$87
	+15%	$131	$123	$115	$106	$98
	0%	$142	$133	$125	$117	$109
	(15%)	$152	$144	$136	$128	$120
	(30%)	$163	$155	$147	$139	$131

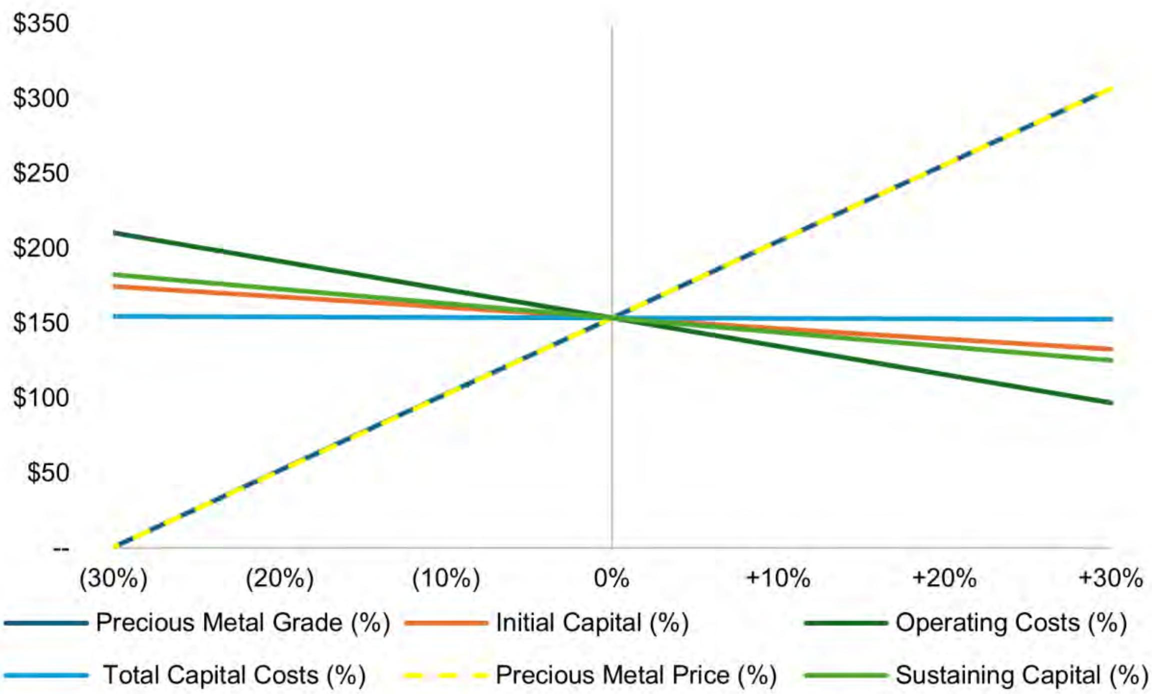
		Initial Capital [%]				
		(30%)	(15%)	0%	+15%	+30%
Sustaining Capital [%]	+30%	48.7%	39.1%	32.0%	26.5%	22.1%
	+15%	56.1%	44.9%	36.8%	30.6%	25.7%
	0%	64.0%	51.1%	41.9%	34.9%	29.4%
	(15%)	72.3%	57.6%	47.3%	39.5%	33.4%
	(30%)	81.1%	64.5%	52.9%	44.3%	37.5%

		Operating Costs [%]				
		(30%)	(15%)	0%	+15%	+30%
Total Capital Costs [%]	+30%	$116	$95	$74	$53	$33
	+15%	$142	$121	$100	$79	$59
	0%	$167	$146	$125	$104	$84
	(15%)	$193	$172	$151	$130	$110
	(30%)	$218	$197	$176	$155	$135

		Operating Costs [%]				
		(30%)	(15%)	0%	+15%	+30%
Total Capital Costs [%]	+30%	30.5%	24.9%	19.3%	13.8%	8.5%
	+15%	41.9%	35.5%	29.2%	22.9%	16.9%
	0%	56.7%	49.2%	41.9%	34.6%	27.6%
	(15%)	76.9%	67.9%	59.1%	50.4%	41.9%
	(30%)	106.2%	95.1%	84.1%	73.2%	62.5%

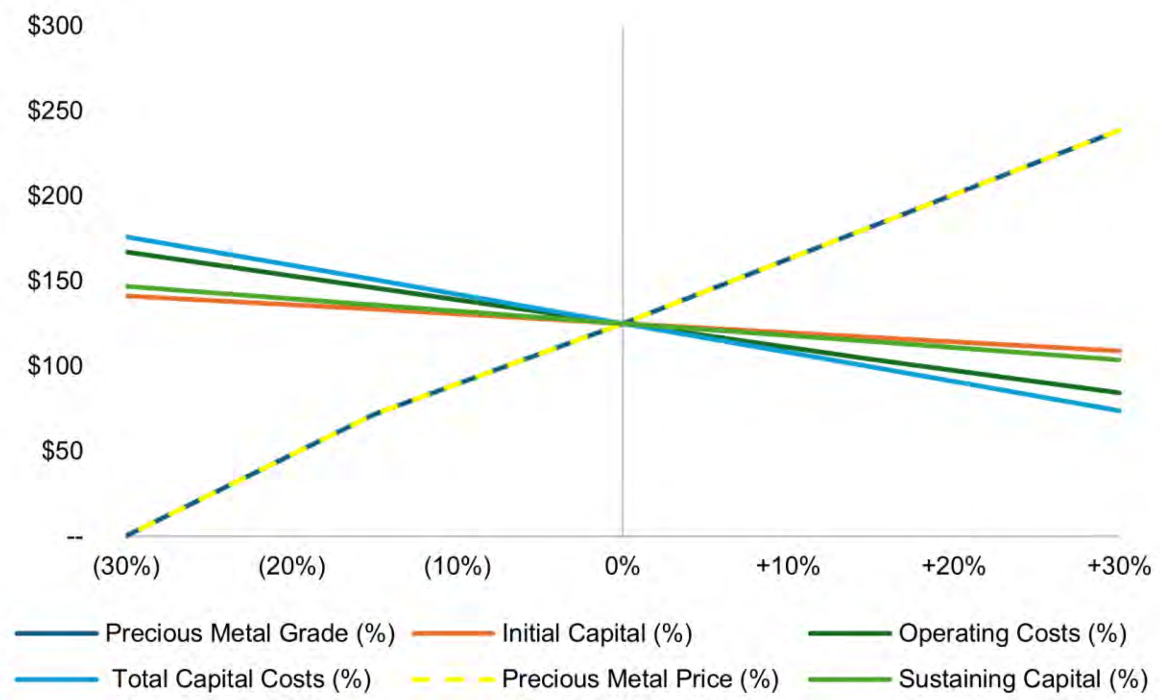
Notes:
NPV = Net Present Value; IRR = Internal Rate of Return; % = percent.

