NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada

Report prepared for

West Red Lake Gold Mines Ltd.

Prepared by

srk consulting

SRK Consulting (Canada) Inc.
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Signed by Qualified Persons:

Cliff Revering, P.Eng. SRK Consulting (Canada) Inc.
Sheila Ulansky, P.Geo. SRK Consulting (Canada) Inc.
Travis O'Farrell, P.Eng. Fuse Advisors Inc.
Stephen Taylor, P.Eng. SRK Consulting (Canada) Inc.
Tim Coleman, P.Eng. SRK Consulting (Canada) Inc.
Brian Prosser, P.Eng. SRK Consulting (U.S.) Inc.
Guy Lauzier, P.Eng. Allnorth Consultants Limited
Chris Dougherty, P.Eng. Nordmin Engineering Ltd.
Bernie Ting, P.Eng. T Engineering
Daniel Ruane, P.Eng. Knight Piésold Consulting Ltd.
Mark Liskowich, P.Eng. SRK Consulting (Canada) Inc.

Reviewed by:

Paul Dagenais, MBA, MASc SRK Consulting (Canada) Inc.

Prepared for

West Red Lake Gold Mines Ltd.
Suite 3123 – 595 Burrard Street
Vancouver, BC V7X 1J1
Canada

Tel.: +1.604.609.6138
www.westredlakegold.com

Prepared by

SRK Consulting (Canada) Inc.
320 Granville Street, Suite 2600
Vancouver, BC V6C 1S9
Canada

Tel.: +1 604 681 4196
Web: www.srk.com

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1 Summary

This report has been prepared by SRK Consulting (Canada) Inc. (SRK) on behalf of West Red Lake Gold Mines Ltd. (WRLG or the Company). The purpose of this report is to provide a technical report that documents all supporting work, methods used and results relevant to a prefeasibility study (PFS) on the Madsen Mine in Ontario, Canada, and that fulfills the reporting requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

1.1 Property Location and Access

The Property is centered at 50.97° North latitude and 93.91° West longitude (UTM Projection NAD83, Zone 15 North coordinates 5646000N, 435000E) within the Baird, Heyson and Dome Townships of the Red Lake Mining District in northwestern Ontario.

The Madsen Mine is located adjacent to the community of Madsen, within the Red Lake Municipality of northwestern Ontario, approximately 565 km by road (430 km direct) northwest of Thunder Bay, Ontario and approximately 475 km by road (260 km direct) east-northeast of Winnipeg, Manitoba. Red Lake can be reached via Highway 105 from Trans-Canada Highway 17. Red Lake is also serviced with daily flights from Thunder Bay and from Winnipeg by Bearskin Airlines.

The mine is accessible from Red Lake via Highway 618, a paved secondary road maintained year-round by the Ontario Ministry of Transportation. The mine is 10 km southwest of the town of Red Lake. A series of intermittently maintained logging roads and winter trails branching from Highway 618 provide further access to other portions of the Property.

1.2 Mineral Tenure and Surface Rights

The Madsen Mine property comprises a contiguous group of 241 tenures, covering an aggregate area of 4,648 hectares in northwestern Ontario. WRLG owns 100% of all mining leases, patents and unpatented claims comprising the mine property.

WRLG owns surface rights in the form of mine property claims, patents and leases. Where WRLG does not hold surface rights, they 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 public use. A single trapping tenure is held over the entire property and WRLG maintains good relations with the tenure holder. Several registered easements for highway and utility lines cross the property.

1.3 History

Gold was originally reported in the Red Lake area in 1897. Intensive exploration of the district followed discovery in 1925 of the gold showings that eventually formed part of the Howey Mine. Since 1927, a total of 28 mines have operated in the Red Lake Mining District, producing 29 Moz of gold at an average recovered grade of 15.6 g/t Au. Approximately 89% of this gold was produced from two mine complexes: Red Lake Mine and Madsen Mine.

1.4 Geological Setting and Mineralization

The Madsen Mine is located within the Western portion of the Archean Superior Province of the Canadian Shield. It 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 is characterized by Mesoarchean and Neoarchean volcanic and plutonic rocks interpreted to have been emplaced within rift and arc-related environments on the continental margin of the Mesoarchean crustal rocks of the North Caribou terrane. The predominantly sedimentary rocks of the English River belt are believed to have accumulated within a synorogenic flysch basin that formed during assembly of the North Caribou terrane with the Winnipeg River terrane to the south during the Uchian Orogeny, ca. 2720-2700 Ma (Percival et al. 2006).

The mine property is underlain by Balmer, Confederation and Huston Assemblage supracrustal rocks. These older rocks are cut by a series of plutonic rocks (post-tectonic Killala-Baird batholith to the west and syn-kinematic Dome and Faulkenham Lake Stocks to the east) and associated smaller sills and dykes.

Most of the historical gold production and most of the current mineral resources at the Madsen Mine are within the Austin, South Austin and McVeigh zones which, along with the 8 Zone, comprise the Madsen deposit. At the scale of the property, these zones all lie within much broader, kilometre-scale planar alteration and deformation corridors that have been repeatedly reactivated during gold mineralization and subsequent deformation and metamorphism. The distribution of gold within these planar structures is almost exclusively within variably altered basalt, and enhanced in close proximity to major lithological contacts, such as ultramafic sills, felsic dykes and felsic volcanic strata.

Controls on mineralization at the Madsen Mine are consistent with a typical orogenic gold system. Many deposit-scale features such as control by lithological/structural contacts and association with felsic dykes are typical in these systems. Recent work indicates that, apart from its early timing of emplacement prior to the dominant regional deformation and metamorphism, the Madsen Mine shares many characteristics with typical orogenic gold deposits, including the Red Lake Mine deposit.

1.5 Exploration, Drilling and Exploration Potential

Since acquiring the Madsen Property in 2023, WRLG has conducted geological mapping, surface rock sampling and glacial till geochemical sampling, all of which were undertaken as part of the 2024 exploration program. The Madsen Mine surface (non-drilling) exploration dataset comprises systematic, property-wide, multifaceted information carefully collected using modern techniques. Combining surface geophysical (magnetic and seismic), geochemical and geological information with historical data and drilling data has allowed for a property-wide geologic map that has formed an important input for sub-surface 3D geologic interpretation supported by the drilling dataset. Delineation of several new surface targets has resulted from compilation of the surface data sets. The surface dataset continues to be refined and informed by infill geological mapping supported by mechanical stripping and by diamond drilling. In the current state it forms a valuable base for geologic interpretation and extrapolation in support of exploration.

The Property has a long history of diamond drilling, dating from initial discovery of the Madsen deposit in the 1930s through until the present day. Documentation of procedures and methods of drilling is sparse prior to the 1990s. All historical exploration and production drill testing on the mine property to date has been by diamond drill coring. Underground drilling from 1937 to 1999 at Madsen Mine employed whole core sampling and most core intervals were sampled for fire assay gold analysis at the on-site mine laboratory. Pure Gold drilled a total of 2,411 diamond drill holes for 399,661 m between 2014 and 2022. These totals include both exploration drilling outside the footprint of the Madsen Mine and definition drilling to support mining operations. Since acquiring the Madsen project in June 2023 and up to May 15, 2024, WRLG completed a total of 146 holes for 11,849 m of BQ diamond drill core (definition) and 59 holes for 8,024 m of NQ diamond drill core (expansion) from underground. WRLG continued definition drilling through the rest of 2024, although that work is not considered in this report. Definition drilling was focused on the Austin and South Austin zones to increase geologic confidence in these areas to a level appropriate for mine development planning. Expansion drilling was focused primarily within the newly defined North Austin zone outside of the existing life-of-mine mineral resource domains, but still in close proximity to existing underground infrastructure. Underground drilling in 2023 and 2024 was completed by Boart Longyear.

Diamond (core) drilling is the most appropriate test method for the mine and this technique has been applied by all operators since early exploration and mining. Historical drilling is tightly spaced (nominally drilled at 6 m centres) within mined-out areas but other largely non-mined areas show evidence of alteration and elevated gold and have been drilled at much broader spacing.

Exploration for gold on the mine property focuses on identifying the planar structures (or shear zones) that were active during gold deposition. Since gold is very heterogeneously distributed within these structures, assessing targets using gold assay data alone will not yield reliable results. The ground in and around the Madsen Mine has high prospectivity for gold and exploration potential exists both within the Madsen deposit and in the adjacent areas (e.g., Russet, Wedge, Fork, Starratt, Gap, Derlak exploration targets).

1.6 Sample Preparation, Analysis and Data Verification

Sampling procedures and methods have evolved significantly over the long history of exploration and mining at the Madsen Mine and specific procedures also varied among operators. The QP is of the opinion that, based on historical information available, the historical 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 undertaken up to WRLG’s acquisition of the property meet or exceed modern best practices. The historical procedures and those undertaken by WRLG are adequate for modern targeting, modelling and resource estimation.

Owing to the long history of exploration and production at the mine, there have been numerous campaigns of data verification, validation and reconciliation. The most comprehensive recorded verification effort (Cole et al., 2010) was conducted during the digitization of the mining-era hardcopy drill logs, prior to Pure Gold’s acquisition of the property. This work was initiated by Claude in 1998, advanced by Placer Dome from 2002 to 2006 and completed by Claude with assistance from SRK during 2008 and 2009. The result was a modern digital database comprising 13,617 historical drill holes with lithological intervals and 550,687 gold assays. This database was the foundation for drill-targeting, geological interpretation and mining by Pure Gold and has been substantially added to and verified since 2014.

The Madsen Mine drilling database is compiled from historical and modern work that spans over 80 years. Available historical hard-copy records were collected and transferred into a modern digital database (Cole et al., 2010). Use and verification of this database shows that it is of high quality, largely free of errors and highly effective, even if assessment of the original data collection methods is not possible. Work by Pure Gold, and subsequently by WRLG, has been conducted with clear data handling protocols and an industry-standard quality control program.

1.7 Mineral Processing and Metallurgical Testing

Historical metallurgical data is available from mill operations dating back to the 1951 Madsen Lake Gold Mines Limited annual report. Gold recovery percentages in the mid-90s were reported at the time. The mill operated for over 40 years with mill throughput ranging from 350 t/d to 850 t/d. In later years, recoveries in the mid-90s continued to be achieved. The present mill was purchased and relocated in the 1990s from Placer Dome's Dona Lake mine. The mill operated at a nominal rate of 600 t/d and used the carbon-in-pulp (CIP) process to recover gold. A 1998 mill report indicated an average annual recovery of 90% at an average gold head grade of 4.2 g/t (Madsen Gold Corp.). The most recent test program, completed in 2018 in support of the 2019 Feasibility Study completed by JDS for Pure Gold, was carried out at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC. A full breakdown of the test program results can be found in the BaseMet (2018) report.

Production data from December 2020 to October 2021 was reviewed, which included head grade, throughput, grind size, gravity recovery, overall recovery and reagent consumption. It was noted that the production data was aligned with the historical metallurgical data and 2018 testwork.

The primary objective of the BaseMet (2018) test program was to confirm the flowsheet and design criteria using the historical data and the existing plant design. Drill core was sent to BaseMet for test work that included sample preparation, mineralogy, comminution gravity concentration, cyanide leach and bulk cyanide leaching to produce material for continuous cyanide destruction.

The BaseMet (2018) test program was done in three phases: Variability Scoping Composites, Year Composites, and Variability Composites. The first phase was scoping variability tests on 12 composites from the five zones to evaluate the metallurgical response using the existing plant flowsheet and historical data. The second phase included test work on composites that at the time of the test program represented Years -1 to 1, 2 to 3 and 4 to 7 of the mine schedule. The final phase tested the optimized flowsheet using 30 variability composites representing the five zones of the deposit.

Based on the results from BaseMet (2018) and 2021 plant operational data, gold doré with no significant levels of deleterious elements can be produced with a primary grind size of 80% passing (P80) 75 µm followed by gravity concentration, 2-hour pre-oxidation, 24-hour cyanide leach, 5-hour carbon-in-pulp (CIP) adsorption, desorption and refining process. Using the blended average recovery of the samples tested based on the 2024 SRK mine plan is estimated to achieve a LOM gold recovery of 95.7%.

1.8 Mineral Resource Estimate

The current, previously disclosed Mineral Resource Estimate (MRE) for the Madsen Mine (Table 1-1) was generated by Mr. Cliff Revering, P.Eng., of SRK Consulting (Canada) Inc., with an effective date of December 31, 2021. The estimate includes Indicated mineral resources of 1,653,000 oz of gold (6.9 Mt at an average grade of 7.4 g/t) and Inferred mineral resources of 366,200 oz of gold (1.82 Mt at an average grade of 6.3 g/t). These mineral resources are reported at a cut-off grade of 3.38 g/t, using a gold price of US$1,800 per ounce, and are constrained by reasonable stope shapes within the Madsen deposit.

This MRE is based on verified historical drilling data and additional drilling data and underground mine development and production data collected by Pure Gold between 2014 and 2022. This MRE is also predicated on a revised geological and mineralization domain model developed in 2021 that incorporates structural controls on mineralization identified through data analysis, grade control programs and mapping of underground exposures by Pure Gold between 2018 and 2022.

Since the effective date of this MRE, additional diamond drilling was conducted. A total of 688 drill holes and 54,122 m of drilling was completed in 2022. An additional 205 drill holes and 19,872 m of drilling was completed by WRLG between October 1, 2023 and May 15, 2024. Based on a review of the results of this drilling it has been determined by Mr. Cliff Revering, QP for the Madsen MRE, that the information obtained will not have a material impact on the MRE presented in this report.

Table 1-1: Mineral Resource Statement, PureGold (Madsen) Mine, Red Lake, Ontario (effective date December 31, 2021)

Classification	Deposit – Zone	Tonnes	Gold Grade (g/t)	Total Gold (troy oz)
Indicated	Madsen – Austin	4,147,000	6.9	914,200
	Madsen – South Austin	1,696,000	8.7	474,600
	Madsen – McVeigh	388,700	6.4	79,800
	Madsen – 8 Zone	152,000	18	87,700
	Fork	123,800	5.3	20,900
	Russet	88,700	6.9	19,700
	Wedge	313,700	5.6	56,100
	Total Indicated	**6,909,900**	**7.4**	**1,653,000**
Inferred	Madsen – Austin	504,800	6.5	104,900
	Madsen – South Austin	114,100	8.7	31,800
	Madsen – McVeigh	64,600	6.9	14,300
	Madsen – 8 Zone	38,700	14.6	18,200
	Fork	298,200	5.2	49,500
	Russet	367,800	5.8	68,800
	Wedge	431,100	5.7	78,700
	Total Inferred	**1,819,300**	**6.3**	**366,200**

Notes:

1) Mineral Resources estimated in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Cliff Revering, P.Eng, Qualified Person.
2) Mineral resources are not mineral reserves and do not have demonstrated economic viability
3) Mineral resources are reported at a cut-off grade of 3.38 g/t Au
4) Mineral resources are reported using a gold price of US$1,800/oz
5) Excludes depletion of mining activity during the period from January 1, 2022 to the mine closure on October 24, 2022 as it has been deemed immaterial and not relevant for the purpose of this report.
6) All figures have been rounded to reflect the relative accuracy of the estimate

1.9 Mineral Reserve Statement

The Madsen Mine has been mined extensively from the mid-1930s to the mid-1970s with more than 8.9 Mt of ore being extracted. Much of the higher-grade material in the mineral resource model is remnants contained in sill pillars and/or immediately adjacent to the historic shrinkage stopes. The mineral reserves (Table 1-2) are contained within a mining area with a strike length of 1,250 m and a 1,200 m vertical extent with a 60° plunge to the SSE. The mineral reserves follow the trend of the historic shrinkage stopes. The strike length of the historic development is 2,000 m with a 1,300 m vertical extent. This presents unique challenges and opportunities for modern mining operations using trackless, mechanized equipment.

Table 1-2: Mineral Reserve Statement, Madsen Mine, Red Lake, Ontario (effective date June 30, 2024)

Classification	Deposit - Zone	Tonnes (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Probable	Madsen - Austin	778	7.37	184
	Madsen - South Austin	861	8.21	227
	Madsen - McVeigh	66	7.37	16
	Madsen - 8 Zone	118	13.38	51
Proven + Probable		**1,823**	**8.16**	**478**

Notes:

1) Mineral Reserves estimated in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Stephen Taylor, P.Eng., Qualified Person.
2) Longhole stope cut-off grade of 4.30 gpt Au based on an estimated operating cost of C$287.34/t including mining, plant and G&A. The mining cost component was benchmarked based on an operating mine in Ontario.
3) Mechanized Cut and Fill stope cut-off grade of 5.28 gpt Au based on an estimated operating cost of C$$354.90/t including mining, plant and G&A.
4) Mineral reserve estimates based on a gold price of US$1,680/oz and an exchange rate of 1.31 C$/US$.
5) Incremental development cut-off grade of 1 gpt Au.
6) A small amount of incremental longhole tonnes were included at a cut-off grade of not less than 3.4 gpt Au, these must be immediately adjacent to economic stopes that will pay for the capital to access area.

The Mineral Reserve Statement presented herein has been prepared for public disclosure according to CIM Best Practice Guidelines (CIM, 2019) and reported as diluted tonnes delivered to the mill. As there are no Measured mineral resources included in the 2021 mineral resource model upon which the PFS mine design is based, there are no Proven mineral reserves included in the Mineral Reserve Estimate.

1.10 Mine Plan, Development and Production

The mine plan for the Madsen Mine is based on the resource model completed by SRK. At present, the Madsen Mine has historic workings covering a 2.3 km strike length to 1,300 m depth.

A significant portion of the higher-grade mineral resources is located in close proximity to the historic workings and can be considered remnant mining targets. These include mineral resources left in place as pillars, not considered mineable at the time, below the cut-off grade at the time, or simply not recognized as ore at the time. Using modern mechanized cut and fill (MCF) mining methods and ground support techniques, a portion of these remnants can be safely extracted today. There are also mineral resources in unmined areas, though these tend to be lower grade than the core zones extracted historically.

A geotechnical assessment was completed using available data, which included an exploration drillhole database, geological underground mapping, and Madsen's ground control management plan. This was supplemented by reports from consultants associated with site feasibility studies and aspects of the project's technical rock mechanics. This data was used to characterize the geotechnical conditions of the rock and support the underground mine and infrastructure design, an evaluation of geotechnical design domains, and the development of geotechnical design guidelines. These guidelines included excavation design parameters, estimates of dilution, as well as ground support requirements.

A number of underground mining methods were considered to deal with different challenges encountered in the various zones, but the mining methods selected for ore extraction at Madsen were narrowed down to:

- Longitudinal Retreat Longhole with ramp access (LH): 32.7% of production
- Mechanized Cut and Fill (MCF) with ramp access: 60.6% of production
- Mechanized Drift and Fill (MDF) with ramp access for 8 Zone: 6.5% of production

Mining method selection was driven primarily by mineralization geometry and continuity, selectivity of method, ability to mechanize the method, proximity to historic workings and anticipated ground conditions. For design purposes, LH was the preferred method of extraction followed by MCF.

Current access to the mine is through the East ramp or West ramp systems. The operation is currently developing a connection between these two ramp systems to improve public safety and haulage efficiency. The connection is expected to be completed in Q1 2025. This connection will eliminate the need for mine vehicles to cross the two municipal roads leading to the town of Madsen when using the West ramp portal. These ramp systems are suitable for modern 40-t haul trucks.

A project is also ongoing to recondition the Madsen Shaft #2 and install a new loading pocket at 12 Level to facilitate hoisting operations and reduce trucking requirements as the active mining areas get deeper. This work is being undertaken by a mining contractor and is expected to be completed in Q3 2025. This shaft is a rectangular timber shaft with small skips and is also acting as the main ventilation intake.

Due to the hoisting capacity limitations of Madsen Shaft #2 to hoist the required ore and waste tonnages from below 12 Level, a new Madsen Shaft #3 has been proposed with construction expected to be completed Q4 2028. This shaft will also help support the increased ventilation flows required by modern diesel equipment. The Madsen Shaft #2 is planned to be reconditioned down to 24 Level to provide cage access to provide emergency egress from the mine, either in the cage or ladderway.

Access for personnel and materials will mainly be via the East portal and ramp systems for the life of mine (LOM) with some movement via the shafts. Once commissioned, the Madsen Shaft #2 will be used to hoist ore and waste to surface until such time as Madsen Shaft #3 is completed.

Ore will be trucked from the work areas to the closest available dump point:

- Surface
- 10 Level grizzly for Madsen Shaft #2
- 18 Level grizzly for Madsen Shaft #3

The total capital and operating development included in the PFS LOM Plan is shown in Table 1-3. Note that the operating development includes MCF/MDF stoping metres.

Table 1-3: Total Capital and Operating Development

Description	Type	Metres
Capital	Lateral	31,870
Capital	Vertical	2,250
Total Capital		**34,120**
Operating	Lateral	49,612
Grand Total		**117,852**

As the mine was previously in commercial production under Pure Gold, the mine is expected to ramp up to 800 tpd and achieve commercial production in Q1 2026 (Table 1-4). WRLG has been executing some pre-production work and test mining to prepare the mine for production, including developing the connection drift, continuing dewatering and beginning rehabilitation of Madsen Shaft #2.

Table 1-4: Madsen Mine production schedule

	Total	2025	2026	2027	2028	2029	2030	2031	2032
Production Rate (tpd)		460	813	796	791	813	813	522	54
Ore Recovered (kt)	1,823	166	293	286	285	293	293	188	20
Head Grade (Au g/t)	8.16	7.00	7.20	7.80	8.00	7.87	7.79	12.29	10.40
Contained Gold (koz)	478.3	37.3	67.8	71.8	73.3	74.1	73.4	74.2	6.5
Recovery (%)	95.7%	94.7%	93.8%	94.3%	95.5%	95.6%	96.5%	98.7%	98.4%
Recovered Gold (koz)	457.9	35.3	63.6	67.7	70.0	70.8	70.8	73.2	6.4

Until the loading pocket on Shaft #2 is operational, all ore will be trucked to surface via ramp. Once the loading pocket is ready, ore will be trucked to the 10 Level grizzly or surface depending on stope location. The proposed Madsen Shaft #3 is expected to be operational by the end of 2028 as mining progresses deeper. Ore will then be trucked to either the 18 Level grizzly for Shaft #3 or the 10 Level grizzly for Shaft #2 depending on stope location. As mining progresses, more of the ore will be hoisted as the near surface stopes are depleted.

Mine sequencing is generally top down by mining area with stope sequence being bottom up within the mining areas.

The overall ventilation strategy for the Madsen Mine will provide control over the fresh air supply and routings, with uncontrolled or free exhaust routings to surface through open stopes, intermediate ore/waste passes, fringe or perimeter raises and decline accesses. The ventilation system will be developed or driven by two exhaust fans installed in the ramp accesses near the surface, and the fresh air will be provisioned by the existing Madsen Shaft #2 and the future Madsen Shaft #3.

The majority of the mining fleet was purchased by the previous owner with some additions having been made by WRLG through purchase and leasing agreements through 2024. At the start of Q2 2025, the mine is expected to have 56 pieces of underground mining equipment available, building up to 70 units by the start of 2026.

The Madsen Mine is expected to utilize an underground workforce averaging 221 people over the LOM Plan. There are currently approximately 140 workers employed, of which approximately 60% live locally. The operation is actively recruiting with a 114-person camp and a new mine dry under construction to accommodate the additional people. Peak labour requirements occur in Q4 2025 as the mine reaches full production. Steady state is achieved in 2026 through 2028, with reductions in the work force starting in 2029 as lateral development requirements taper off.

1.11 Recovery Methods

The processing plant has a design capacity of 800 t/d. The plant is conventional Leach - Carbon in Pulp and consists of the following unit operations:

- One stage semi-mobile jaw crusher (new)
- SAG Mill (existing) and Ball Mill (existing)
- Gravity separation (two existing) and intensive leach system
- Pre-leach thickening to 50% solids
- Pre-oxidation (one existing)
- Leach (five existing) and carbon-in-pulp (six existing)
- 1-tonne carbon plant and gold recovery in a refinery
- Cyanide destruction utilizing two tanks (one standby)
- Tailings pumped to the tailings management facility (TMF)

The crushing circuit is designed to operate at an availability of 50% while the milling, leaching and gold recovery circuits will operate 24 hours per day, 365 d/y at an availability of 95%. The plant will process 33 t/h on average with a target grind size of approximately 80% passing 75 µm at an average LOM head grade of 8.16 g/t Au. Primary grinding will be followed by gravity concentration, 2-hour pre-oxidation, 24-hour cyanide leach, 5-hour carbon-in-pulp (CIP) adsorption/desorption and refining to recover the gold to doré bars. Using the blended average recovery of the samples tested, based on the 2024 SRK mine plan, it is estimated that a LOM gold recovery of 95.7% can be achieved.

After cyanide destruction, the CIP tailings will be pumped to the TMF initially. Starting later in Year 1, a thickened tailings will be pumped into open stopes using a hydraulic backfill system.

1.12 Project Infrastructure

The Madsen Mine is a mature site with an existing underground mine, mineral processing facilities, a shaft, two portals (East and West), a water treatment plant, a tailings area, a rock dump and a general services area. Dewatering of Madsen Shaft #2 has been maintained since WRLG's acquisition in 2023. As part of the mine restart plan, WRLG will be adding surface facilities such as a new mine dry and a mobile crushing unit that are required to allow the operation to restart.

The Madsen Mine is connected to the northwest Ontario power network by aerial distribution power lines. The incoming voltage to the site is from a 44 kV circuit with a 12 MW power supply. The northwest Ontario power transmission network is owned and operated by Hydro One. Red Lake is located at the end of the 115 kV transmission line coming from Ear Falls, Ontario.

The Madsen Mine has an underground leaky feeder system for communications, which will be expanded as required into the new working areas.

The Madsen Mine is being dewatered through the Shaft #2 pumping stations. The pump stations house Mackley 7-stage pumps equipped with 250 kW/4160 V motors. The principal sumps are located at Shaft #2 and spaced out at six-level intervals. The average level interval is 45 m.

Approximately 2.6 Mt of mine rock material will be produced. Mine rock from underground development will be managed in the underground mine as backfill (42%) and stored (58%) in the existing mine rock management facility (MRMF), located adjacent to the TMF. Multiple void historic stopes create good opportunity to store waste rock underground, thereby reducing costs. Mine rock required for additional underground backfill and construction activities will be sourced from this stockpile.

Various backfill methods were considered and it was determined that cemented rock fill (CRF) will not be suitable for filling the legacy voids due to limited access to the area. Later in the mine life, the development waste rock can be used to make CRF or URF (uncemented rock fill) to reduce the amount of rock to be removed from underground. Hydraulic fill can leverage the existing hydraulic fill plant on site. While using hydraulic fill will introduce additional water as the fill materials decant, this option avoids the relatively high tailings reclaim cost with the paste option. The PFS assumes that the backfill will be sent underground through multiple surface boreholes. The capital cost estimate includes the cost for refurbishing the plant, constructing surface piping and the piping cost for the underground distribution system.

Tailings will be managed through a combination of surface storage in the TMF and underground deposition as hydraulic backfill. The TMF at the Project has been in operation since the late 1930s and has gone through several design modifications. The TMF at the Madsen Mine is permitted to discharge tailings and will be expanded to manage a total of 1.5 Mt. The TMF is partitioned into two designated areas, Cell A and the Main TMF. Containment for the first four years of tailings deposition will be provided in Cell A, with the remainder of the tailings managed in the Main TMF. Cell A is fully constructed and ready for operation, including a 4-m dam raise that was completed in summer 2024.

The Madsen Mine will be a shaft-based operation that will utilize the existing Madsen Shaft #2 installation for production in the first three to four years, moving to a new Shaft #3 facility to support ongoing future operations. The existing Madsen Shaft #2 system was constructed in the late 1950s, with completion of the sinking effort in 1958. The finished shaft is a five-compartment rectangular shaft that was designed to accommodate both production and service hoisting with a central manway. Shaft #2 will be used for muck hoisting and as a secondary means of emergency egress. As operational efforts progress, the centroid of the mining operation will move to the northeast more than 1.5 km away from Shaft #2 and to a potential depth of 4,000 feet or more. As such, and with the identified need to develop a large ventilation raise to allow for improved capacity in these areas, it was decided to equip this raise for use as a combined production and service shaft. The new Shaft #3 system would be constructed in Year 3 of the project and will be developed via raisebore. The shaft will be developed in two sections, with the first from 10 Level to surface, and the second from 20 Level to 10 Level, allowing for the surface plant to be constructed and shaft equipping to be undertaken while the second leg is in development. A rock pentice would separate the two legs, then excavated once the lower section of the shaft is completed, allowing for the completion of ground control and shaft furnishing to shaft bottom. The ultimate depth of the shaft will be at or around 4,000 feet (25 Level) via a third lift completed below 20 Level in the future, allowing for hoisting from a greater depth.

1.13 Environmental Studies, Permitting, Social/Community Impact

WRLG is continuing its scientific and engineering studies at the site; consultation with regulators, First Nations and communities; monitoring programs; and detailed project design planning to reopen the mine and processing facility. 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 resource use, improve water quality and bring overall benefit to local communities and First Nations.

WRLG has maintained the permits that existed for the Madsen Mine under previous operators. As the project has advanced, operational enhancements and regulatory changes have required permit updates. Permit status has been confirmed with the Ministry of Mines and Ministry of the Environment, Conservation and Parks (MECP) and the permits and authorizations are in good standing. The mine restart requires neither provincial nor federal Environmental Assessments.

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

1.14 Cost Estimate

Life-of-mine capital costs total C$502.9M (Table 1-5), including C$45.3M in contingency and C$9.1M related to closure costs. Project operating costs total C$588.1M (Table 1-6). The cost estimates were prepared based on pricing information obtained in 2024 and using the SRK LOM production plan.

Table 1-5: LOM capital costs

Description	Total Cost (C$M)
Capital Development	152.4
UG Mobile Equipment	54.0
Allocations to UG Capital	88.5
UG Infrastructure	123.1
Surface Infrastructure	18.4
Processing Capital	0.1
Water/Waste Management Capital	12.0
Contingency	45.3
Closure Costs	9.1
Total	**502.9**

Table 1-6: LOM operating costs

Description	Total Cost (C$M)
Mining	388.1
Processing	137.1
Water/Waste Management	1.1
G&A	61.8
Total	**588.1**

1.15 Economic Analysis

The project generates approximately C$71M in annual free cashflow from 2026 to 2031 (Table 1-7), resulting in an NPV at 5% of C$315M. The IRR associated with this cashflow is 170%. This is primarily due to the fact that capital expenditures projected to return the operation into production were incurred in 2024 and not included in the analysis. The project payback is 1.5 years from the start of production.

The project is most sensitive to changes in gold price, with every 1% change in gold price affecting project NPV by approximately C$11M, while the project is least sensitive to changes in capital costs, with every 1% change in capex affecting project NPV by approximately C$4M.

Table 1-7: Annual cash flow (C$000)

	Total	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**
Gross Revenue	$1,480,356	$129,183	$223,910	$233,630	$216,733	$219,435	$219,340	$226,780	$19,931	$0
Off-site Costs	$1,042	$89	$155	$162	$154	$156	$156	$161	$14	$0
Royalty	$14,793	$1,291	$2,238	$2,335	$2,166	$2,193	$2,192	$2,266	$199	$0
Net Revenue	**$1,464,521**	$127,803	$221,518	$231,133	$214,414	$217,087	$216,992	$224,353	$19,718	$0
Operating Costs	$588,089	$57,083	$89,746	$87,103	$89,478	$88,315	$90,843	$72,774	$12,747	$0
Mining	*$388,079*	*$38,872*	*$57,652*	*$55,702*	*$58,262*	*$56,219*	*$58,747*	*$52,146*	*$10,479*	*$0*
Processing	*$137,146*	*$12,462*	*$22,033*	*$21,555*	*$21,428*	*$22,034*	*$22,034*	*$14,128*	*$1,471*	*$0*
Waste/Water Mgmt	*$1,080*	*$135*	*$135*	*$135*	*$135*	*$135*	*$135*	*$135*	*$135*	*$0*
G&A	*$61,784*	*$5,614*	*$9,926*	*$9,711*	*$9,653*	*$9,926*	*$9,926*	*$6,365*	*$663*	*$0*
Operating Cashflow	**$876,433**	$70,720	$131,772	$144,030	$124,935	$128,772	$126,150	$151,579	$6,971	$0
Capital Costs	$499,421	$101,840	$75,907	$96,546	$95,475	$57,563	$28,339	$26,698	$10,984	$6,075
Mining	*$417,953*	*$83,203*	*$69,246*	*$85,737*	*$80,208*	*$53,388*	*$25,291*	*$20,034*	*$845*	*$0*
Plant and Infrastructure	*$18,483*	*$16,385*	*$390*	*$390*	*$390*	*$390*	*$361*	*$176*	*$0*	*$0*
Waste/Water Mgmt	*$11,993*	*$1,535*	*$307*	*$807*	*$7,819*	*$357*	*$357*	*$407*	*$407*	*$0*
Contingency	*$45,337*	*$12,878*	*$5,998*	*$8,635*	*$9,742*	*$4,020*	*$2,530*	*$1,512*	*$23*	*$0*
Closure	*$9,113*	*$0*	*$0*	*$0*	*$0*	*$0*	*$0*	*$0*	*$3,038*	*$6,075*
Change in Working Cap	*-$3,458*	*-$12,161*	*-$34*	*$977*	*-$2,683*	*-$591*	*-$200*	*$4,569*	*$6,672*	*$0*
Pre-tax Cashflow	$377,011	-$31,120	$55,865	$47,484	$29,460	$71,209	$97,811	$124,881	-$4,013	-$6,075
Tax	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Post-tax Cashflow	**$377,011**	-$31,120	$55,865	$47,484	$29,460	$71,209	$97,811	$124,881	-$4,013	-$6,075

1.16 Other Relevant Data

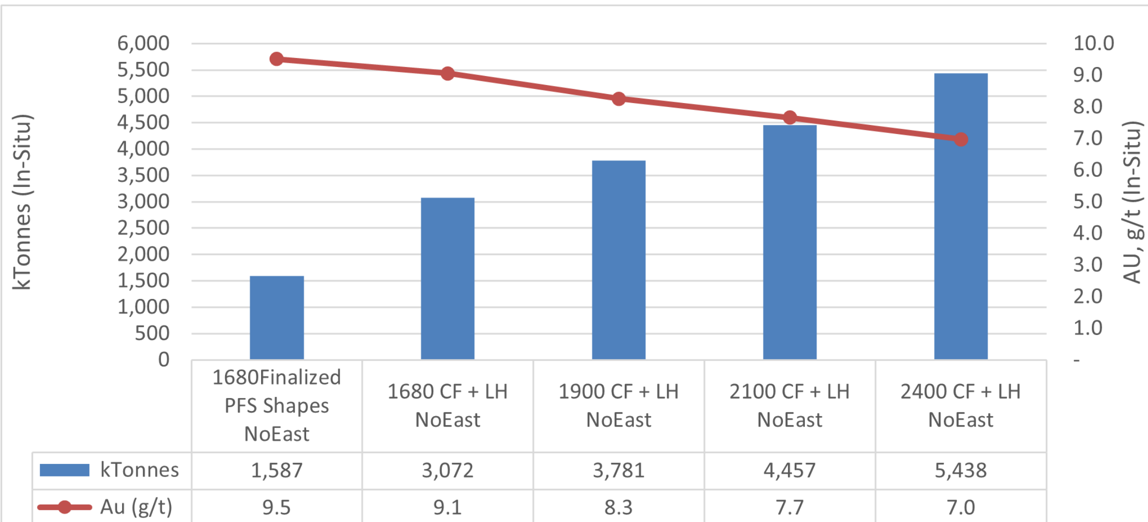
To further understand how the orebody behaved due to changing cut-off grades at various gold prices, two exercises were undertaken (see Section 24):

- Deswik Stope Optimizer (DSO) was used to run additional scenarios at gold prices of US$1,900/oz, US$2,100/oz, and US$2,400/oz, for the entire mine, excluding the East Zone

- A more detailed sensitivity analysis was performed for the East Zone using the existing mine design and gold prices of SU$1,900/oz, US$2,100/oz and US$2,500/oz

Figure 1-1 shows the change in tonnes and grades before accounting for development and modifying factors.

Figure 1-1: Sensitivity to gold price (excluding East Zone)



1.17 Risks and Opportunities

At the time of writing the mine is flooded between 14 Level and 15 Level, so the condition of the historic excavations below 14 Level is unknown. The mine plan assumes some historic track drifts can be reconditioned and utilized. During dewatering, the historic backfill materials may be transported out of the stopes by liquefaction and into the track drifts. There is a risk of impounded water being trapped in various places during the dewatering process.

The 3D as-built models have been built from available data and lack the finer details except in the more recently active mining areas, which may mean that not all of the drifts are represented in the 3D as-built models, leading to unexpected breakthroughs.

As the historic stopes have been modelled from sectional drawings, the exact stope profile between sections is uncertain. The actual location of the stope wall may be several metres from the location shown in the 3D as-built models. This presents both risks and opportunities on a stope-by-stope basis that the resource model has under/over reported the available resource in that area. The historic MCF stopes are planned to approach and sometimes breakthrough into the historic stopes as the mineral resource model indicates areas of good grade left in the immediate walls of these historic stopes. There is a risk that some of these historic MCF areas will not be mineable.

There is no visual distinction between ore and waste, which makes the grade control in the MCF stopes challenging and slows down the MCF stoping rates.

The largest risk to operating and capital costs is labour costs. Many operations are finding it challenging to attract and retain a skilled workforce. The PFS assumes that all operating development, stoping and capital lateral development is executed by company employees. There is a risk of increased labour costs if contractors are used to fill the gaps.

Procurement of critical items and equipment, like power distribution systems (transformers, unit substations, switchgear) and hoisting systems have long lead times that have fluctuated significantly over the past several years. This can impact both cost and schedule depending on demand at the time of execution.

There is an opportunity to extend the mine life and optimize the mine plan as the PFS mine plan is based upon a MRE with an effective date of December 31, 2021. The diamond drilling performed by Pure Gold up until closure in October 2022, plus the diamond drilling and test mining work completed by WRLG since acquiring the project, is not reflected in the PFS mine plan. A considerable amount of Indicated mineral resources that met the COG were not included in the PFS mine plan as these mining areas failed to pay for the required capital development to access them. Improving the continuity and expanding these mineral resources with additional information and/or conversion from Inferred to Indicated mineral resources can likely justify the capital development required to access some of these mining areas in future mine plans.

Gold price is a primary driver of project value. If the realized gold price varies from what was assumed in the financial analysis, project economics could be significantly affected (either negatively or positively).

2 Introduction

This report has been prepared by SRK Consulting (Canada) Inc. (SRK) on behalf of West Red Lake Gold Mines Ltd. (WRLG or the Company). The purpose of this report is to provide a technical report that documents all supporting work, methods used and results relevant to a prefeasibility study (PFS) on the Madsen Mine in Ontario, Canada, and that fulfills the reporting requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

2.1 Basis of Technical Report

This technical report uses the NI 43-101 reporting structure with Qualified Persons (QPs) responsible for the technical content. The report was compiled by SRK and includes contributions from other consultants as outlined in the study scope of work in Section 2.2.

The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This technical report is intended for use by WRLG subject to the terms and conditions of its contract with SRK. The Mineral Resource and Mineral Reserve estimates herein were prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines (CIM, 2019) and classified in accordance with CIM Definition Standards for Mineral Resources and Reserves (CIM, 2014).

2.2 Scope of Work and Responsibility

This technical report was compiled by SRK using contributions from its own consultants, as well as contributions from other consulting companies as listed below:

- Mining Plus – mine design and scheduling
- Allnorth Consultants Limited (Allnorth) – infrastructure
- Nordmin Engineering Ltd. (Nordmin) – shaft design and material handling
- T Engineering – backfill
- Fuse Advisors Inc. (Fuse) – metallurgy, processing, recovery
- Knight Piésold Consulting Ltd. (KP) – water and waste management

Table 2-1 shows the companies involved in preparing each section of this technical report.

Table 2-1: Report responsibilities

Report Section	Description	Responsibility
Section 1	Summary	All
Section 2	Introduction	SRK
Section 3	Reliance on Other Experts	SRK
Section 4	Property Description and Location	SRK
Section 5	Accessibility, Climate, Local Resources, Infrastructure, Physiography	SRK
Section 6	History	SRK
Section 7	Geological Setting and Mineralization	SRK
Section 8	Deposit Types	SRK
Section 9	Exploration	SRK
Section 10	Drilling	SRK
Section 11	Sample Preparation, Analysis and Security	SRK
Section 12	Data Verification	SRK
Section 13	Mineral Processing and Metallurgical Testing	Fuse
Section 14	Mineral Resource Estimates	SRK
Section 15	Mineral Reserve Estimates	SRK
Section 16	Mining Method	SRK
Section 17	Recovery Methods	Fuse
Section 18	Project Infrastructure	Nordmin, Allnorth, KP
Section 19	Market Studies and Contracts	SRK
Section 20	Environmental Studies, Permitting and Social or Community Impact	SRK, KP
Section 21	Cost Estimate	All
Section 22	Economic Analysis	SRK
Section 23	Adjacent Properties	SRK
Section 24	Other Relevant Data	SRK
Section 25	Interpretation and Conclusions	All
Section 26	Recommendations	All
Section 27	References	SRK
Section 28	Glossary	SRK

2.3 Qualified Person Responsibilities and Site Inspections

The QPs preparing this technical report are specialists in the fields of geology, exploration, mineral resource and reserve estimation and classification, geotechnical engineering, hydrogeology, underground mining, metallurgy and mineral processing, mine infrastructure, waste and water management, environment and permitting, capital and operating cost estimation, and mineral economics.

None of the consultants or any associates employed in the preparation of this report has any beneficial interest in WRLG. The consultants are not insiders, associates, or affiliates of WRLG. They are independent of WRLG. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between WRLG and the consultants. The consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The individuals listed in Table 2-2, by virtue of their education, experience, and professional association, are considered QPs as defined in the NI 43-101 standard for this report and are members in good standing of appropriate professional institutions. In addition to their contributions to the Summary (Section 1), Interpretation and Conclusions (Section 25), and Recommendations (Section 26) related to their areas of expertise, the QPs have specific report section responsibilities as summarized in Table 2-2. QP certificates of report authors are provided in Appendix A.

Table 2-2: QP responsibilities and site visits

Qualified Person	Company	Responsibility / Role	Site Visit	Report Section(s)
Cliff Revering, P.Eng.	SRK	Property Description, Accessibility/Climate, History, Geological Setting, Deposit Types, Exploration, Drillings, Sample Preparation, Data Verification, Mineral Resource Estimation, Adjacent Properties,	July 4 - 7, 2022	6, 7, 8, 9, 10, 11 (not including 11.1.6 or 11.3.6), 12 (not including 12.5 or 12.6), 14, 23
Sheila Ulansky, P.Geo.	SRK	Sample Preparation, QAQC	n/a; relied on Cliff Revering	11.1.6, 11.3.6, 12.6
Travis O'Farrell, P.Eng.	Fuse	Metallurgy, Processing, Recovery Methods, Cost Estimate	August 27 - 28, 2024	12.5, 13, 17, 21.2.6, 21.3.3
Stephen Taylor, P.Eng.	SRK	Mineral Reserve Estimation, Mining Methods, Cost Estimate, Economic Analysis, Other Relevant Data	October 9 - 11, 2024	2, 3, 4, 5, 15, 16.1 to 16.3, 16.5 to 16.8 (not including 16.6.6), 19, 21.1, 21.2 (not including 21.2.6 or 21.2.7), 21.3 (not including 21.3.3 or 21.3.4), 22, 24
Tim Coleman, P.Eng.	SRK	Geotechnical Engineering	April 6 - 7, 2022	16.4
Brian Prosser, P.E.	SRK	Ventilation	n/a; relied on Stephen Taylor	16.6.6
Guy Lauzier, P.Eng.	Allnorth	Project Infrastructure, Cost Estimate	February 13 - 14, 2024	18.1 to 18.5, 18.7

Qualified Person	Company	Responsibility / Role	Site Visit	Report Section(s)
Bernie Ting, P.Eng.	T Engineering	Backfill	n/a; relied on Stephen Taylor	18.6
Chris Dougherty, P.Eng.	Nordmin	Project Infrastructure (mine shaft and material handling)	July 1997, 1998, 2009, 2010	18.11
Daniel Ruane, P.Eng.	Knight Piésold	Project Infrastructure (Waste and Water Management), Environment (Closure), Cost Estimate	October 1, 2024	18.8 to 18.10, 20.5 (not including 20.5.1), 21.2.7, 21.3.4
Mark Liskowich, P.Eng.	SRK	Environment and Permitting	n/a; relied on Daniel Ruane	20.1 to 20.4, 20.5.1

2.4 Sources of Information

This report is based on information collected by the QPs during site visits and on additional information provided by WRLG throughout the course of mineral resource estimation and report preparation. Other information was obtained from the public domain. SRK has no reason to doubt the reliability of the information provided by WRLG. This technical report is based on the following sources of information:

- Discussions with WRLG on-site personnel and management
- Inspection of the site, including underground, surface facilities and drill core
- Review of exploration and historical mining data collected by WRLG
- Previous studies completed on the Project
- Test work completed during the course of this study by WRLG, previous owners or by the QPs or their designates
- Additional information from public domain sources

2.5 Effective Dates

The Effective Date of this technical report is January 7, 2025. The Effective Date of the Mineral Resource Estimate is December 31, 2021. The Effective Date of the Mineral Reserve Estimate is June 30, 2024.

2.6 Units of Measure, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or “metric”, except for Imperial units that are commonly used in industry (i.e., troy ounces (oz.) for the mass of precious metals, US gallons per minute (gpm) for water flow rates, etc.).

All dollar figures quoted in this report refer to Canadian dollars (CDN$, CAD$, C$ or $) unless otherwise noted.

This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.7 Declaration

The opinions of the QPs contained herein are based on information collected throughout the course of their investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be more or less favourable.

3 Reliance on Other Experts

In relation to the information contained in Section 4.1 and Section 4.2, the QPs relied on the opinion provided by Bennett Jones LLP on December 31, 2024 (Bennett Jones 2024). The QPs have not researched property title or mineral rights for the Madsen mine and express no further opinion as to the ownership status of this property.

The QPs have not independently verified the environmental, social and other permits required for WRLG operations as referred to in Section 4.3 and Section 4.4. The QPs relied on the information provided by WRLG in this regard, including:

- Amended Environmental Compliance Approval - TMF Discharge (9280-C2XNTQ)
- Environmental Compliance Approval - Dust, Air, and Noise Emissions (5217-BPXK2E)
- Permit to Take Water - #2 Shaft Dewatering (0202-AHJL45)
- Permit to Take Water - Process Pond (6834-BVBRSJ)
- Species at Risk - Myotis (RL-C-001-17)

4 Property Description and Location

4.1 Project Location

The Property is centered at 50.97° North latitude and 93.91° West longitude (UTM Projection NAD83, Zone 15 North coordinates 5646000N, 435000E) within the Baird, Heyson and Dome Townships of the Red Lake Mining District in northwestern Ontario (Figure 4-1).

Figure 4-1: Madsen Mine location map

Source: WRLG (2024)

4.2 Mineral Tenure

The Madsen Mine property comprises a contiguous group of 241 tenures consisting of 226 patented claims, three (3) mining leases, three (3) mining licences of occupation and nine (9) unpatented mining claims covering an aggregate area of 4,648 hectares in northwestern Ontario. Claim data is summarized in Table 4-1 and shown in Figure 4-2.

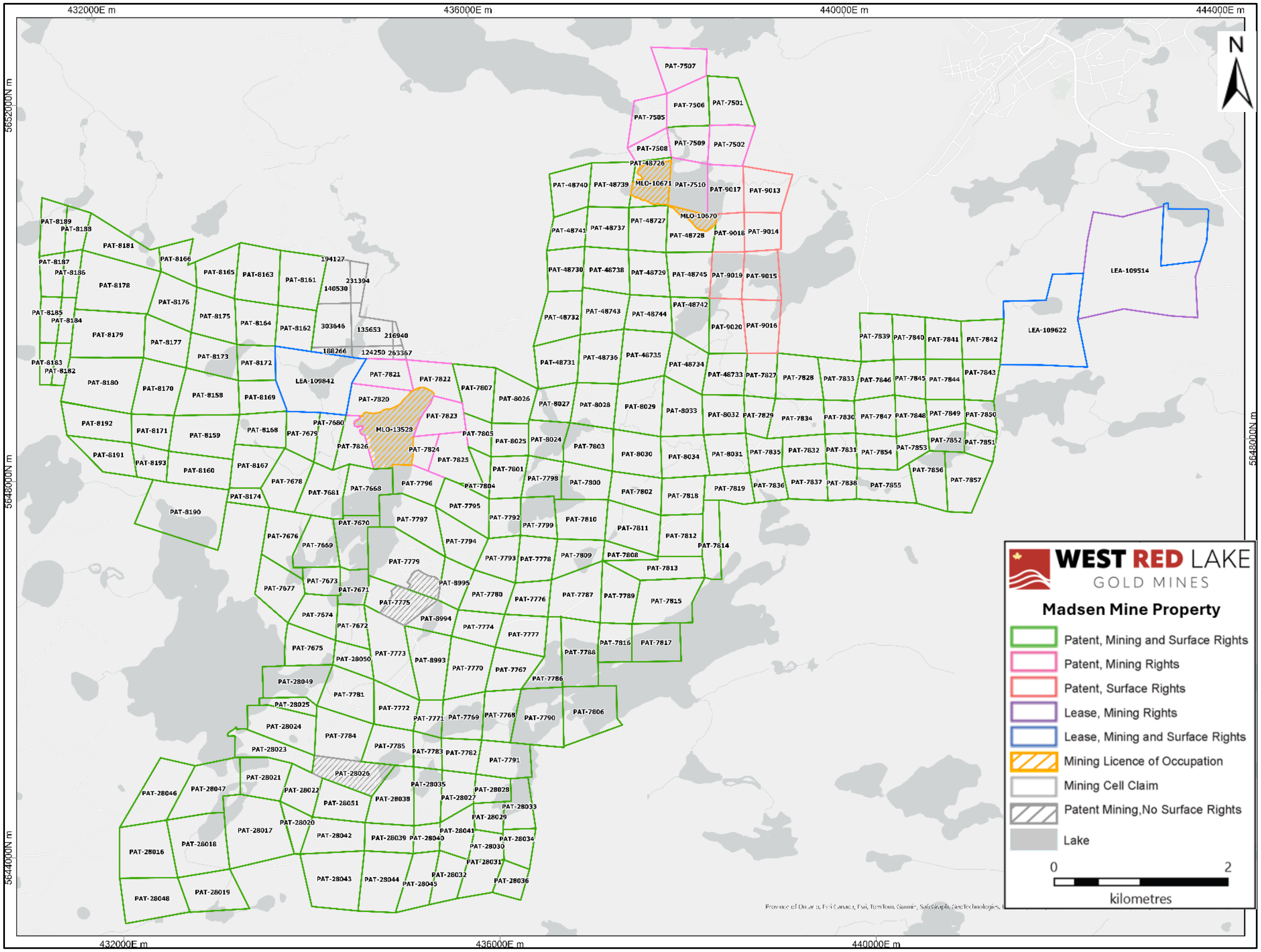
WRLG owns 100% of all mining leases, patents and unpatented claims comprising the mine property. Other than the royalties described in Table 4-2, the authors are unaware of any other royalties, back-in rights, payments or other agreements and encumbrances to which the property is subject.

Table 4-1: Madsen Mine property tenure

Claim No.	No. of Claims	Area (Ha)	Type		Claim No.	No. of Claims	Area (Ha)	Type
Madsen					**Hager**			
PAT-7767 - PAT-7826	60	1167.5	Patented		124250	1	5.8	Unpatented
PAT-8993 - PAT-8995	3	52.4	Patented		135653	1	14.3	Unpatented
MLO-13528	1	37.5	Patented		140530	1	13.7	Unpatented
Grouping Total	**64**	**1257.4**			188266	1	3.3	Unpatented
Starratt - Olsen					194127	1	0.3	Unpatented
PAT-28016 - PAT-28036	21	390.8	Patented		216940	1	1.7	Unpatented
PAT-28038 - PAT-28051	14	309.7	Patented		231394	1	6.6	Unpatented
Grouping Total	**35**	**700.5**			263367	1	1.8	Unpatented
Russet					303646	1	18.1	Unpatented
PAT-7668 - PAT-7681	14	257.3	Patented		LEA-109842	1	52.9	Leased
Grouping Total	**14**	**257.3**			**Grouping Total**	**10**	**118.5**	
Newman-Madsen					**Derlak**			
PAT-48726 - PAT-48745	20	388	Patented		PAT-8024 - PAT-8034	11	219.2	Patented
PAT-7501 - PAT-7502	2	39.4	Patented		**Grouping Total**	**11**	**219.2**	
PAT-7505 - PAT-7510	6	97.6	Patented		**Ava**			
PAT-9013 - PAT-9020	8	142.2	Patented		PAT-7839 - PAT-7857	19	294.8	Patented
MLO-10670 - MLO-10671	2	20.4	Patented		**Grouping Total**	**19**	**294.8**	
Grouping Total	**38**	**687.6**			**Killoran**			
Aiken					LEA-109514	1	108.4	Leased
PAT-8158 - PAT-8193	36	737.4	Patented		LEA-109622	1	95.9	Leased
Grouping Total	**36**	**737.4**			**Grouping Total**	**2**	**204.3**	
Mills								
PAT-7827 - PAT-7838	12	171.2	Patented		**GRAND TOTAL**	**241**	**4648.2**	
Grouping Total	**12**	**171.2**						

Source: WRLG (2025)

Figure 4-2: Madsen Mine tenure map



Source: WRLG (2024)

Table 4-2: Summary of royalty agreements on the Madsen Mine property

Claim No.	No. of Claims	Royalty Holder	Royalty
PAT-7668-7671, PAT-7674, PAT-7676-7681, PAT-8158-8193	47	Franco-Nevada Corporation	1% NSR to a maximum of C$1 million
PAT-7668-7671, PAT-7674, PAT-7676-7681, PAT-8158-8193	47	Canhorn Mining Corporation	1% NSR to a maximum of C$1 million
MLO-10670, MLO-10671, PAT-48726-48745, PAT-7501-7502, PAT-7505-7510, PAT-9013-9020	38	Sandstorm Gold Ltd.	0.5% NSR
MLO-10670, MLO-10671, PAT-48726-48745	22	Franco-Nevada Corporation	1.5% on first 1M oz-equiv; 2% on production beyond first 1M oz-equiv
PAT-7501-7502, PAT-7505-7510	8	My-Ritt Red Lake Gold Mines Ltd	3% NSR
PAT-9013-9020	8	Camp McMann Red Lake Gold Mine Ltd.	3% NSR
PAT-8024-8034	11	Fechi Inc.	3% NSR, 1% purchasable for C$1M
All claims on Madsen Property*	241	Sprott Resource Lending Corp.	1% NSR

Source: WRLG (2024)

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 must be carried out to maintain unpatented mining claims in good standing. Work credits exist on the unpatented claims that form a small part of the mine property.

Patented mining claims (“patents”) confer fee-simple rights to hard-rock mineral tenure and allow extraction and sale of minerals. Most of the 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 they were previously held by the Crown; if not, 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 done at the time of lease conversion. Leases do not require assessment work but are subject to annual rent.

The term expiry dates of WRLG’s current mining leases are January 31, 2035 (LEA-109514) and December 31, 2036 (LEA-109622).

4.3 Surface and Other Rights

Table 4-3 shows the surface rights ownership for mine property claims, patents and leases. WRLG owns surface rights as indicated in the table. Where WRLG does not hold surface rights, they 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. A single trapping tenure is held over the entire property and WRLG maintains good relations with the tenure holder. Several registered easements for highway and utility lines cross the property. The authors are aware of no other conferred rights on the Property.

Table 4-3: Summary of surface rights

Claim No.	No. of Claims	Disposition Type
PAT-28016-28036, PAT-28038-28051, PAT-48726 - PAT-48745, PAT-7501 - PAT-7510, PAT-7668 - PAT-7681, PAT-7767 - PAT-7819, PAT-7827 - PAT-7857, PAT-8024 - PAT-8034, PAT-8158 - PAT-8995, PAT-9013 - PAT-9020	218	Patent, surface, and mining rights
LEA-109842, LEA-109622	2	Lease, surface, and mining rights
124250, 135653, 140530, 188266, 194127, 216940, 231394, 263367, 303646	9	Crown retained surface rights
MLO-10670, MLO-10671, MLO-13528	3	Licence of Occupation, surface, and mining rights
LEA-109514	1	Lease, mining rights only
PAT-7820 - PAT-7826	7	Patent, mining rights only

Source: WRLG (2025)

4.4 Environmental Liabilities

WRLG acquired a legacy mine site with a history of almost a century of exploration and mining. A modern closure plan was created for the operation and additional funding was provided by Pure Gold in 2021. Pure Gold, and subsequently WRLG, have undertaken an extensive site cleanup that has seen significant amounts of waste removed from the site, security and road improvements and revegetation of inactive disturbed areas.

4.5 Permitting

All operational permits are in place for the mine and processing facility in production. The key operational permits are listed in Section 20.

4.6 Other Factors and Risks

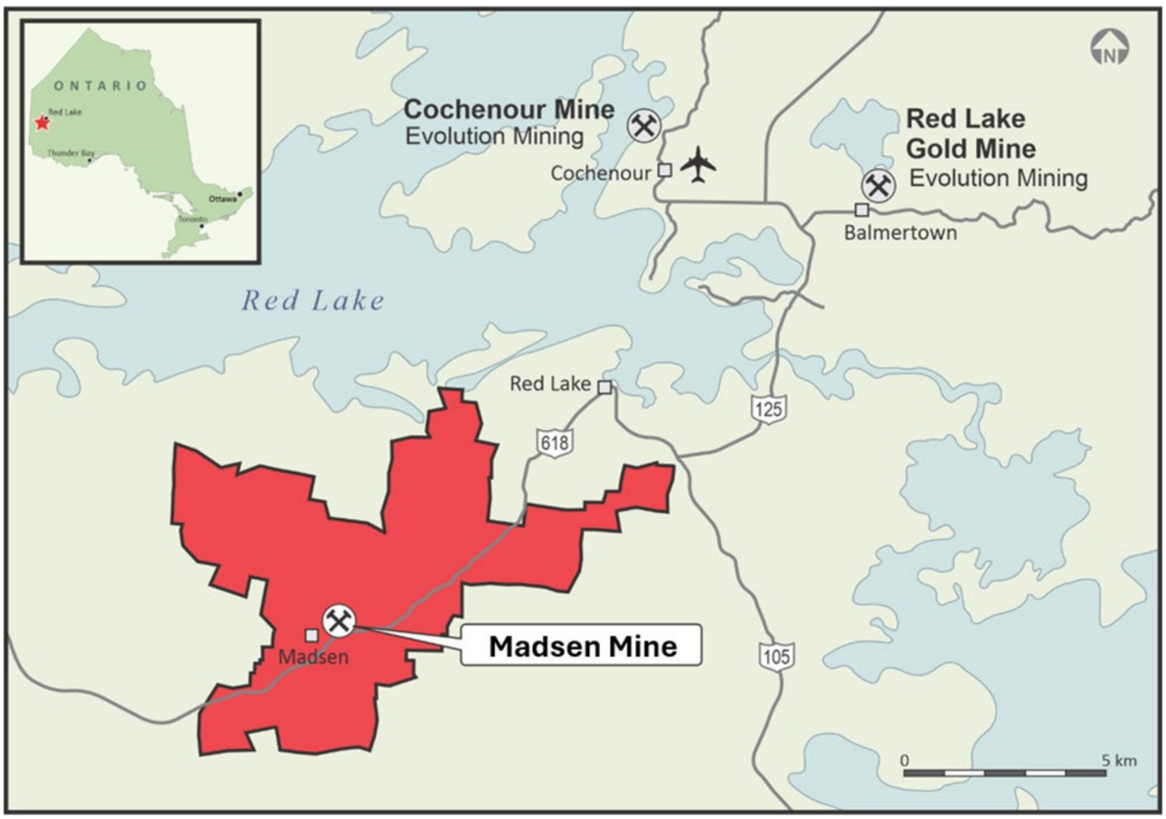
The authors are not aware of any other significant factors and risks that may affect access, title or the right or ability to perform work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Madsen Mine is located adjacent to the community of Madsen, within the Red Lake Municipality of northwestern Ontario, approximately 565 km by road (430 km direct) northwest of Thunder Bay, Ontario, and approximately 475 km by road (260 km direct) east-northeast of Winnipeg, Manitoba. Red Lake can be reached via Highway 105 from Trans-Canada Highway 17 (Figure 4-1). Red Lake is also serviced with daily flights from Thunder Bay and from Winnipeg by Bearskin Airlines.

The mine is accessible from Red Lake via Highway 618, a paved secondary road maintained year-round by the Ontario Ministry of Transportation (Figure 5-1). The mine is 10 km southwest of the town of Red Lake. A series of intermittently maintained logging roads and winter trails branching from Highway 618 provide further access to other portions of the mine property.

Figure 5-1: Property location and main access routes



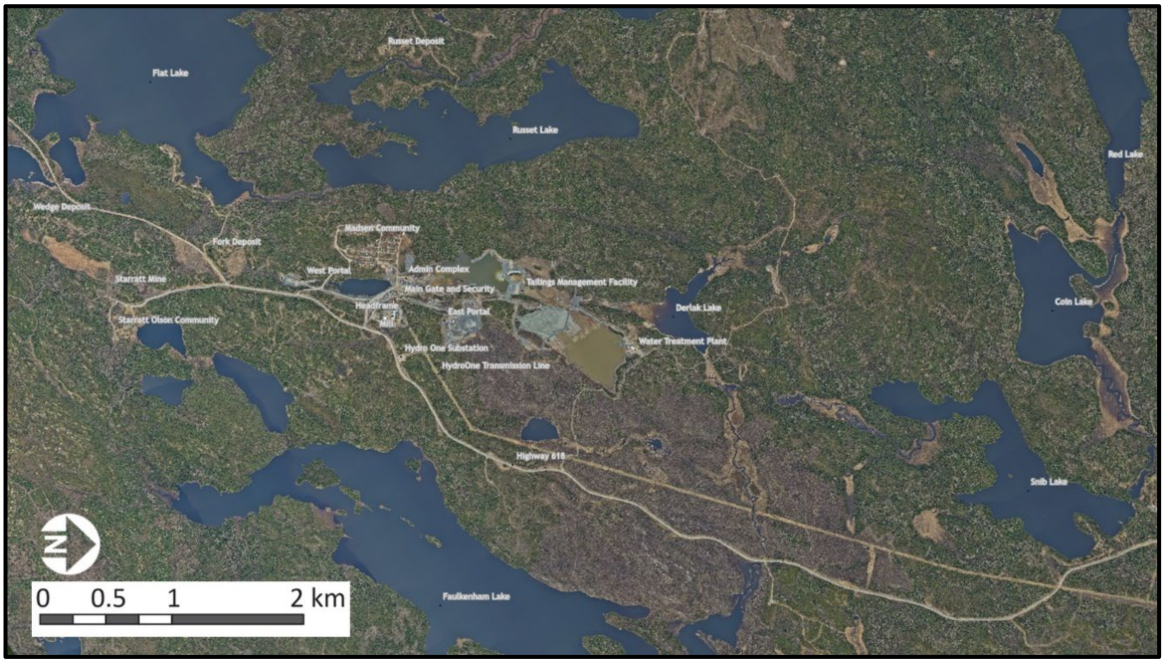
Source: WRLG (2024)

5.1 Local Resources and Infrastructure

Red Lake Municipality, with a population 4,094 (Statistics Canada 2021 Census), comprises six communities: Red Lake, Balmertown, Cochenour, Madsen, McKenzie Island and Starratt-Olsen. Mining and mineral exploration is the primary industry in the area. 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 Madsen Mine staff live and will live in the surrounding communities and out-of-town employees stay in local accommodations in Red Lake.

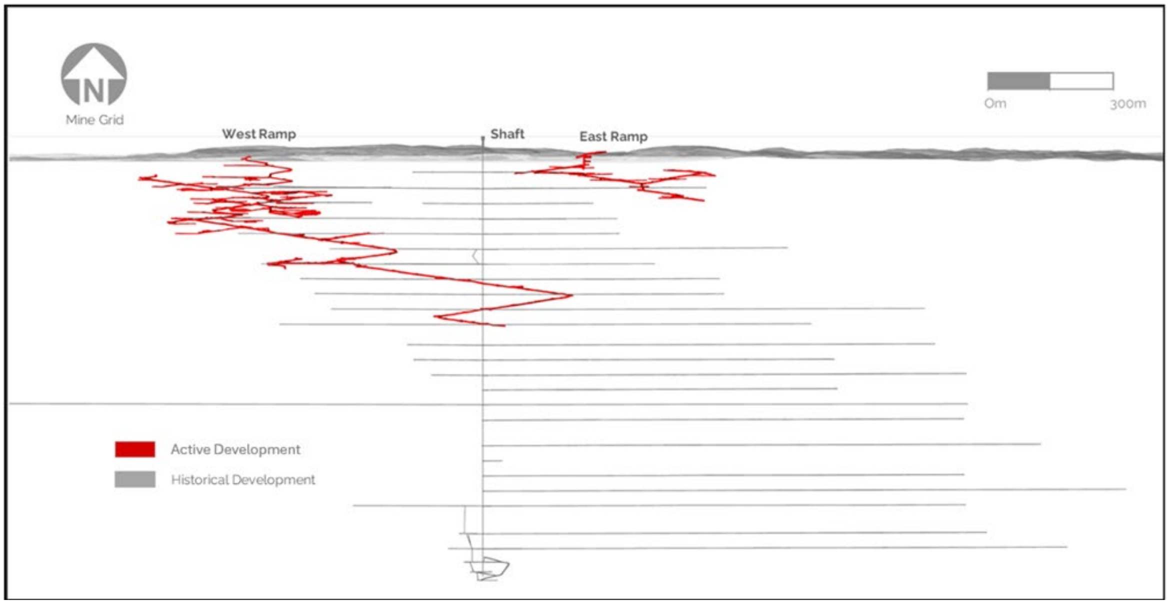
Major infrastructure at the Madsen Mine includes paved highway and secondary road access, ample fresh water supply, hydroelectric power from the provincial grid, an operational processing and tailings facility, two underground access portals and ramps, a 1,273 m shaft and significant underground development along with supporting ancillary surface facilities (Figure 5-2; Figure 5-3).

Figure 5-2: Surface infrastructure overview



Source: WRLG (2024)

Figure 5-3: Underground infrastructure overview



Source: WRLG (2024)

5.2 Physiography and Climate

The terrain within the mine property is gentle to moderate with elevations ranging from 360 to about 430 m above sea level. Topography is dominated by glacially scoured southwest-trending ridges, typically covered with jack pine and mature poplar trees. Swamps, marshes, small streams and small to moderate-size lakes are widespread. Rock exposure varies, but rarely exceeds 15% at ground surface and averages between 5% and 10%. Glacial overburden depth is generally shallow, rarely exceeding 20 m, and primarily consists of ablation till, minor basal till, minor outwash sand and gravel, and silty-clay glaciolacustrine sediments. Vegetation consists of thick second growth boreal forest composed of black spruce, jack pine, poplar and birch.

The climate in the Red Lake area is described as warm-summer humid continental (climate type Dfb according to the Köppen climate classification system). Mean daily temperatures range from -18°C in January to +18°C in July. Annual precipitation averages 70 cm, mainly occurring as summer showers, though including a total of about 2 m of snow. Snow usually starts falling during late October and starts melting during March but is not normally fully melted until late April. Late-season snow in May does occur.

Fieldwork and drilling are possible year-round on the property, although certain wetter areas are more easily accessible in the winter when frozen.

6 History

Gold was originally reported in the Red Lake area in 1897 by R.J. Gilbert of the North West Development Company. Intensive exploration of the district followed discovery in 1925 of the gold showings that eventually formed part of the Howey Mine (Lebourdaix, 1957).

Since 1927, a total of 28 mines have operated in the Red Lake Mining District, producing 29 million ounces of gold at an average recovered grade of 15.6 g/t Au. Approximately 89% of this gold was produced from two mine complexes: Red Lake Mine and Madsen (formerly PureGold) Mine (Malegus et al., 2022).

The exploration and mining history of the Madsen Mine is tabulated in Table 6-1.

The mine property can be divided into five claim groups with separate histories of mining and exploration prior to amalgamation with the mine property over the past forty years: Madsen, Starratt (acquired in 1980), Russet (timing unknown, but acquired between 1989 and 1997), Newman-Madsen (2014) and Derlak (2017). The following sections describe the exploration and mining work carried out by other operators during each main stage of property amalgamation:

- From 1925 until 1980 when the Madsen and Starratt mine properties were combined
- From 1980 until 1998 when Claude acquired the Madsen, Starratt and Russet properties
- From 1998 until 2014 when Laurentian (subsequently renamed Pure Gold) purchased the project and amalgamated the Newman-Madsen claims in 2014 (the Derlak property was added to the property in 2017)
- From 2014 to 2023 by Pure Gold after acquisition of the mine property in 2014

Table 6-1: Exploration and mining history of the Madsen Mine Project

Year	Activity
1925	Gold discovered at Red Lake
1927	First claims staked in Mine area
1935	Madsen Red Lake Gold Mines incorporated; No. 1 shaft sunk to 175 m
1936	Discovery of Austin Zone
1937	Madsen No. 2 shaft sunk to access Austin Zone. Ultimately reaches to 1,273 m with 27 levels
1938	Madsen mill facility initiates 36 years of continuous production
1948	Starratt-Olsen mine opens with production for 8 years
1956	Production halted at Starratt-Olsen mine. Total production of 823,554 tonnes at a recovered grade of 6.19 g/t (163,990 oz Au)
1969	Discovery of the 8 Zone located between levels 22 and 27 of the Madsen Mine
1974	Production halted at Madsen Mine. Total production of 7,593,906 tonnes at a recovered grade of 9.91 g/t (2,416,609 oz Au)
1974	Madsen operation sold to Bulora Corporation
1976	Bulora Corporation files for bankruptcy
1980	E.R. Rowland acquires Madsen and Starratt properties
1990	Red Lake Buffalo Resources acquires Madsen and Starratt properties from Rowland estate; changes name to Madsen Gold Corp. in 1991

Year	Activity
1997	Madsen Gold Corp. commences mining and milling at Madsen after moving Dona Lake mill to Madsen. Production of 8,350 ounces gold
Prior to 1998	Madsen Gold Corp. acquires Aiken and Russet claims and amalgamates with the Madsen and Starratt properties (collectively referred to hereinafter as the Madsen Gold Project)
1998	Claude purchases Madsen Gold Corp. and commences dewatering Madsen workings and mining from the Madsen shaft. Production of 8,930 ounces gold
1998–2000	Claude drills 230 holes (~21,000 m) on the Madsen Gold Project
2001	Placer Dome options the Madsen Gold Project and stops dewatering
2001–2004	Placer Dome drills 115 holes (60,725 m) on the Madsen Gold Project, most on targets outside the Madsen and Starratt mine areas. Discovers Fork and Treasure Box zones
2002	Wolfden acquires the Newman-Madsen Property and explores it in joint venture with Kinross (2002-03; 17 holes; 4,193 m) and Sabina (2004-2011; 48 holes; 18,684 m)
2006	Placer Dome exits Madsen Gold Project, returning it 100% to Claude
2007–2013	Claude drills 346 holes (198,913 m) on the Madsen Gold Project, including >200 holes (>80,000 m) on targets outside the Madsen Mine itself. Dewaters from level 6 (2007) to level 16 (2010) and below; pumping halted in 2013
2012	Sabina purchases 100% interest in Newman-Madsen Property and issues 0.5% NSR to Premier Gold Mines Limited
2012	Sabina drills 13 holes (4,332 m) on Newman-Madsen Property
2014	Laurentian Goldfields Ltd. purchases the Madsen Gold Project from Claude, later renamed PureGold Mine
2014	Laurentian, renamed Pure Gold, purchases the Newman-Madsen Property from Sabina, and amalgamates it into the Mine Property. SRK restates 2009 resource
2014–2018	Pure Gold drills 904 core holes (210,645 m) on the Mine Property
2016	Nordmin completes positive Preliminary Economic Assessment (PEA) for Pure Gold
2017	Pure Gold opens the West Portal and initiates ramp reconditioning
2017	Pure Gold initiates permitting study and environmental baseline work
2017	Pure Gold purchases the Derlak property from Orefinders Resources Inc. and merges it into the Mine Property
2017	Pure Gold completes new resource estimate and positive PEA
2017	Pure Gold completes first Mineral Resource Estimates for Russet South and Fork deposits
2018	Pure Gold completes first Mineral Resource Estimate for the Wedge deposit
2018	Pure Gold completes underground mining and bulk sample program from West ramp
2019	JDS completes positive Feasibility Study for Pure Gold
2019	Pure Gold announces production decision and begins construction on the PureGold Mine and shaft dewatering
2020	Pure Gold commences mining and milling operations; first gold pour
2021	Pure Gold declares commercial production
2022	Pure Gold announces disclosure of an updated Mineral Resource Estimate with an effective date of December 31, 2021
2023	WRLG acquires Pure Gold Mining Inc.

Source: WRLG (2024)

6.1 Early History

6.1.1 Madsen

The first claims staked in the Madsen area date back to 1927, with no work from this period recorded. Marius Madsen staked part of the Madsen Property in 1934 and Madsen Red Lake Gold Mines was incorporated in 1935 (Leduc and Sutherland, 1936). Early prospecting uncovered several gold showings in the area. The work initially focused on an auriferous quartz vein hosted by felsic volcanic rock on claim 11505 near High Lake. The No. 1 shaft was sunk to a depth of 175 m on this zone, and four levels were developed. In 1936, Austin McVeigh located a gold-bearing zone (later determined to be part of the McVeigh deposit) on the northern edge of what is now the Process Water Pond. Drilling on this and an adjacent zone in late 1936 delineated the important Austin zone. The underground development of the Madsen No. 2 shaft commenced in 1937 with the sinking of a three-compartment shaft to a depth of 163 m. The shaft eventually reached a depth of 1,273 m with 24 shaft-accessible levels and three additional ramp-accessible levels totaling 27 underground levels (~67 linear km) of development. The original Madsen mill began operating in August 1938 (Brown and Crayston, 1939) and operated continuously until 1974.

The operation was sold to Bulora Corporation in 1974 and was acquired by E.R. Rowland in 1980.

6.1.2 Starratt and Wedge

The Starratt-Olsen Mine, located approximately 2.2 km southwest of the PureGold Mine, operated from 1948 through 1956 and produced approximately 163,990 ounces of gold from 823,554 metric tonnes at an average recovered grade of 6.19 g/t Au (Malegus et al., 2022).

The original staking and prospecting in the Starratt and Wedge areas dates back to 1926 and 1927, soon after gold was discovered in Red Lake (Kilgour and de Wet, 1948). Only minor work was completed at the time and the claims were allowed to lapse. Claims were staked by David Olsen and R.W. Starratt in 1934 and optioned by Val d'Or Mineral Holdings (Val d'Or) in 1935 (Ferguson, 1965). The early exploration focused on three showings termed the Olsen (OL Zone), De Villiers (DV Zone), and Starratt. Trenching at the OL Zone was carried out by Hollinger Consolidated Gold Mines in 1934 with a total of six trenches that returned up to 19.5 g/t Au over approximately 1 m. The property was optioned by Val d'Or, who tested the OL Zone with three diamond drill holes and 34 trenches totalling 295 m, though the trenching returned erratic values and drilling returned low values over narrow widths (Ferguson, 1965).

Early work carried out by Val d'Or at the DV and CK Zones in 1937 consisted of 21 trenches with a total length of ~125 m (samples returned up to 127.9 g/t Au), 1,766.1 m of diamond drilling in 24 drill holes and the sinking of a 13 m shaft with a 5.2 m crosscut at 9.8 m depth (Holbrooke, 1963; Ferguson, 1965; Ferguson et al., 1971). The DV Zone was originally named after a Val d'Or prospector named W. de Villiers who worked the area in the 1930s. Eight additional holes were drilled during the 1940s to test the DV Zone (Panagapko, 1998; 1999)

Initial exploration at Starratt carried out in the late 1930s included 32 diamond drill holes for 2,109.8 m drilled whereas six holes for 184.8 m were drilled to test the OL Zone (Holbrooke, 1963).

Faulkenham Lake Gold Mines Limited held an option on the Property from Val d'Or between 1938 and 1939 and sunk a shaft to 175 ft at Starratt. Hasaga Gold Mines ('Hasaga') then obtained the property and changed the name to Hasaga No. 2 property (Ferguson, 1965). In 1945, Starratt-Olsen Gold Mines was incorporated to take over the property, however Hasaga retained a 50% interest and directed operations (Ferguson, 1965).

Between 1940 and 1944, the property sat idle due to the Second World War, after which exploration resumed with a drilling campaign in 1944 to 1945 with 52 drill holes (for 6,443.2 m) over a strike length of 600 m, with an additional 792 m of underground drilling (Ferguson, 1965). This successful program delineated mineralization that defined the mine reserves and lead to the incorporation of Starratt-Olsen Gold Mines Limited (Panagapko, 1999). Mine development and operations were carried out between 1948 and 1956 (Holbrooke, 1963), including development on the 800, 1000, 1150 and 1475 levels in the CK Zone and extended on the 1475 Level westward to the OL Zone. Production at the Starratt mine ceased in 1956 when all known reserves were exhausted. The company name was changed to Starratt Nickel Mines Limited in 1957.

Minimal work was carried out on the Starratt-Olsen claims between 1958 and 1998 (Holbrooke, 1963). Two diamond drill holes totalling 193.5 m drilled to test the DV Zone in 1961, followed by 19 holes (SN63-01 to SN63-06 and SN64-01 to SN64-13) for 4,104 m in 1963-64 by Dickenson Mines Limited ('Dickenson') that mainly tested the DV Zone and includes three holes drilled to the west at the OL Zone (Holbrooke, 1963; Panagapko, 1998). Highlights from the Dickenson drilling include 6.25 g/t Au over 3.8 m and 4.88 g/t Au over 12.2 m. These results were not followed up on by Dickenson (Gow, 1989), however Dickenson purchased the property in 1965. E.R. Rowland acquired the property in 1980 and amalgamated it with the Madsen Property.

6.1.3 Russet

The Russet Red Lake Syndicate was formed in 1936 and acquired eight patented mining claims in the southern part of the Russet Lake area and completed limited prospecting work. Russet Red Lake Gold Mines was incorporated in 1943 and acquired the Syndicate's claims and six additional patent claims. Exploration by Russet Red Lake Gold Mines commenced in 1944 with trenching and 24 short holes on Claims 19181 and 19235 just west of Russet Lake (Crayston and McDonough, 1945). This work focused on a seemingly complexly folded zone of iron formations hosted by mafic volcanic rock that crops out on Claim 19235. Work then shifted about 350 m to the east to explore another zone of gold mineralization hosted by altered mafic volcanic rock near the western contact of the Russet Lake Ultramafic. In 1946 and 1947, a total of 105 shallow holes tested both the Main zone and the No. 3 Zone near Russet Lake (Panagapko, 1999), after which the property remained idle until it was amalgamated with the Aiken ground to the west in 1965.

Aiken Red Lake Gold Mines Limited was incorporated in 1945 and acquired 36 patented mining claims previously held by several smaller prospecting syndicates. Work in 1945 consisted of prospecting, trenching and core drilling on the No. 1 and No. 2 veins located on Claims 18728 and 20585, respectively (Ferguson, 1965). No further work was conducted on the property until it was merged with the Russet South property to the east in 1965.

International Mine Services carried out a three-hole drilling program in the No.3 zone area in 1966 (Kuryliw, 1968a). A further 21 holes were completed on the Russet mineralized zones in 1968, based on a geological and structural re-interpretation (Kuryliw, 1968b).

Five holes in 1969 tested the stratigraphy south of the No.3 zone (Panagapko, 1998). During the winter of 1974, a 22-hole program was completed in the No.3 zone area (Tindale, 1974).

Following up on an electromagnetic anomaly identified from an airborne magnetics and electromagnetics survey carried out by Madsen Gold Mines in 1971, bulldozer-trenching, line-cutting, geological mapping, magnetometer survey, electromagnetic EM-17 horizontal loop survey and chip sampling were conducted in the fall of 1974 (Kuryliw, 1975; Tindale, 1975a, b).

One hole was drilled in the northern part of the property in 1977 to test an EM conductor (Tindale, 1977).

6.1.4 Newman-Madsen

Coin Lake Gold Mines Ltd. (“Coin Lake”) acquired the property historically referred to as My-Ritt from Red Lake Bay Mines Ltd. in 1936. Coin Lake completed an intensive program of stripping and trenching from 1936 to 1939 (Chastko, 1972). During this time, a magnetometer survey was completed and at least 22 holes were drilled.

Between 1943 and 1946, Cockeram Red Lake Gold Mines completed a total of 35 diamond drill holes (5,674 m), testing for gold mineralization along strike from the Madsen Mine (Durocher et al., 1987). Results from these drilling programs are not available. Central Patricia Gold Mines Ltd. drilled an additional 14 core holes in 1943 (Durocher et al., 1987).

An area south of Coin Lake was held as part of a land package owned by Rajah Red Lake Gold Mines Ltd in the mid-1950s. In 1957, the company’s charter was cancelled and ownership of the Heyson Township claims was transferred to H.A. Newman. The only recorded work on the Heyson Township claims consists of geological and magnetometer surveys completed in 1959 (Howe, 1960). Mespi Mines Ltd. also completed an aeromagnetic survey over the area in 1959.

Assessment file records are scarce for the time period between 1959 and 1971 but it is known that My-Ritt Gold Mines Ltd. held the property at some point during this time period.

In 1971, Cochenour-Willans Gold Mines Ltd. obtained the property from My-Ritt Gold Mines Ltd. and completed VLF-EM, IP, and soil geochemical surveys, followed by three core holes totalling 528 m (Chastko, 1972). However, the exact location of these holes is unknown and results are unavailable.

6.1.5 Derlak

The following information on Derlak is taken from Durocher et al. (1987). The earliest records on the Derlak property indicate that stripping, trenching, magnetometer surveying and diamond drilling were completed by Derlak Red Lake Gold Mines Limited in 1936–1937. Nine holes (~518 m) tested approximately 500 m of strike length along a porphyry dyke. Mineralized shear zones associated with the dyke contact had a maximum width of 12 m and low gold values.

In 1944, Derlak Red Lake Gold Mines Limited drilled another eight diamond drill holes testing below the same zones without success.

Madsen Red Lake Gold Mines Limited optioned Derlak and drilled 13 holes in 1967 with a maximum assay of 2.3 g/t Au.

6.1.6 Fork

Prior to the discovery of the Fork deposit and subsequent definition drilling conducted by Placer Dome and Claude between 2002 and 2009, only minor surface and underground exploration work was completed in the vicinity of the deposit. Between 1936 and 1944 a series of short drill holes tested the southwestern extension of the Madsen Mine Trend towards the Starratt mine property (Panagapko 1998). Several drill holes from these programs encountered altered rock and quartz veins as well as localized brittle and ductile deformation zones within the Fork deposit area. In the late 1950s underground drilling from the 16-Level of Madsen Mine intersected what is now interpreted to be the down dip projection of the Fork deposit. Drill-hole logs report alteration and mineralized intercepts returning 8.23 g/t Au over 0.85 m and 21.26 g/t Au over 1.53 m. Several follow up fans of drill holes were completed into the altered zone, but no further work was reported until the early 2000s.

6.1.7 Faulkenham

Faulkenham Lake Gold Mines Limited explored and developed a gold showing approximately 1 km southwest of the Starratt-Olsen mine during 1936 & 1937, with a shaft sunk to a vertical depth of 344 feet and developed on three levels. No commercial production is recorded from this site, but mined material was left in an ore dump on surface near the shaft site. Three mineralized veins are reported to exist in the area, with the main vein delineated over a strike length of 900 feet, striking 107° with a dip of 85° towards the north. Vein width was variable, pinching and swelling with a maximum reported width of 24 inches. Maximum grade reported for the vein was 0.362 ounces per ton over a width of 16.75 inches (Horwood, 1940).

6.2 1980 – 1998

6.2.1 Madsen / Starratt

E. R. Rowland controlled the combined Madsen-Starratt property from 1980 to 1988 when Red Lake Buffalo Resources acquired the ground from his estate. Under an option agreement, Noranda Exploration (Noranda) carried out mapping and core drilling between 1980 and 1982 (Noranda Exploration Company Limited, 1982). On the Starratt claims, Noranda’s 11 holes focused on the down-dip extension of the De Villiers vein. Three of these holes hit significant gold mineralization including an interval returning 16.46 g/t Au over 1.55 m.

Red Lake Buffalo Resources was reorganized into Madsen Gold Corp. (‘Madsen Gold’) in 1991. Madsen Gold drilled 29 holes (2,480 m) at Starratt in 1998 (Panagapko, 1998).

Madsen Gold purchased a mill from the exhausted Dona Lake Mine, transported it to Madsen and erected it at the current site. Production commenced in June 1997 with 8,350 ounces produced for the year (Blackburn et al. 1998).

6.2.2 Wedge

In 1981, E.W. Rowland optioned the property to Noranda, now host to the Wedge deposit. Noranda conducted geological mapping and diamond drilling over ~170 m of strike length to a depth of ~150 m in nine holes for 1,332.9 m, which focused on the down-dip extension of the DV Zone mineralization. Three holes intersected significant mineralization, up to 15 g/t Au over 1.5 m (Noranda Exploration Company Limited, 1982; Panagapko, 1998; 1999). Despite intersecting encouraging mineralization, Noranda encountered problems intersecting mineralization comparable to that reported from earlier Dickenson drilling, which was apparently due to Noranda surveying their holes using the Madsen Mine Grid, whereas Dickenson used the Starratt Mine Grid (Gow, 1989). Perhaps because of this, Noranda did not recognize a clear correlation between the various phases of work carried out in the DV Zone (Gow, 1989). Furthermore, Noranda's drilling appears to have undercut the mineralization encountered in the Dickenson drilling.

6.2.3 Russet

In 1985, Aiken-Russet Red Lake Mines Ltd. was amalgamated with several other companies to form Canhorn Mining Corporation. The following year, an airborne electromagnetic survey covering the entire Aiken-Russet property was carried out by Aerodat Ltd and outlined several conductors. Additional work in 1986 included line cutting, ground magnetometer and VLF surveys and limited field examinations before the property was optioned to United Reef Petroleum Ltd. (Butella and Erdic, 1986). United Reef Petroleum carried out an exploration program on the Russet property in 1987 and 1988 (Siriunas, 1989) which included airborne and ground geophysical surveys and a 78-hole drilling program. The majority of the drilling focused on the Russet Main and No.3 zones, but drilling was also directed at various other targets on the property.

The Russet property was acquired by Red Lake Buffalo Resources or Madsen Gold prior to 1998 and combined with the Madsen and Starratt properties.

6.2.4 Newman-Madsen

Between 1981 and 1982, Noranda Inc. completed four holes of unknown length in the central part of the Newman-Madsen claims. The location, orientation and results of the drilling are unavailable. No further exploration on the property was reported until 2002, when the property was acquired by Wolfden Resources Corporation (Wolfden).

6.2.5 Derlak

Selco Inc. optioned the Derlak property and completed geological mapping, magnetometer, VLF-EM, HLEM surveys and six diamond drill holes in 1980–81 (Pryslak and Reed, 1981). No significant gold mineralization was located.

The property was reportedly optioned by Seine Explorco Ltd. in 1981 and by Redaurum Red Lake Mines Ltd. in 1985 but the reports have not been located.

Placer Dome optioned the property in 1997 and undertook IP, magnetometer, geological mapping, and rock sampling surveys (Blackburn et al., 1999). Twelve rock samples (probably selective grab samples) exceeded 10 g/t Au on the western part of the property, including a quartz vein that returned 370 g/t Au. Placer Dome drilled four holes on the property in 1998, intersecting weak quartz-carbonate veining in shear zones without significant gold values.

6.3 1998 – 2014

6.3.1 Madsen / Wedge / Starratt / Russet South

After acquisition of the mine property from Madsen Gold in April 1998, Claude began mining portions of the McVeigh and Austin deposits with access from the West portal and ramp and eventually conducted exploration drilling campaigns across the Madsen, Wedge, Starratt and Russet areas.

In 1998, Madsen Gold / Claude extracted 85,417 tonnes from Madsen, of which 81,740 tonnes were milled for a total production of 8,930 ounces of gold at an average recovered grade of 3.43 g/t Au (Blackburn et al., 1999). Mill recovery was estimated to be 86.75%, with a head grade of 3.91 g/t Au. Mining occurred within the Austin zone between Levels 2 and 5 of the mine and in the McVeigh zone between surface and 2 Level.

Information available for the final seven months ending October 1999 indicate a mill throughput of 99,726 tonnes at a diluted gold grade of 4.39 g/t Au. Reconciliation revealed a significant grade variance, ascribed to excessive mining dilution (Olson et al., 1999).

The mine and mill complex were put on care and maintenance in October 1999. Total recorded production for the historical Madsen Mine, inclusive of that produced by Claude, during the periods 1938 to 1974 and 1998 to 1999, is 7,872,679 metric tonnes at an average recovered grade of 9.69 g/t Au for a production of 2,452,388 ounces of gold (Malegus et al., 2022).

Following acquisition of the property in 1998, Claude compiled all historical geophysical, geological, geochemical and drilling data on the Mine Property (Panagapko, 1998). As part of their surface exploration work, Claude conducted an IP survey over the southwestern portion of the deposit area, consisting of 11.7 line-km of reconnaissance gradient array surveying, 1.8 line-km of follow up gradient array surveying and 2.3 line-km of follow up pole dipole surveying (Warne et al, 1998). This survey successfully outlined resistivity and chargeability anomalies interpreted to be related to silicification and sulphide mineralization, respectively (Panagapko, 1999).

Between 1998 and 2000, Claude evaluated several near surface targets including the McVeigh West, De Villiers and No. 1 shaft zones (Panagapko, 1999). This involved mapping, stripping, trenching, limited test-mining and drilling of 133 holes. Table 6-2 summarizes the extent and distribution of drilling on the Mine Property, between Claude's purchase of the property in 1998 and acquisition by Pure Gold in 2014.

Table 6-2: Distribution of 1998 - 2013 drilling on the Mine Property

Operator	Drilling	Area						Total
		Madsen	Starratt	Fork	Russet	Treasure Box	Other	
Claude (1998-2000)	holes	85	33	-	-	-	15	133
	metres	6,417	na	-	-	-	1,296	7,713
Placer Dome (2001-2005)	holes	12	9	16	6	49	6	98
	metres	15,244	4,830	6,160	3,653	24,356	4,315	58,558
Claude (2007-2013)	holes	108	35	105	5	51	10	314
	metres	93,883	19,344	45,179	3,121	13,573	7,439	182,539
Total (1998-2013)	holes	205	77	121	11	100	42	556
	metres	115,598	24,174	51,339	6,774	37,929	13,051	248,865

Source: D. Baker (2017)

At the McVeigh West area, approximately 750 m west of the No. 2 Shaft, 80 surface holes explored several new zones of gold mineralization extending to at least 90 m below surface. Exploration drilling in the 2-11N and 2-13N raise areas of the McVeigh Zone confirmed the presence of gold-bearing lenses above the known workings on the second level.

The surface expression of the No. 1 Shaft quartz vein system was stripped, mapped and channel sampled, delineating four mineralized lenses on surface. Three benches were mined for approximately 7,920 tonnes of vein and wall rock. An additional waste stockpile of 5,440 tonnes was generated with a reported average grade of 4.83 g/t gold. Fifteen holes were drilled on the No. 1 Shaft vein and several decimetre-scale zones of gold mineralization were intersected. Most holes, however, encountered either minor or no veining at all.

At the DV Zone, a ~150 m x 10-20 m area was cleared between departures 8400E and 8900E, centred along 6090N SMG, which also uncovered the historical shaft at 8490E-6060N. Claude collected 101 grab samples from the stripped area with 40 samples returning greater than 3.1 g/t Au, including 38 samples collected over 33.5 m to the east of the shaft returning an average of 9.55 g/t Au using a capped gold value of 31.25 g/t. Following these positive results, Claude began test mining of the DV Zone by removing 2,940 tons from two benches. The results of the first bench cut concluded that the gold-bearing veins exposed at surface were faulted off 1 m below surface, however quartz veining was present ~30 m from the western end of the trench and continued over 43 m to the east. Waste slashes were taken along the north wall before the second bench was mined to centre the second bench on the vein. Face samples collected from the first bench returned an average 6.5 g/t Au over 2.5 m from 19 samples over 48.8 m (Panagapko, 1999). Claude subsequently conducted two phases of drilling at DV for 29 holes.

Work carried out at the CK Zone in 1998 began with rock sampling of historical trenches, which returned several high-grade results. Mechanical stripping and detailed mapping were carried out and 34 grab and channel samples were collected. The highest value returned from the east end of the trench was 3.34 g/t Au, whereas nine select grab samples of vein material from the western end of the trench returned an average of 38.2 g/t Au. Four holes were drilled to test for continuity of mineralization at depth, however assays only returned weakly anomalous results (Panagapko, 1999). No additional work was carried out in the area until 2003.

In 2001, Claude granted Placer Dome an option to earn 55% of the Mine Property. Placer Dome failed to complete the option requirements and Goldcorp returned the property to Claude in September 2006 following their acquisition of the Placer Dome Red Lake assets.

Most of Placer Dome's efforts (information taken from Crick, 2003; Dobrotin, 2002, 2003, 2004a, b; Dobrotin and Landry, 2001; Dobrotin and McKenzie, 2003) were directed at drilling the Madsen Mine at depth and other broad property-scale targets. Surface mapping and geochemistry and a 45 km^2 airborne magnetic/gravity survey were also completed.

From 2001 to 2005, Placer Dome drilled 98 holes (Table 6-2) to test the footwall stratigraphy of the main auriferous zones within a mafic-ultramafic sequence up-dip of various targets on the property, including: 8 Zone, Starratt, Treasure Box, Russet, and Fork, among others. Several zones of anomalous gold mineralization were encountered and several of these areas remain as high priority targets. Mobile metal ion and conventional soil sampling to the north, west and around Russet Lake in 2001 outlined five relatively small and low magnitude anomalies. Re-logging of historical drill holes and compilation of historical geochemical, geophysical and drill data led to drilling of eight holes (5,028 m) in 2002 on the northern shore of Russet Lake, in an area now referred to as the Treasure Box zone. Of these eight holes, three intersected visible gold and all eight intersected gold grades ranging from 1 to 48 g/t Au. A further 41 holes (19,328 m) were drilled at Treasure Box in 2003 and 2004, with some of the better composites including 9.6 m at 4.58 g/t Au and 4.2 m at 17.9 g/t Au.

Five holes (2,664 m) were drilled on the western shore of Russet Lake in 2002. Four of the holes intersected gold values ranging from 1 to 14.5 g/t Au with a best intercept of 10.6 g/t Au over 1.22 m. A further three holes (2,356 m) were drilled in this area in 2003, outlining a broad corridor of ductile deformation with gold values from 1.0 to 8.83 g/t Au over 0.3 to 1.2 m widths.

Nine holes were drilled in the Starratt / Wedge area in 2003, and although visible gold was encountered in some holes, Placer found the widths were generally narrow and the continuity was irregular (Dobrotin, 2004a). Drilling did encounter anomalous gold values along the footwall of the Russet Lake ultramafic contact in a previously unexplored area now known to be the MJ Zone. This previously unidentified gold-bearing structure was originally called the Footwall Zone and the original targeting criteria included a gravity anomaly and a magnetic signature typical of mafic/ultramafic contacts.

After Claude re-acquired the mine property operatorship in 2006, they focused mainly on drilling (Table 6-2), historical data compilation, and dewatering and rehabilitation of the Mine. Mine dewatering commenced in 2007 and was discontinued in late 2013. Claude drilled 108 holes in the Mine area, both from surface and underground. Their main targets were the down-dip extension of 8 Zone, the McVeigh target near the southwestern extent of known mineralization, near-surface mineralization northeast of the Austin zone in an area known as Apple, and its down-plunge extension.

Claude drilled 51 holes in the Treasure Box area in 2007 (Malegus et al., 2022), testing the system to depths in excess of 350 m. Anomalous gold values were present throughout, with several narrow high-grade zones associated with quartz-tourmaline veining over a strike length of 165 m. The best intersections included 6.05 m grading 12.94 g/t Au and 1.22 m grading 38.47 g/t Au.

Claude began exploring at Starratt-Olsen again in 2008, drilling 31 holes for 15,505.6 m. The first phase of drilling consisted of 18 holes designed to test for prospective structures along 1,500 m of strike length from approximately the MJ Zone in the southwest up to the 86 Zone in the northeast. All holes were terminated in the post-tectonic Killala-Baird Batholith. The most significant assays returned included 0.4 m of 190.29 g/t Au at Starratt and 0.6 m of 10.49 g/t Au at the MJ Zone.

A follow-up phase of drilling (13 holes for 5,100.7 m) tested the MJ Zone with tightly spaced holes. The drilling outlined two narrow parallel shear zones hosting high-grade gold. These holes targeting the MJ Zone also tested the CK Zone nearer surface and returned several significant gold-bearing intercepts.

Claude drilled an additional four holes in 2010 with three holes drilled to test below the historical Starratt Mine workings, including ST 08-32 (4.0 m of 6.5 g/t Au) proximal to the shaft. ST10-33 was drilled 400 m to the northeast of the Starratt mine shaft and returned 2.0 m of 7.0 g/t Au. No further Claude exploration work was carried out on the Starratt-Olsen claim group.

6.3.2 Newman-Madsen

The Newman-Heyson property was explored under a joint venture between Wolfden and Kinross Gold Corporation ("Kinross") in 2002 and 2003. In 2002, the joint venture completed line-cutting, ground magnetics, soil geochemical surveys and six drill holes (1,786 m) testing targets in the Dome stock (Klatt, 2003a). Assay results included rare high-grade intersections including hole KRL-02-05 that intersected 9.25 g/t gold over 3.55 m. In 2003, the joint venture drilled 11 holes (2,407 m) on widely spaced targets, but no gold mineralization was encountered (Klatt, 2003b).

In 2004, Wolfden created the Newman-Madsen project by amalgamation of the My-Ritt, Nova Co, and Newman-Heyson properties. Exploration on Newman-Madsen was completed under a joint venture between Wolfden and Sabina Resources Ltd. ("Sabina Resources"), whereby Sabina Resources earned a 50% interest in the property. In 2004, the joint venture completed a drilling program comprising 31 holes (9,531 m) with Wolfden as operator (Toole, 2005). Drilling intersected gold mineralization along a regional structure. In this area, mineralization is spatially associated with an arsenic soil geochemical anomaly related to the Dome stock granodiorite. This mineralized zone was subsequently termed the Evade zone (Toole, 2005).

In 2006 the joint venture drilled four holes (2,964 m) to test targets along or near the Balmer-Confederation unconformity. All holes intersected anomalous gold values highlighted by an intercept of 22.57 g/t Au over 2.0 m in hole DDH NM06-02 (Long, 2007).

In 2010, the joint venture, under the operatorship of Sabina Gold & Silver Corp. ("Sabina") completed four holes (3,183 m) to test the far northeast extension of the Mine trend stratigraphy at levels significantly deeper than previously explored. Drilling was successful in intersecting the targeted stratigraphy and delineating an area of hydrothermal alteration with significant gold, including a high-grade intercept of 43.51 g/t Au over 0.65 m in hole NM-10-02.

In 2011, the joint venture drilled nine holes (3,006 m) to test targets interpreted to comprise folded mafic and ultramafic rock sequences of the Balmer Assemblage where they are coincident with favourable structures, geochemical signatures, and resistivity anomalies. These targets were selected to test Red Lake Gold Mine High-Grade zone style opportunities and returned a series of anomalous and significant gold values.

In January 2012, Sabina acquired 100% interest in the Newman-Madsen Property for a cash payment of C$500,000 and issuance of a 0.5% net smelter return royalty to Premier Gold Mines Limited. Following this transaction, Sabina drilled 13 holes (4,332 m) testing extensions of the Buffalo mine trend, the Dome Stock contact and the Balmer Assemblage (Sabina Gold and Silver Corp., 2012).

In March 2013, Sabina contracted a 37.4 line-km IP survey using a Volterra-3DIP instrument array in an attempt to delineate the extent of the Buffalo and Madsen trends, and to outline the contact between the Dome stock and the adjacent Balmer Assemblage volcanic rock.

In June 2014 Sabina sold the Newman-Madsen Property to Pure Gold, who amalgamated it into the mine property. Table 6-3 summarizes historical drilling on the former Newman-Madsen Project.

Table 6-3: Summary of drilling on Former Newman-Madsen Project

Operator	Year	No. of Holes	Total Length (m)
Coin Lake Gold Mines Ltd	1930s	~221	unknown
Cockeram Red Lake Gold Mines	1943–1946	45	5,674
Cochenour-Willans Gold Mines Ltd.	1971	3	528
Noranda Inc.	1981–1982	33	unknown
Wolfden Resources Ltd. / Kinross Gold Corporation	2002–2003	17	4,193
Wolfden Resources Ltd. / Sabina Resources Ltd	2004–2006	35	12,495
Premier Gold Mines Ltd / Sabina Gold & Silver Corp.	2010–2011	13	6,189
Sabina Gold & Silver Corp.	2012	13	4,332
Totals		**~380**	**~29,200**

Source: after Cole et al. (2016)

6.3.3 Derlak

Reddick and Lavigne (2012) reported no further exploration on the Derlak property after 1998. A Titan 3D IP survey and drilling of three holes totalling 1,556 m were completed by Orefinders along with limited fieldwork prior to purchase of the property by Pure Gold in 2017. The core drilled by Orefinders was subsequently relogged by Pure Gold and two of the three holes were confirmed to not have tested the Balmer Formation. In the third hole unsampled intervals with prospective quartz veining and mine style alteration were logged and sampled. Low gold values were returned.

6.3.4 Fork

A mineralized lens near the centre of the Fork deposit was discovered by Placer Dome during exploration programs in 2002–2004. In 2003, two holes (for 1,671 m) were drilled on the northeastern part of the target 500 m along trend from the southwest extent of the McVeigh zone. The original targeting criteria was an intersection between an interpreted flexure in the Russet Lake ultramafic rocks and a planar structure interpreted from field mapping and airborne magnetic survey data. Both drill holes intersected a wide zone of strongly altered and deformed mafic and ultramafic rocks with several gold intercepts highlighted by 4.0 g/t Au over 1.2 m (Dobrotin, 2003).

In 2004, Placer Dome drilled an additional 14 holes (for 4,489 m) at Fork with significant intervals including 6.1 g/t Au over 2.8 m and 47.0 g/t Au over 1.3 m. During this drilling program Placer Dome reported that Fork was composed of several southeast plunging shoot structures. Two of these structures (AD and BC zones) hosted deformed gold-bearing blue grey quartz veins proximal to deformation zones within the Russet Lake Ultramafic. The mineralization style was considered analogous to that of the 8 Zone, though the exploration program was unsuccessful in delineating a connection between the Fork deposit and 8 Zone. Systematic drilling up- and down-dip of the AD and BC zones was recommended (Dobrotin, 2004a).

In 2007, Claude Resources completed 17 drill holes at Fork and followed up with extensive drilling in 2008–2009 (105 holes for 45,179 m) (Lichtblau et al., 2009). Drilling in 2008 focused on infilling at 30 m to 40 m spacing and indicated mineralization was spatially related to two subparallel southeast-dipping shear zones that host narrow, discontinuous gold-bearing vein systems over a strike length of 400 m (Lichtblau et al., 2009). Significant intercepts included 13.91 g/t Au over 8.39 m and 15.77 g/t Au over 7.62 m. Additional drilling in 2009 attempted to demonstrate continuity along the interpreted mineralized structures and define the limits of the known mineralization (Cole et al., 2010). Modelled continuity was interpreted to be poor and no resource estimation was completed.

6.4 2014 – 2023 (Pure Gold Mining)

Following acquisition of the Madsen Property in 2014, Pure Gold completed several focused surface exploration campaigns (Table 6-4) comprised of geological mapping and rock and soil sampling with a focus on gaining understanding of gold mineralization on the mine property. An airborne geophysical survey was completed across the property in 2014 to aid in geologic mapping, structural interpretation and targeting. MMI soil sampling was completed across the property from 2014 to 2017. Field programs of mechanical overburden stripping, mapping and rock sampling were completed at the Russet South deposit in 2015, the Dev, Dev Northwest and Roberts targets in 2016 and 2017, the Wedge deposit in 2018 and the Wedge-OL, Dev and Derlak targets in 2019. In addition to this outcrop-scale work, all soil geochemical anomalies detected during the MMI soil sampling campaigns were prospected during the 2017 and 2018 field seasons. During the 2019 field season a systematic evaluation of all surface exploration targets generated by previous work was conducted and the results used to inform follow-up exploration drilling programs in late 2019 and 2020.

Extensive re-logging programs were conducted in 2017 and 2018, in which core drilled by previous operators was geologically logged in a manner consistent with the current geological understanding and coding scheme, re-sampled where appropriate, and photographed. Following this, the core was transported off-site to a newly constructed core storage area on the Russet Lake access road.

A 2D seismic survey was conducted in 2020 with the aim of demonstrating the viability of seismic techniques to detect the structures which host mineralization on the mine property; results were successful in imaging features associated with both the 8 Zone and Austin / South Austin zones of mineralization. Drilling testing of a target emerging from this work (Derlak) intersected low grade gold mineralization within the same lithological and alteration package associated with the actively mined Austin zone, though at 750 m further down plunge.

These exploration programs were successful in contributing significant new geoscience data relied on in concert with extensive historical datasets to develop a new geological model for gold mineralization on the property (Section 8). The sampling programs delineated new gold anomalous zones in all target areas described in Section 7.3 and identified new high-grade gold surface mineralization at several targets. New drilling targets were developed and significant high-grade gold-bearing drill intercepts resulted at Starratt, Fork, Wedge, Treasure Box and Russet South.

Table 6-4: Madsen Property exploration other than drilling 2014 – 2022

Exploration Technique	Year(s)	Target or Prospect	Quantity	Reference
Airborne magnetic survey	2014	Property-wide	1,702.8 line km	CGG (2014)
Drill collar location survey	2014	Property-wide	221 drill collars	Pure Gold database
Geological mapping, rock sampling	2014	Madsen deposit/unconformity, Fork, Madsen North	123 rock	Cooley and Leatherman (2014a)
Geological mapping, rock, and soil sampling	2014	Property-wide and Russet grid sampling	37 rock 117 B horizon soil 505 MMI soil 123 lithogeochem	Baker (2014a)
Geological mapping, rock sampling	2014	Derlak Lake towards Red Lake, Buffalo	79 rock	Cooley and Leatherman (2014b)
Geological mapping, rock, and soil sampling	2014	Mapping at Russet and No. 1 Shaft; MMI sampling at Madsen South, Pumphouse, SPfold and Dev grids	29 rock 2,021 MMI soil 8 lithogeochem	Baker (2014b)
Geological mapping, rock sampling	2015	Flat Lake, Dev, Hasaga, Buffalo, DeVillier, Snib Lake, McVeigh, Coin Lake, Fork, Shore	410 rock, most analysed by portable XRF only	Cooley and Leatherman (2015)
Mechanical stripping, geological mapping, rock sampling	2015	Russet, Dev, Russet North	202 rock, 72 channel, 3,234 MMI soil	Baker and Swanton (2016)
Petrography	2015, 2016	Russet, Madsen	67 thin polished sections	Ross (2015), Leitch (2016)
Mechanical stripping, rock sampling	2015	Russet	78 rock	Pure Gold database

Exploration Technique	Year(s)	Target or Prospect	Quantity	Reference
Mechanical stripping, geological mapping, rock sampling	2016, 2017	Dev, Dev Northwest, Roberts, Roberts South	296 rock	Jones (2016), Pure Gold database
Soil sampling	2016	Property-wide	2481 soil	Pure Gold database
Geological mapping, rock sampling	2017	Property-wide	143 rock	Pure Gold database
Soil sampling	2017	Derlak	686 soil	Pure Gold database
Geological mapping, rock sampling, mechanical stripping	2018	Wedge	125 rock	Pure Gold database
Historical core re-logging	2017, 2018	Property-wide	595 holes 271,429 m	Nuttall (2017), Bultitude (2018)
Geological mapping, rock sampling, mechanical stripping	2019	Property-wide	388 rock	Swanton et al. (2019), Pure Gold database
Channel Sampling	2020	Treasure Box	36 rock	Pure Gold database
2D seismic survey	2020	Property Wide	3 lines, 35 km	HiSeis (2020)

Source: Pure Gold (2022)

6.4.1 Airborne Geophysics and Imagery

In May 2014, Pure Gold commissioned CGG Canada Services, Ltd (CGG) of Mississauga, Ontario to complete a high resolution magnetic airborne geophysical survey over the entire Madsen Property (CGG, 2014). The purpose of the survey was to provide geophysical support for detailed mapping of the geology and structure of the property. In 2020, Hardrock Geophysics Inc was commissioned to produce a high resolution unconstrained 3D magnetic inversion of this data, the results of which model magnetic susceptibility down to a depth of approximately 600 m below the average surface elevation on the property (Penney, 2020).

In June 2016, Pure Gold commissioned KBM Resources, Ltd (CGG) of Mississauga, Ontario to complete a high resolution LIDAR and orthoimagery survey over most of the Madsen Property to provide new high resolution color orthoimagery and topographic control to an absolute vertical accuracy of 15 cm and orthoimage resolution of 0.1m (Mizon, 2016).

In June 2017, Pure Gold purchased Pleiades 1B color satellite imagery from Skywatch which covers the entire Madsen Property and surrounds at 0.5 m resolution.

A new LiDAR survey was flown, again covering the majority of the property, in October 2019 by KBM Resources, in order to provide updates to both the air photo and elevation data for the new surface infrastructure constructed to that point.

Similarly, another LiDAR survey update was conducted in May 2022 by KBM Resources covering the entirety of PureGold property and the surrounding areas. Mean vertical accuracy of this survey is 0.016 m and orthoimage resolution is 0.1 m (Mizon, 2022).

6.4.2 Survey Control

During 2014 Pure Gold completed a property-wide program to survey a selection of historical drill hole locations to improve confidence in using historical drill hole data. Location data were collected with a Trimble ProXRT™ differential GPS receiver with Omnistar real-time correction, which achieved sub-metre precision. In all, 221 historical collars were surveyed from across the property. Many Madsen Gold Corp. historical collars could not be located due to casing being removed.

During 2017, D.S. Dorland Ltd. (Dorland) re-established the mine grid surface survey controls and created a new transformation conversion between the latest federal datum NAD83 CSRS 2010 UTM 15 CGVD 2013 and the historical Imperial Mine Grid (IMG) to the Metric Mine Grid (MMG) (Dorland, 2017). The transformation is a traditional four parameter two-dimensional similarity transformation which has been augmented with additional parameters to ensure that the centre of rotation and scale is at the average location of the surface control points. The mine grid is rotated 59.303861° counterclockwise from UTM North. Following this work, Dorland conducted an underground control survey from the established surface controls down the Madsen ramp to Level 2, adjusted the surface and underground control surveys and placed new wall control points.

In 2018, Pure Gold surveyors continued to establish and maintain the underground survey control network as development proceeded during test mining.

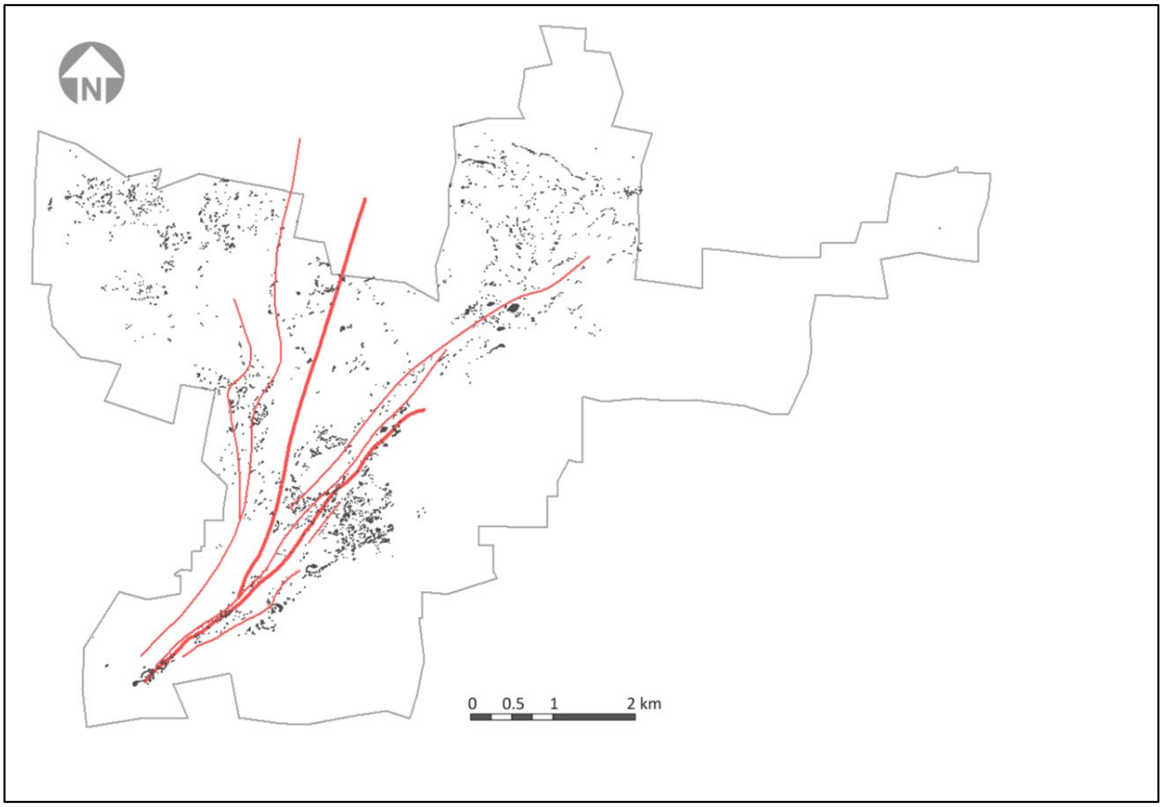
6.4.3 Geological Mapping

Several geological mapping campaigns were completed during the 2014 and 2015 summer field seasons as detailed in reports by Michael Cooley, Lamont Leatherman and Darcy Baker (Cooley and Leatherman, 2014a, b; Baker, 2014a, b; Cooley and Leatherman, 2015; Baker and Swanton, 2016). During the 2017 season, a comprehensive property-wide mapping and prospecting campaign was initiated, designed primarily to follow up on soil and surface rock anomalies (Leidl, 2018). In 2018, mapping and prospecting was largely restricted to the Wedge area in the southwest corner of the property. Detailed mapping of mechanically stripped outcrops in the Wedge, Derlak and Dev areas was conducted in 2019 in conjunction with sampling of these outcrops (Swanton et al., 2019).

GPS-enabled field computers were used to map locations and shapes of outcrop exposures and to collect data on lithology, alteration and structure which has resulted in a database of more than 4,000 individual bedrock outcrops across the property.

The resultant property-wide geological map is summarized in Figure 6-1, which shows the Pure Gold mapped outcrops, including mechanically-stripped outcrops. Mapping has defined structural features (foliations, folds) relating to different deformation events and constrained the timing of gold mineralization relative to these events.

Figure 6-1: Pure Gold outcrop mapping



Note: Mapped outcrop locations in black and target alteration corridors in red.
Source: Pure Gold (2022)

6.4.4 Mechanical Stripping

A series of six outcrops were stripped with an excavator by Pure Gold in 2015 to provide bedrock exposure over key areas where previous drilling had intersected near-surface gold mineralization. Stripped areas were mapped and sampled in detail. The exposure revealed several structural relations and indications of the timing of gold mineralization that were not previously apparent in drill core (Baker and Swanton, 2016).

A reconnaissance outcrop stripping program was completed in the Dev and Roberts areas in 2016 to follow up on a series of gold anomalies in surface grab samples and MMI soil samples. Several prospective zones with similar mineralization style to the Russet deposit were identified and follow-up was recommended (Jones, 2016). More extensive stripping of these outcrops and others in the Dev, Dev Northwest and Roberts areas was conducted in 2017, with channel and grab samples from several of these new exposures returning gold values significant enough to justify further work, including drilling.

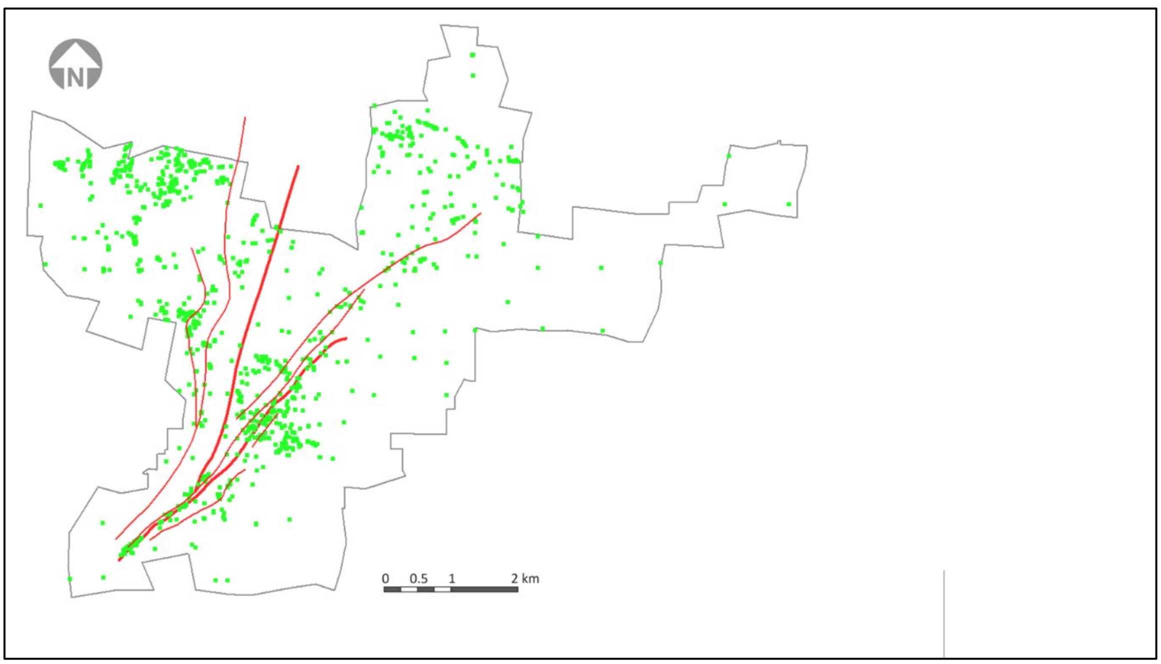
Outcrop stripping in the Wedge DV Zone was carried out in 2018 to provide improved surface geological mapping of the deposit. Four outcrop areas were stripped and mapped with channel and grab samples collected from these outcrops returning values from 0 to 25.9 g/t Au.

A program of further outcrop stripping was conducted in 2019 with the objective of advancing prioritized exploration targets identified from previous programs. A total of five outcrops were stripped and mapped during this program; two in the Wedge-OL area, two in the Dev area and one in the Derlak area. These areas have all been reclaimed by seeding with native species where conditions allowed.

6.4.5 Rock Geochemistry

Up to the end of 2021, Pure Gold collected 2,271 surface rock samples. The majority (1,764) of the samples were analyzed using 4-acid ICP-MS for lithogeochemistry and fire assay for gold. A subset of these samples (53) were analyzed for gold only via fire assay, while another subset (454) were collected for reference purposes during geological mapping and were not geochemically analyzed. Rock sample locations are shown in Figure 6-2. The samples were collected to determine the composition and alteration state of the main lithologic units encountered during mapping, and to determine gold content. Numerous grab and channel samples were collected at natural outcrops and those exposed during mechanical stripping. Industry best practice techniques are applied to the collection of grab and channel samples, however due to the selective nature of the sample collection the results are not considered in themselves to be representative of average gold content of the sampled zone but are rather used as one guide to the prospectivity of a target prior to drilling.

Figure 6-2: Pure Gold rock sample locations



Note: Rock sample locations in green and target alteration corridors in red.
Source: Revering et al. (2022)

6.4.6 Soil Geochemistry

Two soil sampling techniques were trialed by Pure Gold: conventional, B-horizon soil sampling and Mobile Metal Ion (MMI) soil sampling. During the first sampling program in 2014, both types of samples were collected at the same widely-spaced sites across the property. Subsequent surveys focused on collecting follow-up soil samples using only the MMI technique which was deemed to be the most appropriate for most sample sites (Arne, 2014).

MMI soil samples were collected in plastic Ziploc bags from a continuous interval between 10 cm and 25 cm below the organic/inorganic interface. Undecomposed organic material was avoided and excluded from the sample. Depending on the depth to the organic/mineral soil interface and the amount of groundwater, samples were collected with a hand auger or by digging a small pit. Sites were photographed, marked with Tyvek tags and data recorded in field notebooks to be entered into Pure Gold's sample template. Location data was recorded on handheld Garmin GPSs.

Conventional B-horizon soil samples were collected in paper kraft bags from the B soil horizon using a shovel or auger. Undecomposed organic material was avoided and excluded from the sample. Sites were photographed, marked with Tyvek tags and data recorded in field notebooks to be entered into Pure Gold's spreadsheet template. Location data was recorded on handheld Garmin GPSs.

For the initial, property-wide soil sampling program, sample locations were spaced about 1,000 to 500 m apart. For subsequent, follow-up programs MMI samples were collected along east-west grid lines spaced 100 m apart. Sample spacing along these lines was 25 m although sampling sites were modified slightly as appropriate to select a suitable location. Soil sample locations are shown in Figure 6-3.

Pure Gold collected 8,972 MMI soil samples covering the majority of the property that is underlain by Balmer Assemblage rocks. Several regions of anomalous gold exist, including some which are not explained by bedrock geology (these are explained in greater detail in Section 7.3). Given the extremely sensitive nature of the MMI technique, some areas of anomalous gold near historical mine sites may be due to contamination by tailings transported by wind or surface water.

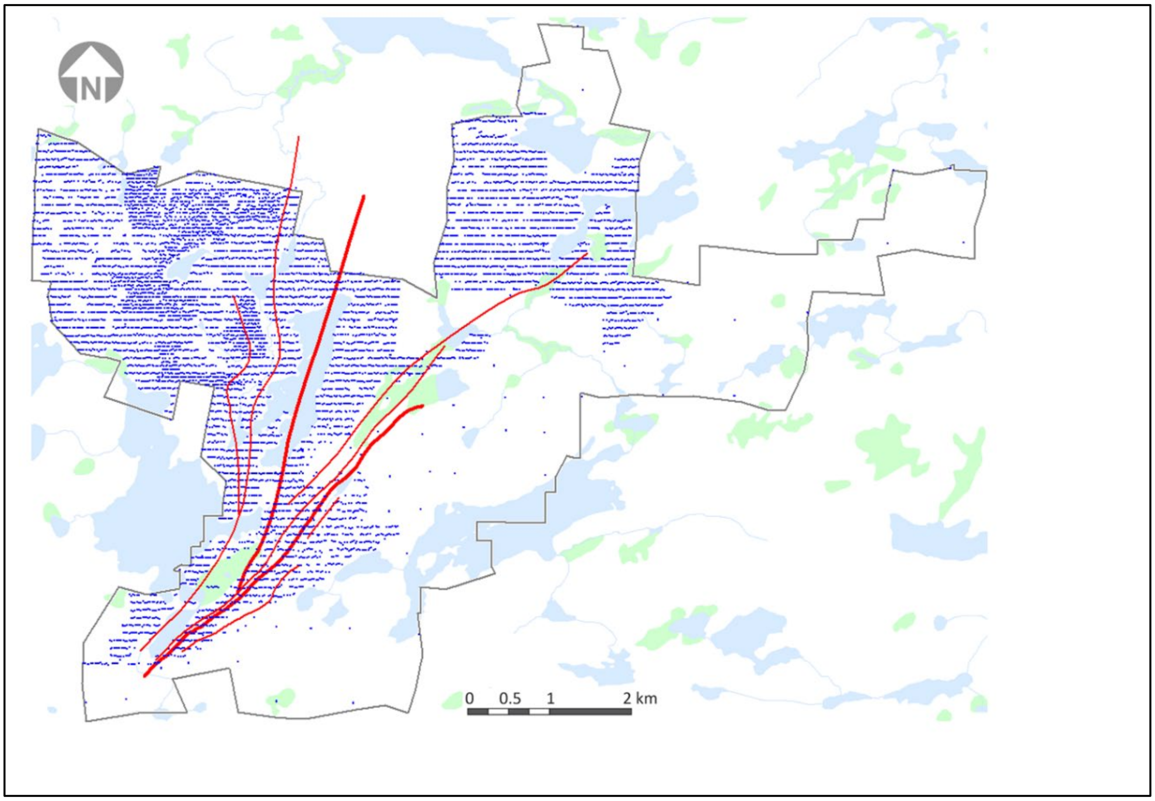
6.4.7 Historical Drill Core Relogging

An extensive historical drill core relogging campaign was initiated in 2017 and completed in 2018. The program focused on drill core produced by Placer Dome and Claude between 2001 and 2010. In total, approximately 271,000 m from 595 drill holes were photographed and recoded, which updated the historical logging codes to the Pure Gold logging scheme. Additionally, several thousand new core samples were collected and analyzed for gold and multi-element ICP geochemistry. Magnetic susceptibility was recorded for 10 holes through the 8 Zone. The processed core was moved to a new core storage site on the Russet Road.

6.4.8 Petrography

Pure Gold completed several petrographic studies of samples selected to characterize timing of mineralization, alteration phases and igneous precursors (Ross, 2015; Leitch, 2016; Ross, 2016). The results of this work have been integrated along with lithogeochemical studies to refine core logging and the geologic mapping scheme.

Figure 6-3: Pure Gold soil sample locations

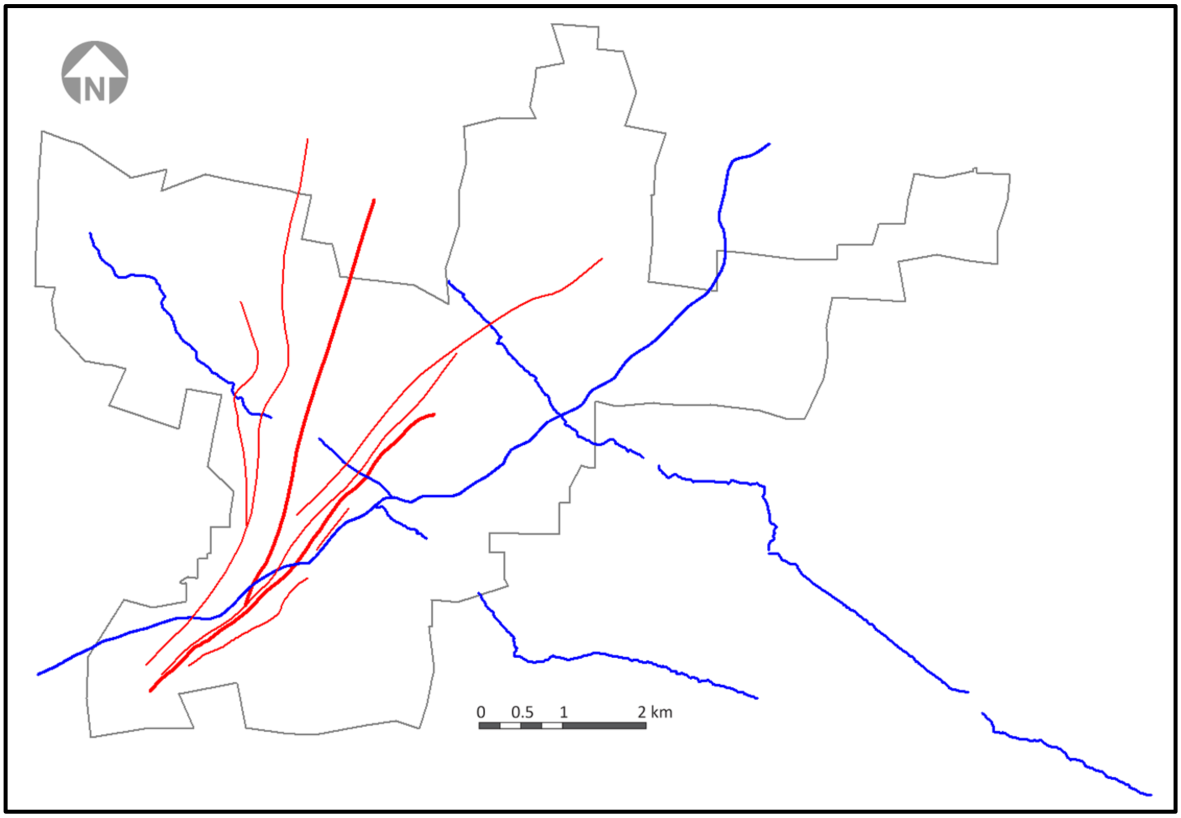


Note: soil survey locations in blue and target alteration corridors in red
Source: Pure Gold (2022)

6.4.9 Seismic Survey

A 2D seismic survey was conducted in 2020 by HiSeis, an Australian company specializing in usage of seismic surveys for hard rock mineral exploration. Three lines were surveyed, for a total of approximately 35 line-km of data with data reported to a depth of 5 km below surface. Two of the lines were oriented northwest-southeast, approximately perpendicular to the strike of stratigraphy and mineralization while the third was oriented northeast-southwest, generally parallel to these trends. The NE-SW line was conducted along Highway 618 and the Suffel Lake Road, paralleling the Balmer-Confederation unconformity and spatially overlapping the unconformity for approximately 1 km. The NW-SE trending lines were conducted along a combination of existing bush roads and newly cut access lines. The southern line was surveyed such that it crosses the trend of the Madsen deposit at the location of both the headframe and the 8 Zone, making it well placed to image both the upper portions of the Austin / South Austin zones and the 8 Zone at depth. The northern line crossed the trend of the alteration corridor hosting the Madsen deposit approximately 1 km northeast of the furthest extent of developed stopes from the PureGold Mine, making it well placed to image any possible along-strike continuity of the deposit (Figure 6-4). The survey identified acoustic impedance responses interpreted to represent alteration associated with gold mineralization. Results were successfully used for drill targeting at Derlak.

Figure 6-4: 2D seismic survey



Note: 2020 Seismic survey lines in blue and target alteration corridors in red.
Source: Pure Gold (2022)

6.4.10 Test Mining and Underground Bulk Sample

Pure Gold collected an underground bulk sample of the McVeigh zone of the Madsen deposit in 2018 during a test mining program designed to gauge lateral and vertical continuity of gold mineralization, to validate the resource model, to provide additional geotechnical information and to assess applicability of potential mining methods. A total of 172 mining faces were blasted from the bottom of the existing ramp between 2 Level and 3 Level for approximately 370 m of new development. Of these faces, 128 were under geology control including 16 slashes and five bench faces. Thirteen raise faces were also blasted from two separate raises. Each face was geologically mapped, photographed and sampled. Two parallel sampling lines were typically completed across each face using a pneumatic chipper. After each blast, ~2.0 kg muck samples were collected in a regular pattern with five individual muck samples taken from every second scoop bucket resulting in one muck sample for every three tonnes of mined material. Each blast typically had an average of about 25 muck samples collected, and all samples from each round were averaged to determine if a particular round met the 4.0 g/t gold cut-off. Additionally, each mining face had test sludge holes drilled into the walls approximately one metre back from the face. A total of 3,008 muck samples, 1,698 chip samples and 208 test hole samples were collected (excluding QAQC samples, which were systematically inserted into each sample batch).

The test mining was advanced using a single-boom hydraulic jumbo drill, typically with one round blasted per day. Blasted rock was mucked using scoop trams with varying bucket capacities (including 2, 3.5 and 6 cubic yards), which was then loaded into a 30 tonne capacity haul truck. Underground survey control was maintained using a total station. In all, 4,953 tonnes of high-grade (>4.0 g/t gold) material was mined and stockpiled in a secure underground location, while the 2,143 tonnes of lower-grade (1-4 g/t gold) material was mined and stockpiled in a secure location on surface. Some un-mineralized mine rock was trucked to surface and placed on the existing un-mineralized mine rock storage area, while the remaining un-mineralized mine rock was used for backfill in historical underground workings.

As mining progressed, drift walls were tested on a minimum of 3 m spacing with a core drill to identify potential for additional gold mineralization. The holes were drilled ~10 to 20 m horizontally with a Bazooka air drill, which returns EW size (25.4 mm or 1 inch) core samples that were logged and photographed prior to whole core sampling. A total of 1,976 m of Bazooka core drilling was completed in 153 holes and 1,711 samples (excluding QAQC) were collected.

6.4.11 Ongoing Exploration Targets

The mineralized zones described in Section 7.3 are all subject to ongoing exploration as targets.

6.4.12 Sampling Methods and Quality

The rock and soil sampling by Pure Gold was systematic and completed according to a set of clearly documented procedures that meet or exceed industry best practices. Rock samples provide an indication of the presence of gold but are biased by available bedrock exposure and sample selection. Similarly, soil samples provide an indication of elevated gold within the overburden, but this material can be transported and is not necessarily indicative of underlying bedrock.

6.4.13 Interpretation

The Madsen Gold Project surface (non-drilling) exploration dataset comprises systematic, property-wide, multifaceted information carefully collected using modern techniques. Combining surface geophysical (magnetic and seismic), geochemical and geological information with historical data and drilling data has allowed for a property-wide geologic map that has formed an important input for sub-surface three-dimensional geologic interpretation supported by the drilling dataset. Delineation of several new surface targets has resulted from compilation of the surface data sets. The surface dataset continues to be refined and informed by infill geological mapping supported by mechanical stripping and by diamond drilling. In the current state it forms a valuable base for geologic interpretation and extrapolation in support of exploration.

6.5 Historical Production

Total recorded production from 1938 to 1974 at the Madsen Mine was approximately 7,593,900 metric tonnes at an average grade of 9.91 g/t Au (approximately 8,371,630 tons at an average grade of 0.289 ounces of gold per ton). Annual production for this period is summarized in Table 6-5 (excludes data from certain periods). This accounted for approximately 2,416,600 ounces of gold.

Table 6-5: Gold production for Madsen Mine from 1938 to 1976

Year	Gold Production (Troy ounces)	Tonnage Milled (short tons)	Year	Gold Production (Troy ounces)	Tonnage Milled (short tons)
1938	n/a	n/a	1958	123,489	302,200
1939	13,909	65,460	1959	118,805	301,999
1940	25,716	140,674	1960	119,084	306,377
1941	30,088	141,109	1961	106,096	301,031
1942	30,971	145,534	1962	100,878	311,705
1943	39,195	146,346	1963	107,131	306,247
1944	33,733	144,179	1964	n/a	n/a
1945	36,825	127,870	1964	94,869	305,823
1946	25,438	98,472	1965	87,632	94,869
1947	34,977	143,371	1967	70,033	277,566
1948	32,421	143,391	1968	56,196	265,268
1949	35,579	150,779	1969	60,579	238,473
1950	65,444	282,050	1970	40,569	184,530
1951	61,687	302,227	1971	44,497	146,162
1952	67,337	304,251	1972	37,696	138,250
1953	82,596	285,018	1973	29,163	126,070
1954	82,333	286,246	1974	2,102	11,112
1955	104,874	295,713	1975	n/a	n/a
1956	100,995	294,913	1976	2,196	12,840
1957	103,181	305,300	Total	2,208,313	7,433,425

Note: Production figures extracted from available Madsen Mine annual reports, 1938 to 1976. n/a = data not available.

Source: Cole et al. (2016)

From 1998 to 1999, Claude Resources began mining portions of the McVeigh and Austin Zones. In 1998, Claude extracted 85, 417 tonnes, of which 81,740 tonnes were milled for a total production of 8,929 ounces of gold at an average recovered grade of 3.43 g/t Au (0.10 ounces per ton gold). Mill recovery was estimated to be 86.75 percent, suggesting a head grade of around 3.91 g/t Au (0.114 ounces per ton gold). Stoping was within the Austin Zone between Levels 2 and 5 of the mine and the McVeigh Zone. Information available for the final seven months ending October 1999 indicate a mill throughput of 99,726 tonnes at 4.39 g/t Au (0.128 ounces per ton) for a total of 13,260 ounces of gold.

After 15 months, the Madsen Mine and mill complex was put on care and maintenance status in October 1999.

6.6 Historical Mineral Resource Estimates

Numerous historical estimates of mineral resource inventories (NI 43-101 non-compliant) have been prepared for the Madsen Mine throughout its history dating back to the start of production in 1938. However, in 2008 Claude Resources Inc. commissioned SRK Consulting (Canada) Inc. (SRK) to prepare a resource estimate to NI 43-101 standards. The technical report titled “Mineral Resource Estimation Madsen Gold Project Red Lake, Ontario, Canada”, dated January 20, 2010, and authored by Cole et al., defined an Indicated resource of 3,236,000 tonnes at an average grade of 8.93 g/t Au, and an Inferred resource of 788,000 tonnes at an average grade of 11.74 g/t Au, using a cut-off grade of 5 g/t Au, a metal price of US$1000 per troy ounce gold and a metallurgical recovery of 94%. The effective date of this Mineral Resource Estimate (MRE) is December 7, 2009.

This historical MRE included available data to September 27, 2009, which consisted of 13,624 drillholes (816,367 metres) drilled between 1936 and 2009 and 4,446 historical underground stope chip samples, 550,687 gold assay records and 620 specific gravity measurements. A total of 16 interpreted mineralized domains within the Austin, South Austin, McVeigh and Zone 8 were used in the estimation workflow, and assay samples were composited to 2-metre lengths within the Austin, South Austin and McVeigh mineralized domains and 1-metre composites within Zone 8 mineralized domains.

Outlier composites were evaluated using log normal distributions from each domain and capped. Variography was completed on capped composite data separately for each zone to determine grade estimation parameters. Search neighbourhoods were adjusted based on variography results. Block gold grades were estimated in two successive estimation passes using ordinary kriging although inverse distance and nearest neighbour estimators were also used for comparison. The first estimation pass considered a search neighbourhood adjusted to full variogram ranges whereas the second estimation pass considered a search neighbourhood adjusted to two times full variogram ranges. For Zone 8, the first and second estimation passes considered search neighbourhoods adjusted to twice and five times the variogram ranges, respectively.

Four grade block models were constructed. The block models for Austin, South Austin and Zone 8 were constructed in Datamine Studio 3 using the sub-blocking function. The block model for McVeigh was constructed in GEMS as a percentage block model. Each block model was populated with a gold grade during the estimation process. Block size in all block models was set at 5x5x5 m in size, except for Zone 8 which was set to a 5x5x2 m block size.

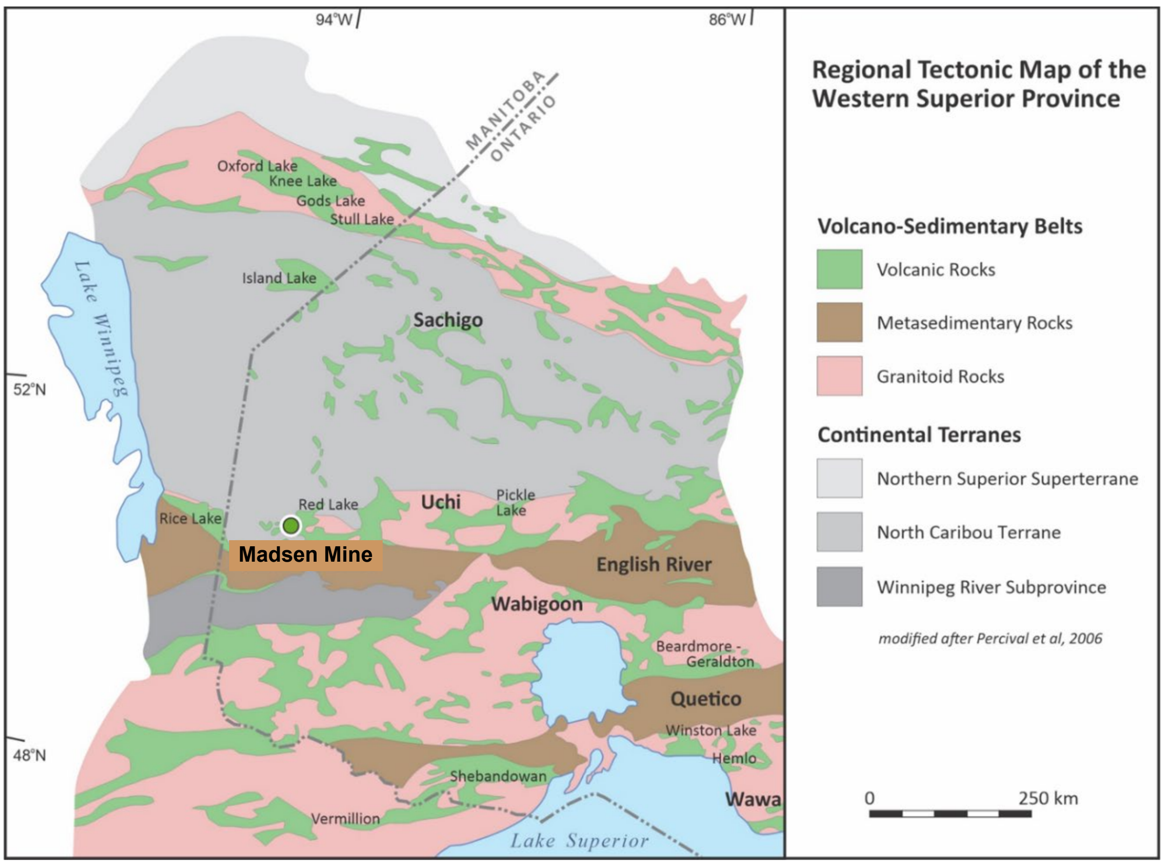
The QP does not consider this historical MRE to be reliable or relevant as additional drilling and sampling, geological investigations and interpretation, and mine production have occurred since the estimate was disclosed in January 2010. As such, this historical MRE is not being treated as a current MRE and is superseded by the mineral resource statement reported herein.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Madsen Mine is located within the Western portion of the Archean Superior Province of the Canadian Shield (Figure 7-1). It 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 is characterized by Mesoarchean and Neoarchean volcanic and plutonic rocks interpreted to have been emplaced within rift and arc-related environments on the continental margin of the Mesoarchean crustal rocks of the North Caribou terrane. The predominantly sedimentary rocks of the English River belt are believed to have accumulated within a synorogenic flysch basin that formed during assembly of the North Caribou terrane with the Winnipeg River terrane to the south during the Uchian Orogeny, ca. 2720-2700 Ma (Percival et al. 2006).

Figure 7-1: Geology of the Western Superior Province



Source: WRLG (2024) after Percival et al. (2006)

7.1.1 Uchi Domain

The Uchi domain (Figure 7-1) is approximately 570 km long by 50 km wide and comprises a series of plutonic rocks discontinuously surrounded by arcuate belts of supracrustal volcano-sedimentary rocks. These supracrustal strata record more than 300 million years of tectonostratigraphic evolution along the southern margin of the North Caribou terrane. The North Caribou basement is inferred to be overlain by 2990–2960 Ma rift-related rocks, 2940–2910 Ma arc volcanic rocks, and cut by 2870–2850 Ma plutonic rocks. The stratigraphic and geochemical characteristics of these Mesoarchean sequences are interpreted to reflect a continental margin setting. A deformation event and unconformity exposed in the Red Lake belt (Sanborn-Barrie et al. 2001) separate the Mesoarchean rocks from Neoarchean strata, the latter including 2745–2734, 2731–2729, and 2722–2719 Ma calc-alkaline volcanic assemblages and younger (<2710 Ma) coarse clastic sedimentary rocks.

Continuously trending packages of supracrustal rocks such as those in the Uchi domain are referred to as greenstone belts. Globally, such Archean belts are responsible for about 18% of historical gold production (Roberts, 1988) and the Uchi domain is a significant contributor. Most Uchi greenstone belts have some recorded historical gold production but all pale by comparison to the well-endowed Red Lake Greenstone Belt which boasts 29.8 million ounces of gold production to the end of 2021 (Malegus et al., 2022).

7.1.2 Red Lake Greenstone Belt

The Red Lake Greenstone Belt is approximately 50 by 40 km and comprises a series of ca. 2990-2700 Ma supracrustal rocks intervening between three main younger granitoid batholiths ranging from 7 km to 20 km across (Figure 7-2). The supracrustal rocks have been stratigraphically divided into eight assemblages and the following descriptions of these are taken from Sanborn-Barrie et al. (2004b).

Balmer Assemblage (Mesoarchean)

The oldest volcanic rocks in the Red Lake greenstone belt comprise predominately tholeiitic mafic and komatiitic ultramafic rocks of the ca. 2990–2960 Ma Balmer Assemblage. Significantly, all of the belt’s major gold resources are hosted in the Balmer Assemblage, near its contact with overlying Neoarchean rocks. The assemblage consists of lower, middle and upper massive to pillowed tholeiitic sequences separated by distinctive felsic and ultramafic volcanic rocks. Minor metasedimentary rocks also occur within the assemblage, mainly as thinly bedded magnetite-chert iron formation.

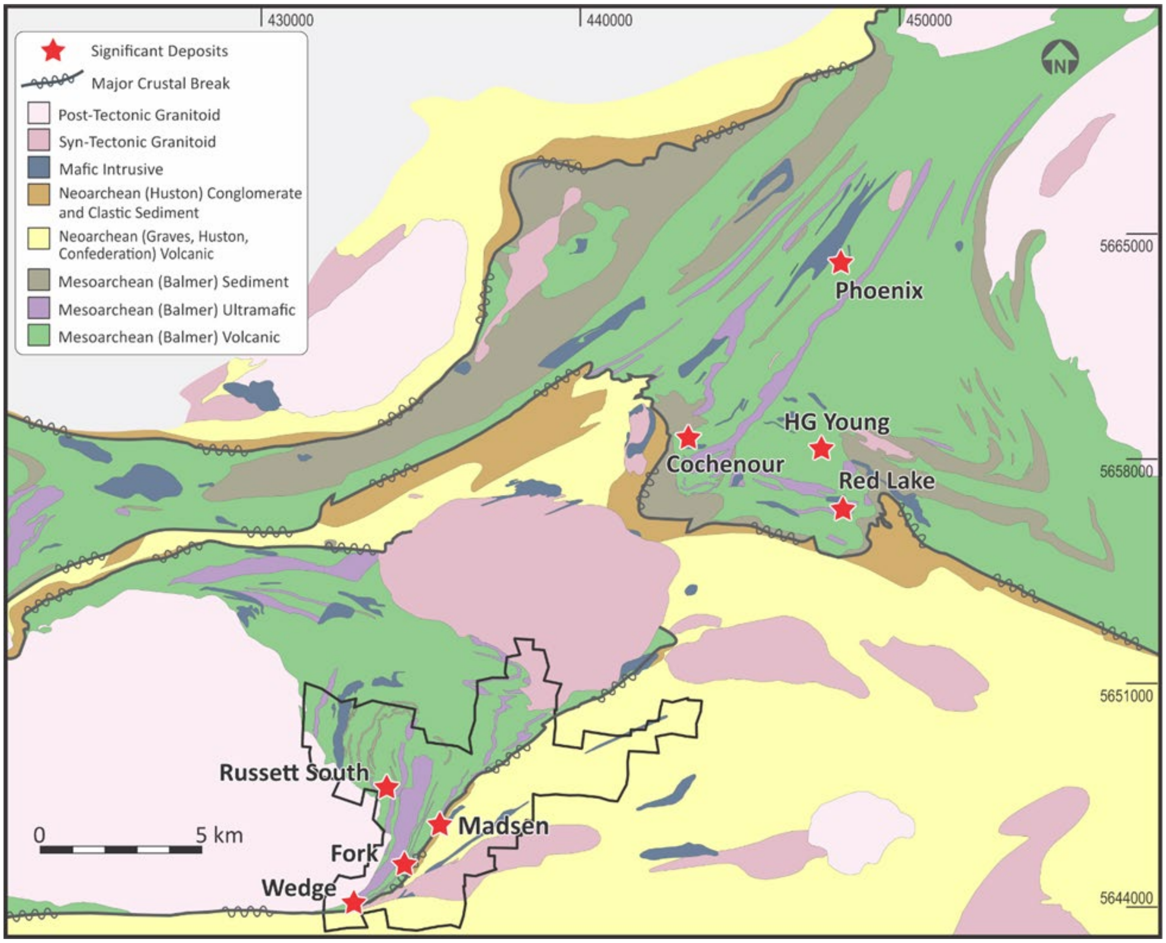
Ball Assemblage (Mesoarchean)

Underlying the northwestern portion of the Red Lake Greenstone Belt is the ca. 2940–2920 Ma Ball Assemblage, comprising a thick sequence of metamorphosed intermediate to felsic calc-alkaline flows and pyroclastic rocks.

Slate Bay Assemblage (Mesoarchean)

The Slate Bay Assemblage extends the length of the belt and lies disconformably on Balmer and Ball assemblage volcanic rocks. It comprises clastic rocks of three main lithological facies varying from conglomerates, quartzose arenites, wackes, and mudstones. Detrital zircon data indicate that the Slate Bay clastic material is mostly derived from Ball Assemblage rocks with minor input from Balmer Assemblage rocks. Based on the youngest zircon ages, the maximum age of deposition for the Slate Bay Assemblage is ca. 2916 Ma, whereas overlying ca. 2850 Ma volcanic rocks (Trout Bay Assemblage) provide a minimum age for deposition (Corfu et al., 1998; Sanborn-Barrie et al., 2004b).

Figure 7-2: Simplified geology of the Red Lake Greenstone Belt with main deposits



Note: Mine property is shown with black outline

Source: WRLG (2024) after Sanborn-Barrie et al. (2004b)

Bruce Channel Assemblage (Mesoarchean)

A thin (<500 m) sequence of calc-alkaline dacitic to rhyodacitic pyroclastic rock, clastic sedimentary rock and banded iron formation is dated at ca. 2890 Ma and assigned to the Bruce Channel Assemblage. Enriched LREE trace element profiles relative to the Balmer assemblage are interpreted to indicate crustal growth at a juvenile continental margin.

Trout Bay Assemblage (Mesoarchean)

The Trout Bay Assemblage was previously correlated with Balmer rocks but represents a distinct sequence in the northwestern part of the belt. It comprises tholeiitic basalt, clastic rock and iron formation. An interbedded, intermediate tuff returned a ca. 2850 Ma age for this assemblage.

Confederation Assemblage (Neoarchean)

Following an approximately 100-million-year hiatus in volcanic activity, the Confederation assemblage records a time of widespread calc-alkaline volcanism from ca. 2748–2739 Ma. A ca.

2741 Ma (Lichtblau et al., 2012) quartz-feldspar-porphyritic lapilli tuff along with a localized conglomerate, form a distinctive basal Confederation assemblage unit within the Madsen Mine area.

Overlying the McNeely sequence in the Confederation assemblage is the Heyson sequence of tholeiitic basalts and felsic volcanic rocks. Isotopic and geochemical data suggest the McNeely rocks were formed during a shallow marine to subaerial arc on the existing continental margin with later intra-arc extension and eruption forming the Heyson sequence.

In the Mine area, the strata of the Confederation and Balmer assemblages are in angular unconformity with opposing facing directions. The Balmer Assemblage was, thus, at least tilted and possibly overturned prior to the deposition of the Confederation Assemblage (Sanborn-Barrie et al., 2001). While no definitive structural break has been identified between the two assemblages, the prevalence of hydrothermal alteration and gold mineralization tracing their contact, combined with distinct styles of deformation commonly mapped on either side of the contact, strongly imply some form of structural control from cryptic buried structures.

Huston Assemblage (Neoarchean)

Following deposition of the Confederation Assemblage, the Huston Assemblage (deposited between approximately 2742–2733 Ma) records a time of clastic sedimentary rock deposition varying from immature conglomerates to wackes. This molasse-style assemblage follows the regional trace of the unconformity separating the Balmer and Confederation assemblages, and further implies tectonic reactivation of some form of cryptic paleo-structure along this boundary. The Huston Assemblage has been compared to the Timiskaming conglomerates in the Timmins camp of the Abitibi greenstone belt that exhibit a similar spatial association with major tectonostratigraphic breaks and gold (Dubé et al., 2004).

Graves Assemblage (Neoarchean)

The ca. 2730 Ma Graves Assemblage comprises andesitic to dacitic pyroclastic tuff on the north shore of Red Lake. It is interpreted to represent the volcanic deposits of a shallow water to subaerial arc complex. It overlies and is locally transitional with the Huston Assemblage.

Intrusive Rocks

Intrusive rocks found in the Red Lake Greenstone Belt generally coincide with the various stages of volcanism described in the assemblage sections above. In the simplest interpretation, these intrusive rocks include the subvolcanic feeders to the extrusive volcanism that occurred at the earth's surface and later magmatic emplacement. These rocks include mafic to ultramafic intrusions during Balmer and Ball time periods, gabbroic sills related to Trout Bay volcanism, felsic dykes and diorite intrusions during the Confederation Assemblage, as well as intermediate to felsic plutons, batholiths, and stocks of Graves Assemblage age.

Confederation-aged magmatic activity evolves from the calc-alkaline suite, reflecting arc-type magmatism, to having sanukitoid affinities, signalling the generation of mantle-derived magmas attributed to slab breakoff (Percival et al., 2012). Syn-kinematic granitoid rocks such as the

McKenzie Island, Dome and Faulkenham Lake stocks as well as the Abino granodiorite (2720-2718 Ma) fall into this latter category and were host to past producing gold mines.

The last magmatic event recorded in the belt is from about 2700 Ma and includes a series of potassium-feldspar megacrystic granodiorite batholiths, plutons and dykes, including the post-tectonic Killala-Baird batholith. The contact between Killala-Baird granodiorite and Balmer Assemblage volcanic rocks is well exposed on the mine property at Flat Lake.

Deformation History

The structural and deformation history of the Red Lake Greenstone Belt is summarized here from the published regional mapping of Sanborn-Barrie (Sanborn-Barrie et al., 2004a; Sanborn-Barrie et al., 2001; Sanborn-Barrie et al., 2000; Sanborn-Barrie et al., 2004b). Note that detailed work on the Madsen Mine has produced a refined structural history, which is discussed in the next section.

The earliest deformation event (denoted as D1) involved non-penetrative deformation which resulted in tilting of Balmer Assemblage rocks prior to Confederation volcanism. Evidence for this is cited as opposed younging directions on either side of an angular unconformity between the Balmer and Confederation assemblages near the mine and within the central portions of the Red Lake Belt.

The first stage of penetrative deformation (D1) (that which has imposed a strong tectonic fabric to the rocks) occurred post Confederation time (after 2.74 Ga). This D1 event resulted in formation of northerly-trending folds (F1) including a NNE-trending fold that trends through the centre of the mine property concordant with the Killala-Baird batholith contact. Sanborn-Barrie suggests that D1 deformation was completed prior to deposition of the ca. 2.73 Ga Graves Assemblage volcanic rocks since these do not seem to be affected by D1 structures.

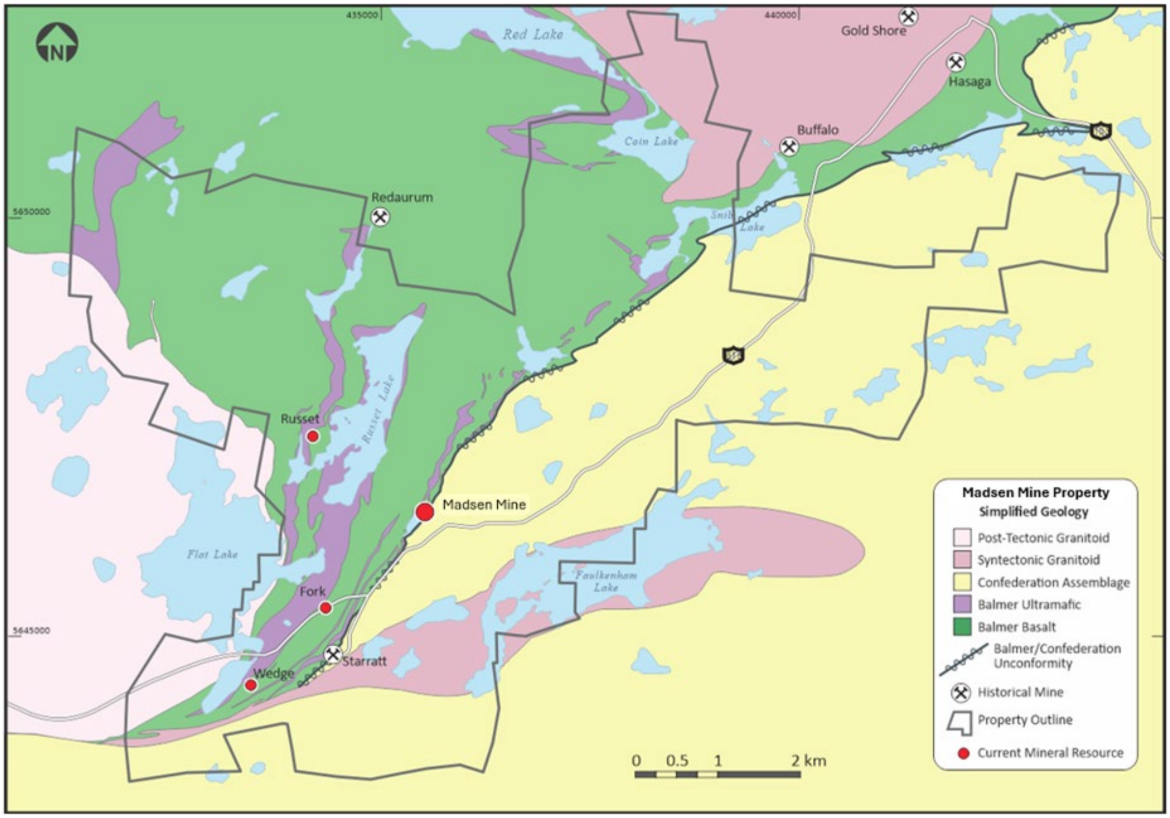
Superimposed on these early structures are E to NE-trending (D_2) structures in the western and central Red Lake Greenstone Belt. These same structures trend SE in the eastern part of the belt. This change in orientation is gradual – consistent with coeval timing rather than an overprinting relationship. Due to the relative absence of mylonitic rocks and strain gradients in these deformation zones, Sanborn-Barrie attributes these structures to a regional bulk strain event, rather than a strongly partitioned crustal-scale shearing event as proposed by an earlier round of researchers (e.g., Andrews et al., 1986, Hugon and Schwerdtner, 1988). The timing of D_2 strain is constrained by the ca. 2.72 Ga Dome Stock which exhibits a weak foliation fabric (S_2) but country rock xenoliths within the pluton also exhibit an intense penetrative S_2. Sanborn-Barrie takes this to mean that the deformation largely predated Dome Stock but continued after stock emplacement which brackets the timing at about 2.72 Ga, linking it to the Uchian orogeny. Since the post 2.70 Ga English River Assemblage conglomerate is deformed by a penetrative.

In summary, Sanborn-Barrie's deformation history of the Red Lake Greenstone Belt involves tilting of Balmer stratigraphy (D0) followed by penetrative foliation development during belt-scale folding (D1) post-Confederation time and lastly, widespread overprinting of D1 structures by localized folding and widespread S_2 foliation.

7.2 Property Geology

The mine property is underlain by Balmer, Confederation and Huston Assemblage supracrustal rocks (Figure 7-3). These older rocks are cut by a series of plutonic rocks (post-tectonic Killala-Baird batholith to the west and syn-kinematic Dome and Faulkenham Lake Stocks to the east) and associated smaller sills and dykes.

Figure 7-3: Simplified geology map of the mine property



Source: WRLG (2024) after geology by (Baker and Swanton, 2016)

The following sections describe the supracrustal, metasomatic (altered), vein and intrusive rock units across the mine property, and which form the basis of geological mapping and drill core logging databases.

7.2.1 Balmer Assemblage Rocks

The oldest rocks underlying the mine property belong to the ca. 2.99–2.96 Ga Balmer Assemblage and comprise: (i) predominantly mafic volcanic and intrusive rocks with minor ultramafic volcanic and intrusive rocks, and (ii) metasedimentary rocks including narrow iron formations which serve as useful stratigraphic markers. Each of the logged and mapped Balmer Assemblage lithologies are described below.

Peridotite

Peridotite (PRDT) sills and flows with komatiitic geochemistry are common within the Balmer Assemblage. These ultramafic bodies are commonly altered to serpentine and magnetite or tremolite-actinolite, but primary intrusive and extrusive features have been identified where original textures are preserved. Spatial relationships, chemical discrimination and primary textures have allowed discrimination into two main units: (i) a series of intrusive or largely intrusive sill-like bodies and (ii) an extrusive unit named the Russet Lake Ultramafic. The PRDT intrusive units have not been identified to host gold mineralization, but the PRDT extrusive units can be an important host rock, particularly in the 8 Zone in the mine, where cut by the Russet shear they host gold-bearing quartz veins.

Pyroxenite

Medium- to coarse-grained pyroxenite (PXNT) occurs within composite sills with PRDT within the Balmer Assemblage. Relict augite has been identified in thin section. The close association of PXNT and PRDT in these sills suggests that PXNT is a product of olivine fractionation during the emplacement of the sills (Mackie, 2016).

Iron Formation

Thin (0.1–1 m) iron formation (IRFM) occurs exclusively within the Balmer Assemblage in the mine area within rare clastic sedimentary packages or more commonly between individual basalt flows. Three types are recognized on the mine property: chert magnetite iron formation, garnet-rich silicate iron formation, and chert sulphide iron formation. Silicate iron formations seem generally less prospective than sulphide iron formations which generally host low-grade (<1 g/t Au) gold mineralization, with much higher grades (>10 g/t Au) present where intersected by mineralized structures.

Metasedimentary Rock

Bedded, clastic metasedimentary rocks (MTSD) of Balmer Assemblage occur as isolated, thin (1-10 m) units hosted within the volcanic package. They typically contain garnet, staurolite, andalusite and amphibole porphyroblasts consistent with an aluminous parent rock.

Basalt

Dark green-brown, fine-grained, unaltered basalt (BSLT) is the most common lithology in the Balmer Assemblage. Basaltic flows are typically massive but are locally pillowed, with rare flow top breccias and hyaloclastite. Unaltered basalt has low prospectively for gold mineralization but altered Balmer basalt is the main host to gold mineralization on the mine property.

Gabbro

Dark grey, massive, equigranular, medium- to coarse-grained gabbro (GBRO) cuts basalt rocks and shows relatively high ratios of MgO:Fe2O3 and Ni:Cr relative to younger Confederation gabbro (O'Connor-Parsons, 2015). Gabbro is not known to be prospective for gold mineralization on the mine property.

7.2.2 Confederation Assemblage Rocks

Felsic Volcanic

Felsic volcaniclastic rock (FVOL) forms the majority of the lower Confederation Assemblage comprising ash, lapilli tuff and juvenile epiclastic rocks sourced from tuffaceous material that commonly directly overlies the quartz crystal-lithic rhyolite tuff (QPXL). FVOL is generally not prospective for gold mineralization at on the mine property.

Intermediate Volcanic

Dark, lustrous, intermediate volcanic rocks (IVOL) overlie the felsic volcaniclastic rocks of the Confederation Assemblage in the mine area. This unit comprises massive and locally pillowed or variolitic flows. This unit is not prospective for gold mineralization on the mine property.

Quartz Crystal and Lithic Rhyolite Tuff

A quartz crystal-rich lithic-crystal tuff (QPXL) forms the majority of the lowest Confederation Assemblage in the Mine area. It has provided a visually distinctive marker interval for both modern and historical geologic study. The unit includes 5–15% quartz phenocrysts and rare flattened lithic fragments in a silica rich, sericitic tuffaceous matrix. QPXL is locally interbedded with lenses of clastic metasedimentary rock and is not prospective for gold mineralization on the mine property. A sample of QPXL collected near the West Portal was dated ca. 2741 Ma (Lichtblau et al., 2012).

Conglomerate

Locally, a pebble-cobble conglomerate (CONG) demarcates the lowermost Confederation Assemblage, conformably underlying the lithic-quartz crystal tuff. This unit appears to be primarily comprised of reworked Balmer assemblage mafic volcanic rocks and likely represents the base of the Neoarchean package resting upon the angular unconformity with intact Balmer rocks beneath. Similar units are also found locally elsewhere in the stratigraphic sequence, above and below the unconformity; the former of which have locally been ascribed to the Huston Assemblage by Sanborne-Barrie et al. (2004b) and Lichtblau et al. (2012).

Metasedimentary Rock

Bedded, clastic metasedimentary rocks (MTSD) are present in both the Balmer and Confederation assemblages as thin (1–10 m) units within volcaniclastic packages. They commonly host garnet, staurolite, andalusite and amphibole porphyroblasts indicating an aluminous parent rock. In the Confederation Assemblage, these units have low gold prospectivity.

Basalt

Dark green-brown, fine-grained, unaltered basalt (BSLT) is the most common lithology in the Balmer Assemblage but is less abundant in the Confederation Assemblage. Basaltic flows are typically massive but variations include pillowed, flow top breccia and hyaloclastic textures. Confederation basalt has low prospectively for gold mineralization and is typically massive.

Gabbro

Dark grey, massive, equigranular, medium- to coarse-grained gabbro (GBRO) is Fe-rich relative to older Balmer gabbro. None of the gabbros identified on the property are prospective for gold mineralization.

7.2.3 Veins

Quartz-Carbonate Veins

Wispy, discontinuous quartz-carbonate veins (VQCB) commonly fill tension gashes and extensional zones in BSLT and GBRO. They do not carry gold and are not associated with gold-bearing structures.

Early Carbonate-Magnetite Veins / Diopside Replacement Veins

White-grey to violet-grey, massive to dismembered, fine-grained, carbonate-magnetite veins (VECB; Figure 7-4), occur only within Balmer Assemblage rocks and were emplaced early in the deformation history based on their degree of deformation and metamorphism. During amphibolite facies metamorphism and metasomatism, these veins were almost completely replaced by a skarn-like assemblage best characterized as light green, massive, coarse crystalline diopside-quartz-amphibole-calcite-biotite veins. This replacement appears to be most pervasive and intense in the eastern parts of the property, perhaps due to thermal contact metamorphism in proximity to the Killala-Baird batholith. Where this process has occurred, the veins are assigned the logging code VNDI as opposed to VECB.

These veins (VECB and VNDI) are the primary vein type associated with gold mineralization. Gold is commonly found in and around these veins at the full range of grades measured on the property, with the best grades coinciding with more siliceous parts of the vein system. Whether this silica (quartz) content is a primary component of the veins or came later in a secondary silicification (+gold) event is unclear.

Quartz Veins

Fine-grained, white to translucent quartz-dominant veins (VNQZ) cut both Balmer and Confederation rocks. These veins do not have a clear association with any gold-bearing structures although a few contain gold.

Blue-Grey Quartz Veins / Pervasive Silicification

Blue-grey to white, massive, recrystallized quartz veins (VBGQ; Figure 7-4) are associated with gold mineralization at the Wedge, Fork and Russet deposits, and at the 8 Zone in the Mine; they have only been identified within Balmer host rocks. This vein set was folded and/or boudinaged by D_2 deformation, indicating a pre-D_2 or syn-D_2 timing of emplacement. These veins are highly prospective for gold mineralization. Gold is present as unevenly distributed, discrete gold grains within the vein mass. Narrow zones of biotite and amphibole are commonly present on the immediate selvedges to these veins. This vein set is interpreted to correlate with pervasive silicification within the Madsen deposit where discrete individual quartz veins are rare.

Figure 7-4: Early veins

Note: Early, blue-grey quartz vein (VBGQ, top) with several flecks of gold. Vein shows typical curviplanar margins consistent with folding and boudinage, characteristic of these veins and evidence of early tectonic timing. VBGQ veins are the main gold host at the Fork, Russet South and Wedge deposits. Early carbonate veins (VECB, bottom) are locally widespread proximal to auriferous zones on the mine property but rarely contain significant gold. Early timing is evidenced by widespread, locally pervasive to complete replacement by metamorphic mineral phases (principally diopside, chlorite and amphibole). Photos of full drill core (top) and sawn half drill core (bottom).

Source: D. Baker (2018) with photos supplied by Equity Exploration Consultants Ltd.

Quartz-Tourmaline Veins

Quartz-tourmaline veins (VQTM) fill tensional fractures in unaltered basalt and gabbro. At the Treasure Box target and other gold prospects in the region (e.g., Buffalo Mine) these veins host bonanza-grade gold. These veins are common across the Red Lake Greenstone Belt particularly proximal to the Dome Stock, suggesting a temporal and genetic relationship. They cut the S_2 foliation, but also occupy shear veins and tension veins within the Dome Stock, so are tectonically syn-kinematic. They cut VBGQ and VECB veins at the historical Redaurum Mine (Figure 7-5).

Figure 7-5: Quartz tourmaline veins

Note: Gold mineralized quartz tourmaline veins (VQTM) cutting deformed gold mineralized VECB and VGBQ at the Redaurum shaft adjacent to the mine property. The VQTM are indicated by red arrows and the VECB and VGBQ by a blue arrow.

Source: D. Baker (2018) with photos supplied by Equity Exploration Consultants Ltd.

7.2.4 Metasomatized Rocks

Balmer Assemblage rocks vary from weakly foliated and undeformed volcanic rocks with well-preserved fine-scale primary volcanic features (e.g., pillow structures, spinifex texture, varioles) to mafic and ultramafic igneous rock that has been pervasively altered, deformed and metamorphosed such that no primary features are discernible. Such rocks are commonly associated with gold mineralization across the property at all known gold-bearing zones except Treasure Box (which is characterized by late quartz-tourmaline veins without significant wall-rock alteration).

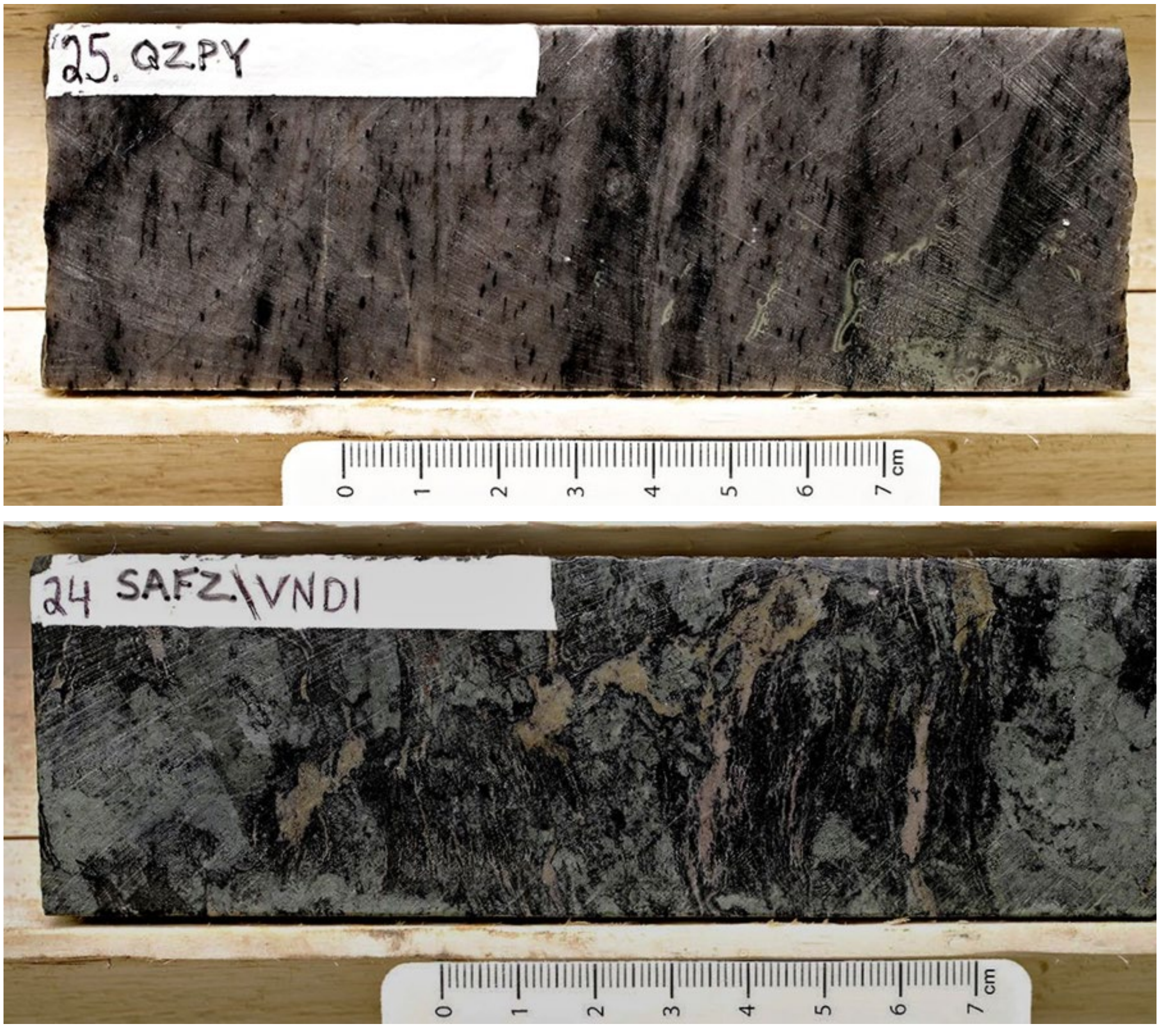
Metasomatized and hydrothermally altered Balmer volcanic rock locally forms coherent units cut by early, planar gold-associated alteration zones as described above. In these most intensely altered rocks the original protolith has been completely recrystalized and primary lithologies are not identifiable. Three codes were developed to describe these most important rocks that are defined by secondary mineral assemblages as described in the following sections.

The distinction between metasomatic mineral phases (those derived from interaction with hydrothermal fluid) and metamorphic mineral phases (those derived from metamorphism generally in an isochemical system) is difficult to discern in these altered units. Regional metamorphism (synchronous with, or late-D_2 deformation) has overprinted the deposits to the degree that the host rocks to the gold-bearing zones are characterized by a complex mineral assemblage that has grown during regional metamorphism and because the host rock surrounding the deposits was altered prior to metamorphism, an assemblage of abundant metamorphic biotite, garnet, amphibole and diopside resulted such that these minerals are useful as proxies for hydrothermally altered Balmer rock. So, these sensu stricto metamorphic minerals are herein treated as alteration indicators and this has proven an effective approach to delineating the alteration envelope surrounding gold mineralization. The three metasomatic rock assemblages identified are described in the following sections.

Strongly Altered and Foliated Zone

Strongly altered and foliated zone (SAFZ) refers to coherent domains of rock that are altered and foliated to a degree that the protolith is completely unrecognizable (Figure 7-6). These domains of intense alteration and strong foliation overprint follow early structural corridors that were exploited by gold bearing fluids and delineate areas known to host gold mineralization. Zones of strong silicification within the SAFZ are especially prospective for gold mineralization.

Figure 7-6: Key mineralization-associated rock types



Note: Two distinct rock types that show close relationship to auriferous zones. Quartz porphyry (QZPY, top) is characterized as light to dark grey, fine-grained and weakly quartz-phyric. QZPY intrusions occur in concordant to planar gold-bearing zones interpreted to be early shear zones. QZPY pre-date gold mineralization but are not commonly mineralized themselves. They are commonly proximal to gold mineralization. Strongly altered and foliated zones (SAFZ, bottom) are characterized by texturally destructive biotite, diopside and potassium feldspar that are considered to result from a metamorphic overprint of a carbonate altered basalt. SAFZ and veins hosted within it are the main gold hosts at the Mine. Photos are of sawn half drill core.

Source: D. Baker (2018) from photographs supplied by WRLG (2024)

The appearance of SAFZ is variable between different areas of the deposit, with the variability controlled by host rock composition, the character of mineralizing fluids and the degree of post-mineral metamorphism / metasomatism. For example, in the southwestern portion of the McVeigh zone a well-developed SAFZ is generally defined by the presence of 1 cm to 2 cm thick wispy bands or ribbons of cream-brown biotite-potassium feldspar (microcline) separating larger bands and lenses of diopside, green amphibole and (locally) quartz and carbonate. This SAFZ typically contains abundant VECB and VNDI veins which are commonly transposed into the main fabric of the foliation. In contrast, SAFZ present in the northeastern, near surface area of the Austin zone is characterized by strong biotite alteration, moderate foliation, weak to moderate background silicification and narrow (0.5 – 2 cm wide) ptygmaticly folded VECB veins. Throughout the deposit the sulphide content of SAFZ is highly variable but ranges up to 10% pyrite-pyrrhotite-chalcopyrite-arsenopyrite. There is generally very limited to no correlation between sulphide content and gold values, with the exception being fine grained arsenopyrite, which can correlate with gold.

The mapping or logging of SAFZ provides an indicator of intensity of alteration within a deposit-scale controlling structure.

Pervasively Altered Basalt

Pervasively altered basalt (BSLA) refers to coherent domains of moderately foliated, pervasively biotite-amphibole altered rock that is generally interpreted to have a mafic volcanic protolith. BSLA is transitional from the more intensely altered and proximal SAFZ. Primary textures can be locally preserved in BSLA.

The BSLA domains are interpreted to represent the marginal envelope of structural alteration corridors that were exploited by hydrothermal gold mineralization. The marginal setting with reduced alteration intensity has moderate potential to host gold mineralization.

Biotite-Amphibole Altered Peridotite

Biotite-amphibole altered peridotite (PRBA) refers to domains of moderately to strongly altered and foliated peridotite first identified within the Russet Lake Shear Zone proximal to the 8 Zone. PRBA is interpreted as the ultramafic-derived equivalent of basalt-derived SAFZ and shares a common association with gold as SAFZ, best exemplified in the 8 Zone.

7.2.5 Plutonic Rocks

Monzonite

Monzonite (MNZT) is grey, unfoliated, medium-grained, equigranular and includes intrusive rocks that are part of the Faulkenham Lake Stock and dykes and sills in Balmer and Confederation rocks thought to have been deposited during emplacement of the Faulkenham. These monzonite bodies are characteristically epidote and hematite altered. The Faulkenham Lake Stock post-dates gold deposition at the mine.

Granodiorite

Granodiorite (GRDI) includes white to light grey, unfoliated, medium- to coarse-grained, equigranular plutonic rock of the Killala-Baird Batholith. The post-tectonic ca. 2704 Ma Killala-Baird Batholith post-dates mineralization and therefore is not prospective for gold.

7.2.6 Dykes and Sills

Intermediate Intrusive

Intermediate intrusive (IINT) are grey, undeformed, fine- to medium-grained dykes that crosscut both Balmer and Confederation group rocks and cut gold mineralization in the Mine. These dykes have sharp, chilled margins and locally tend to strike concordant to the property-wide foliation suggesting they exploited the S_2 structural grain. IINT have been dated at ca. 2698 Ma from the Madsen deposit and ca. 2696 Ma from the Wedge deposit and provide a minimum age for gold mineralization (Dubé et al., 2004). Their spatial distribution, similar composition and age suggest that they may be genetically related to and sourced from the Killala-Baird Batholith.

Mafic Intrusive

Mafic intrusive (MINT) are dark grey, post-tectonic, fine- to medium-grained dykes that cross-cut Balmer and Confederation rocks. They have sharp, typically chilled, margins and post-date mineralization. They are interpreted to be Proterozoic in age.

Quartz Feldspar and Feldspar Porphyry

Quartz-feldspar porphyry (QFPY) and feldspar porphyry (FSPY) dykes are intermediate, grey- pink, unfoliated and are only known to cut Balmer rocks. They are interpreted as post-tectonic, likely Killala-Baird Batholith related. They are not prospective for gold mineralization.

Hornblende Feldspar Porphyry

Hornblende-feldspar porphyry (HFPY) dykes are intermediate, dark grey-pink and are most common in the Russet Lake area and in the East ramp area of the Mine. HFPY dykes have not been found cutting the Confederation Assemblage; however, they are interpreted to post-date the Confederation Assemblage due to their lack of foliation. It is possible that these dykes are a local phase of the FSPY. These dykes post-date mineralization and are not prospective for gold.

Quartz Porphyry

Quartz porphyry (QZPY) refers to a set of felsic, light to medium grey, foliated, quartz-phyric or fine-grained (aphyric) dykes that are spatially associated with gold-bearing zones in the Madsen and Wedge deposits (Figure 7-8). Porphyritic examples contain a few percent rounded, quartz phenocrysts and foliation-parallel biotite aggregates. These dykes are pervasively sericite altered and sodium-depleted (Mackie, 2016). Proximal to gold-bearing zones, early carbonate veins (VECB) cut QZPY dykes and amphibole-quartz-diopside replaces QZPY. Collectively, this is strong evidence that QZPY dykes predate the main gold event. Their stratigraphically discordant nature and parallel occurrence in the same altered structural corridors suggests that these intrusions exploited the same early structures that controlled the gold-bearing hydrothermal systems. Underground exposures confirm that QZPY dykes are locally tightly folded with the penetrative S_2 foliation axial planar to the folds.

Russet Mafic Intrusive

The Russet Mafic Intrusive (RSMI) unit occurs as relatively narrow (1 – 20 m thick) intrusive units within the Russet Lake Ultramafic. It is spatially directly associated with the early altered structures hosting gold mineralization. It seems to be a direct correlative for the QZPY in the basaltic host rocks. The RSMI are medium to fine grained, with a rock mass consisting of plagioclase and amphibole; when coarser grained it can have a gabbroic texture.

Silicification is common and can be associated with up to several percent fine grained disseminated pyrite/pyrrhotite. Visible gold is present in some examples, though rare. The unit is anomalously high in TiO_2 (>0.7%) and Na_2O (>3%).

7.2.7 Structural Geology

Given the significant role that deformation-related structures (e.g., shear zones and fault zones) play in transporting and focusing gold-bearing fluids in orogenic gold systems, determining the structural architecture and deformation history of the mine property was a focus of surface exploration work after Pure Gold acquired the property (Baker, 2014a, b; Baker and Swanton, 2016; Cooley and Leatherman, 2014a, b, 2015). Additionally, oriented core drilling data along with three-dimensional interpretation of major lithological contacts has constrained the relations between the host stratigraphy, gold-bearing structures and deformation features.

Based on outcrop, underground exposure and drill core observations, most supracrustal rocks exhibit a tectonic foliation which is the most common structural element present across the property. The intensity of this foliation varies widely from a decimetre-scale-spaced planar fabric to an intense, sub-millimetre-spaced schistosity with localized shear-related fabrics. Unaltered mafic rock units such as massive and pillowed Balmer basalt (BSLT) typically do not exhibit strong tectonic foliations. By contrast, felsic units of the Confederation Assemblage (FVOL) and altered Balmer basalt readily develop foliations owing to a bulk chemistry that encourages phyllosilicate (mainly sericite) growth during strain-related recrystallization and metamorphism. Such rheological contrasts have led to a significant amount of strain partitioning throughout the belt, making it difficult to distinguish between deformation events and correlate certain structures, like foliations, regionally. Nevertheless, four distinct deformation events can be recognized on the mine property, even if their regional extent is not completely known.

The first phase of deformation, D1, is poorly defined due to a lack of penetrative foliation. The strongest evidence for D1 is the property-scale map pattern showing repetition of Balmer stratigraphic units on the east and west sides of the Russet Lake ultramafic body. Opposing pillow top way-up indicators in both ultramafic and mafic rocks (Atkinson, 1993 and Cooley and Leatherman, 2015) indicate that the Russet Lake ultramafic occupies the core of an isoclinal antiform with an overturned western limb. Type II interference folds have also been recognized near the hinge area to the northeast of this large antiform and further south along its eastern limb at the Wedge (86) target; both confirming the overprint of two folding events (F1 and F2). Although no widespread penetrative foliation developed during the F1 folding event, a central high strain zone of intense foliation development has been identified within the hinge of this large antiform and named the Russet Lake Shear Zone.

The second generation of regional deformation (D_2) on the mine property includes a conspicuous, penetrative regional foliation (S_2) which is generally consistent with the D_2 structural trends of Sanborn-Barrie et al. (2004b). This foliation consistently transects most units on the mine property, including the unconformity with Confederation assemblage rocks. S_2 is axial planar to minor (tens of metres scale) S-shaped folds (F_2) defined by lithological contacts as well as folded mineralized domains within the Madsen Mine (e.g., Horwood, 1940, and current MRE domains).

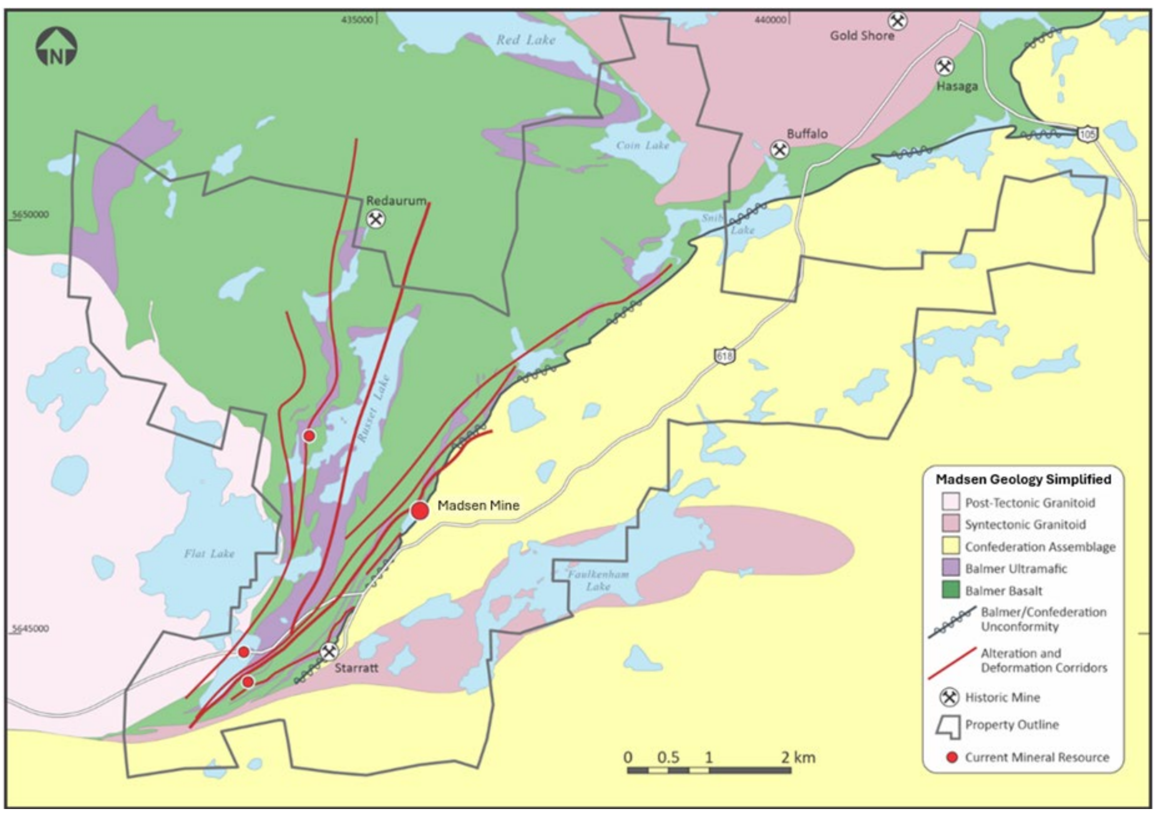
A third deformation event (D_3) is expressed strictly by locally developed Z-shaped folds (F_3) ranging from centimetres to kilometre scale (e.g., folded ultramafic northwest of Coin Lake). These structures have no associated penetrative fabric, and they fold the S_2 foliation along with the affected stratigraphic units. F_3 folds are consistently Z-shaped, in contrast to the opposing S-shaped asymmetry that is characteristic of F_2 folds.

The youngest deformation (D_4) to affect the mine property is localized brittle faulting. Such faults are rare across the property, particularly in the mine area but are common at Starratt where they are characterized by metre-scale intervals of fault breccia and fault gouge recovered in drill core. These are mostly steeply dipping, approximately east-west trending and related to faulting along the southern contact of the Killala-Baird Batholith (e.g., the Liard Lake fault of Sanborn-Barrie et al., 2004b). These faults clearly post-date gold deposition as they locally displace gold mineralized lenses at the north end of Starratt, but offsets seem to be less than a few metres.

7.3 Property Mineralization

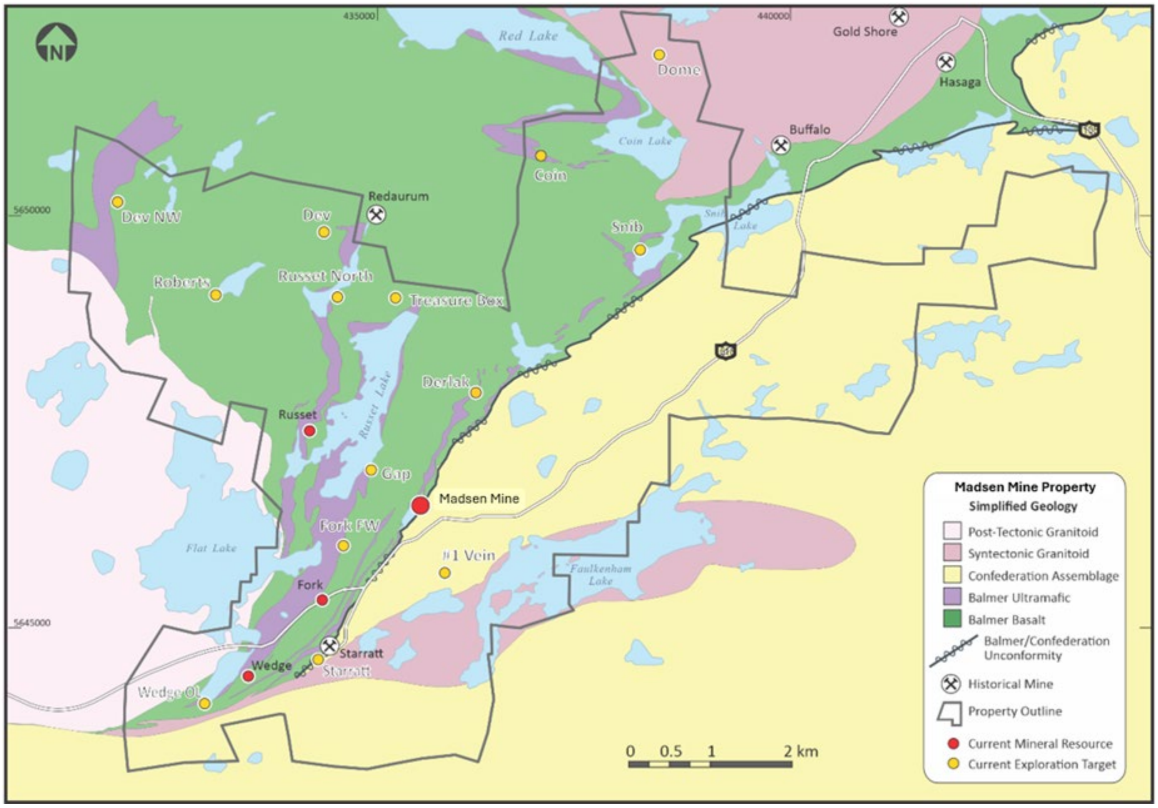
The following sections summarize the geology, geometry and style of the significant gold-bearing alteration corridors and associated targets present on the Madsen Mine property (Figure 7-7 and Figure 7-8). A closer view of the projected distribution of the gold deposits is shown in Figure 7-9.

Figure 7-7: Major alteration corridors associated with gold mineralization



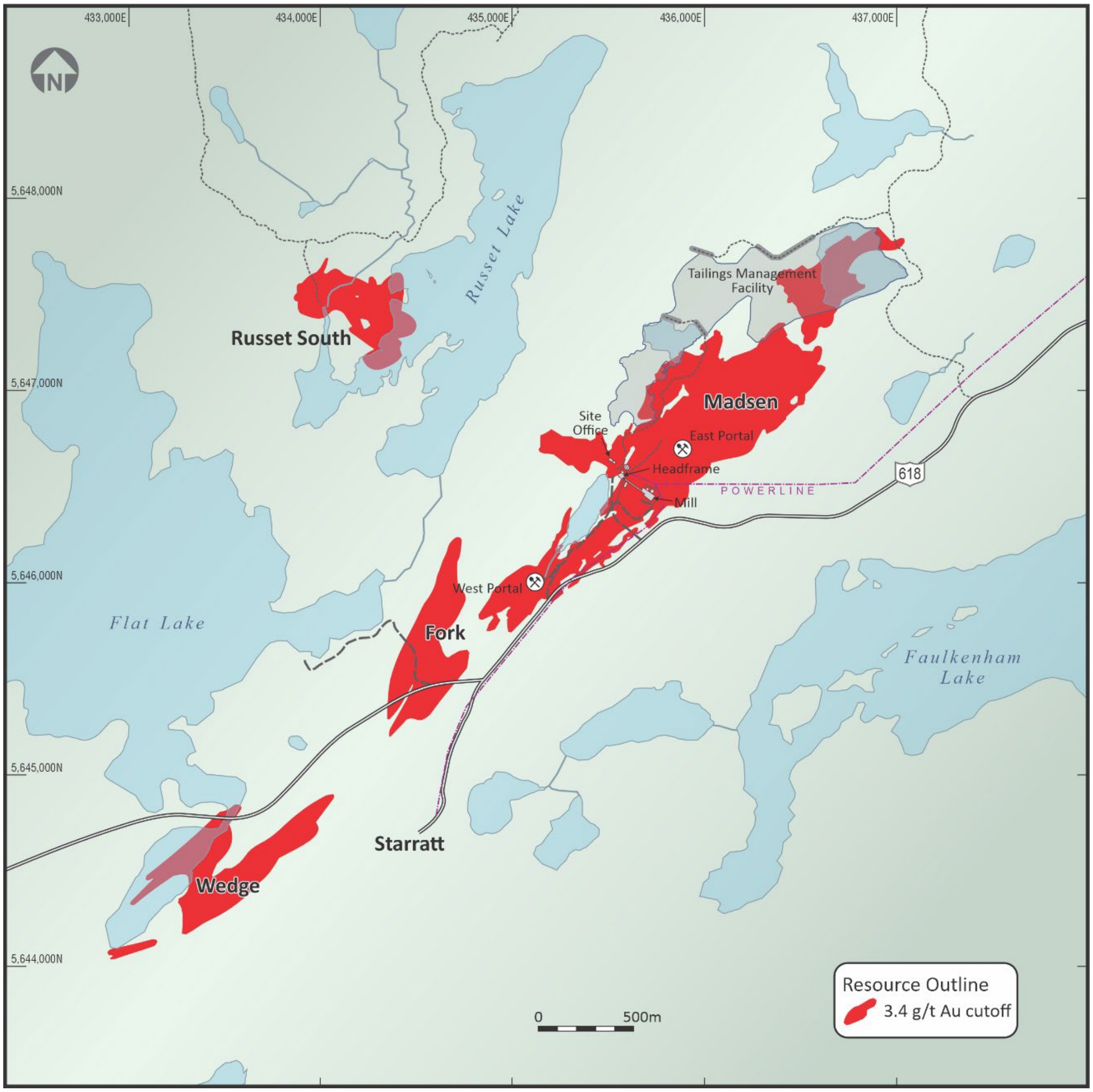
Note: Summary geological map of the mine property. Property outline shown by solid black line.
Source: WRLG (2024)

Figure 7-8: Historical mines, deposits and exploration targets



Note: Summary geological map of the mine property. Property outline shown by solid black line.
Source: WRLG (2024)

Figure 7-9: Plan map of Madsen Mine resource domains



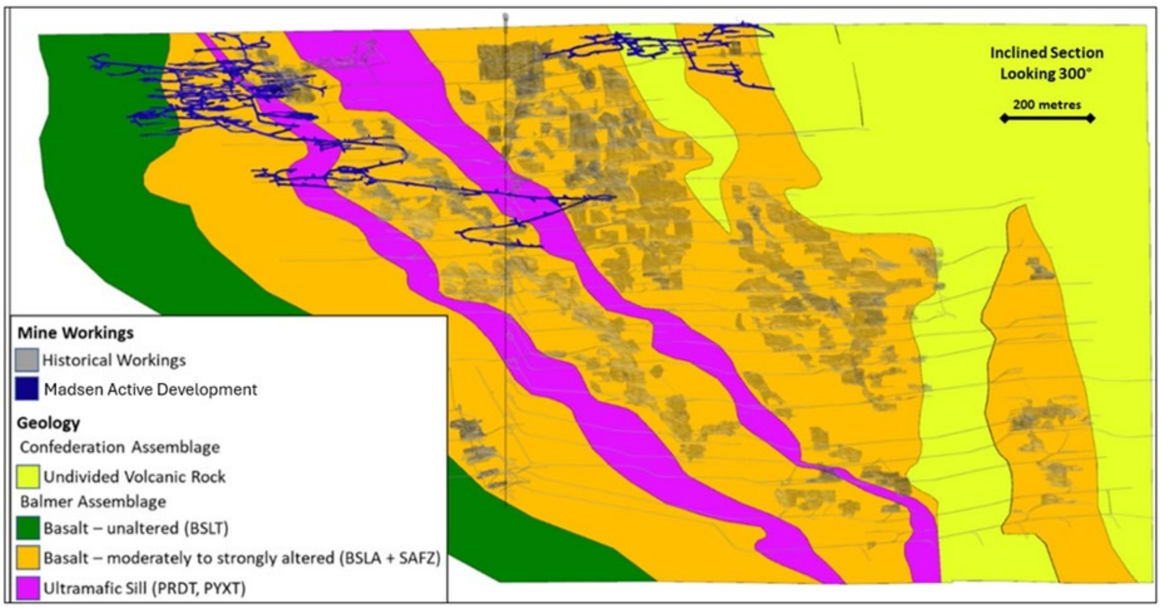
Source: WRLG (2024)

7.3.1 Madsen Deposit – Austin, South Austin and McVeigh Zones

Most of the historical gold production and most of the current mineral resources at the Madsen Mine are within the Austin, South Austin and McVeigh zones which, along with the 8 Zone, comprise the Madsen deposit. At the scale of the property, these zones all lie within much broader, kilometre-scale planar alteration and deformation corridors that have been repeatedly reactivated during gold mineralization and subsequent deformation and metamorphism (Figure 7-7).

The distribution of gold within these planar structures is almost exclusively within variably altered basalt, and enhanced in close proximity to major lithological contacts, such as ultramafic sills, felsic dykes and felsic volcanic strata. The overall plunge of the different deposits occurring close to the Confederation assemblage (e.g., Madsen, Starratt and Wedge) is controlled by the intersection of the mineralized planar structures with the local stratigraphy, as exemplified by its intersection with two barren ultramafic sills in the Madsen Mine (Figure 7-10). Here, the northeastern boundary of the Austin zone coincides with the moderately northeast-plunging intersection between the mineralized structure and the Confederation felsic volcanics. The mineralized structure forms horsetail splays of mineralized lenses as it approaches and finally terminates against the felsic volcanic rocks in the deposit hanging wall. Moving southwest, the mineralized structure cuts downward through the Balmer stratigraphy, cutting the two ultramafic sills. The lines of intersection between the (Figure 7-10) mineralized structures and these two sills plunge northeast and form the boundaries of the Austin (between the Confederation felsic volcanics and the first ultramafic sill) and the South Austin (between the two ultramafic sills) zones.

Figure 7-10: Inclined long section (oblique view towards the northwest) through the Austin and South Austin zones with projected geology



Note: Mined historical stopes (grey) demonstrate gold-bearing zones which show a strong northeast plunge at about 40°. The projected geology shows the strong proximal alteration zones (orange) surrounding high grade gold mineralization (here represented by mined stopes) and, significantly, that these zones are interrupted by ultramafic sills (purple). Long section geology contacts were determined from detailed geological interpretation on 30 level plans from surface to below the deepest developed part of the mine.

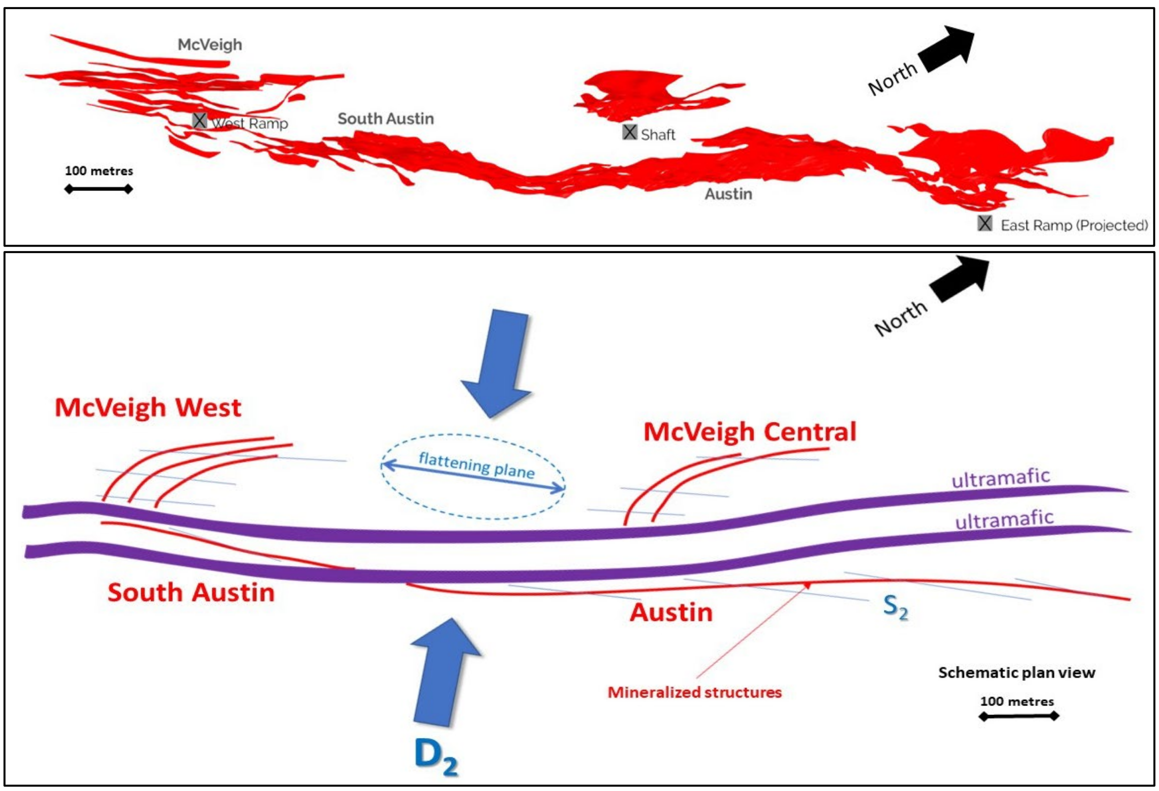
Source: WRLG (2024)

These geometrical features are important for two reasons: (1) they illustrate the fundamental structural control on the overall shape and distribution of the gold bearing system, and (2) the transecting relationship between the mineralized structure and its host stratigraphy contradicts previous interpretations of this deposit's origin as a 'stratabound' gold deposit.