Figure 22-4: Pre-tax Sensitivity Net Present Value Results for the Rowan Project



Notes:
% = percent.

Figure 22-5: Post-tax Sensitivity Net Present Value Results for the Rowan Project



Notes:
% = percent.

22.6 Economic Analysis Summary

The economic analysis was performed assuming a 5% discount rate. The pre-tax NPV discounted at 5% is C$154 million, the IRR is 47.2%, and payback period is 2.4 years. On a post-tax basis, the NPV discounted at 5% is C$125 million, the IRR is 41.9%, and the payback period is 2.4 years. The Project's post-tax breakeven gold price is US$1750/oz and post-tax breakeven gold grade is 5.6 g/t; these breakeven values represent the point at which the Project would be considered uneconomic (i.e., zero discounted profit).

23 ADJACENT PROPERTIES

The QP has not independently verified the information in this section, and this information is not necessarily indicative of the mineralization at the Rowan Property.

23.1 Newman Todd Property Owned by Renegade Gold

The Newman Todd property is located to the south and adjoining the Rowan Property and is 100% owned by Renegade Gold Inc., formerly known as Trillium Gold Mines Inc.. The Newman Todd structure is interpreted to be the southwest extension of the NT Zone, which crosses from the Rowan Property boundary and trends southwest for a distance 2.2 km over the Newman Todd property.

Renegade Gold Inc. began work on Newman Todd property in 2011 and has completed 20,180 m of drilling as of 31 March 2022. In 2003, Redstar Gold began exploring the Newman Todd property with mapping, prospecting and geophysics, which was followed by exploration diamond drilling from 2005 to 2010. As this time, Trillium Gold Mines Inc. optioned the Newman Todd property from Redstar Gold and continued exploration work up to the present time in 2022. Before 2003, the Newman Todd property was sporadically explored by numerous companies since the late 1920s.

Results disclosed from Newman Todd structure drilling indicate gold values within a quartz breccia unit along the contact of a quartz-diorite/quartz porphyry intrusive.

The original Newman Todd property consisting of 13 patented claims covers an area of approximately 198 ha where the Newman Todd structure is situated. In 2020, Trillium Gold Mines Inc. acquired six additional patented claims located adjacent and to the west of the original 13 patented claims; these six claims are known as the Rivard property and have an area of 90 ha, which give a total area of 288 ha for the Newman Todd property (SR 2025).

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information for the PEA.

25 INTERPRETATION AND CONCLUSIONS

The following interpretations and conclusions have been provided by the QPs in their respective areas of expertise, based on the review of data available for this Technical Report.

25.1 Metallurgical Testwork

The metallurgical testwork program for the Rowan deposit was completed on four master composite samples and included chemical and mineralogical characterization, comminution testing, extended gravity recoverable gold testing, gravity leach testing, and cyanide destruction testing.

The ore from the Rowan deposit is characterized by high gravity-recoverable gold, low sulphur content, and no deleterious elements that would preclude successful gold extraction from the material. Gravity recoveries ranged from 28% to 81%, and overall gold extraction ranged from 94.3% to 99.5% after 24 hours. The leaching process was fast, with the majority of gold extracted (i.e., up to 99.3%) within the first 6 hours. High gold recoveries were also maintained at a coarser grind size of 125 microns (µm), indicating the ore is not highly sensitive to grind size within the range tested (i.e., 75 µm to 125 µm). Comminution testing confirmed that the material is hard, with a Bond Work Index (BWi) ranging from 16.2 kWh/t to 18.2 kWh/t.

A design gold recovery of 97% was selected for the Project to account for minor expected losses in a commercial operating environment. The metallurgical testwork results support the use of a conventional gravity-leach-CIP flowsheet for processing of ore from the Project.

25.2 Mineral Resource Estimate

Exploration work conducted on the Rowan Property to date has led to a focus on the historical Rowan Mine area and on the NT Zone where several gold zones exhibited characteristics such as prospective geology, structural setting, and anomalous geochemistry that appear to merit additional work.

A significant occurrence of gold mineralization delineated by diamond drilling at the Rowan Property throughout the long history of exploration and underground production includes:

- A 1.8 km portion of the strike length of the east-west trending Pipestone Bay-St Paul Deformation Zone (PBDZ) in the Rowan Mine area that contains several gold zones, which have been drilled to a depth of approximately 450 metres below ground surface (mbgs).
- A 1 km portion of the northeast trending NT Zone that contains several gold zones, which have been drilled to a depth of approximately 200 mbgs.

Interpretations and conclusions related to the mineral resource estimate include:

- Based on the drilling completed in 2023, sufficient data and geological information was collected to support a reinterpretation of the geology and controls on mineralization at the Rowan deposit.
- The QP reviewed the sample preparation, analysis, and security procedures at the Rowan Property and considers these procedures to be adequate for use in the estimation of mineral resources.
- The QA/QC program for the Rowan deposit as designed is adequate and the drilling information database is suitable for use in a mineral resource estimate.

- The QP has reviewed the drill hole database data adjustments and verification checks completed by SRK and is of the opinion that the drilling information database is adequate for use in the 2025 mineral resource estimate.
- At present, 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.

The 2025 mineral resource estimate for the Rowan Project is summarized in Table 25-1.

Table 25-1: 2025 Mineral Resource Estimate Summary for the Rowan Deposit as of 30 June 2025

Category	Tonnage [t]	Average Grade [g/t Au]	Contained Metal [oz Au)
Indicated	478,707	12.78	196,747
Inferred	421,181	8.73	118,155

Notes:
- CIM (2014) definitions were followed for mineral resources.
- Mineral resources were estimated at a gold cut-off grade of 3.80 g/t using a long-term gold price of US$1,800 per ounce.
- There are no mineral reserves currently estimated at the Rowan deposit.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Mineral resources are reported within vein wireframes at the stated cut-off grade of 3.80 g/t Au.
- Density of 2.8 g/cm^3.
- Numbers may not add due to rounding.

t = tonne; g/t = grams per tonne; Au = gold; oz = ounce; g/cm^3 = grams per cubic centimetre.

25.3 Mining Methods

The proposed underground mine plan was determined to be suitable based on the geotechnical conditions that are present at the Rowan deposit. Underground stope sizes were adequate given the rock mass conditions and the shape factors proposed for the mine. Ground support in drift developments was determined to be a standard pattern bolt and screen, with rock bolt length equivalent to half the span width.

The proposed underground mine plan for the Project shows the potential for a small tonnage, high-grade open stoping mine that will produce approximately 705 kt of ore at an average gold grade of 8.01 g/t that contains approximately 182 koz of gold. Costs were estimated using conservative costs and production rates and will be further refined in future engineering studies.

25.4 Recovery Methods

Ore from the Rowan deposit is amenable to processing by a conventional gravity-leach-CIP flowsheet, which is demonstrated by the metallurgical testwork program completed for the Project. The selected off-site toll milling facility has the ability to process ore from the Rowan deposit without significant modifications and capital expenditure; however, this approach would result in the reduction of overall gold recovery compared to the recovery results demonstrated in the metallurgical testwork program.

A series of targeted upgrades is proposed at the selected off-site toll milling facility to achieve the gold recoveries indicated in the metallurgical testwork at the maximum additional ore throughput of 475 t/d. These upgrades would include installation of a new mobile secondary crusher (i.e., pre-crusher), expansion of the gravity gold recovery circuit, and additional leach and CIP tanks to support a longer residence time. The proposed modifications are modest in scope, integrate with the existing off-site toll milling facility plant layout, and are consistent with the recovery assumptions.

No significant upgrades would be required to the grinding circuit, gold recovery and refining circuits, power supply, water supply and water treatment, or reagent handling systems at off-site toll milling facility to support the Project. The off-site toll milling facility currently includes a tailings management facility that would have sufficient capacity to accommodate the additional tailings generated from ore from the Rowan deposit. However, the next planned dam raise of the existing tailings management facility would need to be advanced by approximately one year. This tailings management facility construction schedule adjustment was incorporated into the Project's schedule and capital cost estimate.

A Crusher and Sampler Tower will be installed at the Project site to prepare ore for processing at the selected off-site toll milling facility and confirm appropriate metallurgical accounting of the ore feed from the Rowan deposit. This equipment is designed to operate 12 hours per day and 365 days per year on day shift only and is sized to support the maximum off-site toll mill throughput of 475 t/d.

25.5 Project Infrastructure

25.5.1 Surface Infrastructure

Project surface infrastructure includes ore storage facilities, ore crushing and sampling facilities, water rock and water management facilities, and water treatment. There are no processing or tailings management facilities located at the Project site.

Ancillary infrastructure at the Project site will include mine dry and administration facilities, maintenance and warehouse facilities, fuel storage and power generation, exploration facilities, housing for personnel, and site roads.

Infrastructure will also include security fencing and gates as required to control access to the Project site.

Further engineering studies will refine the designs and costing of this surface infrastructure.

25.5.2 Waste Rock and Water Management

The design of the WRS and the associated water management infrastructure was developed based on the current best understanding of the Project site and is deemed appropriate for this level of design. Geotechnical design considerations were conservatively assumed, where applicable. The high-level WBM was completed to support Project design in consideration of the estimated underground dewatering inflow volumes.

25.6 Environmental Studies, Permitting, and Social or Community Impact

WRLG is continuing scientific and engineering studies at the Project site; First Nation, community, and regulatory consultations; monitoring programs; and design planning to progress the Project. WRLG has focused its efforts since acquisition on reducing the uncertainty and risk associated with any new mining development and is actively designing operations to minimize water use, improve water quality, and bring overall benefit to First Nations and local communities.

The Project will require a Class EA because it is considered a Resource Stewardship and Facility Development Project in accordance with the Ministry of Natural Resources (MNR 2003). There is no requirement for a federal

EA nor a provincial individual EA. WRLG will engage with all appropriate regulatory authorities to complete the appropriate submissions of any additional federal, provincial, and municipal permits and approvals that will be required for development of the Project.

WRLG has committed to engagement and consultation with local First Nations; municipal, provincial, and federal governments; other stakeholders; and the public throughout all stages of the Project. The intent of this engagement and consultation is to provide all interested parties with opportunities to learn about WRLG, identify concerns, and provide input with the goal of positively enhancing Project planning and development. WRLG recognizes the importance of timely, full, and open discussion of concerns and options associated with the Project and the related concerns those individuals or communities may have regarding Project activities. WRLG will maintain open and honest communications with local communities and individual stakeholders throughout all stages of the Project. WRLG will establish operational practices, both now and into the future, that reflect the values, expectations, and needs of the community in which WRLG operates, and that are based on continued mutually respectful consultation with all First Nations and other stakeholders.

25.7 Markets and Contracts

The Project will produce gold in the form of doré bars with 99.9% gold payable. WRLG and its consultants have conducted no market studies on the sale of gold doré. Existing terms from previous sales by WRLG were used as the basis for the economic modelling.

For the economic analysis of the Project, the gold price was assumed at US$2,500/oz and a USD/CAD exchange rate of 1.35 was used. Ore refining and ore transportation costs total to C$1.50/oz of gold produced. No existing refining agreements or sales contracts are currently in place for the Project.

The QP is of the opinion that the marketing and commodity price information is suitable to be used in the cash flow analysis to support the Technical Report.

25.8 Capital Cost Estimate

The preliminary economic evaluation of the Rowan Project was based on the capital and operating cost estimates outlined in the Technical Report. These cost estimates reflect an underground mining operation, and account for the development of both underground and surface infrastructure, including underground mine development and production, ore storage facilities, crushing and sampling facilities, waste rock management facilities, water management facilities, WTP, and ancillary infrastructure, as well as Owner's costs and appropriate provisions.

The capital cost estimate conforms to a PEA-level estimate with a ±30% accuracy range which is within the boundaries of the Association for the Advancement of Cost Engineering International Class 5 guidelines (AACE International 2012). The capital cost estimate was developed in Q2 2025 using first-principles engineering combined with benchmarking against similar underground mining operations in Ontario. Conceptual engineering to support the capital cost estimate was completed by Fuse Advisors Inc., Knight Piésold Ltd., and Integrated Sustainability Consultants Ltd. Major mechanical equipment and infrastructure costs were derived from vendor quotations, and auxiliary components and indirect costs were based on industry-standard benchmarks.