Similar to other structurally controlled gold deposits, the Madsen deposit comprises second order footwall splays that exhibit a slightly different structural style to the main Austin/South Austin structure. In the footwall of the second ultramafic sill, the McVeigh zones occur as footwall splays, with a steeper dip and more northerly trend, off the main structure (Figure 7-11).This difference in orientation relative to the main zones results in a reversal in the plunge of their intersection with the host stratigraphy, and thus, the McVeigh mineralized shoots plunge to the southwest, instead of northeast like those in the main zone. This same plunge reversal is repeated at Starratt, which has both northeast- and southwest-plunging shoots on either side of a large ultramafic sill, and at Wedge, which plunges predominantly towards the southwest.

The more northerly orientation of the footwall splays also places them at a higher angle to the flattening plane of the D_2 deformation (schematic plan in Figure 7-11). This angular position makes these zones more susceptible to greater degrees of rotation, folding and transposition during the ~northwest-southeast-directed D_2 shortening event compared to the main zones. The impact of this deformation on the McVeigh zones is reduced continuity and more irregular geometries of the mineralized lenses.

Figure 7-11: Inclined view of the Madsen deposit resource domains showing the angular relationship between the McVeigh footwall splays (West and Central) and the main zone formed by Austin and South Austin. Schematic plan illustrates relationships to D_2 shortening.



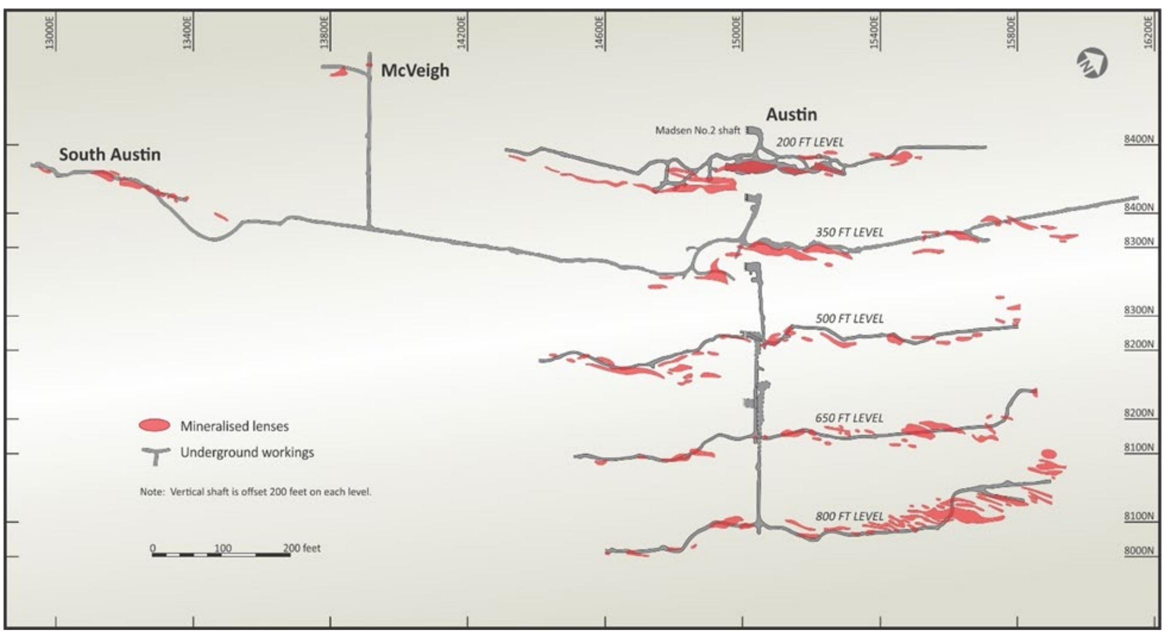
Note: The steeper dip and more northerly trend of the McVeigh zones results in a reversal of the plunge of their intersection with the host stratigraphy, and accounts for the observed reversals in plunge of mineralized shoots in the mine and across the property. In addition, the higher angle of the footwall splays to the flattening plane of the D_2 deformation makes them much more susceptible to rotation, folding and transposition during D_2 than their main zone counterparts.

Source: WRLG (2024)

The Austin and South Austin Zones are open down plunge from the deepest levels of past mine development and from the deepest parts of the current MRE. Drilling in 2011 intersected 14.3 g/t Au over 2.0 m at 825 m below past mining, and drilling by Pure Gold in 2021 returned 1.99 g/t Au over 2.0 m within a package of silicified SAFZ and QZPY lithologies at 500 m down plunge from past mining of the Austin Zone on 18 Level and 250 m along strike from past mining on 24 Level. Drilling in 2017 by Pure Gold returned 34.6 g/t Au over 4.3 m at 240 m below past mining in the South Austin Zone. Detailed drill core review and geological modelling has confirmed alteration and host rock continuity at these depths.

Viewed in more detail (Figure 7-12), the mineralized zones are comprised of trains of lenticular gold-bearing zones that have been variably rotated into the penetrative S_2 foliation. Locally, gold-bearing bodies show stope-scale S-shaped fold geometries (e.g., the 350-foot level in Figure 7-12) consistent with small-scale folds mapped at surface and interpreted in geological level plans. At smaller scales observed underground and in drill core the structural features indicate intense transposition of a pre-existing gold-bearing vein system into the S_2 orientation. In some cases, the enveloping surface of deformed veins is more shallow-dipping than the overall structure and S_2 foliation, indicating that, prior to its transposition, the vein system included both steep and flat veins – a common feature of structurally-controlled gold vein deposits elsewhere (e.g., Sigma-Lamaque, Val d’Or). Similarly, backing out the transposition deformation of the McVeigh zones (west and central) reveals them to be steeply dipping footwall splays of a moderately dipping structure defined by the Austin and South Austin zones – also a common feature of structurally-controlled veins systems.

Figure 7-12: Historically mined gold-bearing shapes from original level plan maps



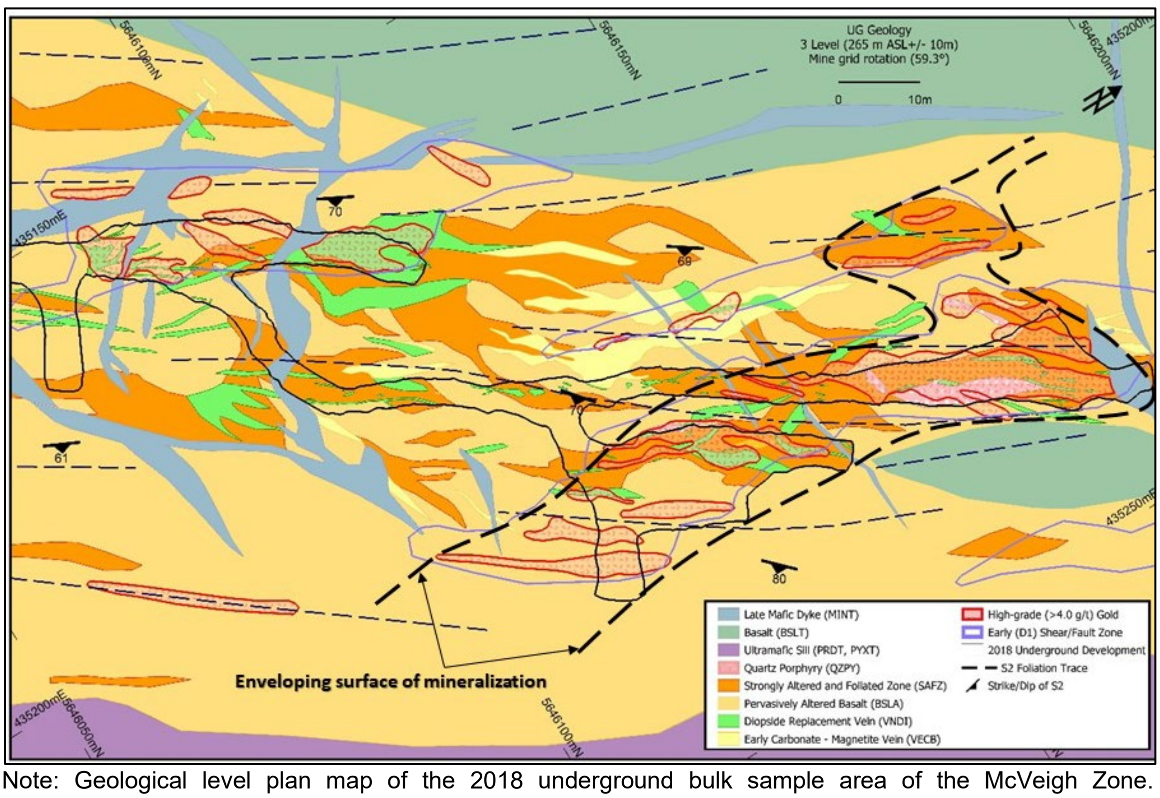
Note: Composite level plan maps of gold-bearing lenses drawn from mining control plans during the earliest periods of mining at the Madsen Mine. The gold-bearing zones shown by red shading were demarcated by mine geologists as high-grade zones. Note that these shapes have two orders of control. At a smaller-scale, gold occurs within a series of left-stepping lenses that are transposed within the S_2 foliation. At a larger scale, these lenses collectively define planar structures that are continuous for many 100s of metres. Levels are offset for effect.

Source: WRLG (2024) after Horwood (1940)

Together, these features (i.e., footwall splays, steep and shallow vein structures, strong alignment with structural intersections) elucidate a history of classic brittle-ductile gold-bearing vein emplacement and alteration, strongly influenced by the presence of local competence contrasts within the host stratigraphy. This gold vein system was subsequently overprinted by an intense ductile transposition deformation (D_2), itself accompanied or followed by amphibolite facies metamorphism. Peak metamorphism post-dated the deformation, as evidenced by the lack of fabric development in the metamorphic mineral assemblage (diopside-amphibole-biotite-garnet). The intense deformation and metamorphism has largely obscured the primary morphology and mineralogy of the deposit, which led to varied and unusual interpretations of its origins by previous workers. The current understanding of the deposit provides a predictive basis for short-term grade control models, exploration and delineation drilling and life of mine resource modelling. Confidence in this interpretation grows steadily with increasing mine development and geological observation.

The 2018 bulk sample project provided significant, detailed information on a small portion of the McVeigh Zone of the Madsen deposit (Figure 7-13) and a test for the geometries and relations described above. Detailed structural observations and data collection has confirmed the relationships between gold-bearing lenses and the penetrative foliation (S_2). High-grade gold lenses form a similar pattern to that recorded on historical level plans where individual tabular lenses show a left-stepping pattern, but the overall enveloping surface of mineralization is continuous across the S_2 fabric, as seen in the Austin Zone (Figure 7-12 and Figure 7-13).

Figure 7-13: Geological level plan map of the 2018 McVeigh Zone bulk sample



Note: Geological level plan map of the 2018 underground bulk sample area of the McVeigh Zone. Interpretation is derived from historical, exploration, resource and bazooka drill hole data as well as detailed underground back and face mapping. Gold-bearing high-grade outlines are determined from detailed face chip sampling as well as drill core analyses. The pattern that emerged is very similar to that present on many historical underground level plans (e.g. Figure 7-12) that is characterized by high-grade gold-bearing lenses that are aligned and transposed parallel to the penetrative S_2 foliation but that overall are contained in planar zones that are oblique to S_2.

Source: Equity Exploration Consultants Ltd. (2019) after drawing by D. Baker and R. Scott.

The penetrative S_2 foliation within the bulk sample area as measured in underground exposures is consistently SE-dipping (average orientation of 038/65, UTM grid) and compares very well with data collected from the same region in oriented drill core.

Small-scale folds within the bulk sample area include tight folds of narrow quartz porphyry dykes and folds of diopside veins. These folds generally plunge -60° towards 110° (UTM) which is similar to the orientation of map-scale folds interpreted from level plan geological interpretations.

In drill core, or at underground face exposures, gold-bearing zones at the Madsen Mine are best identified visually by fine (sub-millimetre) grains of free gold within strong alteration and veining. All high-grade intervals generally contain visible gold on drill core exteriors, although numerous examples exist of high-grade assays where visible gold was only identified within the interior (cut surface) of the core samples. Sulphides (primarily pyrite and pyrrhotite with minor arsenopyrite and chalcopyrite) are relatively common throughout the deposit, though they do not appear to have any direct correlation with gold. It is believed that present sulphide abundance reflects primary sulphide abundance or alteration in the host rock and does not serve as a marker for gold mineralization, suggesting it was not introduced by the mineralizing fluids. Apart from the presence of free gold, pervasive silicification (locally accompanied by discrete quartz veining) and quartz-carbonate (VECB) or diopside (VNDI) veining are the best indicators that a given interval is within a high-grade lens within the mineralized structure.

7.3.2 Madsen Deposit: 8 Zone

The geology and mineralization style of the 8 Zone is somewhat distinct from that of other known zones within the Madsen deposit. Gold in the 8 Zone occurs within strongly altered and veined peridotite of the Russet Lake Ultramafic (see Section 7.2.4 for description of the PRBA unit). By contrast, most gold at the McVeigh, Austin and South Austin Zones is hosted within mafic rocks proximal to generally barren ultramafic units. The 8 Zone has a planar geometry, strikes generally north-south and dips to the east at approximately 45° which is significantly shallower than the other zones. As it is presently modelled, the 8 Zone is approximately 130 m along strike by 700 m down dip and 30 m in thickness.

Within this mineralized plane, gold occurs in highly deformed, centimetre- to metre-scale blue-grey recrystallized quartz veins (VBGQ) located within a corridor of amphibole-biotite alteration (PRBA) which is generally on the order of tens of metres wide. The more intense zones of alteration occur as 1–10 cm intervals of near-total replacement by biotite, an abundance of blue-green amphibole and a well-developed foliation defined by alignment of biotite.

Relogging of historical core from surface and underground holes drilled by past operators at 8 Zone and throughout the Russet Lake Ultramafic along with drilling and geologic modelling by Pure Gold improved the geologic model and understanding of 8 Zone. The deposit lies within the Russet Lake Shear Zone which, significantly, has been modeled along a strike length of 5 km and to nearly 2 km depth (Baker, 2017). This early structure dissects the length of the Russet Lake ultramafic volcanic unit which represents the lowermost Balmer Assemblage rocks on the mine property.

7.3.3 Russet Deposit

Gold at the Russet deposit is hosted within folded and/or boudinaged blue-grey quartz veins that are similar to those that are characteristic of the 8 Zone. At Russet, the veins mostly occur within weakly deformed 10 m-scale wide, planar zones proximal to the northern contact of Russet Lake ultramafic volcanic rocks and on both the hanging wall and footwall of a smaller ultramafic sill parallel to this contact. The veins are most commonly hosted within relatively weakly biotite-amphibole altered basalt, though some occur within ultramafic rock and underlying iron formations. Despite the complicated arrangement of individual veins, due to their transposed nature, zones of high vein density, deformation, alteration and gold mineralization can be defined over hundreds of metres of strike length, trending broadly sub-parallel or at low angle to stratigraphy which is itself broadly folded about south-plunging F2 folds in the Russet Lake area. In places, gold appears to follow spaced axial planar shear zones within a broader F2 fold, but does not extend far from the zone sub-parallel to stratigraphy. Projected to surface, these zones of high vein density extend over a footprint of approximately 650 m by 650 m and have been defined to a vertical depth of 200 m.

7.3.4 Wedge Deposit: 86, DV, CK, MJ and OL Zones

The Wedge deposit comprises four resource zones (DV, CK, MJ and OL) and one mineralized zone (86) that remains at the exploration target stage. All five zones generally correspond with historical surface showings and mineralized areas (Branson, 2019a). Drilling, core re-logging and interpretation by Pure Gold, however, showed that these historical zones all lie within a series of early syn- to pre-D_2 structures (Nuttall, 2017).

The DV and CK Zones lie within the same structure that hosts the Fork Main Zone, but about 900 m along strike to the southwest. The intervening area is prospective for potential resource expansion and this area includes the 86 Zone exploration target. The 86 Zone was explored in 1998 by mechanical stripping and recent mapping of these outcrops (Cooley and Leatherman, 2015) suggests that 86 Zone may represent the southern extension of the Fork deposit as the host rocks are continuous and the style of mineralization is similar. Drilling by Pure Gold (PG17-359) has tested the current southern limit of the Fork deposit along strike to the north of 86 Zone. At 86 Zone, rock sampling by Pure Gold of outcropping iron formation characterized by banded magnetite, pyrrhotite and amphibole has returned highly anomalous gold values. Drilling directly underneath this surface mineralization in 2017 returned multiple intercepts exceeding 5 g/t Au (up to 22.9 g/t Au over 1.1 m). Gold is hosted in quartz veins spatially associated with both iron formation and altered basalt.

In detail, the DV and CK Zones comprise a series of up to three concordant resource shapes across a collective width of 70 m and a maximum strike length of 700 m. At the DV Zone, gold is hosted within discontinuous quartz ± chlorite-amphibole veins (VBGQ veins) with biotite-amphibole-diopside selvedges and minor pyrite, pyrrhotite, chalcopyrite and arsenopyrite (Branson, 2019b). These veins are hosted in weakly altered mafic volcanic rocks or more commonly within moderately to strongly altered mafic volcanic rock (BSLA or SAFZ). At the CK Zone, the geology and mineralization are comparable to the DV Zone, though the host basalt rocks have been cut by quartz porphyry. A key relationship is that the veins and the enveloping alteration zones are transected by and transposed into the main S_2 foliation of the host rocks – an identical relationship to the Madsen deposit.

The OL Zone lies about 450 m southwest along strike from the edge of the CK Zone resource shape in an area characterized by deformed gold-bearing quartz veins hosted in zones of deformed quartz porphyry (QZPY) and strongly altered foliated zones (SAFZ). Outcrop stripping, surface rock sampling and diamond drilling by Pure Gold delineated two parallel trends of alteration and veining separated by approximately 25 m and extending for a strike length of 200 m. The zone is open both along strike and at depth.

The MJ Zone is hosted by two concordant shear zones up to 40 m in width characterized by deformed gold-bearing quartz veins hosted within altered and deformed basalt and peridotite within the Russet Lake Ultramafic. Current drilling has delineated these shear zones over 500 m of strike length and to 320 m depth with the structure remaining open along strike and down-dip.

In addition to being a part of the recognized property-wide structural architecture associated with gold mineralization responsible for mineralization at Madsen, Fork and Russet, the Wedge deposit exhibits similar high-level characteristics to the Madsen deposit (same alteration and structural timing), however gold tends to be more often hosted in discrete quartz veins rather than disseminated within intervals of pervasively silicified rock, as is more common in the Madsen deposit.

The apparent plunge of mineralization along these structures – best demonstrated at the well-tested DV Zone – appears to be associated with the intersection of the structures and major rheological and geochemical contrasts between relatively rigid and massive basalt and adjoining IRFM and ultramafic units (Branson, 2019b). This architecture is comparable to the plunge at the Austin and South Austin zones in the Madsen deposit which are defined by intersection of the mineralized zones and mafic/ultramafic contacts.

7.3.5 Fork Deposit and Fork Footwall Target

The Fork deposit lies within two concordant shear zones spaced 100-150 m apart. These structures strike north-north-easterly and dip about -60°. The upper lens is known as the Main Zone and occurs along a shear zone that is continuous to the southwest with the shear zone that hosts the DV and CK Zones. The distribution of gold within this shear zone is controlled by the intersection with the contacts of minor ultramafic sills and iron formation units within the basalt.

The lower lens has been referred to as the Fork Footwall Zone (and it occurs within the Russet Lake Shear Zone (Baker, 2017). Here the Russet Lake Shear Zone is wholly within ultramafic volcanic rocks of the Russet Lake Ultramafic and gold mineralization is interpreted to be associated with the intersection of the shear with internal flow contacts. Significantly, the Fork Footwall Zone occurs within the same structural/stratigraphic position as the 8 Zone which occurs about 1.8 km down-plunge to the northeast.

A third resource domain (North-South Domain) has been modeled between the Fork Footwall Zone and Fork Main Zone. It is not clear geologically how this relates to the modeled structures but may be a short second-order splay.

The Fork deposit is cut by late, discordant felsic, intermediate, and mafic dikes as in the mine. The mineralized body is curvilinear and is weakly folded by steeply southeast plunging F2 folds. Gold is predominantly associated with deformed quartz veins hosted within an envelope of highly strained and hydrothermally altered rock controlled by shear zones that developed oblique to the host volcanic stratigraphy. Less commonly, gold is found in replacement-style disseminations within altered basalt along and proximal to contacts with interflow iron formation or ultramafic sills. Geochemically, altered rocks at the Fork deposit are sodium-depleted as at the Madsen deposit. The Fork deposit has been drill tested over a 600 m strike length and to a vertical extent of 375 m depth. The mineralized zones are typically 1 m to 5 m thick. The deposit is located approximately 350 m from existing underground development in the West Ramp.

The Fork Footwall target is the sparsely-drilled southwestern extension of the Fork Footwall Zone that particularly targets the intersection of the host Russet Lake Shear Zone structure with the Russet Lake ultramafic and overlying Balmer basalt contact. This 300 m-long target has been tested by 11 drill holes and remains an active target.

7.3.6 Starratt Target

Gold mineralization at the Starratt target is of the same style as at the Madsen Mine. Gold occurs in similar strongly altered and deformed basalt (SAFZ) with the typical biotite-amphibole-diopside assemblage with local silicification and potassium feldspar alteration. The structural setting is also equivalent to the Madsen deposit whereby mineralized zones occur in planar bodies that cut at low oblique angles across the same ultramafic sills. As in the mine, plunge control of mineralization at Starratt is controlled by the intersection of ultramafic units and these interpreted early structures but at Starratt the plunge is steeper owing to the general steepening of the stratigraphy as the Balmer rocks become constricted between the Killala-Baird Batholith and the Faulkenham Lake stock to the southwest.

Historically, the Starratt and Madsen Mines were operated by different companies and original records from Starratt are fragmented such that the historical drill hole database for Starratt is sparse. No original drill logs are available and the existing drill database has been built largely from fragmented original section and plan maps showing selected data only. Nonetheless, available historical information, surface mapping and geophysical interpretation aided drill-targeting as step-outs from mined out areas in 2016. Gold intercepts in deeper holes (such as 34.0 g/t Au over 2.3 m true thickness in PG16-198) demonstrate that Starratt is open at depth. The mineralized lenses at Starratt extend for approximately 1,200 m strike length, vertical depth of 550 m, with a thickness of 2 m to 15 m.

Since the Starratt and Madsen mines were separated by a tenure boundary during their operational history, the area between these historical mines is under-explored especially given that the area is underlain by Balmer basalts intruded by ultramafic sills. The alteration and host stratigraphy appear continuous between the two mines and Starratt is interpreted to be part of the same mineral system albeit on a parallel shear.

7.3.7 Gap Target

The Gap zone is an exploration target along the up-plunge projection of the target setting associated with the 8 Zone along both the Russet Lake Shear Zone to the upper (eastern) contact of the Russet Lake Ultramafic towards the Fork deposit footwall domain and up-dip towards the Russet deposit.

Drilling in the portion of the Gap target between the Fork footwall domain and the 8 Zone has shown that gold mineralization is of a similar style to that at the 8 Zone and is characterized by deformed blue-grey quartz veins (VBGQ), biotite-altered peridotite (PRBA) as well as a silicified mafic intrusive (RSMI) that is characteristic of the Russet Lake Shear Zone. Intercepted gold grades were relatively low with individual samples grading up to 9.9 g/t gold, however the presence of gold, even at low levels, on the same structure as both the 8 Zone and the Fork Footwall zone is considered prospective, and the area remains an attractive exploration target.

Likewise, sparse drilling in the panel between the 8 Zone and Russet deposits has consistently intersected gold mineralization hosted in rocks with alteration similar to both of those zones. This portion of the Gap target also remains an attractive exploration target area.

7.3.8 Derlak Target

This target is defined as the area approximately 1 – 2 km northeast along strike of the Austin and South Austin zones of the Madsen deposit in a similar setting; within high altered altered basalt intruded by an altered felsic dyke (QZPY) between the hanging wall of an ultramafic sill and the footwall of the unconformity with the Confederation Assemblage. The target extends from surface to a vertical depth of approximately 2 km.

On surface it is underlain by anomalous gold in soil samples and pervasive biotite alteration with well developed foliation (BSLA) in stripped outcrops. Drill testing of this area has produced no significant gold assays, but did intersect similar alteration as at surface with the additional favourable indicator of the presence of a QZPY unit. This alteration corridor has been sparsely tested over a strike length of approximately 600 m.

The target has also been tested at a vertical depth of approximately 1.5 km, at a point approximately 750 m down plunge of the mine. The presence of anomalous acoustic impedance reflectors in the 2020 2D seismic survey data suggestive of continuity with the mine alteration and proximity to anomalous gold values at the northeast end of 18 and 22 Levels and in drillhole AD12-03 provided support for a target that was tested by drilling of two deep drillholes. Both drillholes intersected intense mine-style alteration including silicification, altered felsic porphyry (QZPY) and anomalous gold (up to 2 g/t Au) at the target depth. Further drilling is planned as funding allows.

The presence of favourable alteration, lithologies and anomalous gold values over such a large area directly along strike of the mine makes the Derlak zone an attractive area for future exploration work.

7.3.9 #1 Vein Target

Prior to discovery and mining of the main Madsen deposit, gold mineralization was discovered and developed as exploratory workings at the Madsen #1 Vein, located within the Confederation Assemblage approximately 1 km south of the No. 2 headframe. Surface work conducted by Pure Gold confirmed the presence of gold mineralization (often associated with visible gold) in quartz veins which follow a NE-SW trend for several hundred metres. Current work also confirms the validity of a series of grab samples taken by Claude Resources which showed consistent gold mineralization along the trace of the surface exposure of the #1 Vein during a program of limited surface mining in the 1990s.

Drilling by Pure Gold at the #1 Vein target intersected quartz veins containing visible gold within an altered felsic intrusive unit to a vertical depth of 135 m and over a strike length of approximately 100 m. Assay values up to 9.4 g/t Au over 2.0 m were intercepted. The majority of this drilling was planned in close proximity to the historical #1 Vein mine workings and the workings were intersected in several drillholes. Drilling by Pure Gold (to the end of 2021) served to establish an understanding of the nature of mineralization at the #1 Vein and provided a starting point for future exploration activities in this area.

7.3.10 Dev Northwest Target

The Dev Northwest Target is an early-stage exploration target defined by anomalous gold in soil samples, alteration in outcrop and gold mineralization in quartz veins. Follow-up prospecting and mapping on a large gold-in-soil anomaly at the end of the 2015 field season identified quartz veining and silicification in iron carbonate and banded amphibole-biotite altered basalt. Anomalous gold values were returned from limited outcrop grab sampling. In 2017 further mapping, trenching and channel sampling work was completed (Leidl, 2018). Strongly altered BSLA and SAFZ shear zones and prominent sheeted arrays of intersecting quartz veins were identified. The area was re-examined in 2019 and it was recommended that drill testing be conducted on both the vein and alteration zones to test for significant density of gold bearing veins on and around the contact with a nearby ultramafic unit (Swanton et al, 2019).

7.3.11 Dev Target

At the Dev Target, a large D_2 fold defined by magnetic anomalies is cut by several axial planar shear zones. Banded iron formation defines at least three stratigraphic marker units which may fold back on themselves to define an F1 fold hinge (Cooley and Leatherman, 2015). MMI (Mobile Metal Ion) soil data define a significant multi-element, gold-associated anomaly covering a 1500 m by 200 m area (Baker and Swanton, 2016). Programs of mechanized stripping in 2016 and 2019 determined that the most prospective portion of this large soil anomaly is the northeastern corner, where several large zones of iron carbonate veining are present on the margins of deformed iron formation. Both iron formation and iron carbonate veins are locally silicified and channel samples from the 2019 program returned up to 6.5 g/t Au. As these veins are interpreted to be the same as those which host the adjacent Redaurum historical mine (Evolution Mining), follow up drill testing of the zones was recommended (Swanton et al., 2019).

7.3.12 Snib Target

Historically, this area was part of the Newman Madsen Property acquired by Pure Gold in 2014. Pure Gold completed MMI soil sampling over the Snib target which returned anomalous gold in soil values at the northern and southern limits of Snib Lake. These areas are underlain by quartz veining at the contact of folded ultramafic units. Additionally, shearing and strong carbonate alteration are present north of the lake (Cooley and Leatherman, 2015). Historical core drill holes testing the unconformity between the Balmer and the Confederation Assemblages returned intercepts of 22.56 g/t Au over 2.0 m from drill hole NM06-02 and 43.51 g/t Au over 0.65 m from drill hole NM-10-02, both within the Confederation Assemblage. There is approximately 1000 m of poorly tested strike length between the two drill intercepts noted above and the area remains a viable exploration target due to the presence of alteration, proximity to an ultramafic unit and the Balmer-Confederation unconformity and the gold occurrences.

7.3.13 Treasure Box Target

The Treasure Box Target near the north end of Russet Lake is characterized by discontinuous en echelon extensional quartz-tourmaline veins and stockwork veins that locally contain visible gold. Vein swarms vary from 10 m to 70 m wide but individual veins are generally <40 cm thick. Gold in the wall rock adjacent to the veins is negligible. The veins are hosted in a package of moderately altered and weakly deformed basalt and gabbro. The most extensively tested portion of the target was drilled by Placer Dome and Claude to delineate a package of mineralized veins over a strike length of 165 m and to a vertical depth of 250 m with a typical thickness of 35 m. Work by Pure Gold during 2019 recognized that this zone forms part of a larger trend of veining extending nearly for 1 km west of the zone of dense drilling, encompassing several additional stripped outcrops and isolated historical drillholes (Swanton et al, 2019). Drilling of the main Treasure Box target by Pure Gold in 2020 successfully intercepted gold mineralization directly outside the footprint of previous drilling (up to 19.2 g/t Au over 1 m), adding significantly to the understanding of mineralization in the area and demonstrating it to be open to depth and along strike. Additional drilling along the western strike extension of the zone intercepted the same veining system seen in outcrop and is a positive indicator of the exploration potential of the trend.

7.3.14 Dome Target

This is an early-stage exploration target located along the southwest margin of the granodioritic Dome Stock comprising gold in quartz-tourmaline veins. Historical drilling intercepted 1.8 g/t Au over a composite length of 24 m, including 22.6 g/t Au over 0.9 m. Historical work indicates near surface potential for mineralization over a significant strike length. This mineralization is similar in style to that at the adjacent and past-producing Buffalo and Red Lake Goldshore mines on the Hasaga property held by Equinox Gold, as well as Yamana Gold’s North Madsen project.

7.3.15 Roberts Target

The Roberts target is located along a curvilinear north-striking, east-dipping iron formation on the west side of Robert's Lake in the northwestern portion of the mine property. Gold occurs in deformed quartz veins and in sulphidized wall rock adjacent to veins hosted within basalt and sulphide-facies interflow iron formations (Jones 2016). The iron formation beds have been traced for hundreds of metres to the north of the Roberts trenches and for over 1 km to the stripped outcrops at the Roberts South showing. Channel samples from exposed trenches returned up to 10 g/t Au over 2.0 m and isolated grab samples returned up to 59 g/t Au. Pure Gold drilled fifteen holes over a strike length of 125 m, testing directly beneath surface mineralization with the best intercepts grading 3.7 g/t Au over 5.0 m (including 9.8 g/t over 1.0 m) and 2.8 g/t over 3 m (including 7 g/t over 1 m). No testing below 60 m vertical depth has been completed. As drill testing has been confined to a relatively small section of the overall Roberts trend and has been successful in demonstrating the presence of gold mineralization, there is potential for additional exploration success along the trend and at depth.

7.3.16 Russet North Target

The Russet North target has a similar geological setting to the Russet Deposit. This target comprises 100 m of north-south striking iron formation where gold appears to be localized along a folded iron formation-basalt contact. Surface sampling (rocks and soils) as well as drilling has been completed. Drilling intercepted up to 5.5 g/t Au over 1.0 m.

7.3.17 Coin Target

The Coin target is situated at the far northern extent of the mine property, within a region of tight folding defined by east-west axis folds and ultramafic sills cutting Balmer basalt. Anomalous geochemical results from soil and rock sampling in the area is suggestive of mineralization in proximity to the hinges of these folds. Recent drill intercepts and historical drill results from nearby holes on the Evolution mining property to the immediate West make this area an attractive exploration target.

8 Deposit Types

Gold mineralization on the Madsen property is localized along major structural trends, similar to other deposits found throughout the Red Lake district and ranging in scale from localized showings to past-producing or currently producing mines. Based on extensive work, Pure Gold made advances in the geological understanding of these deposits and developed a new interpretation that deviates from those of past workers (e.g., Dubé et al., 2000). As of the effective date of this report, the deposits are now classified as Archean orogenic gold deposits (Groves et al., 1998), though they have been intensely modified by deformation and metamorphism following gold deposition.

8.1 Characteristics

Following Kerrich et al. (2000), orogenic gold deposits are typically associated with crustal-scale fault structures, although the most abundant gold mineralization is typically hosted by lower-order splays from these major structures. Deposition of gold is generally syn-kinematic, syn- to post-peak metamorphism and is largely restricted to the brittle-ductile transition zone. However, deposition over a much broader range of pressure-temperature conditions (200–650^{o}C; 1–5 kbar) has been demonstrated. Host rocks are highly variable, but typically include mafic and ultramafic volcanic rocks, banded iron formation, sedimentary rocks and more rarely granitoid rocks. Alteration mineral assemblages are dominated by quartz, carbonate, mica, albite, chlorite, pyrite, scheelite and tourmaline, although there is much inter-deposit variation.

8.2 Mineralization Model

8.2.1 Madsen Mine Style Gold Mineralization

Controls on mineralization at the Madsen Mine are consistent with a typical orogenic gold system. Many deposit-scale features such as control by lithological / structural contacts and association with felsic dykes are typical in these systems. Smaller-scale features have been used to support the interpretation that the Madsen Mine deposits are an unusual or end-member type of orogenic gold system. For example, Dubé et al. (2000) conclude that the Madsen deposit is a disseminated, stratabound deposit that shares similarities with mafic-hosted gold-skarns and also with higher-temperature Australian deposits. Recent work, however, indicates that, apart from its early timing of emplacement prior to the dominant regional deformation and metamorphism, Madsen Mine shares many characteristics with typical orogenic gold deposits, including the Red Lake Mine deposit.

All significant gold mineralization on the mine property is demonstrably early relative to the most significant, penetrative deformation (D_2) and metamorphic events. Quartz veins at 8 Zone, Wedge and Russet are boudinaged, recrystallized, folded and overprinted by the penetrative S_2 foliation. Mineralized bodies of the Austin, South Austin and McVeigh Zones are locally folded and transposed into S_2. In addition to this intense deformation overprint, the mineralized veins and alteration have been subject to the relatively high temperatures of amphibolite facies metamorphism, which led to extensive recrystallization and growth of the skarn-like mineral assemblage of diopside-amphibole-quartz-biotite. By contrast, more typical deposits of the orogenic gold deposit class are characterized by lower temperature greenschist facies mineral assemblages. It is this intense deformation combined with the amphibolite grade metamorphism that obscured the primary features of the Madsen Mine style gold mineralization and led to suppositions of a syngenetic, or other atypical origins for the deposit. However, numerous structural and petrological features identified by Pure Gold contradict such interpretations and support an orogenic gold deposit classification.

At the property-scale, the mineralized zones follow planar corridors defined by patterns of gold mineralization, alteration and high strain that are continuous on a kilometre-scale (e.g., Austin/ South Austin Zones are planar and continuous over an area of at least 600 by 2,000 m). These planar structures transect their host Balmer stratigraphy and the Balmer / Confederation Assemblage unconformity at a low angle, which refutes the possibility of a syngenetic or stratabound origin. This corridor is also host to discontinuous, coplanar lenses of quartz porphyritic felsic dykes, which predate mineralization but preferentially host significant rinds of mineralization along their contacts – a further indication of an epigenetic origin and structural control for the mineralization.

At the deposit scale, primary local control controls on the localization of gold mineralization include competence contrasts between different lithologies, in particular felsic dykes and ultramafic sills, but also iron formations, which is a common feature of structurally-controlled hydrothermal deposits of all types. These competence contrasts manifest deposit scale controls on the shape and plunge of the deposits at their intersections with mineralizing structures. Reversals in the plunge direction of mineralized shoots are thereby linked to the primary orientations of vein emplacement relative to their host stratigraphy - originally moderately-dipping structures host shoots that tend to plunge east (e.g., Austin and South Austin), whereas originally steep-dipping and more northerly-trending structures tend to plunge southwest (e.g. McVeigh and Wedge). Also, the presence of shallow-dipping veins within the original vein structure, as evidenced by the shallower dip of enveloping surfaces around tightly-folded veins, in conjunction with moderately- and/or steeply-dipping structures, is another feature common in structurally-controlled gold vein systems.

Petrologically, the skarn-like alteration assemblage of the mine style mineralization is an unusual feature in gold vein systems but is now recognized here as a metasomatic overprint on the original quartz-carbonate (+sericite?) vein and alteration mineral assemblage. Numerous examples of preserved quartz-carbonate veins and vein remnants with biotite-rich reaction rims inside massive diopside-amphibole-altered rock have been found, clearly indicating the primary alteration assemblage to be quartz-carbonate rich, as is more typical of orogenic gold deposits. The skarn minerals (diopside, amphibole, biotite and garnet) therefore represent the recrystallized equivalents of the original alteration assemblage, grown at higher temperatures following emplacement.

Thus, the main components of the Madsen deposit mineralization model include:

- Significant gold deposition occurred prior to the main, belt-scale deformation event (D_2) within largely planar structures that have been nearly completely recrystallized by overprinting deformation and metamorphism. An original, pre-D_2 shape of the gold-bearing structural/vein system included a dominant moderately-dipping structure (Austin and South Austin) with subsidiary steeply-dipping footwall splays (McVeigh West and Central), as well as relatively minor shallow dipping veins.

- Geometrically, gold deposits were folded by small-scale, localized folds and were structurally dismembered by transposition and rotation into the penetrative S_2 foliation
- Pervasive, but incomplete replacement of the original quartz-carbonate vein/alteration mineral assemblage to a metasomatic skarn-like assemblage of diopside-amphibole-biotite-garnet (i.e., both assemblages are intimately associated with gold)
- Effective exploration drill targeting requires anticipation of these shapes and expectation of a heterogeneous gold system

8.2.2 Planar, Quartz-Sulphide Vein-hosted Gold

The second style of gold mineralization found on the mine property is hosted by discrete, planar quartz veins with little associated alteration. Setting and orientation of these veins is variable – multiple examples occur proximal to iron formation, but they are also present in both Balmer-age mafic and Confederation-age felsic host rocks. The orientation of these veins is often – though not universally – roughly parallel to the axial plane of D_2 folds. In cases where the veins do not display this orientation, they typically track sub-parallel to a stratigraphic feature such as an iron formation. The key features which distinguish these veins from the earlier veins described above are their lack of associated alteration and co-incident quartz porphyry or non-quartz vein material (e.g., ankerite/diopside). The timing of this vein style is not well constrained, but if the #1 Vein is included in the set, a post-Confederation age is required. The localized association with D_2 axial planes suggests that they were emplaced synchronous with the D_2 deformation event. Examples of this style include the Dev Northwest, Roberts and #1 Vein exploration targets.

8.2.3 Quartz-Tourmaline Vein-hosted Gold

The third style of gold mineralization is associated with quartz-tourmaline veins. These veins show evidence for emplacement under a brittle tectonic setting and typically occur as en echelon vein arrays. As such, these veins have short strike lengths and vein orientations are somewhat chaotic, displaying only a rough alignment of orientations within swarms along large (km-scale) trends.

Despite being generally narrow (cm to 10’s of cm scale) the veins can host high gold grades making swarms of sufficient density an attractive exploration target. Where they are present in the same location as pre D_2 ankerite-quartz veins (e.g., Redaurum deposit), they show a clear cross-cutting relationship with those earlier veins. They do not show evidence of deformation during D_2, suggesting that they post-date that tectonic event. These veins are part of the same system that forms the historical Buffalo deposit, which has been interpreted to be associated with emplacement of the Dome Stock. The primary example of this style on the mine property is the Treasure Box showing.

8.3 Concepts Underpinning Exploration at the Madsen Mine

Exploration for gold on the mine property focuses on identifying the planar structures (or shear zones) which were active during gold deposition. Since gold is very heterogeneously distributed within these structures assessing targets using gold assay data alone will not yield reliable results. Several features in addition to the presence of elevated gold have been identified to demarcate these important structures, including assemblages of alteration mineral phases (i.e., SAFZ and PRBA lithological codes), strong deformation and the presence of the distinctive early VBGQ veins which are characteristically deformed. Importantly, these structures are locally very subtle but careful drill core logging – including relogging of all available historical drill core – has allowed a property scale 3D model of these structures to be built. These structures form the first-order target for exploration drilling and stepping out from known high-grade gold results within these structures forms the second-order targeting criteria. Structural complexities such as jogs or interactions with lithological contacts are also targeted.

9 Exploration

Since acquiring the Madsen Property in 2023, WRLG has conducted geological mapping, surface rock sampling and glacial till geochemical sampling, all of which were undertaken as part of the 2024 exploration program.

9.1 Rock Sampling and Mapping

A geological mapping and grab sampling program was conducted in May 2024, with a focus primarily on the southeastern portion of the property underlain by the Confederation Assemblage. This portion of the property had received limited mapping or prospecting work by previous operators, and the purpose of the current program was to define an effective geological framework for future exploration targeting efforts. The mapping program was conducted by a team of two people over the course of 15 field days. Spatial data (primarily outcrops) were collected by the field team directly into GPS-enabled GIS-enabled field mapping devices, and the results of the mapping were used to update the existing geological map of the Madsen Property to include additional detail in the Confederation Assemblage (Figure 9-1). During the mapping program, the field team collected a total of 69 grab samples all of which were submitted for gold, four-acid multi-element and whole-rock fusion analysis (Table 9-1). In addition to the samples collected in the field, four OREAS 223 CRMs were inserted into the sample stream.

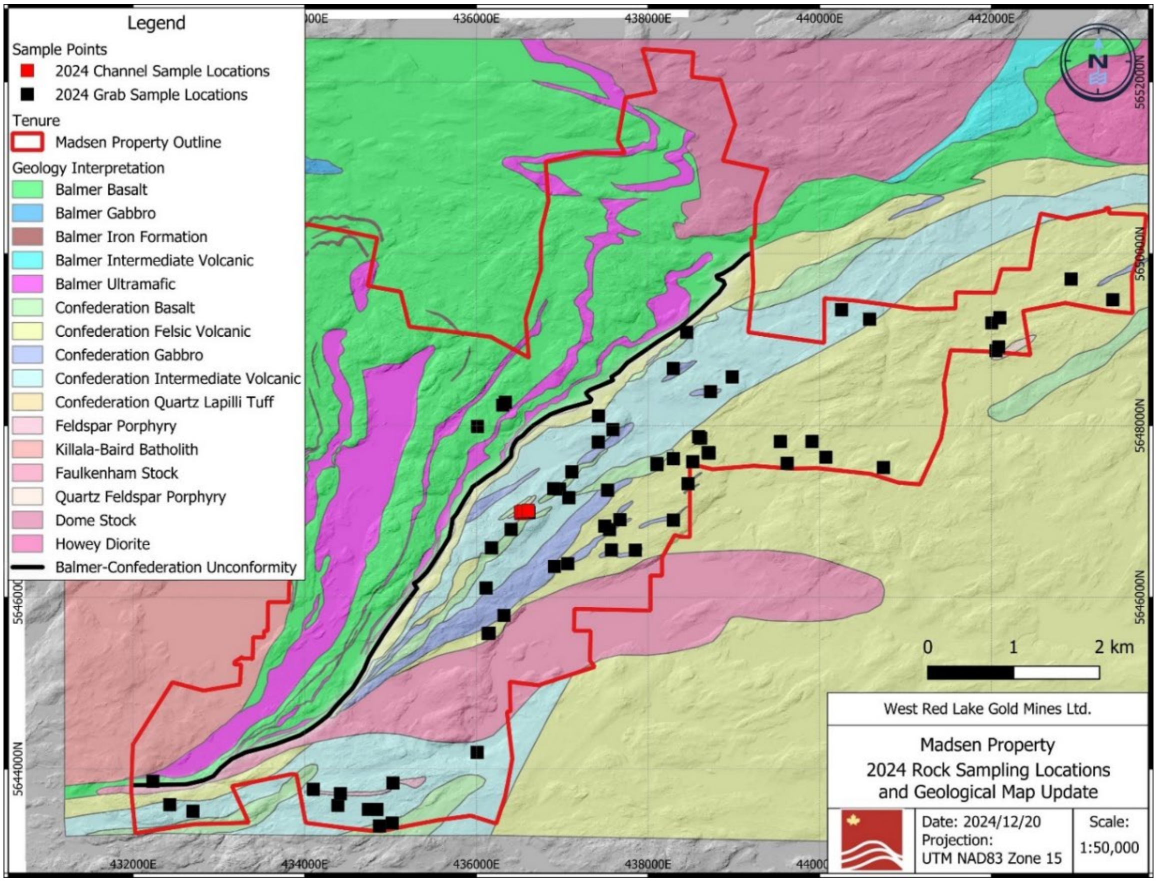
The mapping program identified an area of prospective veining and alteration within the Confederation Assemblage approximately 500 m east of the Madsen Mine East Portal, and a small follow-up channel sampling program was conducted in September 2024 to more effectively sample these veins than was possible during the mapping program. A total of 42 channel samples were collected and submitted (along with two CRMs, one blank and one duplicate) for gold and four-acid ICP analysis.

Table 9-1: 2024 surface rock sampling analytical methods

Analytical Lab	Procedure Type	Method Code	Method Description	Usage
ALS Minerals	Preparation	PREP-31A	Crush to 70% less than 2mm, riffle split off 250g, pulverize split to better than 85% passing 75 microns	Grab Samples
ALS Minerals	Gold Assay	Au-AA23	Fire Assay with AAS finish. 30 g sample	Grab Samples
ALS Minerals	Multi-Element Analysis	ME-MS61	Four acid digestion with ICP-MS finish	Grab and Channel Samples
ALS Minerals	Whole-Rock Analysis	ME_XRF26	Lithium Borate fusion with XRF finish	Grab Samples
SGS Minerals	Preparation	PRP89	Crush to 70% less than 2mm, riffle split off 250g, pulverize split to better than 85% passing 75 microns	Channel Samples
SGS Minerals	Gold Assay	GO_FAA50V10	Fire Assay with AAS finish. 50 g sample	Channel Samples

Source: WRLG (2025)

Figure 9-1: 2024 geological mapping update, grab and channel sample locations



Source: WRLG (2025)

9.2 Surficial Geochemical (Till) Sampling

Two programs of surficial material sampling were conducted on the Madsen Property during the 2024 exploration campaign. The first program was focussed on the Confederation Assemblage and one under-explored portion of the Balmer Assemblage in close proximity to the Blamer-Confederation unconformity. The sampling grid was designed with a nominal sample spacing of 50 m x 300 m (50 m sample spacing along lines spaced 300 m apart). The second program was designed to follow up on anomalies detected during the first program at closer spacing, with sample set out at 50 m spacing along 100 m spaced sampling lines. The second program also entailed expansion of the grid at the original 50 m x 300 m spacing to areas of the Confederation not covered by the first program (Figure 9-2, Table 9-2).

Sampling was carried out using a “Dutch” style handheld auger, with an extension capacity to a maximum sampling depth of 2 m. Sampling was constrained to material of known glacial origin i.e. glacial and fluvio-glacial deposits, with a strong preference given to sampling the un-weathered C horizon. B horizon soils, though less favourable in till sampling were permissible if soil profile development was poor. Sampling was conducted along pre-defined lines oriented generally perpendicular to ice flow direction with pre-defined sampling station points. No sample was taken at a sample station if the preferred sample media was not present within the sampling depth capacity of the hand-held auger (i.e., in deep peat horizons) or if no glacial material was deposited over shallow outcrop. Proposed sites were allowed to be moved within a maximum variance of 15% of the inter-sample distance if the proposed site was deemed unfavourable for sample collection.

A sample weight of approximately 2 kg was taken at each sample site. Depth of samples varied, but typically C horizon till material was collected from ≥ 80 cm in depth. All samples were collected in Hubco Sentry, air dryable sample bags, allowing for reduction in transport weight after partial drying in the field. Sample bags were labelled and tagged with pre-printed sample ID tags provided by ALS Minerals. Navigation to proposed sample locations was carried out using handheld GPS devices. Digital data collection was completed using QField software, along with a purpose built till sampling data collection GeoPackage, contained on a ruggedized ULE phone device. Data collected included: sample media, depth, clast size, interpreted till facies, and environmental data. Daily data uploads from the QField system to a master QGIS workspace were completed to ensure data validation and accuracy prior to a subsequent field day.

All samples were submitted to ALS Minerals in Thunder Bay, Ontario for screening to -63 µm followed by an aqua regia digestion and an ultra-trace multi-element analytical suite, with method codes as detailed in Table 9-3. The only CRM used in the 2024 surficial sampling program was OREAS 46, a multi-element CRM sourced from glacial till in Quebec Canada and suitable for use in trace level analysis with a certified gold value of 1.6 ppb.

Table 9-2: 2024 surficial sampling program details

	Phase 1	**Phase 2**	***Total***
Program Start	May 13, 2024	September 10, 2024	
Program Completion	May 30, 2024	September 27, 2024	
Person-Days	88	61	*149*
Sites Visited	945	903	*1848*
Sites Sampled	706	676	*1382*
Duplicate Samples	40	38	*78*
CRMs	39	38	*77*

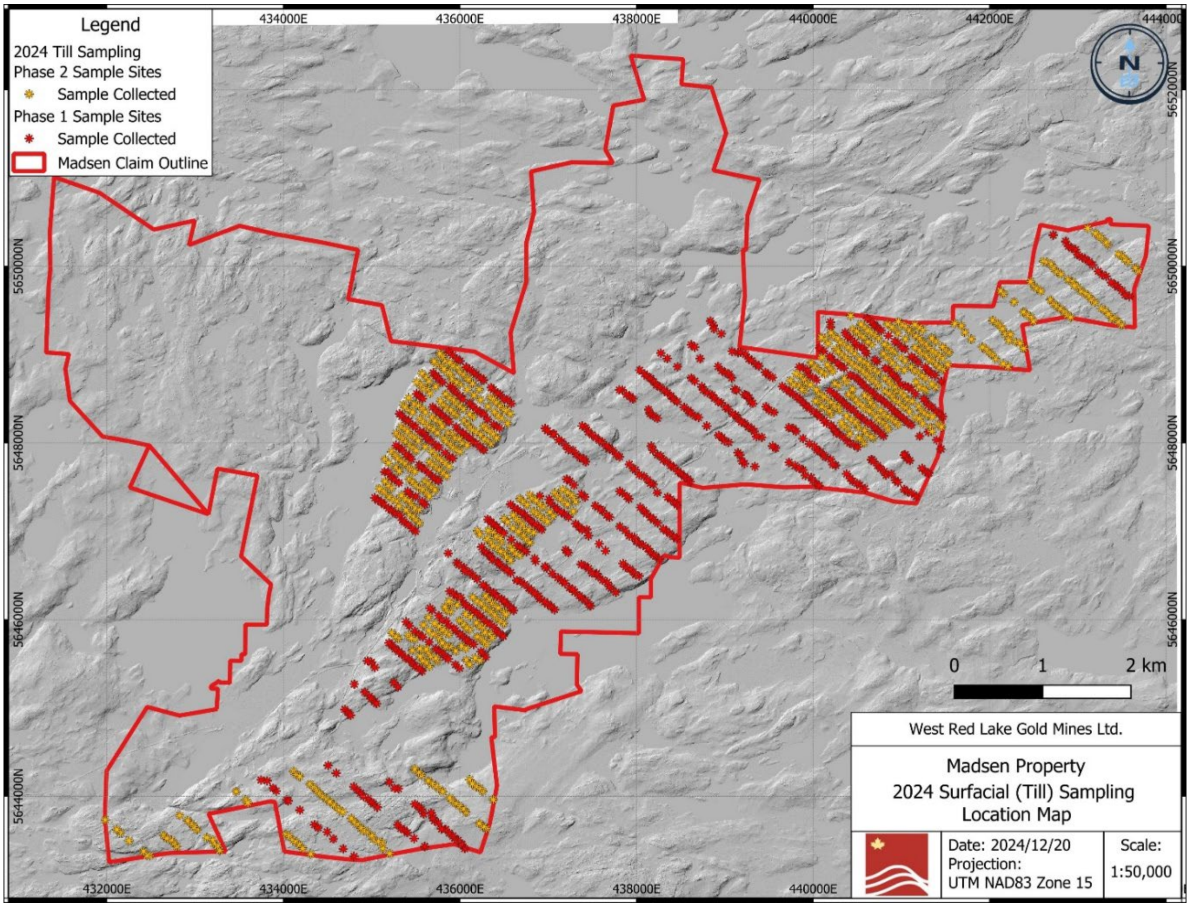
Source: WRLG (2024)

Table 9-3: Surficial sample analytical procedures

Analytical Lab	**Procedure Type**	**Method Code**	**Method Description**
ALS Minerals	Preparation	DRY-22	Dry samples at 60° C
ALS Minerals	Preparation	SCR-51 (63UM)	Screen to -63µm mesh size
ALS Minerals	Multi-Element Analysis	AuME-ST44	Aqua Regia and cyanide leach extraction with ICP-MS analysis. 50 g sample size

Source: WRLG (2024)

Figure 9-2: 2024 Till Sample Locations



Source: WRLG (2024)

10 Drilling

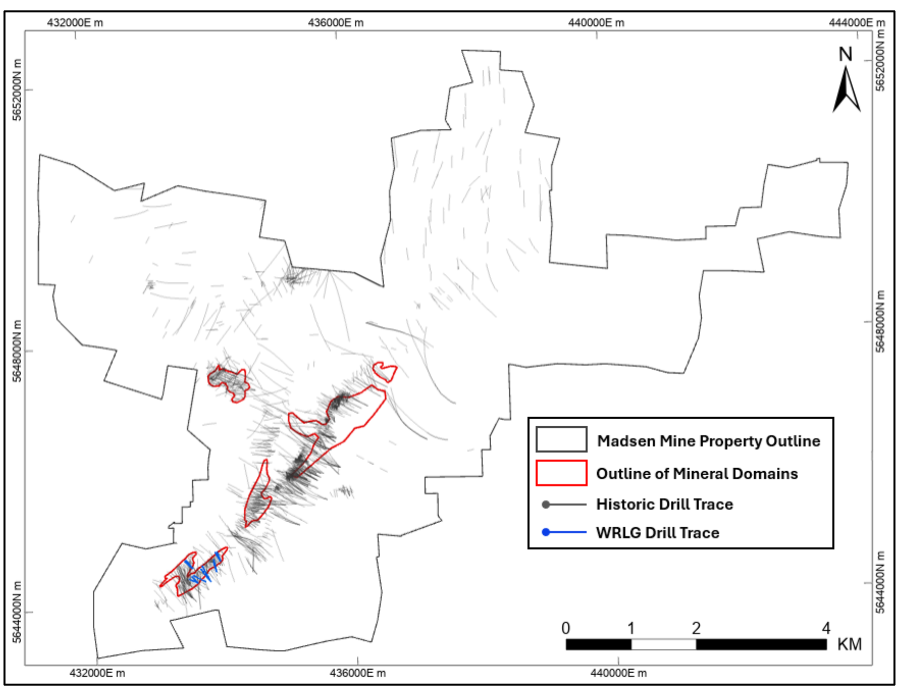
The Property has a long history of diamond drilling, dating from initial discovery of the Madsen deposit in the 1930s through until the present day. Figure 10-1 provides a plan map and Figure 10-2 provides a long section view showing drill holes completed by WRLG and previous operators on the Property as a whole and in the immediate vicinity of the Madsen deposit, respectively. Documentation of procedures and methods of drilling prior to the 1990s is sparse.

10.1 Historical Drilling

All historical exploration and production drill testing on the mine property to date has been by diamond drill coring. Underground drilling from 1937 to 1999 at Madsen Mine employed whole core sampling and most core intervals were sampled for fire assay gold analysis at the on-site mine laboratory. Core was generally sampled at 5-foot intervals. The near complete drill log collection preserved at the mine site office attests to a very systematic approach to surface and underground drilling by Madsen Red Lake Gold Mines during the era of mining at Madsen. Early collar locations are referenced to the Imperial Mine Grid and holes are systematically identified according to the level where it was collared and the grid easting. Use of this grid system persisted by other operators until about 2000 at which time recreational-grade GPS equipment was used to collect collar coordinates for surface drillholes and from 2009 onward a differential GPS system was used.

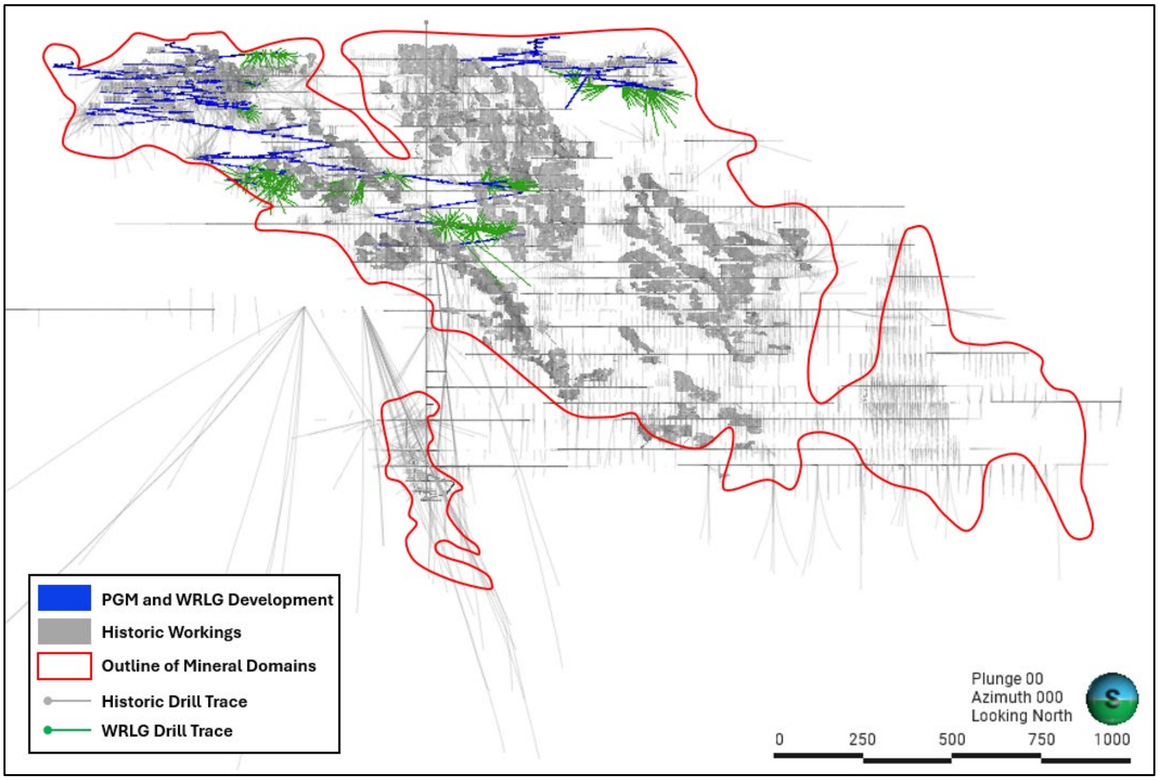
According to archived drill logs, Madsen Gold Corp's drilling campaigns in 1998 recovered BQ-sized core and drilling was completed by Newmac Industries. Downhole survey data were collected by acid dip tests or Sperry Sun.

Figure 10-1: Plan map showing the location and direction (traces) of surface drill holes completed by WRLG and previous operators



Source: WRLG (2025)

Figure 10-2: Longitudinal section view (looking 300°) of Madsen Mine showing location of underground diamond drilling, development, and outlines of mineralized domains



Source: WRLG (2025)

10.2 Placer Dome

Placer Dome completed drilling programs between 2001 and 2004 (Dobrotin 2002, 2003, 2004a, 2004b, Dobrotin and McKenzie, 2003) across the Madsen property. All drilling returned NQ-sized core and was completed by Major Dominik Drilling with a Major 50 or Boyles 37 rig. Downhole surveys were completed using a combination of acid dip tests, Maxibor and EZShot methods. Drill collar locations were recorded by handheld GPS equipment in NAD27 for early programs, but a differential GPS was used in 2004.

10.3 Wolfden and Sabina

Drilling programs on the Newman-Madsen Property between 2003 and 2011 (Klatt, 2003a, 2003b; Toole, 2005; Long, 2007) returned NQ-sized core and were completed by Chibougamau Diamond Drilling Ltd. Generally, downhole surveys were completed using Flexit™ single shot tools and holes were lined up with a handheld Brunton compass.

10.4 Claude

According to information on drill logs and internal written procedures, Claude Resources' drilling programs returned NQ-sized core and were completed by Bradley Brothers Limited. Collar location coordinates were collected with a differential Leica GS50™ DGPS in NAD27. Downhole surveys were completed with Flexit™ and EZShot™ tools and later with DeviFlex™ gyroscopic tools.

10.5 Pure Gold

Pure Gold drilled a total of 2,411 diamond drill holes for 399,661 m between 2014 and 2022. These totals include both exploration drilling outside the footprint of the Madsen Mine and definition drilling to support mining operations. Core size and drilling locations are summarized in Table 10-1.

Drilling within the Madsen deposit was aimed at characterizing the historically mined zones using modern methodologies and on extending the strike and dip extents of known mineralization. Targeted exploration drilling occurred within and adjacent to all of the current resource domains with a focus on resource growth. Additionally, several target areas across the property were tested including Starratt, Fork, Russet and Wedge and initial drill holes tested several regional targets.

Drill holes completed within the resource domains confirmed data contained in the historical mine compilation and allowed a thorough study of the structural geology, geochemistry and alteration of the gold-bearing zones. Information acquired through the latest drilling and supported by surface work was consistent with interpretations that the gold mineralization at Madsen Mine developed early in the tectonic history of the belt and has been deformed and folded.

Surface drilling was completed by Major Drilling in 2014 and early 2015 and by Hy-Tech drilling from early 2015 to 2022, with the exception of short PQ-sized holes drilled in 2018 as part of the geotechnical program for the tailings facility, which were drilled by Boart Longyear. Hy-Tech drilling conducted underground drilling in 2017, 2018, 2020 and part of 2021. For the latter part of 2021, underground diamond drilling was conducted by Boart Longyear. Additionally, in-house Pure Gold staff conducted a portion of the diamond drilling during 2020 and early 2021.

Most surface holes (except the 14 geotechnical holes) were drilled with NQ-sized equipment and core was placed in wooden core boxes. Drill collar casings were preserved and covered with caps labelled with the drill hole name and marked with wooden stakes. Hole collar locations were surveyed post-completion by Pure Gold staff or contractors. From 2014 through 2021 the holes were surveyed using a Trimble ProXRT™ differential GPS receiver with Omnistar™ real-time correction; from late 2021 until 2022 a Trimble R2™ differential GPS receiver was used. Both tools are able to achieve sub-metre precision. Underground holes were drilled with a variety of core sizes and drill rigs. All EW core and a minor amount of AQTK core was drilled with an air powered Bazooka rig, with the majority of AQTK diameter core drilled with an air-powered VAG drill rig. All BQ and NQ diameter core was drilled with electric hydraulic rigs. For all types of underground drilling, core was placed in wooden core boxes at the rig and delivered to surface for logging and sampling. Hole collar locations were marked by the drill crews with numbered metal tags to identify each hole, and then surveyed post-drilling by Pure Gold mine survey staff with the same survey equipment and controls used in all other aspects of mine surveying.

Table 10-1: Pure Gold diamond drilling totals showing core size, year, and drill location

	Year drilled:	2014	2015	2016	2017	2018	2019	2020	2021	2022	Total to Dec 31, 2022
Core Size		**Surface**									
NQ	**# of Holes**	26	27	235	202	143	101	86	151	187	**1,158**
	Total Metres	6,895	5,909	82,525	81,118	33,884	27,246	26,943	25,491	15,862	**305,873**
HQ	**# of Holes**						1				**1**
	Total Metres						135				**135**
PQ	**# of Holes**					14					**14**
	Total Metres					262					**262**
Core Size		**Underground**									
EW	**# of Holes**					153		56	24		**233**
	Total Metres					1,976		1,723	553		**4,251**
AQTK	**# of Holes**							3	164		**167**
	Total Metres							68	11,598		**11,666**
BQ	**# of Holes**								172	501	**673**
	Total Metres								14,018	38,260	**52,278**
NQ	**# of Holes**				58	53		54			**165**
	Total Metres				12,180	5,842		5,174			**23,196**
TOTAL	**# of Holes**	26	27	235	260	363	102	199	511	688	**2,411**
	Total Metres	**6,895**	**5,909**	**82,525**	**93,298**	**41,965**	**27,381**	**33,907**	**51,660**	**54,122**	**397,662**

Source: WRLG (2024)

Down-hole surveys in 2014 and 2015 were initially completed with a Reflex EZ-Shot tool every 20 to 30 m. Drill holes were re-surveyed at completion with a Reflex Gyro survey tool from hole bottom to top. Starting azimuths for the gyroscopic instruments and drill alignments were determined with an azimuth pointing system (APS) GPS based compass in 2014 and 2015. In 2016, survey procedures were improved through the replacement of the APS with gyrocompass devices (initially the Reflex TN-14™, which was subsequently replaced with a DeviCo Devialigner™) for drill alignments and initial gyro orientations and with north seeking downhole gyro tools for all downhole surveying. Survey procedures for surface exploration remained constant from 2014 through 2022, with north seeking single shot surveys taken every 30 m and continuous surveys with initial alignment taken by north seeking gyrocompass done upon completion of drilling. Definition drilling followed the same procedures until late 2021, at which point the procedure was changed such that only a single continuous survey with the downhole gyroscopic surveying tool was completed.

All surface exploration drillcore drilled by Pure Gold between 2014 and 2022 was oriented using a Reflex ACTIII™ core orientation tool. Definition drillcore (either surface or underground) was not oriented.

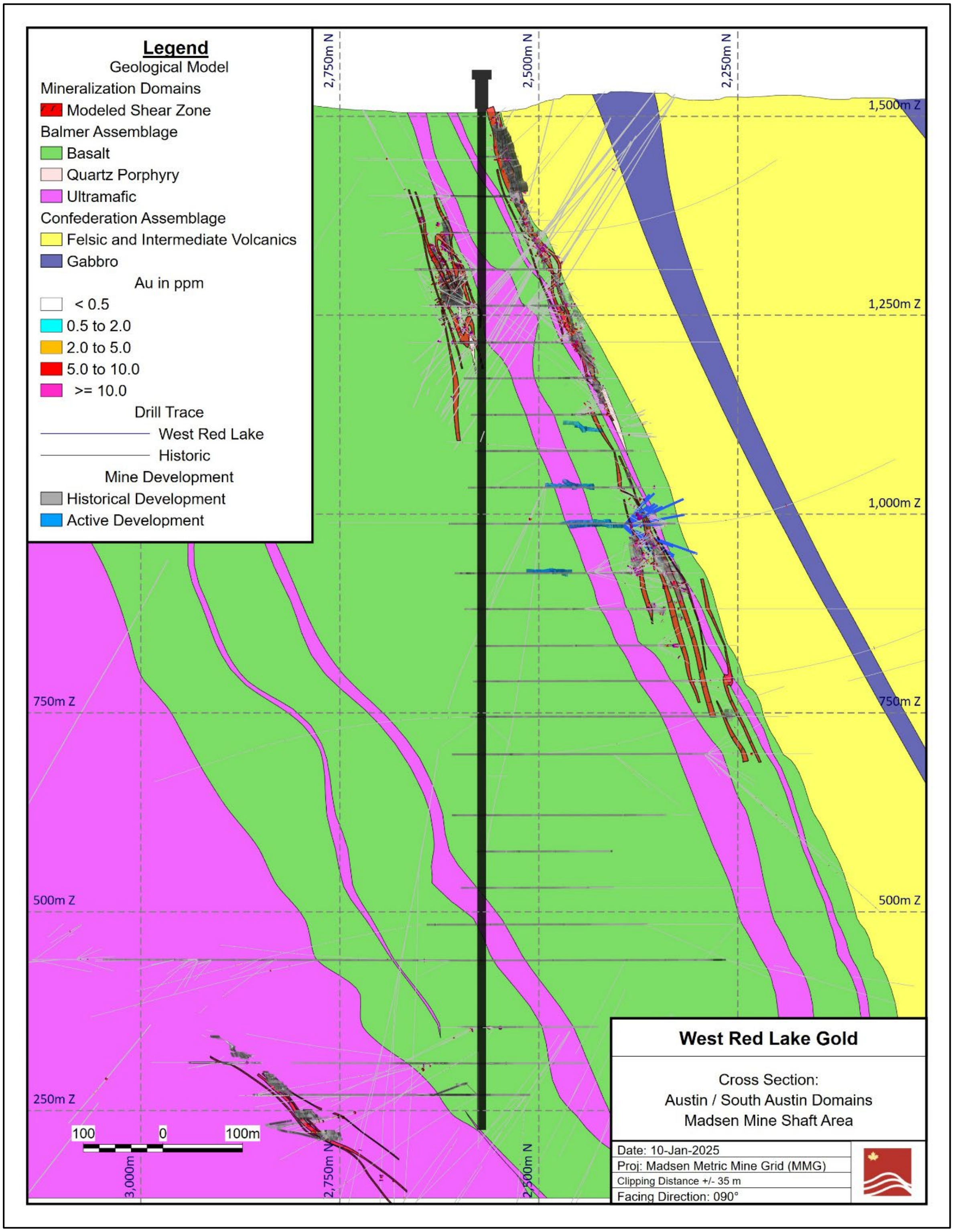
All surface drilling sites were cleared of any cut timber and debris, and sites outside the active mine site were re-contoured and re-seeded with a native seed mix post-drilling.

All drill holes were logged, photographed, and sampled at the mine following procedures very similar to those currently employed by WRLG and fully described in Revering et al. (2024). All data collected during core processing were stored in the Reflex Hub™ (formerly ioHub) cloud database until early 2018 when the database was moved to a Datashed™ SQL database managed by Pure Gold.

10.6 West Red Lake Gold

Since acquiring the Madsen project in June 2023 and up to May 15, 2024, WRLG completed a total of 146 holes for 11,849 m of BQ diamond drill core (definition) and 59 holes for 8,024 m of NQ diamond drill core (expansion) from underground (Figure 10-3 to Figure 10-5). WRLG has continued definition drilling through the rest of 2024, work that is not considered in this report. Definition drilling was focused on the Austin and South Austin zones to increase geologic confidence in these areas to a level appropriate for mine development planning. Expansion drilling was focused primarily within the newly defined North Austin zone outside of the existing life-of-mine mineral resource domains, but still in close proximity to existing underground infrastructure. Underground drilling in 2023 and 2024 was completed by Boart Longyear.

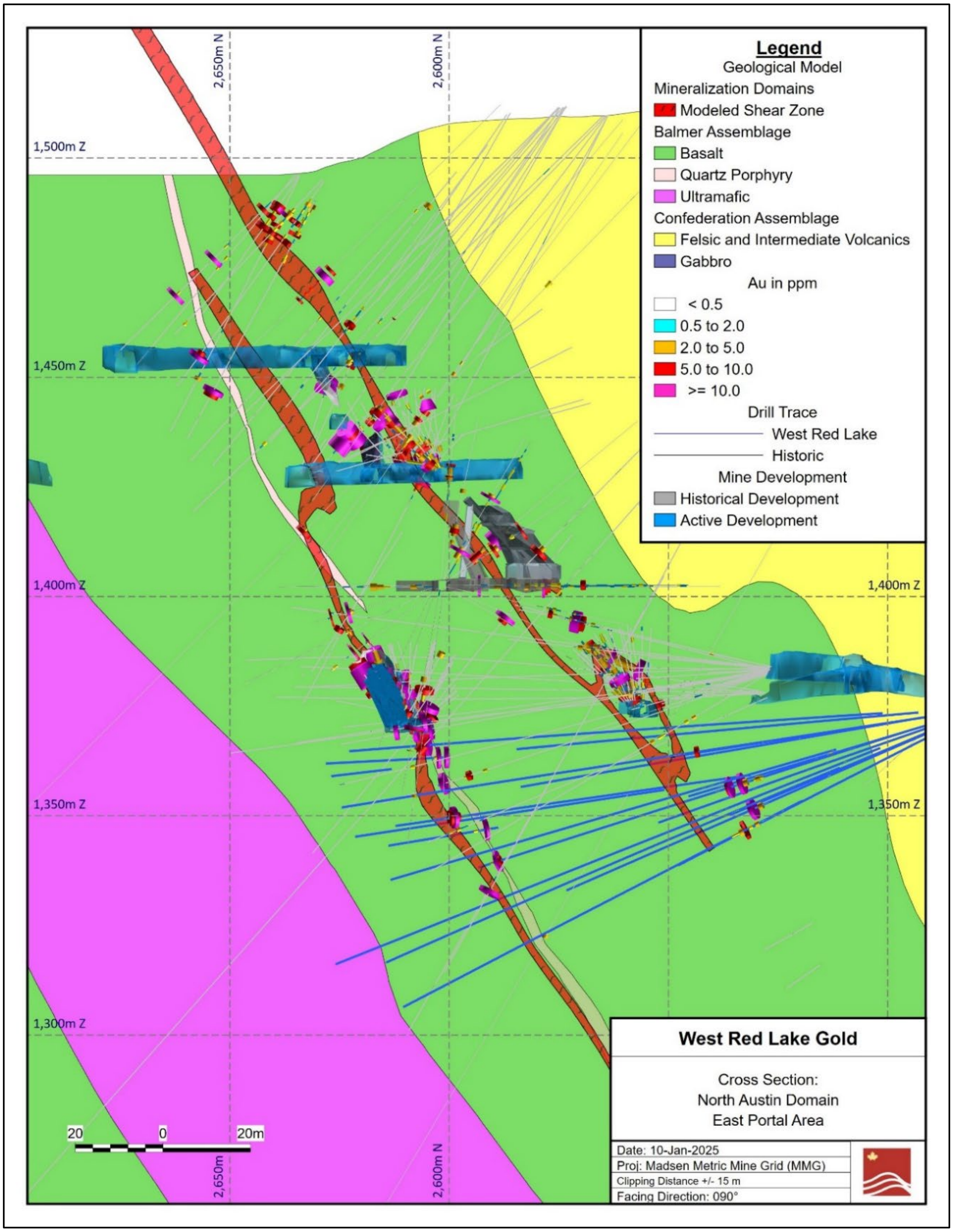
Figure 10-3: Representative cross-section through the Austin-South Austin / Shaft area of Madsen Mine



Source: WRLG (2025)

Notes: Coordinates displayed in local Madsen Mine Grid (MMG). 30 m thick cross-section looking 030° (090° MMG).

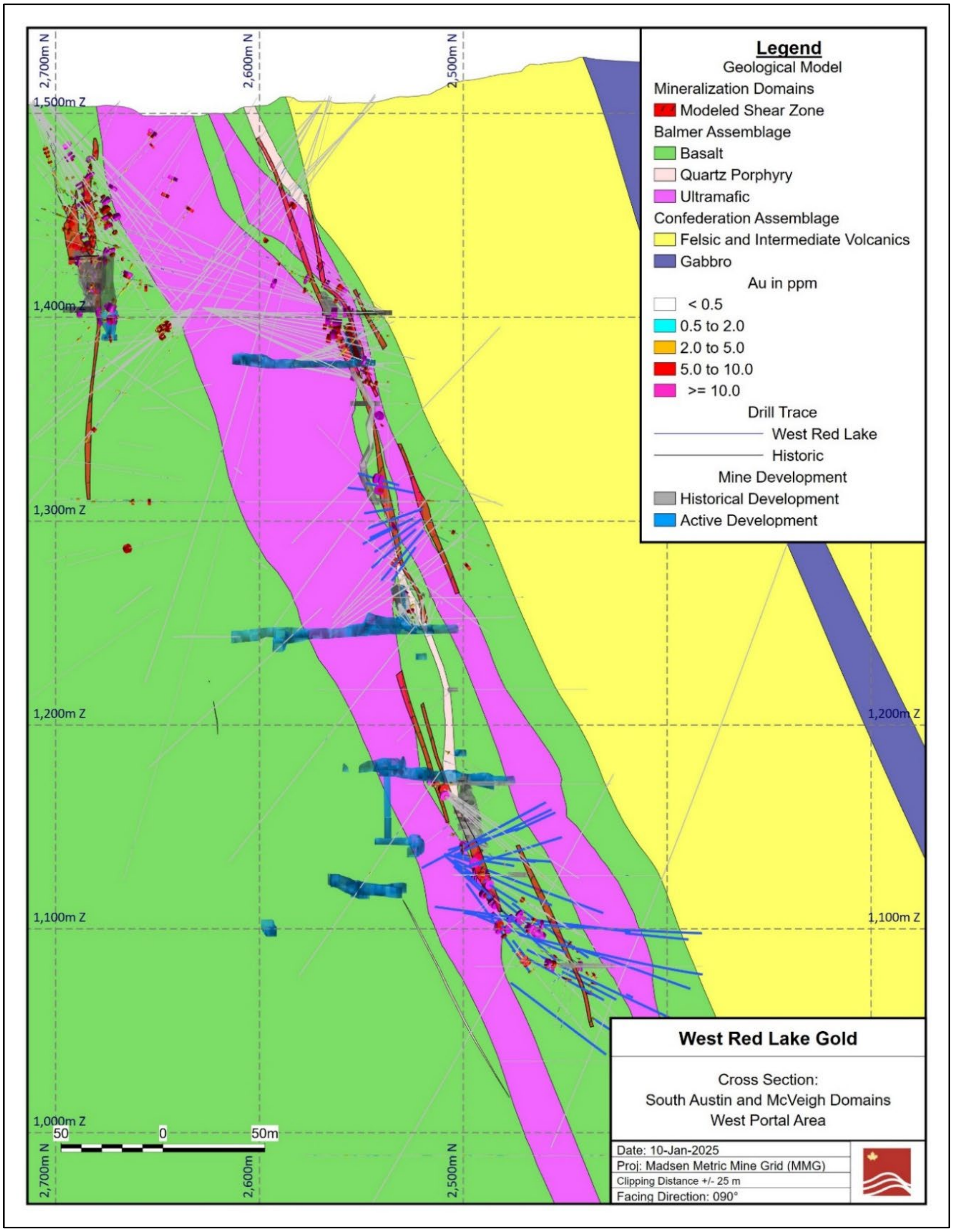
Figure 10-4: Representative cross-section through the East Portal area of Madsen Mine showing mineralized domains within the North Austin Zone.



Source: WRLG (2025)

Notes: Coordinates displayed in local Madsen Mine Grid (MMG). 30 m thick cross section looking 030° (090° MMG).

Figure 10-5: Representative cross-section through the West Portal area of the Madsen Mine showing mineralized domains within the McVeigh and South Austin Zones.



Source: WRLG (2025)

Notes: Coordinates displayed in local Madsen Mine Grid (MMG). 30 m thick cross section looking 030° (090° MMG).

WRLG also completed a surface diamond drilling program at the Wedge target in September 2023, approximately 2 km southwest of the Madsen Mine. Drilling was focused on extending high-grade zones at Wedge and increasing confidence in the existing mineral resource. WRLG drilled a total of 11 holes for 2,996 m of NQ diamond drill core at Wedge. The surface program was successful in extending high-grade shoots within the existing mineral resource. Surface drilling in 2023 was completed by PL Drilling. All surface holes were drilled with NQ-sized equipment and core was placed in wooden core boxes. Drill collar casings were preserved and covered with caps labelled with the drill hole name and marked with wooden stakes. Hole collar locations were surveyed post-completion by WRLG staff or contractors. The holes were surveyed using a Trimble R2TM differential GPS receiver, which is able to achieve sub-metre precision. Underground expansion holes were drilled with NQ-diameter core while definition holes were drilled with BQ-diameter core. For all types of underground drilling, core was placed in wooden core boxes at the rig and delivered to surface for logging and sampling. Hole collar locations were marked by the drill crews with numbered metal tags to identify each hole, and then surveyed post-drilling by WRLG survey staff with the same survey equipment and controls used in all other aspects of mine surveying.

Surface drill rigs were aligned on proper azimuth and dip using a north-seeking DeviCo Devialigner™ for drill alignments and with a DeviCo DeviGyro™ downhole gyro tools for all downhole surveying. Downhole survey shots on surface exploration holes were taken every 30 m to monitor deviation with continuous in-and-out surveys collected upon completion of the hole. Underground drill rigs were aligned with front and back sites which were established by WRLG survey personnel. Continuous in-and-out surveys were collected on all underground drill holes.

All surface exploration and underground expansion drill core was oriented using a Reflex ACTIII™ core orientation tool. Underground definition drill core was not oriented.

All surface drilling sites were cleared of any cut timber and debris, and sites outside the active mine site were re-contoured and re-seeded with a native seed mix post-drilling.

All drill holes were logged, photographed, and sampled at the mine following the procedures fully described in Sections 10.6.1 to 10.6.7. All drill core is geologically logged directly into a laptop or tablet computer using the LogChief™ software platform and synchronized directly from these devices to WRLG’s Datashed database.

10.6.1 Core Processing

Upon initiation of drilling on a hole, the senior geologist assigns a logging geologist and (if required) a geotechnician to the hole. Data collection responsibility for each hole is tasked to the logging geologist and supervised by the project geologist. It is the responsibility of the senior geologist to validate all drill hole data and ensure transfer to the database on completion of the logging and sampling.

10.6.2 Geological Quick Logging

In the case of exploration drillholes, immediately following delivery of core from the drill rig to the core shack at morning shift change the logging geologist assigned to each drill hole completes a quick log of geology and mineralization and the results of the quick log are entered into an online tracking sheet. Observations and interpretation are discussed at daily meetings to enable consistent interpretation and adjustments for the planning of subsequent holes. The emphasis is on the mineralized zones and the potential to host gold.

Quick logging is not completed for resource definition drillholes.

10.6.3 Geotechnical Procedures

All drill core is prepared by a technician prior to geological logging. Procedures vary depending whether the drilling is classed as exploration or definition. For exploration drilling, this preparation work includes reassembly and orientation of drill core pieces, checking and correction of block errors, drawing bottom of hole core orientation marks on core, assigning systematic orientation quality scores to each interval, recording loss of orientation lines and placing down-hole metre marks, as well as measuring recovery, rock quality designation, specific gravity and magnetic susceptibility. All downhole measurements are collected to the nearest centimetre. Procedures are largely the same for definition drilling, except that orientation, RQD and magnetic susceptibility are not recorded. In some selected instances, detailed geotechnical logging is conducted, quantifying joint characteristics and core resistance to breakage.

Each core box is permanently labelled with details of drill hole number, box number and depth interval engraved onto an aluminum tag affixed to the end of the tray. Box intervals are recorded into an Excel file and retained.

10.6.4 Geological Logging

All drill core is geologically logged directly into a laptop or tablet computer using the LogChief™ software platform and synchronized directly from these devices to WRLG's Datashed database. Additionally, geological boundaries and annotations are marked on the core using china markers on the portion of the core to be retained after cutting and sampling. Due to the focus on mineralization, any major structures (primarily large or gold-hosting veins) are reported both in the lithology table as well as in the structure or vein tables. Lithologies are split out for intervals that are greater than 1 m in core length and/or of geological significance. Individual veins 30 cm wide or greater are logged separately. The focus of the geologic logging is to highlight the gold associated alteration zones and to capture lithology, vein and structural data.

10.6.5 Structural Data

All exploration drilling by WRLG has used the Reflex ACTIII ™ core orientation tool. All core intervals in the Balmer Assemblage target units are oriented with a bottom of core mark. Representative average foliation measurements are logged downhole every 10 to 20 m or on major changes in orientation. Strike and dip of key structural features including vein and lithologic contacts, fold axes and lineations are recorded using alpha, beta and gamma angles and software is used to calculate and plot the true strike and dip of structural features. The structural data are then visualized in Leapfrog Geo™ or Micromine™ software.

Core orientation is not performed for core drilled as part of definition programs.

10.6.6 Core Photography

After sample lay-out (but prior to sawing) all drill core is photographed both wet and dry using a professional grade digital camera in a fixed mount with standardized camera settings, lighting and layout. HoleID, core box number, depth blocks, cut lines and sample marks and tags are visible in the photographs. Digital photograph files are renamed to include the hole number, box numbers and depths.

10.6.7 Core Sampling

Sample intervals are laid out by the logging geologist as part of the logging workflow. Samples are not to exceed 1.5 m in length and must be at least 0.5 m in length. The recommended length for samples in zones of mineralization is 0.5 m so as to aid in regularity of composite calculation, though exceptions can be made in cases of dilution by post-mineral units within a mineralized interval. Sample boundaries are set up so as to not to cross lithological contacts, except in cases where it is not possible to maintain minimum sample lengths while respecting these contacts; in such cases sampling may cross lithological contacts. Sample selection from exploration drilling includes all Balmer Assemblage rocks and any altered or veined Confederation Assemblage rocks as well as the lowermost 30 m of Confederation Assemblage against the Balmer Assemblage contact. Sample selection from definition drilling follows the same guidelines and also excludes ultramafic sills to reduce assay volumes.

Core from exploration drilling is sawn in half with a diamond blade core saw along lines marked by the logging geologist. Core samples from definition drilling are sampled as whole core.

10.6.8 Core Storage

After logging, photographing and sampling, the remaining exploration core is labeled with aluminum tags on each box with details of drill hole number, box number and depth interval. The boxes are then cross-stacked, strapped and stored in ordered rows on pallets at a core storage facility on the Madsen project site. All returned pulps and coarse reject material from the assay labs are tarped and also stored in this area for a period of time deemed appropriate (generally 1 year), after which point the pulps and rejects are discarded.

10.7 Interpretation

Diamond (core) drilling is the most appropriate test method for the mine and this technique has been applied by all operators since early exploration and mining. Historical drilling is tightly-spaced (nominally drilled at 6 m centres) within mined-out areas but other largely non-mined areas show evidence of alteration and elevated gold and have been drilled at much broader spacing.

11 Sample Preparation, Analysis and Security

11.1 Sampling, Sample Preparation and Analysis

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

The QP is of the opinion that, based on historical information available, the historical 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 undertaken up to WRLG's acquisition of the property meet or exceed modern best practices. The historical procedures and those undertaken by Pure Gold are adequate for modern targeting, modelling and resource estimation.

Sampling procedures employed by WRLG since acquisition of the property have not been reviewed by the resource QP as no data collected by WRLG has been incorporated into the Mineral Resource Estimate. The sampling information included here is for the sake of completeness and ownership continuity.

11.1.1 Historical Sampling

Drill core, chip and muck sample preparation, analysis and security procedures for historical samples taken during the operation of the mine are not documented and therefore difficult to review. Samples were assayed for gold at the mine laboratory but no information exists regarding laboratory certifications or preparation and assaying procedures. ISO 9000 series standards were first published in 1987, and the ISO 17025 standard was first published in 1999 and as such could not have been applied. Assay results are hand-written or typed on paper logs, level maps and sections. These paper documents have been well catalogued and preserved in an orderly manner at the site.

Sample preparation, analysis and security procedures for limited historical samples taken by Central Patricia Gold Mines and Cockeram Red Lake Gold Mines between 1943 and 1946 and by Noranda Inc. in 1981 and 1982 are unknown. No information exists regarding laboratory certifications but as indicated in the preceding paragraph, such early work predates applicable ISO standards. The preparation and assaying techniques are not documented. These holes are generally not within current resource areas.

11.1.2 Placer Dome

Placer Dome used two primary laboratories for assaying drill core samples collected from the mine. All samples from 2001 to 2006 were assayed by XRAL Laboratory in Toronto, Ontario or ALS Chemex Laboratory in Vancouver, British Columbia. Samples were analyzed for gold by fire assay and 32 or 37 multielement packages with aqua regia acid digest. Drill core sample lengths range from 0.7 to 5 feet.

11.1.3 Wolfden and Sabina

Wolfden submitted drill core samples to Accurassay Laboratories in Thunder Bay, Ontario. Accurassay received ISO 17025 accreditation in 2002 from the Standards Council of Canada. It is unknown which analytical methods were covered under this accreditation.

At Accurassay, samples were prepared using a standard rock preparation procedure consisting of drying, weighing, crushing, splitting, and pulverization. Prepared samples were assayed for gold, platinum, palladium, and rhodium as well as for a suite of base metals using ICP-MS.

Procedures followed by Sabina are recorded in more detail. In 2010 and 2011 Sabina submitted drill core samples to SGS Laboratories (SGS) in Red Lake for sample preparation and analysis. SGS was accredited by the Standard Council of Canada (SCC) to ISO 17025:2005 (accredited laboratory number 598) for gold analysis by fire assay.

All samples were delivered by Sabina personnel to SGS. Sample preparation and assay analysis included crush to 75% passing 2 mm and then pulverizing a 250 g split to 85% passing 75 µm. Samples were assayed by fire assay with an atomic absorption spectroscopy (AAS) finish on 50 g aliquots. A duplicate sample was assayed by SGS as part of their assaying procedures.

In 2012, Sabina submitted drill core samples to Activation Laboratories Ltd. (Actlabs) in Red Lake for sample preparation and analysis. Actlabs was accredited to ISO 9001:2008 by Kiwa International Cert GmbH (certificate number 1109125). Samples were crushed to 90% passing 2 mm after which a 250 g split was pulverized to 95% passing 105 µm. Samples were assayed by fire assay with AAS finish using a 30 g aliquot.

11.1.4 Claude

Claude used four primary laboratories between 2006 and 2012 for drill core analysis. SGS Laboratory in Red Lake and TSL Laboratory located in Saskatoon, Saskatchewan were used from 2006 to May 2008, until Claude identified performance issues with samples submitted to the SGS Laboratory in Red Lake and as a result stopped submitting samples to this laboratory. Starting in 2009, Claude submitted drill core samples to Accurassay Laboratories in Thunder Bay, Ontario but experienced lengthy delays in receiving results. Then in 2010, Claude submitted all drill core samples to ALS Limited (ALS) in Thunder Bay for sample preparation and to ALS Vancouver for assaying. All these laboratories are accredited ISO/IEC Guideline 17025 by the Standards Council of Canada for conducting certain testing procedures, including the procedures used for assaying samples submitted by Claude. These laboratories also participate in proficiency testing programs.

These laboratories all used standard rock sample preparation procedures involving coarse crushing dried sample, pulverization of 500 g subsamples to 90% passing 150 mesh screens (105 µm). All core samples were assayed for gold using a standard fire assay procedure on pulverized subsamples with an atomic absorption finish. Samples assaying more than 1.0 g/t Au were re-analyzed by fire assay with a gravimetric finish. Samples assaying greater than 5.0 g/t Au were re-analyzed using screen metallic fire assay procedures.

11.1.5 Pure Gold

During 2014, 2015, 2016, 2018 and 2019 Pure Gold submitted all exploration drill core and surface rock samples to ALS Minerals (ALS) Laboratory in Thunder Bay and Vancouver for sample preparation and analysis, respectively. During these programs, Pure Gold submitted pulp duplicate samples to SGS Laboratory in Burnaby, British Columbia for check assay testing. In 2017, Pure Gold submitted all drill core and surface rock samples to SGS Minerals Services (SGS) in Red Lake for sample preparation and gold analysis, with additional analyses conducted at SGS's Vancouver facility. Owing to capacity limitations in Red Lake, some samples were diverted to the SGS Laboratories in Lakefield and Burnaby for preparation and analysis after being delivered to the Red Lake laboratory. During the 2018 underground bulk sample program, Pure Gold submitted all underground drill core, muck and chip samples to the SGS laboratory in Red Lake for sample preparation and gold analysis. During the 2020, 2021 and 2022 surface exploration drilling programs, samples were submitted to both ALS and SGS for analysis, while core from definition drilling programs (both surface and underground) during 2021 and 2022 was submitted to SGS. Table 11-1 summarizes analytical labs used by Pure Gold by year and sample source.

Table 11-1: Summary of analytical labs used by Pure Gold by year and sample source

	Sample Source		
Year	**Exploration Drilling & Surface Sampling**	**Definition Drilling**	**Chip, Muck and Testhole Sampling**
2014	ALS		
2015	ALS		
2016	ALS		
2017	SGS	SGS	
2018	ALS	SGS	SGS
2019	ALS		
2020	ALS & SGS		SGS
2021	ALS & SGS	SGS	SGS
2022	ALS & SGS	SGS	SGS

Source: SRK (2024)

The ALS laboratory in Vancouver is ISO 9001:2008 and CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005) certified by the Standards Council of Canada (SCC) for the analytical methods used on the mine samples (accredited lab 579). The SGS laboratory is CAN-P-1579, CAN-P-1587, and CAN-P-4E (ISO/IEC 17025:2005) certified by the SCC for the analytical methods used on the mine samples (accredited lab 744).

Samples were dried and crushed to 70% of the sample passing a 2 mm screen (method CRU-31). Initial crushing was followed by a Boyd rotary split of a 1 kg subsample (method SPL-22Y), and pulverization of the split in a ring mill to better than 85% of the ground material passing through a 75 µm screen (method PUL32).

A summary of the analytical laboratories and methods used by Pure Gold for drill core analysis is provided in Table 11-2. Sample pulps were shipped by ALS from the Thunder Bay preparation laboratory to the ALS laboratory in Vancouver for analysis. Assays for gold were by a 30 g aliquot fire assay followed by aqua regia (HNO3-HCl) digestion and measurement by atomic absorption spectroscopy (AAS, method Au-AA23). Samples in which the gold concentration exceeded 5 ppm were re-assayed from the same pulp by method Au-GRA21, fire assay of a 30 g aliquot, parting with nitric acid (HNO3) followed by gravimetric gold determination. In cases of significant visible gold in samples, the complete interval including shoulder samples was re-assayed by metallic screen fire assay (method Au-SCR24). This method was also manually selected in some instances in 2014 and 2015 where high assay values were returned from Au-GRA21 results. In addition to the gold assays, multi-element geochemical trace level analyses were completed by induction coupled plasma-atomic emission spectroscopy (ICP-AES, method ME-ICP61) following digestion by hydrofluoric (HF), nitric (HNO3) and perchloric (HClO4) acids followed by a hydrochloric (HCl) acid leach.

Table 11-2: Summary of analytical methods and labs used by Pure Gold for drillcore analysis

Year	Lab					
	ALS			SGS		
	Preparation	Gold Analysis	Multi-element	Preparation	Gold Analysis	Multi-element
2014	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61			
2015	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61			
2016	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61			
2017	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61	G_PRP89	GE_FAA313, GO_FAG303, GO_FAS31K, GO_FAS51K	GE_ICM40B
2018	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61	G_PRP89	GE_FAA313, GO_FAG303, GO_FAS31K, GO_FAS51K	GE_ICM40B
2019	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61			
2020	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61	G_PRP89	GE_FAA313, GO_FAG303, GO_FAS31K, GO_FAS51K	GE_ICM40B

Year	Lab					
	ALS			SGS		
	Preparation	Gold Analysis	Multi-element	Preparation	Gold Analysis	Multi-element
2021	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61	G_PRP89	GE_FAA30V5, GE_FAA30V10, GO_FAG30V, GO_FAS30M	
2022	PREP31	Au-AA23, Au-GRA21, Au-SCR24	ME-MS61	G_PRP89	GE_FAA30V5, GE_FAA30V10, GO_FAG30V, GO_FAS30M	

Source: SRK (2024)

As routine external quality control methods for the samples re-assayed by method Au-SCR24 were not practical, for this method Pure Gold relied on the internal quality control performed by ALS and a comparison with the initial assays by methods Au-AA23 and Au-GRA21.

The SGS laboratory in Red Lake is CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005) certified for the analytical methods used on the mine samples (accredited lab 598). The SGS laboratory in Vancouver is CAN-P-1587, CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005) certified for the analytical methods used on the mine samples (accredited lab 744). The SGS laboratory in Lakefield is CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005) certified for the analytical methods used on the mine samples (accredited lab 184).

Samples were submitted with the preparation code G_PRP89, as part of which samples were dried and weighed (method G_WGH79) and crushed to 75% of the sample passing a 2 mm screen (method G_CRU21, method G_CRU22 where sample weight is >3.0 kg). Initial crushing was followed by a split (to obtain a sample weight of 1.0–1.5 kg), and then pulverization of the split in a chromium steel bowl to better than 85% of the ground material passing through a 75 µm screen (method PUL47).

Analysis for gold was conducted at the SGS laboratory in Red Lake. During 2017, 2018, 2020, part of 2021 and 2022, analysis was by a 30 g fire assay with an atomic absorption spectroscopy finish (methods GE_FAA313 & GE_FAA30V5). In cases where the assay value returned greater than 5 ppm Au, a follow-up gravimetric analysis was conducted (30 g fire assay with a gravimetric finish, method GO_FAG303). In cases where gold was identified during core logging, a screen metallic gold analysis was conducted in addition to the AAS and gravimetric analytical procedures (screen to 106 µm followed by fire assay, method codes GO_FAS31K and GO_FAS51K for samples less than 1 kg and greater than 1 kg respectively). During late 2021 and 2022, this suite of methods was streamlined to using method GE_FAA30V10 for all gold analyses, with GO_FAG30V (a replacement code for the method GO_FAG303 used previously) triggered if a value of greater than 100 ppm Au was returned.

In addition to the gold assays, 49-element geochemical trace level analyses were completed in the Burnaby laboratory by induction coupled plasma-atomic emission spectroscopy (ICP-AES) and induction coupled plasma mass spectrometry (ICP-MS) following digestion by hydrofluoric (HF), nitric (HNO3), perchloric (HClO4) and hydrochloric (HCl) acids (method GE_ICM40B).

11.1.6 West Red Lake Gold Mines

During 2023 and 2024, WRLG submitted all underground drill core to SGS Minerals Services (SGS) in Red Lake for sample preparation and gold analysis with additional analyses conducted at SGS's Burnaby (B.C.) facility. Surface exploration drill core was submitted to SGS in Red Lake for gold analysis and ALS Vancouver via ALS Thunder Bay for additional analyses (Table 11-3).

Table 11-3: Summary of analytical methods and labs used by WRLG for drillcore analysis

Year	Lab					
	ALS			SGS		
	Preparation	Gold Analysis	Multi-element	Preparation	Gold Analysis	Multi-element
2023	-	-	ME-MS61	G_PRP89	GO_FAA50V10 GO_FAG50V, GO_FAS50M	GE_ICP40Q12
2024	-	-	ME-MS61	G_PRP89	GO_FAA50V10 GO_FAG50V, GO_FAS50M	GE_ICP40Q12

Source: WRLG (2024)

Drilling completed underground at the Madsen Mine consisted of BQ-sized diamond drill core for definition drill programs and oriented NQ-sized diamond drill core for exploration focused drilling. All drill holes are systematically logged, photographed, and sampled by a trained geologist at the Madsen Mine core processing facility. Minimum allowable sample length is 0.5 m. Maximum allowable sample length is 1.5 m. Control samples (certified standards and uncertified blanks), along with duplicates, are inserted at a target 5% insertion rate. Results are assessed for accuracy, precision, and contamination on an ongoing basis. The BQ-sized drill core is whole core sampled. The NQ-sized drill core is 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 is sampled consistently utilizing the orientation line as reference. For those samples containing visible gold ("VG"), a trained geologist supervises the cutting/bagging of those samples, and ensures the core saw blade is 'cleaned' with a dressing stone following the VG sample interval. Bagged samples are then sealed with zip ties and transported by Madsen Mine personnel directly to SGS Natural Resource's Facility in Red Lake, Ontario for assay.