The initial capital cost for the Project was estimated to be C$70.4 million, which includes C$13.2 million in contingency. The LOM sustaining capital was estimated to be C$102.6 million, which includes of C$23.7 million in contingency. The reclamation and closure cost was estimated to be C$3.2 million, which excludes contingency and taxes.

The capital cost estimate forms the foundation for further technical and financial evaluations in future study phases.

25.9 Operating Cost Estimate

The operating cost estimate for the Project was developed in Q2 2025 and is considered to be accurate within a range of ±30% which is within the boundaries of the Association for the Advancement of Cost Engineering International Class 5 guidelines (AACE International 2012) and appropriate for a PEA.

Mine operating costs were developed from first principles and applied directly to the Project mine production schedule. Processing (i.e., crushing and sampling) costs were determined using vendor quotes and benchmarked values from similar crushing and material handling operations. General and administrative costs were estimated based on benchmarks from comparable projects of similar scale.

All operating costs are presented in Canadian dollars, with a USD/CAD exchange rate assumption of 1.35 where applicable. No escalation for inflation or currency fluctuation was applied unless otherwise noted.

The operating cost estimate includes both on-site and off-site costs:

- On-site costs: underground mining, waste rock management, water management, water treatment, and crushing and sampling expenses.
- Off-site costs: toll milling, ore transportation to an off-site toll milling facility, and general and administrative expenses.

The total operating costs over the LOM were estimated to be C$213.6 million, with an average annual operating cost of C$42.7 million. The total operating unit cost for the LOM was estimated to be C$302.8/t processed, which includes:

- on-site operating cost: C$207.1/t processed; and
- off-site operating cost of C$95.7/t processed.

25.10 Economic Analysis

The economic analysis for the Project was completed assuming 5% discount rate, a gold price of US$2,500/oz, and a USD/CAD exchange rate of 1.35. This gold price was determined to be appropriate as it reflects a reasonable scenario to test Project sensitivity for potential investor returns while remaining within the range of recent gold price trends.

The economic analysis used the following key assumptions:

- Construction will occur over 12 months.
- Operations will occur over five years.
- Results were based on 100% ownership of the Project by WRLG.
- All values are expressed in Q2 2025 Canadian dollars.

- No inflation or escalation was considered.
- Revenues were modeled to occur in the period when the material was processed.
- Costs and taxes were calculated for each period in which they occur.
- Costs from Operations incurred in the pre-production period were capitalized.
- Exclusion of sunk costs (e.g., exploration and resource definition costs, engineering fieldwork and study costs, environmental baseline study costs, construction costs); however, pre-development costs were utilized for tax deductions.
- All cash flows were discounted to the start of Construction using a mid-period discounting convention.
- Project revenue will be derived from the sale of gold doré bars.
- All gold doré will be sold in the same year gold is produced.
- No contractual arrangements currently exist for the processing of ore at an off-site toll milling facility.

The pre-tax NPV discounted at 5% is C$153.8 million, the IRR is 47.2%, and the payback period is 2.4 years. The post-tax NPV discounted at 5% is C$125.3 million, the IRR is 41.9%, and the payback period is 2.4 years. Cumulative post-tax unlevered free cash flow was estimated to total C$155 million.

The sensitivity analysis results indicate that NPV for the Project is most sensitive to gold price and gold mill head grade.

This PEA is preliminary in nature and includes 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, and there is no certainty that the PEA will be realized. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

This Technical Report was based on the 2025 mineral resource estimate for the Project, which is comprised of 63% indicated resources and 37% inferred resources by tonnes.

A summary of the Project economic analysis is provided in Table 25-2 and the cumulative post-tax discounted free cash flow for the Project is shown on Figure 25-1.

Table 25-2: Economic Analysis Summary for the Rowan Project

Description	Unit	Life of Mine Total or Life of Mine Average
General		
Gold Price	US$/oz	2,500
Exchange Rate	USD/CAD	1.35
Life of Mine	years	5
Total Ore Mined	kt	705
Total Toll Mill Feed	kt	705
Production		
Mill Head Grade	g/t	8.01
Head Grade Contained Gold	koz	181.6
Mill Recovery Rate	%	97
Total Gold Recovered	koz	176.2
Total Average Annual Gold Production	koz	35.2

Table 25-2: Economic Analysis Summary for the Rowan Project (continued)

Description	Unit	Life of Mine Total or Life of Mine Average
Operating Costs		
On-Site Costs	**C$/t milled**	**207.1**
Rowan Mining Cost	C$/t milled	190.0
Waste Rock and Water Management	C$/t milled	0.5
Water Treatment	C$/t milled	4.3
Crushing and Sampling	C$/t milled	12.3
Off-Site Costs	**C$/t milled**	**95.7**
Toll Milling	C$/t milled	67.4
Trucking to Toll Mill	C$/t milled	15.0
General and Administrative	C$/t milled	13.3
Ore Refining and Ore Transportation	C$/t oz	1.5
Royalty	%	2.5
Cash Costs[(a)]	US$/oz Au	962.5
All-In Sustaining Cost[(b)]	US$/oz Au	1,407.8
Capital Costs		
Initial Capital	C$M	70.4
Sustaining Capital	C$M	102.6
Closure Costs	C$M	3.2
Total Capital Costs	C$M	176.2
Economic Model Results		
Pre-tax NPV @ 5%	C$M	153.8
Pre-tax IRR	%	47.2
Pre-tax Payback	years	2.4
Post-tax NPV @ 5%	C$M	125.3
Post-tax IRR	%	41.9
Post-tax Payback	years	2.4