Samples are then prepped by SGS, which consists of drying at 105°C and crushing to 75% passing 2 mm (SGS Code: G_PRP89). A riffle splitter is then utilized to produce a 500 g course reject for archive. The remainder of the sample is then pulverized to 85% passing 75 microns from which 50 g is analyzed by fire assay and an atomic absorption spectroscopy (AAS) finish (SGS Code GO-FAA50V10). Samples returning gold values greater than 100 g/t Au are reanalyzed by fire assay with a gravimetric finish on a 50 g sample (SGS Code GO_FAG50V). Samples with visible gold are also analyzed via metallic screen analysis (SGS code: GO_FAS50M). For multi-element analysis, samples are 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 Quality Management System that complies with ISO/IEC 17025.

Exploration drilling completed on surface at the Madsen Mine consisted of oriented NQ-sized diamond drill core. All drill holes are systematically logged, photographed, and sampled by a trained geologist at the Madsen Mine core processing facility. Minimum allowable sample length is 0.5 m. Maximum allowable sample length is 1.5 m. Control samples (certified standards and uncertified blanks), along with duplicates, are inserted at a target 5% insertion rate. Results are assessed for accuracy, precision, and contamination on an ongoing basis. The NQ-sized drill core is 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 is sampled consistently utilizing the orientation line as reference. For those samples containing visible gold ("VG"), a trained geologist supervises the cutting/bagging of those samples, and ensures the core saw blade is 'cleaned' with a dressing stone following the VG sample interval. Bagged samples are then sealed with zip ties and transported by Madsen Mine personnel directly to SGS in Red Lake, Ontario for assay.

Samples are then prepped by SGS, which consists of drying at 105°C and crushing to 75% passing 2 mm (SGS Code: G_PRP89). A riffle splitter is then utilized to produce a 500 g course reject for archive. The remainder of the sample is then pulverized to 85% passing 75 microns from which 50 g is analyzed by fire assay and an atomic absorption spectroscopy (AAS) finish (SGS Code GO-FAA50V10). Samples returning gold values greater than 10 g/t Au are re-analyzed by fire assay with a gravimetric finish on a 50 g sample (SGS Code GO_FAG50V). Samples with visible gold or returning gold values greater than 30 g/t Au are also analyzed via metallic screen analysis (SGS code: GO_FAS50M). For multi-element analysis, samples are sent to ALS's facility in Vancouver, British Columbia via Thunder Bay, Ontario and analyzed via four-acid digest with an atomic emission spectroscopy (ICP-AES) finish for 48-element analysis on 0.25 g sample pulps (ALS code: ME-MS61). ALS Geochemistry analytical laboratories operate under a Quality Management System that complies with ISO/IEC 17025.

11.2 Sample Security

11.2.1 Historical Sampling

During the historical sampling period, sample security procedures employed are unknown.

11.2.2 Claude

Claude implemented chain of custody and sample security procedures in 2006 as documented and directly observed by Cole et al. (2010). Procedures generally involved sample handling by appropriately qualified staff, controlling access to sampling facilities and documentation of sample dispatch and receipt at laboratories.

11.2.3 Pure Gold

Pure Gold personnel employed the following security and chain of custody procedures:

i. Core is placed in wooden core boxes by drilling contractors, covered with wooden lids, and sealed with fiber tape

ii. Core boxes are delivered to the logging facility by drill crew members at twice daily shift changes via truck or mine equipment

iii. Core shack personnel open and sort core boxes for logging

iv. Core awaiting logging or sampling is stored in wooden racks in the core shack

v. Core is sampled and bagged into pre-labelled sample bags by samplers under the supervision of core logging geologists and the project geologist or by the geologists themselves

vi. Sample bags are placed inside pre-labelled rice sacks

vii. Rice sacks containing bagged samples are sealed and palletized (or placed within plastic shipping totes or dedicated collection points) within the core shack

viii. Palletized containers of rice sacks are shipped directly from the core shack to laboratory preparation facilities. For programs utilizing ALS, Manitoulin Transport of Winnipeg, Manitoba transported pallets to the ALS Minerals laboratory in Thunder Bay, Ontario for sample preparation. For programs utilizing SGS, samples bags are collected from site directly by SGS personnel and driven to their Red Lake facility

ix. Access to the core logging facility is restricted to authorized staff

x. Analytical instructions are included with each shipment with copies sent by email. ALS and SGS are instructed to report any discrepancies between sample lists as shipped and as received at the laboratory

11.3 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation and assaying processes. They are also important to prevent sample mix-up and monitor the voluntary or inadvertent contamination of samples. Assaying protocols typically involve regular duplicate and replicate assays and insertion of quality control samples. Check assaying is typically performed as an additional reliability test of assaying results. This typically involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

Several operators prior to 2009 make mention of the existence of quality control programs but provide few details. Implementation of rigorous analytical quality control measures for Madsen Mine geochemical samples began in 2009 by Claude. Original records of these programs are sparse but SRK had direct access to the Claude work program and has reported on the procedures adopted and results obtained (Cole et al., 2010).

11.3.1 Historical Period

Analytical quality control measures implemented by operators during early exploration activities or at the mine laboratory during the operation of the Madsen Mine (1936–1976) are unknown. Analytical quality control measures implemented by Claude between 1998 and 2000 are undocumented.

11.3.2 Placer Dome

Placer Dome annual project reports indicate that analytical quality control measures were implemented, however the details of these measures and the analytical quality control data were not transferred to Claude in 2006 and as such no documentation is available.

11.3.3 Wolfden and Sabina

Wolfden and Sabina implemented external analytical quality control measures on drill core sample analysis. The procedures are unknown and data prior to 2006 are unavailable. Implemented measures from 2006 included insertion of control samples (blank and standard reference material) into the sample stream on regular intervals. A sample blank was inserted every 25 samples and a standard inserted every 75 samples.

The blank material was sourced from an outcrop in the southwest corner of Wolfden's Bonanza/Follansbee property. Representative samples of this material were assayed by Accurassay Laboratories to ensure suitability. The performance of the blank material is unknown.

A 2006 drilling report noted that two different standards were used, SK21 that had a certified assay of 4.048 g/t Au and SN16 that had a certified assay of 8.367 g/t Au. Certificates are not available and the source of the standards is unknown. The report suggests performance issues with standard SK21 as the average assay value was approximately 10% higher than the accepted value. However, only 21 assay results are available, which is too few to extract meaningful statistical information from the results.

Sabina submitted blank and standard material in the sample stream at a rate of one quality control sample every 20 samples. No information is available detailing the type and source of the reference material.

11.3.4 Claude

The exploration work conducted by Claude after 2006 was carried out using a quality assurance and quality control program in line with industry best practices. Standardized procedures were used in all aspects of exploration data acquisition and management including mapping, surveying, drilling, sampling, sample security, assaying and database management (Cole et al, 2010).

Claude relied partly on the internal analytical quality control measures implemented by the primary laboratories. Assay results for quality control samples inserted by the primary laboratories were submitted with routine assaying results and reviewed for consistency by Claude personnel.

Additionally, Claude implemented external analytical quality control measures to monitor the reliability of the assaying results delivered by the primary laboratories. External control samples (blanks, field or CRM samples or field duplicate) were inserted at a rate of approximately 13% within each batch of samples submitted for preparation and assaying.

Field duplicate samples were inserted at a rate of one in 50 for all batches of drilling samples submitted for assaying. Duplicate core samples were collected by splitting in half the remaining split core over the same length.

For the drilling program in 2009, Claude used four reference control samples purchased from Rocklabs. The silica sand blank material was sourced from Accurassay.

In 2010, Claude changed some of the standard reference materials used during the drill programs. A total of seven gold standards were alternated. Certified blanks included material from Rocklabs and Canadian Resource Laboratories.

A blank and a standard were inserted every 20 samples. The inserted standard typically alternated between three medium- to low-grade standards (SG40, SL46 and SH41). In addition, a high-grade standard and a blank were inserted after any sample containing visible gold.

No independent laboratory check assay tests are documented. Field duplicate samples were collected at a rate of one in 50 samples. Laboratory duplicate samples were not collected.

11.3.5 Pure Gold

For all Pure Gold drilling programs, Madsen Mine personnel implemented a Quality Assurance and Quality Control (QAQC) program comprising insertion of blank, CRM and duplicate samples into the drill core or rock sample streams. Results of gold analyses on these samples are monitored and corrective measures implemented where deficiencies are identified.

Field duplicate and preparation duplicate samples are alternately inserted every 20 samples. Field duplicates are obtained by quartering the core and submitting the two quarters in sequence to the laboratory. Preparation duplicates consist of a second split of the coarse reject of the selected sample and are collected by the laboratory during the sample crushing stage. Preparation duplicates are assigned the sample number immediately succeeding the original and in shipping are represented by a labeled empty bag containing the assigned sample tag. A list of preparation duplicates and instructions for preparation are included with each sample submittal form.

Blank sample material consists of commercially available marble landscape rock. An average weight of 2 kg is submitted for each blank sample. Blank samples are routinely inserted every 20 samples, with two additional blanks inserted following samples containing visible gold.

Standards used by Pure Gold between 2014 and 2022 ranged from low-, medium- and high-grade standards for routine analysis, with a higher-grade gold standard for samples with visible gold. These standards were selected to cover all potential analytical gold methods. Pre-packaged packets are used where available. Three primary standards were inserted on a rotating basis in roughly equal proportions every 20th sample, and the fourth high-grade standard was inserted when visible gold was identified in core. The standards used in these categories varied, dictated largely by availability of standards from commercial suppliers. Standard IDs, along with the supplier and certified gold values are listed in Table 11-4. Pure Gold requested extra cleaning of both crusher and pulverizer (ALS Codes: WSH-21 and WSH-22) during sample preparation of samples collected from within mineralized intervals (including shoulder samples).

Table 11-4: CRMs used by Pure Gold (2014–2022)

Supplier	Standard ID	Year(s) in use	Use Case	Gold Assays (ppm Au)	
				Certified Value	SD
CDN Labs	CDN-GS-1M	2016 - 2017	Low Grade	1.07	0.05
CDN Labs	CDN-GS-1T	2017 - 2021	Low Grade	1.08	0.05
CDN Labs	CDN-GS-1V	2017, 2019 - 2021	Low Grade	1.02	0.1
CDN Labs	CDN-GS-22	2016 - 2021	High Grade following VG	22.94	0.56
CDN Labs	CDN-GS-5F	2014 - 2015	High Grade	5.27	0.17
CDN Labs	CDN-GS-6E	2016 - 2021	High Grade	6.06	0.15
CDN Labs	CDN-GS-16	2017, 2019 - 2021	High Grade following VG	16.48	0.63
Ore Research	OREAS 17c	2014 - 2015	Medium Grade	3.04	0.08
Ore Research	OREAS 214	2016 - 2021	Medium Grade	3.03	0.08
Ore Research	OREAS6pc	2015	Low Grade	1.52	0.07
Ore Research	OREAS214	2016 - 2021	High Grade	3.03	0.082
Ore Research	OREAS216B	2017, 2020 - 2021	Hight Grade	6.66	0.158
Ore Research	OREAS 221	2017, 2020 - 2021	Low Grade	1.06	0.036
Ore Research	OREAS 222	2021	Low Grade	1.22	0.033
Ore Research	OREAS 226	2017, 2020 - 2021	High Grade	5.45	0.126
Ore Research	OREAS 229b	2020 - 2021	High Grade	11.95	0.288
Ore Research	OREAS 241	2021 - 2021	High Grade	6.91	0.309

Supplier	Standard ID	Year(s) in use	Use Case	Gold Assays (ppm Au)	
				Certified Value	SD
Ore Research	OREAS 242	2021 - 2021	High Grade	8.67	0.215
Ore Research	OREAS 243	2021	High Grade	12.39	0.306
Rocklabs	SG56	2014 - 2016	Low Grade	1.027	0.01
Rocklabs	SH55	2016	Low Grade	1.375	0.05
Rocklabs	SL61	2015 - 2016	High Grade	5.931	0.06
Rocklabs	SQ 36	2014 - 2016	High Grade following VG	30.04	0.02
Rocklabs	SQ87	2016	High Grade following VG	30.87	0.21

Source: Pure Gold (2022)

As part of its QAQC program, Pure Gold regularly commissioned independent specialists to report on performance of QAQC samples. Overall compliance rates for these samples are acceptable but given the considerable number of quality control samples submitted, numerous areas for improvement were highlighted by these independent reviewers and recommendations were made to Pure Gold management. These were addressed through sample re-analysis, discussion with laboratory management and through improvements in core shack and sampling protocols. For example, some carry-over of gold was detected within blank samples in 2016 but with the insertion of extra blank samples and requests for quartz washes of crushing equipment, this effect was largely mitigated.

All Pure Gold drilling data was verified as it was loaded to the Datashed™ database, including quality control samples. Reports illustrating performance of quality control samples were automatically generated through this process. A review of these reports indicated acceptable performance. Failures were identified and addressed by Pure Gold upon receipt of analytical certificates.

To monitor database integrity, Pure Gold routinely commissioned independent drill hole database reviews. Mackie (2015, 2017) reviewed sub-sets of drill holes completed by various operators and concluded that the drill hole database is of high quality and reliability and is a reasonable rendition of historic data. In 2019, Pure Gold commissioned a third independent review on the database (Murphy, 2019). Murphy identified some discrepancies between the Datashed™ database and an earlier database version, but all such issues were explained by further investigation and were concluded to be minor and generally attributable to the process of data migration to Datashed™. Murphy concluded that the database and structure were acceptable and a reasonable rendition of the historical and modern data.

11.3.6 West Red Lake Gold

For all WRLG drilling programs, Madsen Mine personnel implemented a QAQC program comprising insertion of blank, CRM and duplicate samples into the drill core or rock sample streams. Results of gold analyses on these samples are monitored and corrective measures implemented where deficiencies are identified.

Field duplicate and preparation duplicate samples are alternately inserted every 20 samples. Field duplicates are obtained by quartering the core and submitting the two quarters in sequence to the laboratory. Preparation duplicates consist of a second split of the coarse reject of the selected sample and are collected by the laboratory during the sample crushing stage. Preparation duplicates are assigned the sample number immediately succeeding the original and in shipping are represented by a labeled empty bag containing the assigned sample tag. A list of preparation duplicates and instructions for preparation are included with each sample submittal form.

Blank sample material consists of commercially available marble landscape rock. An average weight of 2 kg is submitted for each blank sample. Blank samples are routinely inserted every 20 samples, with two additional blanks inserted following samples containing visible gold.

Standards used by WRLG between 2023 and 2024 ranged from low-, medium- and high-grade standards for routine analysis, with a higher-grade gold standard for samples with visible gold. These standards were selected to cover all potential analytical gold methods. Pre-packaged packets are used where available. Three primary standards were inserted on a rotating basis in roughly equal proportions every 20th sample, and the fourth high-grade standard was inserted when visible gold was identified in core. The standards used in these categories varied, dictated largely by availability of standards from commercial suppliers. Standard IDs, along with the supplier and certified gold values are listed in Table 11-5. WRLG requested extra cleaning of both crusher and pulverizer (ALS Codes: WSH-21 and WSH-22) during sample preparation of samples collected from within mineralized intervals (including shoulder samples).

Table 11-5: CRMs used by WRLG (2023–2024)

Supplier	Standard ID	Year(s) in use	Use Case	Gold Assays (ppm Au)	
				Certified Value	SD
CDN Labs	CDN-GS-1M	2016 - 2017	Low Grade	1.07	0.05
CDN Labs	CDN-GS-1T	2017 - 2021	Low Grade	1.08	0.05
CDN Labs	CDN-GS-1ZA	2024	Low Grade	1.37	0.04
CDN Labs	CDN-GS-5Y	2024	Medium Grade	5.21	0.16
CDN Labs	CDN-GS-7L	2024	Medium - High Grade	7.91	0.18
CDN Labs	CDN-GS-22	2024	High Grade VG	22.94	0.56
Ore Research	OREAS 239	2024	Low - Medium Grade	3.55	0.09
Ore Research	OREAS216B	2024	Medium - High Grade	6.66	0.158
Ore Research	OREAS 241	2024	Medium - High Grade	6.91	0.309

Source: WRLG (2025)

All WRLG drilling data have been verified as it was loaded to the Datashed™ database, including quality control samples. Reports illustrating performance of quality control samples are automatically generated through this process. A review of these reports indicates acceptable performance. Failures were identified and addressed by WRLG upon receipt of analytical certificates.

12 Data Verification

Owing to the long history of exploration and production at the mine, there have been numerous campaigns of data verification, validation and reconciliation. The most comprehensive recorded verification effort (Cole et al., 2010) was conducted during the digitization of the mining-era hardcopy drill logs, prior to Pure Gold's acquisition of the property. This work was initiated by Claude in 1998, advanced by Placer Dome from 2002 to 2006 and completed by Claude with assistance from SRK during 2008 and 2009. The result was a modern digital database comprising 13,617 historical drill holes with lithological intervals and 550,687 gold assays. This database was the foundation for drill-targeting, geological interpretation and mining by Pure Gold and has been substantially added to and verified since 2014.

12.1 Performance of CRM, Blank and Duplicate Samples

A database with historical and Pure Gold QAQC sample data, including certified reference materials (N = 32,163), blanks (N = 19,666) and duplicates (N = 20,592) was evaluated. Table 12-1 summarizes QAQC performance.

12.1.1 Certified Reference Materials

Data for the 32,163 Certified Reference Materials (CRMs) provided by Pure Gold include "external" CRM samples inserted by the operators as well as "internal" standards inserted by the analytical laboratories. The following summary provides an overview for a subset of CRMs inserted by Pure Gold between 2014 and 2021 into the sample streams of drill holes that start with "PG", "PGB", "PGC", PGP", "PGT", or "PGU" (i.e., all holes drilled by Pure Gold). The total number of CRMs in this subset is 10,197 (here referred to as the "2014-21 PG subset") of which 333 were CRMs that had fewer than 100 insertions and 204 are considered to be erroneously labelled or mishandled. These CRMs were removed from the subset to take the total down to 9,660.

First pass calculation of Z-scores showed a "failure" rate of 5% (N = 482), where a "failure" is defined as any CRM analysis returning a Z-score of greater than 3 or less than -3. Particularly high failure rates were recorded for CDN-GS-22 (12%) as well as OREAS standards OREAS 216b (11%), OREAS 222 (22%), OREAS 226 (12%) and OREAS 229b (11%). Z-scores for these 9,660 CRMs average 0.17 and 95% of the data averages 0.15, indicating that, overall, there is no systematic bias. There is some positive or negative bias on shorter time scales (Figure 12-1).

CRMs indicate that analyses are unbiased and sufficiently accurate.

12.1.2 Blanks

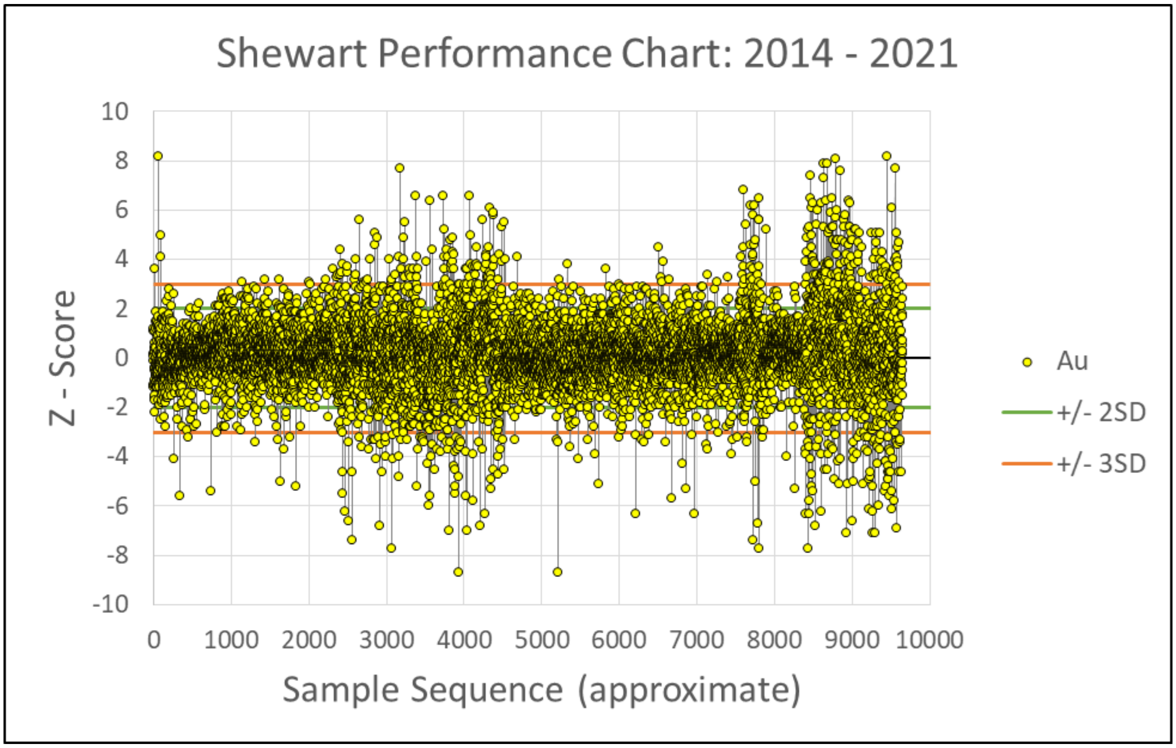
The 2014-21 PG subset also includes 10,676 blank samples, 85 (0.8%) of which exceed a threshold of 10 times the lower detection limit (LDL) that is typically used to indicate contamination (Figure 12-2). Some of these anomalous blanks include a sample that returned 7 g/t Au, and 58 others that returned between 0.1 to 5.8 g/t Au. Most of these samples can be readily explained by high-grade results in the preceding samples, which is typical laboratory performance where samples are characterized by coarse gold. Pure Gold implemented a procedure of inserting multiple sequential blanks following samples where gold was visually identified by the logging geologist to address this contamination. Typically, the contamination tails off through the multiple blanks. Other contaminated blanks are in fact duplicates of core samples mislabelled as blanks and may also reflect sample handling errors (Equity Exploration Consultants Ltd., 2022).

Table 12-1: Overview of CRM Performance for a Subset of 6,100 Samples, with Failure Rates Calculated using the Standard Deviation (SD) of current results

Supplier	CRM ID	Certified Mean	SD	N in subset	N failures	Failure %
CDN Labs	CDN-GS-1M	1.07	0.045	1108	21	1.9%
CDN Labs	CDN-GS-1T	1.08	0.05	898	20	2.2%
CDN Labs	CDN-GS-1V	1.02	0.049	367	22	6.0%
CDN Labs	CDN-GS-22	22.94	0.56	260	31	11.9%
CDN Labs	CDN-GS-5F	5.27	0.17	111	4	3.6%
CDN Labs	CDN-GS-6E	6.06	0.17	2089	87	4.2%
OREAS	OREAS 214	3.03	0.082	2179	89	4.1%
OREAS	OREAS 216b	6.66	0.16	793	88	11.1%
OREAS	OREAS 221	1.06	0.04	450	7	1.6%
OREAS	OREAS 222	1.22	0.03	161	35	21.7%
OREAS	OREAS 226	5.45	0.13	480	59	12.3%
OREAS	OREAS 229b	11.95	0.29	142	15	10.6%
Rocklabs	SG56	1.027	0.033	225	1	0.4%
Rocklabs	SH55	1.375	0.045	208	2	1.0%
Rocklabs	SL61	5.931	0.177	189	1	0.5%
Total				**9660**	**482**	**5.0%**

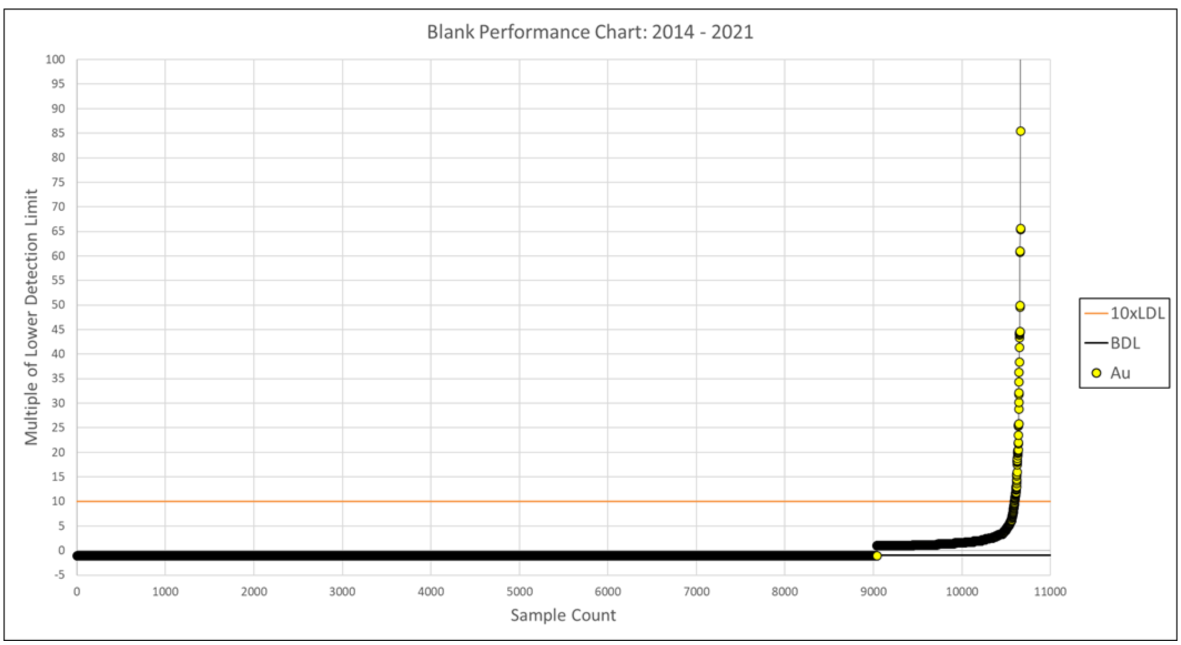
Source: Equity Exploration Consultants Ltd. (2022)

Figure 12-1: Shewart plot for 9,660 CRMs (yellow)



Source: Equity Exploration Consultants Ltd. (2022)

Figure 12-2: Blank Assays for Pure Gold's 2014-2021 Drilling. 10X detection limit threshold used as indicating a blank failure.



Source: Equity Exploration Consultants Ltd. (2022)

12.1.3 Duplicates

The 2014-21 PG subset also includes 17,401 duplicates, comprising 3,825 core (or field), 5,991 crush (or preparation) and 7,585 pulp (or lab) duplicate samples. All core duplicate pairs show a correlation coefficient (ρ) of 0.83 and an average coefficient of variance (CV_{AVR}) of 39%, which is at the upper limit of "acceptable" as defined by Abzalov (2008). The CV_{AVR} can be improved to 36% by eliminating duplicate pairs where at least one sample returned below detection. Removal of three additional outliers improves ρ from 0.83 to 0.85 and r^2 from 0.69 to 0.73 (Table 12-2).

Crush duplicates show a slightly stronger correlation (ρ = 0.88) than the core duplicates, and slightly better CV_{AVR} of 33%. This is consistent with the increased homogeneity of crush duplicates relative to core. Elimination of pairs with at least one sample below detection improves CV_{AVR} to 28% so that it falls within the "acceptable" range for orogenic gold deposits. Removal of two outliers improves ρ to 0.93 and r^2 from 0.94 to 0.96 (Table 12-2).

Table 12-2: Summary Statistics for Duplicate Samples

Duplicate Type	All data				<dl, outliers removed			
	N	ρ	R^2	CV_{AVR}	N	ρ	R^2	CV_{AVR}
Core or Field	3825	0.83	0.69	39%	2548	0.85	0.73	36%
Crush or Preparation	5991	0.94	0.88	33%	4017	0.96	0.93	28%
Pulp or Lab	7585	0.98	0.97	29%	5124	1.00	0.99	22%

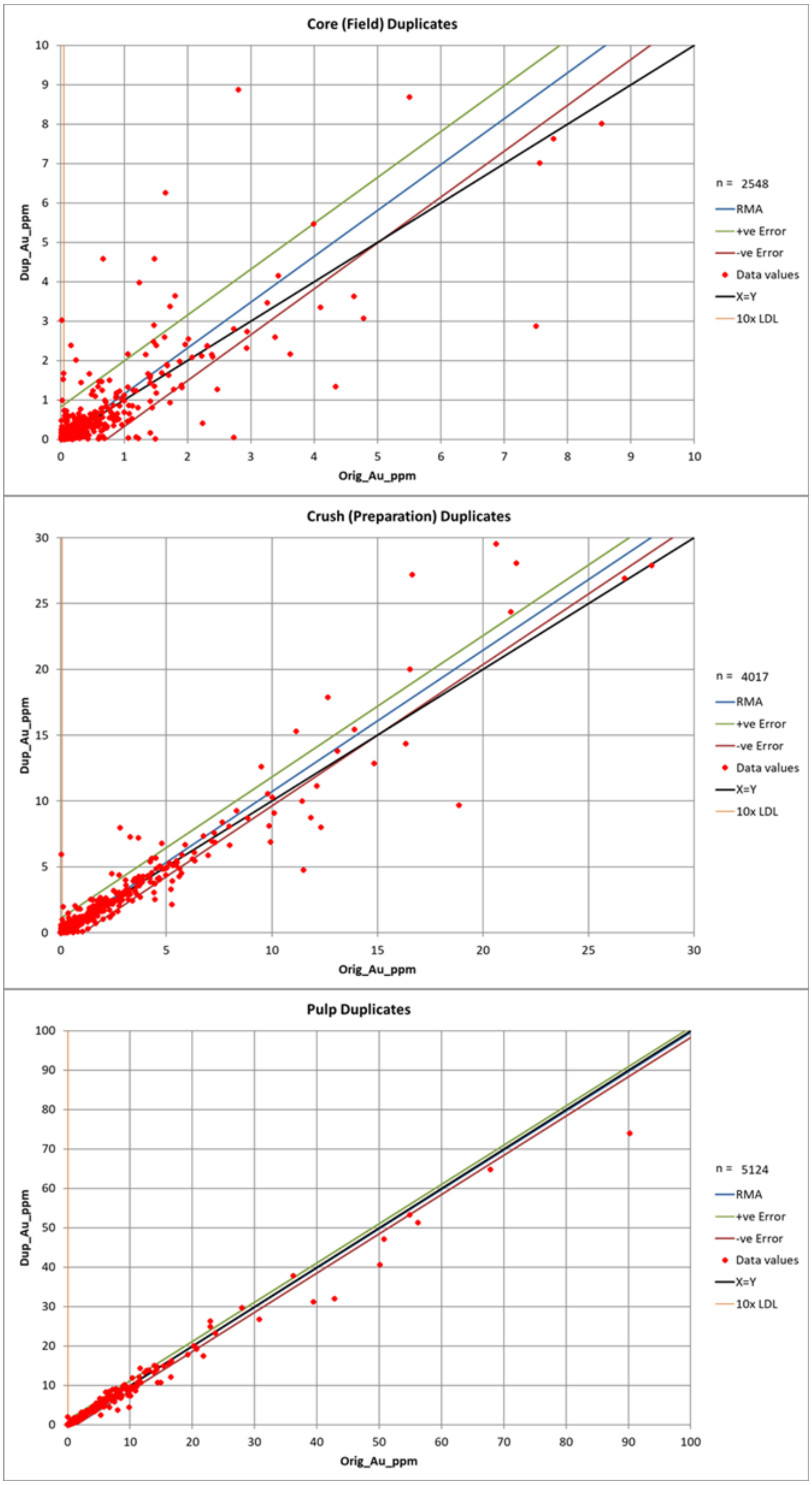
Source: Equity Exploration Consultants Ltd. (2022)

Pulp duplicates show a stronger correlation (ρ = 0.97) than crush duplicates and a CV_{AVR} of 29%, although this is still above the 10-20% range considered best practice to be acceptable by Abzalov (2008). This is consistent with the increased homogeneity of pulp duplicates relative to crush duplicates. Removal of all duplicate pairs with at least one sample below detection, as well as one poorly correlated outlier, improves correlation to ρ = 1.00 and CV_{AVR} to 22%.

Removal of duplicate pairs with outliers and samples below detection show the expected trend of improved duplication from core through crush to pulp duplicates (Table 12-2). CV_{AVR} for core and crush duplicate pairs fall within the "acceptable" values of Abzalov (2008), with CV_{AVR} for pulp duplicates just above the 20% "acceptable" limit.

Scatter plots showing the original and duplicate assays for core, crush and pulp duplicates are presented in Figure 12-3. Note that samples below detection and outliers have been removed from these scatter plots.

Figure 12-3: Scatter plots (original and duplicate assays) for Core, Crush and Pulp Duplicates



Source: Equity Exploration Consultants Ltd. (2022)

12.2 Database Validation based on Logged Lithological Intervals

The drill hole database includes over 259,500 downhole lithological intervals coded according to schemes current to the era of drilling as digitized during database construction. Given that the database contains drill holes spanning over 85 years of exploration and lithological coding schemes that have varied with time, some interpretation of historical codes was required prior to geological interpretation. Upon acquisition of the project, Pure Gold constructed a code equivalence table and recoded the database using the modern coding scheme, whilst also preserving historical codes. The recoded scheme was deemed useful and sufficiently consistent to accurately interpret the most important geological units, including the lower Confederation Assemblage contact, altered rocks, ultramafic sills and quartz porphyry intrusive rocks. Several minor changes to the recoding of the historical lithological codes have been made to refine the interpreted codes. The historical codes continue to be preserved in a separate database field.

12.3 Drill Hole Location and Survey Data

During 2014, Pure Gold completed a property-wide program to survey a selection of historical drill hole locations to improve confidence in using historical drill hole data. Location data were collected with a Trimble ProXRT differential GPS receiver with Omnistar real-time correction, which achieved sub-metre precision. In all, 221 historical collars were surveyed from across the property. Many Madsen Gold Corp. historical collars could not be located due to casing being removed.

As described in Section 9.2, Pure Gold completed a property-wide program to confirm the location of historical drill hole locations. Earlier surface and underground drill hole locations could not be confirmed but, in general, the database coordinates of underground drill hole collars are consistent with the geological units encountered in Pure Gold drilling and there is no evidence that any systematic shift or errors exist in the database. Importantly, this confirmation has largely been completed in UTM coordinate space, which tests the conversion from the historical local coordinate system (Imperial Mine Grid).

Conversely at the Russet deposit, most pre-1998 drill hole collars seem poorly located based on logged lithologies and could not be located or positively identified on the ground primarily due to lack of locatable suitable geo-reference points on the local grids. Since these could not be systematically surveyed and therefore verified, the early historical intercepts for Russet have not been used in the estimation of the resource, which has relied entirely on holes drilled by Pure Gold, Claude and Placer Dome.

Downhole survey methods for the earliest (mostly underground) drilling were rudimentary compared with modern gyroscopic survey methods. Historical drill logs indicate that a variety of methods were employed or in some cases, no downhole surveying was completed. Magnetic survey methods are problematic on the mine property owing to highly magnetic rock types (ultramafic and iron formation units) which prompted Claude and Pure Gold to implement gyroscopic downhole tool technology. As such, the locations of the downhole drill traces will have variable precision based on the era of drilling, the survey method used and the length of hole (longer holes deviate more). Given that most underground drilling – and particularly the closely-space resource definition holes – are short holes, deviation is expected to be small. The longer exploration holes tend to be more recent holes that are well-surveyed by modern gyroscopic tools. Pure Gold re-surveyed (downhole) many important holes drilled by previous operators that only had magnetic survey data. Underground mining since 2018 has exposed many exploration and resource definition drill holes and their known locations were generally in agreement with locations calculated from downhole surveys.

SRK is of the opinion that the collar locations for the historical drill hole database are well-compiled, have been translated accurately from mine grid coordinates and are adequate for mineral resource estimation. Downhole survey data exhibit variable precision in line with the technology used at the time of drilling, but most recent drill holes have been surveyed with high-precision gyroscopic tools.

12.4 Data Verification by SRK

The Madsen Mine drilling database is compiled from historical and modern work that spans over 80 years. Records of quality control or data handling procedures are poorly documented prior to 2009. Approximately 11 years of sporadic effort was required to capture and translate the available historical hard-copy records into a modern digital database (Cole et al., 2010). Use and verification of this database shows that it is of high quality, largely free of errors and highly effective even if assessment of the original data collection methods is not possible. Work by Pure Gold was conducted with clear data handling protocols and an industry-standard quality control program.

Mr. Cliff Revering conducted a site visit to the Madsen Mine from July 4 to July 7, 2022, during which time he reviewed select drill cores to review the dominant lithological domains and local mineralization controls and visited exposures of the mineralized system in underground development available at the time of the site visit. Mr. Cliff Revering also reviewed selected historical hard-copy records to compare against entries within the drill hole database and found no inconsistencies. He can confirm that the description of the geology, mineralization and mineralization controls, and the drilling, logging, sampling and data collection techniques described are consistent with observations made in the field during his site visit.

12.5 Metallurgy Data

In 2018, approximately 870 kg of drill core intervals from the Madsen deposit were collected by Pure Gold. A review of the location of the drill holes and intervals collected were plotted against the areas to be mined and were found to be spatially representative with sufficient variability in gold head grade. Intervals chosen for the 2018 test program were sent to BaseMet Laboratories in Kamloops, BC for test work in support of the 2019 Feasibility Study. The drill core intervals were used to create variability composites that represented the five different zones in the Madsen deposit and composites representing the Years -1, 1, 2, 3 and Years 4 to 7, based on the PEA mine schedule. In 2018 approximately 179 kg of drill core intervals from a number of drill holes at varying depths and gold head grades were collected to create composites representing three satellite deposits; Fork, Russett and Wedge. A preliminary investigation by BaseMet was completed to assess the metallurgical performance using the Madsen Mine flowsheet. For this study, the gold performance of the samples tested in 2018 were compared to the operational data collected by

Pure Gold from the restart of the mill in December 2020 to the end of 2021. The month end reports from this period and gold recoveries achieved from the zones processed through the plant were reviewed by Ms. Kelly McLeod, P.Eng. and have been further reviewed by Mr. Travis O’Farrell, P.Eng.

12.6 Qualified Person Comment on Data Verification

It was Kelly McLeod’s opinion that there is sufficient information to predict the potential gold recoveries for the Madsen deposit that will be mined in the future. This opinion has been further corroborated by Travis O’Farrell who also examined the metallurgy data and believes there is sufficient information to predict gold recoveries for the Madsen deposit.

Mr. Cliff Revering reviewed and analyzed the results of data compilation and verification programs conducted by previous companies and QPs and accepts the results of these programs. Based on this review and analysis, along with the additional data verification conducted directly by Mr. Cliff Revering, he is of the opinion that the Madsen Mine drill hole database is adequate to support the current geological interpretation of the Madsen Mine deposits and to support the estimation of mineral resources.

Ms. Sheila Ulansky reviewed the 2023 to 2024 Madsen Mine assays and accepts the results of these programs based on independent analyses and checks. Minor, non-material data issues will need to be addressed prior to including the 2023 and 2024 results into an updated mineral resource.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Historical metallurgical data is available from mill operations dating back to the 1951 Madsen Lake Gold Mines Limited annual report. Gold recovery percentages in the mid-90s were reported at the time. The mill operated for over 40 years with mill throughput ranging from 350 t/d to 850 t/d. In later years, recoveries in the mid-90s continued to be achieved. The present mill was purchased and relocated in the 1990s from Placer Dome's Dona Lake mine. The mill operated at a nominal rate of 600 t/d and used the carbon-in-pulp (CIP) process to recover gold. A 1998 mill report indicated average annual recovery of 90% at an average gold head grade of 4.2 g/t (Madsen Gold Corp.). The most recent test program, completed in 2018 in support of the 2019 Feasibility Study completed by JDS for Pure Gold, was carried out at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC. A full breakdown of the results for the test program can be found in the BaseMet 2018 report.

The following section pertains to the results used as the basis for the process design and recovery method presented in Section 17 of this report. This discussion will include a summary of the results from the BaseMet (2018) test program.

Based on the results from Base Met (2018) and 2021 plant operational data, gold doré with no significant levels of deleterious elements can be produced with a primary grind size of 80% passing (P80) 75 µm followed by gravity concentration, 2-hour pre-oxidation, 24-hour cyanide leach, 5-hour carbon-in-pulp (CIP) adsorption, desorption and refining process. Using the blended average recovery of the samples tested, based on the 2024 SRK mine plan, it is estimated that a LOM gold recovery of 95.7% can be achieved.

13.2 BaseMet (2018) Test Program Summary

The primary objective of the test program was to confirm the flowsheet and design criteria using the historical data and the existing plant design. Drill core was sent to BaseMet for test work that included sample preparation, interval assaying, gravity concentration, cyanide leach and bulk cyanide leaching to produce material for continuous cyanide destruction. Process optimization test work was also conducted to further the understanding and optimization of the processing characteristics in support of the 2019 Feasibility Study.

The test program was done in three phases: Variability Scoping Composites, Year Composites, and Variability Composites. The first phase was scoping variability tests on 12 composites from the five zones to evaluate the metallurgical response using the existing plant flowsheet and historical data. The second phase included test work on composites that at the time of the test program represented Years -1 to 1, 2 to 3 and 4 to 7 of the mine schedule. The final phase tested the optimized flowsheet using 30 variability composites representing the five zones of the deposit.

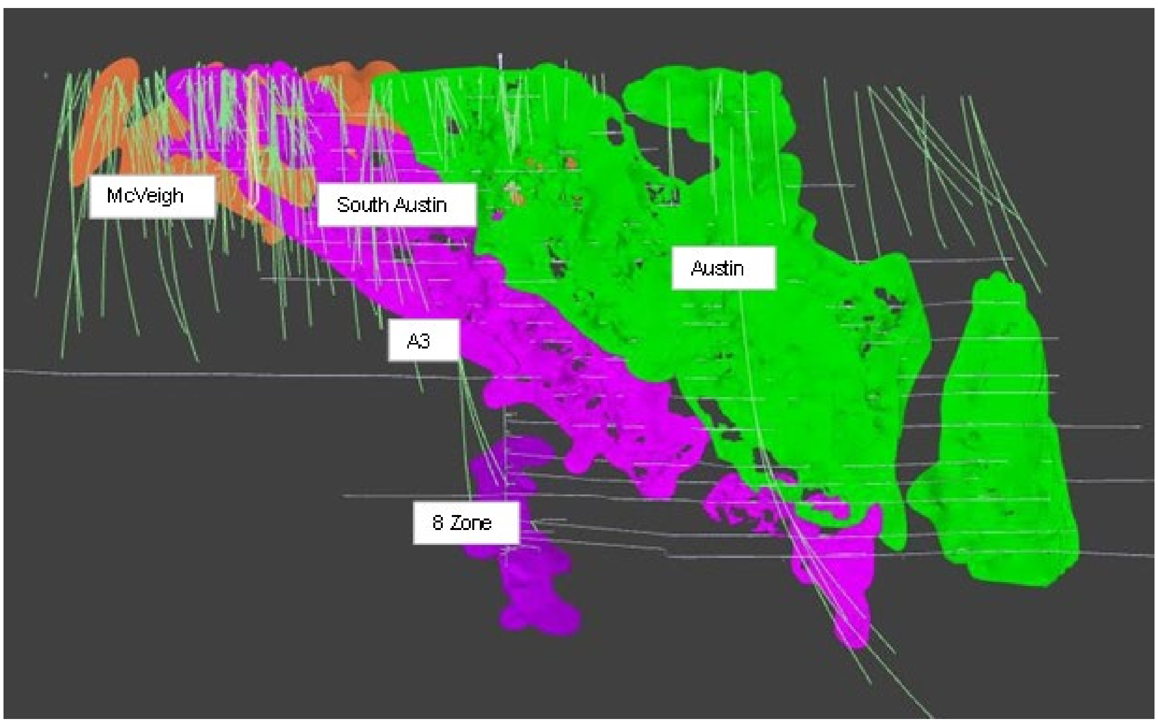
13.2.1 Sample Selection

All samples were received in February 2018 by BaseMet in two forms. Approximately 870 kg arrived as cut core (1/4 and 1/2 core), and about 14.9 kg arrived as coarse rock samples. In total, 286 individual core interval samples and 20 coarse rock samples were received. The drill core was collected from McVeigh, Austin, South Austin (including the A3 domain) and 8 Zones. Hangingwall and footwall intervals were included for dilution.

The drill core was initially inspected and weighed. Each interval was then individually stage crushed to a nominal 3.36 mm (6 mesh). The crushed material was blended and a 250 g sample was riffle split and pulverized for subsequent assaying. The split was packaged, labelled clearly and shipped to an alternative assay laboratory. Blanks and standards were inserted prior to shipment. In addition, screen metallics assays were also conducted for gold.

Figure 13-1 shows the location of the five zones in the Madsen deposit that were tested in the BaseMet program.

Figure 13-1: Longitudinal section through Madsen deposit showing resource domain wireframes (looking mine grid north)



Source: Pure Gold (2018)

13.2.2 Scoping Variability Composites

Twelve scoping variability composites representing Austin, South Austin (including A3 domain), McVeigh and 8 Zones were generated to create spatially representative samples based on mineralogy, grade and location in the deposit. The details of the construction of each composite are presented in Table 13-1.

Table 13-1: Scoping Variability Composite Construction

Composite	Variability Composites	Mass in Composite (kg)
McVeigh West Top	PGU-0004	1.41
	PGU-0018	1.17
	PGU-0024	2.95
	PGU-0033	2.5
	PG16-154	1.97
McVeigh West Middle	PG16-098	0.68
	PG16-112	0.99
	PG16-117	2.03
	PG16-121	0.77
	PG16-138	2.52
	PG16-153	1.08
	PG16-163	0.86
	PG16-185	1.06
McVeigh West Lower	PG16-253	3.31
	PGU-0044	6.69
McVeigh Central	PG16-054	0.86
	PG16-056	2.64
	PG16-058-A	2.76
	PG16-058-B	2.87
	PG16-060	0.87
Austin Central	PG16-055	2.64
	PG14-025	0.8
	PG16-054	2.6
	PG16-060	1.19
	PG16-126	0.75
	PG16-108	1.07
	PG16-262	0.94
Austin East	PG16-111	2.15
	PG16-114	2.16
	PG16-115	3.74
	PG16-227	1.95
Austin South Top	PGU-0035	0.5
	PGU-0037	0.37
	PG14-002	0.61
	PG16-148	1.91
	PG16-150	0.99
	PG16-151	1.34
	PG16-195	3.48

Composite	Variability Composites	Mass in Composite (kg)
	PG16-279	0.8
Austin South Middle	PG16-066	4.89
	PG16-203	2.74
	PG17-459	2.37
Austin South Deep	PG17-456	10.00
A3 Upper	PG16-231	3.97
	PG17-304	6.03
A3 Lower	PG16-282	3.1
	PG16-288	1.29
	PG17-320	5.6
8 Zone	MUG-12-20	3.96
	MUG-09-02B	0.78
	MUG-09-04	2.86
	MUG-12-26	2.41

Source: Base Met (2018)

Head Assays

The samples and head assays of the twelve composites are shown below in Table 13-2.

Table 13-2: Head assays of scoping variability composites

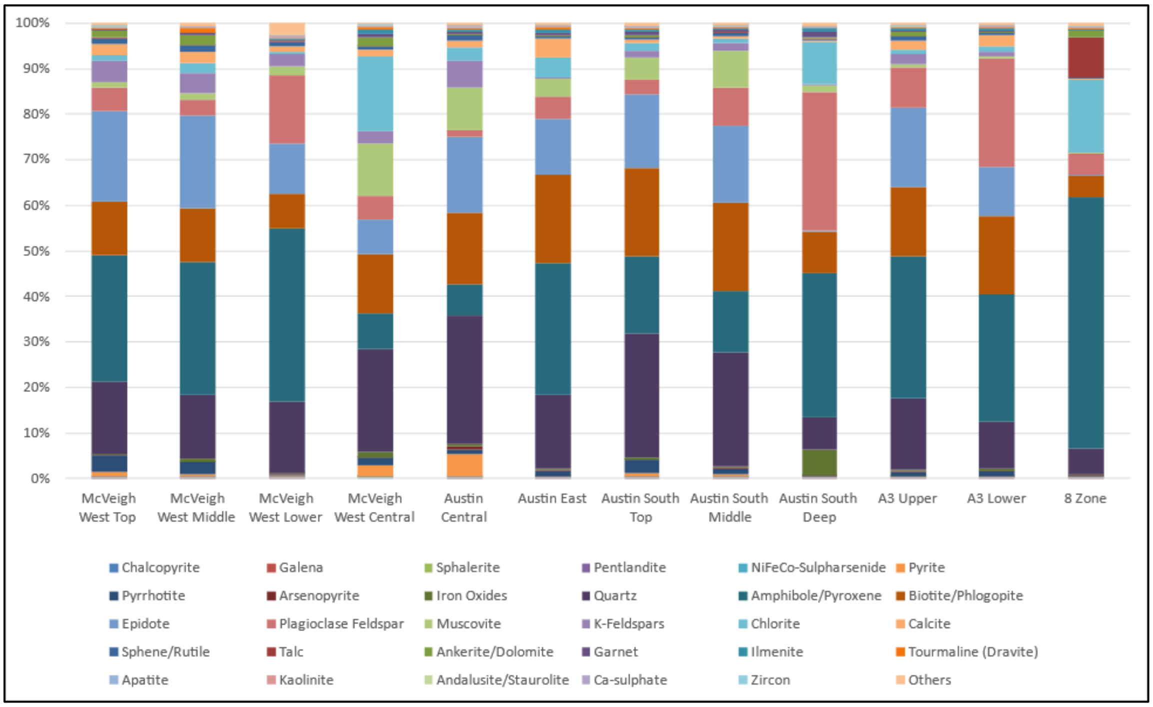
	Assay					
Composite	**Au (g/t)**	**Ag (g/t)**	**Fe (%)**	**S (%)**	**C (%)**	**As (g/t)**
McVeigh West Top	5.42	2	5.4	2.11	0.72	128
McVeigh West Middle	3.75	1	4.7	1.24	0.94	65
McVeigh West Lower	9.42	1	1.6	0.35	0.28	41
McVeigh Central	4.62	2	7.5	2.02	0.47	334
Austin Central	5.72	1	6.0	2.78	0.44	2,330
Austin East	2.53	<1	5.1	0.43	0.69	813
Austin South Top	15.3	<1	5.9	1.56	0.21	271
Austin South Middle	1.36	<1	4.5	0.80	0.15	920
Austin South Deep	9.6	<1	6.5	0.06	0.06	6
A3 Upper	15.3	2	4.1	0.56	0.52	26
A3 Lower	4.08	<1	3.8	0.53	0.44	201
8 Zone	40.2	<1	1.8	0.10	0.27	413

Source: Base Met (2018)

Mineralogy

The twelve scoping variability composites were sent out for un-sized Bulk Mineral Analysis (BMA) using QEMSCAN. The results show the un-sized estimation of the mineral composition. Figure 13-2 is a summary of the mineral content for each of the composites.

Figure 13-2: Mineral content - scoping variability composites



Source: BaseMet (2018)

As shown above the main minerals are Amphibole / Pyroxene and Quartz. The main sulphide minerals are Pyrrhotite and Pyrite, averaging 1.4% and 1.0% of the mass in each sample, respectively. Small amounts of Arsenopyrite were also observed.

Comminution Results

Crushing Work Index determination tests were conducted on selected coarse rock samples. Bond Ball Mill Work Index (BWi) and SAG Mill Comminution (SMC) test were carried out on the twelve scoping variability composites. The Crushing Work Index (CWi) for the sample was 13.2 kWh/t, indicating a moderately hard material.

BWi tests were conducted at a closing screen sizing of 106 microns (µm). The results ranged between 9.7 kWh/t to 17.1 kWh/t and averaged 14.4 kWh/t, indicating that the material is moderately hard. The SMC tests resulted in an average drop weight index (DWi) of 10.7 kWh/m^3. Similar to the BWi, the results indicate a moderately hard material with respect to SAG milling. Three abrasion index (Ai) tests were completed on McVeigh (0.256 g), Austin (0.282 g) and A3 (0.260 g). The samples were moderately abrasive with an average Ai of 0.266 g. The comminution results are shown below in Table 13-3.

Table 13-3: Comminution results - scoping variability composites

Sample ID	Size Fraction Tested (mm)	DWi kWh/m^3	DWi %	Mia kWh/t	Mih kWh/t	Mic kWh/t	A	b	sg	ta	SMC	F_{80} µm	P_{80} µm	Gpr	BWi kWh/t
McVeigh West Top	22.4 - 19.0	10.57	91	26.9	21.8	11.3	78.2	0.34	2.81	0.25	12.39	2380	78	1.34	14.2
McVeigh West Middle	22.4 - 19.0	10.58	91	25.5	20.6	10.7	71	0.40	2.97	0.25	12.51	2312	76	1.33	14.1
McVeigh West Lower	22.4 - 19.0	10.23	89	25.3	20.4	10.5	88.2	0.32	2.91	0.25	12.35	2377	78	1.52	12.8
McVeigh Central	22.4 - 19.0	12.63	98	29.6	24.8	12.8	46.1	0.51	2.95	0.21	13.79	2465	78	1.10	16.6
Austin South Top	22.4 - 19.0	10.62	91	26.5	21.5	11.1	60.7	0.44	2.87	0.24	12.58	2316	77	1.31	14.4
Austin South Middle	22.4 - 19.0	11.14	93	27.7	22.7	11.7	72.5	0.35	2.85	0.23	12.85	2466	73	1.28	14.2
Austin South Deep	22.4 - 19.0	10.18	88	25.1	20.2	10.5	48.2	0.60	2.92	0.26	12.23	2431	77	1.15	16.0
Austin East	22.4 - 19.0	12.47	97	28.8	24.2	12.5	66.3	0.36	2.99	0.21	13.83	2348	75	1.19	15.3
Austin Central	22.4 - 19.0	12.48	97	29	24.3	12.6	61.6	0.39	2.98	0.21	13.74	2399	77	1.06	17.1
A3 Upper	22.4 - 19.0	11.00	93	26.6	21.7	11.2	61.6	0.43	2.94	0.23	12.89	2475	72	1.07	16.2
A3 Lower	22.4 - 19.0	9.85	86	25.2	20.1	10.4	57.5	0.50	2.84	0.26	12.00	2383	73	1.46	12.8
8 Zone	22.4 - 19.0	6.96	54	18.5	13.8	7.1	43.9	0.96	2.93	0.37	10.14	2162	81	2.23	9.7

Source: BaseMet (2018)

Baseline Flowsheet Evaluation – Gravity Leach Results

The scoping variability composites were used to evaluate the response of the samples to the baseline flowsheet using the existing plant design and operating parameters as a starting point for grind size, leach time and reagent dosage.

The samples were prepared to a nominal particle size of 80% passing (P80) of 75 µm, passed through a Knelson gravity concentrator followed by hand panning and leaching of the gravity tailings. A total of 20 gravity and cyanide leach tests were completed. The conditions were varied to see the response at various pH levels, with pre-oxidation and addition of lead nitrate.

The baseline leach parameters included a primary P80 of 75 µm, 10.5 pH, oxygen sparging to maintain the dissolved oxygen (DO) at greater than 20 mg/L, and 1000 ppm NaCN. Leaching test work was carried out on gravity tailings. All tests were completed in closed bottles on rolls, allowing constant agitation of the pulp as the sample leached for 48 hours. Cyanide levels, dissolved oxygen (DO), and pH were monitored and controlled throughout each test. Kinetic sampling was done at 2, 6, 24 and 48 hours.

The gold extraction at time 0 indicates the percent gold reporting to the Knelson concentrator. The highest gravity recovery was seen for 8 Zone at 82%. The gold extraction for all tests ranged between 86% to 99% after 24 hours, the average cyanide consumption was 0.7 kg/t and the average lime consumption was 0.2 kg/t. The optimization test work focused on the effect of pH, pre-oxidation, lead nitrate (LN) addition and primary grind size on gold recovery and leach kinetics. A two-hour pre-oxidation showed significant improvement to the leach kinetics and lower cyanide consumption, but overall extraction was similar to the baseline results. Further optimization test work was completed on the year composites in Phase 2 of the program.

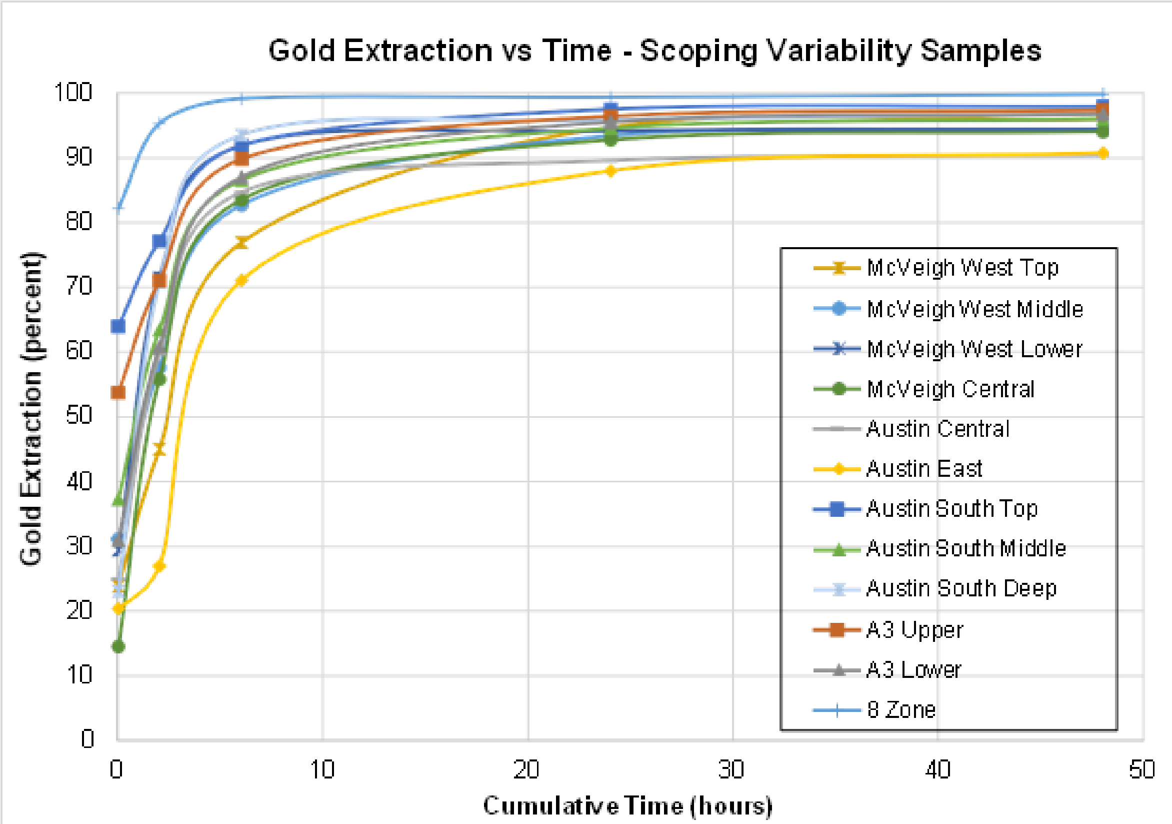
The results are summarized in Table 13-4, and gold extraction versus time using the baseline conditions for the 12 composites are illustrated in Figure 13-3.

Table 13-4: Bottle roll leach results (BaseMet 2018)

Sample ID	Test No.	Conditions	Gold Extraction - percent cumulative					CN Tls Grade (Au g/t)	Consumption (kg/t)		Recalc Head (Au g/t)	Assay Head (Au g/t)	Final 48hr Solution Values - ppm		
			0	2	6	24	48		NaCN	Lime			Cu	Fe	As
McVeigh West Top	1	BL	23.9	45.0	77.0	94.8	96.0	0.24	0.99	0.25	5.91	6.15	31.8	50.1	0.7
McVeigh West Middle	2	BL	31.1	57.6	82.8	93.4	94.3	0.20	0.66	0.25	3.47	3.82	24.7	34.0	0.1
	19	2hr Pre-Ox	41.0	75.5	90.5	94.3	94.4	0.27	0.52	0.23	4.78	3.82	15.7	17.1	0.3
	20	LN, pH 10	36.3	75.7	91.5	94.4	94.9	0.28	1.06	0.00	5.46	3.82	23.9	57.0	0.1
McVeigh West Lower	3	BL	29.4	71.5	92.0	94.2	94.5	0.31	0.31	0.17	5.49	9.01	16.0	16.5	0.4
McVeigh Central	4	BL	14.5	55.8	83.6	92.8	94.1	0.36	1.02	0.35	5.98	4.55	58.0	42.3	0.9
	13	2hr Pre-Ox	9.5	79.5	87.4	90.5	92.2	0.36	0.83	0.40	4.51	4.55	41.0	24.0	0.8
	14	LN, pH 10	8.9	85.3	88.9	91.7	92.8	0.31	1.44	0.00	4.21	4.55	60.8	70.0	0.2
Austin Central	5	BL	25.0	59.8	84.7	89.6	90.5	0.56	0.98	0.32	5.85	5.44	10.6	56.0	15.6
	15	2hr Pre-Ox	26.8	78.8	90.7	92.6	94.0	0.36	0.84	0.49	5.87	5.44	22.3	33.3	11.8
	16	LN, pH 10	28.2	81.2	89.5	93.6	93.7	0.38	0.85	0.00	5.98	5.44	23.2	78.5	8.5
Austin East	6	BL	20.4	27.0	71.1	88.0	90.8	0.23	0.52	0.29	2.43	2.62	11.4	23.9	11.1
	17	2hr Pre-Ox	20.9	72.8	81.9	86.4	86.8	0.48	0.50	0.24	3.64	2.62	11.2	20.7	20.0
	18	LN, pH 10	18.8	79.6	84.6	87.9	88.0	0.44	0.73	0.00	3.61	2.62	15.3	42.0	6.7
Austin South Top	7	BL	64.0	77.2	92.0	97.6	98.0	0.30	0.70	0.31	15.0	17.7	21.4	32.8	1.6
Austin South Middle	8	BL	37.3	63.5	86.6	94.6	96.0	0.09	0.58	0.19	2.26	1.41	11.5	23.5	9.1
Austin South Deep	9	BL	23.1	70.7	93.6	95.9	96.8	0.18	0.64	0.23	5.60	9.14	7.9	22.3	<0.1
A3 Upper	10	BL	53.8	71.1	89.9	96.5	97.4	0.40	0.43	0.19	15.4	20.8	13.9	20.4	<0.1
A3 Lower	11	BL	31.0	60.9	87.1	95.6	96.8	0.13	0.35	0.18	4.16	3.86	9.5	11.5	0.9
8 Zone	12	BL	82.3	95.4	99.2	99.4	99.9	0.01	0.35	0.21	7.10	30.5	5.3	2.7	22.9

Source: BaseMet (2018)

Figure 13-3: Effect of grind size on gold extraction



Source: BaseMet (2018)

13.2.3 Year Composites

The year composites were constructed from the scoping variability samples to represent the material expected to be mined in Years -1 to 1, Years 2 to 3 and Years 4 to 7. Composite construction details are presented in Table 13-5.

The PEA mine schedule at the time was used as a guide for target grade and ratio of material from each of the five zones to be mined. Test parameters from Phase 1 on scoping variability composites were used as the baseline conditions for Phase 2 on the year composites with the addition of a 2-hour pre-oxidation. The test work investigated the metallurgical response with varying parameters such as grind size, gravity concentration, cyanide concentration, lead nitrate addition, pH, pulp density and carbon-in-pulp (CIP). Most of the optimization test work was carried out on year composites 1 and 2 due to the limited sample size of year composite 3.

Table 13-5: Year composite construction details

Composite	Scoping Variability Composites	Mass in Composite (kg)	Percentage of Composite
Year -1 to 1	McVeigh West Top	45.9	12%
	McVeigh West Middle	84.0	21%
	McVeigh Central	55.9	14%
	Austin Central	87.00	22%
	Austin South Top	121.7	31%
Year 2 to 3	McVeigh West Middle	72.5	53%
	Austin Central	18.3	13%
	Austin South Top	46.7	34%
Year 4 to 7	McVeigh West Top	45.3	15%
	McVeigh West Middle	71.9	24%
	Austin Central	12.4	4%
	Austin South Top	165.7	55%
	8 Zone	7.3	2%

Source: BaseMet (2018)

Head Assays

The year composites representing Years -1 to 1, Years 2 to 3 and Years 4 to 7 were generated to represent the material expected to be mined as indicated in the PEA mine schedule at the time. The samples and head assays are shown below in Table 13-6.

Table 13-6: Head assays of year composites

Composite	Assay					
	Au (g/t)	**Ag (g/t)**	**Fe (%)**	**S (%)**	**C (%)**	**As (g/t)**
Year Composite 1	7.24	2	6.2	1.86	0.62	339
Year Composite 2	5.94	2	6.7	2.21	0.38	1,285
Year Composite 3	10.2	2	5.4	1.19	0.36	1,064

Source: BaseMet (2018)

Mineralogy

The year composites were sent out for Un-sized Bulk Mineral Analysis (BMA) using QEMSCAN. Table 13-7 summarizes the mineral content for each of the composites.

Similar to the scoping variability composites, the main minerals were amphibole/pyroxene, quartz, biotite and phlogopite. The main sulphide minerals were pyrrhotite and pyrite.

Table 13-7: Mineralogy of year composites

Minerals	Year Composite 1	Year Composite 2	Year Composite 3
Chalcopyrite	<0.1	<0.1	<0.1
Sphalerite	<0.1	<0.1	<0.1
Pyrite	1.8	2.8	1.8
Pyrrhotite	2.2	1.3	0.8
Arsenopyrite	0.2	0.2	0.3
Iron Oxides	1.0	0.5	0.5
Ilmenite	0.2	0.3	0.4
Quartz	17.4	24.6	16.9
Amphibole / Pyroxene	23.8	14.9	29.4
Biotite / Phlogopite	15.7	19.6	13.9
Epidote	16.2	15.7	8.7
Plagioclase Feldspar	3.6	2.3	2.9
K-Feldspars	3.5	3.5	2.2
Chlorite	4.9	4.8	9.9
Muscovite	2.6	5.1	3.0
Andradite	0.6	0.6	0.3
Talc	0.5	0.7	6.0
Calcite	2.1	1.5	1.1
Dolomite / Ankerite	1.2	0.2	0.7
Rutile / Anatase	1.2	0.8	0.5
Apatite	0.1	0.2	0.2
Others	1.0	0.3	0.4
Total	100	100	100

Source: BaseMet (2018)

Comminution Results

BWi tests were conducted on each year composite at a closing screen sizing of 106 microns (µm). The results indicated the material is moderately hard for the first four years and moderately soft in the remaining years. The results were 14.6 kWh/t, 14.8 kWh/t and 13.0 kWh/t for year composites 1, 2 and 3, respectively.

Gravity Recoverable Gold (GRG) Test Work

The year composites were each evaluated for gravity recoverable gold (GRG). The samples were tested at a nominal size of 850 µm and run through three Knelson gravity concentration tests, each with a grinding step down to P80 of 53 µm. Gold recovery ranged between 74% and 90% with gold grade ranging between 79 g/t and 105 g/t. The test parameters were evaluated, and it was recommended to include two concentrators to handle the total feed to the mill in the existing grinding circuit, followed by intensive leaching of the concentrate.

Flowsheet Optimization

The year composites were evaluated over a range of parameters. The first few tests included the baseline conditions at primary grind sizes of 53 µm, 75 µm and 106 µm. In addition to varying grind size, cyanide concentration, lead nitrate, pH, pulp density, air versus oxygen sparging and CIL were investigated.

Primary Grind

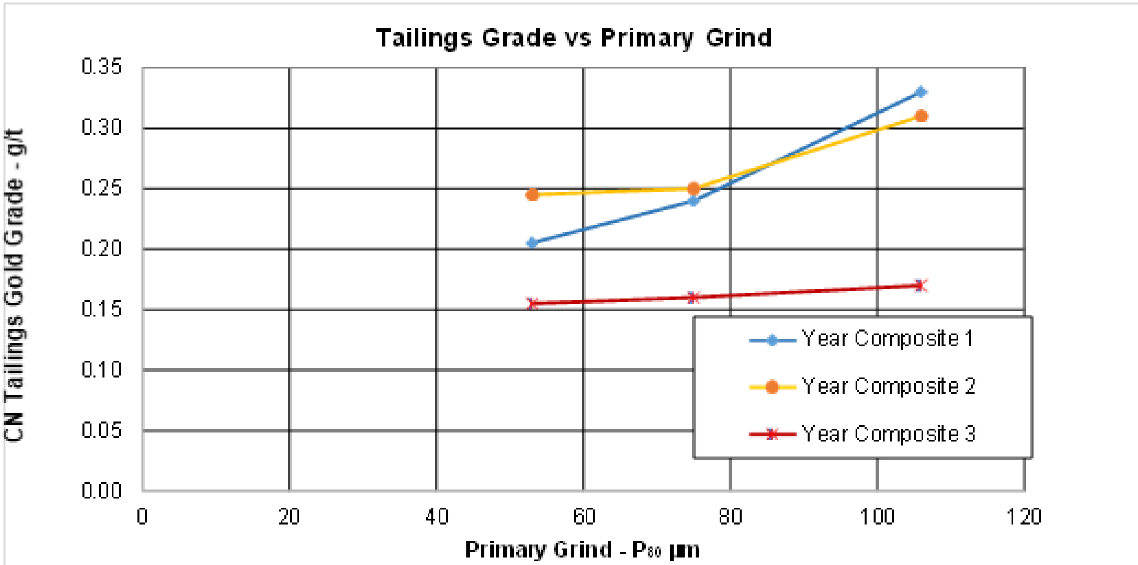
At a grind size above 75 µm, an increase in the tailings grade was observed. There was no correlation observed between grind size and gold extraction, although tailings grade appeared to increase for year composites 1 and 2 at grind sizes of 80 µm and above. The results are summarized in Table 13-8 and illustrated in Figure 13-4 below. Based on these results, a primary grind size of 75 µm was used in subsequent tests.

Table 13-8: Gold extraction versus primary grind size

Composite	Test	Primary Grind (µm)	Consumption (kg/t)		Recovery (%)		Tails Grade (g/t)	Recalc Head (g/t)	Final 48hr Solution Values (ppm)		
			NaCN	Lime	24 h	48 h			Cu	Fe	As
Year Composite 1	21	106	0.39	0.13	94.0	95.3	0.33	6.9	16.9	8.9	1.0
	22	75	0.46	0.13	96.1	96.8	0.24	7.4	20.0	13.7	0.9
	23	53	0.56	0.11	95.2	96.1	0.21	5.3	34.0	15.5	1.6
Year Composite 2	24	106	0.48	0.18	97.9	98.3	0.31	18.6	15.6	17.9	5.6
	25	75	0.55	0.18	95.8	96.5	0.25	7.1	16.2	19.0	5.2
	26	53	0.67	0.22	98.2	98.7	0.25	18.4	26.8	21.9	5.7
Year Composite 3	27	106	0.54	0.27	97.7	98.2	0.17	9.3	11.6	12.0	9.0
	28	75	0.56	0.25	95.1	98.5	0.16	10.4	10.6	13.4	10.0
	29	53	0.64	0.27	96.6	98.2	0.16	8.6	12.2	17.0	9.4

Source: BaseMet (2018)

Figure 13-4: Effect of grind size on gold extraction



Source: BaseMet (2018)

Cyanide Concentration and Lead Nitrate Addition

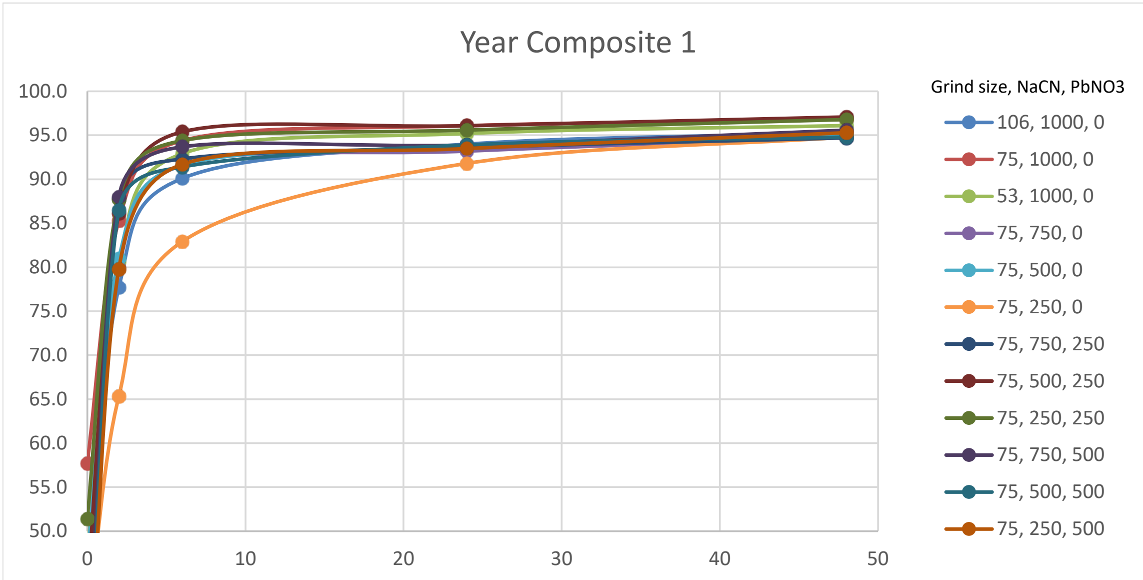
Tests were conducted at varying cyanide concentrations ranging from 250 ppm to 1000 ppm and lead nitrate addition between 0 ppm to 500 ppm. The higher cyanide and lead nitrate additions did not significantly improve gold extraction, but the tests conducted without lead nitrate did not perform as well as those with lead nitrate. The pH, pulp density, air sparging and CIL did not seem to have a significant effect on gold performance.

The final set of tests looked at gold extraction at a primary grind size of 75 µm with varying cyanide dosages and without gravity concentration. Gold extraction was not improved with higher cyanide dosage. The overall extraction was between 96% and 97%, and the leach kinetics slowed without the gravity stage.

A review of the results shows that tests 39 and 48 produced the best results for overall gold extraction. The optimized test conditions included a primary grind of 75 µm, gravity concentration, 2-hour pre-oxidation, 500 ppm NaCN, and 250 ppm lead nitrate.

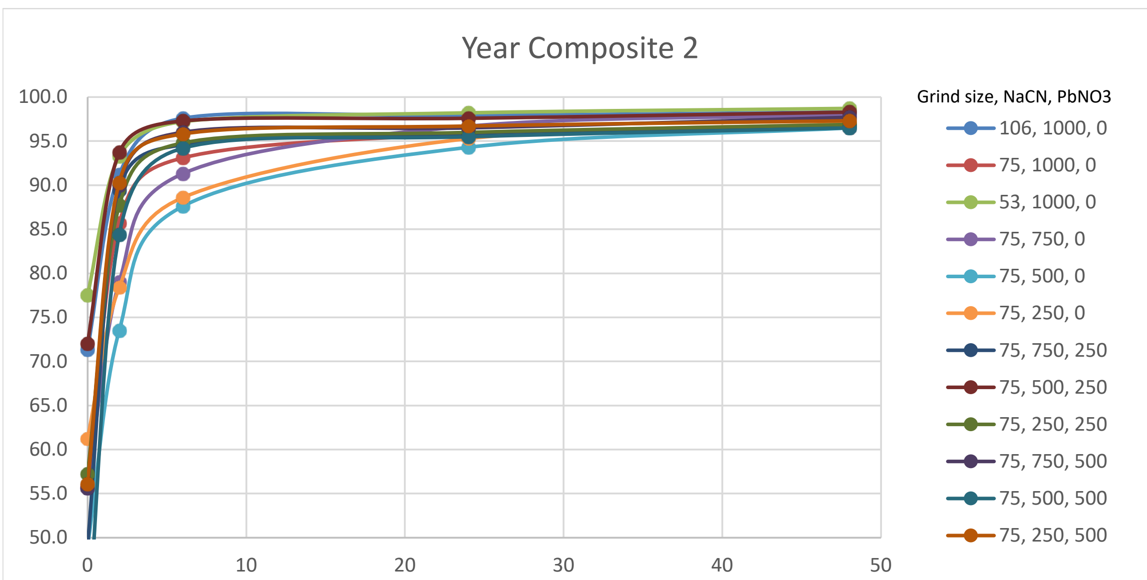
The results are illustrated in Figure 13-5 to Figure 13-7.

Figure 13-5: Gold extraction versus time (Year composite 1)



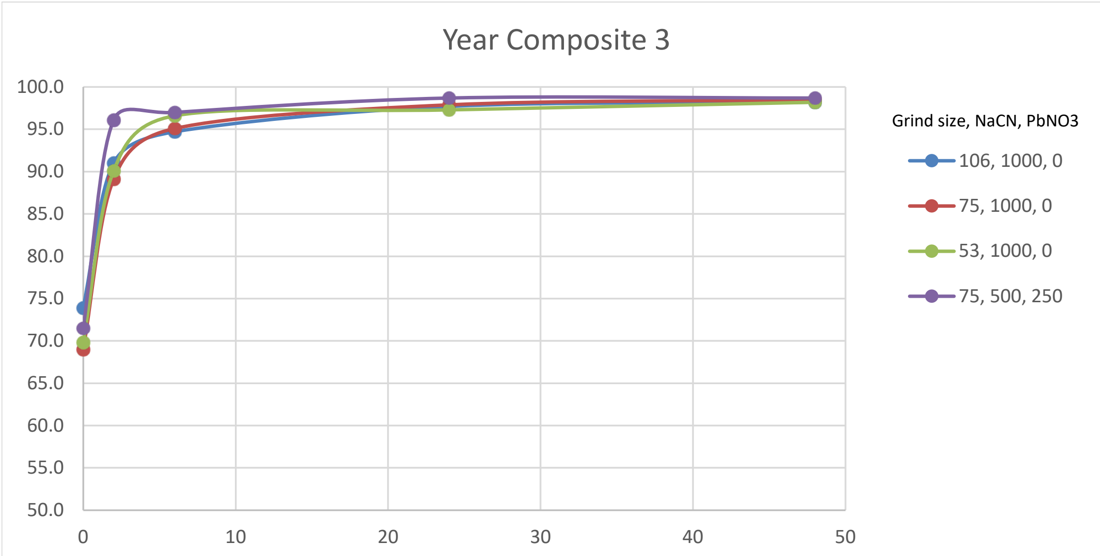
Source: BaseMet (2018)

Figure 13-6: Gold extraction versus time (Year composite 2)



Source: BaseMet (2018)

Figure 13-7: Gold extraction versus time (Year composite 3)

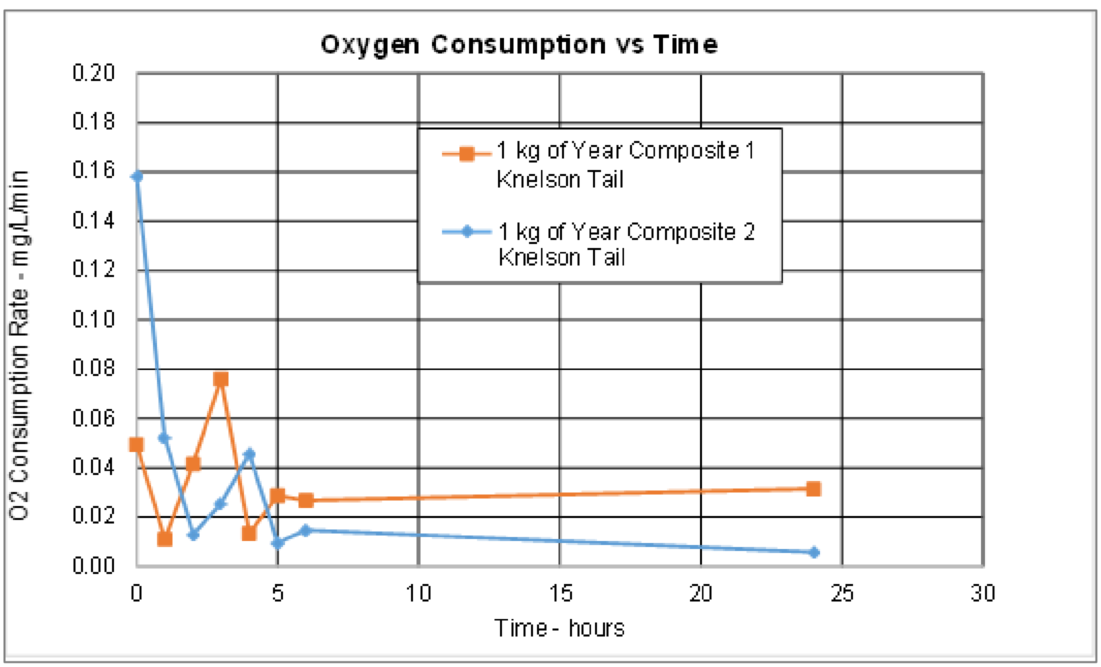


Source: BaseMet (2018)

Oxygen Uptake Test Work

Subsamples of year composites 1 and 2 were used for the oxygen uptake evaluations. Test 80 and 81 utilized the G.M. Fraser procedure and optimized test conditions. Over a 24-hour period, dissolved oxygen measurements were taken at one minute intervals for the first 15 minutes of each hour for hours 0-6 and then at 24 hours. The consumption was based on maintaining 20 mg/L dissolved oxygen. Figure 13-8 shows that most of the oxygen demand is in the first five hours.

Figure 13-8: Oxygen uptake rate results



Source: BaseMet (2018)

Carbon Adsorption Test Work

To determine gold loading rates and constants for a specific carbon type, equilibrium carbon loading (Table 13-9) and sequential triple contact carbon-in-pulp tests (Table 13-10) were conducted using pregnant leach slurry from tests 74 (year composite 1) and 77 (year composite 2). The results indicated that carbon loading of approximately 4,500 g/t gold can be achieved at a carbon concentration of 25 g/L.

Table 13-9: Equilibrium carbon loading results

Test	Sample	Equilibrium Loading at Au in Solution	
		g Au / tonne Carbon	Au in Solution (g/tonne)
74	Year Composite 1	4,369	1.00
		3,223	0.50
		2,156	0.20
77	Year Composite 2	5,052	1.00
		3,650	0.50
		2,376	0.20

Source: BaseMet (2018)

Table 13-10: Sequential triple contact carbon-in-pulp results

Test	Sample	Fleming Kinetic Constants		
		Feed Solution Au (ppm)	k - hr^{-1}	n
75	Year Composite 1	1.83	53.8	0.98
78	Year Composite 2	5.99	71.4	0.79

Source: BaseMet (2018)

Cyanide Destruction Test Results

Feed for the cyanide destruction test work was created from year composite 1 and year composite 2. Cyanide detoxification of the leach tailings slurry was done using the SO_2/air method. The produced pulp from year composite 1 contained 134 mg/L weak acid dissociable cyanide (CN_{WAD}), 10.9 mg/L Fe, and 2.2 mg/L Zn. This sample was used to determine the optimized conditions, then those conditions were applied to year composite 2.

Continuous cyanide destruction test work was completed using the SO_2/air process to produce a treated product with less than 5 mg/L CN_{WAD}. A series of continuous cyanide destruction tests were then completed to establish the cyanide destruction circuit design criteria and understand the effect of reagent dosage on the oxidation of cyanide using year composite 1.

The cyanide pulp produced during the test program responded well to the SO_2/air cyanide destruction process, producing a treated pulp with less than 5 mg/L CNWAD when a $SO_2:CN_{WAD}$ ratio of 5:1 was used with 30 mg/L copper sulphate added as a catalyst and a reactor retention time of 60 minutes. The results are shown in Table 13-11. The conditions used in test C5 were incorporated into the process design for the cyanide destruction circuit.

Table 13-11: Cyanide destruction test results

Sample ID	Test No.	Test Parameters				Final Solution Assays				
		pH	Retention Time (min)	SO_2 g./g CN_{WAD}	Cu (mg/L)	CN_{WAD} (ppm)	Cu (ppm)	Fe (ppm)	Ni (ppm)	Zn (ppm)
Year Comp 1	Feed	10.5	-	-	-	134.0	29.8	10.9	3.18	2.2
	C1	8.7	92.8	4	15	85.5	-	-	-	-
	C2	8.5	90.8	6	15	9.0	1.80	0.3	0.2	<0.01
	C3	8.5	88.2	6	30	4.0	0.76	0.2	0.04	<0.01
	C4	8.6	89.1	5	30	0.41	0.06	0.3	0.02	<0.01
	C5	8.5	60.0	5	30	4.7	9.24	1.51	0.08	<0.01
Year Comp 2	Feed	10.4	-	-	-	200	40.6	44.9	4.51	2.92
	C1	8.35	56.3	5	30	0.29	0.46	1.6	<0.1	<0.1

Source: BaseMet (2018)

13.2.4 Variability Testing

A total of 48 variability samples (VC) were constructed from McVeigh, Austin, South Austin (including A3 domain) and 8 Zones. Due to low sample mass, only 30 of the variability composites were subject to testing in this program. The composites were tested using the optimized test conditions to confirm metallurgical response. A breakdown of the construction of the variability composites is presented in Table 13-12.

Table 13-12: Variability composite construction

Variability Composites	Composite ID	Variability Composites	Composite ID
PGU-0004	VC-1	PG16-262	VC-27
PGU-0018	VC-2	PG16-111	VC-28
PGU-0024	VC-3	PG16-114	VC-29
PGU-0033	VC-4	PG16-115	VC-30
PG16-154	VC-5	PG16-227	VC-31
PG16-098	VC-6	PGU-0035	VC-32
PG16-112	VC-7	PGU-0037	VC-33
PG16-117	VC-8	PG14-002	VC-34
PG16-121	VC-9	PG16-148	VC-35
PG16-138	VC-10	PG16-150	VC-36

Variability Composites	Composite ID	Variability Composites	Composite ID
PG16-153	VC-11	PG16-151	VC-37
PG16-163	VC-12	PG16-195	VC-38
PG16-185	VC-13	PG16-279	VC-39
PG16-253	VC-14	PG16-066	VC-40
PGU-0044	VC-15	PG16-203	VC-41
PG16-054	VC-16	PG17-459	VC-42
PG16-056	VC-17	PG17-456	VC-43
PG16-058-A	VC-18	PG16-231	VC-44
PG16-058-B	VC-19	PG17-304	VC-45
PG16-060	VC-20	PG16-282	VC-46
PG16-055	VC-21	PG16-288	VC-47
PG14-025	VC-22	PG17-320	VC-48
PG16-054	VC-23	MUG-12-20	VC-49
PG16-060	VC-24	MUG-09-02B	VC-50
PG16-126	VC-25	MUG-09-04	VC-51
PG16-108	VC-26	MUG-12-26	VC-52

Source: BaseMet (2018)

Head Assays

The samples and head assays are shown in Table 13-13.

Table 13-13: Variability composite head assays

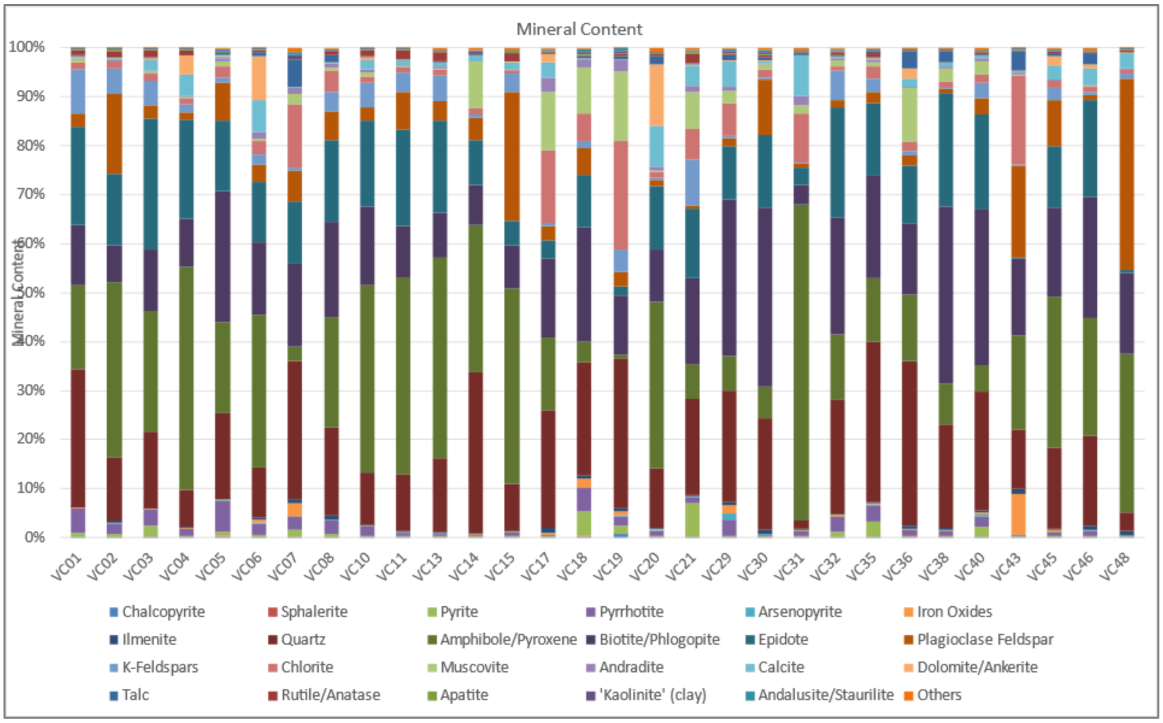
	Assays				
	Au (g/t)	**Ag (g/t)**	**Fe (%)**	**S (%)**	**C (%)**
VC 1	14.42	5	5.70	2.66	0.08
VC 2	2.61	1	3.00	1.29	0.12
VC 3	3.77	2	5.55	2.56	0.58
VC 4	2.98	2	3.01	0.74	1.38
VC 5	1.62	2	9.40	2.89	0.08
VC 6	4.74	2	4.55	1.20	4.83
VC 7	12.03	2	42.25	2.12	0.04
VC 8	3.55	1	6.15	1.40	0.10
VC 10	3.55	<1	4.50	0.81	0.59
VC 11	2.01	4	2.32	0.38	0.26
VC 13	4.19	5	1.48	0.20	0.22
VC 14	5.98	3	1.31	0.06	0.19
VC 15	7.42	2	1.75	0.43	0.31
VC 17	5.18	4	5.85	0.14	0.70
VC 18	2.17	1	10.55	4.64	0.08
VC 19	2.10	<1	8.20	1.97	0.02
VC 20	0.81	<1	3.71	0.56	2.70
VC 21	11.06	4	6.70	4.17	0.78
VC 29	7.27	<1	9.40	1.68	0.67
VC 30	3.48	<1	6.05	0.20	0.10
VC 31	6.22	<1	3.35	0.35	1.22
VC 32	4.74	4	6.35	1.76	0.08
VC 35	28.94	1	7.55	3.10	0.07
VC 36	2.59	2	3.63	0.59	0.32
VC 38	0.08	2	6.15	0.40	0.13
VC 40	3.44	1	6.75	1.98	0.12
VC 43	10.84	1	11.85	0.05	0.10
VC 45	28.80	3	4.41	0.52	0.77
VC 46	4.29	1	4.85	0.59	0.49
VC 48	3.59	1	2.99	0.08	0.63

Source: BaseMet (2018)

Mineralogy

The variability composites were sent out for Bulk Mineral Analysis (BMA) using QEMSCAN. The results provide mineral composition. Figure 13-9 is a summary of the mineral content for each of the composites. The main minerals are Amphibole / Pyroxene and Quartz, and the main sulphides are pyrrhotite and pyrite with trace amounts of arsenopyrite. All samples contained less than 10 percent sulphide minerals.

Figure 13-9: Variability composite mineralogy



Source: BaseMet (2018)

Comminution Results

BWi tests were conducted at a closing screen sizing of 106 µm. The results ranged between 9.5 kWh/t to 17.1 kWh/t and averaged 13.8 kWh/t, indicating that the material is moderately hard. Due to insufficient mass, bond ball mill work index testing was only carried out on seventeen of the samples. The results are shown in Table 13-14.

Table 13-14: Variability composite comminution results

Zone	Sample ID	Variability ID	F80 (µm)	P80 (µm)	Gpr	Bwi (kWh.t)
McVeigh West Top	PGU-0004	1	1951	80	1.50	13.5
	PGU-0024	3	1989	80	1.44	13.9
	PGU-0033	4	1726	80	1.30	15.4
	PG16-154	5	2123	76	1.31	14.5
McVeigh West Middle	PG16-112	7	1841	81	1.18	16.7
	PG16-117	8	1877	80	1.41	14.2
	PG16-138	10	1926	76	1.42	13.7
	PG16-153	11	1899	75	1.44	13.5
	PG16-185	13	2098	80	1.43	13.9
McVeigh West Lower	PGU-0044	15	1908	80	1.95	10.9
McVeigh Central	PG16-056	17	1915	76	1.40	13.9
	PG16-058-A	18	1834	65	1.21	14.3
	PG16-058-B	19	1816	80	1.14	17.1
Austin South Top	PG16-148	35	1924	80	1.46	13.9
	PG16-195	38	2110	64	1.38	12.4
Austin South Deep	PG17-456	43	2148	80	1.49	13.4
A3 Lower	PG17-320	48	1878	69	2.05	9.5

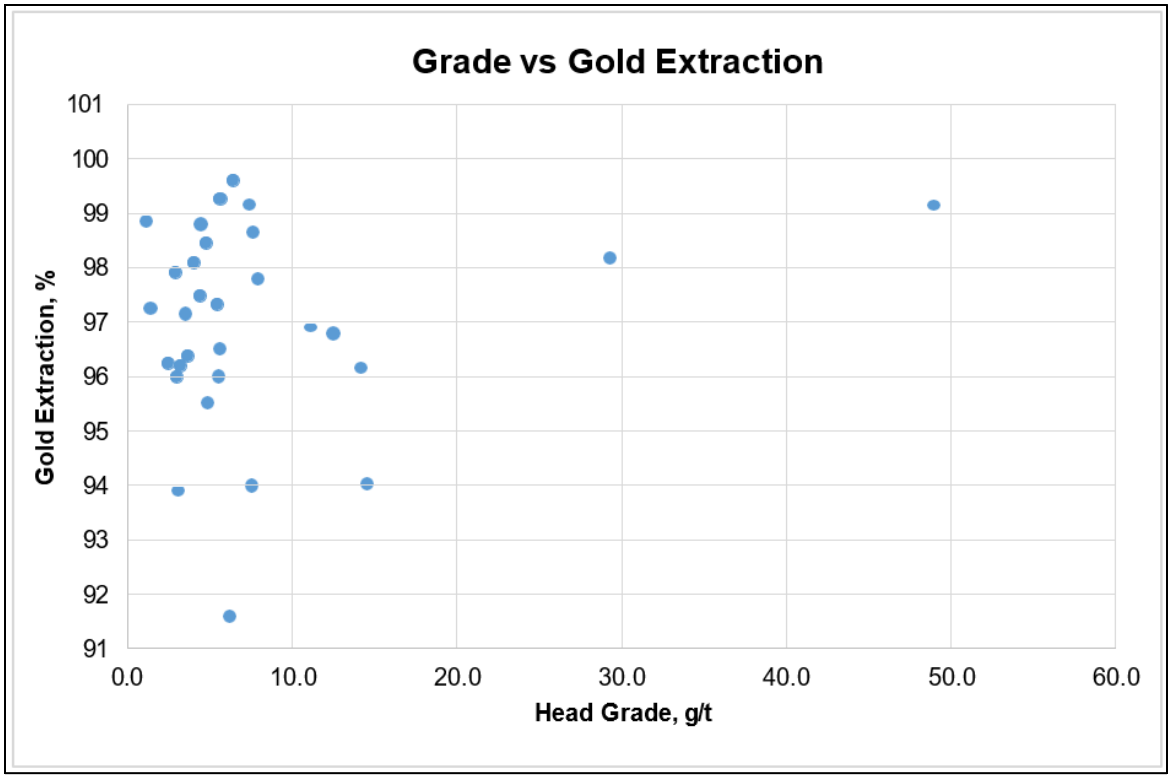
Source: BaseMet (2018)

Variability Gravity Leach Tests

The variability samples were prepared to a primary P80 of 75 µm and passed through a Knelson concentrator. The gravity tailings were treated for 2 hours with oxygen and then leached at a 10.5 pH, with oxygen sparging to maintain the dissolved oxygen (DO) greater than 20 mg/L, 500 ppm NaCN and 250 g/t lead nitrate. All tests were completed in closed bottles on rolls, allowing constant agitation of the pulp as the sample leached for 48 hours. Kinetic sampling was done at 2, 6, 24 and 48 hours.

The average gold extraction for the 30 variability samples was 95.4% after 24 hours, and the average recovery from the feed to the gravity concentrate was 45.7%. The NaCN and lime consumptions averaged 0.24 g/t and 0.42 g/t after 24 hours, respectively. No correlation was noted between head grade and gold extraction as shown in Figure 13-10. The gravity leach results are summarized in Table 13-15.

Figure 13-10: Variability composite – head grade versus gold extraction



Source: BaseMet (2018)

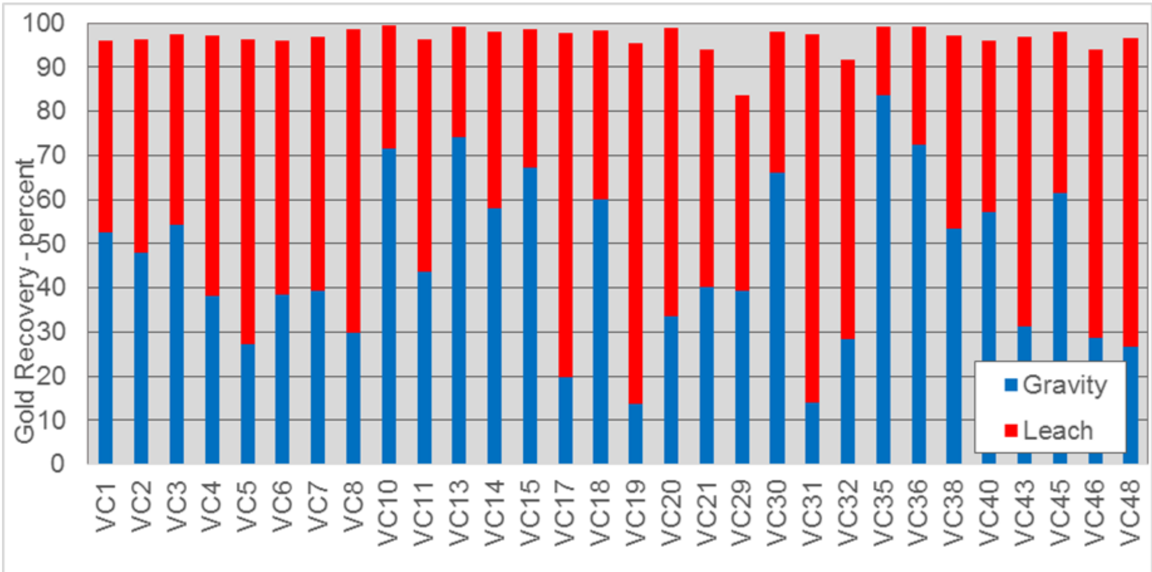
Table 13-15: Variability composite – gravity leach results

Zone	Variability Composite	ID	Test	Gravity Rec'y	Gold Extraction		Silver Extraction		Consumption (kg/t)			Recalc. Head Grade (g/t)	
				%	**24 h**	**48 h**	**24 h**	**48hr**	**NaCN (24 h)**	**NaCN (48 h)**	**Lime**	**Au**	**Ag**
McVeigh West Top	PGU-0004	VC1	82	52.5	96.2	96.1	51.5	54.0	0.27	0.31	0.35	14.1	5.10
	PGU-0018	VC2	83	48.0	96.6	96.4	22.7	22.9	0.11	0.17	0.30	3.58	2.10
	PGU-0024	VC3	84	54.4	98.0	97.5	58.0	58.5	0.12	0.28	0.36	4.35	1.90
	PGU-0033	VC4	85	38.2	93.4	97.2	19.5	18.6	0.18	0.28	0.28	3.50	1.50
	PG16-154	VC5	86	27.1	92.2	96.3	53.2	56.5	0.31	0.50	0.44	2.39	1.10
McVeigh West Middle	PG16-098	VC6	87	38.4	97.3	96.0	49.0	47.7	0.17	0.36	0.54	2.97	0.90
	PG16-112	VC7	88	39.4	96.9	96.4	75.7	76.5	0.20	0.29	0.53	12.4	1.70
	PG16-117	VC8	89	29.7	98.2	98.7	53.7	54.2	0.40	0.44	0.43	4.30	1.10
	PG16-138	VC10	90	71.5	99.5	99.6	77.4	81.7	0.18	0.28	0.37	6.36	0.40
	PG16-153	VC11	91	43.5	101.3	96.2	15.3	14.8	0.18	0.24	0.26	3.14	2.30
	PG16-185	VC13	92	74.2	98.6	99.1	25.1	25.3	0.25	0.26	0.27	7.35	0.70
McVeigh West Lower	PG16-253	VC14	93	58.0	99.5	98.1	3.4	3.1	0.16	0.17	0.24	3.94	4.90
	PGU-0044	VC15	94	67.2	99.1	98.6	7.2	5.8	0.36	0.43	0.27	7.51	2.50
McVeigh Central	PG16-056	VC17	95	19.7	96.7	94.9	20.1	20.3	0.23	0.36	0.49	7.80	5.00
	PG16-058-A	VC18	96	59.9	96.3	98.4	36.5	39.8	0.58	0.80	0.69	4.63	2.60
	PG16-058-B	VC19	97	13.8	93.2	95.5	28.3	28.6	0.53	0.80	0.63	4.75	4.50
	PG16-060	VC20	98	33.5	90.9	98.8	22.9	31.5	0.31	0.37	0.52	1.06	0.70
Austin Central	PG16-055	VC21	99	40.1	94.1	94.0	60.2	63.8	0.35	0.41	0.59	14.5	5.50
Austin East	PG16-114	VC29	101	39.3	83.2	83.5	65.6	68.3	0.23	0.32	0.63	7.53	1.60
	PG16-115	VC30	102	66.0	94.3	97.9	26.6	28.1	0.17	0.21	0.34	2.87	0.70
	PG16-227	VC31	103	13.9	91.7	97.3	27.3	29.8	0.18	0.18	0.44	5.42	0.70
Austin South Top	PGU-0035	VC32	104	28.3	91.0	91.6	86.4	88.5	0.28	0.34	0.31	6.15	4.40
	PG16-148	VC35	105	83.6	98.5	99.1	35.6	38.3	0.30	0.43	0.47	48.9	3.90
	PG16-150	VC36	106	72.4	98.7	99.3	26.1	27.6	0.16	0.17	0.39	5.57	2.20
	PG16-195	VC38	107	53.3	84.6	97.2	9.8	10.8	0.20	0.29	0.33	1.26	1.80
Austin South Middle	PG16-066	VC40	108	57.2	95.2	96.0	41.5	42.6	0.21	0.24	0.42	5.49	2.10
Austin South Deep	PG17-456	VC43	109	31.2	97.9	96.9	35.3	34.9	0.14	0.24	0.37	11.0	2.40
A3 Upper	PG17-304	VC45	110	61.5	98.2	98.1	41.2	42.8	0.19	0.32	0.49	29.2	6.30
A3 Lower	PG16-282	VC46	111	28.7	94.1	93.9	35.8	38.6	0.11	0.18	0.34	3.02	1.30
	PG17-320	VC48	112	26.8	95.9	96.5	55.0	57.2	0.07	0.14	0.37	5.51	1.90
Average				**45.7**	**95.4**	**96.6**	**38.9**	**40.4**	**0.24**	**0.33**	**0.42**	**8.02**	**2.5**

Source: BaseMet (2018)

The proportion of gold recovery between gravity recovery and leach recovery for each of the variability composites is shown in Figure 13-11.