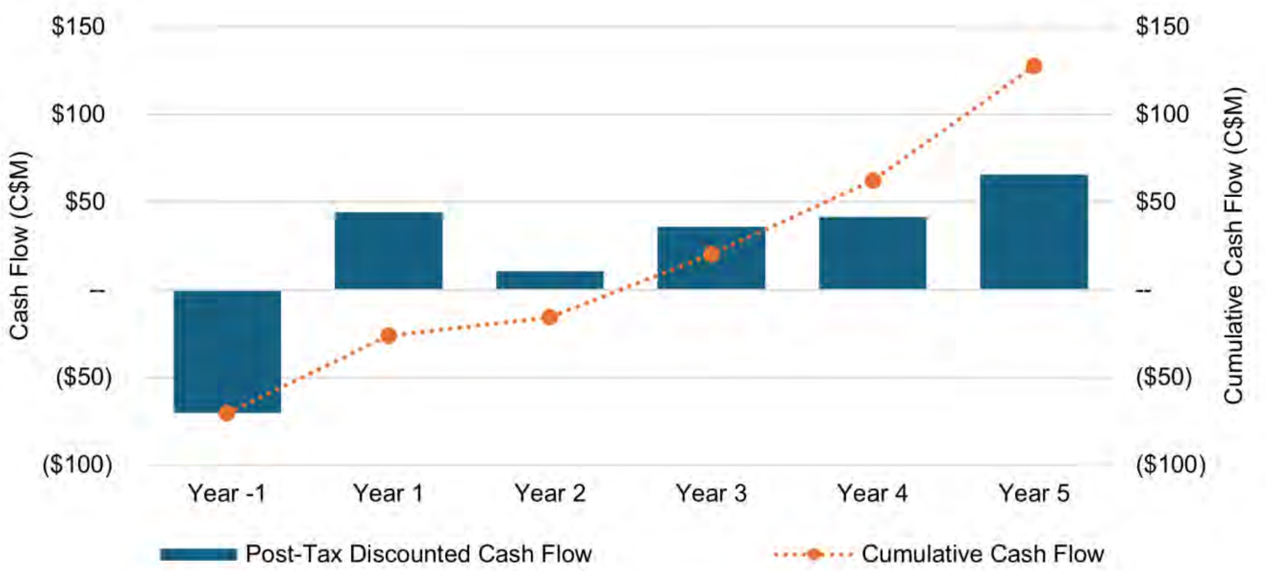
Notes:

a) Cash costs consist of mining costs, processing costs, mine-level G&A, ore refining charges and royalties.

b) All-In Sustaining Cost includes cash costs plus sustaining capital and closure cost value.

US$/oz = United States dollars per ounce; USD/CAD = United States Dollar to Canadian Dollar; kt = kilotonne; g/t = grams per tonne; koz = thousand ounces; C$/t = Canadian dollars per tonne; oz = ounce; Au = gold; C$M = million Canadian dollars; NPV = Net Present Value; IRR = Internal Rate of Return.

Figure 25-1: Post-tax Discounted Free Cash Flow for the Rowan Project



Notes:
C$M = million Canadian dollars.

25.11 Risks and Opportunities

The following summary of risks and opportunities associated with the Project involves forward-looking statements that are based on reasonable expectations and informed by the recent past. Readers are cautioned that such forward-looking statements involve uncertainties and unknowns that may cause actual outcomes to differ from those implied by these forward-looking statements. This Technical Report is preliminary in nature and further work is required to mitigate the following risks.

25.11.1 Risks

25.11.1.1 Geology and Mineral Resource

The Rowan deposit is geologically complex, and it is possible that additional exploration data may result in changes to the understanding of deposit geology. As the current mineral resource includes inferred mineral resources that are estimated on the basis of limited geological evidence and sampling, there is no assurance that further exploration would upgrade these resources to indicated or measured resource categories.

Mineral resources are not mineral reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of the 2025 mineral resource estimate will be converted into mineral reserves. Inferred mineral resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as mineral reserves.

Mineral resource estimates may be materially affected by the data quality, natural geological or mineralogical variability, metallurgical recovery variability, as well as accuracy of the economic assumptions, including gold price, mining costs, and processing costs.

25.11.1.2 Metallurgical Testwork and Recovery Methods

The metallurgical testwork completed for the Project may not fully represent the orebody's spatial and mineralogical variability and could potentially lead to lower-than-expected gold recoveries in a full-scale processing operation.

The proposed modifications to the process flow sheet of the off-site toll milling facility may change depending on additional testwork completed on ore from the Rowan deposit.

25.11.1.3 Mining and Underground Geotechnical Considerations

The feasibility of the underground mine for the Project could be negatively affected by significant changes in key factors, including ore grade, rock quality, and groundwater inflow rates. The assumptions made in this Technical Report relating to the underground mine and these factors were considered conservative to mitigate these possibilities.

Potential underground geotechnical risks for the Project include an underestimation of the effect of foliation on the stope stability and underground development. No in-situ stress measurements were performed to support this Technical Report, and the effects of rock bursting have not been assessed. The effects of the hydrogeology regime on the performance of stopes and underground development were not assessed and could negatively affect ore recovery. Poor ground conditions and damaged zones have not been created in a 3D wireframe to date, and the presence of such zones could negatively affect ore recovery.

25.11.1.4 Project Infrastructure

The ground conditions and stability of the Portal Pad, water management pond berms, and other infrastructure areas are unknown as a geotechnical program has not been completed to date.

The slopes and heights of the ore storage facilities and WRS may change as slope stability analysis was not completed at the time of this Technical Report since there is currently no relevant geotechnical information.

The water management pond storage capacity, WTP treatment rates, and WTP design may change as additional site-specific information is collected, including meteorological and hydrology data, contact water chemistry data, and expected groundwater inflows.

25.11.1.5 Market Studies

The marketing terms considered in the Technical Report were based on terms observed in comparable gold projects in Ontario, as well as existing terms from previous sales by WRLG. No marketing studies have been completed, and no discussions have been held with any marketing companies in determining the marketing terms to date.

25.11.1.6 Cost Estimates

There are no contracts established with any equipment suppliers, power suppliers, fuel suppliers, or marketing companies at this time. Vendor quotes were received for underground mobile equipment, surface mobile equipment, and major processing equipment. However, the actual prices of this equipment may vary at the time of Project construction and execution.

Challenges related to workforce availability, infrastructure procurement, and operational efficiency could affect Project costs and timelines.

25.11.1.7 Economic Analysis

The economic analysis has not considered the risk of the Project to gold price fluctuation, market demand, market competition, inflation, or other unexpected events that could significantly affect the economics or overall schedule of the Project.

25.11.2 Opportunities

25.11.2.1 Mineral Resource

Mineral resources may be increased by investigating gold mineralization located on the periphery of the current geological model for the Project.

25.11.2.2 Mining and Underground Geotechnical Considerations

Underground mining and geotechnical opportunities are that the empirical dilution estimates may overestimate the external dilution as the moderate depth and competent rock mass of the Rowan deposit may provide a more stable stope than considered in the geotechnical analysis and the underground mine design.