Figure 13-11: Variability composites – gravity recovery versus leach recovery



Source: BaseMet (2018)

13.3 2021 Plant Operation Review

Production data from December 2020 to October 2021 was reviewed, which included head grade, throughput, grind size, gravity recovery, overall recovery and reagent consumption. The throughput weighted average recovery was 95.1% for the period, with almost half the gold being recovered in the gravity circuit. The Dec-2020 to Oct-2021 production data is presented in Table 13-16.

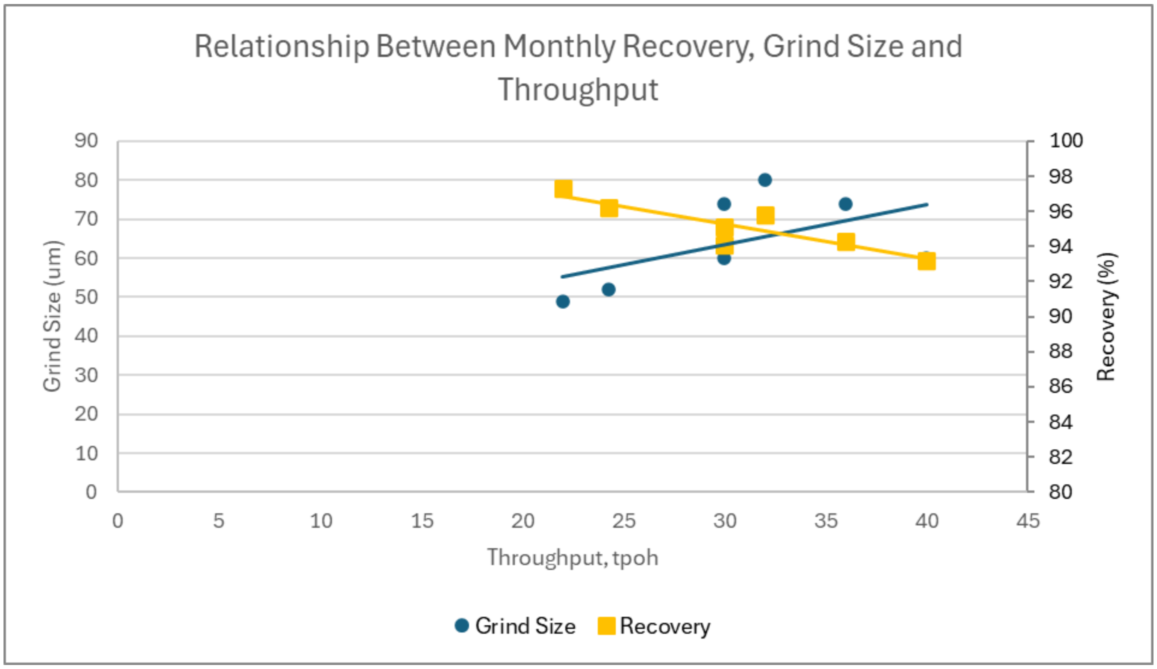
As is typical, a relationship exists between throughput, grind size and recovery, as presented in Figure 13-12. Higher throughput causes a coarser grind, resulting in less liberation and lower recovery. Based on the recovery as a function of throughput, at 800 tpd (33 tpoh), a recovery of approximately 95% is expected.

Table 13-16: Plant operation data (2021)

	Dec-20	Jan-21	Feb-21	Mar-21	Apr-21	May-21	Jun-21	Jul-21	Aug-21	Sep-21	Oct-21
Tonnes Milled, Reconciled Production	3,535	12,405	16,436	19,563	12,792	16,220	17,300	21,787	22,262	19,717	9,683
Head Grade (g/t)	7.05	3.71	2.64	2.5	4.26	5.53	3.45	5.31	5.6	3.36	3.91
Recovery (%)	96.6	95.3	95.1	95.1	97.3	96.2	94.1	95.8	94.3	93.2	95.2
Estimated Production (Oz)	775	1,408	1,328	1,493	1,706	2,777	1,804	3,560			
Throughput (t per operating hour)	15	20.1	30	30	22	24.3	30	32	36	40	40
Circuit Grind (P80)			51	60	49	**52**	**74**	80	74	60	45
Gravity Circuit Recovery				40.7	**56.7**	**50.7**	**40**	45.7	**45**	**53**	46

Source: Madsen Plant Operational Data (2021)

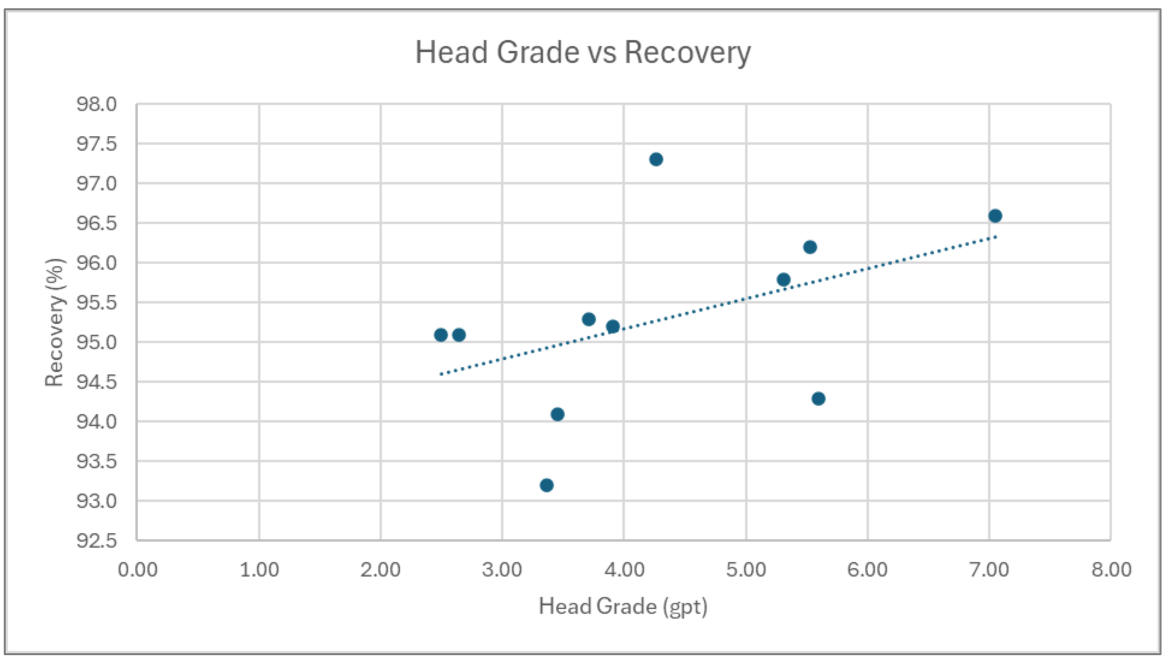
Figure 13-12: Plant operation recovery and grind size as a function of throughput (2021)



Source: Madsen Plant Operational Data (2021)

Another relationship exists between feed grade and recovery, as presented in Figure 13-13, with higher grade ore having higher recovery.

Figure 13-13: Monthly average grade versus recovery



Source: Madsen Plant Operational Data (2021)

13.4 Recovery Estimates

The existing plant was designed based on the results from the BaseMet (2018) test program and includes a single stage of crushing followed by a SAG and ball mill to achieve a P80 of 75 µm. Two gravity concentrators are installed prior to the ball mill circuit to collect any gravity recoverable gold, followed by a leach/CIP circuit, stripping and refining. Further details on the recovery methods are presented in Section 17.

Test work averages of gold recovery for each deposit, the 2021 production recovery (at 4.14 gpt average head grade), and the estimated LOM recovery based on the 2024 mine schedule (at 8.16 gpt average head grade) are summarized in Table 13-17. Test work averages of gold recovery for each deposit were applied to the 2024 SRK mine schedule to predict a LOM average recovery estimate of 95.7%. The 2024 SRK mine schedule has an average LOM head grade of 8.16 gpt, higher than the 2021 average head grade of 4.14 gpt. The grade recovery curve presented in Figure 13-12 points to a recovery of 96.6% at a head grade of 8.16 gpt, thus supporting using a recovery higher than the 2021 production recovery of 95.1%. The LOM average recovery estimate of 95.7% provides the basis for the economic analysis presented in Section 22.

Table 13-17: Gold recovery estimate

Description	Average Extraction (based on test work results) % Au
Austin	91.9
South Austin	95.4
A3	96.2
McVeigh	96.9
8 Zone	99.4
2021 Production Recovery (avg 4.14 gpt)	95.1
LOM Average Recovery (2024 mine schedule, avg 8.16 gpt)	**95.7**

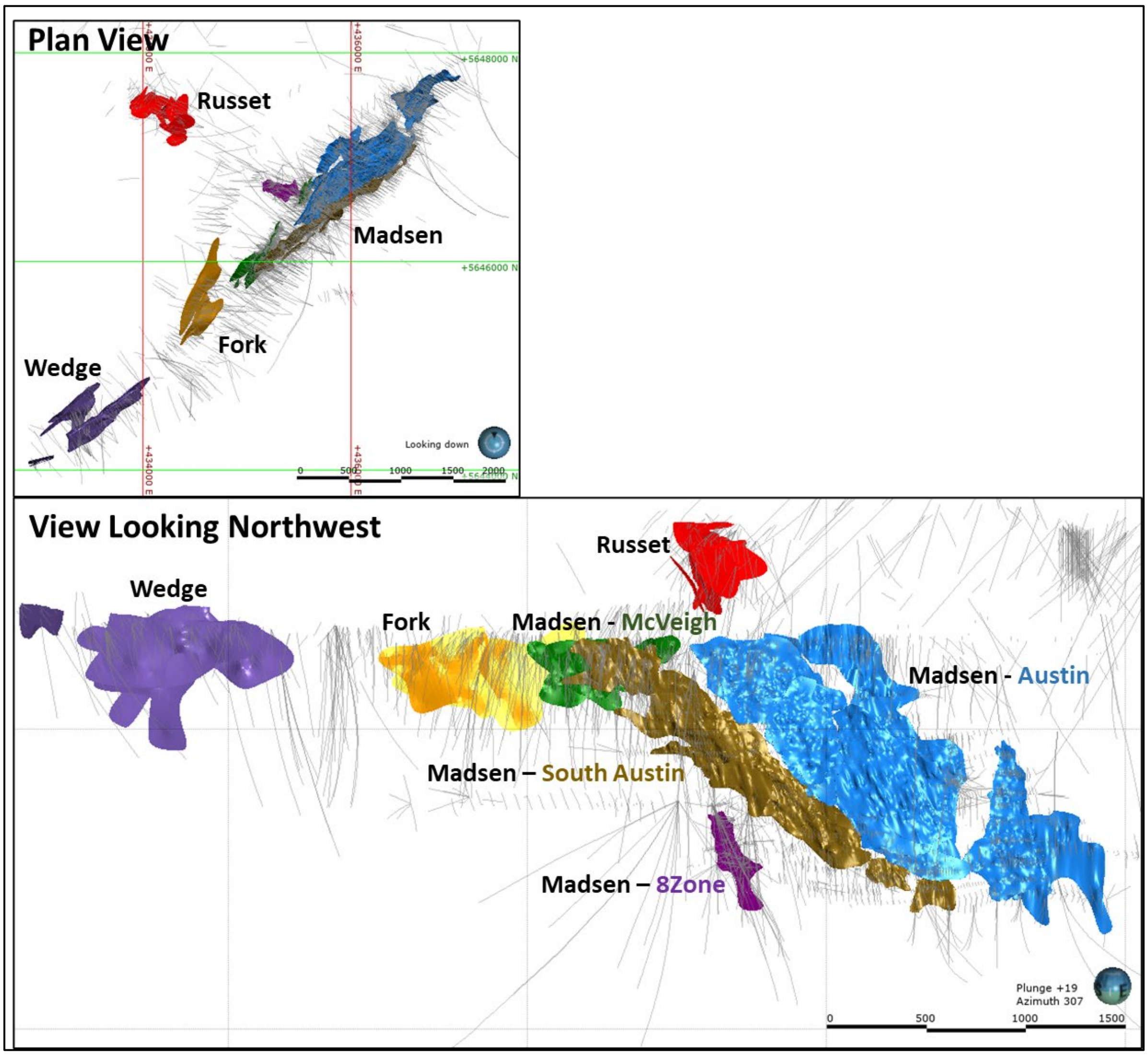
Source: BaseMet (2018)

14 Mineral Resource Estimates

14.1 Summary

The mineral resource statement presented herein represents a previously disclosed mineral resource estimate (MRE) prepared for the PureGold Mine (now Madsen Mine), with an effective date of December 31, 2021, in accordance with the Canadian Securities Administrators' National Instrument 43-101. The mine comprises the Madsen, Fork, Russet and Wedge deposits as depicted in Figure 14-1. The previous mineral resource estimate prepared for the mine with an effective date of February 5, 2019 was prepared by Ginto Consulting Inc. (see Makarenko et al., 2019).

Figure 14-1: Location map of PureGold (Madsen) Mine zones (with drill hole traces in grey)



Source: WRLG (2024)

The mineral resource models prepared by SRK consider a total of 13,621 boreholes which intersect the interpreted mineralized domains. A total of 13,192 boreholes are located within the Madsen deposit, 110 within the Fork deposit, 119 within the Russet deposit and 200 within the Wedge deposit. As well, 20,682 production chip samples collected during recent mining operations between 2018 and 2021, and 5,750 historical chip samples (circa 1969 to 1974) were used for mineral resource estimation. The mineral resource estimation work was completed by Mr. Cliff Revering, P.Eng., and peer reviewed by Dr. Oy Leuangthong, P.Eng. The effective date of the mineral resource statement is December 31, 2021.

Since the effective date of the mineral resource, additional diamond drilling was conducted until the mine closure on October 24, 2022. A total of 688 drill holes and 54,122 m of drilling was completed in 2022. An additional 205 drill holes and 19,872 m of drilling was completed by WRLG between October 1, 2023 and May 15, 2024. Based on a review of the results of this drilling it has been determined by Mr. Cliff Revering, Qualified Person for the Madsen MRE, that the information obtained will not have a material impact on the MRE presented in this report.

This section describes the resource estimation methodology and summarizes the key assumptions considered in the estimation workflow. In the opinion of Mr. Cliff Revering, the resource estimate reported herein is a reasonable representation of the gold mineral resources found in the PureGold Mine (now Madsen Mine) at the current level of drilling and sampling. The mineral resources have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (November 2019) and are reported in accordance with the Canadian Securities Administrators’ National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.

The database used to develop the geological model and mineral resource estimates for the PureGold Mine (now Madsen Mine) has been reviewed by Mr. Cliff Revering, and he is of the opinion that the current drilling information is sufficiently reliable to interpret the geology and mineralization controls of the PureGold Mine (now Madsen Mine) and that the assay data are sufficiently reliable to support the estimation of mineral resources.

Seequent’s Leapfrog Geo™ software was used to construct the geological model for the mine, and Maptek’s Vulcan™ and Seequent’s Leapfrog Edge™ software were used for mineral resource estimation.

14.2 Resource Database

The mine drillhole database contains 16,406 drill holes totalling over 1.3 Mm of drilling (Table 14-1); 2,488 of these drill holes (identified as Exploration and Production holes in Table 14-1) were drilled by Pure Gold between 2014 and 2021 totalling 351,356 m of drilling. The database contains 761,206 gold (Au) assays that have been amassed from drill holes to the end of 2021, of which 167,815 were drilled by Pure Gold since 2014.

Table 14-1: Drill hole database summary

Program Type	Number of Holes	Total Metres	Number of Au Assays
Exploration	1,190	319,292	141,991
Historical	13,958	992,745	593,391
Production	1,258	32,064	25,824
TOTALS	**16,406**	**1,344,101**	**761,206**

Source: WRLG (2024)

In addition to the drilling data available for the 2021 MRE update, a production chip samples database was provided by Pure Gold. A total of 26,432 production chip samples with associated Au assays were used in the MRE process, of which 20,682 were collected by Pure Gold since mining recommenced in 2018 (Table 14-2). The majority of Pure Gold chip samples are located in the McVeigh and South Austin zones, where mining activities by Pure Gold were focused. An additional 5,750 historical chip samples from the 8 Zone were also incorporated into the 2021 MRE update. Historical chip samples from the Austin and South Austin areas of the mine were excluded from the MRE workflow due to uncertainty with data quality and verification; these are also excluded from the total presented in Table 14-2.

Table 14-2: Production chip sample database summary

Year	Number of Chip Samples	Total Sample Length (m)
Historical	5,750	4,598
2018	1,698	1,064
2020	3,588	2,371
2021	15,396	11,869
TOTALS	**26,432**	**19,901**

Source: WRLG (2024)

14.3 Geological Model

The geological models used as the basis for the 2021 MRE were updated in 2021 by Pure Gold with assistance from SRK. These incorporated additional geological understanding of the mineralization controls obtained through underground mining and geological data collection programs during active mine production from 2018 to 2021. Three-dimensional modeling of the mineral resource domains was conducted using Leapfrog Geo^{TM} software and generally used a 3 g/t Au cut-off and 2 m minimum width criteria for mineralization domain interpretation. Areas of lower grade and/or waste material were incorporated into the mineralized domain model to facilitate interpretation of mineralization continuity related to vein packages across the deposit area.

The Madsen deposit is the largest of the deposits and comprises four zones identified as the Austin, South Austin, McVeigh and 8 Zone, and has a total of 50 separate mineralized domains. The Fork, Russet and Wedge deposits each have a total of 3, 15 and 6 individual mineralized domains, respectively. Table 14-3 provides a volume summary of the mineralized domains within each deposit. Historical mining of the Madsen deposit was focused within the Austin and South Austin zones, with relatively minor production from the McVeigh and 8 zones, as depicted in Figure 14-2.

Table 14-3: Mineralized domain volume summary

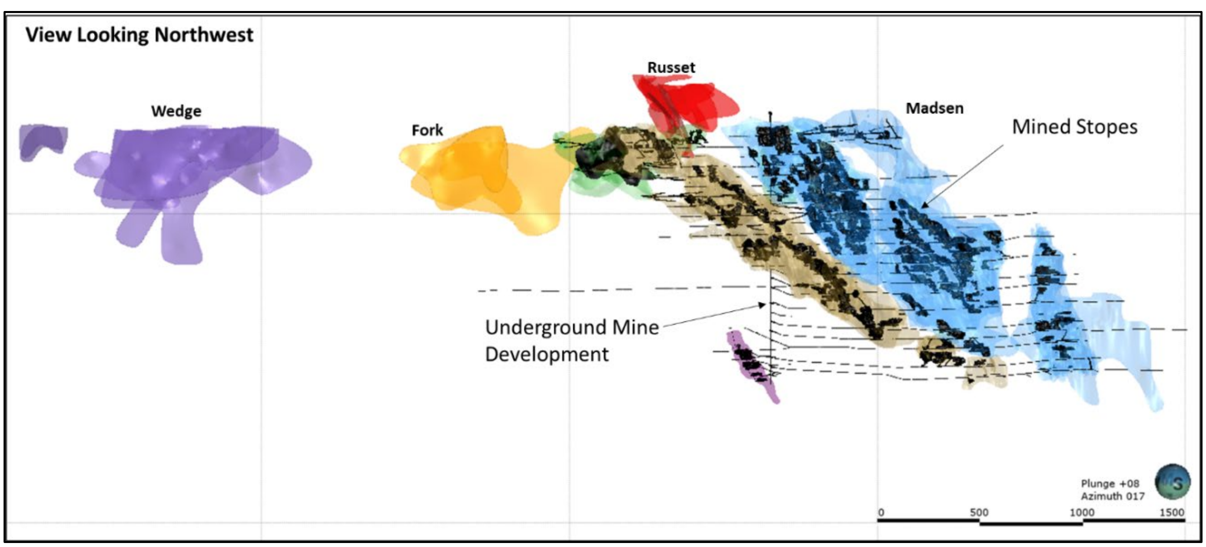
Deposit	Zone	Domain	Volume m^3 (000)
Madsen	Austin	111	1,690.2
Madsen	Austin	112	1,322.4
Madsen	Austin	113	662.4
Madsen	Austin	114	391.5
Madsen	Austin	115	293.8
Madsen	Austin	121	533.8
Madsen	Austin	122	992.3
Madsen	Austin	123	1,300.4
Madsen	Austin	124	1,121.8
Madsen	Austin	131	295.5
Madsen	Austin	132	371.4
Madsen	Austin	133	312.6
Madsen	Austin	134	270.0
Madsen	Austin	135	74.4
Madsen	Austin	141	335.7
Madsen	Austin	142	407.8
Madsen	Austin	151	375.4
Madsen	South Austin	301	113.8
Madsen	South Austin	302	42.6
Madsen	South Austin	303	41.7
Madsen	South Austin	304	80.2
Madsen	South Austin	305	105.6
Madsen	South Austin	311	0.2
Madsen	South Austin	312	153.5
Madsen	South Austin	313	123.1

Deposit	Zone	Domain	Volume m^3 (000)
Madsen	South Austin	321	1,218.5
Madsen	South Austin	322	1,226.5
Madsen	South Austin	323	311.9
Madsen	South Austin	324	270.1
Madsen	South Austin	331	53.5
Madsen	South Austin	332	62.9
Madsen	South Austin	333	38.8
Madsen	McVeigh	211	141.2
Madsen	McVeigh	212	131.8
Madsen	McVeigh	213	181.0
Madsen	McVeigh	214	35.7
Madsen	McVeigh	215	7.3
Madsen	McVeigh	216	2.6
Madsen	McVeigh	217	356.3
Madsen	McVeigh	218	229.5
Madsen	McVeigh	219	60.9
Madsen	McVeigh	220	127.3
Madsen	McVeigh	221	151.6
Madsen	McVeigh	222	143.1
Madsen	McVeigh	223	81.8
Madsen	McVeigh	224	238.4
Madsen	8 Zone	810	133.6
Madsen	8 Zone	820	62.7
Madsen	8 Zone	830	104.2
Madsen	8 Zone	840	84.3

Deposit	Domain	Volume m^3 (000)
Fork	FW	852.0
Fork	HW	395.8
Fork	NS	305.9
Russet	V1	71.1
Russet	V2	29.0
Russet	V3	136.7
Russet	V4	41.5
Russet	V5	5.7
Russet	V6	112.4
Russet	V7	175.6
Russet	V8	104.8
Russet	V9	21.4
Russet	V10	123.3
Russet	V11	23.2
Russet	V12	81.1
Russet	V13	34.2
Russet	V14	64.3
Russet	V15	108.3
Wedge	DVCK_1	486.8
Wedge	DVCK_2	1,046.4
Wedge	DVCK_3	904.8
Wedge	MJ_1	739.7
Wedge	MJ_2	415.1
Wedge	MJ_3	365.1

Source: WRLG (2024)

Figure 14-2: Geological model of the mine deposits, including historical mine development and mined stopes in the Madsen deposit

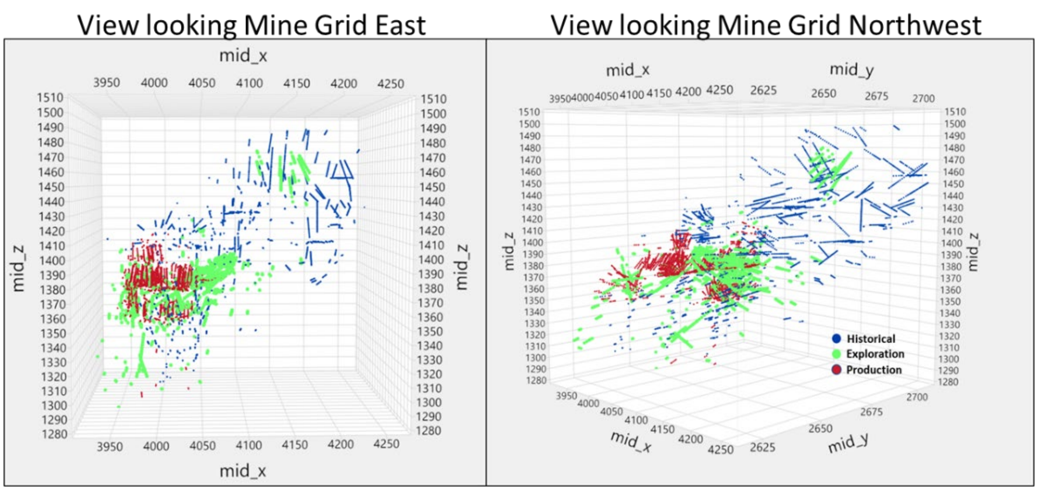


Source: WRLG (2024)

14.4 Historical Data Comparisons

To assess the compatibility of historical assay data relative to recent assay data collected by Pure Gold, Mr. Cliff Revering compared datasets within several mineralized domains. As an example, Figure 14-3 provides a summary of assay data within the Madsen – McVeigh Zone mineralized domain 213. Assay data were segregated into three data populations: 1) Pure Gold Exploration, 2) Pure Gold Production and 3) Historical assays. As seen in Figure 14-3, Pure Gold Production assays (red) are clustered within mined stope locations and were collected for the purpose of grade control ahead of mining, whereas Pure Gold Exploration (green) and Historical (blue) assays are more comparable datasets spatially. The assay summary statistics for the three data populations provided in Figure 14-3 demonstrate that Pure Gold Exploration and Historical assay data populations have similar average grade characteristics and grade distributions, whereas the Pure Gold Production data population is biased high, relative to the Exploration and Historical data, due to its preferential location within mined stopes targeted above an economic cut-off grade. It should be noted that the summary statistics provided in Figure 14-3 are length-weighted and have incorporated a grade cap of 187 g/t Au to mitigate the impact of high-grade outliers within the data populations.

Figure 14-3: Pure Gold and historical assay data comparison within the Madsen – McVeigh Zone mineralized domain 213



Summary Statistics (length weighted)

Exploration

Quantiles		
100.0%	maximum	187.00
99.5%		65.50
97.5%		18.79
90.0%		5.70
75.0%	quartile	1.72
50.0%	median	0.21
25.0%	quartile	0.04
10.0%		0.01
2.5%		0.00
0.5%		0.00
0.0%	minimum	0.00

Summary Statistics	
Mean	2.79
Std Dev	10.36
Std Err Mean	0.24
Upper 95% Mean	3.26
Lower 95% Mean	2.33
N	1745.00

Historical

Quantiles		
100.0%	maximum	187.00
99.5%		63.72
97.5%		16.87
90.0%		5.83
75.0%	quartile	1.99
50.0%	median	0.34
25.0%	quartile	0.17
10.0%		0.09
2.5%		0.00
0.5%		0.00
0.0%	minimum	0.00

Summary Statistics	
Mean	2.52
Std Dev	8.02
Std Err Mean	0.18
Upper 95% Mean	2.87
Lower 95% Mean	2.16
N	2978.00

Production

Quantiles		
100.0%	maximum	187.00
99.5%		100.00
97.5%		32.90
90.0%		8.68
75.0%	quartile	2.63
50.0%	median	0.38
25.0%	quartile	0.08
10.0%		0.02
2.5%		0.01
0.5%		0.00
0.0%	minimum	0.00

Summary Statistics	
Mean	4.66
Std Dev	15.07
Std Err Mean	0.29
Upper 95% Mean	5.22
Lower 95% Mean	4.10
N	2959.00

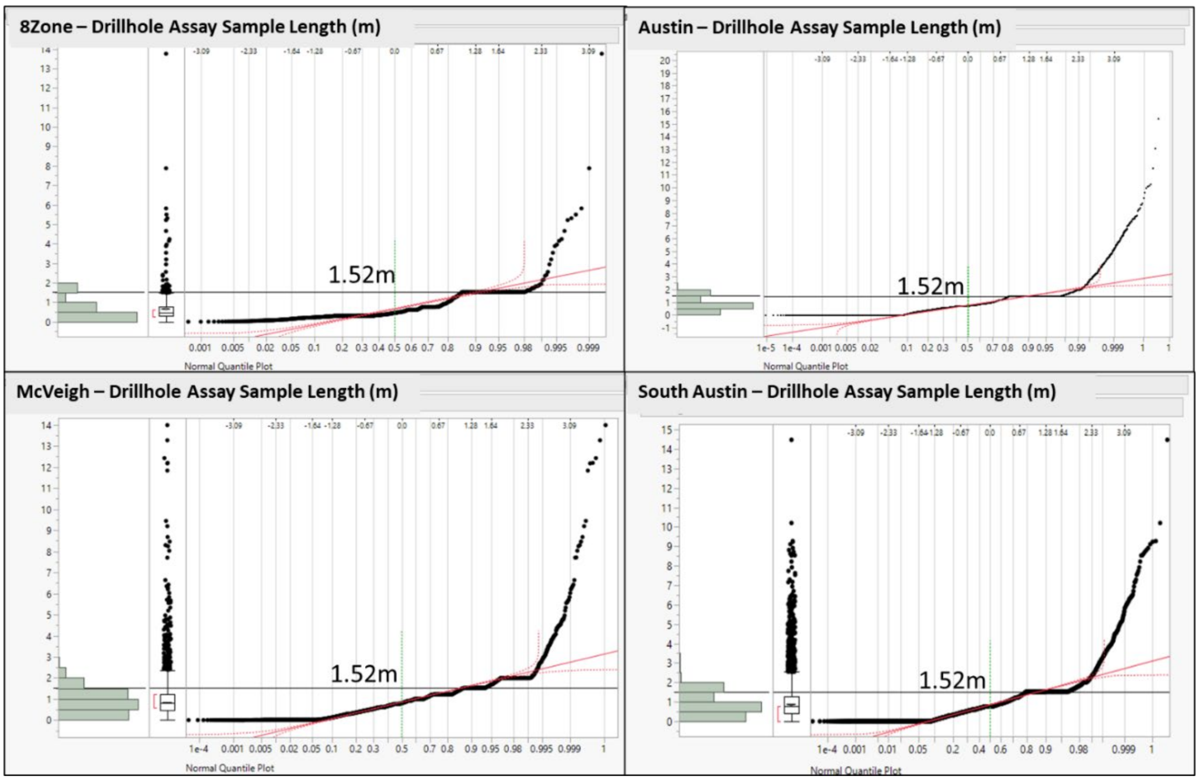
Source: WRLG (2024)

14.5 Compositing

14.5.1 Madsen

Assay samples for the Madsen deposit were composited to 1.52 m (5 ft) lengths within the mineralized domain boundaries, and all residual composites smaller than 0.76 m in length were added to the adjacent composite interval. As shown in Figure 14-4, over 95% of all drill hole assay sample lengths were collected using a 1.52 m sample length or smaller, and therefore supported the selection of a 1.52 m composite length.

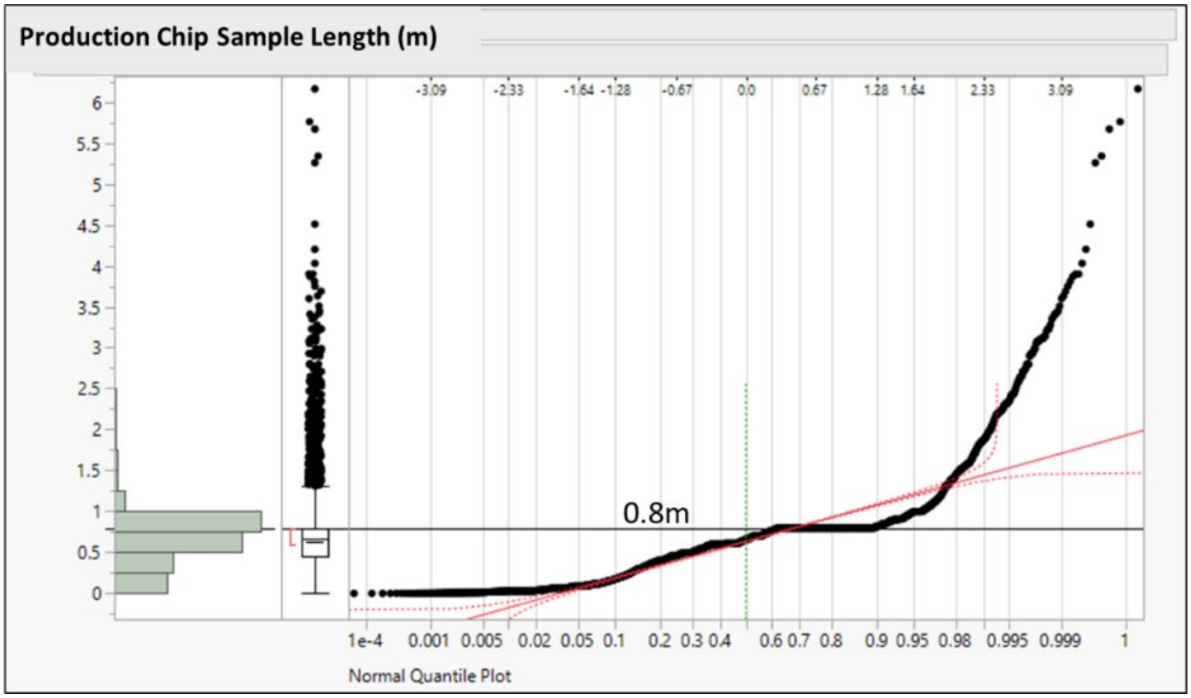
Figure 14-4: Madsen deposit drill hole assay sample length distributions (by mineralized zone)



Source: WRLG (2024)

Production chip samples used in the estimation process were predominately sampled at a 0.80 m sample length or smaller (Figure 14-5), and therefore were also composited to a 1.52 m composite length.

Figure 14-5: Madsen deposit – production chip sample assay length distributions



Source: WRLG (2024)

Summary statistics of drill hole and production chip samples raw assay data (by mineralized domain) for the Madsen deposit are provided in Table 14-4 and Table 14-5, respectively, with summary statistics for the composited (uncapped) assay data provided in Table 14-6.

Table 14-4: Madsen deposit drill hole raw gold assay (g/t) summary statistics (length-weighted, by mineralized domain)

Mineralized Domain	Domain Code	# of Samples	Mean	StDev	Min	Max	CV
Austin_HG1_V1	111	22,149	5.12	25.82	0.00	4,395.4	5.04
Austin_HG1_V2	112	15,685	4.02	10.11	0.00	964.8	2.52
Austin_HG1_V3	113	4,573	1.26	4.69	0.00	238.9	3.72
Austin_HG1_V4	114	3,072	2.28	7.07	0.00	279.1	3.10
Austin_HG1_V5	115	1,499	1.37	4.24	0.00	167.3	3.10
Austin_HG2_V1	121	2,881	1.36	5.13	0.00	149.5	3.77
Austin_HG2_V2	122	9,499	3.54	11.37	0.00	648.0	3.21
Austin_HG2_V3	123	14,676	4.38	26.03	0.00	2,534.7	5.94
Austin_HG2_V4	124	10,366	2.24	11.93	0.00	750.9	5.31
Austin_HG3_V1	131	3,558	2.88	7.32	0.00	173.5	2.54
Austin_HG3_V2	132	4,447	4.28	16.13	0.00	1,309.4	3.77
Austin_HG3_V3	133	3,824	3.86	13.34	0.00	720.0	3.46
Austin_HG3_V4	134	2,503	3.15	10.53	0.00	573.9	3.34
Austin_HG3_V5	135	252	2.75	9.15	0.00	160.5	3.33
Austin_HG4_V1	141	2,213	1.59	4.39	0.00	123.4	2.76
Austin_HG4_V2	142	2,031	1.67	9.84	0.00	332.8	5.88
Austin_HG5_V1	151	1,007	3.09	13.34	0.00	438.9	4.32
McVeigh_HG1_3565_FW	211	66	2.99	7.37	0.00	55.2	2.47
McVeigh_HG1_3565_HW	212	25	5.26	18.69	0.00	79.4	3.55
McVeigh_HG1_BSZ	213	7,682	3.93	23.76	0.00	1,175.0	6.05
McVeigh_HG1_FW	214	1,433	1.37	8.10	0.00	264.9	5.90
McVeigh_HG1_FW2	215	312	0.94	2.85	0.00	33.9	3.03
McVeigh_HG1_FW3	216	133	0.51	1.45	0.00	12.0	2.83
McVeigh_HG1_HW	217	2,406	2.18	5.41	0.00	100.0	2.49
McVeigh_HG1_HW2a	218	1,677	3.89	37.73	0.00	2,406.8	9.71
McVeigh_HG1_HW2b	219	1,411	2.39	9.45	0.00	221.1	3.96
McVeigh_HG1_MAIN	220	1,824	2.17	8.71	0.00	251.1	4.02
McVeigh_HG2_V1	221	1,900	3.28	11.24	0.00	333.3	3.43
McVeigh_HG2_V2	222	1,763	3.22	8.77	0.00	222.9	2.72
McVeigh_HG2_V3	223	1,325	1.17	4.07	0.00	137.5	3.49
McVeigh_HG2_V4	224	561	5.02	92.07	0.00	3,884.6	18.33
SouthAustin_A3_V1	301	184	0.91	1.28	0.00	6.2	1.40
SouthAustin_A3_V2	302	603	3.23	14.81	0.00	213.3	4.59
SouthAustin_A3_V3	303	261	4.29	14.48	0.00	133.7	3.38
SouthAustin_FINGA	304	752	3.82	9.13	0.00	188.9	2.39
SouthAustin_FW_V1	305	1,807	6.32	29.64	0.00	857.8	4.69
SouthAustin_HG1_V1	311	2,228	2.57	11.58	0.00	382.0	4.51
SouthAustin_HG1_V2	312	2,170	2.73	12.36	0.00	543.8	4.52
SouthAustin_HG1_V3	313	1,133	1.73	6.94	0.00	100.0	4.01
SouthAustin_HG2_V1	321	13,516	6.14	31.74	0.00	2,063.3	5.17
SouthAustin_HG2_V2	322	10,996	4.10	18.84	0.00	977.1	4.60
SouthAustin_HG2_V3	323	2,951	1.53	6.28	0.00	197.5	4.11
SouthAustin_HG2_V4	324	2,785	3.52	15.06	0.00	578.1	4.28
SouthAustin_HG3_V1	331	737	6.65	20.49	0.00	559.5	3.08
SouthAustin_HG3_V2	332	1,129	11.77	80.06	0.00	2,390.7	6.80
SouthAustin_HG3_V3	333	506	3.07	19.73	0.00	582.2	6.42
8Zone_V1	810	1,269	13.88	130.30	0.00	6,661.0	9.39
8Zone_V2	820	715	6.86	61.16	0.00	2,731.9	8.91
8Zone_V3	830	682	0.75	6.97	0.00	377.1	9.29
8Zone_V4	840	616	1.90	11.92	0.00	486.2	6.25

Source: WRLG (2024)

Table 14-5: Madsen deposit chip sample raw gold assay (g/t) summary statistics (length-weighted, by mineralized domain)

MinDomain	Domain Code	of Sample	Mean	StDev	Min	Max	CV
Austin_HG1_V2	112	5	0.21	0.20	0.01	0.56	0.93
Austin_HG1_V5	115	34	12.11	48.07	0.06	350.02	3.97
Austin_HG4_V1	141	76	1.64	4.24	0.00	27.51	2.58
Austin_HG4_V2	142	526	2.92	8.26	0.00	138.53	2.83
McVeigh_HG1_3565_FW	211	62	0.23	0.37	0.00	1.96	1.60
McVeigh_HG1_3565_HW	212	33	0.50	1.05	0.00	8.92	2.08
McVeigh_HG1_BSZ	213	3302	7.83	49.62	0.00	2022.78	6.34
McVeigh_HG1_FW	214	579	3.92	46.17	0.00	1235.29	11.79
McVeigh_HG1_FW2	215	26	0.05	0.04	0.01	0.24	0.74
McVeigh_HG1_FW3	216	17	0.09	0.21	0.01	1.01	2.46
McVeigh_HG1_HW	217	1324	2.95	15.88	0.00	1019.84	5.39
McVeigh_HG1_HW2a	218	605	6.39	71.31	0.00	2259.66	11.16
McVeigh_HG1_HW2b	219	465	3.60	12.79	0.00	189.69	3.56
McVeigh_HG1_MAIN	220	1328	2.13	4.52	0.00	91.50	2.12
SouthAustin_HG1_V1	311	869	10.91	154.99	0.00	5083.89	14.20
SouthAustin_HG1_V2	312	137	0.68	0.99	0.01	9.04	1.45
SouthAustin_HG1_V3	313	52	1.90	3.42	0.00	22.42	1.80
SouthAustin_HG2_V1	321	231	2.35	6.45	0.00	66.90	2.74
SouthAustin_HG2_V2	322	48	0.52	0.76	0.01	4.47	1.47
SouthAustin_HG2_V3	323	75	1.29	7.51	0.01	72.69	5.82
SouthAustin_HG2_V4	324	58	2.12	3.52	0.01	25.34	1.66
8Zone_V1	810	3993	26.05	153.57	0.00	8003.94	5.89
8Zone_V2	820	1808	23.98	101.51	0.00	2685.59	4.23
8Zone_V3	830	162	3.10	6.56	0.00	70.63	2.11
8Zone_V4	840	37	3.31	3.55	0.00	16.46	1.07

Source: WRLG (2024)

Table 14-6: Madsen deposit 1.52 m composited gold summary statistics (g/t, uncapped, by mineralized domain)

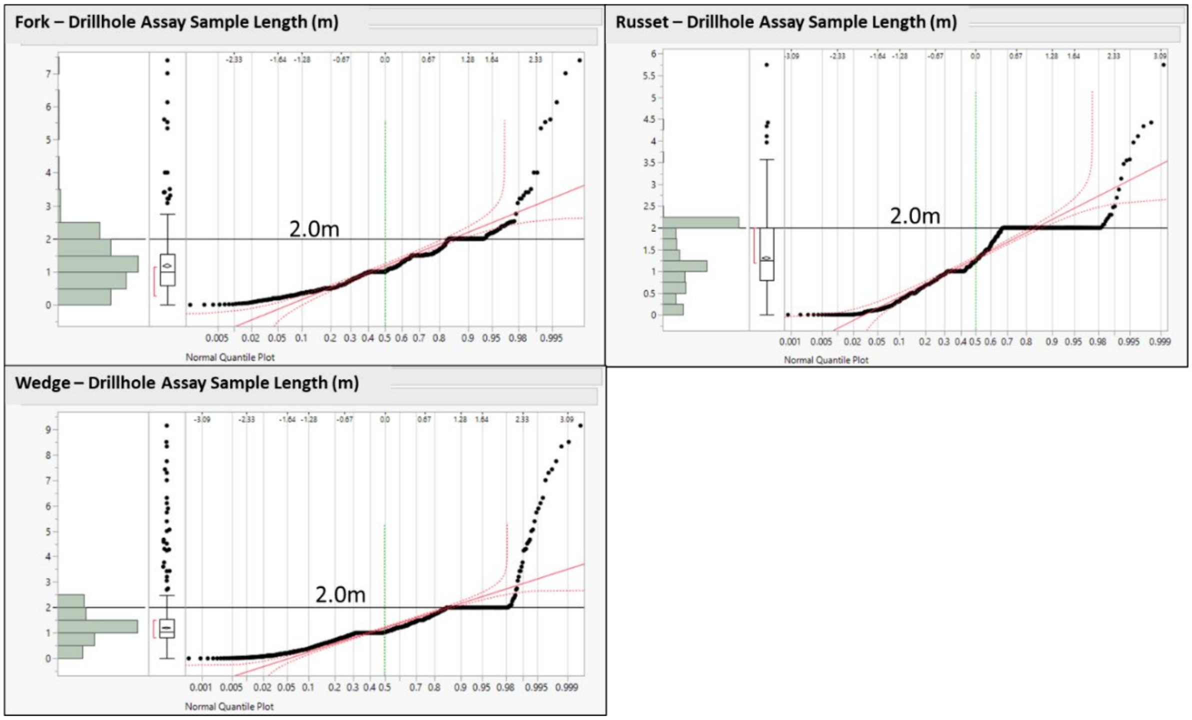
MinDomain	Domain Code	# of Comps Total	# of Comps DDH	# of Comps Chips	Mean (uncapped)	StDev	Min	Max	CV
Austin_HG1_V1	111	12141	12141	0	5.11	19.14	0.00	1098.80	3.75
Austin_HG1_V2	112	8042	8040	2	4.01	8.84	0.00	250.30	2.21
Austin_HG1_V3	113	3003	3003	0	1.28	3.11	0.00	73.52	2.43
Austin_HG1_V4	114	1581	1581	0	2.27	5.74	0.00	118.31	2.53
Austin_HG1_V5	115	956	940	16	1.56	5.76	0.00	149.58	3.70
Austin_HG2_V1	121	1660	1660	0	1.38	4.26	0.00	93.60	3.10
Austin_HG2_V2	122	5276	5276	0	3.56	8.57	0.00	219.52	2.41
Austin_HG2_V3	123	8623	8623	0	4.40	22.69	0.00	1698.47	5.16
Austin_HG2_V4	124	6536	6536	0	2.24	8.01	0.00	288.19	3.58
Austin_HG3_V1	131	1786	1786	0	2.92	6.58	0.00	143.66	2.25
Austin_HG3_V2	132	2149	2149	0	4.24	10.02	0.00	181.30	2.36
Austin_HG3_V3	133	1976	1976	0	3.85	8.59	0.00	118.00	2.23
Austin_HG3_V4	134	1357	1357	0	3.13	8.25	0.00	142.90	2.64
Austin_HG3_V5	135	122	122	0	2.74	6.52	0.00	38.68	2.38
Austin_HG4_V1	141	1218	1189	29	1.58	3.65	0.00	44.38	2.30
Austin_HG4_V2	142	1489	1247	242	1.87	8.73	0.00	270.75	4.66
Austin_HG5_V1	151	519	519	0	3.08	10.89	0.00	229.58	3.53
McVeigh_HG1_3565_FW	211	64	38	26	1.89	5.09	0.00	36.88	2.69
McVeigh_HG1_3565_HW	212	29	16	13	2.90	11.04	0.00	59.04	3.81
McVeigh_HG1_BSZ	213	5776	4395	1381	4.77	22.89	0.00	679.22	4.80
McVeigh_HG1_FW	214	1095	824	271	1.96	16.76	0.00	519.33	8.55
McVeigh_HG1_FW2	215	208	201	7	0.94	2.62	0.00	26.99	2.79
McVeigh_HG1_FW3	216	78	78	0	0.44	1.53	0.00	12.00	3.44
McVeigh_HG1_HW	217	1831	1302	529	2.42	7.12	0.00	169.94	2.94
McVeigh_HG1_HW2a	218	1157	895	262	4.50	31.78	0.00	868.45	7.07
McVeigh_HG1_HW2b	219	936	743	193	2.61	9.67	0.00	148.71	3.70
McVeigh_HG1_MAIN	220	1612	1102	510	2.22	7.28	0.00	193.50	3.28
McVeigh_HG2_V1	221	1002	1002	0	3.28	11.86	0.00	333.26	3.62
McVeigh_HG2_V2	222	932	932	0	3.31	7.65	0.00	88.91	2.31
McVeigh_HG2_V3	223	755	755	0	1.17	3.24	0.00	38.28	2.76
McVeigh_HG2_V4	224	287	287	0	4.37	51.93	0.00	879.26	11.88
SouthAustin_A3_V1	301	105	105	0	0.94	1.37	0.00	6.17	1.46
SouthAustin_A3_V2	302	340	340	0	3.24	12.97	0.00	175.99	4.01
SouthAustin_A3_V3	303	140	140	0	4.48	14.57	0.00	133.71	3.25
SouthAustin_FINGA	304	339	339	0	3.88	9.08	0.00	73.87	2.34
SouthAustin_FW_V1	305	989	989	0	6.46	24.54	0.00	462.02	3.80
SouthAustin_HG1_V1	311	1646	1274	372	4.55	56.66	0.00	2141.28	12.45
SouthAustin_HG1_V2	312	1240	1183	57	2.64	8.31	0.00	160.11	3.15
SouthAustin_HG1_V3	313	650	629	21	1.86	6.27	0.00	61.80	3.37
SouthAustin_HG2_V1	321	7550	7445	105	6.05	23.26	0.00	1038.78	3.85
SouthAustin_HG2_V2	322	6574	6554	20	4.09	14.51	0.00	425.37	3.55
SouthAustin_HG2_V3	323	1792	1756	36	1.51	5.30	0.00	122.04	3.51
SouthAustin_HG2_V4	324	1372	1350	22	3.46	12.43	0.00	149.80	3.59
SouthAustin_HG3_V1	331	338	338	0	6.79	16.16	0.00	165.83	2.38
SouthAustin_HG3_V2	332	519	519	0	11.91	74.11	0.00	1470.48	6.22
SouthAustin_HG3_V3	333	256	256	0	3.07	10.98	0.00	134.19	3.57
8Zone_V1	810	1963	488	1475	21.52	98.63	0.00	1968.24	4.58
8Zone_V2	820	919	236	683	18.60	82.68	0.00	1410.82	4.44
8Zone_V3	830	443	388	55	1.07	4.26	0.00	59.85	3.96
8Zone_V4	840	279	264	15	1.87	5.78	0.00	73.05	3.09

Source: WRLG (2024)

14.5.2 Fork, Russet and Wedge Deposits

Assay samples for the Fork, Russet and Wedge deposits were composted to 2.0 m lengths due to the use of larger sample lengths in these deposits as shown in Figure 14-6. Composites were generated within the mineralized domain boundaries, and all residual composites smaller than 1.0 m in length were added to the adjacent composite interval.

Figure 14-6: Satellite deposits drill hole assay sample length distributions (by deposit)



Source: WRLG (2024)

Summary statistics of the raw assay data (by mineralized domain) for the Fork, Russet and Wedge deposits are provided in Table 14-7, with summary statistics for the composited (uncapped) assay data provided in Table 14-8.

Table 14-7: Fork, Russet and Wedge drill hole raw gold assay (g/t) summary statistics (length-weighted, by mineralized domain)

Deposit	Mineralized Domain	# of Samples	Au_gpt	StDev	Min	Max	CV
Fork	FW	190	0.92	5.02	0.0025	61.12	5.48
Fork	HW	294	1.74	6.70	0.0025	169.09	3.85
Fork	NS	244	2.09	9.97	0.0025	125.19	4.77
Russet	V1	67	2.45	5.54	0.0025	35.90	2.26
Russet	V2	47	2.83	9.82	0.0025	56.20	3.47
Russet	V3	154	0.52	2.82	0.0025	23.20	5.41
Russet	V4	47	5.93	22.21	0.005	132.00	3.74
Russet	V5	28	2.93	8.22	0.0025	42.20	2.81
Russet	V6	130	3.42	12.97	0.0025	115.83	3.79
Russet	V7	206	0.70	5.88	0.0025	80.72	8.44
Russet	V8	188	0.36	2.18	0.0025	17.67	6.13
Russet	V9	26	1.00	2.63	0.0025	11.25	2.64
Russet	V10	123	1.73	4.70	0.0025	27.20	2.72
Russet	V11	36	3.88	12.33	0.0025	83.80	3.18
Russet	V12	49	4.46	9.93	0.0025	46.40	2.22
Russet	V13	7	6.24	9.02	0.0025	19.65	1.45
Russet	V14	61	0.87	1.96	0.0025	9.15	2.26
Russet	V15	39	1.95	3.82	0.023	13.75	1.96
Wedge	DVCK_1	484	1.65	10.52	0.0025	132.47	6.38
Wedge	DVCK_2	626	1.04	3.87	0.0003	37.49	3.70
Wedge	DVCK_3	561	2.06	17.64	0.0003	354.00	8.56
Wedge	MJ_1	196	0.58	3.52	0.0025	32.90	6.07
Wedge	MJ_2	129	3.11	14.55	0.0025	185.62	4.67
Wedge	MJ_3	80	2.33	11.07	0.0025	94.60	4.75

Source: WRLG (2024)

Table 14-8: Fork, Russet and Wedge deposits 2.0 m composited gold summary statistics (g/t, uncapped, by mineralized domain)

Deposit	Mineralized Domain	# of Comps	Au_gpt	StDev	Min	Max	CV
Fork	FW	124	0.75	2.89	0.00	29.08	3.88
Fork	HW	149	1.69	5.00	0.00	50.48	2.96
Fork	NS	146	2.10	6.26	0.00	45.24	2.98
Russet	V1	25	2.46	5.50	0.01	35.90	2.23
Russet	V2	18	2.92	8.45	0.00	35.68	2.89
Russet	V3	122	0.52	2.02	0.00	21.59	3.87
Russet	V4	17	6.68	18.99	0.01	76.02	2.84
Russet	V5	11	2.93	6.69	0.01	20.13	2.29
Russet	V6	60	3.51	9.56	0.00	44.62	2.72
Russet	V7	164	0.70	3.37	0.00	40.37	4.84
Russet	V8	143	0.36	1.65	0.00	17.52	4.63
Russet	V9	11	1.00	1.90	0.01	5.65	1.90
Russet	V10	65	1.73	3.89	0.00	27.20	2.24
Russet	V11	14	3.87	8.43	0.01	31.87	2.18
Russet	V12	23	4.57	8.30	0.00	35.75	1.81
Russet	V13	5	6.24	8.30	0.00	19.00	1.33
Russet	V14	27	0.89	1.73	0.00	6.69	1.93
Russet	V15	29	1.95	2.93	0.03	13.75	1.50
Wedge	DVCK_1	281	1.68	7.84	0.00	76.41	4.67
Wedge	DVCK_2	369	1.05	3.12	0.00	33.48	2.97
Wedge	DVCK_3	320	2.07	16.48	0.00	354.00	7.97
Wedge	MJ_1	148	0.58	2.63	0.00	28.65	4.53
Wedge	MJ_2	75	3.05	10.74	0.00	108.50	3.51
Wedge	MJ_3	53	2.16	9.61	0.00	94.60	4.45

Source: WRLG (2024)

14.6 Evaluation of Outliers

Grade capping is a technique used to mitigate the potential effect that a small population of high-grade sample outliers can have during grade estimation. These high-grade samples are not considered to be representative of the general sample population and are therefore “capped” to a level that is more representative of the general data population. Although subjective, grade capping is a common industry practice when performing grade estimation for deposits that have significant grade variability.

Outlier analysis for the Madsen deposit was conducted on the 1.52 m composites and on 2.0 m composites for the Fork, Russet and Wedge deposits. Grade capping analysis was conducted separately for each mineralized domain. Histograms and normal quantile plots were generated for each data population and used to assess appropriate grade capping thresholds. Composites were capped prior to grade estimation. A summary of grade capping thresholds and capped summary statistics are provided in Table 14-9 and Table 14-10.

Table 14-9: Madsen deposit grade (g/t) capping summary of 1.52 m gold composites

MinDomain	Domain Code	# of Comps	# Comps Capped	Au_g/t (capped)	StDev (capped)	Min	Max (capped)	CV (capped)	% Grade Reduction
Austin_HG1_V1	111	12141	18	4.78	10.26	0.00	130.00	2.15	6%
Austin_HG1_V2	112	8042	4	3.97	8.18	0.00	122.00	2.06	1%
Austin_HG1_V3	113	3003	11	1.20	2.04	0.00	17.00	1.71	7%
Austin_HG1_V4	114	1581	6	2.14	4.13	0.00	33.00	1.93	6%
Austin_HG1_V5	115	956	8	1.31	2.30	0.00	15.00	1.76	16%
Austin_HG2_V1	121	1660	5	1.28	2.95	0.00	29.00	2.30	7%
Austin_HG2_V2	122	5276	9	3.49	7.51	0.00	74.00	2.16	2%
Austin_HG2_V3	123	8623	8	4.12	11.44	0.00	174.00	2.78	6%
Austin_HG2_V4	124	6536	12	2.14	6.31	0.00	75.00	2.95	5%
Austin_HG3_V1	131	1786	15	2.75	4.86	0.00	29.00	1.77	6%
Austin_HG3_V2	132	2149	15	3.93	7.04	0.00	50.00	1.79	7%
Austin_HG3_V3	133	1976	9	3.73	7.50	0.00	55.00	2.01	3%
Austin_HG3_V4	134	1357	4	2.96	6.38	0.00	53.00	2.15	5%
Austin_HG3_V5	135	122	2	2.53	5.49	0.00	25.00	2.17	8%
Austin_HG4_V1	141	1218	2	1.56	3.44	0.00	30.00	2.20	1%
Austin_HG4_V2	142	1489	24	1.39	2.97	0.00	15.00	2.13	26%
Austin_HG5_V1	151	519	5	2.62	4.12	0.00	22.00	1.57	15%
McVeigh_HG1_3565_FW	211	64	2	1.39	2.44	0.00	8.00	1.76	27%
McVeigh_HG1_3565_HW	212	29	1	1.20	2.80	0.00	10.00	2.34	59%
McVeigh_HG1_BSZ	213	5776	23	4.12	12.14	0.00	122.00	2.95	14%
McVeigh_HG1_FW	214	1095	9	1.27	3.59	0.00	28.00	2.82	35%
McVeigh_HG1_FW2	215	208	4	0.77	1.61	0.00	7.00	2.08	18%
McVeigh_HG1_FW3	216	78	1	0.36	0.93	0.00	5.00	2.63	19%
McVeigh_HG1_HW	217	1831	8	2.22	4.40	0.00	37.00	1.98	8%
McVeigh_HG1_HW2a	218	1157	8	2.93	6.96	0.00	55.00	2.38	35%
McVeigh_HG1_HW2b	219	936	12	2.10	5.05	0.00	32.00	2.40	20%
McVeigh_HG1_MAIN	220	1612	6	2.06	4.84	0.00	40.00	2.36	7%
McVeigh_HG2_V1	221	1002	2	2.97	5.63	0.00	46.00	1.89	9%
McVeigh_HG2_V2	222	932	9	3.06	5.77	0.00	36.00	1.89	8%
McVeigh_HG2_V3	223	755	8	1.04	2.23	0.00	13.00	2.14	11%
McVeigh_HG2_V4	224	287	2	1.31	3.01	0.00	19.00	2.30	70%
SouthAustin_A3_V1	301	105	0	0.94	1.37	0.00	6.17	1.46	0%
SouthAustin_A3_V2	302	340	10	1.91	3.45	0.00	16.00	1.81	41%
SouthAustin_A3_V3	303	140	2	3.50	8.02	0.00	40.00	2.29	22%
SouthAustin_FINGA	304	339	9	3.40	6.71	0.00	29.00	1.97	12%
SouthAustin_FW_V1	305	989	4	5.93	18.09	0.00	161.00	3.05	8%
SouthAustin_HG1_V1	311	1646	4	2.78	9.05	0.00	99.00	3.25	39%
SouthAustin_HG1_V2	312	1240	2	2.50	6.23	0.00	69.00	2.49	5%
SouthAustin_HG1_V3	313	650	4	1.78	5.65	0.00	43.00	3.17	4%
SouthAustin_HG2_V1	321	7550	8	5.76	16.16	0.00	238.00	2.81	5%
SouthAustin_HG2_V2	322	6574	5	3.94	11.50	0.00	152.00	2.92	4%
SouthAustin_HG2_V3	323	1792	2	1.46	4.47	0.00	57.00	3.06	3%
SouthAustin_HG2_V4	324	1372	6	3.33	11.14	0.00	96.00	3.35	4%
SouthAustin_HG3_V1	331	338	5	6.01	11.26	0.00	55.00	1.87	11%
SouthAustin_HG3_V2	332	519	7	7.29	12.41	0.00	63.00	1.70	39%
SouthAustin_HG3_V3	333	256	5	2.18	4.53	0.00	23.00	2.08	29%
8Zone_V1	810	1963	9	17.72	49.70	0.00	400.00	2.80	18%
8Zone_V2	820	919	9	14.22	37.84	0.00	260.00	2.66	24%
8Zone_V3	830	443	3	0.83	1.54	0.00	11.50	1.86	23%
8Zone_V4	840	279	2	1.73	4.38	0.00	36.00	2.52	7%

Source: WRLG (2024)

Table 14-10: Fork, Russet and Wedge deposits grade (g/t) capping summary of 2.0 m gold composites

Deposit	Mineralized Domain	# of Comps	# of Comps Capped	Au_g/t (capped)	StDev (capped)	Min	Max (capped)	CV (capped)	% Grade Reduction
Fork	FW	124	5	0.51	1.66	0.00	5.00	3.26	32%
Fork	HW	149	2	1.39	4.06	0.00	14.00	2.93	18%
Fork	NS	146	5	1.66	5.71	0.00	18.00	3.45	21%
Russet	V1	25	2	1.98	4.62	0.01	9.70	2.33	20%
Russet	V2	18	2	1.63	5.24	0.00	11.50	3.21	44%
Russet	V3	122	1	0.47	2.21	0.00	10.00	4.71	10%
Russet	V4	17	1	2.50	5.39	0.01	11.00	2.15	63%
Russet	V5	11	1	0.90	1.95	0.01	3.00	2.16	69%
Russet	V6	60	3	2.54	8.54	0.00	21.00	3.36	28%
Russet	V7	164	2	0.43	1.37	0.00	5.20	3.17	38%
Russet	V8	143	1	0.28	1.36	0.00	6.80	4.83	21%
Russet	V9	11	0	1.00	2.76	0.01	5.65	2.77	0%
Russet	V10	65	1	1.63	4.73	0.00	13.40	2.90	6%
Russet	V11	14	1	1.99	3.41	0.01	6.20	1.71	49%
Russet	V12	23	2	3.17	5.90	0.00	11.80	1.86	31%
Russet	V13	5	1	3.76	5.32	0.00	8.00	1.41	40%
Russet	V14	27	2	0.80	1.96	0.00	4.20	2.46	11%
Russet	V15	29	1	1.72	2.94	0.03	7.20	1.71	12%
Wedge	DVCK_1	281	5	1.05	4.49	0.00	17.50	4.28	38%
Wedge	DVCK_2	369	1	1.04	4.15	0.00	23.00	4.01	1%
Wedge	DVCK_3	320	2	1.38	7.49	0.00	51.50	5.43	33%
Wedge	MJ_1	148	3	0.35	1.21	0.00	4.00	3.45	40%
Wedge	MJ_2	75	3	2.00	4.70	0.00	14.50	2.35	34%
Wedge	MJ_3	53	2	1.34	3.19	0.00	12.50	2.39	38%

Source: WRLG (2024)

14.7 Variography

Grade continuity analysis of gold mineralization was conducted using capped composites for each mineralized domain. Variogram analysis was conducted using Seequent's Leapfrog Edge™ software. Variogram parameters used for grade interpolation are provided in Table 14-11 and Table 14-12. Normal score variograms were generated for mineralized domains that contained limited assay composites, and for smaller mineralized domains no variograms could be generated due to insufficient data.

Table 14-11: Madsen deposit variogram parameters (by mineralized domain)

Mineralized Domain	Domain Code	Vulcan Rotations			Normalised Nugget	Structure 1					Structure 2				
						Normalised sill	Structure	Range (m)			Normalised sill	Structure	Range (m)		
		Bearing	Plunge	Dip				Major	Semi-major	Minor			Major	Semi-major	Minor
Austin_HG1_V1	111	300	49	48	0.30	0.49	Spherical	5.528	5	3.5	0.21	Spherical	43	30	10
Austin_HG1_V2	112	299	49	48	0.30	0.55	Spherical	5	7.5	4	0.15	Spherical	40	20	11.2
Austin_HG1_V3*	113	297	49	48	0.12	0.50	Spherical	2.5	2	5.5	0.38	Spherical	23	23	12
Austin_HG1_V4	114	29	59	-28	0.30	0.38	Spherical	10	3	7	0.32	Spherical	23	23	9
Austin_HG1_V5	115	347	64	8	0.30	0.60	Spherical	3.5	19	6.5	0.10	Spherical	30	22.8	7.8
Austin_HG2_V1*	121	9	61	-2	0.31	0.50	Spherical	5	3	8	0.19	Spherical	26	21	11
Austin_HG2_V2	122	57	47	-43	0.30	0.63	Spherical	11.4	9	6	0.07	Spherical	45	45	13
Austin_HG2_V3	123	343	62	21	0.25	0.59	Spherical	6	4	2.5	0.16	Spherical	35	25	8
Austin_HG2_V4	124	341	61	21	0.30	0.57	Spherical	4	7	4	0.13	Spherical	35	35	10
Austin_HG3_V1	131	340	62	25	0.30	0.51	Spherical	5	11	4	0.19	Spherical	30	30	10
Austin_HG3_V2	132	32	64	-23	0.30	0.53	Spherical	4.5	10	2.6	0.17	Spherical	35	35	7
Austin_HG3_V3	133	350	65	17	0.25	0.51	Spherical	4	3	4.5	0.24	Spherical	26	20	9
Austin_HG3_V4	134	332	62	30	0.25	0.50	Spherical	6	3	3	0.25	Spherical	33	15	7.05
Austin_HG3_V5	135	311	54	34	No Variogram										
Austin_HG4_V1	141	333	54	23	0.20	0.69	Spherical	6.5	6	6	0.11	Spherical	20	20	10
Austin_HG4_V2	142	66	30	-50	0.15	0.69	Spherical	4	4	4	0.16	Spherical	25	12	8
Austin_HG5_V1*	151	68	58	-45	0.31	0.62	Spherical	5	12	5	0.07	Spherical	35	35	16
McVeigh_HG1_3565_FW	211	356	75	4	No Variogram Model										
McVeigh_HG1_3565_HW	212	301	56	48	No Variogram Model										
McVeigh_HG1_BSZ	213	66	65	-68	0.25	0.57	Spherical	3.5	1.6	1	0.18	Spherical	15	10	7
McVeigh_HG1_FW	214	77	34	-63	0.30	0.59	Spherical	3	6	3.5	0.11	Spherical	15	15	7
McVeigh_HG1_FW2	215	355	66	11	No Variogram										
McVeigh_HG1_FW3	216	324	47	39	No Variogram										
McVeigh_HG1_HW	217	62	56	-47	0.30	0.24	Spherical	3	5	1.5	0.46	Spherical	20	13	5
McVeigh_HG1_HW2a	218	60	58	-45	0.30	0.60	Spherical	2	4	6	0.10	Spherical	20	15	10
McVeigh_HG1_HW2b*	219	16	67	-7	0.11	0.75	Spherical	5	2	4	0.15	Spherical	15	20	10
McVeigh_HG1_MAIN	220	79	36	-66	0.25	0.54	Spherical	7	3	4.5	0.21	Spherical	20	15	10
McVeigh_HG2_V1	221	274	29	69	0.30	0.47	Spherical	8	5	3	0.23	Spherical	34	30	15
McVeigh_HG2_V2	222	55	54	-58	0.20	0.48	Spherical	2.5	3	3	0.32	Spherical	25	15	5
McVeigh_HG2_V3*	223	280	43	72	0.09	0.59	Spherical	10	4	7	0.31	Spherical	30	15	8
McVeigh_HG2_V4	224	14	68	-27	No Variogram Model										
SouthAustin_A3_V1	301	318	69	54	No Variogram Model										
SouthAustin_A3_V2	302	339	74	34	No Variogram Model										
SouthAustin_A3_V3	303	330	61	32	No Variogram Model										
SouthAustin_FINGA	304	334	61	29	No Variogram Model										
SouthAustin_FW_V1	305	64	70	-48	0.20	0.63	Spherical	3.72	5	5	0.17	Spherical	15	15	7
SouthAustin_HG1_V1	311	96	19	-69	0.25	0.51	Spherical	2	1	3	0.25	Spherical	15	9	5
SouthAustin_HG1_V2	312	47	62	-33	0.30	0.63	Spherical	3.5	4	3.1	0.07	Spherical	15	15	8
SouthAustin_HG1_V3*	313	52	61	-37	0.21	0.56	Spherical	5.5	6	5	0.23	Spherical	28	28	9
SouthAustin_HG2_V1	321	301	51	68	0.25	0.61	Spherical	7	5	8	0.14	Spherical	30	30	12
SouthAustin_HG2_V2	322	323	63	39	0.25	0.53	Spherical	4.5	4	4	0.22	Spherical	30	22	9
SouthAustin_HG2_V3	323	297	47	68	0.30	0.61	Spherical	3	8	3	0.09	Spherical	15	15	9
SouthAustin_HG2_V4*	324	298	50	66	0.21	0.64	Spherical	5.5	3	3	0.15	Spherical	20	15	9
SouthAustin_HG3_V1	331	325	57	33	No Variogram Model										
SouthAustin_HG3_V2*	332	299	44	52	0.35	0.55	Spherical	14	2	6	0.10	Spherical	33	18	12
SouthAustin_HG3_V3	333	308	51	45	No Variogram Model										

*Denotes Normal Score Variogram Model

Source: WRLG (2024)

Table 14-12: Fork, Russet and Wedge deposits variogram parameters (by mineralized domain)

Deposit	Mineralized Domain	LF Directions			Normalised Nugget	Structure 1					Structure 2				
		Dip	Dip Azimuth	Pitch		Normalised sill	Structure	Range (m)			Normalised sill	Structure	Range (m)		
								Major	Semi-major	Minor			Major	Semi-major	Minor
Fork	FW	70	115	92	No Variogram Model										
Fork	HW	61	122	40	0.25	0.75	Spherical	40	20	15	No 2nd Strcuture				
Fork	NS	81	109	77	0.2	0.80	Spherical	40	30	15					
Russet	V1	34	134	93	No Variogram Model										
Russet	V2	38	133	10											
Russet	V3	67	36	121											
Russet	V4	39	141	85											
Russet	V6	31	116	67											
Russet	V7	49	134	56											
Russet	V8	44	97	99											
Russet	V10	37	129	50											
Russet	V11	35	121	72											
Russet	V12	38	157	52											
Russet	V14	36	105	141											
Russet	V15	57	33	105											
Wedge	DVCK_1	60	147	22	0.05	0.36	Spherical	25	24	17	0.59	Spherical	50	30	18
Wedge	DVCK_2	67	147	135	0.1	0.32	Spherical	20	27	22	0.58	Spherical	125	55	22
Wedge	DVCK_3	73	142	112	0.1	0.45	Spherical	17	36	10	0.44	Spherical	125	70	20
Wedge	MJ_1	63	146	115	No Variogram Model										
Wedge	MJ_2	68	145	169	0.1	0.90	Spherical	90	70	15	No 2nd Strcuture				
Wedge	MJ_3	70	147	143	No Variogram Model										

Source: WRLG (2024)

14.8 Block Model Configuration

Separate block models were generated for the Madsen, Fork, Russet and Wedge deposits, with block model configuration details summarized in Table 14-13. All block models were generated using a parent block size of 5 x 3 x 5 m and were sub-blocked to 0.5 x 0.5 x 0.5 m resolution for the Madsen deposit and 0.3125 x 0.375 x 0.3125 m resolution for the Fork, Russet and Wedge deposits for volumetric reporting. Grade interpolation was conducted at the parent block size of 5 x 3 x 5 m.

Table 14-13: Block model configuration parameters

Parameters	**Deposit**	**X (m)**	**Y (m)**	**Z (m)**
Parent Block Size	**Madsen**	5	3	5
Sub-Block Size		0.5	0.5	0.5
Base Point*		3500	1910	-100
Boundary Size		2900	1242	1700
Rotation		90°		
Parent Block Size	**Fork**	5	3	5
Sub-Block Size		0.3125	0.375	0.3125
Base Point**		434180	5645175	-220
Boundary Size		485	1230	660
Rotation		110°		
Parent Block Size	**Russet**	5	3	5
Sub-Block Size		0.3125	0.375	0.3125
Base Point**		433800	5647060	-15
Boundary Size		760	681	445
Rotation		90°		
Parent Block Size	**Wedge**	5	3	5
Sub-Block Size		0.3125	0.375	0.3125
Base Point**		433110	5643750	-310
Boundary Size		1515	639	760
Rotation		50°		
**Coordinates specified in local metric mine grid reference datum ("MMG")*				
***Coordinates specified in UTM NAD83 reference datum*				

Source: WRLG (2024)

14.9 Grade Estimation

Gold grades were interpolated into the block models predominantly using ordinary kriging ("OK") where sufficient sample density was available, or inverse distance ("ID2") for mineralized domains that had insufficient sample density to generate robust variograms. Grade estimation for each domain was conducted using multiple passes, with successively expanding search criteria in subsequent estimation passes. Locally varying anisotropy ("LVA") models were used for grade estimation within the Madsen, Fork and Wedge deposits, to align search orientations more accurately with the geometry of the mineralized domains. LVA was not used within the Russet deposit due to the generally simple geometries of the mineralized domains within the deposit.

Chip samples collected within production stopes and used for block grade estimation within the Madsen deposit were restricted to a 7.5 x 7.5 x 5 m search ellipse (i.e., Pass 1) to mitigate the area of influence of these samples to the immediate vicinity of the production stopes. All subsequent estimation passes used only drill hole assay samples to estimate block grades.

A summary of the estimation parameters used for grade interpolation in the Madsen, Fork, Russet and Wedge deposits is provided in Table 14-14 and Table 14-15. As noted in Table 14-15, a secondary outlier restriction technique (i.e., clamping) was used during the estimation process for the Russet V6 and Wedge DVCK_3 mineralized domains to mitigate the potential for over-estimation of grade within these domains. Gold grades were capped to the thresholds listed beyond the distance percentage of the search ellipse range.

Table 14-14: Madsen deposit estimation parameters

Zone	Mineralized Domain	Interpolant	Estimation Pass	LVA	Ellipsoid Ranges			Number of Samples			Data Source Used
					Maximum	Intermediate	Minimum	Minimum	Maximum	Max per Hole	
Austin	All domains excluding 135	OK	Pass 1	yes	15	15	7.5	8	12	3	DDH, Chips
			Pass 2	yes	30	30	10	8	12	3	DDH
			Pass 3	yes	60	60	15	4	6	3	
			Pass 4	yes	90	90	20	2	6	3	
	135	ID2	Pass 1	yes	15	15	7.5	8	12	3	DDH, Chips
			Pass 2	yes	30	30	10	8	12	3	DDH
			Pass 3	yes	60	60	15	4	6	3	
			Pass 4	yes	90	90	20	2	6	3	
McVeigh	211, 212, 215, 216, 224	ID2	Pass 1	yes	15	15	7.5	8	12	3	DDH, Chips
			Pass 2	yes	30	30	10	8	12	3	DDH
			Pass 3	yes	60	60	15	4	6	3	
			Pass 4	yes	90	90	20	2	6	3	
	217,219, 221, 222, 223	OK	Pass 1	yes	15	15	7.5	8	12	3	DDH, Chips
			Pass 2	yes	30	30	10	8	12	3	DDH
			Pass 3	yes	60	60	15	4	6	3	
			Pass 4	yes	90	90	20	2	6	3	
	213, 214, 218, 220,	OK	Pass 1	yes	7.5	7.5	5	12	24	3	DDH, Chips
			Pass 2	yes	15	15	7.5	8	12	3	DDH
			Pass 3	yes	30	30	10	8	12	3	
			Pass 4	yes	60	60	15	4	6	3	
			Pass 5	yes	90	90	20	2	6	3	
South Austin	301, 302, 303, 304, 331, 333	ID2	Pass 1	yes	15	15	7.5	8	12	3	DDH, Chips
			Pass 2	yes	30	30	10	8	12	3	DDH
			Pass 3	yes	60	60	15	4	6	3	
			Pass 4	yes	90	90	20	2	6	3	
	305, 312,313, 321, 322, 323, 324, 332,	OK	Pass 1	yes	15	15	7.5	8	12	3	DDH, Chips
			Pass 2	yes	30	30	10	8	12	3	DDH
			Pass 3	yes	60	60	15	4	6	3	
			Pass 4	yes	90	90	20	2	6	3	
	311	OK	Pass 1	yes	7.5	7.5	5	12	24	3	DDH, Chips
			Pass 2	yes	15	15	7.5	8	12	3	DDH
			Pass 3	yes	30	30	10	8	12	3	
			Pass 4	yes	60	60	15	4	6	3	
			Pass 5	yes	90	90	20	2	6	3	
8 Zone	810	OK	Pass 1	yes	7.5	7.5	5	12	24	3	DDH, Chips
			Pass 2	yes	15	15	7.5	8	12	3	DDH
			Pass 3	yes	30	30	10	8	12	3	
			Pass 4	yes	75	75	20	4	6	2	
	820	OK	Pass 1	yes	7.5	7.5	5	12	24	3	DDH, Chips
			Pass 2	yes	15	15	7.5	8	12	3	DDH
			Pass 3	yes	30	30	10	8	12	3	
			Pass 4	yes	60	60	20	4	6	2	
	830, 840	OK	Pass 1	yes	7.5	7.5	5	12	24	3	DDH, Chips
			Pass 2	yes	15	15	7.5	8	12	3	DDH
			Pass 3	yes	30	30	10	8	12	3	
			Pass 4	yes	60	60	20	4	6	2	
			Pass 5	yes	90	90	25	2	6	2	

Source: WRLG (2024)

Table 14-15: Fork, Russet and Wedge deposits estimation parameters

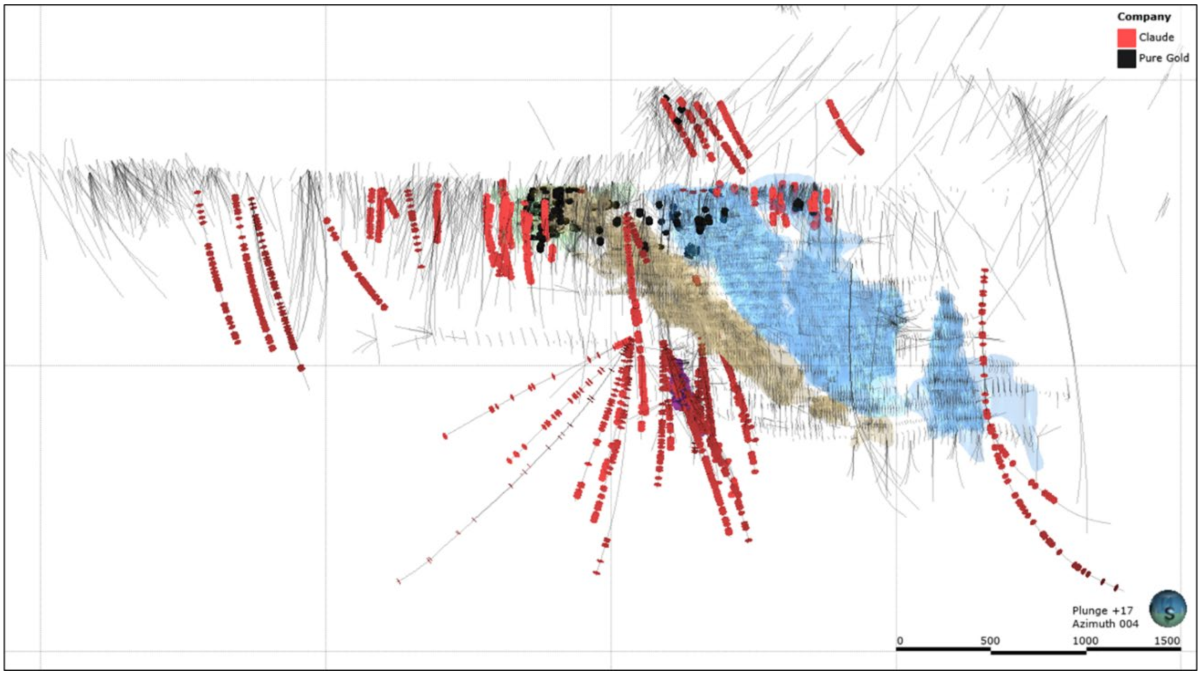
Deposit	Mineralized Domain	Interpolant	Estimation Pass	LVA	Ellipsoid Ranges			Number of Samples			Outlier Restrictions		
					Maximum	Intermediate	Minimum	Minimum	Maximum	Max per Hole	Method	Distance	Threshold
Fork	FW	ID2	Pass 1	yes	40	40	10	6	9	3	None		
		ID2	Pass 2	yes	80	80	20	6	9	3	None		
		ID2	Pass 3	yes	120	120	20	4	6	2	None		
		ID2	Pass 4	no	120	120	20	1	3	1	None		
Fork	HW, NS	OK	Pass 1	yes	40	20	15	6	9	3	None		
		OK	Pass 2	yes	80	40	20	6	9	3	None		
		OK	Pass 3	yes	120	60	20	4	6	2	None		
		OK	Pass 4	no	120	60	20	1	3	1	None		
Russet	V1, V2, V4, V10, V11, V12, V14, V15	ID2	Pass 1	no	50	30	20	6	9	3	None		
		ID2	Pass 2	no	100	60	25	6	9	3	None		
		ID2	Pass 3	no	100	60	25	4	6	2	None		
		ID2	Pass 4	no	100	60	25	1	3	1	None		
Russet	V3, V7, V8	ID2	Pass 1	no	50	30	20	6	9	3	None		
		ID2	Pass 2	no	100	60	25	6	9	3	None		
		ID2	Pass 3	no	150	90	30	4	6	2	None		
		ID2	Pass 4	no	150	90	30	1	3	1	None		
Russet	V6	ID2	Pass 1	no	50	30	20	6	9	3	None		
		ID2	Pass 2	no	100	60	25	6	9	3	Clamp	50%	8.7
		ID2	Pass 3	no	100	60	25	4	6	2	Clamp	50%	8.7
		ID2	Pass 4	no	100	60	25	1	3	1	Clamp	50%	8.7
Wedge	DVCK_1	OK	Pass 1	yes	50	30	18	6	9	3	None		
		OK	Pass 2	yes	75	45	25	6	9	3	None		
		OK	Pass 3	yes	100	60	25	4	6	2	None		
		OK	Pass 4	no	100	60	25	1	3	1	None		
Wedge	DVCK_2	OK	Pass 1	yes	63	26	15	6	9	3	None		
		OK	Pass 2	yes	125	55	22	6	9	3	None		
		OK	Pass 3	yes	188	83	33	4	6	2	None		
		OK	Pass 4	no	188	83	33	1	3	1	None		
Wedge	DVCK_3	OK	Pass 1	yes	63	35	15	6	9	3	None		
		OK	Pass 2	yes	125	70	20	6	9	3	Clamp	50%	13.5
		OK	Pass 3	yes	188	105	25	4	6	2	Clamp	34%	13.5
		OK	Pass 4	no	188	105	25	1	3	1	Clamp	34%	13.5
Wedge	MJ_1, MJ_3	ID2	Pass 1	yes	50	30	20	6	9	3	None		
		ID2	Pass 2	yes	100	60	25	6	9	3	None		
		ID2	Pass 3	yes	150	90	30	4	6	2	None		
		ID2	Pass 4	no	150	90	30	1	3	1	None		
Wedge	MJ_2	OK	Pass 1	yes	45	35	15	6	9	3	None		
		OK	Pass 2	yes	90	70	15	6	9	3	None		
		OK	Pass 3	yes	135	105	20	4	6	2	None		
		OK	Pass 4	no	135	105	20	1	3	1	None		

Source: WRLG (2024)

14.10 Density

A total of 5,924 specific gravity (SG) measurements have been collected across the mine deposit area, of which 2,636 were collected by Claude Resources (circa 2012) and an additional 3,288 have been collected by Pure Gold (Figure 14-7). Both data sets of SG measurements were obtained from drill core using the water displacement method.

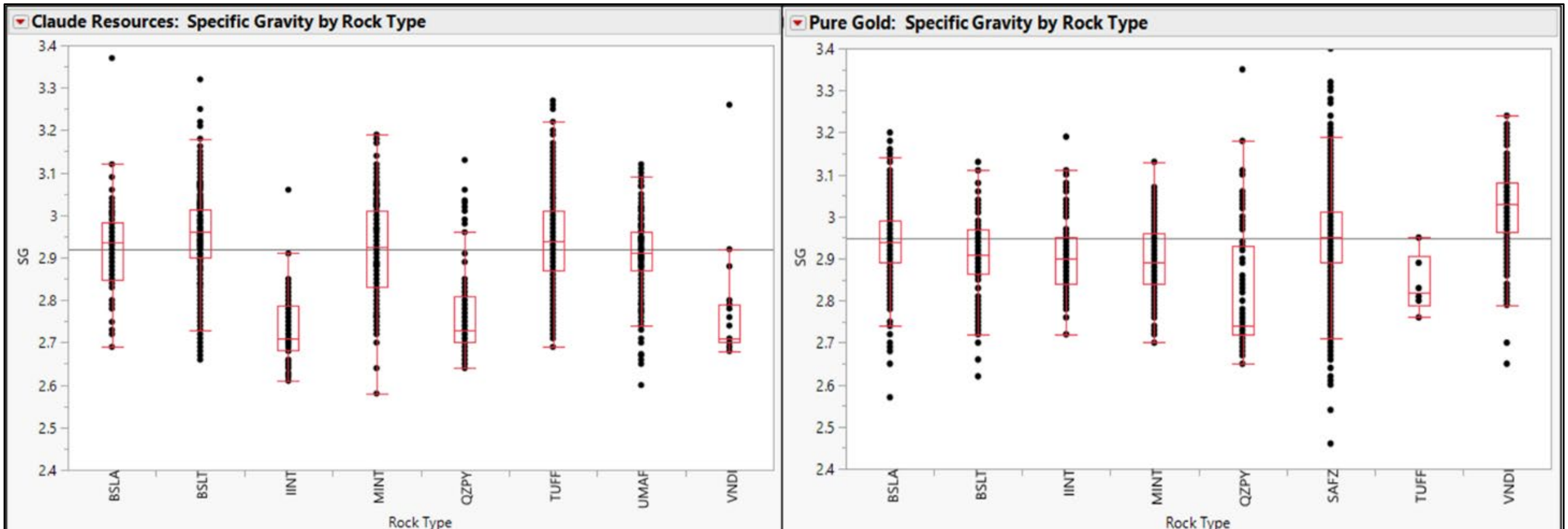
Figure 14-7: Specific gravity sample locations by company



Source: WRLG (2024)

Summary statistics of SG measurements grouped by company data set and major rock type are provided in Figure 14-8. Principal host lithologies for mineralization are identified as “BSLA” and “SAFZ”. A total of 591 SG measurements are located within the mineralized wireframes of the mine deposits (577 located within the Madsen deposit) and have an average SG value of 2.94. Therefore, a global density value of 2.94 g/cm^3 is used for the mineral resource estimates for all deposits.

Figure 14-8: Summary of specific gravity by rock type



Company	Rock Type	Number of Samples	Mean	StDev	Minimum	Median	Maximum	Company	Rock Type	Number of Samples	Mean	StDev	Minimum	Median	Maximum
Claude	BSLA	78	2.92	0.11	2.69	2.94	3.37	Pure Gold	BSLA	468	2.94	0.10	2.57	2.94	3.85
Claude	BSLT	712	2.95	0.11	2.10	2.96	3.32	Pure Gold	BSLT	113	2.90	0.10	2.62	2.91	3.13
Claude	IINT	64	2.73	0.08	2.61	2.71	3.06	Pure Gold	IINT	107	2.91	0.09	2.72	2.90	3.19
Claude	MINT	404	2.92	0.11	2.58	2.93	3.19	Pure Gold	MINT	141	2.90	0.09	2.70	2.89	3.13
Claude	QZPY	87	2.77	0.11	2.64	2.73	3.13	Pure Gold	QZPY	81	2.82	0.14	2.65	2.74	3.35
Claude	TUFF	539	2.94	0.10	2.69	2.94	3.27	Pure Gold	SAFZ	1770	2.95	0.10	2.37	2.95	3.52
Claude	UMAF	458	2.91	0.08	2.60	2.91	3.12	Pure Gold	TUFF	6	2.84	0.07	2.76	2.82	2.95
Claude	VNDI	19	2.77	0.14	2.68	2.71	3.26	Pure Gold	VNDI	244	3.02	0.10	2.65	3.03	3.24

Source: WRLG (2024)

14.11 Model Validation

Block model validation was conducted using multiple techniques, including;

- Visual inspection of estimated block grades relative to composite grades
- Swath plot analysis of grade profiles between ordinary kriged (OK) and nearest-neighbour (NN) block estimates
- Statistical comparison of global average estimated block grades and declustered composite grades, per mineralization domain
- Change of support analysis

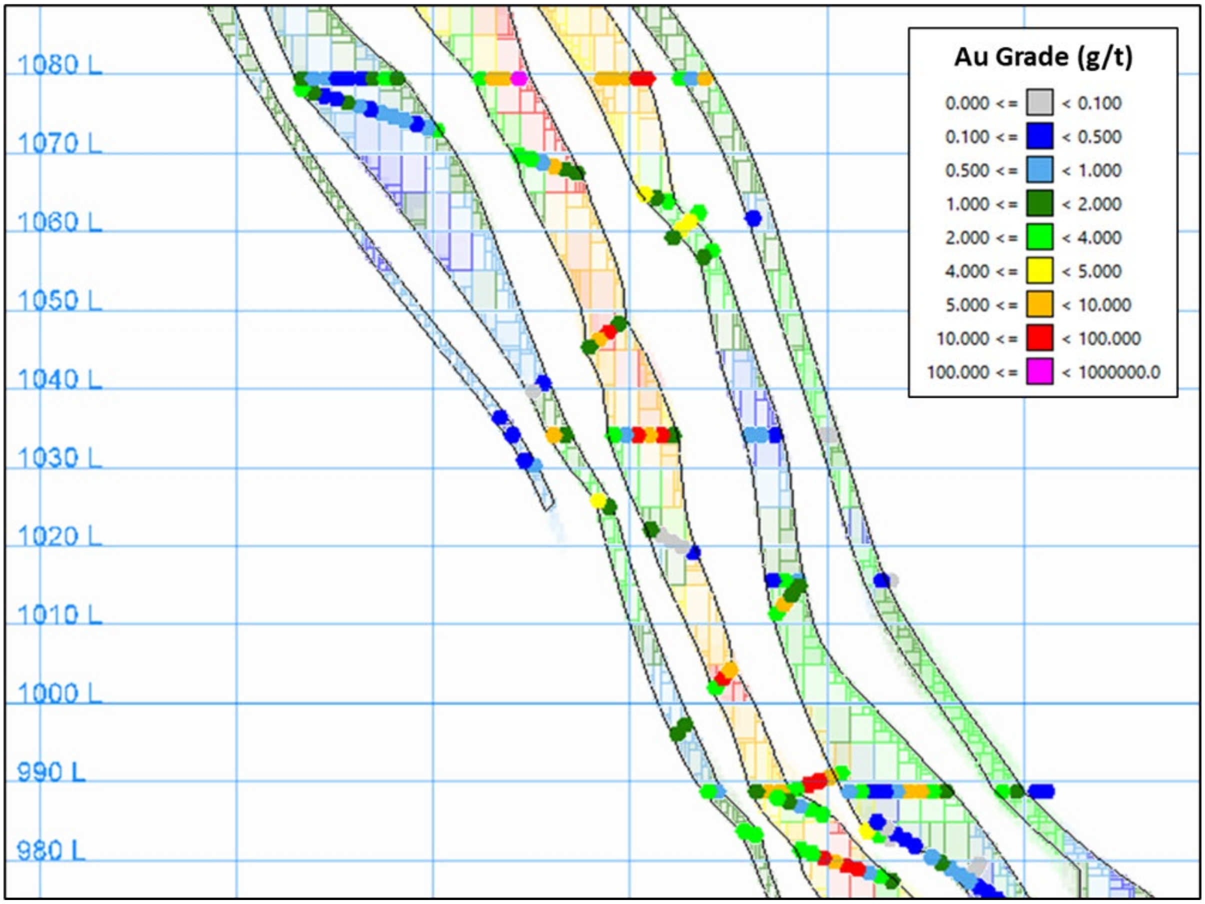
Cross-sectional comparisons of interpolated block grades versus sample composites for the Madsen deposit are provided in Figure 14-9 and Figure 14-10. Reasonable correlation between the block estimates and composite data can be observed.

Swath plot comparisons of interpolated Au grades from the OK and NN models for several large, mineralized domains within the Madsen deposit are provided in Figure 14-11 and Figure 14-12. Reasonable correlation between the OK and NN models is observed on these plots, with the OK models showing a greater level of smoothing in the grade profile, which is to be expected for this estimation technique.

Figure 14-13 provides a comparison of global average estimated Au grades between OK and NN models by deposit-zone and mineralized domain. Generally, there is good agreement between the OK and NN estimated Au grades. For some volumetrically smaller mineralized domains, such as Madsen-8 Zone domain 820, a more pronounced discrepancy is observed between the OK and NN models associated with search distance restrictions placed on historical production chip samples within the OK estimation workflow. For mineralized domains within the Russet deposit, discrepancies between the OK and NN models are associated with low numbers of sample composites within the domains and the smoothing effect of the OK estimation technique.

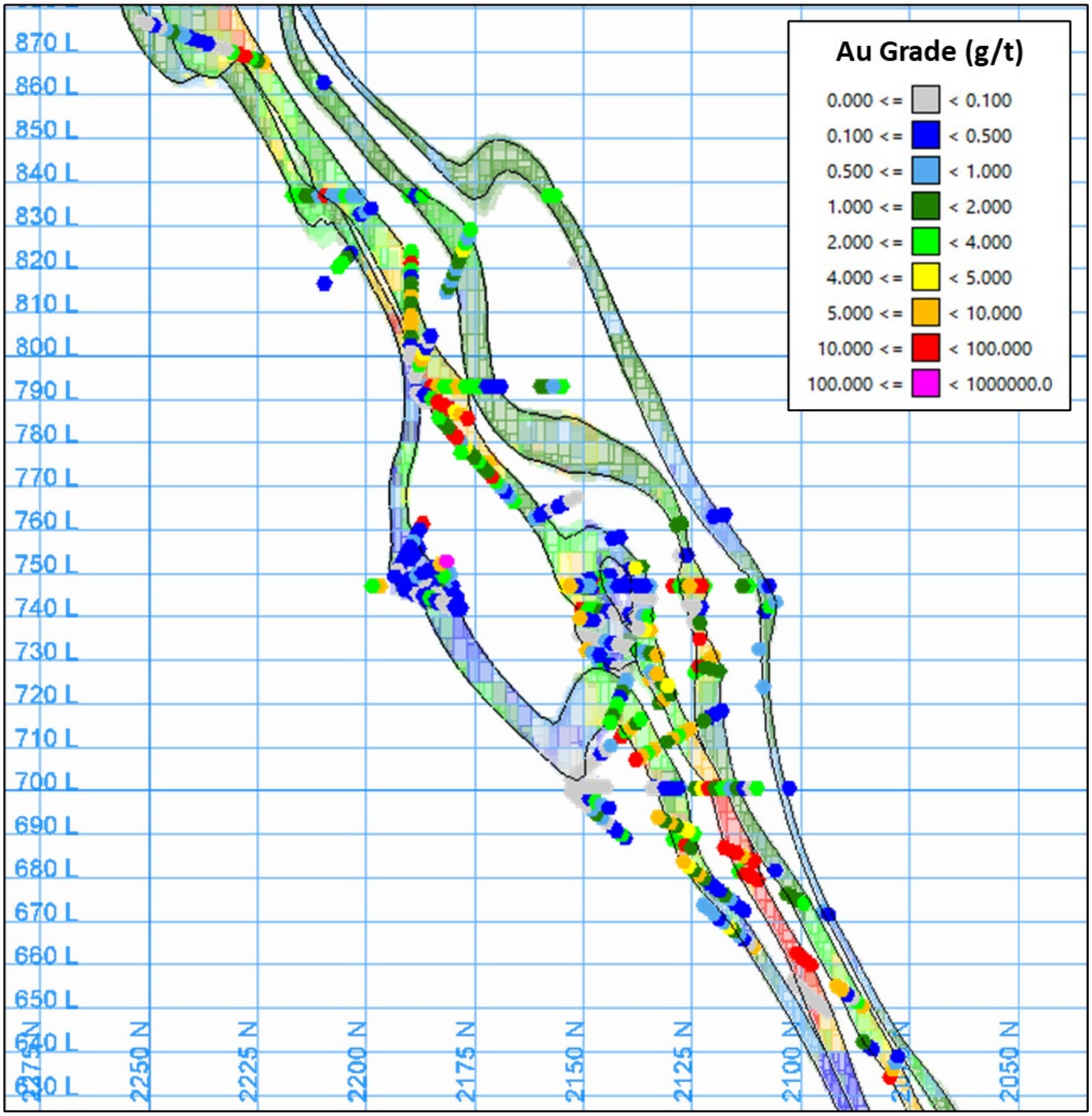
Change of support analysis was conducted on several larger zones within the Madsen deposit to evaluate the volume-variance relationship (i.e., “smoothing”) between the declustered sample composites and estimated block grades. A discrete Gaussian model was used to generate grade distributions of the declustered sample composites at the same support as the block model, and then plotted relative to the block model grade distributions on Q-Q and grade-tonnage plots as shown in Figure 14-14 and Figure 14-5 for Madsen-Austin Zone domains 111 and 123, respectively. In both examples there is good correlation between the block support adjusted, declustered sample composites (i.e., Declus+COS) and block model (BM) grade distributions, thus indicating an appropriate amount of smoothing in the block model grade estimates.