25.11.2.3 Metallurgical Testwork and Recovery Methods

Detailed metallurgical testwork could improve the process flow sheet, increase processing efficiency, and increase gold recovery, which may reduce processing costs.

25.11.2.4 Project Infrastructure

There is an opportunity to house personnel for the Project in Red Lake, Ontario and provide daily transportation to the Project site; this approach would remove the requirement of a full Personnel Camp on site.

26 RECOMMENDATIONS

26.1 Drilling and Mineral Resource Estimate

The following drilling and mineral resource related activities are recommended for the Rowan Property:

- Drilling should be focused on mineralized shoot geometries and inferred resource areas where there are isolated gold intercepts at or above the gold cut-off grade of 3.80 g/t used in the 2025 mineral resource estimate. Also, closer-spaced drilling within the mineralized shoots could provide sufficient data to develop continuity for variographic analysis.
- The composite gold grades for the mineral resource estimate are capped at the appropriate level for this Technical Report; however, high-grade transitions should be explored by domain in the next mineral resources estimate update.
- Additional drilling should be completed to expand the Rowan deposit at depth, including both down dip and down plunge.
- The current drilling information is captured and stored using the Geotic software system. Typically, this drilling data is then transferred to the Leapfrog software system. A central database should be established in addition to the two systems currently being used.
- Bulk density determinations should be routinely completed in mineralization and waste areas in any future drilling.
- Future metallurgical test work at the Rowan deposit should include additional drilling and sample selection based on the updated vein model.

26.2 Mineral Processing and Metallurgical Testwork

The metallurgical assessment has been completed to a PEA level. Further metallurgical testwork recommendations to bring the Project to a PFS level would include:

- Variability testing: Following optimization of processing parameters, variability testing is required for confirmation of the gold recoveries achieved during the 2023 metallurgical testwork program (Base Met Labs 2024). This testing would be completed for the various veins, spatial areas, and mineralogy of the Rowan deposit. The samples should cover all expected grade ranges, including down to the proposed cut-off grade and possibly below this grade, to establish the potential future use of low-grade stockpiles. This testwork would include full circuit simulation studies to cover the anticipated mine schedule, particularly in the initial years of production.
- Toll mill flowsheet testing: Once the off-site toll mill facility is identified, the toll mill flowsheet and processing parameters should be tested to confirm gold recovery from the Rowan deposit and to confirm if modifications to the toll mill flowsheet would be required.
- Blended feed testing: Once the off-site toll mill facility is identified, the toll mill flowsheet and processing parameters should be tested on a blend of Rowan deposit and toll-mill ores to confirm there are no negative effects on gold recovery.

26.3 Geotechnical Studies to Support Mine Engineering

The following geotechnical studies and activities are recommended in subsequent study phases:

- Detailed structural logging or televiewer logging to determine the orientation of geological structures in the area of the underground mine.
- Detailed study of the geotechnical domains that are not general to lithologies but consider the spatial distribution of rock masses.
- Determine if there are any 3D structural or lithological connections between the weak and/or fractured zones.
- Rock strength laboratory testing to determine rock parameters for major lithologies.
- Point load testing on 3 m intervals that extend 25 m above and below the proposed underground development.
- Investigate the presence and impact of foliations on the rock mass quality.
- Quality assurance/quality control (QA/QC) of geotechnical core logging to date to verify the geotechnical information is being correctly and accurately captured.
- Additional drilling into the footwall of the mineral deposit for underground mine infrastructure.
- Ongoing geotechnical logging and training of field staff.
- Detailed ground support designs that account for local structural and ground conditions for all proposed underground infrastructure.

26.4 Underground Mine Engineering

The following mine engineering and design work is recommended in subsequent study phases:

- Refine the mining method selection and stope design parameters based on the results of a more advanced geotechnical investigation.
- Complete a dilution and mining loss study to determine the optimal stope height and to support more advance mine planning activities.
- Complete a hydrogeological assessment to confirm groundwater inflows to the underground workings and to confirm the mine dewatering infrastructure requirements.
- Complete additional drilling to assess the potential of including additional mining zones that are in close proximity to the PEA mine design.
- Complete a more detailed analysis of the ventilation air flow, velocity, vent raise dimensions, and fan power requirements for the Project.
- Complete further engineering work for underground infrastructure to improve confidence in capital costs.
- Advance the mine plan to a PFS-level of study with the incorporation of an updated resource block model, and updated geotechnical, metallurgical, and hydrogeological parameters, as well as updated commercial terms, environmental considerations, and capital and operating costs.

26.5 Recovery Methods

It is recommended that the proposed processing modifications to the off-site toll milling facility are further evaluated and refined through additional engineering and economic analysis in subsequent study phases. Specific recovery method recommendations:

- Advance conversations with the existing off-site toll milling facilities in the Red Lake area and conduct further analyses to determine which facility would be most likely to offer the best toll milling terms, require the least amount of processing modifications, and achieve the targeted gold recoveries from Rowan deposit ore.
- Confirm the design and sizing of the new mobile secondary crusher (i.e., pre-crusher) to verify consistent mill feed.
- Complete detailed engineering for the gravity circuit expansion as well as the additional leach and CIP tanks to validate capital cost estimates and installation timelines.
- Monitor tailings deposition rates at the selected off-site toll milling facility to confirm tailings management facility dam raise timing.

26.6 Waste Rock and Water Management

Recommendations for the next phase of engineering for the waste rock and water management facilities are summarized below:

- Foundation investigation and subsequent design of the water management pond berms and the Portal Pad.
- Foundation investigation within the water management pond basin footprints to inform the necessity of the liners for the seepage underdrain system.
- Suitability assessment of waste rock from the underground for use as construction material or identify local appropriate construction material borrow sources.
- Collection of site-specific meteorological and hydrology data to refine seasonal runoff values and event design storms.
- Refinement of underground dewatering rates and water treatment release conditions to confirm the required storage capacity for water management ponds, WTP design, and WBM results.
- Develop a full closure plan for the Project based on the final design configuration.

26.7 Environmental Studies and Permitting

Studies and activities are recommended to support the Class EA submission as well as the additional permit and approval submissions required for development of the Project. These studies and activities will support the assessment of potential Project effects on the receiving environment and will include:

- Continue engagement and consultation with local First Nations; municipal, provincial, and federal governments; other stakeholders; and the public.
- Continue the characterization of the physical environment (e.g., air quality, surface water, groundwater).
- Continue the geochemical characterization of waste rock (e.g., static testing, kinetic testing).