Figure 14-9: Cross-section comparison of interpolated Au grades vs Au composites in the Madsen Deposit along Mine Grid 4900 Easting (looking east)



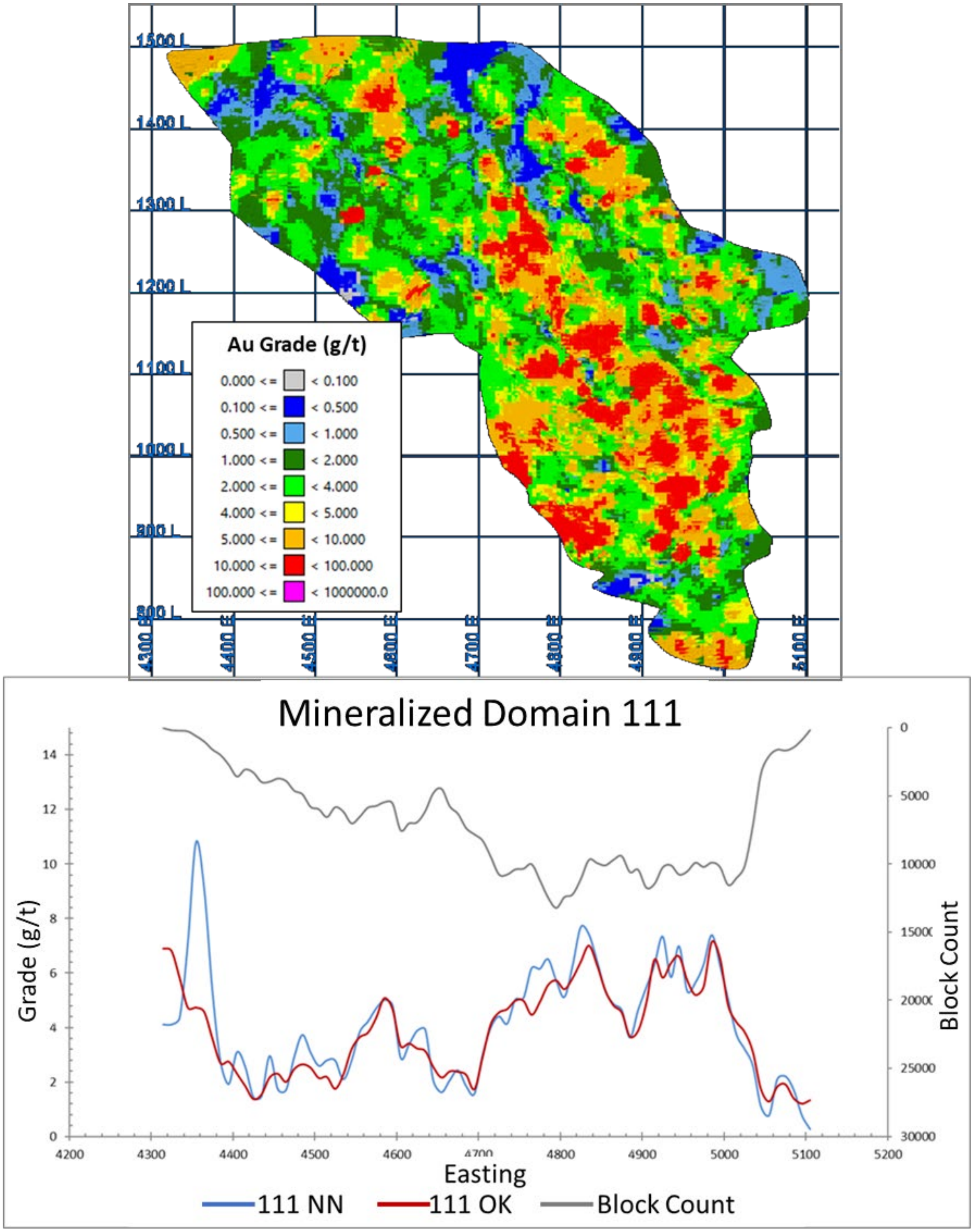
Source: WRLG (2024)

Figure 14-10: Cross-section comparison of interpolated Au grades vs Au composites in the Madsen Deposit along Mine Grid 5400 Easting (looking east)



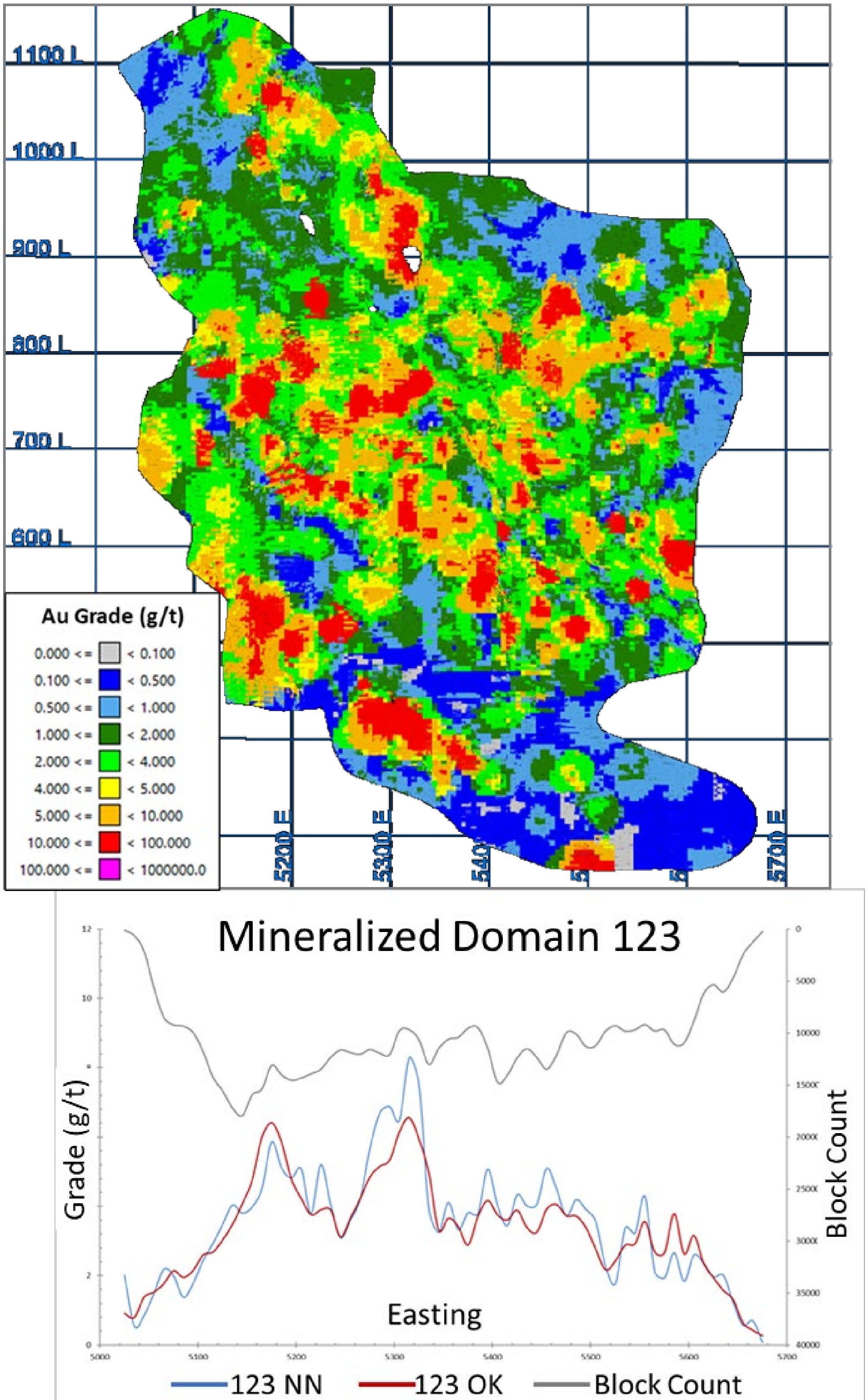
Source: WRLG (2024)

Figure 14-11: Madsen Deposit – Austin Zone Mineralized Domain 111 swath plot comparison of Au (g/t) grade for OK and NN block model estimates



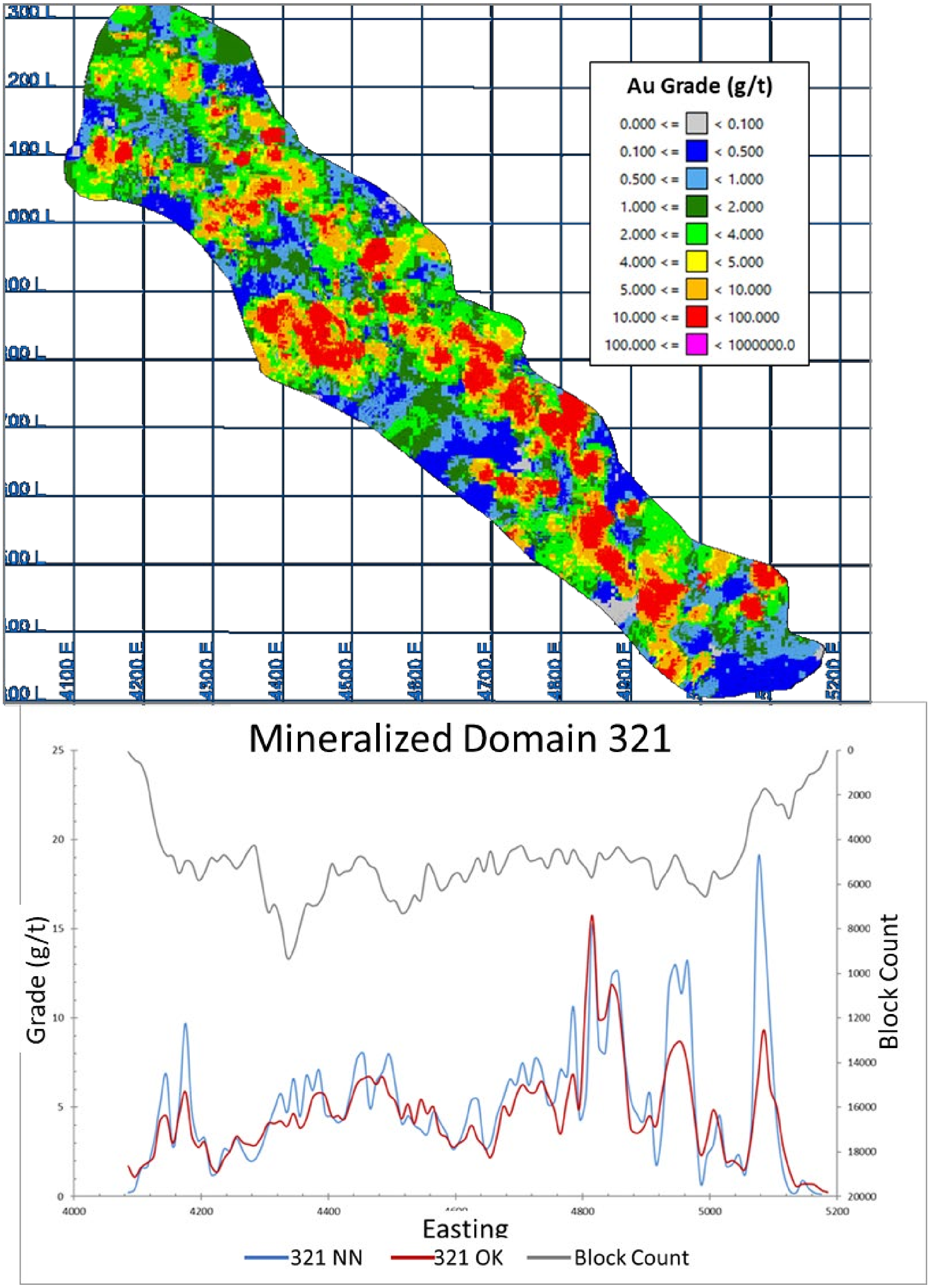
Source: WRLG (2024)

Figure 14-12: Madsen Deposit – Austin Zone Mineralized Domain 123 swath plot comparison of Au (g/t) grade for OK and NN block model estimates



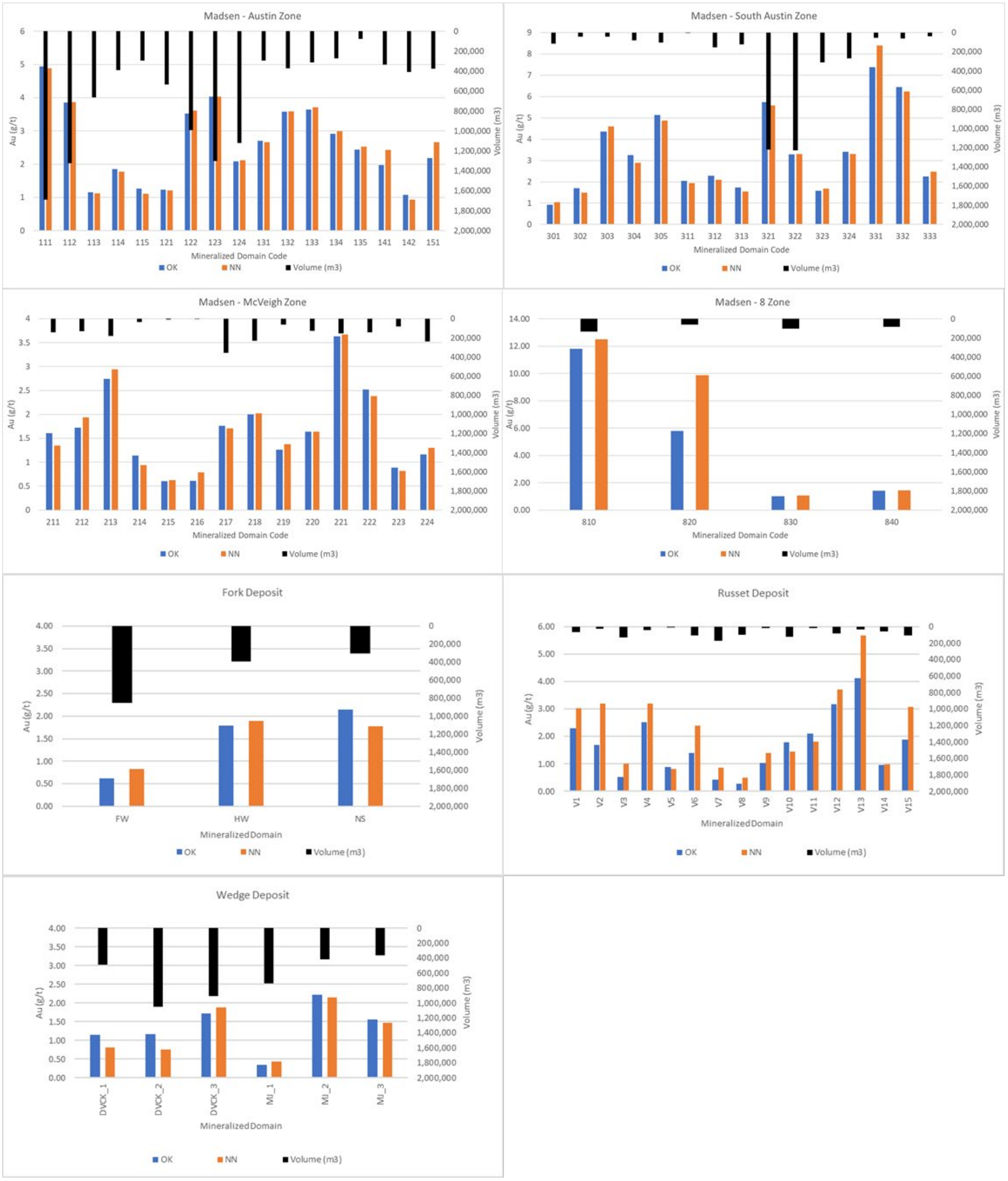
Source: WRLG (2024)

Figure 14-13: Madsen Deposit – South Austin Zone Mineralized Domain 321 swath plot comparison of Au (g/t) grade for OK and NN block model estimates



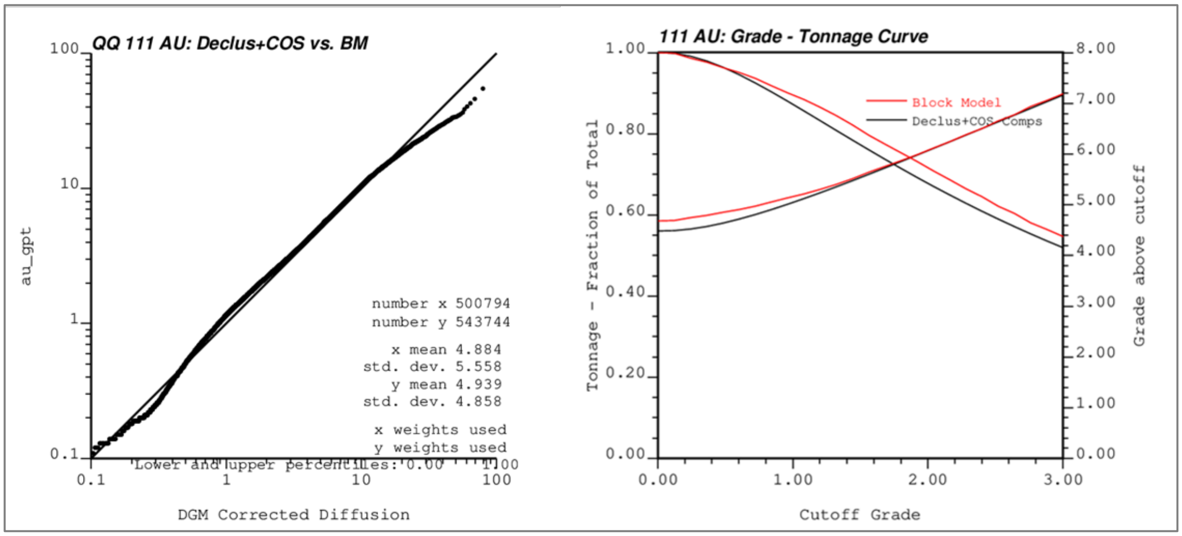
Source: WRLG (2024)

Figure 14-14: Global average grade (Au g/t) comparison between ordinary kriged (OK) and nearest-neighbour (NN) estimated grades by deposit-zone and mineralized domain



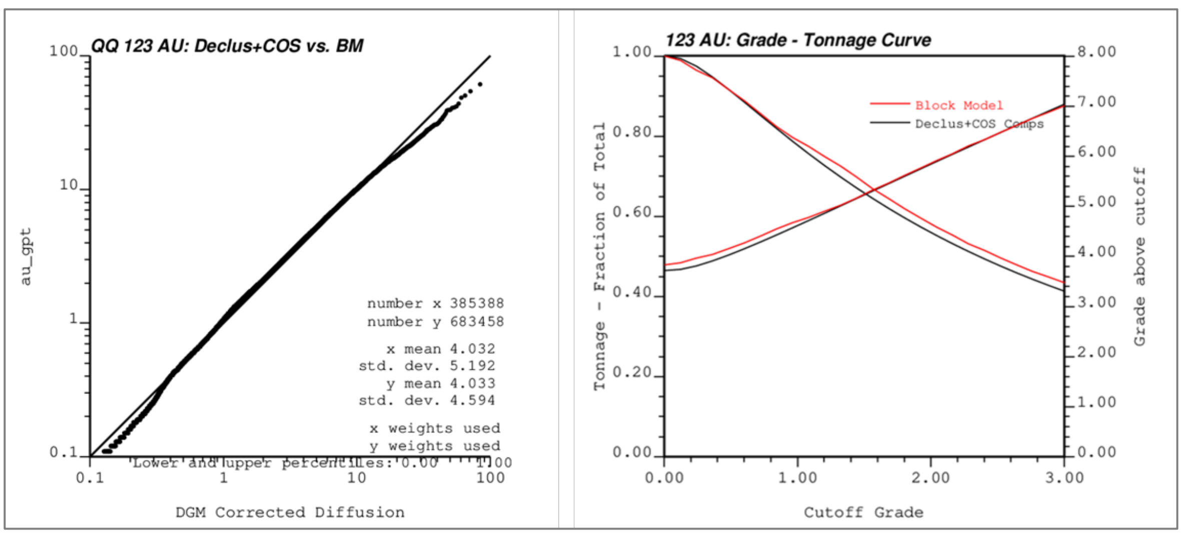
Source: WRLG (2024)

Figure 14-15: Madsen Deposit – Austin Zone Mineralized Domain 111 Change of Support Analysis



Source: WRLG (2024)

Figure 14-16: Madsen Deposit – Austin Zone Mineralized Domain 123 Change of Support Analysis



Source: WRLG (2024)

14.12 Mineral Resource Classification

Block model quantities and grade estimates for the mine were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Mr. Cliff Revering, P.Eng., an independent qualified person for the purpose of National Instrument 43-101.

Mineral resource classification is typically a subjective concept, and industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate semi-contiguous areas of similar resource categories.

Mr. Cliff Revering is satisfied that the geological models honour the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. Mineral resource classification criteria considered the following components:

- Quality of the data used to support mineral resource estimation
- Confidence in the geological interpretation of the mineralized zones
- Average drill hole spacing within the deposits
- Estimation parameters including the number of drill holes and assay composites used to estimate a block

Madsen deposit blocks were classified within the Indicated resource category in areas where the average drill hole spacing was 25 m or less, and blocks were estimated with a minimum of two drill holes and at least four assay composites. All other estimated blocks were classified in the Inferred resource category. McVeigh Zone mineralized domains 211, 212 and 216 were classified entirely as Inferred mineral resources due to limited data available within these mineralized domains.

Wedge deposit blocks were classified within the Indicated resource category in areas where the average drill hole spacing was 40 m or less, and blocks were estimated with a minimum of two drill holes and at least three assay composites. Additional Wedge deposit blocks were classified as Inferred mineral resources provided they did not satisfy Indicated resource criteria and the average drill hole spacing was 80 m or less. The MJ1 mineralized domain was classified entirely as an Inferred resource due to limited available data.

Fork deposit blocks within mineralized domains HW and NS were classified within the Indicated resource category in areas where the average drill hole spacing was 40 m or less, and blocks were estimated with a minimum of two drill holes and at least three assay composites. Blocks within mineralized domain FW and all other blocks within domains HW and NS were classified as Inferred mineral resources.

Russet deposit blocks within mineralized domains V1, V6, V10 and V12 were classified within the Indicated resource category in areas where the average drill hole spacing was 40 m or less, and blocks were estimated with a minimum of two drill holes and at least three assay composites. Blocks within these domains were classified as Inferred mineral resources provided they did not satisfy Indicated resource criteria and the average drill hole spacing was 80 m or less. Mineralized domains V2, V3, V4, V7, V8, V11, V14 and V15 were classified as Inferred mineral resources due to limited available data within these domains. All other mineralized domains were excluded from the classified mineral resource.

The classification criteria used for the Fork, Russet and Wedge deposits reflect the reduced geological complexity associated with the interpreted mineralized domain geometries within these deposits compared to Madsen.

14.13 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (2019) define a mineral resource as:

"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. SRK considers that portions of the mine are amenable for underground extraction.

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" within the Madsen deposit, SRK used a stope optimizer and reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from the underground mine. The mining parameters were selected based on review of the existing underground mine operation and are summarized in Table 14-16. The reader is cautioned that the results of this analysis are used solely for the purpose of testing the "reasonable prospects for eventual economic extraction" by an appropriate mining method and do not represent an attempt to estimate mineral reserves. The results of this analysis are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

Table 14-16: Assumptions used for defining reasonable prospects for economic extraction

Parameter	Value	Unit
Gold Price	$1,800	US$ per ounce
Foreign Exchange Rate	1.30	C$/US$
Stope Heights	3, 6 & 12	metres
Stope Length	10	metres
Minimum Stope Width	2	metres
Maximum Stope Width	20	metres
UG Mining Costs	$108.60	C$ per tonne mined
Process Costs	$73.30	C$ per tonne of feed
G&A Costs	$54.10	C$ per tonne of feed
Mining Recovery	95%	percent
Process Recovery	95%	percent
Cut-off Grade	3.38	g/t Au

Source: SRK (2022)

MSO shapes were not generated for the Fork, Russet and Wedge deposits, however the same resource cut-off grade determined for the Madsen deposit was applied for resource reporting purposes. In general, the distribution of classified blocks above a cut-off grade of 3.38 g/t are located with multiple contiguous to semi-contiguous zones within each mineralized domain.

The mineral resource statement for the PureGold (Madsen) Mine deposits is provided in Table 14-17, with an effective date of December 31, 2021. The mineral resources have been adjusted to reflect the removal of all historical and recent production to the end of December 2021 and are reported as undiluted mineral resources at the stated cut-off grade of 3.38 g/t Au.

The mining activity from the effective date of this mineral resource until the closure of the PureGold (Madsen) Mine has been deemed immaterial. Based on the mining records, 164,604 tonnes of ore at 3.8 g/t grade were processed, resulting in the production and sale of 20,301 ounces of gold. This production figure is not considered significant for the purpose of this report and the mining activity during the period from January 1, 2022 to the mine closure on October 24, 2022 will not have a material impact on the mineral resource estimates presented in this report.

Since the effective date of the MRE, additional diamond drilling was conducted until the mine closure on October 24, 2022. A total of 688 drill holes and 54,122 m of drilling was completed in 2022. An additional 205 drill holes and 19,872 m of drilling was completed by WRLG between October 1, 2023 and May 15, 2024. Based on a review of the results of this drilling, Mr. Cliff Revering has determined that the information obtained will not have a material impact on the MRE presented in this report.

Table 14-17: Mineral Resource Statement, PureGold (Madsen) Mine, Red Lake, Ontario, effective date December 31, 2021

Classification	Deposit – Zone	Tonnes	Gold Grade (g/t)	Total Gold (troy oz)
Indicated	Madsen – Austin	4,147,000	6.9	914,200
	Madsen – South Austin	1,696,000	8.7	474,600
	Madsen – McVeigh	388,700	6.4	79,800
	Madsen – 8 Zone	152,000	18	87,700
	Fork	123,800	5.3	20,900
	Russet	88,700	6.9	19,700
	Wedge	313,700	5.6	56,100
	Total Indicated	**6,909,900**	**7.4**	**1,653,000**
Inferred	Madsen – Austin	504,800	6.5	104,900
	Madsen – South Austin	114,100	8.7	31,800
	Madsen – McVeigh	64,600	6.9	14,300
	Madsen – 8 Zone	38,700	14.6	18,200
	Fork	298,200	5.2	49,500
	Russet	367,800	5.8	68,800
	Wedge	431,100	5.7	78,700
	Total Inferred	**1,819,300**	**6.3**	**366,200**

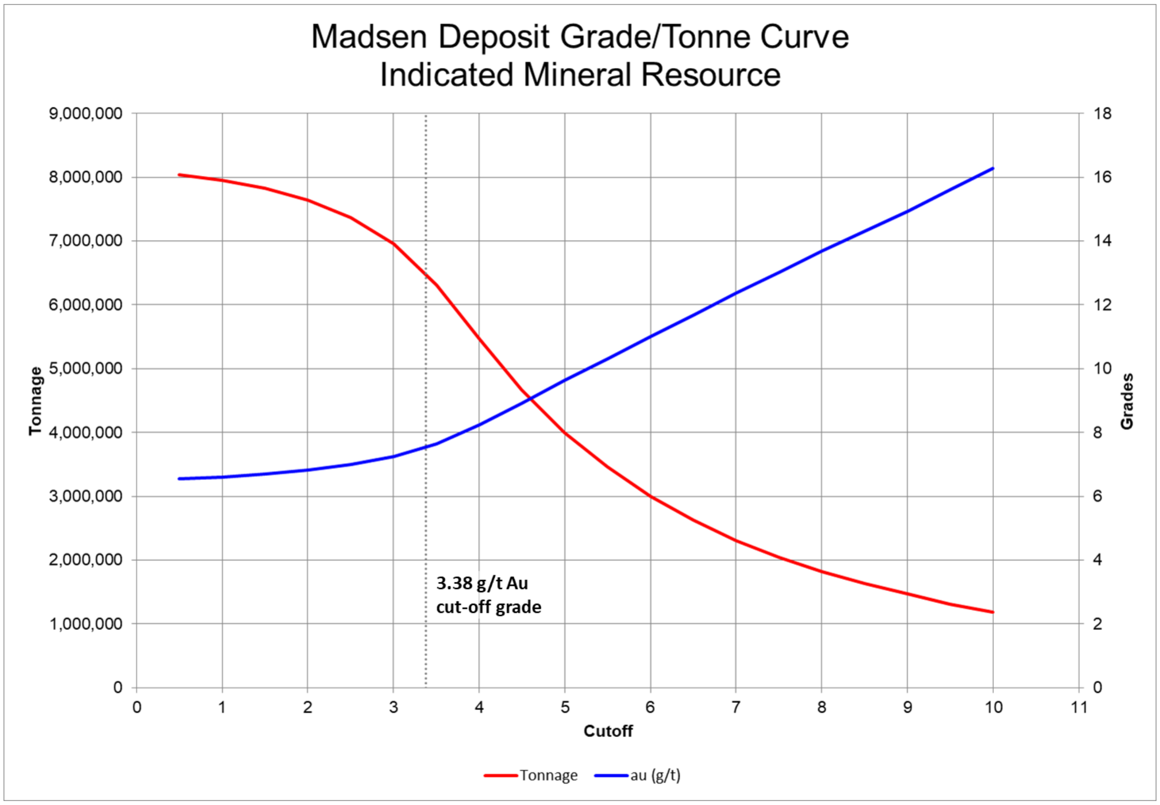
Notes:

1) Mineral Resources estimated in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Cliff Revering, P.Eng., Qualified Person
2) Mineral resources are not mineral reserves and do not have demonstrated economic viability.
3) Mineral resources are reported at a cut-off grade of 3.38 g/t Au
4) Mineral resources are reported using a gold price of US$1,800/oz
5) Excludes depletion of mining activity during the period from January 1, 2022 to the mine closure on October 24, 2022 as it has been deemed immaterial and not relevant for the purpose of this report.
6) All figures have been rounded to reflect the relative accuracy of the estimate

14.14 Grade Sensitivity Analysis

The Madsen Mine mineral resources are sensitive to the selection of the reporting cut-off grade. Figure 14-17 provides a grade-tonnage curve for the Madsen deposit Indicated mineral resource estimate.

Figure 14-17: Grade Tonnage Curve for the Madsen Deposit (Indicated mineral resource)



Source: WRLG (2024)

14.15 Reconciliation to Previous Mineral Resource Estimate

The previous mineral resource estimate for the PureGold (Madsen) Mine was prepared by Ginto Consulting Inc. with an effective date of February 5, 2019 (see Makarenko et al., 2019). A comparison of the current and previous mineral resource estimates is provided in Table 14-18.

Comparison between the two mineral resource estimates shows similar tonnages for Indicated and Inferred resources, however with a reduction in the average gold grade and contained gold content within the current mineral resource estimate. This reduction in average grade and contained gold content is reflective of the operational experience gained through active mining since 2020 and changes incorporated into the 2021 MRE update, including a revised geological interpretation, additional drilling and production data, revised grade capping analysis and estimation parameters, revised mineral resource classification criteria and a lower cut-off grade for mineral resource reporting.

Table 14-18: Summary comparison of the current and previous mineral resource estimates

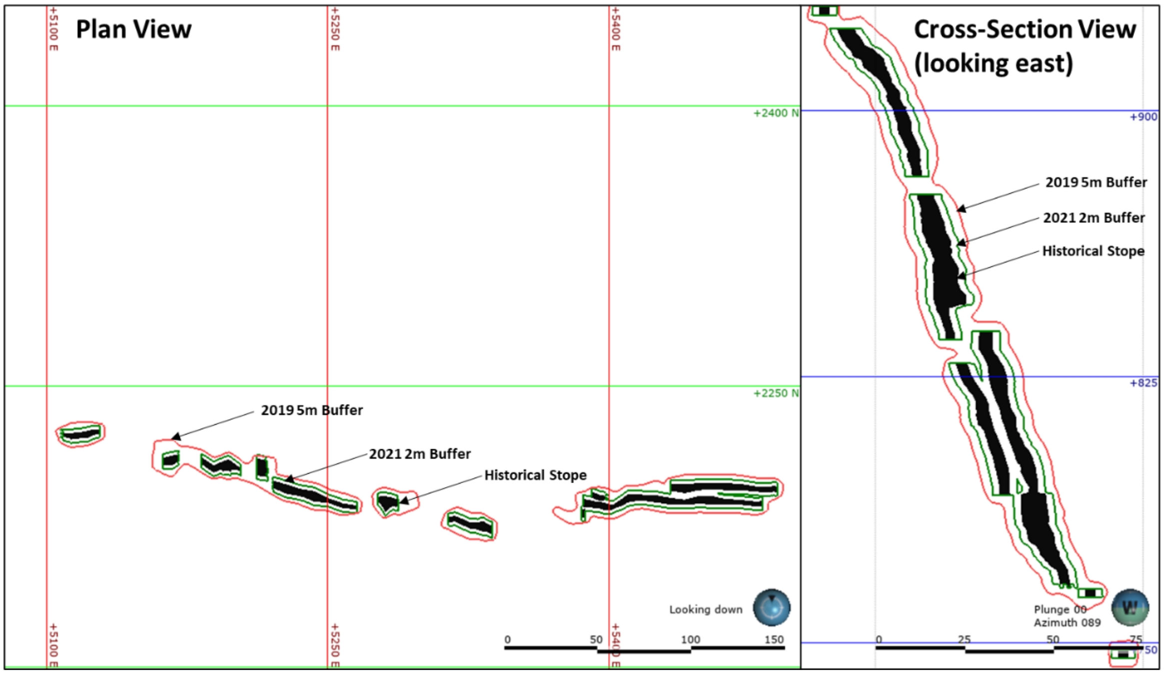
Indicated Mineral Resource	December 31, 2021 MRE	February 5, 2019 MRE
Tonnes	6,909,900	7,196,000
Gold Grade (g/t)	7.4	8.9
Gold Troy Ounces	1,653,000	2,063,000

Inferred Mineral Resource	December 31, 2021 MRE	February 5, 2019 MRE
Tonnes	1,819,300	1,880,000
Gold Grade (g/t)	6.3	7.7
Gold Troy Ounces	366,200	467,000

Source: WRLG (2024)

Additional changes incorporated into the 2021 MRE update included modification of the buffer or exclusion zone around historical stopes from a 5 m buffer (in all directions) to a 2 m buffer applied to the hanging and footwall for historical stopes (Figure 14-18), and inclusion of a 20 m crown pillar. Test mining adjacent to historical stopes during 2020 and 2021 by Pure Gold demonstrated that successful extraction of mineralization adjacent to historical stopes (along strike and down dip) was feasible and supported the removal of an exclusion zone along strike and down dip of historical stopes.

Figure 14-18: Comparison of 2019 and 2021 buffer zones around historical stopes



Source: WRLG (2024)

15 Mineral Reserve Estimates

15.1 Summary

The Madsen Mine has been mined extensively from the mid-1930s to the mid-1970s with more than 8.9 Mt of ore being extracted and many kilometres of track drifts, raises and shafts to contend with. Much of the higher grade material in the mineral resource model is remnants contained in sill pillars and/or immediately adjacent to the historic shrinkage stopes. The mineral reserves are contained within a mining area with a strike length of 1,250 m with a 1,200 m vertical extent with a 60° plunge to the SSE. The mineral reserves follow the trend of the historic shrinkage stopes. The strike length of the historic development is 2,000 m with a 1,300 m vertical extent. This presents unique challenges and opportunities for modern mining operations using trackless, mechanized equipment.

This section summarizes the key assumptions, parameters, and methods used in the preparation of the Mineral Reserve estimate for the Madsen Mine Pre-Feasibility Study (PFS). The Mineral Reserve Statement presented herein has been prepared for public disclosure according to CIM Best Practice Guidelines (November 2019) and reported as diluted tonnes delivered to the mill.

As there are no Measured mineral resources included in the 2021 mineral resource model the PFS mine design is based upon, there are no Proven mineral reserves included in the mineral reserve estimate.

15.2 Key Assumptions, Parameters and Methods

15.2.1 Access and Mining Methods

Access to the mine is currently by ramp from surface with the historic Madsen #2 Shaft being reconditioned for hoisting from 12 Level. A new hoisting shaft is envisioned to hoist from deeper in the mine. Trackless versions of the longitudinal retreat longhole (LH) and mechanized cut and fill (MCF) mining methods were selected.

15.2.2 Cut-off Grades

Stopes were designed using a LH cut-off grade (COG) of 4.30 gpt and a MCF cut-off grade of 5.28 gpt, using a gold price of US$1,680/oz, with calculations as shown in Table 15-1. These break-even cut-off grades (BECOG) were applied as a diluted COG as the stope optimization included dilution for both LH and MCF stopes.

Table 15-1: Break-Even cut-off grade parameters and calculation

Break-Even Cut-off Grade Estimate		**LH COG**	**MCF COG**
	unit	**Estimate**	**Estimate**
Mining	CD$/t	$186.38	$253.94
Milling	C$/t	$51.08	$51.08
G&A	C$/t	$49.88	$49.88
Site total operating cost/tonne milled	**C$/t**	**$287.34**	**$354.90**
Gold price - 3 yr avg as of Feb 15/2013	US$/oz	$1,680	$1,680
Exchange Rate	C$:US$	1.31	1.31
Au Payable	(%)	99.995%	99.995%
Au Refining	US$/oz	$0.00	$0.00
Royalty	(%)	0.0%	0.0%
Value of Au in dore	US$/oz	$1,680	$1,680
Value of Au in dore	US$/gram	54.01	54.01
Process recovery	(%)	95.0%	95.0%
Value of Au in plant feed	US$/gram	51.31	51.31
Value of Au in plant feed	C$/gram	67.22	67.22
Plant feed grade (diluted BECOG)	**Au gpt**	**4.30**	**5.28**

Source: SRK (2024)

Mining costs were benchmarked from an operating mine in Ontario. Mill costs and G&A were based upon historic site costs. Historic costs were classified as fixed/variable and adjusted to account for inflation, differences in mining rate and other factors.

Incremental cut-off grades (ICOG) were also estimated for LH and MCF stopes using the same method (see Table 15-2). The difference is that the milling costs and G&A costs included are only the variable portion. Fixed costs are assumed to be covered by the stope shapes generated using the BECOG.

The incremental stope shapes are used to fill in gaps between stope shapes or extend strike length, improving continuity for LH stoping areas. For incremental stopes shapes to be considered for inclusion in the Mineral Reserve Estimate, they must be continuous with a set of economic stopes that will pay for the capital costs to access the mining area and require minimal operating development.

Additionally, low grade development material was included in the Mineral Reserve Estimate if the grade exceeded 1.0 gpt.

Incremental material should not displace economic material in the mill. If the mill is full either adjust cut-off grades or stockpile incremental material. The mill name plate capacity is 1,089 tpd versus the mine production schedule of 800 tpd. The variance between nameplate capacity and production schedule is because of permitting constraints, as the mill is only permitted under ECA regulations for 800 tpd.

Table 15-2: Incremental cut-off grade parameters and calculations

Incremental Cut-off Grade Estimate		COG	COG
	unit	Estimate	Estimate
Mining	C$/t	$186.38	$253.94
Milling	C$/t	$24.49	$24.49
G&A	C$/t	$15.23	$15.23
Incremental operating cost/tonne milled	**C$/t**	**$226.10**	**$293.66**
Gold price - 3 yr avg as of Feb 15/2013	US$/oz	$1,680	$1,680
Exchange Rate	C$:US$	1.31	1.31
Au Payable	(%)	99.995%	99.995%
Au Refining	US$/oz	$0.00	$0.00
Royalty	(%)	0.0%	0.0%
Value of Au in dore	US$/oz	$1,680	$1,680
Value of Au in dore	US$/gram	54.01	54.01
Process recovery	(%)	95.0%	95.0%
Value of Au in plant feed	US$/gram	51.31	51.31
Value of Au in plant feed	C$/gram	67.22	67.22
Plant feed grade (diluted ICOG)	**Au gpt**	**3.40**	**4.40**

Source: SRK (2024)

15.3 Mineral Reserve Estimation Process

Stope optimization was performed using the Deswik Stope Optimizer (DSO) software, with the mine design and schedule being completed using Deswik.CAD and Deswik.Sched.

15.3.1 Treatment of Inferred Mineral Resource

Best practice is to treat Inferred mineral resources as waste for purposes of stope optimization. There are a number of methods to achieve this. In this case, it was decided to create a new grade field in the block model with the Inferred mineral resource grades set to 0.00 gpt Au for use as the optimization field.

When running DSO, small amounts of Inferred mineral resources may be included within the final shapes due to boundary effects. When setting up the Deswik.Sched file, the final stope shapes are re-evaluated against the original block model grade field, which then have the modifying factors applied to determine the estimated final milled tonnes and head grades.

The total Inferred mineral resources included within the final Mineral Reserve Estimate is 1.37%.

15.3.2 Mine Design Process

The following optimization process was followed to generate the final stope shapes upon which the PFS mine plan and financial analysis is based.

- Review block model and add new grade field for stope optimization, set grade of all Inferred mineral resource blocks to zero. There are no Measured mineral resources or Unclassified material in the block model.
- LH mine design criteria applied in DSO, including:
    - Sub-level spacings, nominally 20 m, adjusted to match historic track drift elevations using gradient polylines
    - Both sub-shapes and full shapes generated
    - Sub-stope lengths of 10 m along strike
    - Minimum mining width 1.6 m undiluted, 2.2 m diluted
    - Dilution included as 0.3 m ELOS applied to hangingwall and footwall
- MCF mine design criteria applied in DSO, including:
    - Fixed height of 3.5 m
    - Fixed 3 m sub-stope lengths
    - Minimum mining width 3.0 m undiluted, 3.8 m diluted
    - Dilution included as 0.4 m ELOS (average) applied to hangingwall and footwall using the variable overbreak or slough (VOS) feature in DSO
- LH stope sub-shapes were generated in two passes:
    - Pass 1 generated LH stopes with diluted BECOG ≥ 4.3 gpt,
    - Pass 2 generated incremental LH stopes with diluted BECOG from 3.4 to 4.29 gpt
- MCF sub-shapes were then generated with a diluted BECOG ≥ 5.28 gpt
- No incremental MCF sub-shapes were generated
- The two passes of LH and the MCF sub-shapes sets were then combined, cleaned and edited to create a clean set of sub-shapes. Cleaning included:
    - Removing any overlaps
    - Removing Pass 2 incremental LH sub-shapes not adjacent to Pass 1 stope shapes
    - Removing material that did not meet geotechnical criteria for mining method
    - Removing MCF cuts (multiple sub-shapes) with a volume less than 100 m^3
- Preliminary economic checks were then done on all zones and mining areas to ensure;
    - Mining area had sufficient value to pay for the required capital development to access each mining area as these are very spread out
    - Ensure MCF mining areas immediately adjacent to historic open voids could pay the capital cost to backfill these voids with cemented backfill
- A series of reviews and further refinements to the final set of stope sub-shapes were conducted including:

  - Geotechnical review
  - Zone by zone review and manual edits to stope sub-shapes to account for offset to historic stopes, rill lines, etc.
  - Replacing LH sub-shapes with MCF in some areas to improve economics
  - Ensuring all sub-shapes depleted properly to account for mining since the mineral resource model was finalized (Dec 31, 2021)
- The final set of LH stope sub-shapes was then compiled into full stope shapes based on geotechnical guidelines for LH strike length and MCF stope sub-shapes were compiled into MCF stope cuts
- The development design was then completed and brought into Deswik.Sched along with the finalized stope set to create the PFS mine plan
- Deswik.Sched setup includes calculations for:
  - Additional dilution added to LH stopes based on depth
  - Accounting for mining recovery
  - Classifying development as ore or waste based on the 1.0 gpt incremental development cut-off grade
- A series of reviews and checks were completed on the final mine design and schedule in Deswik before finalizing the Mineral Reserve Estimate.

15.4 Modifying Factors

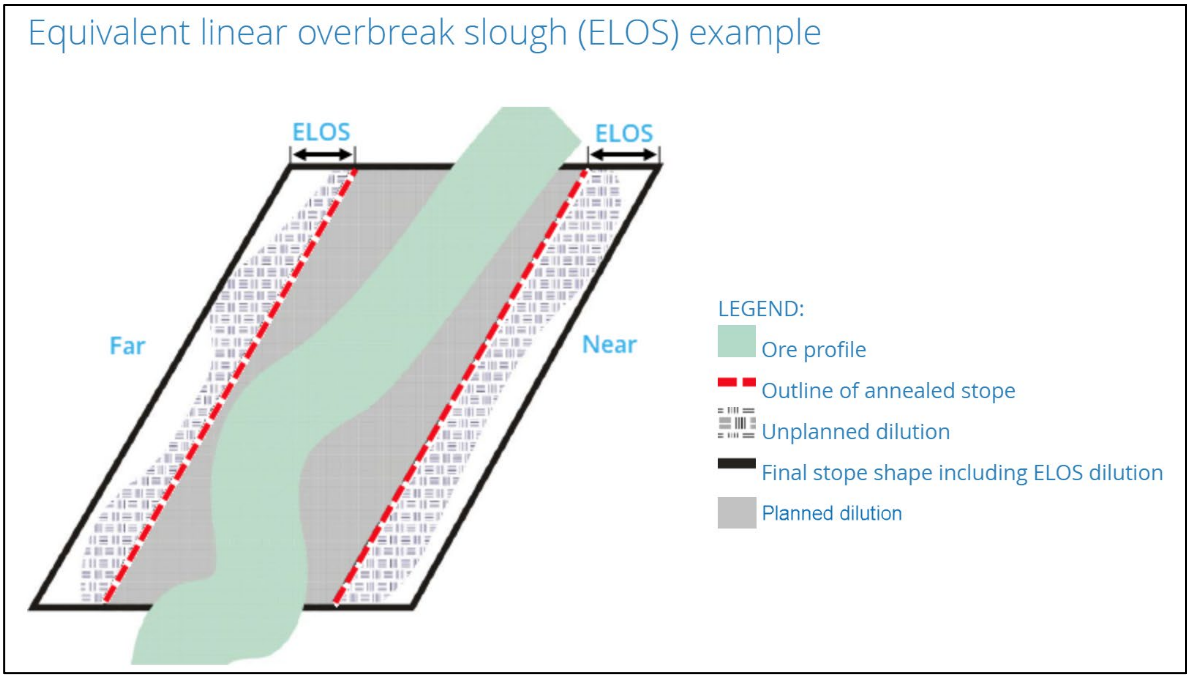
Mineral Resources were converted to Mineral Reserves by applying the appropriate modifying factors, as described herein, to the final DSO shapes created during the mine design process.

The diluted tonnage and grade of each DSO shape is based on the resource block models. All mineral reserve estimates are expressed as "dry” tonnes (i.e., no moisture) and are based on the density values stored in the block model.

Internal (planned) dilution is included within the DSO shapes as is a base external (unplanned) dilution; both are at block model grades. Any portion of the DSO shape outside the block model would have the default density (2.94) and zero grade applied.

The base external dilution for LH was included in DSO as a 0.3 m ELOS added to both the hangingwall and footwall of the shape as shown in Figure 15-1.

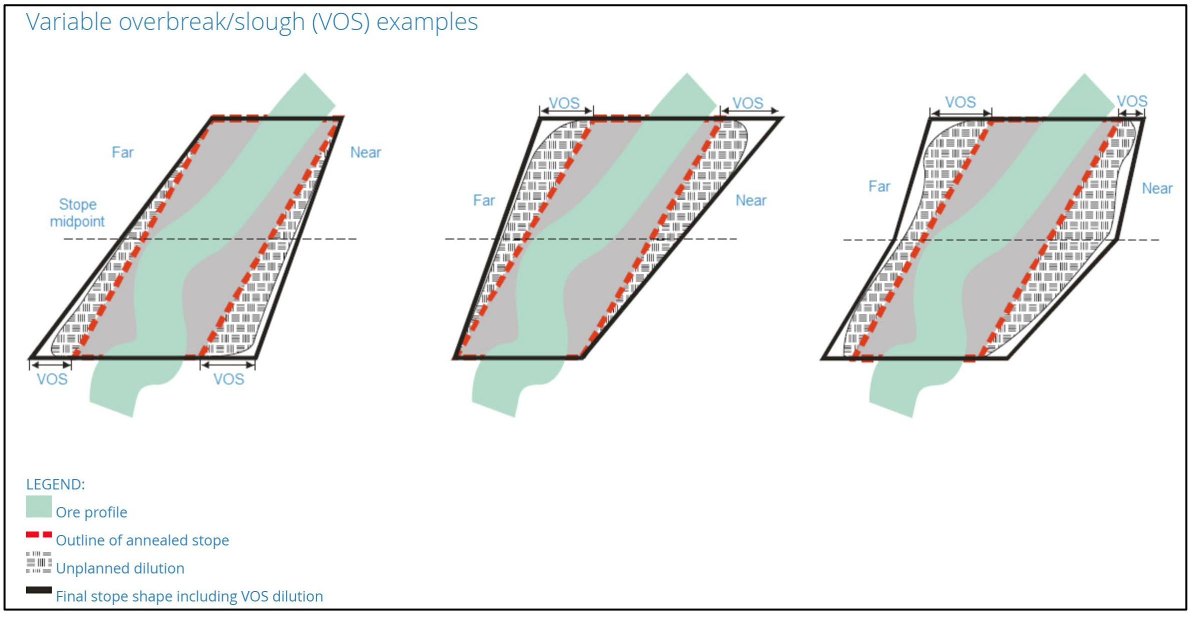
Figure 15-1: Example of Equivalent Linear Overbreak Sough (ELOS)



Source: Deswik (2023) [Source Link]

The base external dilution for MCF was included in DSO as an average 0.4 m VOS added to both the hangingwall and footwall of the shape as shown in Figure 15-2.

Figure 15-2: Example of Variable Overbreak Slough (VOS)



Source: Deswik (2023) [Source Link]

Based on recommendations by the geotechnical team, additional external dilution was added for LH stopes only, in Deswik.Sched based on depth by adding additional ELOS at zero grade. The total ELOS is shown in Table 15-3.

Table 15-3: Total ELOS by depth

Depth	Maximum Hydraulic Radius Allowed	Total ELOS
Surface to 500 m depth	9.4	0.6
500 m to 750 m depth	8.6	0.8
Below 750 m depth	6.7	1.2

Source: SRK (2024)

The mining recovery factors represents how much of the diluted stope material will reach the mill based on mining method.

Table 15-4 lists the external dilution by source and mining recovery for each mining method.

Table 15-4: Modifying Factors by Mining Method

Mining Method	Source	External Dilution	Mining Recovery
LH	Included in DSO	27%	
	Additional with depth	9%	
	Total	**35%**	**95%**
MCF	Included in DSO	24%	
	Backfill Dilution	5%	
	Total	**29%**	**97%**

Source: SRK (2024)

15.5 Mineral Reserve Estimate

This Mineral Reserve Estimate with an effective date of June 30, 2024 has been prepared in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Stephen Taylor, PEng., Principal Mining Engineer with SRK Consulting (Canada) Inc. in his role as Independent Qualified Person.

Table 15-5: Mineral Reserve Statement, Madsen Mine, Red Lake, Ontario, effective date June 30, 2024

Classification	Deposit - Zone	Tonnes (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Probable	Madsen - Austin	778	7.37	184
	Madsen - South Austin	861	8.21	227
	Madsen - McVeigh	66	7.37	16
	Madsen - 8 Zone	118	13.38	51
Proven + Probable		**1,823**	**8.16**	**478**

Notes:

1. Mineral Reserves estimated in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Stephen Taylor, P.Eng., Qualified Person.
2. Longhole stope cut-off grade of 4.30 gpt Au based on an estimated operating cost of C$287.34/t including mining, plant and G&A. The mining cost component was benchmarked based on an operating mine in Ontario.
3. Mechanized Cut and Fill stope cut-off grade of 5.28 gpt Au based on an estimated operating cost of C$354.90/t including mining, plant and G&A.
4. Mineral reserve estimates based on a gold price of US$1680/oz and an exchange rate of 1.31 C$/US$.
5. Incremental development cut-off grade of 1 gpt Au.
6. A small amount of incremental longhole tonnes were included at a cut-off grade of not less than 3.4 gpt Au, these must be immediately adjacent to economic stopes that will pay for the capital to access area.

15.6 Relevant Factors

SRK is not aware of any relevant factors that would materially change the current Mineral Reserve Estimate.

16 Mining Methods

16.1 Introduction

The PFS life-of-mine (LOM) plan for the Madsen Mine is based on the resource model completed by SRK Consulting (Canada) Inc. as detailed in Section 14 of this report. At present, the Madsen Mine has historic workings covering a 2.3 km strike length to 1300 m depth.

A significant portion of the higher-grade mineral resources is located in close proximity to the historic workings and can be considered remnant mining targets. These include mineral resources left in place as pillars, not considered mineable at the time, below the cut-off grade at the time, or simply not recognized as ore at the time. Using modern MCF mining methods and ground support techniques, a portion of these remnants can be safely extracted today.

There are also mineral resources in unmined areas, though these tend to be lower grade than the core zones extracted historically.

A number of underground mining methods were considered to deal with different challenges encountered in the various zones. A partial list of the methods considered includes:

- Longitudinal Retreat Longhole with ramp access, a common mining method
- Longitudinal Retreat Longhole from captive sub-level with raise access similar to methods used at the Dome mine (Historic track mine - Newmont) in the 1990s to mid-2000s
- Sub-Level Open Stoping with ramp access or Alimak raise access similar to methods used at Eagle River mine (Wesdome)
- Alimak Longhole with horizontal drilling similar to method used at Seabee mine (SSR Mining) in the late-2010s
- Mechanized Cut and Fill with ramp access (a common mining method)
- Captive Cut and Fill with raise access similar to methods used at the Dome mine (Newmont) in the 1960s to mid-2000s

The mining methods selected for ore extraction at Madsen were narrowed down to:

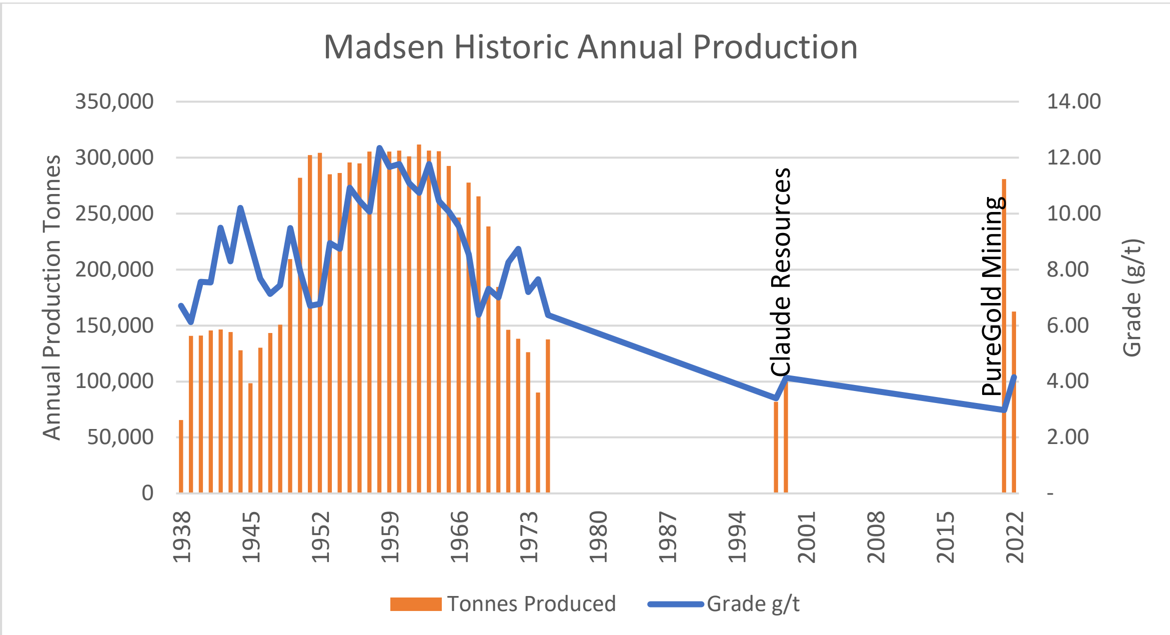
- Longitudinal Retreat Longhole with ramp access (LH)
- Mechanized Cut and Fill (MCF) with ramp access
- Mechanized Drift and Fill with ramp access for the 8 Zone

Mining method selection was driven primarily by mineralization geometry and continuity, selectivity of method, ability to mechanize the method, proximity to historic workings and anticipated ground conditions. For design purposes, LH was the preferred method of extraction followed by MCF.

16.2 Historic Mining Context

The Madsen Mine is a past producing mine in the Red Lake mining camp, which has a long history of underground narrow vein gold mining. The site was mined extensively from the mid-1930s to the mid-1970s as a shaft access mine with track haulage, using a shrinkage mining method. Figure 16-1 summarizes the historical production data available for the Madsen mine.

Figure 16-1: Madsen Mine historic production statistics



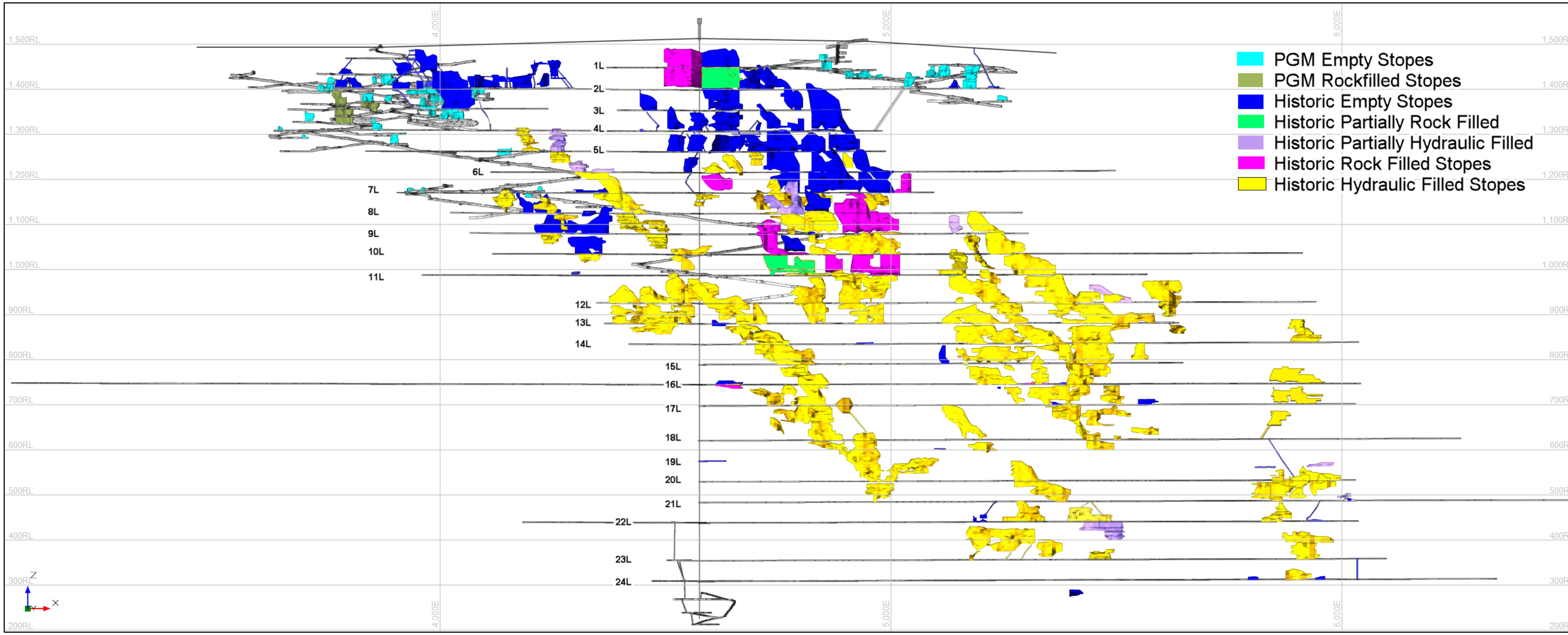
Source: Geology Ontario (2025)

The historic stopes were either not backfilled, backfilled with waste rock or backfilled with an uncemented hydraulic backfill.

Figure 16-2 shows a long section of the mine with the various backfill types identified. The more recent stopes mined by Pure Gold were either left open or rockfilled. These tend to be LH or Uppers stopes. Pure Gold built and commissioned a hydraulic backfill plant, but did not put it to use.

Many of the larger historic stopes located in the upper Austin zone above 8 Level were left empty, with some being partially or completely rockfilled. Most of the historic stopes below 11 Level were backfilled with uncemented hydraulic backfill.

Figure 16-2: Long section of Madsen Mine historic workings (looking north)



Source: SRK (2025)

16.3 Remnant Mining in a Flooded Historic Mine

Remnant mining can be defined as the recovery of mineralized material previously left behind for various reasons during the normal course of mining operations. In essence, the PFS Life of Mine Plan is wrapping a modern mechanized mine around a historic track mine where the core of the good mineralization has already been extracted.

Mining remnants in a historic mine that was flooded for decades poses a number of challenges including:

- General lack of good information on historic flooded areas as most of the detailed stope plans and sections are no longer available
- As-built 3D stope models were created from sectional data, the exact profile of the stope between sections is uncertain
- Dealing with open stopes that may be unstable or already caved
- Dewatering stopes backfilled with saturated uncemented hydraulic backfill without mobilizing the backfill (the mine is currently dewatered to between 14 Level and 15 Level)
- Dealing with falls of ground and other surprises while slashing the historic track drifts
- Dealing with unexpected drifts and raises, the as-builds are certainly missing many raises
- Historic sill pillars unlikely to be thick enough to slash 3 mW x 3 mH track drifts to 5 mW x 5 mH trackless drifts, need to drive bypass drifts around these areas
- The amount of ramp and lateral development required to redevelop the mine and access new mining areas across the long strike length generates more waste rock than ore, adding to the materials handling challenge and capital development costs

16.4 Ground Conditions and Stability

A geotechnical assessment was completed for the Madsen Mine using available data, which included an exploration drillhole database, geological underground mapping, and Madsen’s ground control management plan. This was supplemented by reports from JDS, Mining Plus, and other consultants associated with site feasibility studies and aspects of the project’s technical rock mechanics. This data was used to characterize the geotechnical conditions of the rock and support the underground mine and infrastructure design, an evaluation of geotechnical design domains, and the development of geotechnical design guidelines. These guidelines included excavation design parameters, estimates of dilution, as well as ground support requirements. The various elements of the geotechnical evaluation and findings are discussed in more detail in the following sections.

Madsen is a steeply dipping, narrow vein deposit. Foliation is prevalent in most of the rock units within the deposit and generally follows the dip of the orebody. It is less visible in the Strongly Altered Foliated Zone (SAFZ) unit and does not exist in the mafic dykes. The main part of the deposit consists of the Austin, South Austin, McVeigh and A3 vein systems, with generally similar rock mass characteristics. The 8 Zone sits apart from the other vein systems and is generally strongly altered. The rock mass here is generally weaker, characterized by soft, squeezing ground.

16.4.1 Data Sources and Quality

The main sources of data that SRK used to complete the assessment on the Madsen Mine are introduced and discussed in the following subsections.

2018 Feasibility Study

A Feasibility Study (FS) was completed in 2018 for the Madsen mine site and compiled by JDS Energy & Mining Inc. (JDS). The geotechnical assessment was presented as a series of slides by JDS. Minimal additional rock mechanics data has been collected since this study, thus the FS data was, in many cases, used as baseline data and supplemented or re-evaluated.

Historic Mine Model

SRK has reviewed and incorporated the historical data provided by WRLG, where relevant to the scope of this report. Mine development, stope solids, and backfill types were provided for the existing underground workings. The geometry and backfill of the historic workings were considered in recommendations associated with pillars and proximity of planned new workings to historic workings.

Exploration Drillhole Database

WRLG has provided data for exploration drilling at the Madsen site. Logging has mainly focused on geological logging, with the only geotechnical parameters collected being recovery and rock quality designation (RQD). There are varying levels of detail in the core logging data depending on the year that holes were drilled and the purpose of the drillhole at the time. Core photos were also provided for visual review of the diamond drill core.

Most of the core recoveries are over 95% (over 90% of the intervals had a core recovery percentage greater than 95%), which correlates well with the understanding of the rock mass. Recorded RQDs were also generally high, with over 80% of the intervals having an RQD percentage of over 80%.

Geological Mapping Data

Madsen provided SRK with geological face mapping data. The vast majority of this data were measurements of foliation, with a minimal quantity of joints and dykes. The average foliation orientations collected in diamond drill core appear to have similar dips but vary in dip direction by approximately 15 degrees. It is possible that foliation changes slightly in orientation with depth, which the core data captures, but near-surface crown pillar study data and face mapping data from the dewatered section of the historic workings does not.

Ground Control Management Plan

Madsen provided a ground control management plan (GCMP) dated March 2022. The GCMP outlined geotechnical data used as input into ground support selection and mine planning. This was used as a baseline for some of the recommendations in this report, in conjunction with historic site experience.

Additional Data

One useful additional source of data was a spreadsheet that tracked stope reconciliations comparing planned stopes and their associated tonnes with cavity monitoring scan (CMS) tonnes obtained following the completion of each stope. This data was available for stopes mined between January 2021 and September 2022. While mining methods used in the extraction of these stopes may not align with future mining, the comparison gave a sense of variability in recovery and overbreak. SRK saw a large range of variability in actual stope volumes compared to planned stope volumes with some stopes overbreaking and underbreaking up to 60% from plan. Fewer than 60% of the stopes mined during this period varied in volume (underbreak or overbreak) by less than 20%.

Data Limitations

Minimal rock mechanics data has been collected at Madsen since the completion of the 2018 FS study. SRK observed some data gaps that should be addressed as the project is advanced. These data gaps include the following:

- Spatial extents of geotechnical underground mapping and RQD data are very limited.
    - No discontinuity orientation data available for 8 Zone
    - Minimal DDH RQD or other geotechnical data coverage in the lower mine area (below approximately 500 m from surface)
- Structural orientation joint data is limited to three televiewed holes and limited underground near-surface mapping. With the rock mass being generally competent, joint data is considered to be important for kinematic stability assessments. The main zone is expected to be structurally controlled but there has been a minimal focus on collecting structural data.
- Quality of data in the 2018 FS study requires confirmation (RQD data QAQC required, PLT data not on standard core, lab testing on core from only two holes, Q' data from split core).
    - Rock strength data should be re-evaluated. There is concern that some lithological units may not have been sufficiently sampled and that samples may not be representative of the extents of the deposit. There may also be sampling bias in the weaker units, with more competent intervals being selected for sampling to aid in success of test completion.
- Historical observations noted on mine plans in 8 Zone do not align with the quantified characteristics of the rock mass in this area. With high grades and significant ounces expected to be recovered in this zone, it should be further investigated.
- There is considerable uncertainty around the in-situ stress conditions. There is a possibility that the k ratio is greater than that assumed, and the major principal stress is at an oblique angle or near perpendicular to the orebody. Both of these conditions would increase the risk of rock bursting.

The following work is recommended to address the data gaps:

- Targeted triple tube oriented geotechnical drilling and core logging to collect geotechnical data over the full extents of the planned underground workings. New logging data should be compared to historic logging data collected on split core to assess the validity of the older data.

- Comprehensive geomechanical testing should be performed to supplement historic data. This should involve point load testing of full diameter core as well as lab testing. Lab tests should include tests that can accommodate both weaker and more competent lithologies.
- More mapping of open workings as these become available/ dewatered. Current mapping is limited to near-surface exposures.
- A stress sensitivity numerical modeling study should be undertaken to evaluate the impact of a more adverse principal stress direction.
- A stress measurement campaign should take place at the mine to confirm previous assumptions regarding the stress gradient.

16.4.2 Geotechnical Domains

The 2018 FS had separated domains by lithology and alteration (Table 16-1).

Table 16-1: 2018 FS rock mass characteristics

Domain		No. of Runs	Avg. Recovery	Avg. RQD	Fracs/m	Q' (Barton & Grimstad, 1993) [AB]					RMR (Bieniawski, 1976) [C]		
						Min Q'	20th %	Avg. Q'	80th %	Max Q'	Min. RMR_{76}	Avg. RMR_{76}	Max. RMR_{76}
Litho Based Geotech Domains	Basalts	164	100.0	97.5	1.7	1.9	10.9	15.4	17.0	33.9	50	68	76
	Intrusive Dikes	14	99.3	95.2	2.8	5.2	8.0	13.3	16.4	20.5	59	67	71
	Russet Lake UMAFs	190	99.6	95.9	2.0	2.8	10.4	14.5	16.7	32.9	53	67	75
	Venus UMAFs	35	99.8	88.3	3.7	2.4	3.3	4.6	5.6	7.4	52	57	62
	Veins	6	100.8	98.2	1.5	10.5	10.7	13.6	16.5	16.5	65	67	69
	Confederation	26	99.8	95.3	2.2	3.4	8.4	13.7	16.3	33.0	55	67	76
Alteration / Mineralization Based Geotech Domains	Austin	45	99.9	95.9	3.5	2.8	5.5	11.5	16.5	35.0	53	64	76
	McVeigh	98	99.9	97.5	2.2	1.9	10.8	14.4	16.7	33.9	50	67	76
	South Austin	16	100.0	94.6	2.2	6.7	9.8	18.1	28.9	33.9	61	69	76
	8 Zone HG HW	32	99.9	97.7	1.6	7.5	8.3	14.0	16.6	17.1	62	63	63
	8 Zone HG	17	97.0	86.9	3.6	2.8	4.8	6.7	8.4	12.2	53	61	67
	8 Zone HG FW	34	99.8	95.4	1.9	10.9	15.6	18.3	21.3	32.9	66	70	75
	All Peridotite	211	99.6	95.0	2.3	2.4	7.9	12.4	16.6	32.9	52	66	75
	All SAFZ	48	99.8	98.1	2.3	3.4	8.0	14.5	16.9	34.6	55	67	76

[A] Assuming 3 joint sets (Jn = 9).
[B] Q' is calculated by setting the Joint Water Factor (Jw) and Stress Reduction Factor (SRF) both equal to 1 in the Q equation
[C] RMR76' calculated from Q' based on Bieniawski 1976 equation ($RMR_{76} = 9 \times \ln Q + 44$).

Source: JDS (2018)

Some of the lithologies and alteration domains have similar rock mass properties and SRK recommends simplifying domains on an operational level to two categories only, described below.

- **Good to Fair Rock** – This includes Basalts, Veins, the Confederation unit, Peridotite, all SAFZ, and Russet Lake UMAFs
- **Poor to Very Poor Rock** – This includes all rock in 8 Zone, Venus and other weak ultramafic units

Table 16-2 presents proposed properties for the two categories. Ground support recommendations as part of this PFS study use these two categories. Stope design utilizes the average properties for “good to fair rock”.

Historical UCS testing has not been standard, with point load testing having been performed on a combination of split core, and more recently, whole core.

Table 16-2: Geotechnical operational categories

Ground Support Classification (Geotechnical Domains)	Q' (Barton & Grimstad, 1993) [A,B]	Rock Mass Strength (MPa)
	Avg. Q' ranges	Avg.
Good to Fair Rock	4.0 to 20	107 (R4 – R5)
Poor to Very Poor Rock	0.4 to 4.0	*(R3 – R4)
* No PLT/UCS data in this category from the SRK guided 2024 DDH program. Strengths are in the R3 to R4 range		

Source: SRK (2025)

Squeezing ground conditions are expected in large parts of the 8 Zone mining based upon observations made during previous mining and recorded on contemporaneous mine plans. Steel and timber sets were extensively used in development to the 8 Zone ore zone. SRK have adjusted the rock mass properties to the "poor to very poor rock" domain in the 8 Zone to better represent the anticipated squeezing conditions.

Dykes have been observed in core and in underground workings but represent a very minor portion of the total rock mass. They may play a role in aspects like seismic assessments but are not likely to play a role in daily mine operations.

16.4.3 Structural Geology and Hydrogeology

There have been very few discrete brittle fault structures encountered at Madsen due to its location within the Canadian Shield. Faults at the Madsen site tend to be discrete with minimal to no alteration associated with the faults. Adverse conditions are not expected to be associated with fault zones and no changes to mining practices or ground support are required based on the current understanding of the fault characteristics at the Madsen site. Major fault structures do not play a role in geotechnical domains.

No hydrogeological data has been provided to SRK for review as part of the rock geotechnical evaluation. It is assumed that underground workings will be dewatered and production mining areas will be dry by the time extraction takes place. It is recommended that further hydrogeological investigation be performed to gain a better understanding of water sources, flows and risks.

16.4.4 Geotechnical Conditions

Rock Mass Conditions

The main deposits demonstrate similar rock mass conditions as characterized by the following four main lithological units:

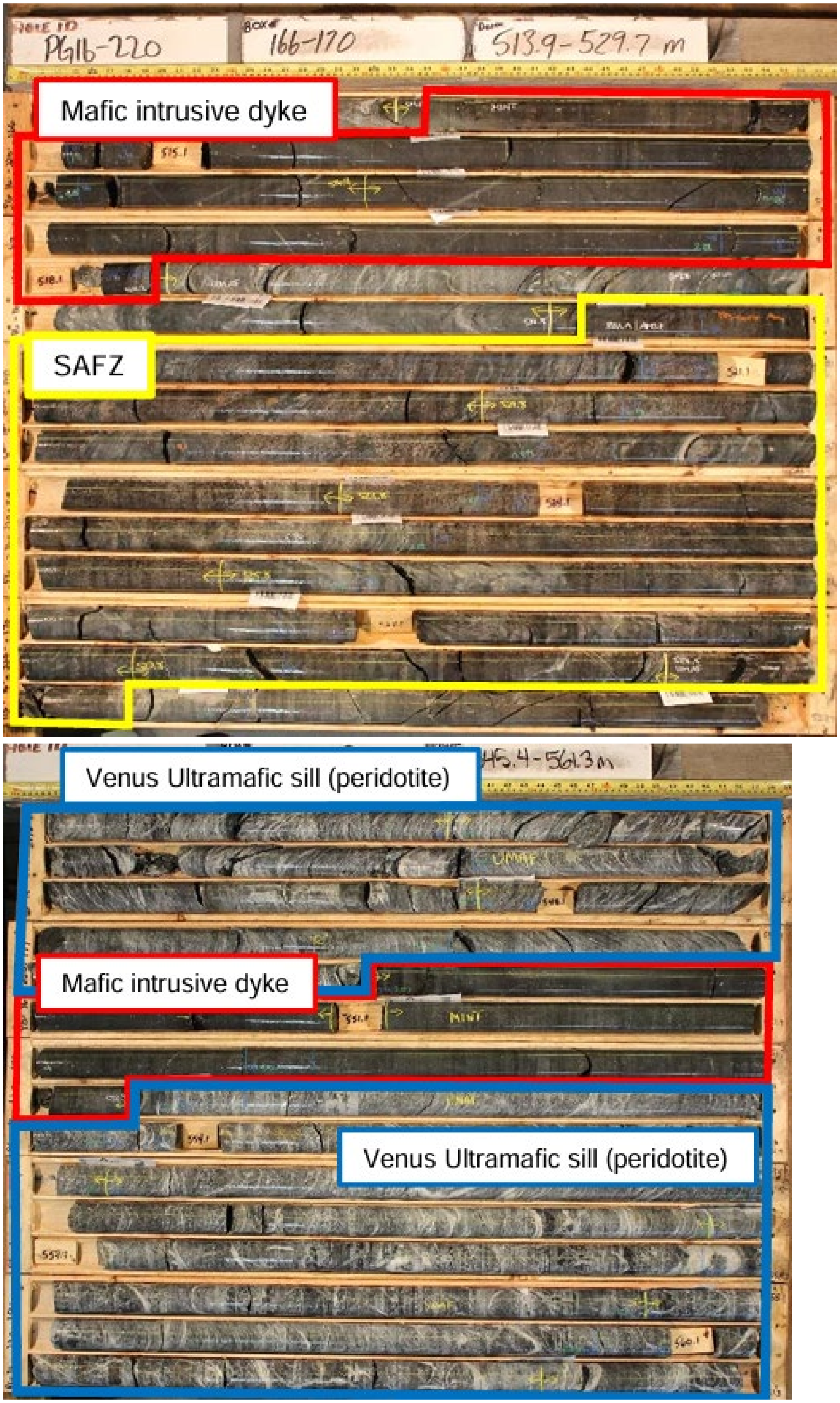
- **Balmer Basalts** – a very competent rock mass with high intact rock strength and appear as slightly blocky to massive (Figure 16-3). Sub-units are defined by alteration, though both the altered basalt (BSLA) and the non-altered basalt (BSLT) have similar geotechnical characteristics.

Figure 16-3: Drill core demonstrating the basalt rock mass

Source: JDS (2018)

- **Strongly Altered Foliated Zones (SAFZ)** – the SAFZ unit (Figure 16-4), containing the gold mineralization, is well understood through extensive drill programs. This unit is a competent rock mass, driven by foliation that dips steeply south, parallel to the orebody.
- **Intrusive Dykes (mafic and intermediate)** – the dykes are characterized as a good, blocky rock mass. The unit is well jointed with talc alteration near contacts.
- **Ultramafics (UMAF)** – UMAF unit is sub-defined by five main sills (Venus, Pluto, Neptune, Mercury and Jupiter) with Venus being the closest to the stopes. Rock mass can vary from very poor to fair rock, caused by poor joint conditions, heavy foliation, and talc alteration.

Figure 16-4: Drill core demonstrating the SAFZ, dyke and ultramafic rock masses

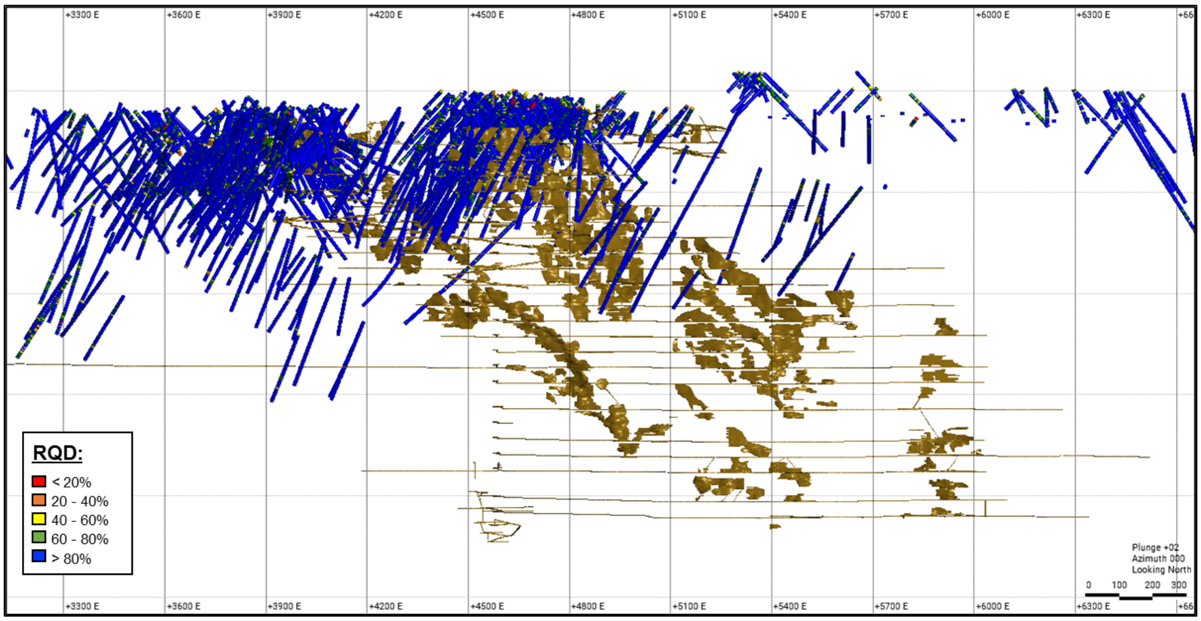


Source: JDS (2018)

RQD Data

RQD data was collected in some of the DDHs on site. Overall, great coverage and rock quality were recorded in the upper zones of the mine, but data is very limited below depths of approximately 500 m below surface. Recorded RQD values were generally high, with over 80% of the intervals having an RQD percentage of over 80% (Figure 16-5).

Figure 16-5: Drillhole database showing RQD



Source: SRK (2025)

Structural Domain Joint Sets

Structural joint data sources are limited to near-surface mapping and ATV surveys within three drill holes. For FS-level assessments, further collection and assessment of discontinuity data is required. This will include a combination of underground mapping and the use of down hole ATV surveys to collect additional discontinuity data.

Stress

The Madsen Mine does not have a recorded history of seismic activity; however, the Red Lake mining district has been known to produce large seismic events, particularly from the Red Lake and Campbell Mines east of the Madsen Mine.

Planned stopes at Madsen extend to approximately 1500 m below ground surface where the stress magnitudes will be greater than that in the historically mined areas.

The maximum principal in-situ stress, σ_1, was based upon recent interpretation of in-situ stresses from the Red Lake Mine, which suggests the following:

- Maximum Principal Stress, σ_1, at an azimuth of 100º
- Minimum Principal Stress, σ_3, is vertical (k = 1.9)
- Intermediate Principal Stress, σ_2, is horizontal (k = 1.1)

SRK reviewed the risk of rock bursting for the Madsen Mine (SRK, 2024). The findings are summarized as follows:

- Madsen does not have records of a rock bursting problem historically, but Red Lake Mine experience in similar ground conditions has shown that bursting can become a problem at high stress and greater depths. Significant rock bursting has occurred in the Red Lake mining district starting at depths near 500 m.
- The rock is generally of high strength and is brittle. At low stress levels, this is advantageous but at high stress can result in greater rock burst risk. The rock mass has the potential to burst if mining induced stress becomes high enough.
- The strong brittle rock mass in the mafic dykes and orebody have potential to burst under high stress conditions, while domains with greater foliation are likely to have a slightly lower rock burst risk.
- A large part of the planned mining exists in abutments and pillars created by previous mining. These areas will be under greater stress than historic mining at Madsen and are likely to be of higher seismic risk than previous mining.
- There is considerable uncertainty around the in-situ stress conditions. There is a possibility that the k ratio is greater than that assumed, and the major principal stress is at an oblique angle or near perpendicular to the orebody. Both conditions would increase the risk of rock bursting.

Field and Laboratory Strength Testing

The rock mass at the Madsen site is generally strong, stiff, and anisotropic.

Geomechanical laboratory testing was last performed in association with the 2019 FS (JDS, 2019). This data was used as part of the 2024 PFS assessment.

Field data included point load testing (PLT) data was collected on rock core. Most of the PLT database is associated with the 2019 study. Additional PLT data has been collected at site in 2024. This data is limited in spatial distribution to the focus of resource drilling locations. Generally, this data aligns with the 2019 database.

Laboratory strength testing was performed on core from two DD holes in 2018. Table 16-3 summarizes the results of this testing.

Table 16-3: Summary of laboratory testing program (2018 Drillholes PGU-0072 and PGU-0073)

Rock Type and Grouping	UCS (MPa)	No. Tests	E* (GPa)	v*	E** (GPa)	v**	γ (kg/m³)
Basalt (BSLA)	134	4	93.5	0.23	100.9	0.16	2,997
Basalt (BSLT)	137	8	93.1	0.21	91.4	0.06	2,963
BSLA & BSLT combined	136	12	93.4	0.22	97.7	0.12	2,975
BSLA & BSLT excluding foliation breaks	150	9	93.0	0.22	98.0	0.12	2,977
Venus (UMAF PRDT/PYXT)	132	6	86.2	0.32	108.5	0.32	3,031
Veins (VNDI)	215	1	235.8	0.2	220.7	0.10	3,217

* Tangent calculation method.
**Secant calculation method

Source: JDS (2019)

As part of the 2019 study, JDS used a correlation factor of 14.6 to convert PLT test results to UCS values. The PLT data used in the 2019 study and converted UCS values for various lithologies is described in Table 16-4. Only the 2019 summarized data was available to SRK - raw lab and PLT data were not available.

Table 16-4: Summary of UCS values correlated from PLTs for different lithologies and test break types

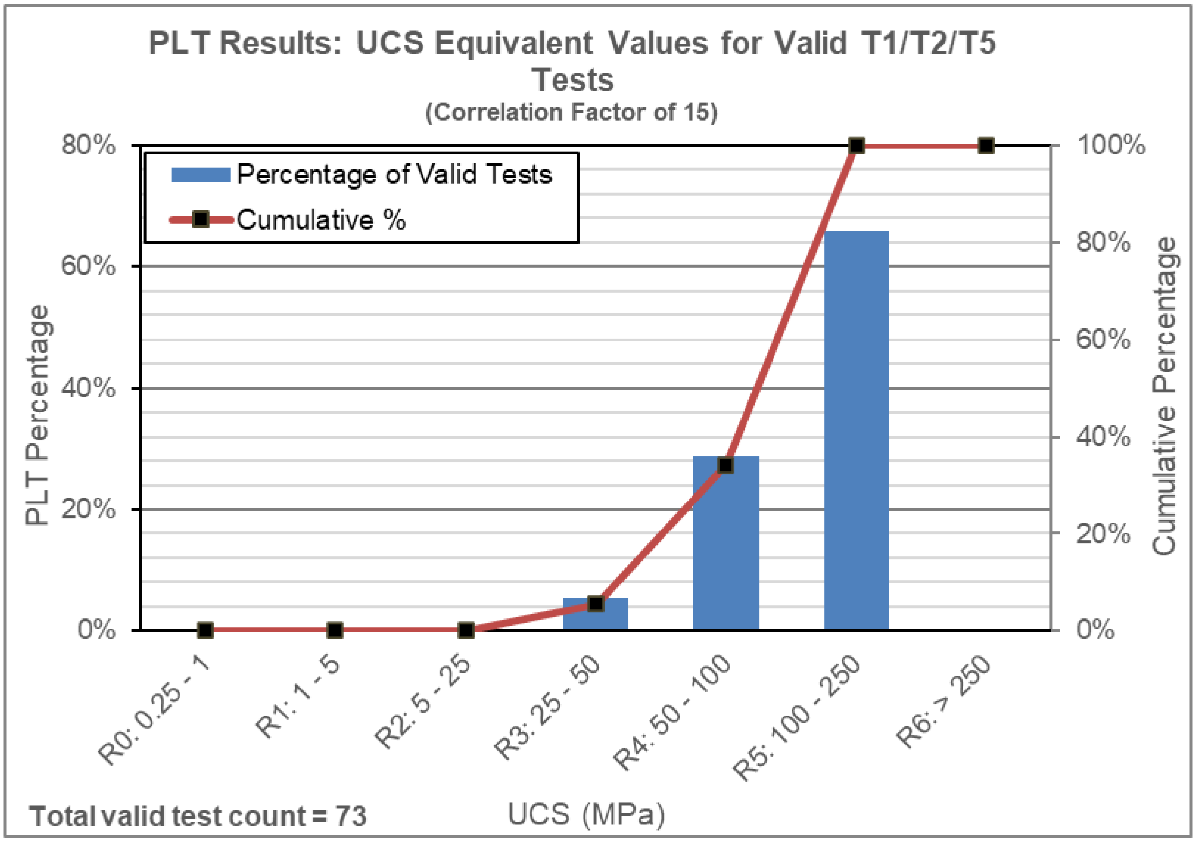
Lithology Solid	No. Tests	T1 (Valid Test) UCS (MPa)	T3 (failed on pre-existing defect) UCS (MPa)	T1 & T3 Combined UCS (MPa)	T3:T1 A (Strength)	% T3 Tests B
Balmar Basalts	74	140	62	110	44%	39%
SAFZ	104	155	80	126	63%	38%
Venus UMAF	29	159	34	103	21%	45%
Russet Lake UMAF	101	138	35	81	25%	55%
Intrusive Dykes	9	205	94	180	46%	22%
Confederation	47	169	54	101	32%	60%

Source: JDS (2019)

From the recent 2024 summer drill program, PLT results were consistent with the 2019 study (JDS, 2019). A total of 73 valid tests (T1, T2, T5) were completed across 14 drillholes, between 15 m before and after the orebody intercept within the drillholes. SRK has not received laboratory data and has used the conversion factor of 15 from the 2019 study to estimate UCS strengths from the PLT data.

The 2024 PLT results (Figure 16-6) are mainly in the Basalts and SAFZ, with little to no whole core PLT tests completed in the Ultramafics unit. UCS values presented in the 2019 study were undertaken on split core. Additional PLT and geomechanical testing is recommended to increase the confidence in rock mass strengths, specifically in the Ultramafics and the 8 Zone.

Figure 16-6: 2024 Drill Program Valid PLT Results (T1, T2, T5)



Source: JDS (2019)

16.4.5 Underground Excavation Design

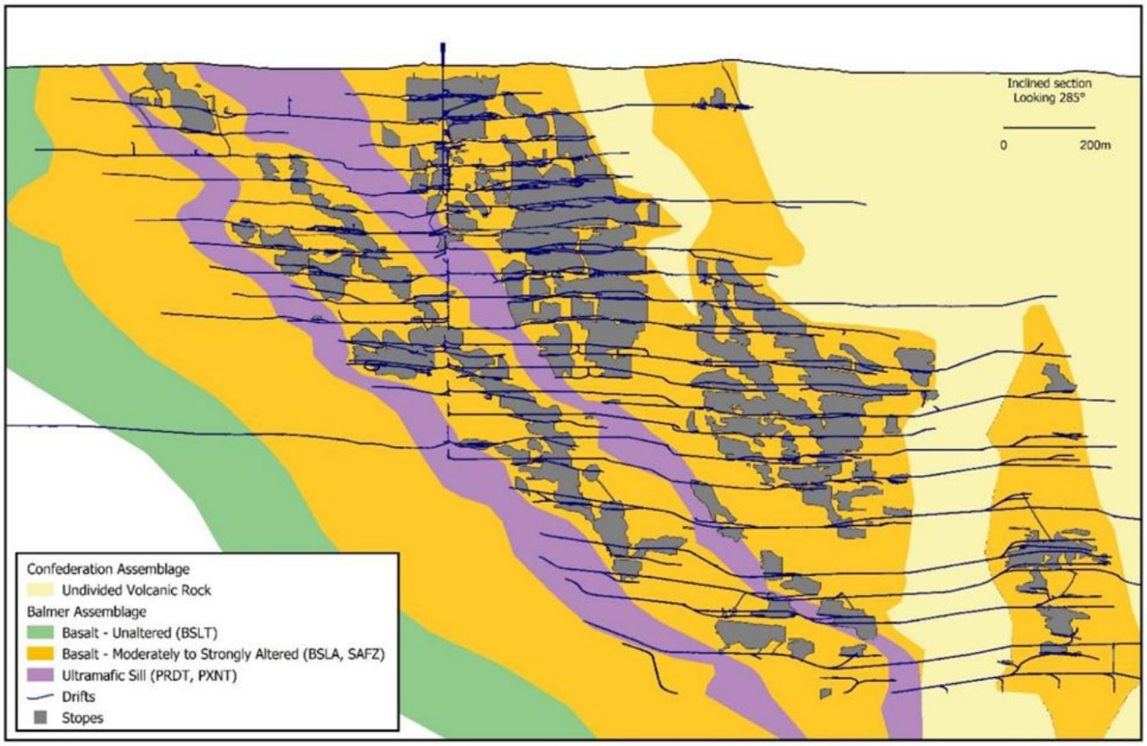
Excavation stability assessments have been completed using well-established empirical and semi-empirical relationships and engineering experience. These relationships enable estimates to be made of the expected mining conditions and support requirements based on a detailed description of the rock mass, excavation geometry, and prevailing stress conditions. The design procedure involves two steps: i) the quality of the rock mass is rated using a pre-defined classification system, and then ii) the expected performance of the underground openings is predicted using an empirically derived stability correlation with the rock mass quality.

Production Excavation Design Spans

The following summarizes the geotechnical, orebody geometry, and practical mine design inputs, around which the geotechnical stability and design evaluation was conducted.

The Madsen Mine exploits a steeply dipping narrow vein deposit. Most historical gold production and most of the current reserves and resources are within the Austin, South Austin, McVeigh, and 8 Zones. The distribution of gold within these planar structures is primarily controlled by the intersection of the structures with basalt/ultramafic lithological contacts (Figure 16-7). The ore zones tend to be tabular, with minor s-shaped fold geometries.

Figure 16-7: Inclined long section through the Austin and South Austin zones with projected geology



Source: JDS (2019)

The 8 Zone differs from the other mineralized zones in geology and geotechnical properties. This zone occurs within strongly altered and veined peridotite of the Russet Lake Ultramafic while the other ore zones tend to be hosted within mafic host rocks. The 8 Zone has a planar geometry, strikes generally north-south, and dips to the east at approximately 45º, which is significantly shallower than the other zones.

Stope Design

For excavations in which personnel access is not required, such as longhole stopes, designs were assessed using the modified Matthews stability curve after Stewart and Forsyth (1995). A range of stope dimensions were evaluated for stability and dilution. A fixed sub-level spacing of 20 m (floor to floor) was used for all mining zones with maximum strike length, stope span, and geotechnical dilution determined three depth ranges.

Empirical estimates using the estimated linear overbreak and sloughing (ELOS) approach and benchmarking have been used to come up with the dilution estimates for the long hole open stopes (Clark, 1998). A summary of the results of this assessment is presented in Table 16-5.

Table 16-5: Stope dimensions and dilution

Depth	Sill to Sill Height (m)	Max. Strike Length (m)	Maximum Span (m)	Total ELOS
Surface to 500 m depth	20	50	6	0.6
500 m to 750 m depth	20	30	6	0.8
Below 750 m depth	20	20	6	1.2

Source: SRK (2025)

The Madsen Mine has extensive historic workings and new mining is often planned within close proximity of historic development and stopes. Different backfill material and backfill practices have been used in the historic stopes. Stopes that have been described as “partially filled” should be treated as open stopes as the fill volumes are unknown.

Historic mine workings consist of drifts, and stopes. WRLG provided a 3D model of these mine workings that was incorporated into the geotechnical assessment. Required stand-off distances from these workings have been set based on the rock mass quality surrounding these workings. When old workings are anticipated to be intersected, cover holes must be drilled in advance of the development to confirm their location and to determine if they are filled, what they are filled with, and to determine if these excavations have been dewatered.

Specifically for new planned cut and fill (C&F) and drift and fill (D&F) mining, excavations would have smaller areas of exposure to historic mining. Where new C&F or D&F is planned in proximity to fully filled historic stopes, it is considered acceptable to mine adjacent to decanted hydraulic fill or rock fill. In these cases, the use of shotcrete and bolts is strongly recommended on the exposed historic fill to help stabilize the wall of exposed fill. Consideration should be given to using shorter round lengths to reduce impact on the historic fill, especially at the start of production until the performance of the historic fill material is better understood.

Backfill

A detailed description of the backfill options considered and eventually selected for implementation at the Madsen Mine are discussed in Section 18.6.

Longhole stoping requires backfill to manage stability and achieve the planned extraction. Backfill is required in longitudinal stopes and closure areas where mining progresses towards previously mined areas. All excavations should be tight filled to minimize the space for potential failure and rock settlement. Planned backfill for the Madsen Mine includes waste rockfill and cemented hydraulic fill.

Rockfill will consist of blasted waste rock on levels and will be placed in newly mined stopes. It is recommended that at least every second stope (50%) of the stopes along each level are fully filled to ensure no long open strike spans are left.

Backfill in MCF mining blocks will predominantly be comprised of waste rockfill with the exception of 8 Zone in which hydraulic fill will be predominant.

Pillars

Rib Pillars

The planned backfill in the PFS study includes unconsolidated rock fill in stopes. It is recommended that rib pillars are utilized in the stoping areas at the end of the strike length of sets of stopes. Rib pillars should be a minimum of 5 m wide or double the span (width to length ratio of 2:1). These rib pillars should extend the full height of the stopes. Rib pillars are also recommended where Alimak raises are used for access. The same rules as between stopes should apply to pillar dimensions between a stope and an Alimak raise.

Sill Pillars

Sill pillars are recommended where mining under historic stopes, regardless of historic fill type. A sill pillar that is a minimum of 7.5 m or two times the stope span will be maintained at the narrowest pillar point.

Crown Pillar

No new data has been collected to supplement crown pillar assessments. Crown pillars should be designed to a Class F or G stability rating based on Carter (2008). Generally, for stopes less than 8 mW x 50 mL with a dip greater than 70º, it is recommended that a minimum crown pillar of 25 m of intact bedrock be maintained.

16.4.6 Excavation and Support Design

Design spans (3 to 5 m) for which personnel access is required have been reviewed based on the critical span design curve presented by Ouchi et al. (2004). In the static stress condition, the excavations in the "good to fair rock" domain are expected to remain stable with standard ground support (i.e., rockbolts and mesh). Additional ground support (i.e., rockbolts, mesh, and shotcrete) will be required in the "poor to very poor rock" domain to maintain a stable operating span. In some cases, shorter round lengths and spiling may be required in the 8 Zone and weaker ultramafic units. Determining exact ground support specifications for such situations should use site-specific assessment by site rock mechanics personnel. Squeezing ground conditions are anticipated in some of the 8 Zone based upon observations of ground behavior recorded on historic mine plans.

Ground support design for the Madsen Mine has considered excavation geometries, required timelines for effectiveness of ground support, seismic conditions, and corrosion estimates. The ground support types, sizes, and patterns align with industry standards.

Design Inputs

The following inputs are based on the current understanding of the excavation design. Each category applies to all lithologies.

- Waste ramp development and truck access levels will be 5 mW x 5 mH
- Long term waste levels and ramps with no truck access will be 3.5 mW x 3.5 mH

- Longhole ore sill drifts on ore will be 4 mW and on average 3.5 mH
- Normal Mechanized Cut and Fill ore drives will be 4.4 mW and on average 3.5 mH
- Mechanized Cut and Fill ore drives against hydraulic or rock fill will be 3.2 mW and on average 3.5 mH
- Mechanized Cut and Fill ore drives specifically in 8 Zone will be 3.5 mH and on average 3.6 mW; Two passes in wider sections of the 8 Zone will be taken resulting in span on 7.2mW

Standard ground support recommendations have been made for average ground conditions assuming no issues with stress and seismicity. While it is currently uncertain whether mine seismicity will be encountered, it is believed that it is likely that higher stresses at depth will require the use of dynamic ground support. For this reason, it is assumed that 10% of development ground support will require adjustments for seismicity and this will consequently double the costs of this ground support.

Support Recommendations

Generally ground conditions are expected to be good to fair in most of the lithologies and zones at the Madsen Mine. Poor to very poor ground conditions are expected within ultramafic lithological units and within the 8 Zone. On average, 10% of the excavations are anticipated to be within poor to very poor ground, with the exception of 8 Zone where 100% of the development is expected to be in weak, squeezing ground conditions.

Table 16-6 summarizes the ground support recommendations for the planned excavations at the Madsen Mine.

Table 16-6: Standard ground support recommendations in horizontal development

Details		Waste Development				Ore Development						
		Ramp development & Truck Access Levels		Long Term Levels & Ramps (no truck)		LH Sill Drifts on Ore		MCF drives (Normal)		MCF against Hydraulic/Rockfilled void		MCF in 8 Zone (Squeezing ground)
Configuration		Good to Fair Rock	Poor to Very Poor Rock	Good to Fair Rock	Poor to Very Poor Rock	Good to Fair Rock	Poor to Very Poor Rock	Good to Fair Rock	Poor to Very Poor Rock	Good to Fair Rock	Poor to Very Poor Rock	Poor to Very Poor Rock
	% of Planned Development	**90%**	**10%**	**90%**	**10%**	**90%**	**10%**	**90%**	**10%**	**90%**	**10%**	**100%**
	Height (m)	5.0	5.0	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5
	Width (m)	5.0	5.0	3.0	3.0	4.0 (avg.)	4.0 (avg.)	4.4 (avg.)	4.4 (avg.)	3.2 (avg.)	3.2 (avg.)	7.2 (avg.)
	Round Length (Break)	3.8	3.8	3.0	3.0	3.4	3.4	2.4	2.4	2.4	2.4	2.4
	Diameter/ span (m)											
	Back profile	Arched back	Arched back	Flat back	Flat back	Flat back	Flat back	Flat back	Flat back	Flat back	Flat back	Flat back
Back & Wall Bolts	Primary - Type	Resin rebar	Super Swellex (Pm 24)	Resin rebar	Super Swellex (Pm 24)	Resin rebar	Swellex (Pm 12)	Mechanical bolts	Swellex (Pm 12)	Mechanical bolts	Swellex (Pm 12)	Super Swellex (Pm 24)
	Primary - Diameter	22 mm (#7)	37 mm	22 mm (#7)	37 mm	22 mm (#7)	28 mm	5/8" FH	28 mm	5/8" FH	28 mm	37 mm
	Primary - Length	1.8 m	2.4 m	1.5m	1.5m	1.5m	1.5m	1.5 m	1.5 m	1.5 m	1.5 m	1.5 m
	Primary - Spacing	1.5 m square pattern	1.5 m square pattern	1.5 m square pattern	1.5 m square pattern	1.5 m square pattern	1.5 m square pattern	1 m square pattern	1 m square pattern	1 m square pattern	1 m square pattern	1 m square pattern
Screen	Type	Galvanized welded wire mesh	Galvanized welded wire mesh	N/A	Welded wire mesh	N/A	Welded wire mesh	N/A	Welded wire mesh	Welded wire mesh	Welded wire mesh	Welded wire mesh
	Gauge	6-gauge	6-gauge		6-gauge		6-gauge		6-gauge	6-gauge	6-gauge	6-gauge
	Wire spacing	100 x 100 mm	100 x 100 mm		100 x 100 mm		100 x 100 mm		100 x 100 mm	100 x 100 mm	100 x 100 mm	100 x 100 mm
	Distance from floor	1.5 m	Floor to floor	1.5 m	Floor to floor	1.5 m	Floor to floor	1.5 m	Floor to floor	One wall, overlap floor and back 1m	Floor to floor	Floor to floor
Shotcrete	Requirement	N/A		N/A		N/A		N/A				
	Thickness		50 mm		50 mm		50 mm		50 mm	50 mm	50 mm	50 mm
	Distance from floor		To floor		To floor		To floor		To floor	To floor	To floor	To floor

Source: SRK (2025)

Note that these support recommendations assume average ground conditions and if extremely poor ground is encountered, site-specific stability assessments and ground support designs will be necessary. It is recommended that vertical development and intersections be avoided in poor ground. Four-way intersections should generally be avoided due to increased excavation spans.

Table 16-7 summarizes ground support recommendations for raises and intersections.

Table 16-7: Standard ground support recommendations for raises and intersections

Support Type	Details	Alimak raise	Other raises	Intersections
	Height (m)	3		Up to 6.5 m
	Width (m)	3		
	Round Length (Break)	2.4		
	Diameter/ span (m)		3.4	up to 8 m
Back & Wall Bolts	Primary - Type	Resin rebar	Resin rebar	Resin rebar
	Primary - Diameter	22 mm (#7)	22 mm (#7)	22 mm (#7)
	Primary - Length	1.5 m	1.5 m	2.4 m
	Primary - Spacing	1.0 m dice-5 pattern	1.0 m dice-5 pattern	1.5 m square pattern
Screen	Type	Galvanised Chain Link Mesh	N/A	Welded wire mesh
	Gauge	6-gauge		6-gauge
	Wire spacing	50 x 50 mm		100 x 100 mm
	Distance from floor	On each face, as well as walls		1.5 m
Shotcrete	Requirement	N/A	N/A	N/A
	Thickness			
	Distance from floor			
Cable bolts	Type	N/A	N/A	Bulbed & tensioned/ plated
	Diameter			Single strand
	Length			3.8 m
	Spacing			2.0 m square pattern

Source: SRK (2025)

16.4.7 Geotechnical Risk

There is poor spatial coverage in geotechnical data across the Madsen deposit. From the current understanding of the rock mass, conditions are expected to be similar within similar lithologies, but this needs to be confirmed with data. There is also currently limited geotechnical data for the 8 Zone and the majority of the understanding of ground conditions comes from historic plans describing ground conditions and ground support.

Geotechnical structural data is limited. In the previous feasibility study structural data was collected in only three holes using acoustic televiewer (ATV) surveys. Geotechnical structural data was also collected in core that was specifically geotechnically logged, but this core had been split prior to logging, which is not standard procedure and may have induced error.

Underground mapping has been mostly conducted near to surface in workings above the flooded elevation. Geotechnical mapping should be extended as the mine is dewatered and as new development advances.

Most of the PLT data collected in the 2019 study was also performed on split core, which is not the standard test methodology. PLT data should be collected for all lithological units on whole core to confirm this data. Additional laboratory testing is also recommended. Some lithologies have been well sampled, with representative quantities of samples, but other lithologies have either not been sampled or have insufficient tests performed to understand strength variability.

Stress conditions are currently not well understood. Historical experience suggests that there has been little mining-induced seismicity at the Madsen Mine, but this could change as mine extraction increases and there is a greater volume of material extracted at depth. Other sites in the region have experienced rock bursting and their experience may be replicated at depth at the Madsen site.

The rock at Madsen is generally high strength and brittle. At low stress levels, this is advantageous but at high stress can result in greater rock burst risk. The rock mass has potential to burst if mining induced stresses become high enough. The strong brittle rock mass in the mafic dykes and orebody have potential to burst under high stress conditions, while domains with greater foliation are likely to have a slightly lower rock burst risk. A large part of the planned mining exists in abutments and pillars created by previous mining. These areas will be under greater stress than historic mining at Madsen and are likely to be higher hazard than previous mining.

There is considerable uncertainty around the in-situ stress conditions. There is a possibility that the k ratio is greater than that assumed, and the major principal stress is at an oblique angle or near perpendicular to the orebody. Both of these conditions would increase the risk of rock bursting.

16.4.8 Conclusions and Recommendations

It is recommended that geotechnical drilling and core logging be performed to better cover the extents of the planned mining. Specifically ground conditions in the 8 Zone have not been well quantified.

Mining within the 8 Zone is anticipated to vary greatly from the rest of the mine. Squeezing ground conditions are expected in the ultramafic units here. Historic development utilized timber sets, spiling, steel sets and shotcrete. It is anticipated that development rates will be slower than in other parts of the mine, with the need for shorter rounds and greater support.

A stress sensitivity numerical modeling study should be undertaken to evaluate the impact of a more adverse principal stress direction as the current assumed stress regime has not been well understood. A stress measurement campaign should take place at the mine to confirm previous assumptions regarding the stress gradient. The production plan should also be modeled to assess changing stress conditions and seismic risk and optimize the stope sequence.

The mine should employ seismic monitoring for advance warning of increasing seismic hazards. The design of the seismic system should place emphasis on coverage of mining areas with greater anticipated seismic risk, such as areas with high historic extraction, stopes in deeper parts of the mine and pillars.

16.5 Underground Mining Method(s)

16.5.1 Key Mine Design Inputs and Assumptions

The following key design inputs and assumptions were made for purposes of the PFS:

- That the dewatering plan will be successful in minimizing the mobilization of historic backfill
- That the historic track drifts immediately above/below historic stopes will not be usable in the mine design until inspected, assumed that bypass required
- That the historic voids that need to be backfilled to allow mining in close proximity can be successfully backfilled
- That historic ore passes near Madsen #2 shaft are not usable
- That historic track drifts are generally stable
- That the 3D as-builts are generally correct
- That operation will continue moving towards full mechanization as this is a requirement for successfully mining the historic MCFs and 8 Zone
- Limited steady state production rate of 800 tpd of ore, even though the mill will not full be (1,089 tpd name plate)
- Assumed all waste rock not used as backfill will be hauled or hoisted to surface
- Assumed that no waste rock was co-mingled with the cemented hydraulic fill
- That the operation will be able to attract and retain a skilled workforce

16.5.2 Access and Materials Handling

Current access to the mine is through either the East ramp or West ramp systems. The operation is currently developing a connection between these two ramp systems in order to improve public safety and haulage efficiency. This connection will eliminate the need for mine vehicles to cross the two municipal roads leading to the town of Madsen when using the West ramp portal. The connection is expected to be completed in Q1 2025. These ramp systems are suitable for modern 40t class haul trucks with the mine recently receiving two Epiroc MT42 trucks.

A project is also ongoing to recondition the Madsen Shaft #2 and install a new loading pocket at 12 Level to facilitate hoisting operations and reduce trucking requirements as the active mining areas get deeper. This work is being undertaken by a mining contractor and is expected to be completed in Q3 2025. This shaft is a rectangular timber shaft with small skips and is also acting as the main ventilation intake.

Due to the hoisting capacity limitations of Madsen Shaft #2 to hoist the required ore and waste tonnages from below 12 Level, a new Madsen Shaft #3 has been proposed with construction expected to be completed Q4 2028. This shaft will also help support the increased ventilation flows required by modern diesel equipment.

The Madsen Shaft #2 is planned to be reconditioned down to 24 Level to provide cage access in order to provide emergency egress from the mine, either in the cage or ladderway.

Access for personnel and materials will mainly be via the East portal and ramp systems for the life of mine with some movement via the shafts. Once commissioned, Madsen Shaft #2 will be used to hoist ore and waste to surface until such time as Madsen Shaft #3 is completed.

Ore will be trucked from the work areas to the closest available dump point:

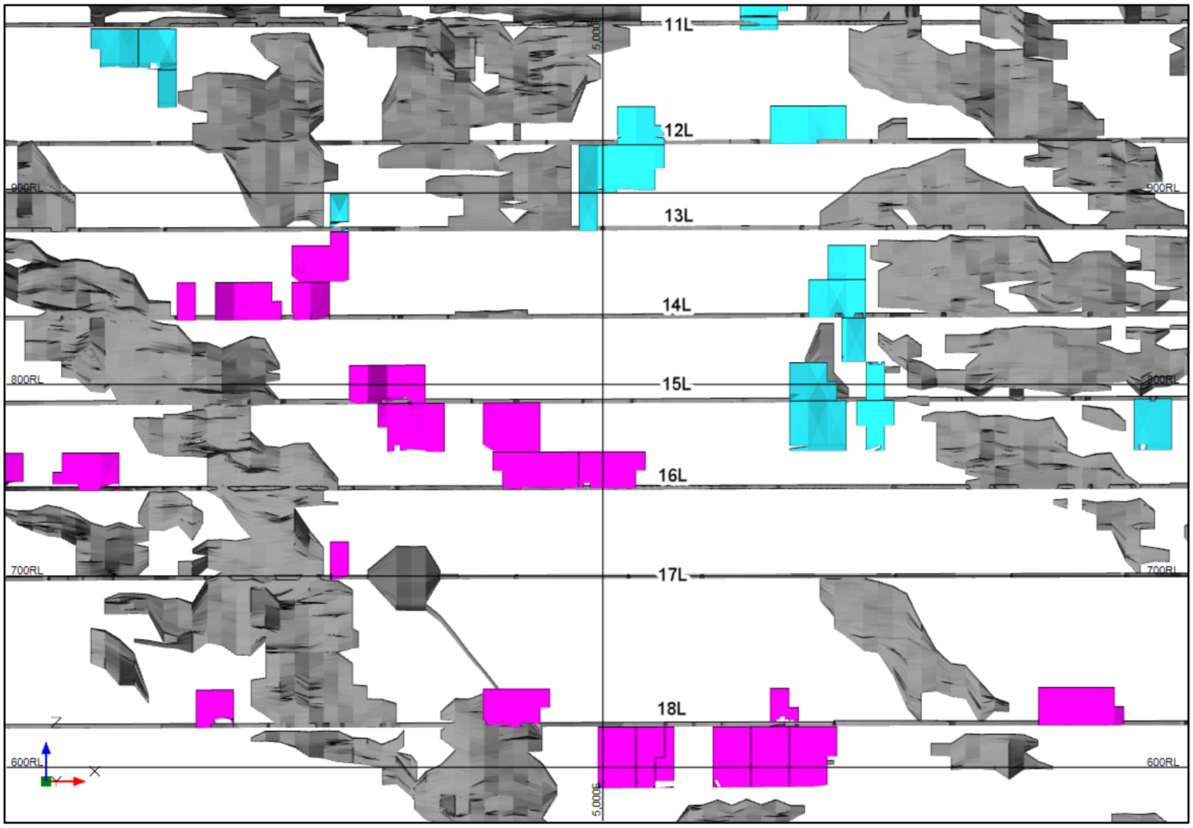
- Surface
- 10 Level grizzly for Madsen Shaft #2
- 18 Level grizzly for Madsen Shaft #3

16.5.3 Planned Mining Methods

One of the factors driving mining method selection is selectivity due to the nature of the ore body and remnant mining in and around historic workings. The ore body is very erratic with a high nugget effect, therefore mining areas are not continuous and tend to be quite small with the largest longhole (LH) zone being 65 m along strike and 32 m high.

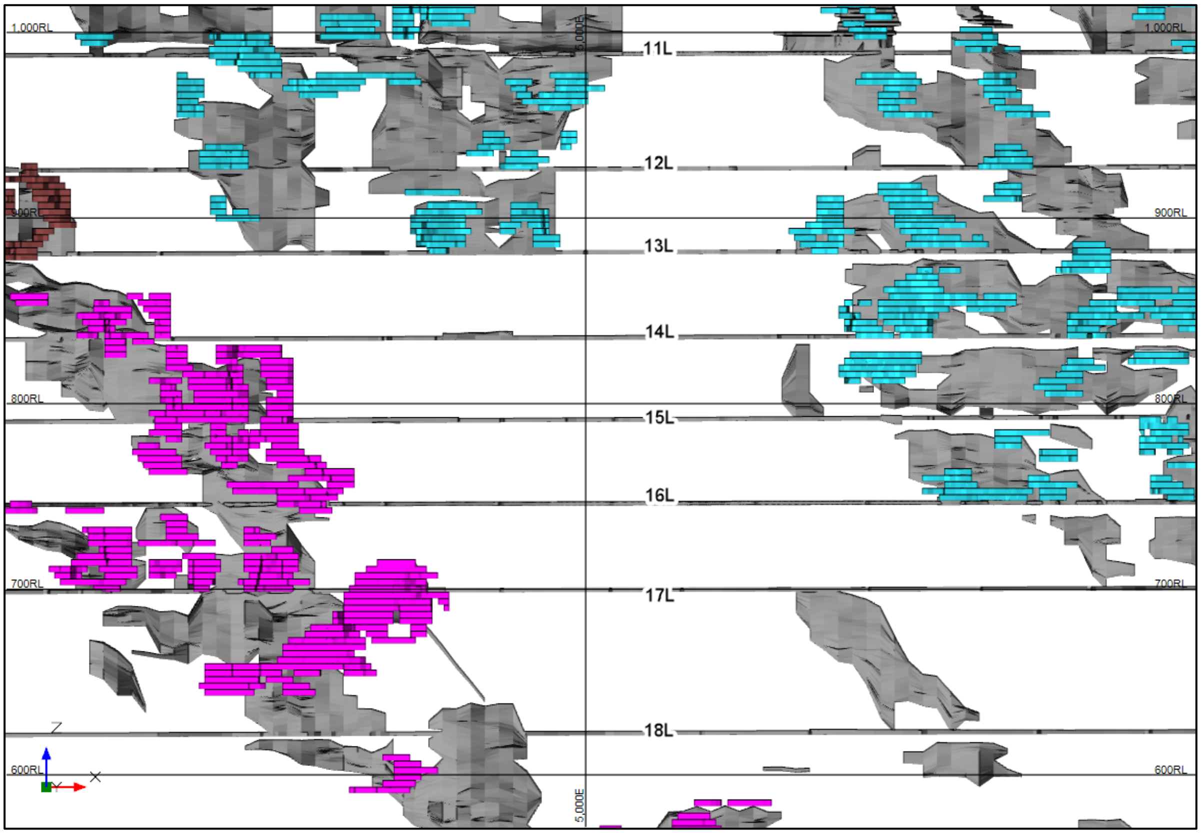
Most LH mining areas are 1-3 stopes along strike and 1-3 stopes high as illustrated in Figure 16-8, which shows the LH stope designs for the Austin Central (Cyan) and Austin South Deep (Magenta) zones. The MCF stopes are more selective but still create mining areas with relatively short strike lengths as illustrated in Figure 16-9. The 8 Zone Mechanized Drift and Fill (MDF) zone is the most continuous and highest grade of all the zones, but also has the worst ground conditions.

Figure 16-8: Example of LH stopes in Austin Zone



Source: SRK (2025)

Figure 16-9: Example of MCF stope in Austin Zone



Source: SRK (2025)

Table 16-8 shows the planned mix of mining methods and the associated tonnes and grade included in the Mineral Reserve Estimate. There is 4 kt of capital development which was classified as incremental ore as described in Section 15 and is assumed to be sent to the mill.

Table 16-8: PFS mining method ratios

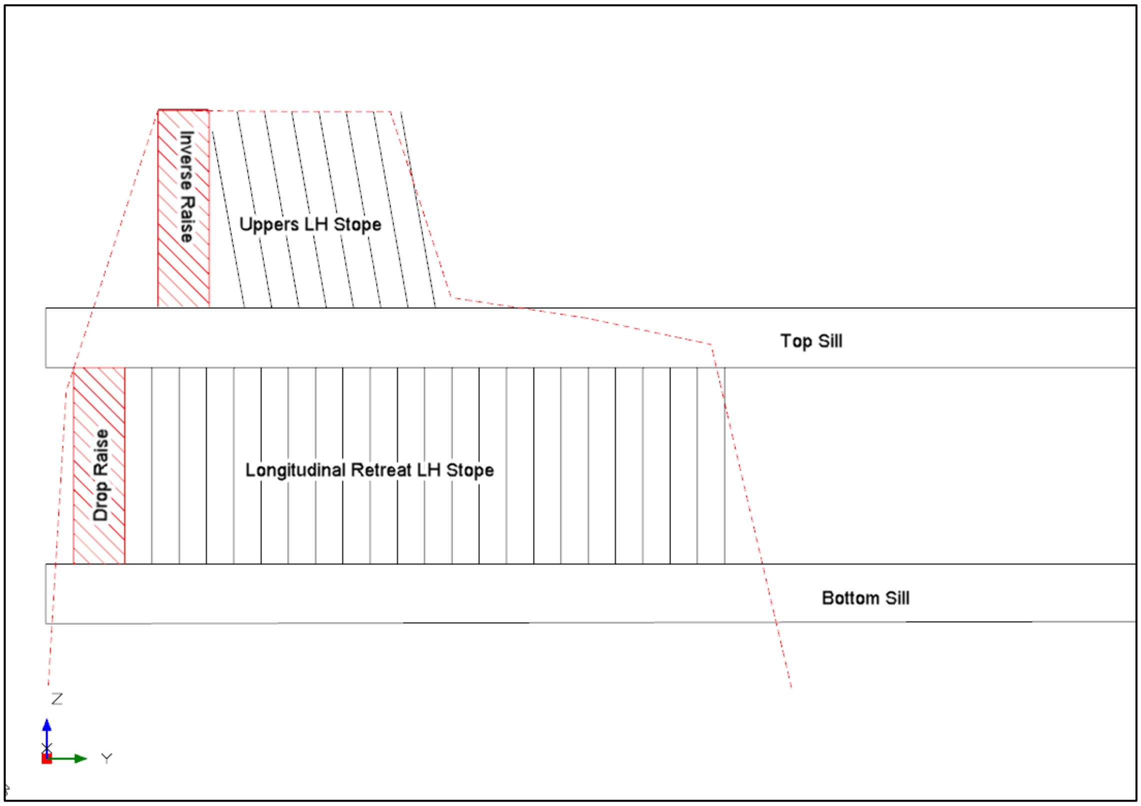
Mining Method	Tonnes (kt)	Gold Grade (g/t)	Percentage
Longhole	597	5.85	32.7%
Normal MCF	316	6.82	17.4%
Historic MCF	788	9.69	43.2%
8 Zone MDF	118	13.38	6.5%
Incremental Capital	4	1.46	0.2%
Total	**1,823**	**8.16**	**100%**

Source: SRK (2025)

Longitudinal Retreat Longhole (LH)

Longhole stopes will utilize longitudinal retreat methods with larger, full level stopes being drilled using down holes from the Top Sill to improve quality control on the drilling. A conventional 1.8 m x 1.8 m drop raise will be drilled and blasted at the extents of the mining area as a slot raise (Figure 16-10). All drilling will be conducted using a top hammer production drill to drill 64 mm diameter blast holes. Production rings will typically have three holes per ring on a 1.34 m spacing with a 1.6 m burden between production rings. Average true width of the LH stopes is 2.8 m in-situ with an additional 0.6 m of ELOS included in the DSO shapes. Therefore, average diluted stope true width is expected to be 3.4 m as shown in Figure 16-11. A drill factor of 3.28 tonnes/m drilled was calculated with a powder factor of 0.81 kg/t.

Figure 16-10: Long section of example LH area



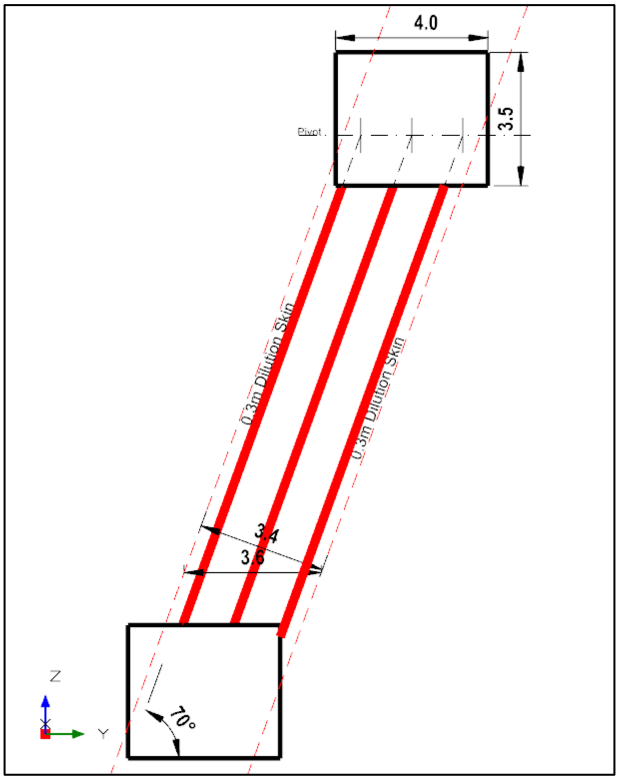
Source: SRK (2025)

The stopes will be blasted using packaged emulsion loaded using Swedish loaders and mucked using remote capable 7-t class LHDs.

Backfill will be Unconsolidated Rock Fill (URF) when conditions allow, Cemented Hydraulic Fill (CHF) if URF cannot be used.

Due to the nature of the ore body, quite a few Uppers LH stopes are included in the plan with the same general parameters. The only difference being the slot raise will be drilled and blasted as an inverse raise and the production holes will be drilled blind from the bottom. These stopes are generally not filled.

Figure 16-11: Typical LH drill section

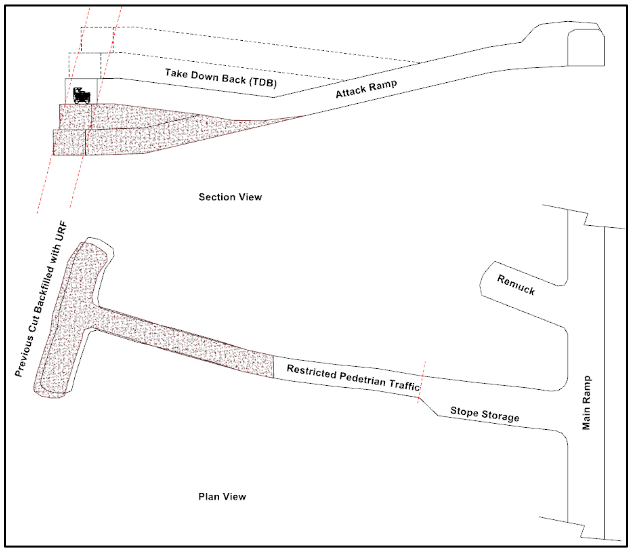


Source: SRK (2025)

Normal Mechanized Cut and Fill (MCF)

The normal MCF stopes are the portion of an MCF stope that is not adjacent to historic workings. An MCF stope may be partially Normal MCF and partially Historic MCF. The difference is in the ground support and round lengths. An illustration of the MCF method is shown in Figure 16-12.

Figure 16-12: Typical mechanized cut and fill (MCF) stope (plan and section)



Source: SRK (2025)

Mining progresses as follows:

1. Attack ramp is driven to the bottom elevation of the stoping block.
2. The first cut of the MCF stope is driven full face along strike from the attack ramp to the ore body extents in both directions.
    a. Driven under Geology control using one boom jumbo with telescopic feed (2.4 m to 3.65 m steel length) and 7-t or 10-t class LHD.
    b. Assumed short rounds (2.4 m) for the first year of production as the grade control procedures are developed and refined to deal with fact that there is no visual distinction between ore and waste.
    c. Assumed longer 3.0 m length rounds starting Q2 2026
    d. Drift height 3.5 m to allow use of mechanized bolter to install screen and 1.5 m ground support (5/8” Forged Head Mechanical Bolts or equiv.)
    e. Explosives will be ANFO
3. The completed cut is backfilled with Unconsolidated Rock Fill (URF), or possible cemented hydraulic fill (CHF), as tight to back as possible.
    a. Waste rock from development will be brought into the stope by LHD and then compacted using a Rammer-Jammer, see Figure 16-13
    b. If there will be a future mining block below the first cut, the backfill will be CHF with a higher binder content and sill matt installed instead of URF
4. Once the cut is backfill, the back of the attack ramp will be drilled and blasted with the jumbo and dropped on the floor as backfill and new ground support installed. This is called a take-down back (TDB).
5. Repeat steps 1-4 until top cut mined out.
    a. Only difference from first cut is that subsequent cuts will not be full face, but “breasting” such that the gap between the backfill from the previous cut and the back of the previous cut acts as a free face
6. Each attack ramp typically provides access to 5 cuts at 3.5 mH each.

Figure 16-13: Example of a Rammer Jammer

Source: SRK (2025)

The PFS LOM Plan has assumed that all of the MCF stopes will be mined using a horizontal slashing method ("Breasting") as the Historic MCF and 8 Zone will require this due to ground conditions. There is an opportunity that a portion of the Normal MCF stopes could be recovered using a vertical slashing method ("Jumbo Uppers"), which has the advantage of slightly higher productivity and lower costs when used in areas with suitable ground conditions.

Average width of the Normal MCF stopes is 3.6 m in-situ with an additional 0.8 m of ELOS included in the DSO shapes. Therefore, average diluted stope is expected to be 4.4 mW x 3.5 mH by 16-20 mL. Advance rate in ore for the first year (2.4 m rounds) was estimated at 3.9 m/d, assuming two faces available 80% of the time and that multiple work areas are available to the crew. This equates to 1.8 rounds per day on ore or 191 tpd. With 3.0 m rounds, this advance rate increases to 5.0 m/d. This equates to 1.8 rounds per day on ore or 245 tpd.

The stopes will be blasted using ANFO and mucked using 7-t class LHDs.

Historic MCF

Historic MCF stopes are the portion of MCF stopes that are adjacent to historic workings, an MCF stope may be partially Normal MCF and partially Historic MCF. The difference is in the ground support and round lengths.

The Historic MCF stopes will utilize short rounds (2.4 m) only and have heavier ground support requirements with the addition of 50 mm of in-cycle shotcrete to be applied to the wall closest to the adjacent stope.

Average width of the Historic MCF stopes is similar to the Normal MCF stopes with an average diluted stope width of 4.3 mW x 3.5 mH by 16-20 mL. Advance rate was estimated at 2.8 m/d assuming two faces available 25% of the time, as many of these areas are end access instead of centre access due to the proximity of the historic stopes. This equates to 1.25 rounds per day on ore or 134 tpd.

The stopes will be blasted using ANFO and mucked using 7-t class LHD's. In-cycle shotcrete will be applied using a remote spray arm and a mechanized bolter will be used to install ground support to minimize exposure of personnel.

The historic stopes can be categorized into three groups in order of descending difficulty:

- Open voids that will be filled with CHF before mining of the adjacent historic MCF
- Historic voids backfilled decades ago with unconsolidated rockfill that must be fully drained before mining of the adjacent historic MCF
- Historic voids backfilled decades ago with uncemented hydraulic fill that must be fully drained before mining of the adjacent historic MCF

The historic voids backfilled with uncemented hydraulic fill are the most at risk of mobilizing the backfill during dewatering operations and in terms of ensuring the backfill is fully drained before beginning to mine in close proximity.

8 Zone Mechanized Drift and Fill (MDF)

The quality of the rock mass within the 8 Zone mineralization is expected to be 'Poor' to 'Very Poor' with intense talc alteration and foliation being common within this zone. Squeezing ground conditions are expected in large parts of the 8 Zone mining based upon observations made during previous mining and recorded on contemporaneous mine plans. Previous reports indicate that the historical mining encountered ground control challenges due to what was described as a talc schist encountered near the hanging wall of the ore body.

Average width of the 8 Zone stopes is considerably wider than any of the MCF stopes higher in the mine at 7.2 mW with strike lengths up to 85 m long.

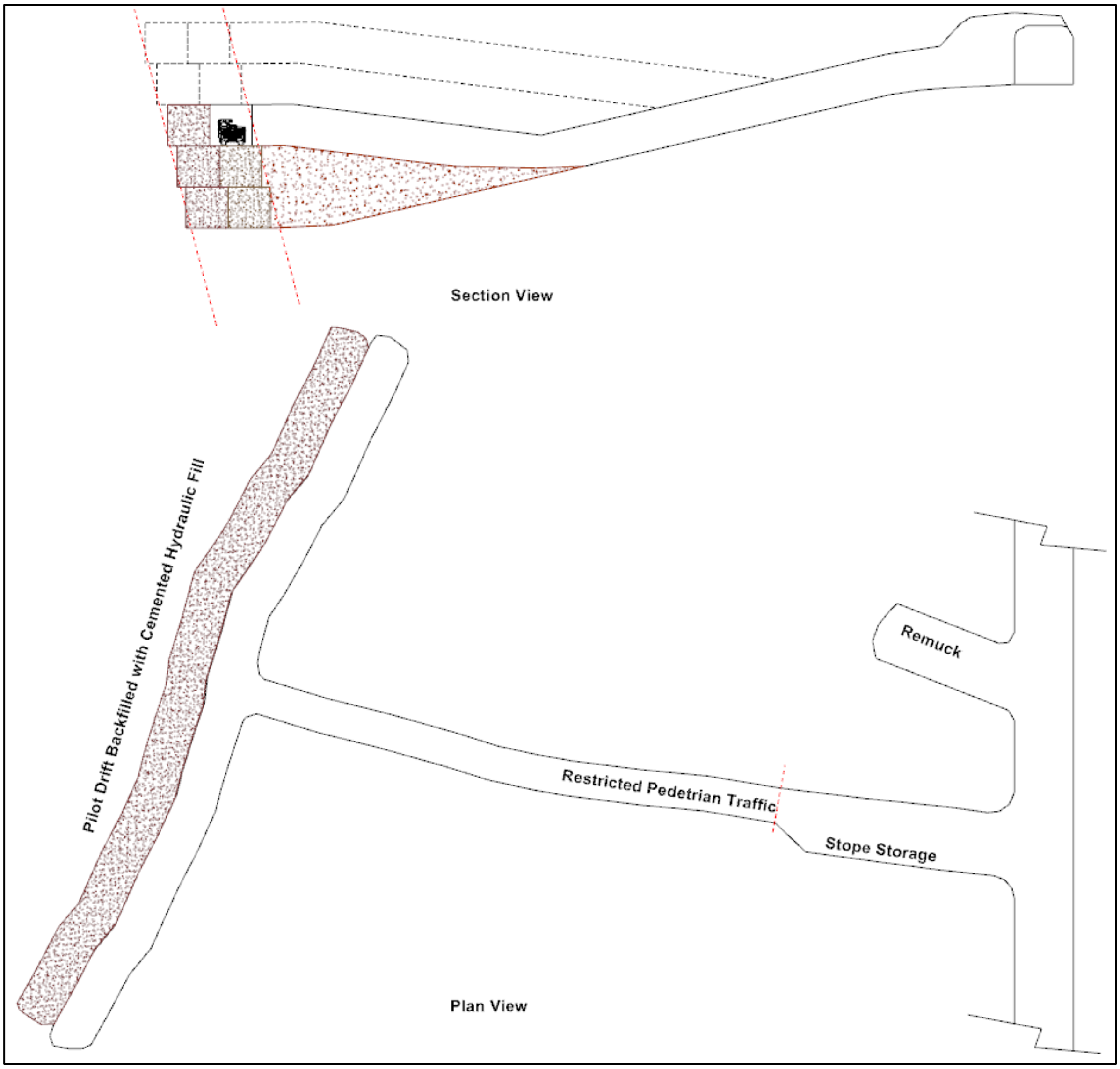
Based on the expected ground conditions and the wider stope widths, a fully MDF method is proposed, which is very similar to the MCF method described above. The average pilot drift will be 3.6 mW x 3.5 mH (see Figure 16-14).

The 8 Zone stopes will utilize short rounds (2.4 m) only and have heavier ground support requirements with the 50 mm of in-cycle shotcrete to be applied both walls and the back and ground support suitable for squeezing ground such as Super Swellex.

Advance rate was estimated at 3.9 m/d assuming two faces available 80% of the time, with smaller rounds (2.4 mL x 3.6 mW x 3.5 mW). This equates to 1.8 rounds per day on ore or 159 tpd.

The stopes will be blasted using ANFO and mucked using 7-t or 10-t class LHDs. In-cycle shotcrete will be applied using a remote spray arm and a mechanized bolter will be used to install ground support to minimize exposure of personnel.

Figure 16-14: Typical mechanized drift and fill (MDF) stope (plan and section)



Source: SRK (2025)

Mining progresses as follows:

1. Attack ramp is driven to the bottom elevation of the stoping block
2. An initial pilot drift is driven along strike following contact from the attack ramp to the ore body extents in both directions
    a. Following the footwall (FW) contact is likely preferred in this case due to the talc shist near the hanging wall (HW) contact
    b. Keep pilot drift narrow to control squeezing, PFS assumed 3.6 m wide, adjust width based on ground conditions

   c. Short rounds (2.4 m) to control squeezing and for grade control
   d. Drift height 3.5 m to allow use of mechanized bolter to install screen and 1.5 m ground support (Pm24 Super Swellex or equivalent)
   e. In-cycle shotcrete using remote spray arm (50 mm floor to floor)
3. Once pilot drift completed, install free draining backfill barricade and tight fill pilot drift with cemented hydraulic backfill
   a. Pilot drift driven at -3% grade to enable tight filling
   b. 5% binder assumed for hydraulic backfill to achieve 150 kPa strength
   c. May have to backfill pilot drift in sections as ore outlines for 8 Zone are very erratic
4. Once backfill in pilot drift cured sufficiently, drive a second cut beside pilot drift and repeat process until full width mined
5. Once whole cut is backfilled, the back of the attack ramp is slashed down and re-supported to access the next cut elevation
6. Repeat steps 1-5 until top cut mined out
7. Each attack ramp typically provides access to five cuts at 3.5 mH each

16.6 Mine Design

16.6.1 Development Design Criteria

All planned development is designed for use by trackless equipment:

- Ramps are 5 mW x 5 mH with a maximum gradient of +/-15%
- Slashing of track drifts from nominal 3 mW x 3 mH to 5 mW x 5 mH at track gradient, nominally +/-0.3%
- Level access drifts where trucks are expected to operate are 5 mW x 5 mH, variable grade
- Back at selected ramp intersections will be enlarged to accommodate truck loading in the intersection
- Any internal ramps or access drifts in waste where trucks are not expected to operate are 3 mW x 3.5 mH with pedestrian traffic restricted
- Level-to-level drop raises for ventilation are 3 m x 3 m, drilled with Epiroc Simba E70 top hammer drill
- Larger ventilation raises are designed to be raisebored at 3.0 m diameter or 3.65 m diameter as recommended by Ventilation team
- Ore and waste pass raises are designed as 3 m x 3 m Alimak raises

16.6.2 Stope Design Criteria

Longitudinal Retreat Longhole:

- Longhole Sill drifts designed at 4.0 mW x 3.5 mH to allow for parallel drilling at typical stope widths,
- Larger stopes will be drilled from topsill using downholes
- Smaller stopes may be taken as Uppers stopes and drilled from bottom sill
- Nominal 20 m sub-level spacing, adjusted to match historic track drift elevations using gradient polylines
- Minimum mining width 1.6 m undiluted, 2.2 m diluted
- Dilution included as 0.3 m ELOS applied to both hangingwall and footwall
- Additional dilution added based on depth:
  - <500 m depth – ELOS = 0.6 m
  - 500 m to 750 m depth – ELOS = 0.8 m
  - >750 m depth – ELOS = 1.2 m

Normal MCF:

- Round lengths have been limited to 2.4 m for the first year of production while the grade control practices are being developed and refined
- Round lengths increase to 3.0 m after the first year
- Fixed height of 3.5 m to allow use of mechanized bolter
- Minimum mining width 3.0 m undiluted, 3.8 m diluted
- Dilution included as 0.4 m ELOS (average) applied to both hangingwall and footwall using the variable overbreak or slough (VOS) feature in DSO
- The productivity assumption accounts for the erratic nature of the ore body with typical Normal MCF stope cut averaging 4.4 mW x 3.5 mH x 16-20 mL.

Historic MCF:

- Round lengths have been limited to 2.4 m for all Historic MCF stopes due to expected ground conditions
- Fixed height of 3.5 m to allow use of mechanized bolter
- Minimum mining width 3.0 m undiluted, 3.8 m diluted
- Dilution included as 0.4 m ELOS (average) applied to both hangingwall and footwall using the variable overbreak or slough (VOS) feature in DSO

- The productivity assumption accounts for the erratic nature of the ore body with typical Historic MCF stope cut averaging 4.3 mW x 3.5 mH x 16-20 mL.

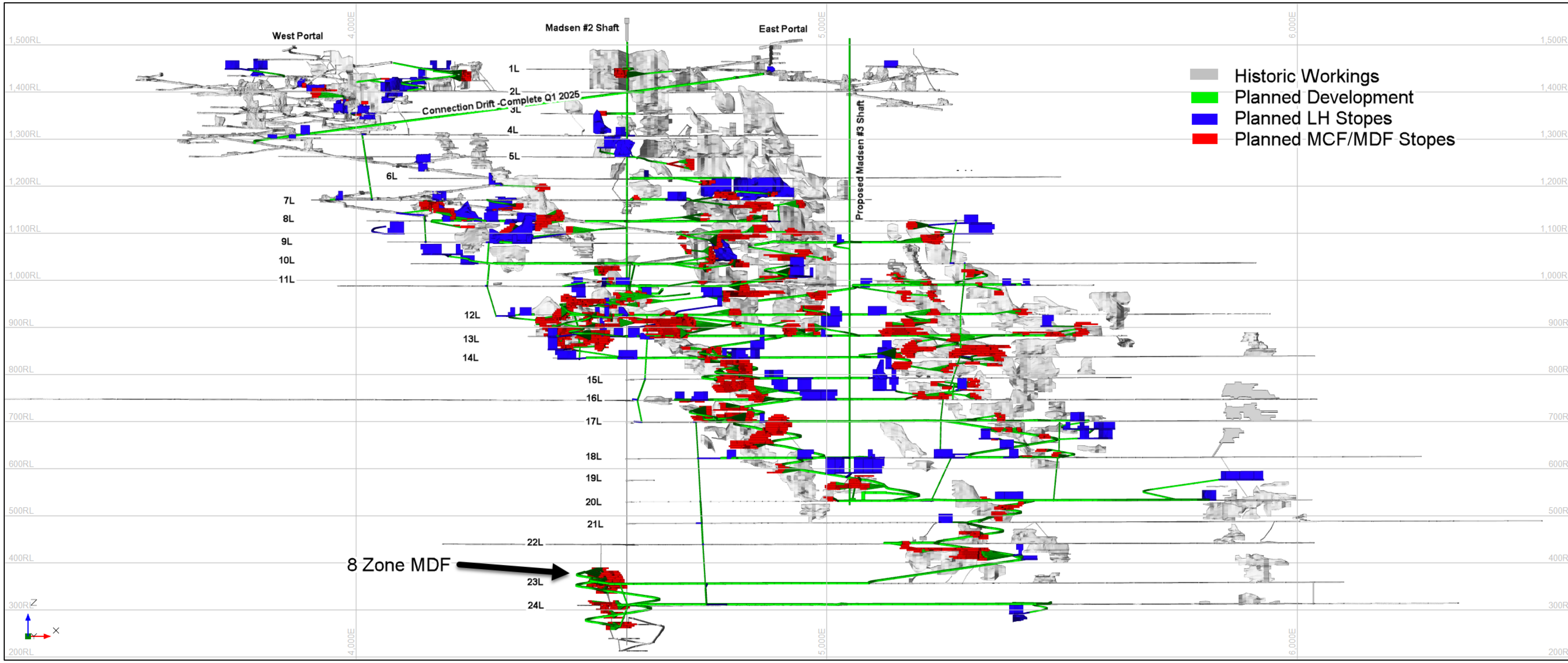
8 Zone MDF:

- Round lengths have been limited to 2.4 m for all 8 Zone MDF stopes due to expected ground conditions
- Fixed height of 3.5 m to allow use of mechanized bolter
- Minimum mining width 3.0 m undiluted, 3.8 m diluted
- Dilution included as 0.4 m ELOS (average) applied to both hangingwall and footwall using the variable overbreak or slough (VOS) feature in DSO
- The productivity assumption accounts for the erratic nature of the ore body with typical MDF stope cut averaging 7.2 mW x 3.5 mH x 65 mL, taken in at least two passes of 3.6 mW.

16.6.3 3D Underground Mine Model

A long section of the Madsen Mine showing the historic workings, planned development and planned stopes is shown in Figure 16-15. The connection drift, which will be completed in Q1 2025, is also shown.

Figure 16-15: Long section of Madsen Mine LOM Plan (looking north)



Source: SRK (2025)

16.6.4 Development Profiles

The total capital and operating development included in the PFS LOM Plan is shown in Table 16-9 with Figure 16-16 showing the total lateral development metres per day. Note that the operating development includes MCF/MDF stoping metres.

Table 16-9: Total capital and operating development

Description	Type	Metres
Capital	Lateral	31,870
Capital	Vertical	2,250
Total Capital		**34,120**
Operating	Lateral	49,612
Grand Total		**117,852**

Source: SRK (2025)

The total capital lateral development is shown in Table 16-10 with Figure 16-17 showing the total capital lateral development metres per day. Note that new ramp and level development is marked up 10% to account for development such as level sumps, electrical cutouts, refuge stations, truck loading areas, etc. that have not been designed as part of the PFS.

Table 16-10: Total capital lateral development

Description	Size	Mark-up	Metres
Ramp	5mW x 5mH	10%	7,350
Level Access	5mW x 5mH	10%	12,190
Slashing of Track Drifts	5mW x 5mH	0%	12,330
Total			**31,870**

Source: SRK (2025)

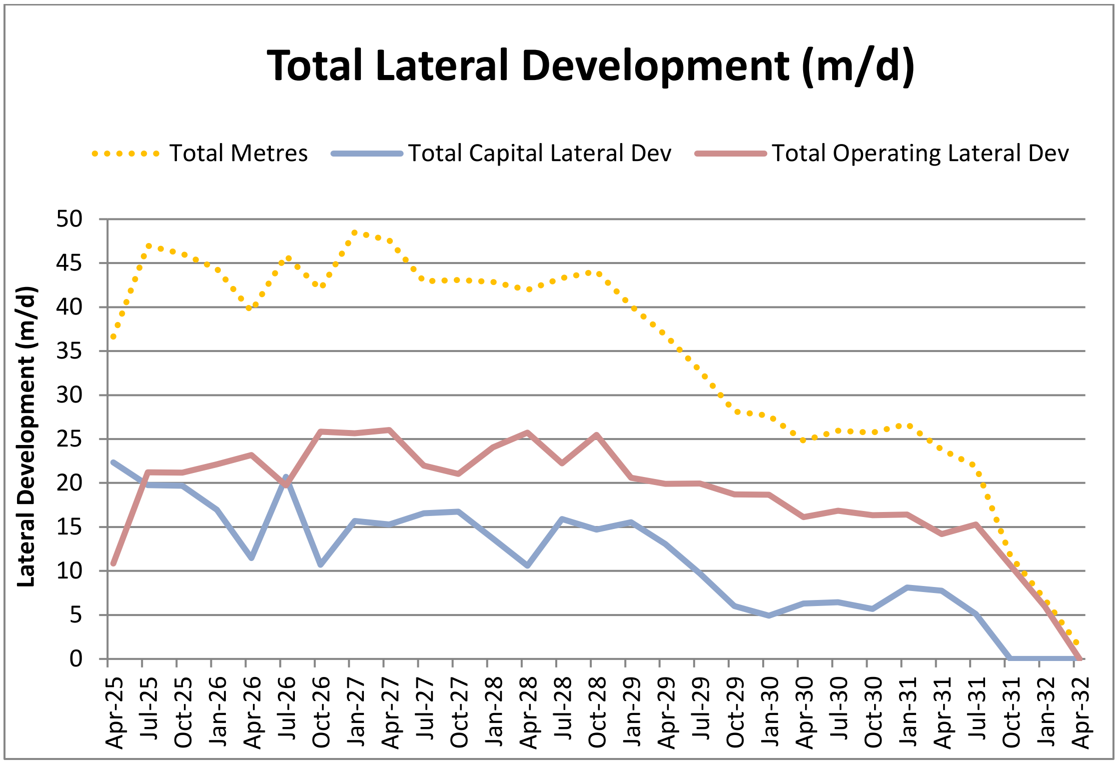
The total capital vertical development is shown in Table 16-11.

Table 16-11: Total capital vertical development

Description	Type	Size	Metres
Ore/Waste Passes	Alimak	3m x 3m	170
Ventilation Raises	Drop Raise	3m x 3m	890
	Raisebore	3.0m dia.	1,050
	Raisebore	3.65m dia.	140
Total			**2,250**

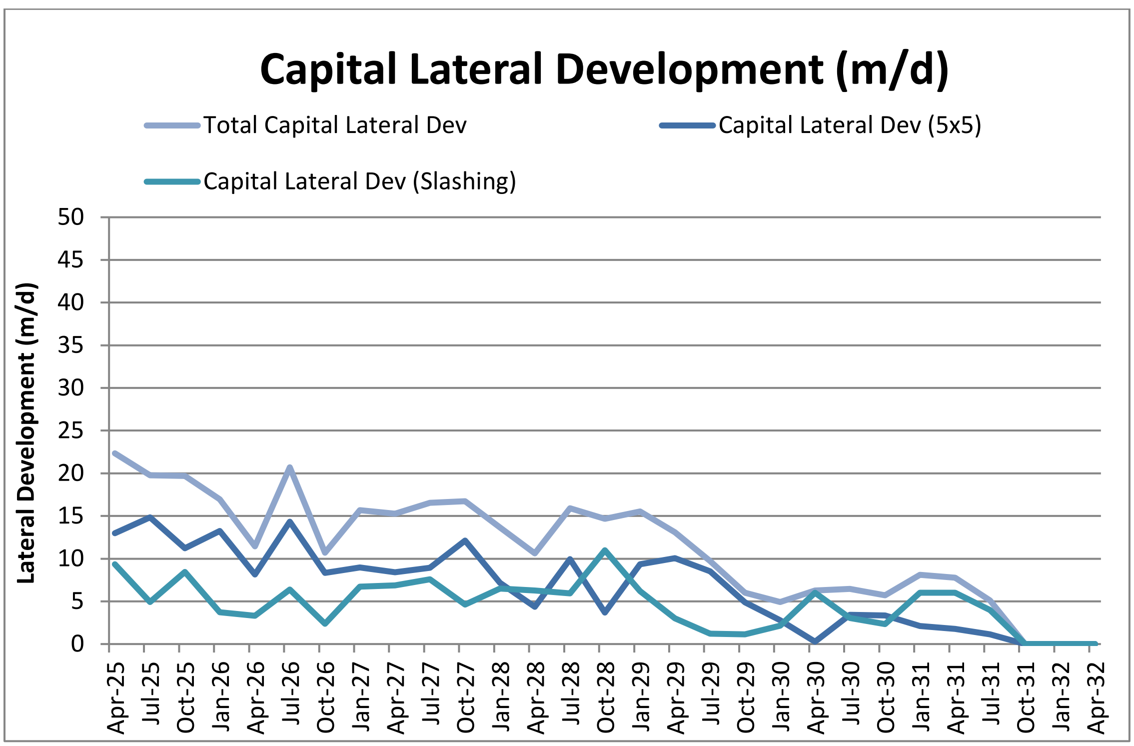
Source: SRK (2025)

Figure 16-16: Total lateral development profile



Source: SRK (2025)

Figure 16-17: Capital lateral development profile



Source: SRK (2025)

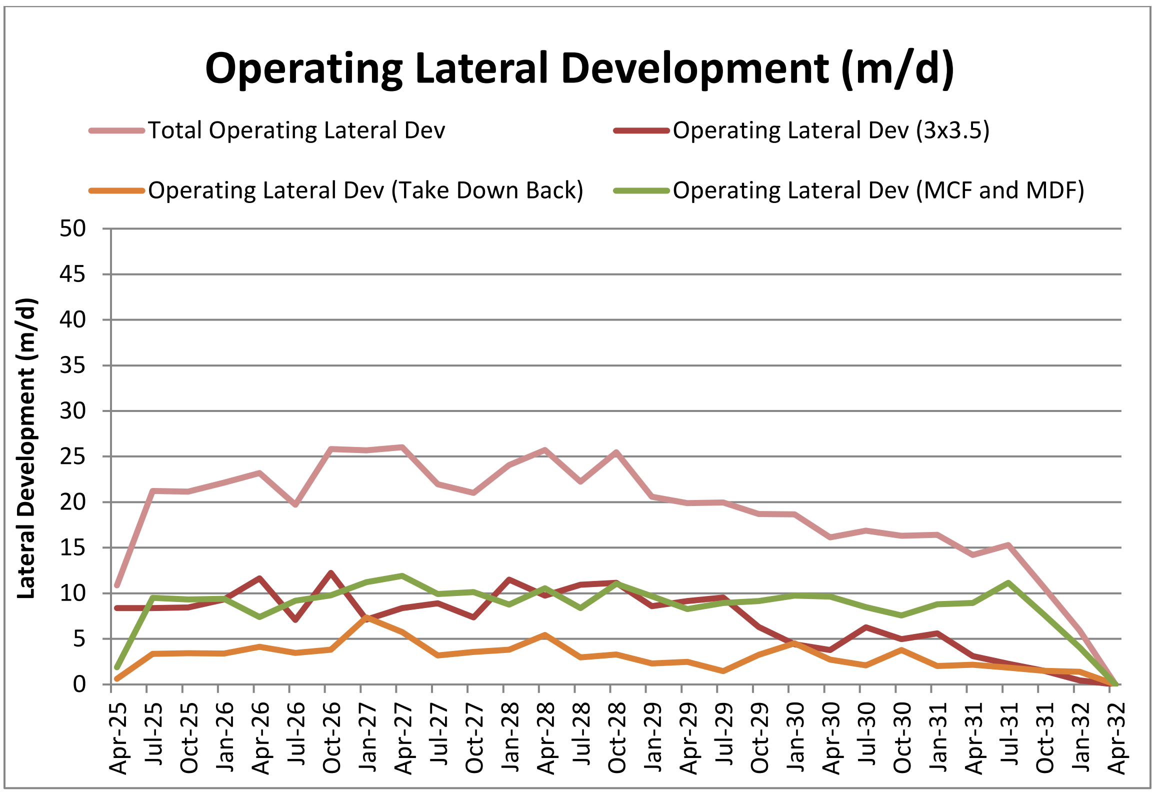
The total operating development included in the PFS LOM Plan is shown in Table 16-12 with Figure16-18 showing the total lateral development metres per day.

Table 16-12: Total Operating Lateral Development

Description	Size	Ore (m)	Waste (m)	Total (m)
Access Drifts, Attack Ramps & TDBs	3.0mW x 3.5mH TDB average 1.75mH	3,251	4,582	7,833
LH Sill Drifts	4.0mW x 3.5mH	2,392	3,099	5,491
MCF/MDF Stoping	Variable width x 3.5mH	26,794	-	26,794
Total		**32,437**	**17,175**	**49,612**

Source: SRK (2025)

Figure 16-18: Operating lateral development profile

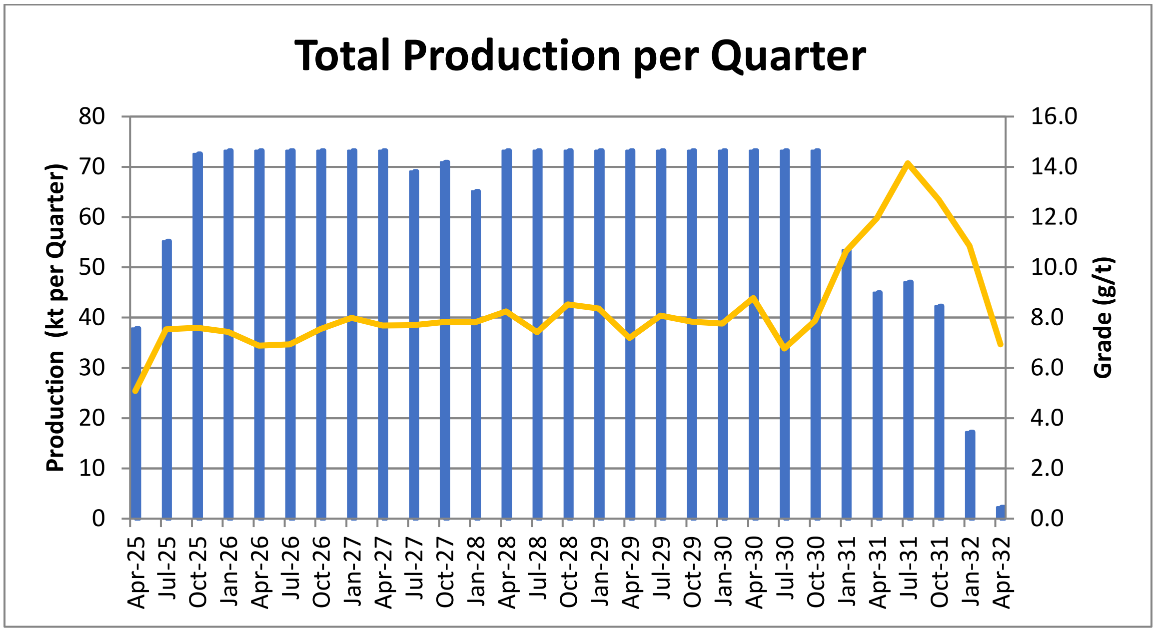


Source: SRK (2025)

16.6.5 Production Profile

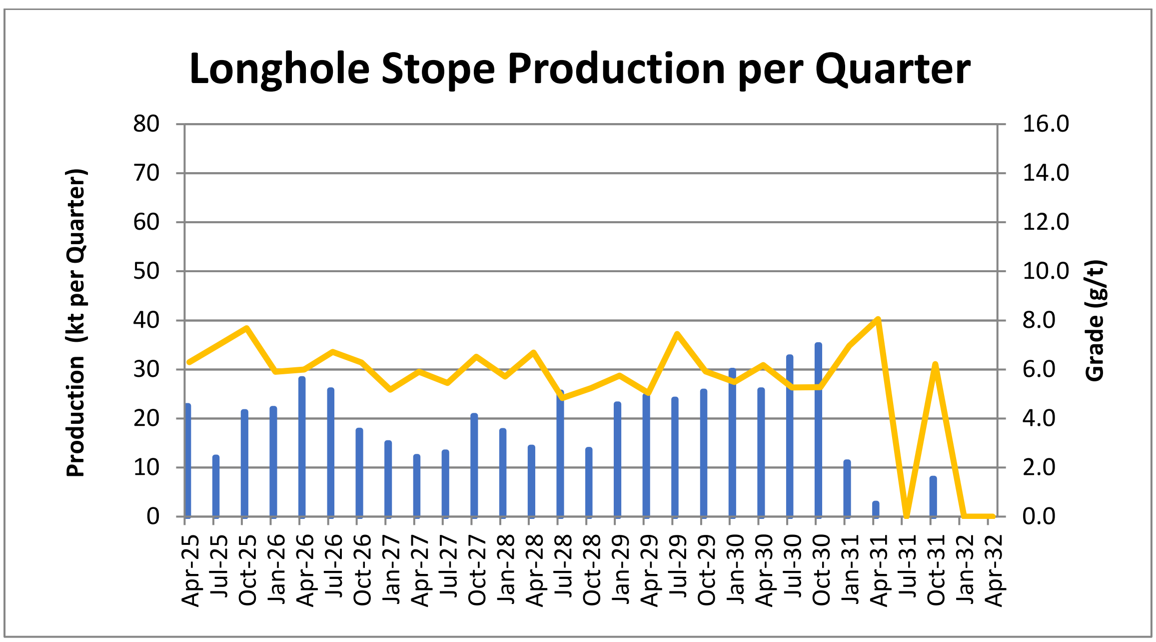
The PFS LOM Plan is based on a Probable mineral reserve of 1.823 Mt grading 8.16 gpt Au. The production rate has been levelled at 800 tpd of ore as shown in Figure 16-19, with Figure 16-20 and Figure 16-21 showing the production from LH stopes and MCF/MDF stopes, respectively. The increased grade shown in 2031/32 is due to mining of the 8 Zone MDF stopes.

Figure 16-19: Total production profile



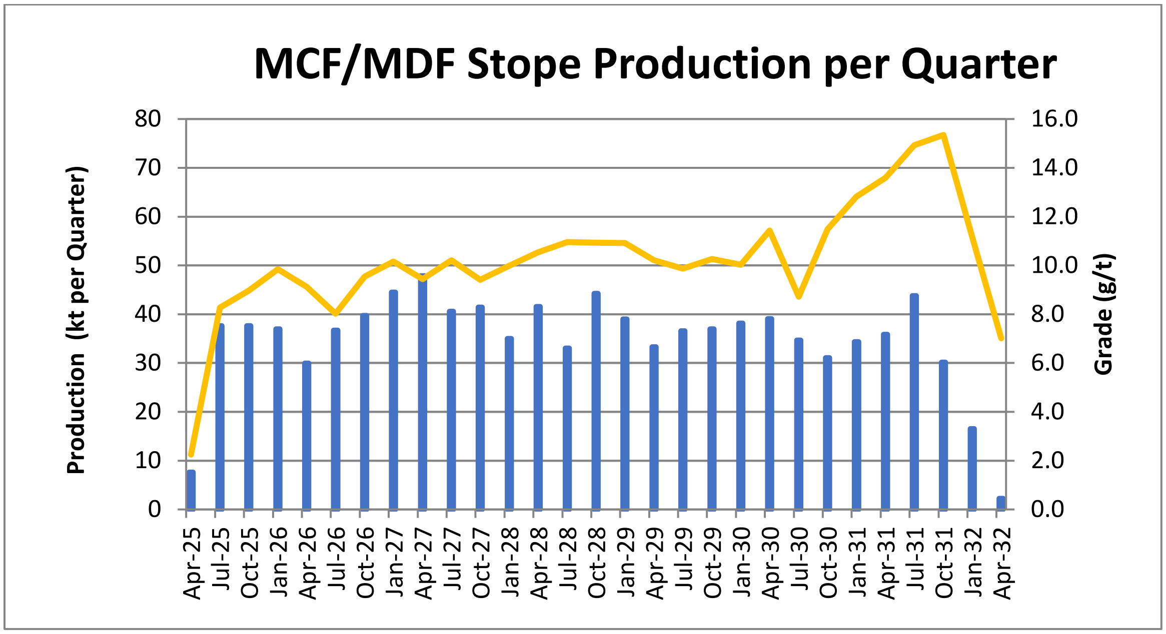
Source: SRK (2025)

Figure 16-20: Longhole stope production profile



Source: SRK (2025)

Figure 16-21: MCF/MDF production profile



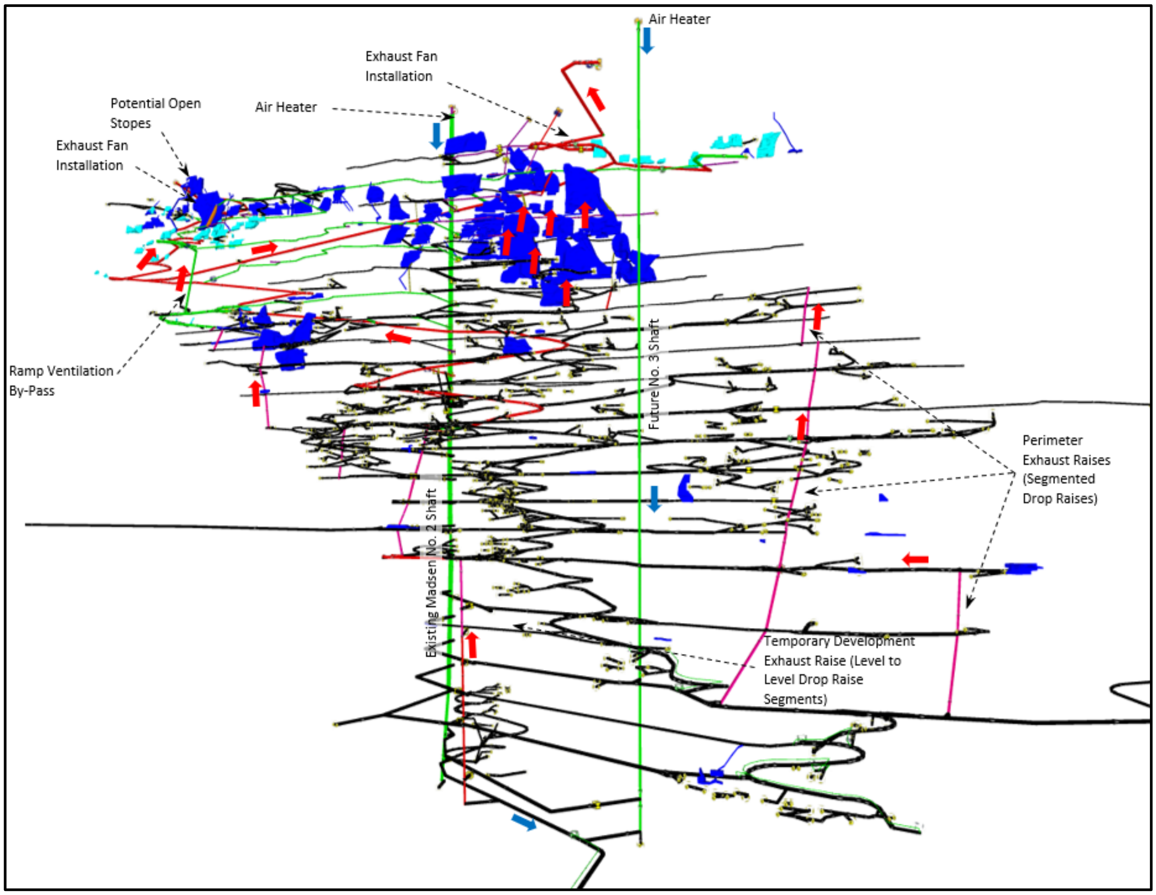
Source: SRK (2025)

16.6.6 Ventilation

The Madsen Mine has been in operation in various forms since 1937, which has resulted in a significant amount of open workings and both lateral and vertical infrastructure. The challenge with respect to the ventilation system required to be developed to support this type of an environment is that not all of the open airways are currently known. To support the production in the mining zones the ventilation system will need to have consistency. The levels accessing the mining zones will be rehabilitated or slashed to a larger dimension to support the new/reorganized mining areas. During the rehabilitation time, the degree to which stopes are open, and the various components of vertical infrastructure not identified on the as-builts but required to support historic mining activity, can be mapped and brought into the overall ventilation design.

The overall ventilation strategy for the Madsen Mine will provide control over the fresh air supply and routings, with uncontrolled or free exhaust routings to surface through open stopes, intermediate ore/waste passes, fringe or perimeter raises and decline accesses. The ventilation system will be developed or driven by two exhaust fans installed in the ramp accesses near the surface, and the fresh air will be provisioned by the existing Madsen Shaft #2 and the future Madsen Shaft #3. The overall ventilation layout is identified in Figure 16-22.

Figure 16-22: General ventilation layout

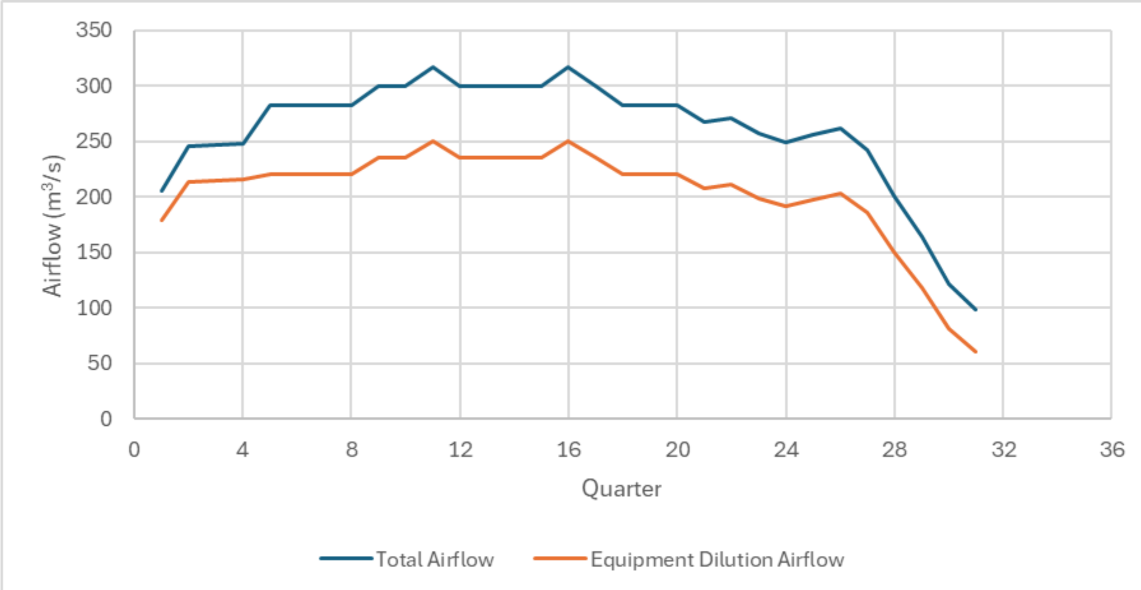


Source: SRK (2025)

Airflow Requirement

The airflow requirement through the mine is estimated by applying a utilization or availability factor for each piece of equipment and then multiplying the value by a standard diesel dilution factor, 0.06 m^3/s/kW. This type of airflow allocation approach assumes that not all equipment will be in use or in the mine at all times or operating under diesel power, rather it establishes an overall average airflow requirement. Haul trucks were assumed to have an 85% utilization, LHDs 75%, and all other ancillary equipment were provided at 25%. A general leakage value of 15% was used, and an additional 25 m^3/s for facilities like the crusher. The utilization factors incorporated for this design are based on both experience and similar projects. The higher impact equipment are assumed to have a much higher utilization rate, whereas the lower impact ancillary equipment are assumed to operate less. The overall airflow requirement is shown in Figure 16-23.

Figure 16-23: Overall system airflow requirement

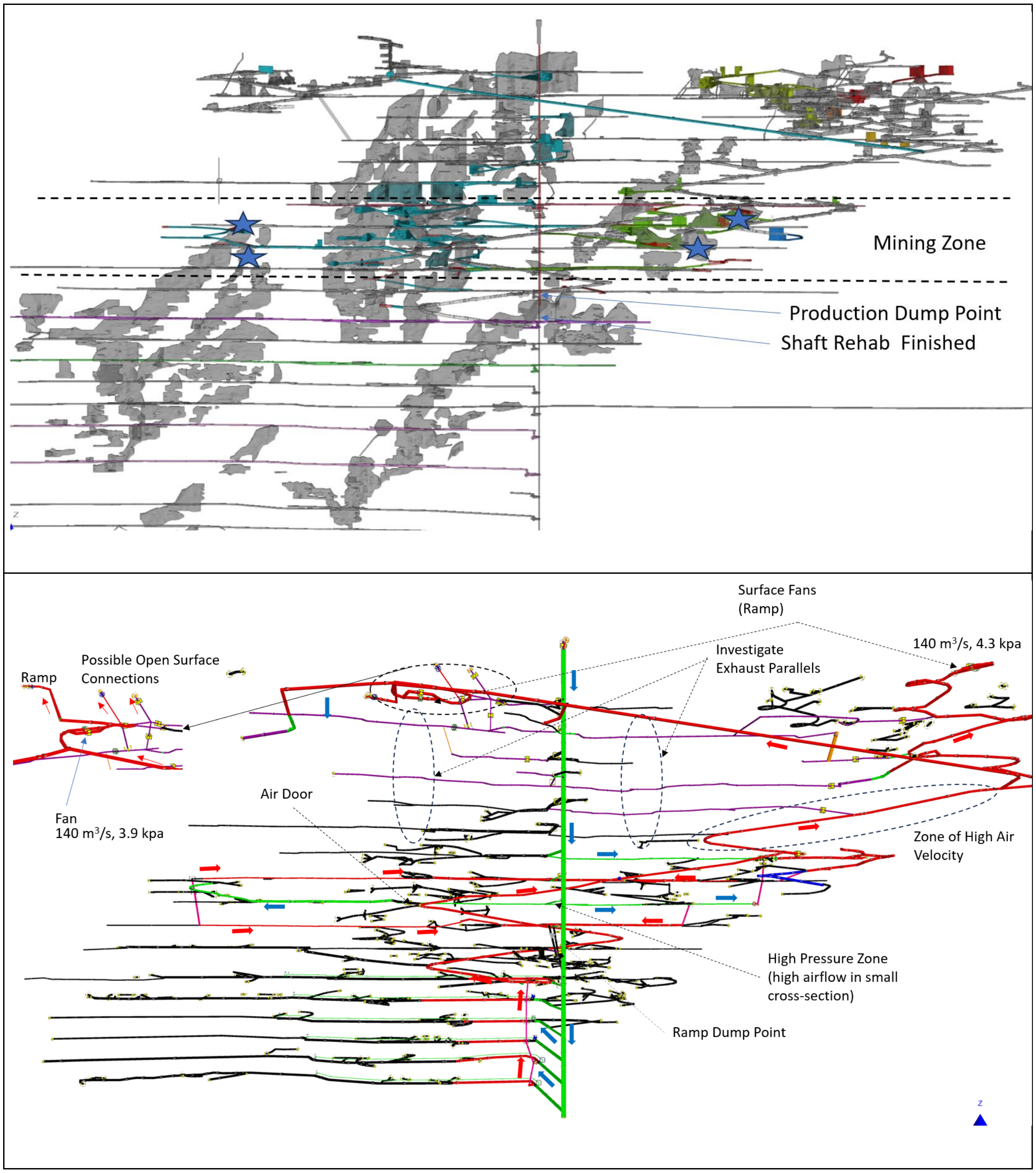


Source: SRK (2025)

Each basic production stope or development would require approximately 10 m^3/s delivered to support the operating LHD, with approximately 28 m^3/s required to support areas with both a truck and LHD in operation.

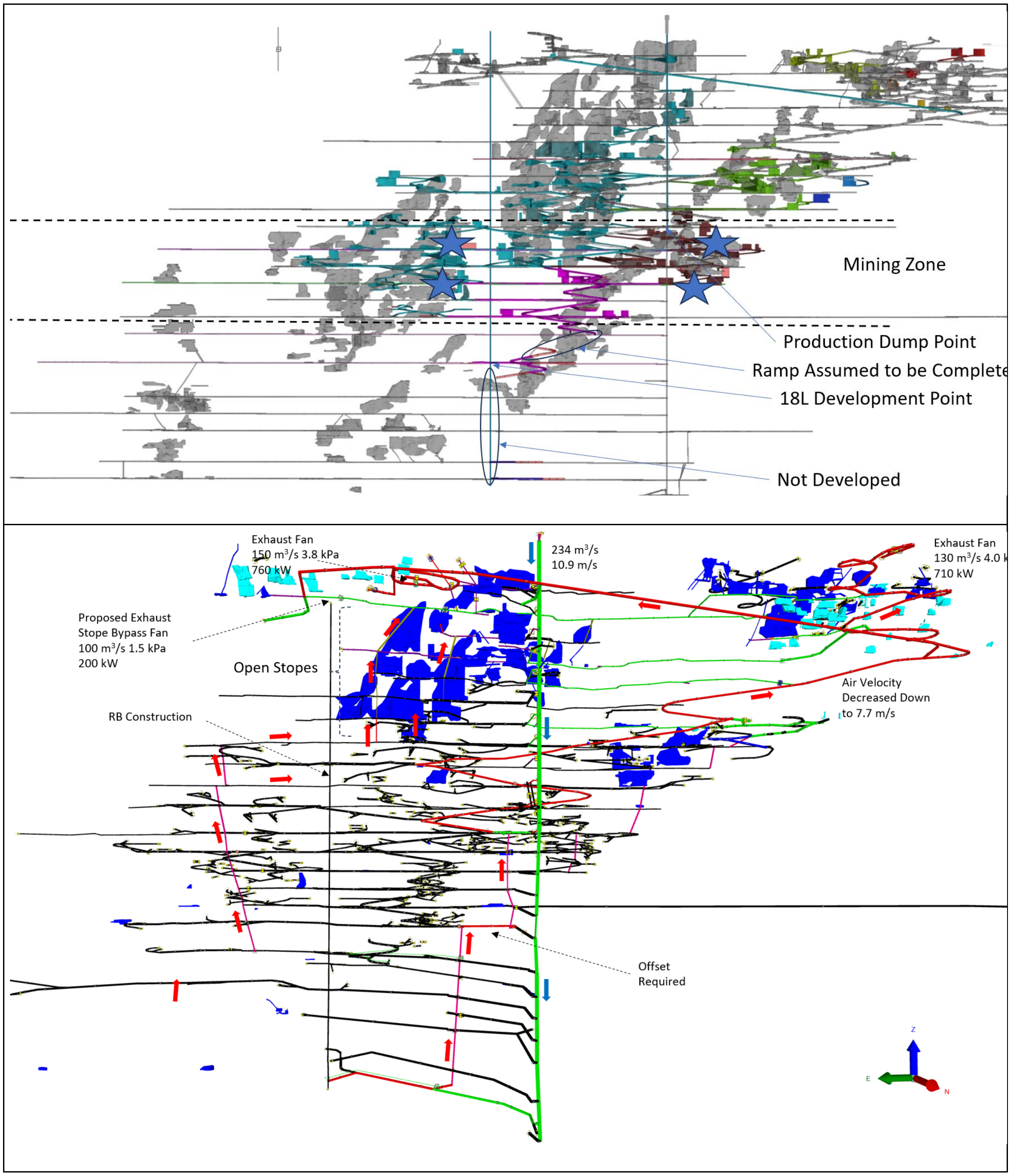
A series of three time phases were modeled to represent the initial production/development before and after Shaft #2 rehabilitation down to 12 Level, Shaft #3 development, and after Shaft #3 is fully developed as shown in Figure 16-24, Figure 16-25 and Figure 16-26.

Figure 16-24: End of Year 2025 before/after 12 Level rehab



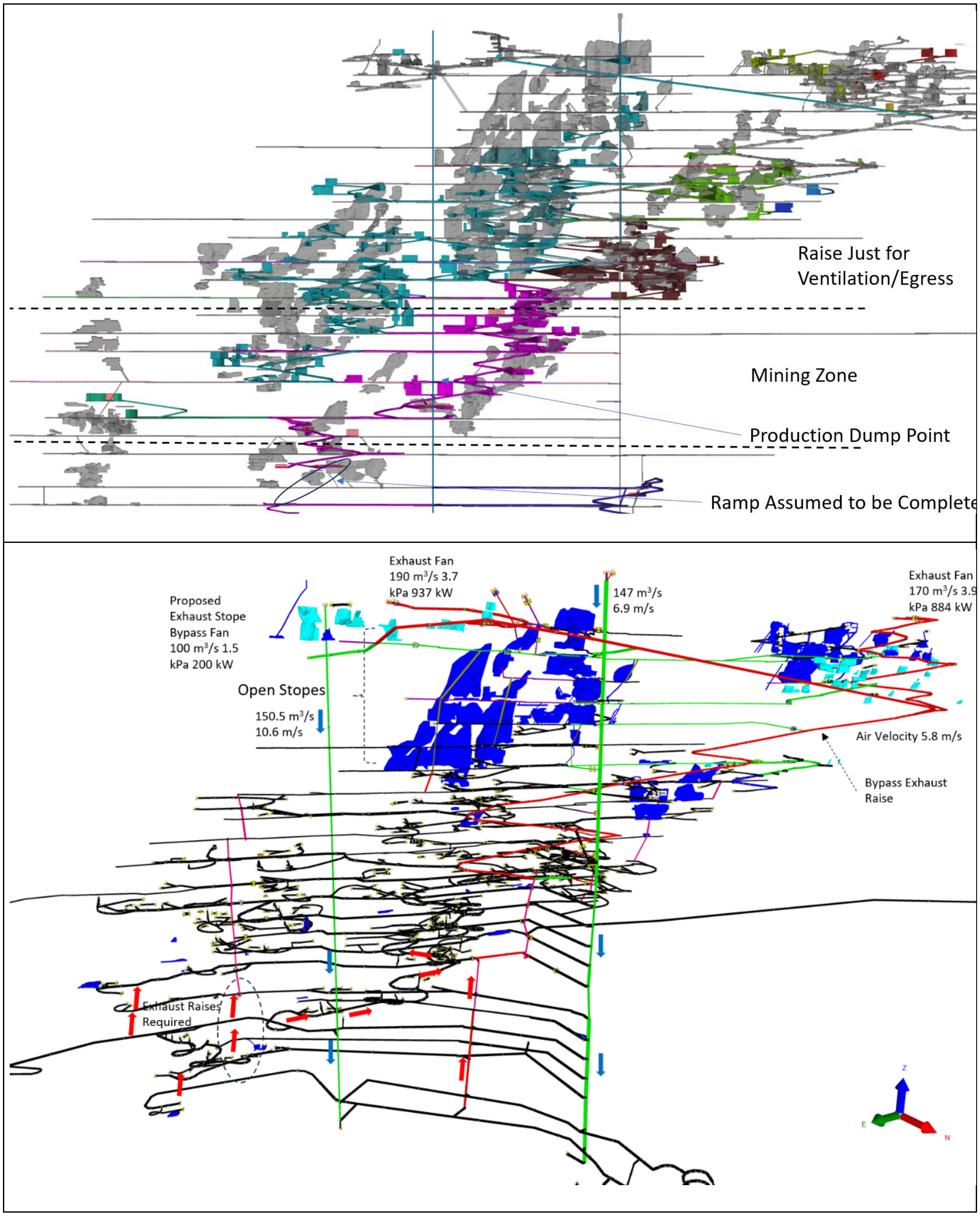
Source: SRK (2025)

Figure 16-25: Shaft #3 construction



Source: SRK (2025)

Figure 16-26: Shaft #3 fully integrated



Source: SRK (2025)

Ventilation Notes

There are several design constraints placed on the ventilation system that will require interim temporary ventilation configurations to support production, haulage and development activities.

Maximum Airflow Through Shaft #2

The airflow downcasting Shaft #2 is held to approximately 235 m^3/s to limit the maximum air velocity to 11 m/s. As the airflow descends the shaft, the air velocity will decrease through leakage and compression. In the early time periods leakage from Shaft #2 will be less impactful because of the limited shaft connections. If the maximum air velocity is increased to 12 m/s then the maximum airflow through achieved through the shaft could be approximately 250 m^3/s, while 13 m/s would provide approximately 270 m^3/s. The air velocity through Shaft #2 is conservative because it is assumed that the entirety of the ladderway compartment is removed from the shaft opening.

Possible Early Year Airflow Reduction

During the early years the haul trucks will be drawing ore/waste directly to surface. During this time there will be periods when the trucks will not be operating in the underground environment. This may provide a slight reduction in the airflow requirement.

Quarter 11 and 16 Airflow Requirement Spikes

The elevated airflow requirements in Quarters 11 and 16 result from spikes in the quantity of operating diesel equipment. It may be possible to refine the schedule to smooth the peaks.

Temporary Reduction in Facility Ventilation

Because all of the fresh air is drawn down Shaft #2 any facilities developed in the shaft accesses would be provided with fresh air before the airflow is contaminated by any diesel equipment. As the mining areas extend deeper the airflow would be drawn directly into the haulage ramp (limiting its full utilization), however, in the early time phases the airflow would be drawn from the shaft closer to the working levels and active haulage ramp area.

Air Velocity Through Haulage Ramp

The air velocity through the haulage ramp has the potential to become excessive. The haulage ramp will be functioning as an exhaust, in parallel with any open stope, ore pass, or internal raise. These have not been formally designated yet but have been incorporated into the model with generally placed internal raises. An additional internal raise will likely be required to decrease the air velocity in the ramp.

Auxiliary Ventilation

The basic ventilation system draws airflow from the shaft, across the level, to the perimeter exhaust raise, then up through both the connected perimeter exhaust raises, open stopes, and upper haulage ramp system. A short auxiliary ventilation system will be required to draw airflow from the level access into the stope, and a longer auxiliary ventilation system will be required to draw airflow from the shaft to the level rehabilitation/development heading. A 1 m diameter duct system was incorporated for both systems for both simplicity and for the minimization of pressure/power. A short summary of the auxiliary fan sizes is shown in Table 16-13.

Table 16-13: Auxiliary ventilation fans

Auxiliary Ventilation System	Power (kW)
Stope Ventilation	75
Long Development (Twin 100 kW)	200
Alcove	15

Source: SRK (2025)

Air Heating

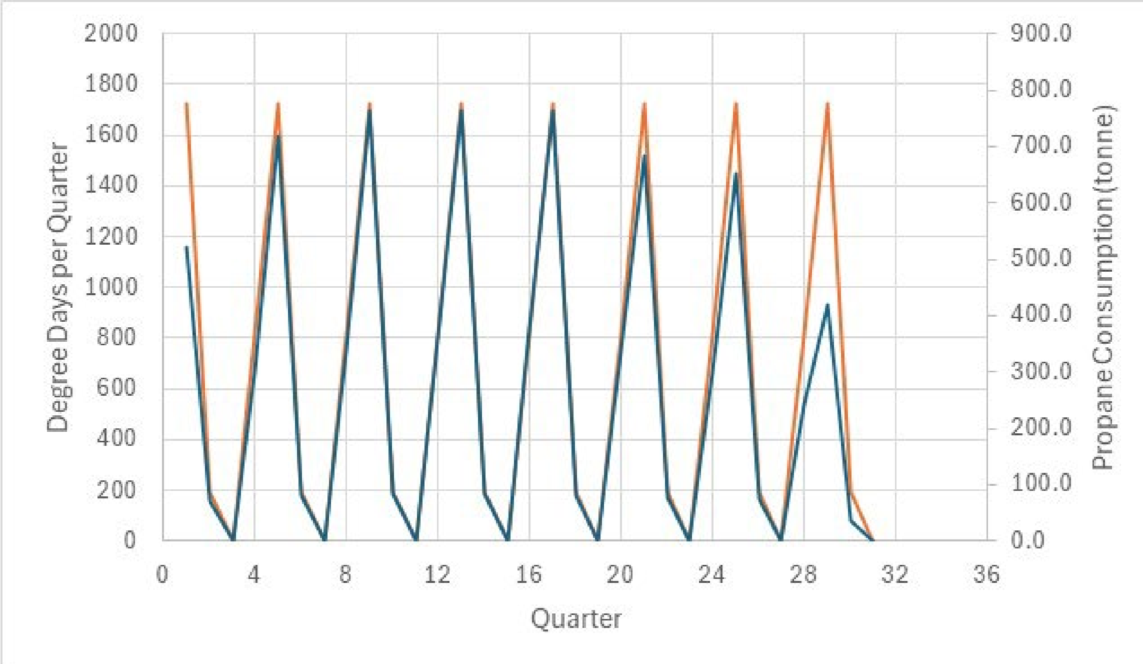
The project site in the Red Lake mining district typically experiences freezing temperatures during the months of October to April. Typical weather station values were obtained for Red Lake, Ontario from degreedays.net. These values were used to identify the operating duty of the air heaters. The maximum heater duty point is based on a low temperature of -34°C providing heating to +5°C as shown in Table 16-14, the quarterly degree days and propane consumption is shown on Figure 16-27. To support full air heating at the lowest temperatures an installation capable of providing a maximum 7.06 MW would be required as shown in Table 16-14 for the top of Shaft #2 and Shaft #3. The air heaters could be reduced if during the peak low temperature time periods the airflow through the ventilation system is reduced. This would require a temporary alteration to the operating equipment fleet.

Table 16-14: Air heater maximum duty calculation (two required for mine)

Parameter	Value
Airflow	150.0 m^3/s
Air density	1.15 kg/m^3
Air density at -40°C	1.20 kg/m^3
Temperature rise	39 °C (from -34 to +5)
Mass flow of air (m_a)	180.0 kg/s
Heat required ($q_a = m_a C_{p\Delta} T$)	7.06 MW (= MJ/s)
Heat required	24.07 MBTU
Calorific value of liquid propane	25.3 MJ/l
LPG propane flow rate	0.2789 l/s

Source: SRK (2025)

Figure 16-27: Quarterly degree days and propane consumption



Source: SRK (2025)

16.7 Mine Production Schedule

16.7.1 Project Development Phases

As the mine was previously in commercial production under Pure Gold, the mine is expected to ramp up to 800 tpd and achieve commercial production in Q1 2026. WRLG has been executing some pre-production work and test mining to prepare the mine for production, including developing the connection drift, continuing dewatering and beginning rehabilitation of Madsen Shaft #2.

First development ore is expected in Q2 2025 with commercial production expected in Q1 2026 when Madsen Shaft #2 loading pocket is commissioned. Until the loading pocket is operational, all ore will be trucked to surface via ramp. Once the loading pocket is ready, ore will be trucked to the 10 Level grizzly or surface depending on stope location. As mining progresses, more of the ore will be hoisted as the near surface stopes are depleted.

The proposed Madsen Shaft #3 is expected to be operational by the end of 2028 as mining progresses deeper. Ore will then be trucked to either the 18 Level grizzly for Shaft #3 or the 10 Level grizzly for Shaft #2 depending on stope location. As mining progresses, more of the ore will be hoisted as the near surface stopes are depleted.

16.7.2 Mine Sequencing

Mine sequencing is generally top down by mining area with stope sequence being bottom up within the mining areas.

16.7.3 Development Advance Rate

The single face advance rates shown in Table 16-15 were used in the PFS LOM Plan.

Table 16-15: Single Face Development Advance Rates

Description	Size	Unit	Rate
Ramps	5.0 mW x 5.0 mH	m/d	4.5
Slash Track Drift	5.0 mW x 5.0 mH	m/d	6.0
Level Access	3.0 mW x 3.5 mH	m/d	6.0
Sill Drifts	3.0 mW x 3.5 mH	m/d	3.5
TDBs	3.0 mW x 1.75 mH	m/d	6.8
Raises	Variable	m/d	2.0

Source: SRK (2025)

16.7.4 Stope Production Rates

The LH stopes will be backfilled with URF unless the situation requires CHF for sequencing. As there are many small LH mining areas, the CHF cure time is not material to the overall mine production rate as there is a lot of flexibility in sequencing. Table 16-16 shows the full cycle productivity for LH stopes with URF and CHF backfill.

Table 16-16: Longhole Stope Production Rates

Description	Quantity	Units	Rate	Unit Rate	LH w/ URF	LH w/ CHF
Stope Preparation	3	days			3 days	3 days
Drilling Slot	447	m	180	m/d	3 days	3 days
Drilling Production	670	m	180	m/d	4 days	4 days
Blasting					5 days	5 days
Mucking	3500	t	600	tpd	6 days	6 days
Delays					3 days	3 days
Backfill Preparation					0 days	2 days
Backfill URF	2,381	t	350	tpd	7 days	0 days
Backfilling	2,381	t	675	tpd	0 days	4 days
Backfill Cure					0 days	14 days
Total	**3500**	**t**			**30 days**	**43 days**
Full Cycle Productivity (tpd)					**116**	**82**

Source: SRK (2025)

The normal MCF stope productivity is shown in Table 16-17 for 2.4 m rounds and Table 16-18 for 3.0 m rounds and an average strike length of 20 m.

Table 16-17: Normal MCF production rate (2.4 m rounds)

Description	Quantity	Units	Rate	Unit Rate	Normal MCF
Attack Ramp	46	m	3.5	m/d	13 days
Cut #1	906	t	191	tpd	5 days
Backfill URF	616	t	350	tpd	2 days
TDB #1	9	m	6.8	m/d	1 days
Cut #2	906	t	191	tpd	5 days
Backfill URF	616	t	350	tpd	2 days
TDB #2	18	m	6.8	m/d	3 days
Cut #3	906	t	191	tpd	5 days
Backfill URF	616	t	350	tpd	2 days
TDB #3	27	m	6.8	m/d	4 days
Cut #4	906	t	191	tpd	5 days
Backfill URF	616	t	350	tpd	2 days
TDB #4	36	m	6.8	m/d	5 days
Cut #5	906	t	191	tpd	5 days
Backfill URF	616	t	350	tpd	2 days
Total	**4,528**	**t**			**59 days**
Full Cycle Productivity (tpd)					**76**

Source: SRK (2025)

Table 16-18: Normal MCF production rate (3.0 m rounds)

Description	Quantity	Units	Rate	Unit Rate	Normal MCF
Attack Ramp	46	m	3.5	m/d	13 days
Cut #1	906	t	245	tpd	4 days
Backfill URF	616	t	350	tpd	2 days
TDB #1	9	m	6.8	m/d	1 days
Cut #2	906	t	245	tpd	4 days
Backfill URF	616	t	350	tpd	2 days
TDB #2	18	m	6.8	m/d	3 days
Cut #3	906	t	245	tpd	4 days
Backfill URF	616	t	350	tpd	2 days
TDB #3	27	m	6.8	m/d	4 days
Cut #4	906	t	245	tpd	4 days
Backfill URF	616	t	350	tpd	2 days
TDB #4	36	m	6.8	m/d	5 days
Cut #5	906	t	245	tpd	4 days
Backfill URF	616	t	350	tpd	2 days
Total	**4,528**	**t**			**54 days**
Full Cycle Productivity (tpd)					**84**

Source: SRK (2025)

The Historic MCF stope productivity is shown in Table 16-19 for 2.4 m rounds and an average strike length of 20 m.

Table 16-19: Historic MCF production rate (2.4 m rounds)

Description	Quantity	Units	Rate	Unit Rate	Historic MCF
Attack Ramp	46	m	3.5	m/d	13 days
Cut #1	885	t	134	tpd	7 days
Backfill URF	602	t	350	tpd	2 days
TDB #1	9	m	6.8	m/d	1 days
Cut #2	885	t	134	tpd	7 days
Backfill URF	602	t	350	tpd	2 days
TDB #2	18	m	6.8	m/d	3 days
Cut #3	885	t	134	tpd	7 days
Backfill URF	602	t	350	tpd	2 days
TDB #3	27	m	6.8	m/d	4 days
Cut #4	885	t	134	tpd	7 days
Backfill URF	602	t	350	tpd	2 days
TDB #4	36	m	6.8	m/d	5 days
Cut #5	885	t	134	tpd	7 days
Backfill URF	602	t	350	tpd	2 days
Total	**4,425**	**t**			**69 days**
Full Cycle Productivity (tpd)					**64**

Source: SRK (2025)

The 8 Zone MDF stope productivity is shown in Table 16-20 for 2.4 m rounds and an average 65 m strike length.

Table 16-20: 8 Zone MDF Production Rate (2.4 m rounds)

Description	Quantity	Units	Rate	Unit Rate	8 Zone MDF
Attack Ramp	46	m	3.5	m/d	13 days
Pilot Drift #1	2,408	t	159	tpd	15 days
Backfill Preparation					2 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Complete Cut #1	2,408	t	159	tpd	15 days
TDB #1	9	m	6.8	m/d	1 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Pilot Drift #2	2,408	t	159	tpd	15 days
Backfill Preparation					2 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Complete Cut #2	2,408	t	159	tpd	15 days
TDB #2	18	m	7	m/d	3 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Pilot Drift #3	2,408	t	159	tpd	15 days
Backfill Preparation					2 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Complete Cut #3	2,408	t	159	tpd	15 days
TDB #3	27	m	7	m/d	4 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Pilot Drift #4	2,408	t	159	tpd	15 days
Backfill Preparation					2 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Complete Cut #4	2,408	t	159	tpd	15 days
TDB #4	36	m	7	m/d	5 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Pilot Drift #5	2,408	t	159	tpd	15 days
Backfill Preparation					2 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Complete Cut #5	2,408	t	159	tpd	15 days
Backfill Preparation					2 days
Backfill CHF	1,638	t	675	tpd	2 days
Backfill Cure					3 days
Total	**24,079**	**t**			**245 days**
Full Cycle Productivity (tpd)					**98**

Source: SRK (2025)

16.8 Mining Operations

16.8.1 Underground Mining Fleet

The majority of the mining fleet was purchased by the previous owner with some additions having been made by WRLG through purchase and leasing agreements. Most of the equipment shown on Table 16-21 is currently being leased with only the service truck and ATVs being rented. The plan includes the Epiroc M2C jumbo being returned and replaced as the rebuilt unit is unreliable. The two Epiroc ST2G LHDs are also to be returned and are being replaced by new CAT R1300 LHDs.

Table 16-21: Equipment currently leased or rented

Company	Equipment	Rent/Lease	No.
AM&T	Getman 4000 Anfo Loader	Lease	1
Epiroc	Epiroc MT42 Truck	Lease	2
AM&T	Epiroc M2C Jumbo	Lease - Replace	1
Kovatera	Kovatera K200 Service Truck	Rent	1
Resource	4x4 ATV Kubota RVT X1140	Rent	1
Toromont	CAT R1700 LHD	Lease	1
MacLean	MEM Scissor Lift	Lease	2
AM&T	Epiroc ST2G LHD	Lease - Return	2

Source: SRK (2025)

Table 16-22 shows the expected fleet composition as of March 31, 2025 and at steady state for 2026 onward. To improve safety and productivity in general and enable the safe mining of MCF stopes in close proximity to the historic workings and 8 Zone, the mine is moving towards greater mechanization. This includes the use of one boom jumbos equipped with split feeds for MCF stoping, replacing manual bolting from scissor lifts with mechanized bolters and purchasing a mechanized production drill and a remote spray arm for shotcreting.

Table 16-22: Underground mining fleet

Function	Type	As of March 31, 2025[1]	Steady State 2026+
Primary Equipment			
2 Boom Jumbo	Diesel/Electric	3	3
1 Boom Jumbo	Diesel/Electric	1	4
2 or 3 Boom Longtom	Pneumatic	3	3
Mechanical Bolter	Diesel/Electric	0	5
14-t Development LHD	Diesel	1	1
10-t Development/Production LHD	Diesel	4	6
7-t Development/Production LHD	Diesel	2	5
40-t Truck	Diesel	2	2
30-t Truck	Diesel	3	3
20-t Truck	Diesel	1	1
Production Drill (Manual Top Hammer)	Pneumatic	3	3
Production Drill (Mechanized Top Hammer)	Diesel/Electric	0	1
Secondary Equipment			
LHD with Forks	Diesel	1	1
Scissor Lift	Diesel	5	5
ANFO Loader (Development)	Diesel	2	2
Boom Truck	Diesel	2	2
UG Grader	Diesel	1	1
Telehandler	Diesel	1	1
Remote Spray Arm	Pneumatic	1	1
Dry Shotcrete Machine	Pneumatic	2	2
Personnel Carrier	Diesel	4	4
UG Tractor	Diesel	2	2
Service Vehicle	Diesel	2	2
Light Vehicles	Diesel	10	10
TOTAL #		**56**	**70**

[1] Equipment on site or ordered as of March 31, 2025

Source: SRK (2025)

16.8.2 Underground Work Force

The Madsen Mine is expected to utilize an underground workforce averaging 221 people over the PFS LOM Plan as shown in Table 16-23. There are currently approximately 140 workers employed, of which approximately 60% live locally. The operation is actively recruiting with a 114-person camp and a new mine dry under construction to accommodate the additional people.

Peak labour requirements occur in Q4 of 2025 as the mine reaches full production while significant amounts of capital development are ongoing to access new mining areas. Steady state is achieved in 2026 through 2028, with reductions in the work force starting in 2029 as lateral development requirements taper off.

Table 16-23: Underground work force

	Peak (Q4 2025)	Average (2026-28)	Average LOM
Development Miners	69	45	37
Production Miners	70	70	63
Haulage and Shaft	29	33	30
Mine Services	20	16	16
Mine Maintenance	45	48	45
Technical Services	23	23	22
Supervision & Management	8	9	9
Total	**265**	**244**	**221**

Source: SRK (2025)

17 Recovery Methods

Madsen includes five different lenses – Austin, South Austin, A3, McVeigh and 8 Zone. The 2018 test programs completed at Base Metallurgical Labs in Kamloops, BC (BL288) and the recently operating Madsen plant have demonstrated that gravity concentration followed by pre-oxidation, cyanide leach, carbon adsorption/desorption and electrowinning can yield gold extraction in the range of 95.7%.

At steady state operation, an average of 800 tpd of material will be processed in a plant that consists of primary crushing, followed by grinding to 80% passing 75 µm using a semi-autogenous grinding (SAG) mill and ball mill. Gravity concentration will recover gold from the SAG screen undersize and ball mill discharge. Cyclone overflow will be thickened to 50% solids in a pre-leach thickener, then pre-aerated with oxygen followed by a 24-hour cyanide leach at a cyanide concentration of 150-170 ppm and a pH of 11.0 in five leach tanks. Gold in solution will then be recovered via carbon-in-pulp (CIP) adsorption in six CIP tanks with a residence time of five hours, followed by acid wash, elution, and refining to produce gold dore on site.

After cyanide destruction, the CIP tailings will be pumped to the TMF initially. Starting later in Year 1, a thickened tailings will be pumped into open stopes using a hydraulic backfill system.

17.1 Madsen Plant

The plant, including the expansion equipment, consists of the following operational units:

- Crushing: primary semi-mobile jaw crusher to produce a final product P80 in the range of 75 mm
- Crushed Material Bin and Reclaim: a 600-t live capacity bin feeding onto the SAG mill feed conveyor
- Primary Grinding: a semi-autogenous grinding (SAG) mill operating in open circuit
- Secondary Grinding: a ball mill in reverse closed circuit with a cluster of hydro-cyclones, producing a final product P80 of 75 µm
- Gravity Concentration: two semi-continuous gravity concentrators to recover gravity recoverable gold from the SAG screen undersize and ball mill discharge
- Intensive Leach: an intensive leach reactor to leach gold from the gravity concentrate for further processing in the refinery to produce gold doré
- Pre-leach Thickening: a high-rate thickener to achieve an underflow solids density of 50% prior to leaching
- Pre-oxidation: one agitated tank sparged with oxygen to oxidize the slurry prior to leaching
- Cyanide Leaching: five agitated leach tanks, with approximately 24 hours of retention time to leach gold into solution
- Carbon in Pulp (CIP): six CIP tanks, providing approximately 5 hours of retention time to adsorb gold–cyanide complexes onto the pores of activated carbon

- Carbon Elution and Regeneration: acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold-rich solution, and thermal regeneration of carbon to remove organic foulants
- Gold Refining: precious metal electrowinning (sludge production), filtration, drying, and refining to produce gold doré
- Cyanide Destruction: two agitator tanks, one operating and one on standby, to reduce the CN_{WAD} (weak acid dissociable) concentration in the CIP tailings to less than 5.0 mg/L with sodium metabisulphite, lime, copper sulphate and oxygen
- Final Tailings: the slurry is pumped to the TMF or deposited underground

17.2 Plant Design Criteria

The process design criteria and mass balance have been developed based on the existing plant data and testwork data. Key process design criteria are summarized in Table 17-1.

Table 17-1: Key process design criteria

Description	**Units**	**Nominal Value**	**Source**
Daily Throughput	t/d	800	Client - SRK Mine Plan
Crushing Plant Availability	%	50	Client – Permitting Requirement
Crushing Plant Throughput	t / oph	67	Engineering Calculation
Process Plant Availability	%	95	Client Plant Operating
Process Plant Throughput	t / oph	35	Engineering Calculation
LOM Average Grade	g/t	8.16	Client - SRK Mine Plan
LOM Average Gold Extraction	%	95.7	Client Plant Operating Data and Test Work (BL288), SRK Mine Plan
Comminution Data			
Bond Crushing Work Index	kWh/t	13.2	BaseMet (2018), BL0288
Axb	-	23 to 42	BaseMet (2018, 2023), BL0288
Bond Ball Mill Work Index	kWh/t	9.7 to 17.1	BaseMet (2018, 2023), BL0288
Bond Abrasion Index	g	0.24	BaseMet (2018), BL0288
Process Design			
Crusher Feed, F_{100}	mm	500	Client Operational Data
Crusher Product, P_{80}	mm	75	Client Operational Data, Vendor Simulation
Grinding Product, P_{80}	µm	75	Client Operational Data and BL288
Gravity Concentration	-	2 – XD20	Vendor Simulation
Pre-Oxidation	h	1	Client Operational Data and BL288, SRK Mine Plan
Leach	h	24	Client Operational Data and BL288, SRK Mine Plan
	stages	5	Engineering Calculation
CIP	h - stages	5 – 6 tanks	Vendor Model, Operational Data

Description	Units	Nominal Value	Source
Carbon Plant	g Au/t carbon	3,310	Vendor Model, Operational Data
	t strip/day	1	Engineering Calculation
Cyanide Destruction	h	0.75 to 1	BaseMet (2018, 2023), BL0288
$CN_{[WAD]}$ Discharge Target	mg/l $CN_{[WAD]}$	<5	Client

Source: SRK (2025)

17.3 Overall Process Flowsheet and Layout

The overall process flow diagram is shown in Figure 17-1, while the plant layout is provided in Figure 17-2.