- Continue the baseline studies for the biological environment (e.g., aquatic plant and animal communities, terrestrial plant and animal communities, species at risk).
- Complete an archaeological assessment to identify any artifacts or archaeological values in the area of the Project.
- Assess the potential effects of the Project on the physical, biological, and social environments.
- Continue to progress the submission of the Class EA as well as submissions of additional federal, provincial, and municipal permits and approvals required for development of the Project.

26.8 Cost Estimate for Recommended Work

It is recommended to continue developing the Project through subsequent study phases; the proposed estimated cost to advance the Project through the PFS stage is summarized in Table 26-1.

Table 26-1: Estimated Cost for a Pre-feasibility Study for the Rowan Project

Recommended Work	Work Description	Estimated Cost [C$M]
Drilling and mineral resource estimate	In-fill drilling and conversion drilling; Rowan deposit geological model update; and mineral resource estimate update.	3.0
Mineral processing and metallurgical testwork	Variability testing; toll mill flowsheet testing; and blended feed testing.	0.2
Geotechnical studies to support mine engineering	Geotechnical-specific drilling for infrastructure and crown pillar study; detailed structural logging or televiewer logging to determine structure orientation; rock strength laboratory program; and geotechnical model development.	0.4
Mine engineering	Underground mine design update; ventilation design update; and mine services design update including dewatering, electrical, and safety systems.	0.5
Recovery methods	Verify pre-crusher size; gravity circuit expansion design update; and leach and CIP tank design update.	0.1
Waste rock and water management	Site investigation test pit program design; WBM update; WRS and water management pond berm stability analysis; and site layout design update.	0.2
Environmental studies and permitting	Environmental baseline studies; waste rock geochemical characterization; engagement and consultation; approval and permitting submission development.	1.7
Pre-feasibility study engineering and report	Infrastructure design; material handling design; economic modelling; capital and operating cost estimates; material take-offs development; and reporting.	0.8
Contingency (15%)		1.1
Total Recommended Work Cost		**8.0**

Notes:
WBM = water balance model; WRS = Waste Rock Stockpile.

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28 QUALIFIED PERSON CERTIFICATES

Certificate of Qualified Person
Grant Carlson, P.Eng.

I, Grant Carlson, P.Eng., certify that I am employed as a Director, Technical Services with Fuse Advisors Inc., with an office address of 595 Burrard Street, Suite 3113, Three Bentall Centre, Vancouver, BC, V7X 1J1.

This certificate applies to the technical report titled "Rowan Project NI 43-101 Technical Report and Preliminary Economic Assessment, Ontario, Canada" that has an effective date of 30 June 2025 (PEA or Technical Report).

1. I graduated from the University of British Columbia in 2007 with a Bachelor of Applied Science in Mining Engineering.
2. I am a Professional Engineer registered with Engineers and Geoscientists of BC (No. 36395).
3. I have practiced my profession for 18 years. I have been directly involved in mining operations in Canada, in similar climatic conditions as the Rowan Project, as a planning engineer, shift supervisor, mine manager, chief operating officer, and mining consultant. I have worked on, visited, and carried out due diligences on precious metal, base metal, diamond, and coal mining operations around the world including Russia, Madagascar, Australia, Eritrea, Lesotho, Ghana, Sierra Leon, Brazil, Peru, Chile, Argentina, Suriname, Guyana, Mexico, the United States, and Canada. Recent relevant experience includes mine planning on Gold Mountain Mining Corp's Elk Gold Project, which is a narrow vein, high-grade gold deposit similar to the Rowan deposit, which is also a toll milling operation. I have also acted as the Qualified Person for the Preliminary Economic Assessment on First Mining Gold's Springpole Project, also in northern Ontario approximately 130 km from the Rowan Project.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
5. I have not visited the Rowan Project site.
6. I am responsible for Sections 1.1, 1.9, 1.11, 1.12, 1.13, 1.14, 1.15, 1.16, 2, 3, 15, 16 (excluding 16.7), 18 (excluding 18.3 and 18.4), 19, 20 (excluding 20.5), 20.5.4, 21, 22, 24, 25.3, 25.5 (excluding 25.5.2), 25.6, 25.7, 25.8, 25.9, 25.10, 25.11.1.3, 25.11.1.4, 25.11.1.5, 25.11.1.6, 25.11.1.7, 25.11.2.2, 25.11.2.4, 26.4, 26.7, 26.8, and 27 of the Technical Report.
7. I am independent of West Red Lake Gold Mines Ltd. as independence is defined in Section 1.5 of NI 43-101.
8. I have been previously involved with the Rowan Gold Project as a consultant in 2023 and 2024 developing conceptual mine plans.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: 4 May 2026

(signed) "*Grant Carlson*"

Grant Carlson, P.Eng.

Certificate of Qualified Person
John Sims, C.P.G.

I, John Sims, C.P.G., certify that I am employed as a Professional Geologist with Sims Resources LLC, with an office address of PO Box 131 Missoula, Montana 59806.

This certificate applies to the technical report titled "Rowan Project NI 43-101 Technical Report and Preliminary Economic Assessment, Ontario, Canada" that has an effective date of 30 June 2025 (PEA or Technical Report).

1. I Graduated from the University of Montana in 1993 with a Bachelor of Applied Science (B.A.Sc.), Geology.
2. I am a Professional Geologist registered with AIPG, CPG #10924
3. I have practiced my profession for 40 years and have been directly involved in the building numerous mineral resource models some of which include Dvoinoye in Russia, Fruta del Norte in Ecuador and Lobo Marte in Chile; leading the re-modeling at Tasiast in Mauritania and Bald Mountain in Nevada; assisting Paracatu in the establishment of its world-class grade control program.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
5. I visited the Rowan Project site from 20 February 2024.
6. I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.8, 1.16, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.11.1.1, 25.11.2.1, 26.1, 26.8, and 27 of the Technical Report.
7. I am independent of West Red Lake Gold Mines Ltd. as independence is defined in Section 1.5 of NI 43-101.
8. I have no prior involvement with Rowan Project.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: 4 May 2026

(signed) "*John Sims*"

John Sims, C.P.G.

Certificate of Qualified Person
Travis O'Farrell, P.Eng., MBA

I, Travis O'Farrell, P.Eng., certify that I am employed as a Director, Projects and Processing with Fuse Advisors Inc., with an office address of 595 Burrard Street, Suite 3113, Three Bentall Centre, Vancouver, BC, V7X 1J1.

This certificate applies to the technical report titled "Rowan Project NI 43-101 Technical Report and Preliminary Economic Assessment, Ontario, Canada" that has an effective date of 30 June 2025 (PEA or Technical Report).

1. I graduated from McGill University in 2010 with a Bachelors of Chemical Engineering.
2. I am a Professional Engineer registered with EGBC (No. 46026).
3. I have practiced my profession for 15 years. I have been corporate metallurgist for mining operations in Canada and metallurgist and/or process consultant for numerous gold and base metal concentrators around the world including Canada, USA, Mexico, Brazil, New Caledonia, Suriname, Ghana. To name a few specific examples, I was the corporate metallurgist for Equinox Gold with CIL and CIP plants in Brazil and Ontario, which are at higher throughputs, but use very similar processes.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
5. I have not visited the Rowan Project site.
6. I am responsible for Sections 1.7, 1.10, 13, 17, 18.3, 25.1, 25.4, 25.11.1.2, 25.11.2.3, 26.2, 26.5, 26.8, and 27 of the Technical Report.
7. I am independent of West Red Lake Gold Mines Ltd. as independence is defined in Section 1.5 of NI 43-101.
8. I have not had prior involvement with the Rowan Project.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: 4 May 2026

(signed) "*Travis O'Farrell*"

Travis O'Farrell, P.Eng.

Certificate of Qualified Person
Paul Hughes, P.Eng.

I, Paul Hughes, P.Eng., certify that I am employed as a Geotechnical Engineer with PHC Inc., with an office address of 940 Fresno Place, Coquitlam, BC, V3J 6G6.

This certificate applies to the technical report titled "Rowan Project NI 43-101 Technical Report and Preliminary Economic Assessment, Ontario, Canada" that has an effective date of 30 June 2025 (PEA or Technical Report).

1. I graduated from the University of British Columbia in 2004 with a Bachelor of Applied Science – Geological Engineering
2. I am a Professional Engineer registered with Engineers & Geoscientists of British Columbia (No. 36997).
3. I have practiced my profession for 20 years. I have been directly involved in the study of rock mechanics. To name a few specific examples, I have acted as Qualified Person for the rock mechanics aspects of the underground Kudz de Kayah Mine and Keno Hill Silver District Project, and the open pit Hualilan Project.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
5. I visited the Rowan Project site from 24 June 2023 to 27 June 2023.
6. I am responsible for portions of Sections 1.9, 16.7, 25.11.1.3, 25.11.2.2, 26.3, 26.8, and 27 of the Technical Report.
7. I am independent of West Red Lake Gold Mines Ltd. per the definition of independence within Section 1.5 of NI 43-101.
8. I have no prior experience with the Rowan Project prior to the commencement of the technical report.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: 4 May 2026

(signed) "*Paul Hughes*"

Paul Hughes, P.Eng. Ph.D.

CERTIFICATE OF QUALIFIED PERSON

Daniel Ruane, P.Eng.

This certificate applies to the technical report titled "Rowan Project NI 43-101 Technical Report and Preliminary Economic Assessment, Ontario, Canada" that has an effective date of 30 June 2025 (PEA or Technical Report).

I, Daniel Ruane, P. Eng., do hereby certify that:

1. I am currently employed as a Specialist Engineer with Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
2. I graduated from the National University of Ireland, Galway with a Bachelor of Engineering in Civil Engineering in 2010 and from the University of Strathclyde and the University of Glasgow with a Master of Science in Geotechnics in 2011. I have practiced my profession continuously since 2011. My experience includes tailings and waste and water management for mining projects in North and South America and Europe.
3. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia, License No. 42894.
4. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
5. I have not visited the Rowan Project site.
6. I am a co-author of the Technical Report, responsible for Sections 1.9, 1.11, 18.4 (excluding 18.4.10), 20.5 (excluding 20.5.4), 25.5.2, 25.11.1.4, 26.6, 26.8, and 27.
7. I am independent of West Red Lake Gold Mines Ltd. as independence is defined in Section 1.5 of NI 43-101.
8. I have not had prior involvement with the subject property.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: 4 May 2026

(signed) "*Daniel Ruane*"

Daniel Ruane, P. Eng.
Specialist Engineer | Associate

Certificate of Qualified Person
A.J. MacDonald, P.Eng.

I, A.J. MacDonald, P.Eng., certify that I am employed as a Vice President, Engineering with Integrated Sustainability Consultants Ltd., with an office address of 620-1050 West Pender St., Vancouver, BC, V6E3S7

This certificate applies to the technical report titled “Rowan Project NI 43-101 Technical Report and Preliminary Economic Assessment, Ontario, Canada” that has an effective date of 30 June 2025 (PEA or Technical Report).

1. I graduated from Queen’s University, Kingston, Canada (B.Sc., 2005) and Carleton University, Ottawa, Canada (M.A.Sc., 2007)
2. I am a Professional Engineer registered with Engineers and Geoscientists British Columbia (member 45872). I am also a member of Association of Professional Engineers and Geoscientists of Alberta, Association of Professional Engineers and Geoscientists of Saskatchewan, Professional Engineers Ontario, Professional Engineers Yukon, Engineers and Geoscientists New Brunswick, Professional Engineers and Geoscientists of Newfoundland and Labrador, and Nevada Board of Professional Engineers and Land Surveyors
3. I have practiced my profession for 19 years. I have been involved or associated with the mining industry since 2007. I have participated in dozens of mining and other resource sector projects, with a particular focus on water treatment process engineering, primarily in Western North America. My experience spans all phases of project delivery including preliminary analysis, conceptual design, detailed design, construction, commissioning and optimization of infrastructure at industrial water treatment facilities in Canada and around the world.
4. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
5. I have not visited the Rowan Project site.
6. I am responsible for Sections 1.11, 18.4.10, 25.11.1.4, 26.6, and 27 of the Technical Report.
7. I am independent of West Red Lake Gold Mines Ltd. as independence is defined in Section 1.5 of NI 43-101.
8. I have had no previous involvement with the Rowan Project.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: 4 May 2026

(signed) "*A.J. MacDonald*"

A.J. MacDonald, P.Eng.