Figure 17-1: Madsen Plant process flow diagram

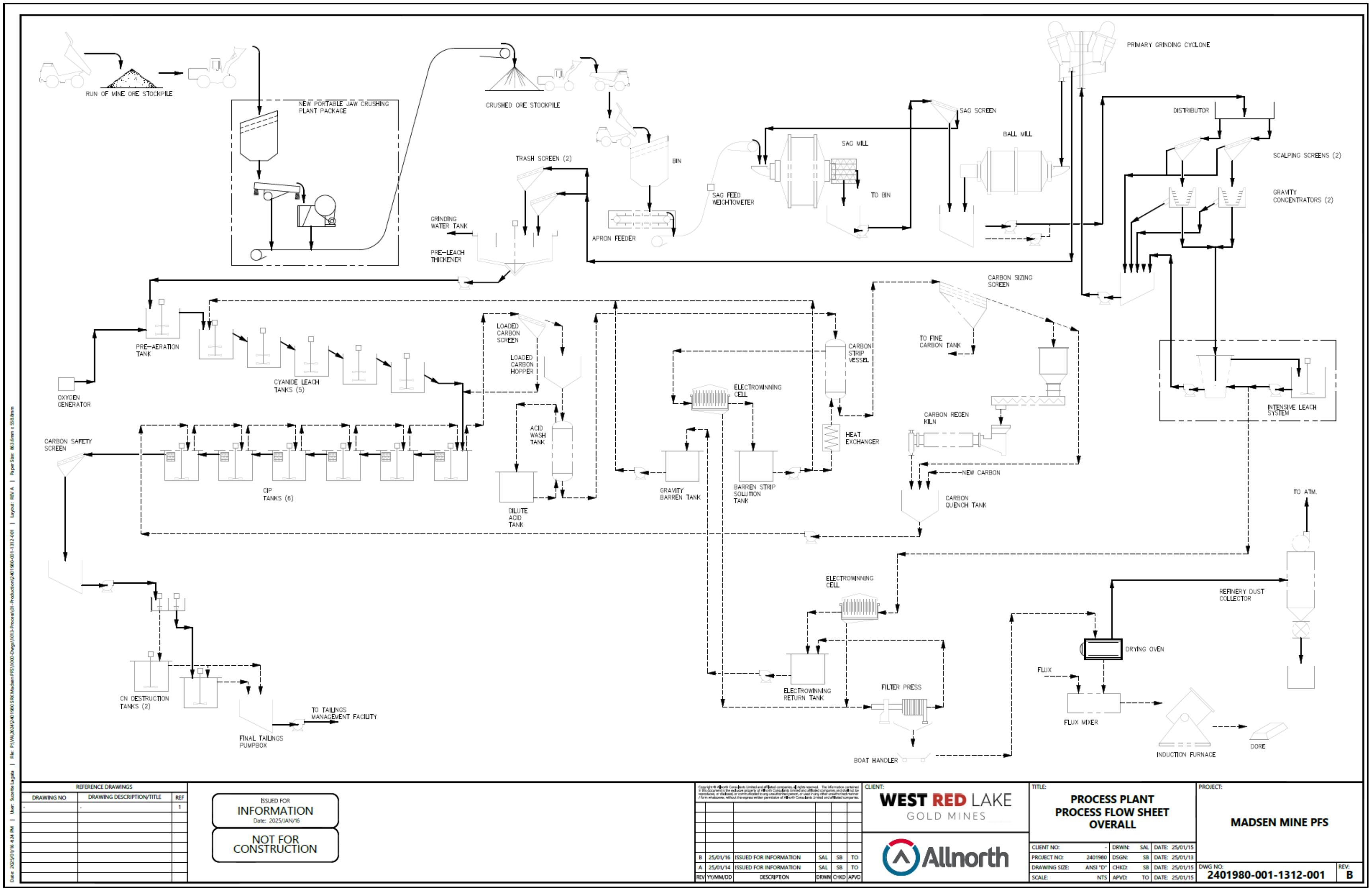
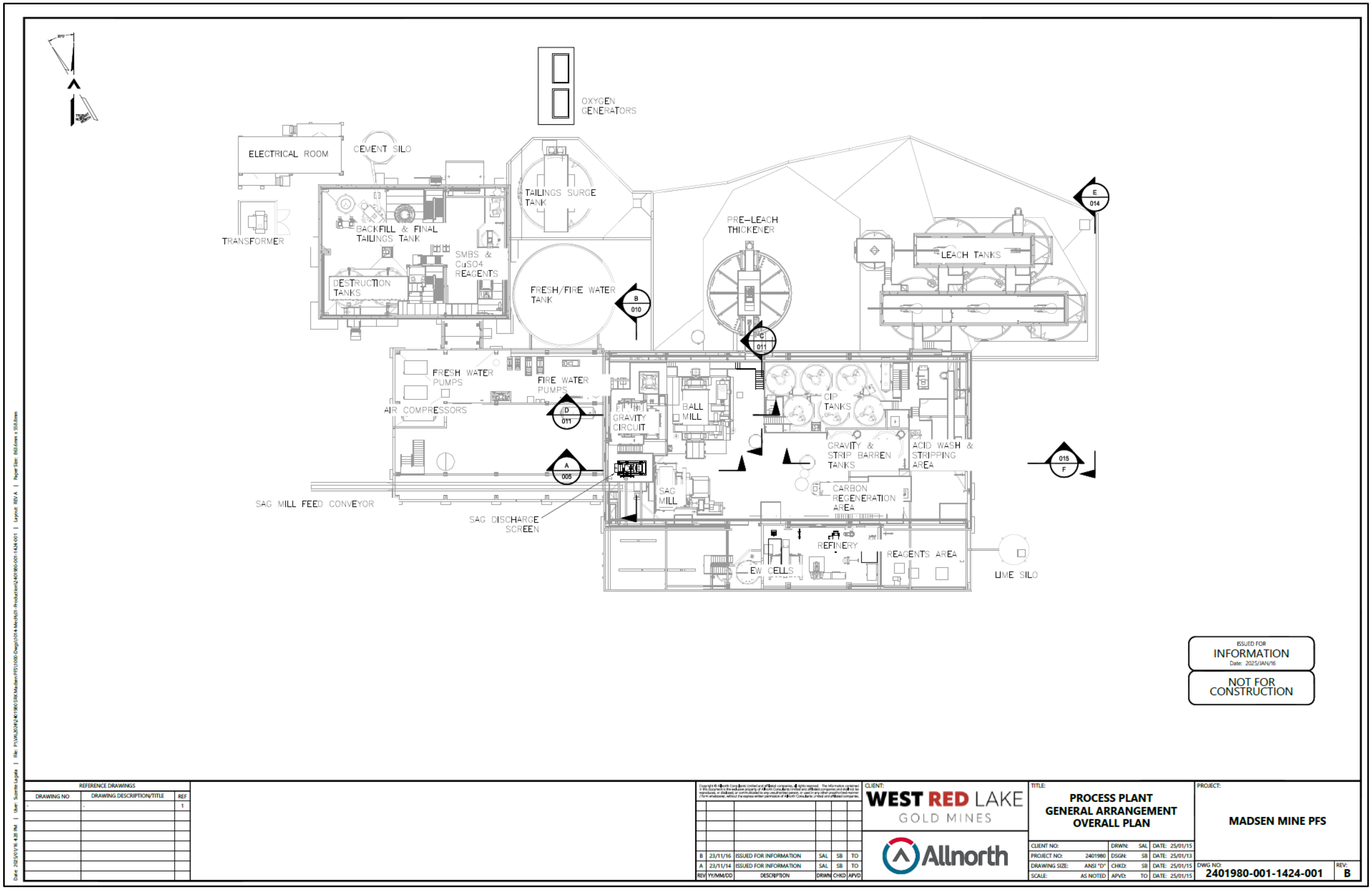


Figure 17-2: Madsen Plant layout



17.4 Process Plant Description

17.4.1 Primary Crushing

Run of Mine (ROM) feed, F100 of 500 mm, will be crushed using a new semi-mobile crushing station to produce mill feed with an average P80 of 75 mm. The crusher product will be stockpiled and loaded into trucks for transport to the head frame, where a front-end loader will transfer the crushed material to the SAG feed conveyor feed bin. The crushed ore will be withdrawn by a feeder onto the SAG mill feed conveyor.

The semi-mobile crushing station includes a dump pocket, vibrating grizzly feeder, jaw crusher and discharge conveyor. Material will be loaded into the dump pocket. A vibrating grizzly feeder will draw material from the dump pocket at a rate of 67 t/h and oversize will feed directly into a jaw crusher with an installed power of 112 kW. The undersized material will bypass the crusher and feed directly onto the discharge conveyor. The primary crushing stage will produce a product P80 of approximately 75 mm at a crusher close side setting (CSS) of 76 mm.

The crushed product, P80 of 75 mm, will be stockpiled using a stacking conveyor, and a front-end loader will feed trucks for transport to the crushed material storage bin. The stacking conveyor (grasshopper) will have the flexibility to stack crushed material into different piles if required.

17.4.2 Grinding

The existing grinding circuit includes a primary SAG mill followed by a secondary ball mill. A gravity concentration circuit is installed in the grinding circuit to recover any gravity recoverable gold. The primary SAG operates in closed circuit with a screen, while the secondary ball mill will operate in reverse closed circuit with a cluster of hydro-cyclones. The SAG mill and ball mill discharge will be processed through the gravity circuit with the gravity tailings feeding the cyclone cluster.

Primary Grinding

The grinding circuit will be fed by reclaimed material from the crushed material storage bin. The crushed material will feed the existing 4.9 m diameter x 1.5 m long 560 kW SAG mill via the SAG mill feed conveyor. A new belt-scale on the feed conveyor will monitor feed rate. Reclaim water will be added to the SAG mill to maintain the slurry charge in the mill at a constant density of 70% solids. Slurry will flow through the discharge trommel with openings into the SAG mill pump box. The ground slurry will then be pumped to the existing sampler and flow by gravity to the gravity feed pump box.

Gravity Concentration

SAG discharge will flow into the gravity feed pump box from the sampler and combine with the ball mill discharge before being pumped to the two gravity concentrator scalping screens. An existing pump will feed the two existing scalping screens. With an aperture size of 2 mm, the feed screen will remove any oversize particles prior to gravity concentration. The screen oversize will flow to the cyclone feed pump box. The screen undersize will feed the semi-continuous gravity concentrators. Using high gravitational forces, high density gravity recoverable gold will collect in the concentrate cone, while lower density material will flow out of the tailings discharge port and combine with the gravity feed screen oversize in the ball mill cyclone feed pump box.

The gravity concentrators will operate in 40- to 60-minute cycles. During a cycle, gravity recoverable gold collects in the concentrate cone. At the end of the cycle, the gravity concentrator feed is diverted to the gravity tailings stream, and the concentrate cone will be flushed with water, sending the concentrate to an intensive leach reactor for further concentration and leaching.

Secondary Grinding

Tailings from the gravity circuit will flow into the ball mill cyclone feed pump box and be pumped up to a cluster of four (three operating/one standby) 250 mm hydro-cyclones for size classification. The coarse underflow will flow by gravity to the ball mill for additional grinding, while the fine overflow, at a final product P80 of 75 µm, will report to the pre-leach thickener via one trash screen. The hydro-cyclones have been designed for a 300% circulating load, and minor modifications will be required at the higher throughput.

Cyclone underflow will feed the existing 2.9 m diameter x 4.9 m long overflow ball mill with an installed power of 560 kW. Ground slurry will overflow from the ball mill onto a trommel screen attached to the discharge end of the mill. The trommel screen oversize, consisting mainly of scats, will discharge into a trash bin for removal from the system, while the undersize will flow into the gravity feed pump box.

17.4.3 Pre-Leach Thickening

Cyclone overflow will flow onto the vibrating trash screen for removal of trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to the 10 m diameter pre-leach thickener. Flocculant solution will be added to the thickener feed to promote the settling of fine solids. The high-rate thickener will thicken the slurry to 50% solids. The thickener underflow will be pumped to the pre-oxidation circuit, while the thickener overflow will flow by gravity into the process water tank to be used as make-up water in the grinding circuit.

17.4.4 Leaching

Pre-leach thickener underflow will be pumped to the pre-oxidation tank prior to leaching. Oxygen will be sparged into the bottom of the agitated tank and slurry will be conditioned for approximately 2 hours to oxidize sulphide minerals. Piping modifications are being completed to dose lime in the pre-oxidation stage to adjust pH prior to adding cyanide in the leach circuit.

Based on metallurgical testing and operational data, pre-oxidation will help reduce the consumption of dissolved oxygen during cyanidation, improving metallurgical recovery. It will also reduce sodium cyanide (NaCN) consumption by preventing the formation of thiocyanate and complexing some of the heavy metals, such as iron. This step will also reduce reagent consumption downstream in the cyanide destruction circuit.

After pre-oxidation, the slurry flows to the first of six 7.3 m diameter x 8.2 m high agitation leach tanks. The site noted previously that the circuit ran at a low sodium cyanide concentration, between 150 to

170 ppm, and achieved the target tailings gold grade. Lime slurry is added to the leach circuit to maintain protective alkalinity at a design pH of 11.0, preventing the creation of hydrogen cyanide gas (HCN). Previously, oxygen was not added into the leach tanks. An additional oxygen generator was added before the plant was shut down, and there is now sufficient oxygen supply to sparge oxygen into the leach tanks. Piping modifications will be completed in spring 2025 to distribute oxygen to the leach tanks. Slurry from the leach circuit will then flow to the CIP circuit for carbon adsorption.

17.4.5 Carbon-in-Pulp (CIP)

Leached slurry will flow into the first of six 3.6 m diameter x 4.8 m high CIP tanks. Each tank has been installed with an agitator and inter-stage screen for retaining activated carbon. As the slurry flows through the CIP tanks, gold-cyanide complexes will be adsorbed onto the pores of activated carbon. The plant was operating the CIP circuit at a carbon concentration of approximately 25 g/L, with the concentration higher at 50 g/L in the first and last tanks to maximize adsorption. A circuit audit is recommended once the plant is in operation to improve carbon adsorption and lower carbon concentration.

As the slurry proceeds through the circuit, metal values in the solution will progressively decrease. The carbon will be transferred countercurrent to the slurry flow to maximize gold recovery. Regenerated carbon, with the highest adsorption potential, will be introduced to the last CIP tank, interacting with the lowest concentration of gold. Loaded carbon, with the lowest adsorption potential, will be in the first CIP tank, interacting with the highest concentration of gold. The plant has the capacity to process one tonne of carbon per day and, if needed, twice per day. Loaded carbon from the first CIP tank will be pumped to the loaded carbon screen where the slurry will be separated, and the carbon transferred to the acid wash circuit. The separated slurry will then flow by gravity back into the first CIP tank. Fresh activated carbon from the carbon regeneration circuit will be pumped into the last tank in the CIP train.

The slurry from the sixth CIP tank will be pumped using the existing pump box. The tailings stream from the last CIP tank will flow onto a stationary safety screen to capture any carbon particles that may have escaped the CIP circuit. Captured carbon particles will be collected in bins, and tailings will be pumped to the cyanide destruction circuit.

17.4.6 Carbon Processing

The carbon processing plant has been designed to process one tonne per day of loaded carbon, producing gold doré. One batch of carbon is processed through the acid wash, elution and regeneration circuits on a daily basis.

Acid Wash

Loaded carbon from the CIP circuit flows to a 1-tonne capacity acid wash vessel constructed of fibre-reinforced plastic (FRP). The carbon is treated with a circulating 3% hydrochloric acid (HCl) solution to remove calcium deposits, magnesium, sodium salts, silica, and fine iron particles. Organic foulants, such as oils and fats, are unaffected by the acid and will be removed after the elution step in the carbon regeneration circuit using a horizontal electric kiln.

During an acid wash cycle, the carbon is first rinsed with fresh water. HCl solution is then pumped from the dilute acid tank upward through the acid wash vessel, overflowing back into the dilute acid tank. The carbon is then rinsed with fresh water to remove the remaining acid and any mineral impurities.

The acid washed carbon is moved from the acid wash vessel into the elution vessel using transport water. Carbon slurry will discharge directly into the top of the elution vessel. Typically, only one acid wash and elution cycle will take place per day.

Carbon Stripping (Elution)

The carbon stripping (elution) process will utilize barren strip solution to strip the loaded carbon, creating a pregnant gold solution that will be pumped through the electrowinning cells for gold recovery. The solution exiting the electrowinning cells will be circulated back to the barren solution tank and circulated through the strip vessel or once the process is completed, stored in the barren tank for reuse.

The strip vessel is a carbon steel pressurized tank with a capacity to hold approximately one tonne of carbon. During the strip cycle, a solution containing approximately 1% sodium hydroxide and 0.1% NaCN, at a temperature of 140°C (284°F), will be pumped up through the strip vessel at a pressure of 450 kPa (65 psi). Solution exiting the top of the vessel will be cooled by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold barren solution prior to passing through the solution heater. An electric boiler will be used as the primary heating source.

To ensure the temperature of the solution leaving the strip circuit is below 80°C a trim heat exchanger will be added prior to the electrowinning stage. Currently the strip solution takes significant time to pre-heat, as the original design had immersion heaters in the pregnant and barren strip solution tanks that were never installed. The addition of immersion heaters to the barren strip solution tank has been included in the expansion budget to shorten pre-heat time and allow for more efficient stripping.

Carbon Regeneration

The carbon regeneration circuit will thermally regenerate the stripped carbon, re-activating the pores and removing any organic foulants, such as oils and fats. Fresh activated carbon will be added to account for any carbon lost during the adsorption and desorption processes.

A recessed impeller pump will transfer the stripped carbon from the strip vessel to the carbon sizing dewatering screen. The 1.5 m diameter vibratory screen doubles as a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity into the regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. Periodically, the carbon fines will be filtered and collected into bags for disposal.

A horizontal electric kiln with residual heat dryer will be utilized to treat one tonne of carbon per day, equivalent to 100% regeneration of stripped carbon. The regenerated carbon from the kiln will flow by gravity into the carbon quench tank, cooled by fresh water and/or carbon fines water, and pumped back to the CIP circuit.

To compensate for carbon losses from attrition and impact, fresh carbon will be added to the carbon attrition tank and mixed with fresh water to activate the carbon pores. The fresh carbon will combine with the regenerated carbon discharging from the kiln.

17.4.7 Electrowinning and Refining

Pregnant solution from the strip circuit and intensive leach reactor will be pumped to the existing refinery for electrowinning in separate circuits, producing a gold sludge. The sludge from each circuit will then be filtered, dried, and refined separately in an electric induction furnace, producing gold doré bars.

Pregnant strip solution will be pumped through electrowinning cells. Gold will plate on stainless steel cathodes, while the barren solution will flow back to the barren solution tank. To prevent a build-up of impurities, a daily bleed of barren solution will be pumped to the leach circuit.

A separate electrowinning circuit processes the pregnant solution from the intensive leach circuit.

Gold rich sludge will periodically be washed off the cathodes into the electrowinning sludge tank using high pressure water. Once the tank is filled, the sludge will be drained, filtered, dried, mixed with fluxes, and smelted in the induction furnace, producing gold doré. This process will take place within the secure and supervised area, and the precious metal product will be stored in a vault for shipping off site.

17.4.8 Cyanide Destruction

The cyanide destruction circuit will consist of two, 4 m diameter x 5 m high mechanically agitated tanks, each with a capacity to handle the full slurry flow for the required residence time of one hour. Cyanide will be destroyed using the SO_2/oxygen process. Treated slurry from the circuit will then be pumped to the final TMF.

The cyanide destruction circuit will treat CIP tailings slurry, process spills from various contained areas, and process cyanide containing bleed streams.

Oxygen will be sparged from near the bottom of the tanks. Lime slurry will be added, if necessary, to maintain the optimum pH of 8.0 to 8.5 and copper sulphate ($CuSO_4$) will be added as a catalyst, maintaining on average 30 mg/L concentration in solution. A sodium metabisulphite (SMBS) solution, at 5:1 SO_2:CN_{WAD}, will be dosed into the system as the source of SO_2. This system has been designed to reduce the CN_{WAD} concentration to below 5.0 mg/L.

Treated slurry from the cyanide destruction circuit will be pumped to the final TMF.

17.4.9 Reagent Handling and Storage

Reagents consumed within the plant are prepared on-site and distributed via the reagent handling systems. These reagents include sodium cyanide (NaCN), lime, lead nitrate (Pb_2NO_3), hydrochloric acid (HCl), caustic soda (NaOH), copper sulphate ($CuSO_4$), sodium metabisulphite (SMBS), Antiscalant, flocculant and activated carbon. All reagent areas are bermed with sump pumps that will transfer spills to various appropriate tanks within the plant, except for flocculant, which will be returned to the flocculant storage tank. The reagents will be mixed, stored, and delivered to end users within the plant. Dosages are controlled by flow meters and manual control valves. The reagents will be delivered in dry form, except for HCl and Antiscalant, which are delivered as solutions. Table 17-2 summarizes the reagents used in the process plant and their estimated annual consumption rates.

Table 17-2: Consumables

Description	Delivered Form	Estimated Annual Usage (t/yr)
NaCN	1 tonne bags (dry)	152
Lime	2 tonne bags (dry)	225
Pb_2NO_3	50 kg bags (dry)	37
HCl	208 L drums (36% liquid)	95
NaOH	50 kg bags (dry)	47
$CuSO_4$	50 kg bags (dry)	40
SMBS	500 kg bags (dry)	422
Antiscalant	50 kg barrels	9
Flocculant	25 kg bags (dry)	12
Activated Carbon (based on carbon stripped)	50 kg bags (dry)	18

Source: BaseMet (2018)

17.4.10 Air Supply

The compressor room has instrument and plant air supplied by two compressors and associated dryers and filters. Air receivers are located throughout the plant building. Oxygen generators supply oxygen to the pre-oxidation, leach, and cyanide destruction circuits.

17.4.11 Water Supply

Plant Water – supplied from the TMF via the fire/plant water tank. Overflow water from the pre-leach thickener will be used predominantly in the primary and secondary grinding circuits to dilute slurry to the required densities.

Fresh Water – for the process plant is pumped from the polishing pond. Fresh water is used as reagent make-up water, gland water, process make-up water as required, and cooling water services.

18 Project Infrastructure

18.1 Summary

The Madsen deposit was discovered in 1937 and the Madsen Mine commenced production a year later with sinking of the Madsen #2 shaft, which ultimately reached a depth of 1,273 m with production from 27 levels.

The Madsen project is a mature mine site with an existing underground mine and mineral processing facilities.

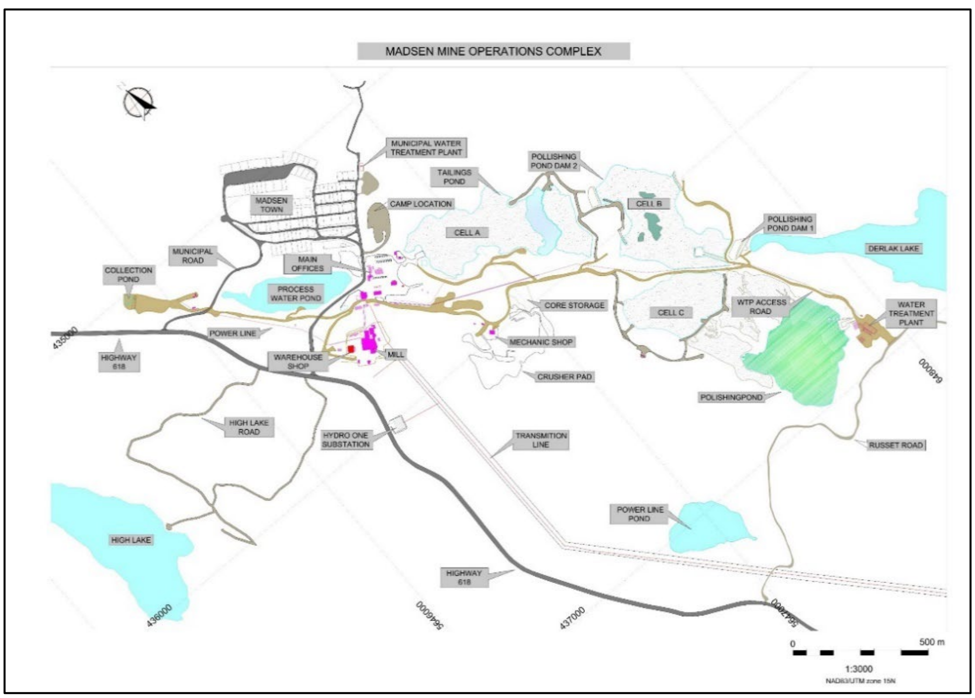
The Madsen Mine is an underground gold mine with a 1,089 t/d processing plant (Mill), a shaft, two portals (East and West), a water treatment plant, a tailings area, a rock dump and a general services area.

The mine plan envisions a modern, fully mechanized mining operations. Two ramps exist to provide access to the underground. The Madsen production shaft is being retrofitted and prepared for hoisting of ore to surface by 2027.

As part of the mine restart plan, WRLG will be adding surface facilities such as a new mine dry, a mobile workshop, and a mobile crushing unit that are required to permit the operation to restart.

The Madsen Mine has been kept dewatered since its acquisition by WRLG in 2023. At the time of writing, dewatering is currently between 14 Level and 15 Level. The site plan is shown in Figure 18-1.

Figure 18-1: Madsen property plan



Source: Allnorth (2025)

18.2 Access Roads and Logistics

The Madsen Mine is located in the Red Lake district of northwest Ontario. The town of Madsen is located 8 km to the southwest of the town of Red Lake.

The road to the Madsen Mine is part of the Kings Highway system, a provincial highway. It is an all-weather paved road named Highway 618. This highway starts at Red Lake and terminates two kilometres to the west of Madsen at Starratt-Olsen.

18.3 Power

The Madsen Mine is connected to the northwest Ontario power network by aerial distribution power lines. The incoming voltage to the site is from a 44 kV circuit with a 12 MW power supply.

The northwest Ontario power transmission network is owned and operated by Hydro One. Red Lake is located at the end of the 115 kV transmission line coming from Ear Falls, Ontario.

The Madsen project has some older equipment dating back to the early fifties and seventies. This 2300 V equipment is located in the compressor room and the headframe. The hoisting system and the East Portal operate on 2300 V switchgear that need to be modernized.

The voltages utilized at the Madsen Mine are summarized in Table 18-1.

Table 18-1: Madsen property voltages

Voltage	Usage	Notes
44 kV	Main incoming Hydro One voltage	
13.8 kV	N/A	Not presently available on site
4160 V	Compressors and large motor	Site medium voltage distribution for surface and underground
2300 V	Compressor (old installation) / East Portal	Existing on site since 1957
600 V	Electric Motors	Standard voltages for electrohydraulic equipment
440 V DC	Hoist	This installation will require modernization to put the hoist back into operation

Source: Allnorth (2025)

A power consumption trade-off was conducted (Table 18-2) using various mill tonnages. Diurnal variations in demand, resulting from the underground operations were also evaluated based on historical data provided by Hydro One. Under low plant tonnages, the plant was assumed to operate once the stockpiles were sufficiently large. The main consumer for power is mine ventilation. There is presently no-load shedding or load sharing agreement with Hydro One or other mines.

Table 18-2: Estimated power consumption based on plant throughput

Throughput Rate	Estimated avg consumed power (kWh/t)	Estimated max avg power consumption* at 95% CL (kWh/t)	Diurnal Mine Variation (kWh)	Design Basis (+ 10% for new equipment) (kWh)
400 t/d	2,670	3,344	268	3,973
800 t/d	5,339	6,688	535	7,945
1100 t/d	7,341	9,196	735	10,941

* Without a power management plan
Source: Allnorth (2025)

The estimated average power demand for the mine site was based on a 1,100 t/d site throughput (a slight round up from the plant nameplate capacity of 1,089 t/d) will be 10,941 kW. The present contracted demand from Hydro One for the site is 12,000 MW. The 44 kV to 4160 V switchgear at the process plant is less than 10 year old and can be upgraded to perform some load management of the individual feeders. As the loads increase, especially with the addition of fans, the mine would benefit from load shedding of the fans and pumping cycles. The major energy consumers in the underground mine are the ventilation fans. Pumping is the second largest consumer and drilling equipment is third.

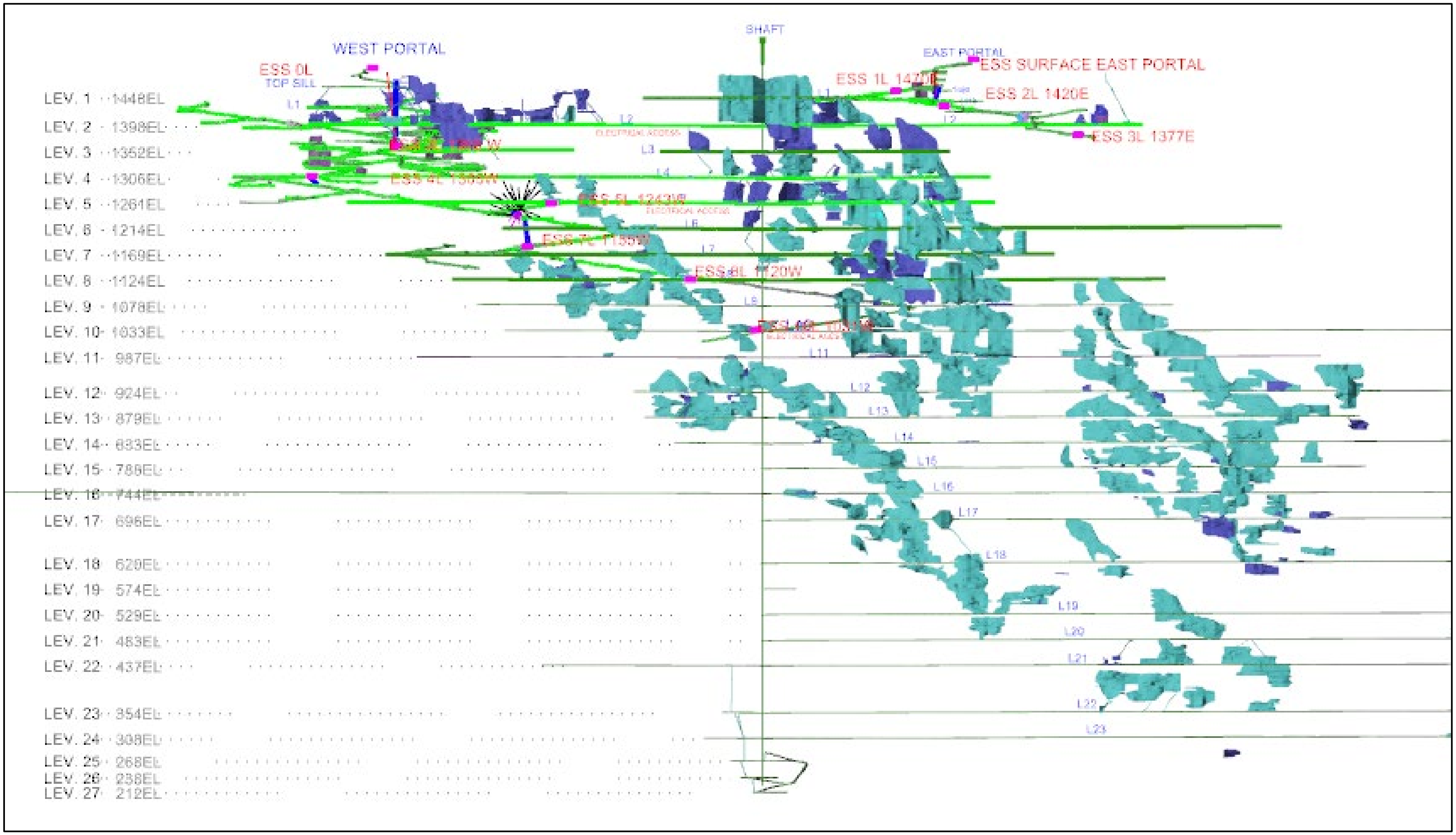
The underground power distribution is presented in Figure 18-2. The mine is fed from the switchgear located at the mill. Underground power is supplied with three phase cables. The standard transformer configuration is modular skid mounted 1000 kVA units with a bank of eight breakers. The low voltage is 600 V distributed to the final consumers. The average distance between the transformer stations is 400 m. The transformers are relocated into the mining areas as the mining progresses deeper into the mine. When a transformer is relocated, the remaining electrical consumers previously fed from the transformer are tied into another transformer with a low voltage cable. The feeders to the underground are described in Table 18-3.

Table 18-3: Underground feeders

Feeder	Utility
West Portal Powerline	The West portion of the mine is fed with a 4160 V aerial line and then with cables to the Electrical Substations (ESS) located in the mine. As mining progresses and deepens, it is expected that most of the electrical consumers will move from the west to the Shaft Feeders.
Shaft #2 Feeders	There are presently two main feeders in the shaft that will in the future feed most of the necessities of the mine. The mining is progressing from the West Portal Area to the Shaft Area. A third feeder goes down to the 6th level only.
110 Raise Feeders	This feeder feeds the east part of the mine as there is no direct connection between the east and the west part of the mine
East Portal Feeder	This feeds some shops at the east portal and some pumps at the portal. It is presently fed at 2300 V but plans are underway to convert all the 2300 V to 4160 V.
Future Redundant Feeder	A redundant feeder to the 12 Level substation is required. This redundant path would start at the 110 raise and reach 12 Level at Shaft #2.
Shaft #3 Feeders	The construction of Shaft #3 will include the installation of two main feeders to permit the development of the mine at depth. One feeder would reach 9 Level and the second feeder would reach 20 Level on Shaft #3.

Source: Allnorth (2025)

Figure 18-2: Electrical substation distribution (looking north)

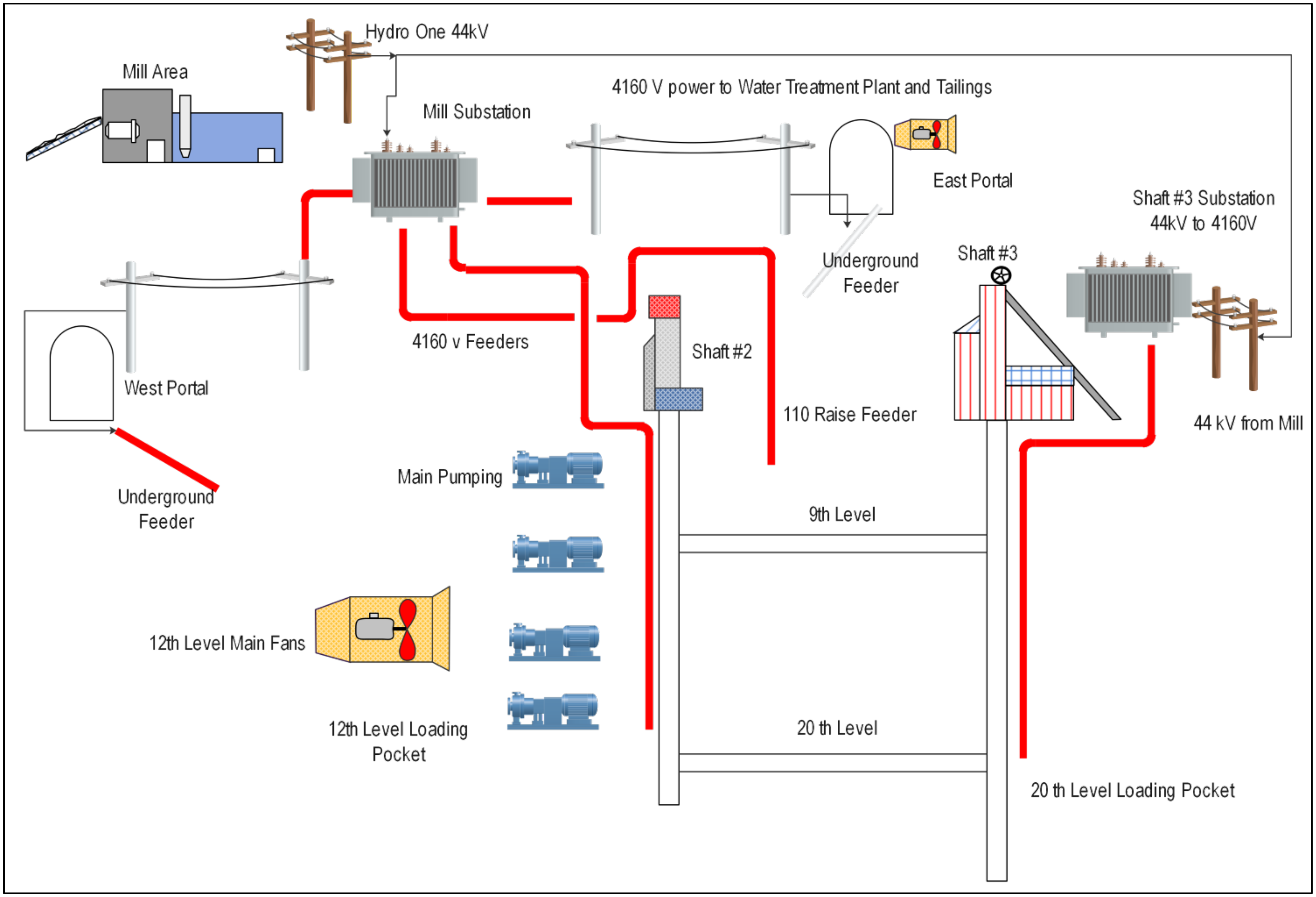


Source: Allnorth (2025)

The proposed future power distribution concept is found in Figure 18-3. The main features of the future power distribution are the following:

- Shaft #2, known as the Madsen Shaft, will remain the main power feeder to the underground in the initial years of operation. As the mine progresses at depth, the mining activity will gradually move from the West Portal towards Shaft #2. When the mining has reached the Shaft #3 area, the underground mine transformers will be fed from the feeders installed in Shaft #3. Once Shaft #3 is operational, the production hoisting from Shaft #2 will be reduced or stopped. However, the Shaft #2 hoist will be required for the maintenance of the main pumps located at the shaft. Water from Shaft #3 will be pumped across to Shaft #2.
- The Shaft #2 feeders are in first years of operation mainly used for the mine ventilation, and the mine's primary pumping. Shaft #2 and its pumping stations will remain active throughout the life of the mine.
- The West Portal feeds most of the underground in the initial years of operation. Most of the transformers are fed from this combination aerial line and underground feeder.
- Once the connection between the East Portal and the West Portal has been completed, a redundant loop can be created to ensure redundancy in the power distribution system. Presently, part of the East Portal is fed using an antiquated 2300 V transformer which is planned to be phased out.
- A power feeder exists in the 110 Raise area. This feeder provides power to the east sector of the mine. This feeder is a 4160 V feeder and should be extended as mining progresses to create a second redundant power loop. The redundant loop is generally required to avoid shutting down sections of the mine when major changes or upgrades are required or where a cable has been damaged.
- A 44 kV power line is proposed to feed Shaft #3 once it has been constructed. A 44 kV to 4160 kV transformer will be installed close to the headframe to permit feeding the hoisting equipment, compressor and surface equipment. Additionally, two shaft feeders would be installed to feed the 9 Level and 20 Level for mining operations. Figure 18-3 illustrates two possible routings for the power line from the existing Mill 44 kV circuit to the Shaft # 3 substation.

Figure 18-3: Simplified future power distribution



Source: Allnorth (2025)

18.4 Underground Communications and Control

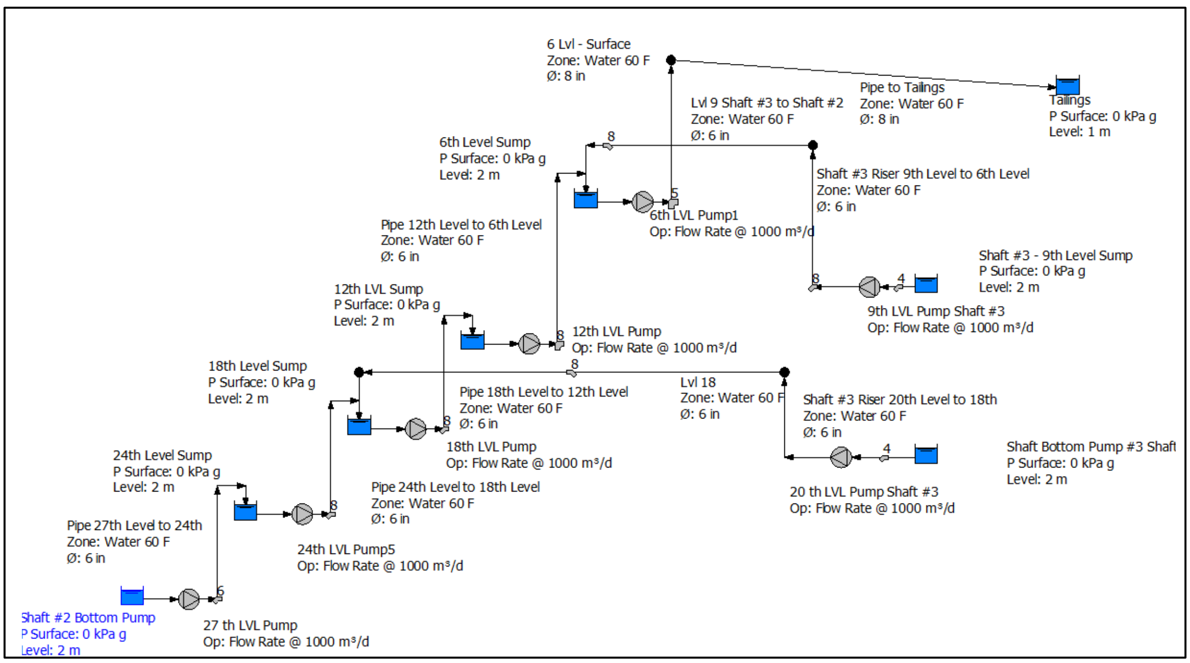
The Madsen Mine presently has an underground leaky feeder system for communications, which will be expanded as required into the new working areas. Repeaters and cable are required. The future investment will be required to extend the existing system.

There is presently no automation planned for the mining operations at the Madsen Mine. No allowance has been made for a fiber optic network for the underground operations.

18.5 Main Pumping System

The main pumping system schematic may be found in Figure 18-4. The Madsen Mine is being dewatered through the Shaft #2 pumping stations. The pump stations are equipped with Mackley 7-stage pumps equipped with 250 kW/4160 V motors. The principal sumps are located at Shaft #2 and spaced out at six-level intervals. The average level interval is 45 m.

Figure 18-4: Main pumping stations Shaft #2 / #3



Source: Allnorth (2025)

The pumping system is a multistage pumping system due to the vertical height of the mine. A total of six pumping stages are required. The horizontal pumps receive water from sumps located in the vicinity of the shaft. The piping in the shafts feed the water to the next shaft sump stage above in the sequence. The mine waters are pumped to the reclaim pond where they are recycled or treated for discharge along with the surface water and plant process water.

The dewatering rate for the mine is 1,000 m^3/d. The 1,000 m^3/d includes the recharge rate and the dewatering volumes of the existing underground workings, which are limited by the capacity of the water treatment plant. The site water balance is positive and the water treatment plant limits the permitted pumping rate. Rates up to 1,300 m^3/d are possible during the summer seasons.

Shaft #2 as of March 2024 was currently dewatered just below 12 Level and the LOM plan requires that the entire mine be dewatered to 27 Level. It is planned that the dewatering pumping strategy will remain in place throughout the life of the mine. The increase in capacity of the tailings water treatment plant will permit a higher discharge rate from the mine.

As part of the dewatering system, water from 9 Level of the new Shaft #3 will be pumped across to the Shaft #2 pumping system. Another pumping station is also planned for 20 Level.

As most of the development will have been completed in the Shaft #2 area, the water from Shaft #3 on levels 9 and 20 will have a natural gradient towards the existing Shaft #2.

The shaft dewatering piping will be steel and the horizontal low-pressure runs will be HDPE. The selected pumps will match the original Mackley multi-stage horizontal pumps made of stainless steel.

Hydraulic backfill is planned to be used in mine and this will add an additional pumping load. The existing pumping system can handle the water but not the sediment load. It will be necessary to ensure that all water reaching the shaft has been decanted. The shaft pumps are clear water pumps and cannot handle the slimes directly from the return water from the hydraulic filling. The shaft sumps are small and cannot accept sediment. Due to their small size and difficult access, the removal of slimes from the sumps is difficult. Additional wear on the main pumps is to be expected when the sediment load increases.

Secondary sumps will be located closer to the working areas and the sediments settled prior to arriving at the shaft. This intermediate sump is necessary to minimize the slimes arriving at Shaft #2.

18.6 Backfill

18.6.1 Backfill Method and Process Description

Several backfill methods were considered, including unconsolidated rock fill (URF), cemented rockfill (CRF), cemented paste backfill (CPB) and cemented hydraulic fill (CHF). URF is the primary backfill as the development waste rock can be placed directly into the stopes, which minimizes truck haulage and hoisting of waste rock to the surface. CRF may be employed to solve specific problems, but none is planned currently.

CPB was originally considered as a means of backfilling certain historic voids and to backfill some of the longhole stope where it does not make sense to use URF. CPB can be produced using reclaimed tailings, mixed with cement and water with a prefabricated modular plant. There are numerous examples of this method being successfully employed. WRLG selected CHF for this purpose as there is an existing CHF plant located adjacent to the mill that was built by the previous owner but not commissioned. This CHF plant was designed specifically for the Madsen Mine and requires minimal capital to bring it into operation.

To produce CHF, the mill tailings will be directed to a surge tank at approximately 44% solids by weight (wt%), which will feed the backfill plant. From the surge tank, the tailings slurry will be diluted to 30 wt% in a cyclone dilution feed tank. This dilution tank, with the addition of process water, provides the flexibility to adjust the percent solids of the slurry fed to the cyclone cluster. The cyclone will separate the slurry into two streams: the overflow (OF) and the underflow (UF). The UF will flow by gravity into the CHF mixing tank. On the other hand, the OF will report to a pump box, which will send the fines as a slurry to the tailings management facility (TMF).

The cement silo will dispense cement into the CHF mixer based on the backfill mix design selected by the operator. Using a weigh feeder, the cement is fed based on the solids flow from the cyclone UF. In the CHF mixing tank, the UF is mixed with cement binder, and the slurry percent solids can be further adjusted to the backfill target mix design by adding a portion of the OF.

From the backfill plant, the CHF will be pumped through a heat-traced, aboveground pipeline to a borehole. From the borehole, the pipeline will connect to the underground distribution system, which will consist of a network of boreholes and level piping underground. Through the underground distribution system (UDS), the CHF will flow to the stopes underground.

When backfill is not required, the design will allow the tailings to bypass the backfill system and be diverted to the TMF.

18.6.2 Backfill Materials and Mix Design

The mill tailings are expected to contain fine particles that may hinder quick drainage when placed underground as cemented hydraulic fill. Therefore, the backfill plant is equipped with a hydrocyclone cluster to deslime the tailings. The hydrocyclone separates the proportion of fines in the tailings, thereby improving the permeability of the fill.

Computer simulations were used to select the appropriate cyclone, followed by a pilot-scale cyclone desliming test to confirm the selection. During commissioning, the cyclone cluster will be calibrated to optimize performance.

Using UCS data and the various strength requirements, the cement percentages required in the mix designs are calculated. The next section discusses the mix designs.

Table 18-4 presents the target strengths used in the design of the backfill mix designs. Notably, Mix C will be used in areas in front of the barricades within large voids from legacy mine workings. The higher cement content will allow for a faster settling time, allowing the formation of a plug to facilitate the introduction of additional fill. If the backfill set time is not a limiting factor, there may be an opportunity to reduce the cement %.

Generally, about 20% by volume of the hydraulic fill will drain as decant water. This will reduce the fill volume after it is consolidated. For example, every 100 m^3 of hydraulic fill introduced will produce 80 m^3 of fill mass. Depending on the geometry and the material properties, further consolidation of around 10% reduction by volume may occur in the stope.

Table 18-4: Target strengths for mix designs A, B and C

Mix Design	Target strength (kPa)	Cement (% by weight)	Application
Mix A	460	7	Long hole stope
Mix B	150	5	Drift and fill
Mix C	600	10	Plug

Source: T Engineering (2025)

18.6.3 Underground Distribution System

The UDS will consist of a network of boreholes, level piping and stope piping. The surface boreholes will be drilled and cased with 4-inch schedule 80 carbon steel piping. The boreholes will connect the surface to underground and link different levels at the backfill stations. From the backfill stations, the level piping will extend to each mining area using carbon steel piping. The final 100 m of piping before the stopes may be switched to HDPE DR11 piping, depending on the geometry.

Pressure sensors will be located at strategic locations to monitor the flow conditions throughout the system. Furthermore, blast tees and dump valves will be incorporated into the UDS to discharge the backfill materials in the event of a pipe blockage. To safeguard the UDS from over-pressurization, rupture discs will be installed in the backfill stations.

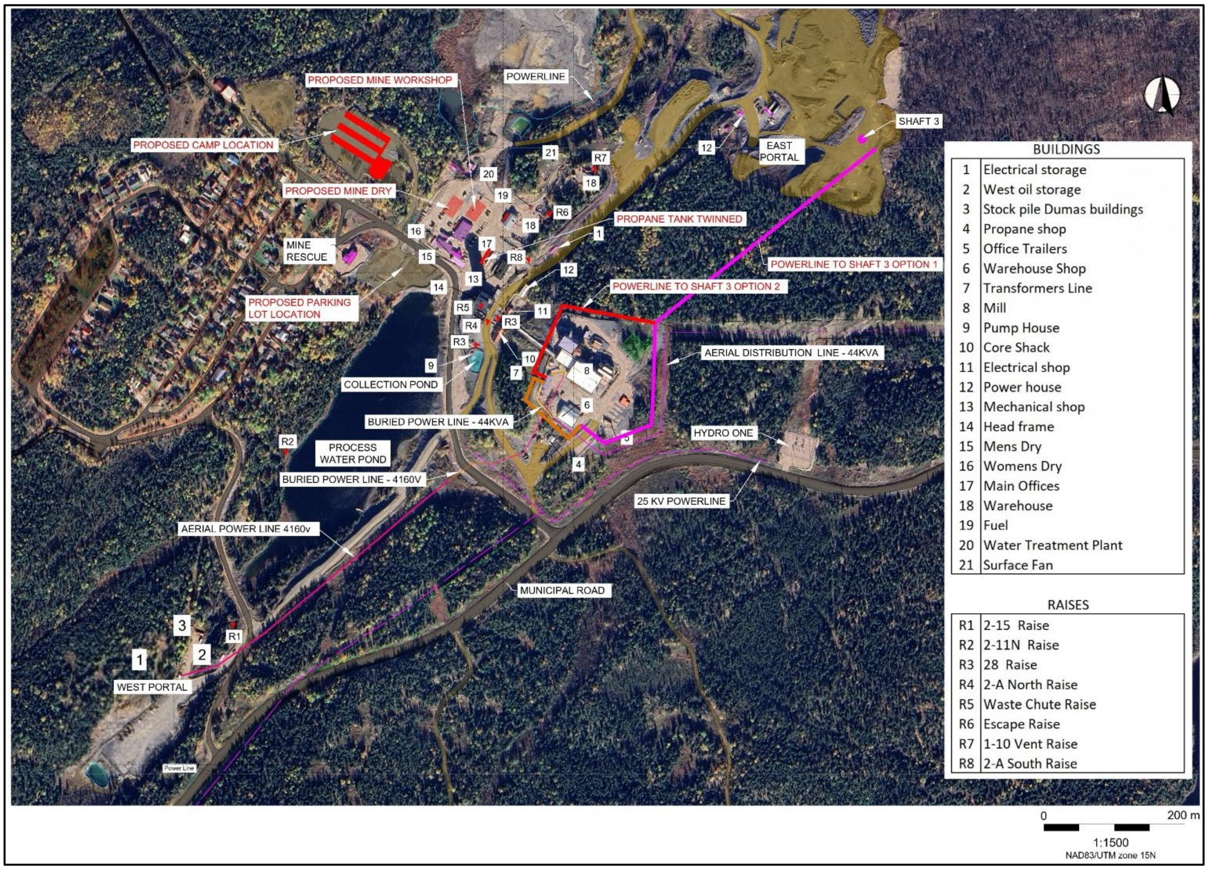
18.7 Site Buildings

The Madsen mine is a mature site and has all the necessary facilities to operate. Under previous ownership several trailer type buildings were rented and then removed in 2021. The site buildings can be seen in Figure 18-5, with black text representing pre-existing structures and red text highlighting the recent or proposed buildings required for the project.

To reduce the reliance on external suppliers for fixed assets and lower the operating cost, WRLG have opted to purchase and install the following:

- Mine dry (190 men / 40 women)
- Mine camp (114-person capacity)
- Mobile workshop (maintenance of the mobile equipment)
- A mobile crusher to be installed at the East Portal

Figure 18-5: Location of site buildings



Source: Allnorth (2025)

The offices and office complexes that will be required to support operation of the Madsen mining and milling operations are summarized in Table 18-5.

The mill dry and office space are not expected to increase in size as there is no expected increase in manpower at the mill.

18.7.1 Mill and Power Distribution Centre

The mill building is preexisting and it will not change in size or configuration. The property's main 44 kV Hydro One is located at the Mill all the power for the site is distributed from this substation at 4160 V and 600 V.

18.7.2 Administration Office

The existing building incorporates the administration, maintenance, mine operation, environmental, safety, geology and engineering but does not meet the required office space. Once the new mine dry has been built, the mine operations and mine maintenance departments would be relocated to this facility.

Table 18-5: Madsen office requirements

Complex / Building	Number of Offices	Office Occupancy
Administration		
Office Multiple Occupancy	5	14
Office Single Occupancy	11	14
First Aid		
Office Multiple Occupancy	1	1
Mill Building		
Office Multiple Occupancy	3	8
Office Single Occupancy	5	5
Mine Operations Complex		
Office Multiple Occupancy	6	26
Office Single Occupancy	9	10
Warehouse		
Office Multiple Occupancy	1	2
Grand Total	**41**	**80**

Source: Allnorth (2025)

18.7.3 Analytical Laboratory

The analytical laboratory was previously contracted out and the samples were sent to Red Lake for analysis. WRLG plan to continue using external analytical services provided by a laboratory in the region. Environmental samples or exploration drill samples will continue to be analyzed by external specialized laboratories.

18.7.4 Mine Dry/Change House

The change house or mine dry will be in the Mine's Operation Trailer Complex. The change houses will be utilized by the mining, mechanical, electrical, site services and technical services personnel. The change house will have a section for the male employees (190 people) and a section for the female employees (40 people). Additional space has been considered for specialized mining contractors, specialized contract mechanics and technicians, diamond drillers as well as visitors.

The process personnel have their dry at the mill site and there are no plans to expand this facility.

18.7.5 Training Facility

The training facility will be part of the Mine's Operation Trailer Complex. It will be a room for up to four people and will be utilized by the mine trainer to perform induction courses established by WRLG.

18.7.6 Warehouse

A small tent-style warehouse already exists at Madsen. Most of the materials used for mining will be stored outdoors on racks. Only high-value items will be stored inside or in containers.

18.7.7 Security and Site Access

This trailer complex already exists and does not need to be increased in size. It is connected to the administration complex through a corridor.

18.7.8 Mobile Equipment Workshop

Under previous ownership there was no workshop for the mobile equipment on surface. Most maintenance was performed underground. Small workshops on surface are inadequately sized to be able to properly maintain the equipment. As WRLG has opted to reduce the number of contractors, the mobile maintenance functions will be performed by mine personnel.

The new 500 m^2 mobile equipment workshop will be located within 100 m of Shaft #2. This building will be of steel construction with a concrete foundation.

The building will be equipped with the following:

- Maintenance personnel offices
- Tool crib and component repair station
- Two maintenance bays
- One wash bay
- One lubricating station
- A welding area

This facility will permit access to mechanics and maintenance contractors and will not be affected by the working schedule of the underground mining operations.

18.7.9 Propane Storage

The site is presently equipped with 2 x 30 000 US gallon propane tanks that are primarily utilized for the mine heater located at the headframe. Additional heating capacity was added in 2024 for mine heating, the mobile workshop and the new mine dry. Most of the site buildings are heated with propane. The propane is supplied by truck from the depot in Red Lake or directly from the south from a distribution centre.

18.7.10 Wastewater Treatment Plant

A wastewater treatment plant already exists on the Madsen site. This plant serves the town of Madsen as well as the mine site and is the responsibility of WRLG to operate and maintain. The wastewater treatment plant has the capacity to treat the additional load from the new camp. The wastewater treatment plant facility capacity has been validated by a third-party engineering firm.

18.7.11 Core Shack

A fully equipped core shack is located within 30 m of the Shaft #2 headframe. No changes are planned for this building.

18.7.12 Emergency Services

The town of Madsen has a fire station located 150 m from the Administration Area. In case of a fire, the fire truck is available for use at the mine. Only extinguishers will be installed in the administrative area. No hydrants are planned for this area.

The mill building is equipped with fire hydrants and fire extinguishers.

The fire station also serves as the mine rescue station for the mine. The Red Lake area is well equipped in case of emergency as several larger mines are located within 15 km of the Madsen Mine.

Red Lake has both a hospital service and an ambulance service located 12 km from the mine site.

18.7.13 Fuel Supply and Storage

The estimated fuel storage requirements for site are summarized in Table 18-6. It is assumed that diesel exhaust fluid (DEF) will be used if Tier 4 engines are used in the mine.

Table 18-6: Site fuel storage requirements

Tank Function	Nominal Size (L)	Tanks (ea)	Actual Volume (L)	Diameter (cm)	Tank Length (cm)
Fuel	35000	2	38846	229	894
Gasoline	10000	1	9972	163	536
DEF	4600	1	4607	130	376

Source: Allnorth (2025)

18.7.14 Water Supply

The domestic water consumption for Madsen is already supplied from the community water source. Table 18-7 outlines water consumption for the site. The water supply should be rated for 15 m^3/day for mine operations.

The water requirements for the mine camp will be approximately 27 m^3/day for all services, which include the kitchen, washing machines and personal hygiene.

Table 18-7: Mine site domestic water consumption

% Usage Factor		Workers x Usage	Shower	Toilet	Urinal	Sink	Washing Machine	Kitchen Sink	Other Consumers	Litres Consumed / day
0.65		188.5	12							2,262
1.30		377.0		10						3,770
0.60		174.0			1					174
2.20		638.0				4				2,233
# Uses		Machine x Uses								
6.0		1					117			702
8.0		1						19		152
3.0		1							150	450
		Total								**9,743**

Source: Allnorth (2025)

18.7.15 Mine Camp

WRLG is constructing a 114-person camp located 150 m to the northwest of the mine administration area, with completion expected in March 2025. The lack of affordable housing, transportation logistics and the lack of specialized personnel has prompted the mine to provide accommodations to the out-of-area workers. The workers will work on a rotation basis. The rooms are single occupant quarters with bed and washroom facilities. The power, water, and sewer services are all within 50 m of the location of the mine camp.

The camp will be equipped with sleeping quarters, a cafeteria, washing facilities and common areas.

The heating for the mine camp will be propane. The camp will also be equipped with an emergency generator

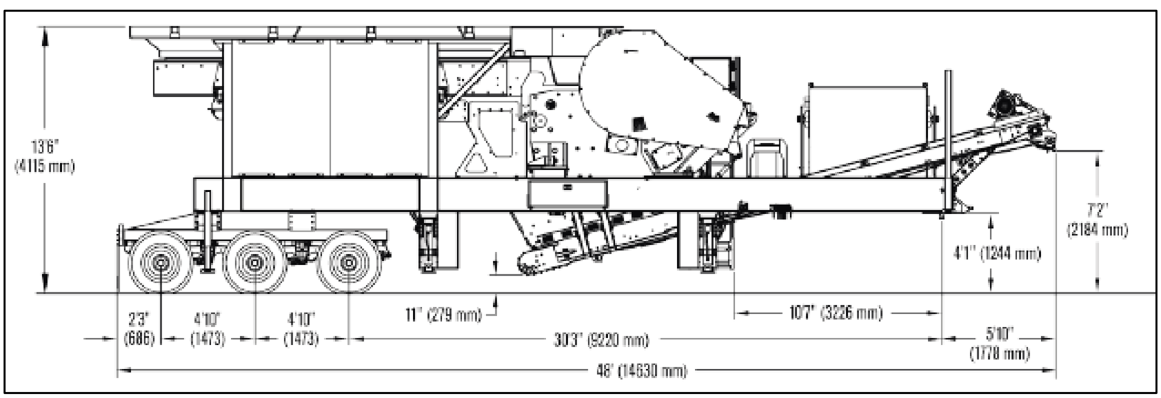
18.7.16 Process Water

The raw water for the process plant is pumped from the process water pond identified in Figure 18-5, located east of the administration area.

18.7.17 East Portal Crusher

The Cedarapids® CRJ3042 plant combines the popular JW42 jaw crusher with a high stroke, 42” x 20’ (1067 x 6096 mm) vibrating grizzly feeder. This plant includes a hopper design that can handle severe primary crushing applications. The unit is mobile but will be positioned at the East Portal. The crusher will be fed with a loader. The ore coming from the West Portal and East Portal will be crushed and then delivered to the plant feeder located at the headframe. The view of the crusher is found in Figure 18-6.

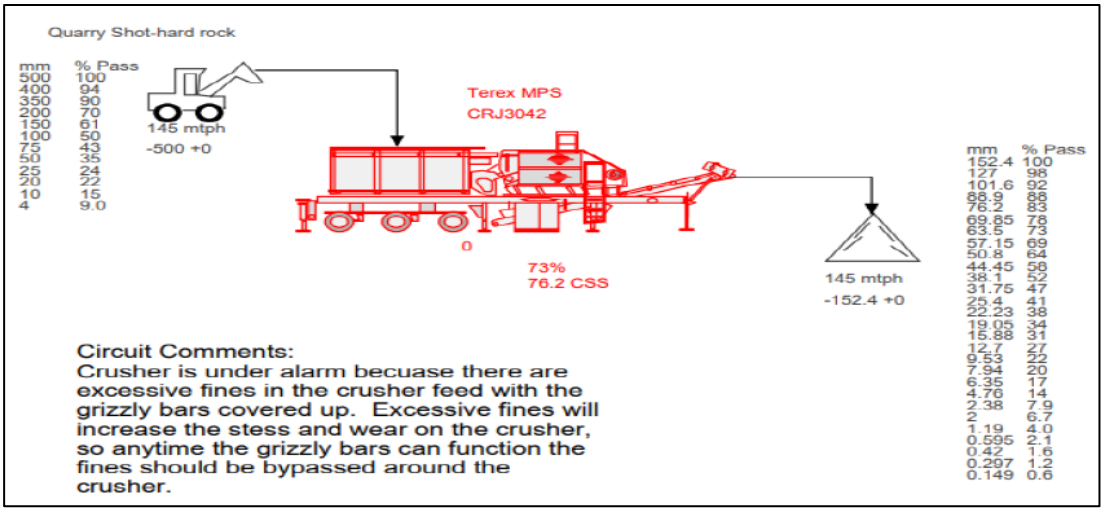
Figure 18-6: East Portal jaw crusher



Source: Allnorth (2025)

The crusher is rated for 147 t/h and is designed to produce a P80 -80 mm product. The particle size distribution is found in Figure 18-7.

Figure 18-7: Crusher product specifications



Source: Allnorth (2025)

18.7.18 Shaft #3 Substation

A single 10 MVA 44kV to 4160 V transformer will be required for Shaft #3. An additional 4 MW 4160 V to 600 V transformer will also be required to provide the 600 V motors at the headframe. The substation will be necessary to supply the 4160 V power for the underground as the mine develops to the east.

18.8 Waste Rock Storage

Approximately 2.6 Mt of Mine Rock material will be produced from underground mining development and will primarily be managed in the underground mine as backfill or stored in the existing Mine Rock Management Facility (MRMF) located adjacent to the Tailings Management Facility (TMF). Multiple void historic stopes create good opportunity to store waste rock underground, thereby reducing costs. Mine Rock required for additional underground backfill and construction activities will be sourced from this stockpile.

Approximately 230,000 m^3 of Mine Rock will be used for construction activities during operations. Mine Rock used for construction activities is expected to be overhaul material and placed directly at the construction fronts to avoid double-handling, where possible. Non-potentially acid generating (Non-PAG) Mine Rock will also be used at closure to cover the TMF and this material will be sourced from the MRMF. It is expected that all of Mine Rock in the MRMF will be as part of the closure of the TMF and no stockpile will remain at closure. The general high level design parameters are summarized below.

- Total Mine Rock Mined = 2.6 Mt
- Mine Rock to Mine Rock Management Facility (MRMF) = 1.43 Mt
- Mine Rock Direct to Construction Activities = 0.51 Mt
- Mine Rock to Underground Backfill = 0.63 Mt

The MRMF is currently operational with Mine Rock placed on the south side of the facility. Surface water run-off naturally drains to Cell A from the south side of the MRMF. The facility footprint will be expanded to the north with the northern portion of the facility naturally draining to the Main TMF (Cell C) and surface water run-off from the north of facility directed to the Polishing Pond. The MRMF is designed with 10 m-high lifts and typical 10 m-wide benches (1.4H:1V bench face slopes). The toe elevation of the dump is approximately at 394 masl. The maximum crest is at 430 masl and the maximum height of the waste rock along the typical cross section is approximately 30 m with a resulting overall slope of approximately 2.5H:1V.

18.9 Tailings Storage

18.9.1 General

Tailings will be managed through a combination of surface storage in the TMF and underground deposition as hydraulic backfill. The TMF at the Project has been in operation since the late 1930s and has gone through several design modifications. The TMF at the Madsen Mine is permitted to discharge tailings and will be expanded to manage a total of 1.5 Mt. The TMF is partitioned into two designated areas, Cell A and the Main TMF. Containment for the first four years of tailings deposition will be provided in Cell A, with the remainder of the tailings managed in the Main TMF. Cell A is fully constructed, including a 4-metre dam raise that was completed in summer 2024, and is fully permitted to receive tailings.

The Main Dam will be constructed downstream of the existing polishing pond dams in Year 4 to raise the perimeter crest elevation of the TMF and provide storage capacity for the remaining tailings. The two areas of the TMF offer sufficient capacity for supernatant ponds, surplus water storage, and flood water management.

18.9.2 Design Basis

The high-level design basis for the TMF design is summarized below:

- Mill Site Elevation = 420 masl
- Mill Throughput (typical) = 800 tpd
- Operating days/year = 360
- Mill availability = 95%
- Ore Mined = 1.8 Mt
- Total Tailings = 1.8 Mt
- Total Tailings to the TMF = 1.5 Mt
- Total Tailings to the Underground Backfill = 0.3 Mt
- Mill Tailings pumped to the TMF in single tailings pipe

18.9.3 Tailings Characterization

A laboratory testing program was conducted in 2018 to determine the geotechnical characteristics of the tailings. Average dry densities for the composite tailings samples range from approximately 1.3 t/m^3 at very low stresses to 1.6 t/m^3 at high stresses. Dry densities for the overflow tailings samples, range from approximately 1.0 t/m^3 at low stresses to 1.4 t/m^3 at high stresses.

Lorax Environmental Services (Lorax) conducted a metal leaching (ML) and acid rock drainage (ARD) assessment on the tailings. Static test data indicate that the existing tailings can be generally classified as potentially acid generating (PAG), with net potential ratios (NPRs) of <1 for most samples. Measurements of acidic paste pHs and the presence of minerals that form under acidic conditions (e.g., jarosite) indicate the presence of localized acidic conditions in the near-surface unsaturated materials. Groundwater quality data for wells screened in the saturated tailings show circum-neutral to basic pH and very high levels of bicarbonate alkalinity, indicating that any localized acidity generated in the surface tailings is effectively neutralized along subsurface pathways in the saturated zone. Overall, given the current circum-neutral to alkaline drainage pH observed for the TMF, and the proposed reclamation and closure strategies that inhibit oxygen ingress (vegetative covers), acidic drainage from tailings is not expected to be a concern over the long term.

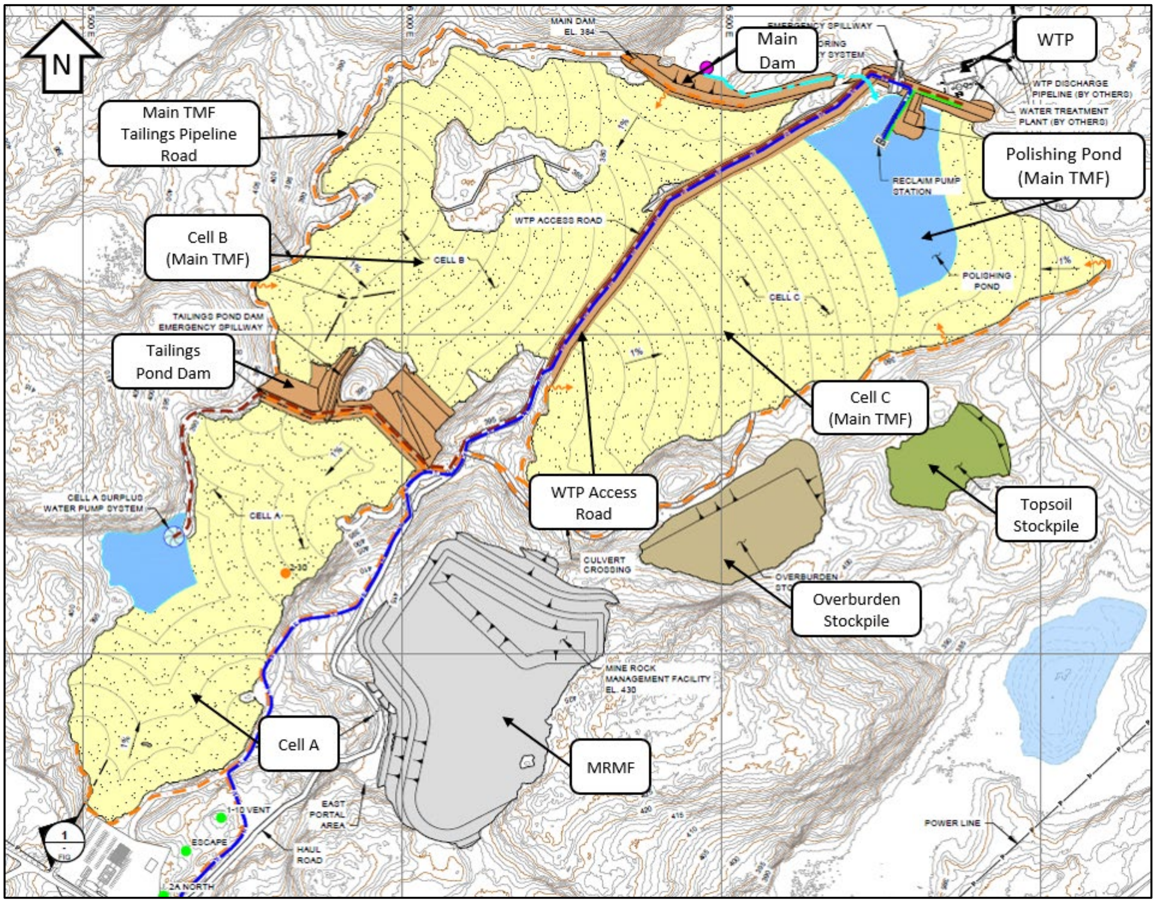
18.9.4 Tailings Management

The TMF is designed with two separate impoundment areas for the management of the tailings and water management:

- **Cell A:** Tailings will initially be deposited within Cell A which is confined by the TP Dams. The TP Dams were raised in 2024 to a final crest elevation of 394 masl. Cell A will provide storage for approximately four years (Year 1 to Year 4) of tailings deposition in addition to the operating pond and EDF flood volumes. Extreme inflows exceeding the EDF storage allowance capacity in Cell A will discharge through an emergency spillway to the Polishing Pond. Cell A is designed as a solids retention facility and is not designed to be used for long term storage of surplus water.
- **Main TMF:** The Main TMF is located downstream of Cell A and is confined by the Main Dam. The Main TMF includes the areas previously referred to as Cell B, Cell C, and the Polishing Pond. The Main Dam will be constructed up to a final crest elevation of 384 masl. The Main TMF will provide storage for tailings deposited beyond Year 4 (Year 5 to Year 8) while maintaining sufficient capacity for contact water storage in the Polishing Pond and the EDF flood volume. Extreme inflows exceeding the storm storage allowance capacity in the Main TMF will discharge through an emergency spillway with an invert elevation of 382.5 masl. Long term accumulation and storage of contact water at the mine will be managed in the Polishing Pond, which will become part of the Main TMF once construction of the Main Dam is complete.

The TMF general arrangement upon the end of operations (Year 8) is provided in Figure 18-8.

Figure 18-8: TMF general arrangement (Year 8)



Source: Knight Piésold (2025)

18.9.5 Main Dam

The Main Dam will be constructed up to a final crest elevation of 384 masl and will provide additional storage capacity in the Main TMF. The Main Dam will be founded primarily on bedrock with up to approximately 10 to 15 m of foundation excavation required to remove unsuitable foundation materials including tailings, organic deposits, and a glaciolacustrine clay layer. The Main Dam will be located downstream of the existing Polishing Pond Dams and will be sufficiently distanced as to not destabilize the downstream toe of the Polishing Pond Dams during the foundation excavation.

The Main Dam will be constructed and operated as a water retaining dam. The majority of the fill for the Main Dam will be Mine Rock and is expected to be overhaul material sourced from underground mining operations. Additional Mine Rock for construction of the Main Dam may also be sourced from the MRMF. The upstream face of the dam will include a layer of filter sand, which will function as a geomembrane liner bedding layer. The geomembrane liner will be installed on a non-woven geotextile layer on top of the filter sand material and will be anchored into the bedrock foundation using a concrete plinth. The embankment will include appropriate filter and transition zones to prevent the downstream migration of tailings and fine material.

Seepage rates through the Main Dam are expected to be minimal. Seepage flows through the Main Dam will be minimized in future operations by keeping ponded water away from the dam structures and by lining the upstream face of the dam with a 2 mm high-density polyethylene (HDPE) geomembrane liner. The liner system includes a layer of non-woven geotextile below the geomembrane for protection from the adjacent materials.

The Basin Seepage Recovery System will be constructed at the upstream toe of Main Dam to collect tailings seepage flows. This system will be excavated, backfilled with coarse drain rock, and will include inclined riser pipe wet wells constructed on the upstream face of the Main Dam. The drainage systems will convey water to the seepage collection sumps for flow measurement and recycle back to the pond. The Water Monitoring and Recovery System (WMRS) will be constructed downstream of the Main Dam to depress the water table during early operations and prevent pressure build up behind the liner. Derlak lake is located downstream of the Main Dam and groundwater from the lake will be collected in the WMRS and either pumped back to the TMF or discharged downstream if water quality objectives are achieved. The WMRS will also perform as a redundant seepage collection and recovery system if required.

18.9.6 Emergency Spillways

The operating basis for the TMF includes temporary storage of the EDF. This storage provision has been incorporated into the staging of Cell A for the Cell A catchment and within the Main TMF for the entire TMF catchment. Extreme inflows larger than the EDF exceeding the available storage capacity will discharge by way of the emergency spillway constructed at Cell A and an emergency spillway to be constructed at the Polishing Pond. The primary objective of the emergency spillways is to protect the integrity of the dams during an emergency, and they are not intended to be used at any stage during operations.

18.9.7 Mechanical Systems

The tailings distribution system is designed to deliver mill tailings to the TMF and to facilitate development of tailings beaches along the inside perimeter of the TMF. The system will consist of three primary components: a tailings pump station, tailings distribution pipe, and discharge spigots.

The tailings distribution system and the configuration of discharge spigots will evolve during operations as the TMF dams and beaches develop and as operating procedures are refined. Tailings discharge will be rotational, whereby a spigot (or multiple spigots) will operate for a short period of time i.e. one month, and then active discharge is moved to the next spigot.

The reclaim system consists of a pump station and a single pressurized pipe to deliver water from the Polishing Pond to the Process Plant. The reclaim pump station consists of two floating vertical turbine pumps (one operational and one standby).

The surplus water system consists of a pump station and a single pressurized pipe to deliver water from Cell A to the Polishing Pond. The system has been designed to keep Cell A dewatered during normal operations. The surplus water pump station consists of two floating centrifugal pumps (one operational and one standby).

The Water Treatment Plant (WTP) supply system consists of a single pressurized pipe to deliver water from the Polishing Pond to the WTP. The WTP supply system pump station consists of two floating vertical turbine pumps (one operational and one standby). The pumps share a floating barge with the reclaim system.

18.10 Surface Water Management and Water Balance

18.10.1 Water Management Strategy

Water management is a critical component in the overall waste management strategy. Mine contact water, tailings slurry water, underground mine dewatering flows, and groundwater accumulating in the TMF will be stored and managed in the Polishing Pond (Main TMF), as described below.

The operating basis for the TMF under normal operating conditions involves temporary seasonal storage of all mine contact water, as well as temporary storage of the EDF, which corresponds to the runoff resulting from the 1/200-year return period 24-hour precipitation event. This storage provision is incorporated into the design and staging of both Cell A and the Main TMF. Extreme event inflows over the EDF volume will discharge through emergency spillways constructed in Cell A and the Main TMF.

The volume of water in the Polishing Pond is expected to fluctuate seasonally. Surplus water will be treated and released downstream of the TMF seasonally during the permitted release period to maintain sufficient storage capacity in the Polishing Pond. Surplus water will be passively treated in the Polishing Pond and actively treated in the WTP prior to being released at the designated water discharge point in Coin Creek during the permitted water release period. The Main TSF was sized to account for the seasonal variation in the Polishing Pond volume and provide sufficient capacity for underground mine dewatering accumulation during winter operations. Seepage flows collected in the Basin Seepage Recovery System constructed at the upstream toe of Main Dam will be pumped back to the Polishing Pond. Reclaim water will be pumped directly from the Polishing Pond to the Process Plant.

Tailings will be deposited from multiple discharge locations to maximize the tailings filling into Cell A and to promote the development of the pond at the western side of the facility where the Cell A Surplus Water Pumping System has been constructed. Maintaining the pond at the western side of the facility will keep ponded water away from the TP Dams and from the partially decommissioned vent raise located in the eastern side of the facility. Tailings in the Main TMF will initially be deposited from the Main Dam to develop a tailings beach upstream of the dam. This will reduce seepage, prevent ponding immediately upstream of the dam, and will fill the storage area between the existing Polishing Pond Dams and the Main Dam.

18.10.2 Water Balance

Water balance modeling indicates the TMF will operate in a surplus during operations. Surplus water will be managed within the Polishing Pond prior to being treated and released to the downstream environment. The amount of surplus water to be managed varies over the operating life of the Project and is primarily dependent on the underground dewatering rates. Average annual treatment and discharge volumes are expected to range from 0.5 million cubic metres per year (Mm³/year) to 1.2 Mm³/year.

The TMF water balance demonstrates that the water management strategy can be executed over the full range of variable climatic conditions, based on the inputs presented. This includes achieving the reclaim water requirements during prolonged dry climate cycles and providing sufficient storage capacity within the TMF to manage surplus water during prolonged wet climate cycles without the uncontrolled release of surplus water to the downstream environment during normal operating conditions. The TMF water balance model results indicate that the Polishing Pond will provide sufficient storage of contact water from Years 1 to 8. Future work should include monitoring the TMF during operations to confirm the tailings parameters and storage requirements, thus maintaining the Polishing Pond water levels below the maximum operating water levels throughout the entire life of mine.

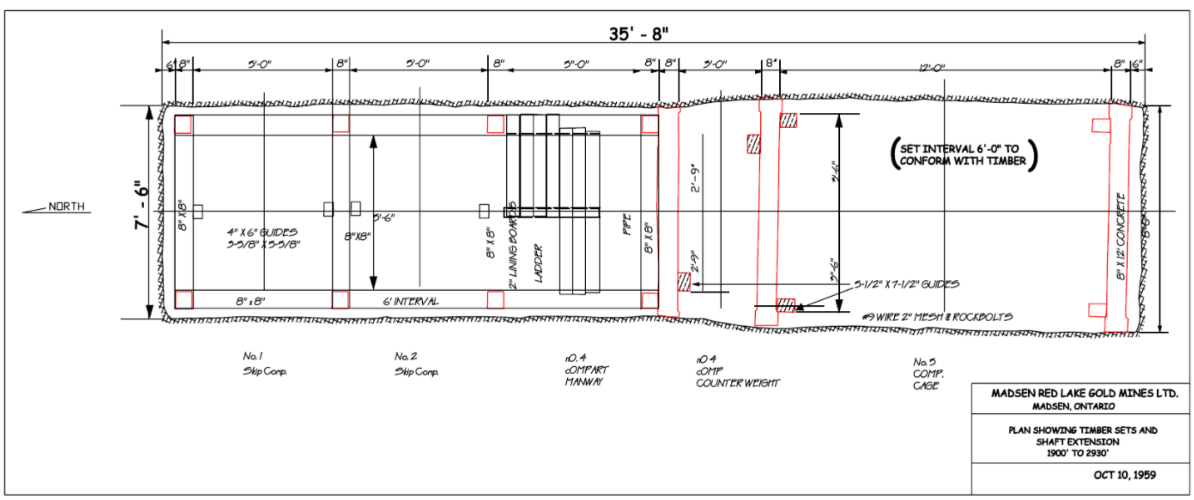
18.11 Shaft Infrastructure

The Madsen Mine will be a shaft-based operation that will utilize the existing Madsen Shaft #2 installation for production in the first three to four years, moving to a new Shaft #3 facility to support ongoing future operations.

18.11.1 Madsen Shaft #2 Refit

The existing Madsen Shaft #2 system was constructed in the late 1950s, with completion of the sinking effort in 1958. The finished shaft is a five-compartment rectangular shaft that was designed to accommodate both production and service hoisting, with a central manway, as shown in Figure 18-9.

Figure 18-9: Madsen Shaft #2 plan arrangement



Source: Nordmin (2025)

The finished shaft was developed to a depth of 3,910 feet, with completion in mid-1958, allowing for operation from 24 Level of the mine. The original design was conceptually designed to include additional levels extending to a final depth of 5,200 feet but was never deepened after 1958. The shaft is timber furnished, with concrete dividers within the cage compartments.

The shaft supported two hoisting plants: a tower-mounted Koepe production hoist in a skip/skip configuration and a tower-mounted Koepe service hoist in a cage/counterweight configuration. The production hoisting compartments and the cage/counterweight compartments are separated by a tightlined timber manway that extends the length of the finished shaft, and also accommodates services and utilities.

The planned use of Shaft #2 as part of WRLG's operation will be as a muck hoisting shaft only, with provision for second means of egress via a skip/cage conveyance. As such, only the two skip compartments and the manway will be in use. The cage and counterweight compartments will be used exclusively for ventilation purposes.

The rehabilitation of the Shaft #2 facility will include several specific efforts:

- Replacement of components of the existing Shaft #2 ore bin loadout, including gates, draw points and feeders
- Rehabilitation and refit of the Shaft #2 production hoisting compartments and manway, with spot replacement of shaft timber, guides, wedges and ground control as required. The shaft recovery effort will be completed in two parts; the first down to 13 Level (-2,050 feet) to allow for the re-commissioning of the 12 Level loading pocket, and a continued dewatering/ rehabilitation effort down to 24 Level (-3,800 feet) to provide access to the shaft for a second means of egress from that level
- Removal and replacement of the 12 Level loading pocket
- Installation of loop dividers and crash sets to support the use of the Koepe friction hoist for operations, as discussed in the next section

This work will allow for the full production rate of 800 tonnes per day ore with additional capacity to handle waste as needed from the 12 Level loading pocket.

18.11.2 Shaft #2 Hoisting Facility

The Shaft #2 hoisting system consists of an 11'-0" diameter tower-mounted Koepe (friction) type hoisting plant, equipped with two (2) 405 kW motors. The system, as procured in the late 1950s, is designed for use with four (4) headropes of 1" diameter, with three (3) tail ropes.

This hoisting plant was fitted with Lebus-type drum shells in the late 1990s to allow it to operate as a single-drum hoist with one rope to support shaft recovery and dewatering efforts. However, the drive system suffered a transformer failure, which then limited the overall power use to 500 kW rather than the original 810 kW.

To utilize this hoisting plant for production purposes, the work required to complete the re-commissioning includes the following elements.

- Removal of the rope and shells currently installed, returning the hoist to its intended Koepe configuration
- Mechanical refitting of the rope grooves on the hoist drum as well as general mechanical maintenance of critical systems/components

- Replacement of the current drive system in its entirety. This includes new incoming transformer and rectification, new motors and digital control systems.
- Inspection and refitting of the conveyances. There are two conveyances currently on site: one skip and one skip/cage system, which allow this hoist to be used as a second means of egress. These conveyances will be shipped to FL Smidth for inspection, repair and recertification.
- Procurement and installation of new hoist ropes and rope connections
- As this is a Koepe-type hoisting plant, hoisting will only be possible from one level, as these systems have fixed distances between conveyances. As such, commissioning for use at 24 Level would require purchase of a new set of head and tail ropes for the hoisting plant.

ABB has completed a detailed inspection of the hoisting plant and drive and provided a proposal that lays out the required work and pricing, which is the basis of the costs of the major procurements for this effort.

18.11.3 Shaft #2 Surface Modifications

The existing headframe and bin will be utilized essentially as is for the new operation, with the following replacements/upgrades being required to allow their use:

- The existing bin drawpoints and feeder system, which allow ore to move from the bin to the mill facility via the main transfer conveyor, will be replaced. The new system will make use of a vibratory feeder installation to allow muck to be loaded onto the conveyor.
- Other minor structural improvements and upgrades to address items like missing grout, and oxidized parts, will be made. There is no requirement for major structural modifications, repairs or upgrades.

This work will be completed ahead of the commissioning of the hoisting plant, and the entire system commissioned for use.

18.11.4 Shaft #3 Facility

As operational efforts progress for the Madsen Mine, the centroid of the mining operation will move to the northeast, as mining follows the deposit structure down plunge. This centroid will move the focus of operations more than 1.5 km away from Shaft #2 and to a potential depth of 4,000 feet or more as it is open at depth, which will require extensive rework of existing headings to allow for muck and service haulage to these areas. As well, continued dependence on ramp-based access for personnel and consumables would result in limitations on crew utilization rates and time-on-face.

Through a review of these factors, and the identified need to develop a large ventilation raise to allow for improved capacity in these areas, the conceptual review of equipping this raise for use as a combined production and service shaft was completed and proved feasible and advantageous.

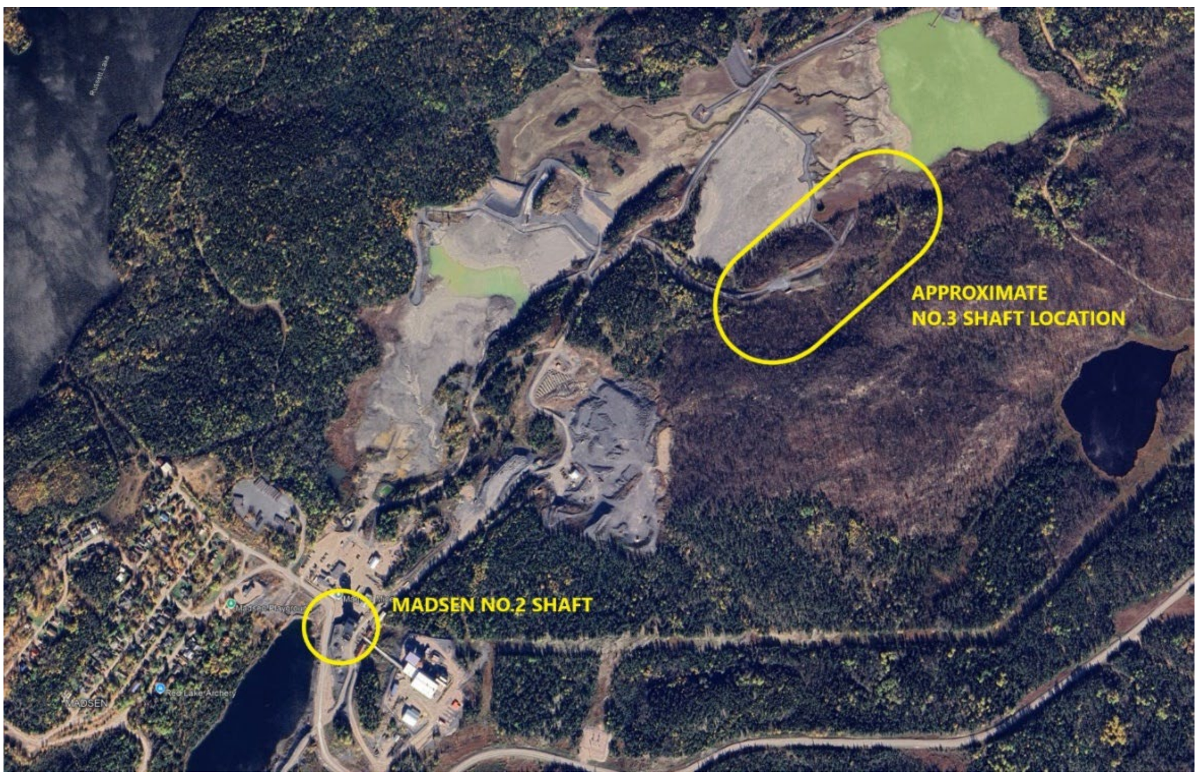
The new Shaft #3 system would be constructed in Year 3 of the project and would be developed via raisebore to both improve cost and schedule for its development. In this concept, the shaft would be developed in two sections, with the first from 10 Level to surface, and the second from 18 Level to

10 Level, allowing for the surface plant to be constructed and shaft equipping to be undertaken while the second leg was in development. A rock pentice would separate the two legs, then excavated once the lower section of the shaft is completed, allowing for the completion of ground control and shaft furnishing to shaft bottom.

The ultimate depth of the shaft will be at or around 4,000-foot (25 Level) via a third lift completed below 20 Level in the future allowing for hoisting from a greater depth. Further deepening may be an option as further work is completed to define the orebody's extents along strike and at depth.

The approximate location of Shaft #3 is shown in Figure 18-10. This location will be further defined through additional geological exploration to better define deeper resources as well as geotechnical efforts to confirm the lithology and rock mass characteristics in the deeper extents of the deposit in the coming two years.

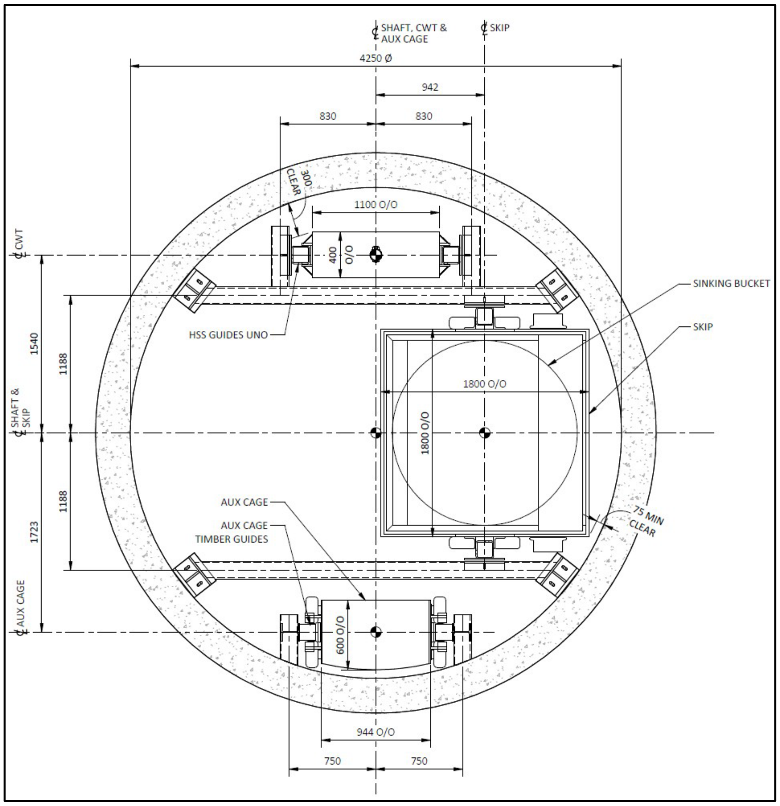
Figure 18-10 - Approximate location of the planned Shaft #3 facility



Source: Nordmin (2025)

The Shaft #3 facility will be constructed as a raisebored 4.25 m-finished diameter shaft, which will be furnished to allow for the use of a skip/cage conveyance for production and service needs and an auxiliary cage that will be used both as off-shift personnel movement as well as emergency second means of egress. The shaft plan for this new facility is shown in Figure 18-11.

Figure 18-11 - Conceptual plan of Shaft #3



Source: Nordmin (2025)

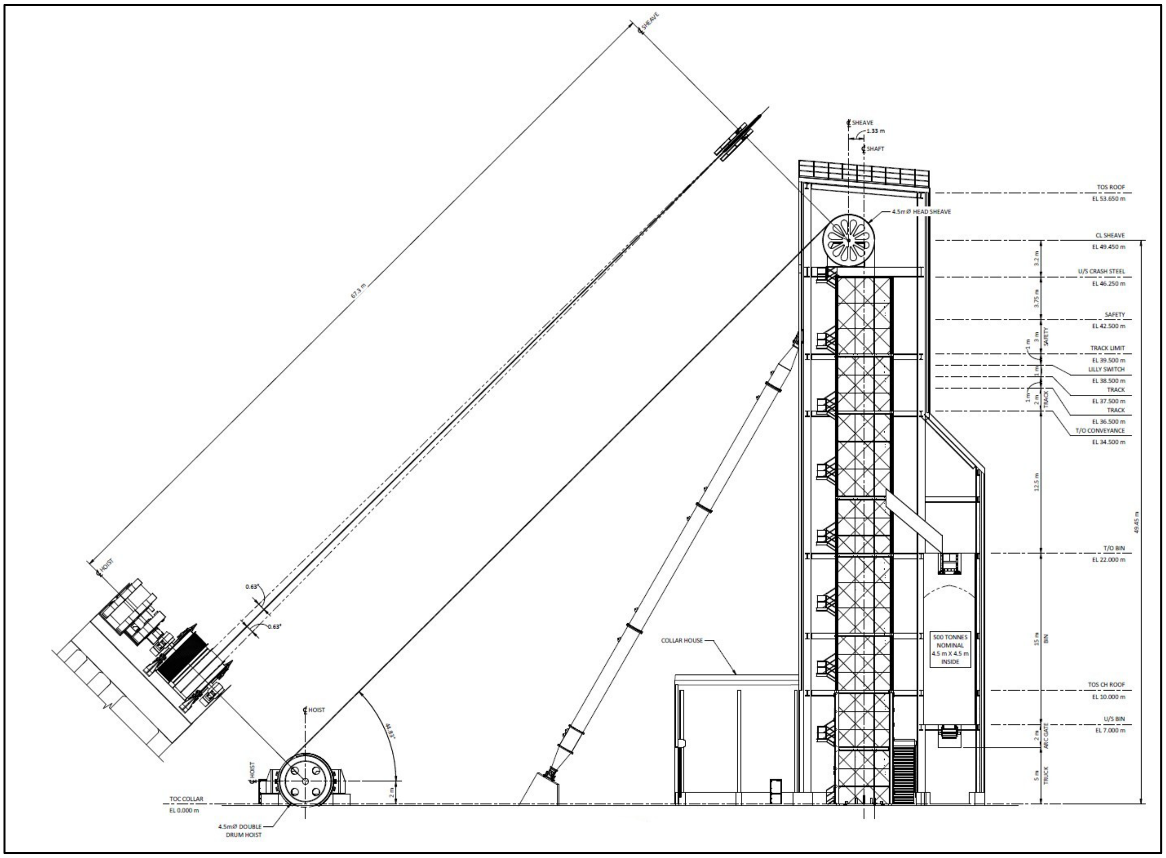
The Shaft #3 hoisting plants will be a double-drum production/service hoist to support a skip/cage in balance with a counterweight to limit power requirements, and a single drum auxiliary hoist. The characteristics of these two hoists are shown in Table 18-8.

Table 18-8: Shaft #3 hoisting system design (Source: Nordmin, 2025)

SYSTEM PARAMETER	PRODUCTION HOIST	SERVICE HOIST	AUXILIARY HOIST
HOIST CONFIGURATION	Double drum hoist	Double drum hoist	Single drum hoist
ONTARIO CODE	REGULATION 854, CLAUSE 228	REGULATION 854, CLAUSE 228	REGULATION 854, CLAUSE 228
	metres	metres	metres
DIAMETER	4.5	4.5	2.4
WIDTH	1.5	1.5	1.5
	kg	kg	kg
CONVEYANCE PAYLOAD	12,000	12,000	3,000
TARE WEIGHT	11,340	11,340	3,000
COUNTERWEIGHT	18,144	18,144	-
NUMBER OF ROPES	1	1	1
ROPE DIAMETER	44.45 mm	44.45 mm	25 mm
ROPE CONSTRUCTION	CASAR VERSAPLAST 1960 MPa	CASAR VERSAPLAST 1960 MPa	CASAR TURBOPLAST 1960 MPa
ROPE WEIGHT	9.61 kg/m	9.61 kg/m	2.90 kg/m
BREAKING STRENGTH	1,839 kN	1,839 kN	554 kN
FOS REQUIRED – CAPPEL	5.0	5.0	5.0
FOS REQUIRED - SHEAVE	7.5 (MEASURED LOAD)	7.5 (RESTRICTED LOAD)	7.5 (RESTRICTED LOAD)
FOS ACTUAL – CAPPEL	5.36	5.36	5.93
FOS ACTUAL – SHEAVE	8.03	8.03	9.41
TREAD PRESSURES	3.44 MPa (less than 3.5 MPa required)	3.44 MPa (less than 3.5 MPa required)	3.01 MPa (less than 3.5 MPa required)
HOISTING SPEED	12.19 m/s	12.19	4.57 m/s
ACCELERATION	0.91 m/s2	0.76 m/s2	0.61 m/s2
DUMP TIME	20	60	60
CYCLE TIME	149.33 seconds	189.33 seconds	344.83 seconds
PRODUCTION RATE	tonnes	tonnes	tonnes
- HOURLY, DRY	144.6	-	-
- HOURLY, 2% WATER	141.8	-	-
- DAILY, WET (@ 10 HRS)	1418	-	-
DRIVE SIZING	kW	kW	kW
- RMS DRIVE NEEDS	1,678	1,678	321
- RMS DRIVE SIZING	1,700	1,700	350

The surface plant to support the hoisting system will consist of a structural steel headframe, equipped with a two-cell bin system that has a capacity of 500 tonnes per cell. This allows for separation of ore from waste for rehandling on surface. The bins will be unloaded using surface trucks that will then be routed to an existing truck dump at the Shaft #2 headframe for movement through to a new surface crushing plant and the existing processing facility. An elevation of the headframe is shown in Figure 18-12.

Figure 18-12 - Elevation/section of Shaft #3 headframe and main hoist



Source: Nordmin (2025)

18.11.5 Ore Handling System

The ore handling system for the two shafts will be very similar to one another. In both cases, the general ore flow will be handled in the following fashion.

Ore will be delivered to dump points via truck. The dump points will be grizzly and rockbreaker protected to allow muck to be screened and larger pieces broken up for handling.

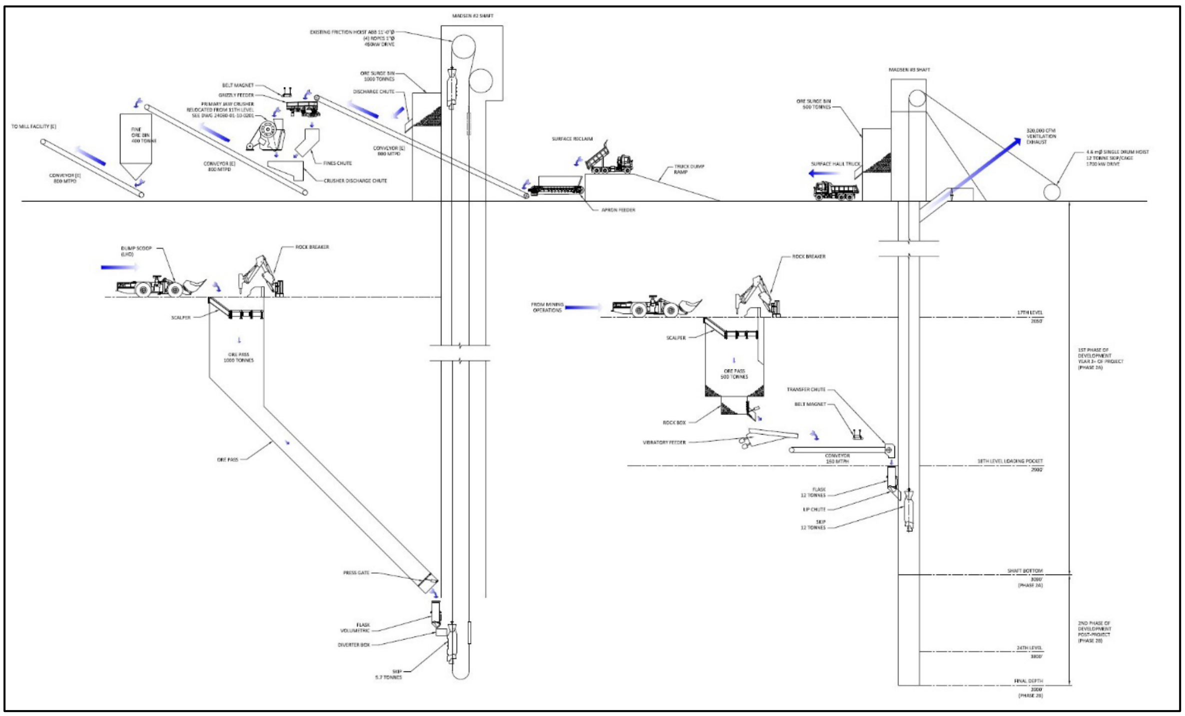
Below each dump point, a raise or bin will be excavated to store several hundred tonnes of material. In the case of Shaft #2, the existing dump on 11 Level will be rehabilitated through the former crusher station and to the loading pocket, providing upwards of 1000 tonnes of storage. At Shaft #3, the raise system will provide 500 tonnes of storage.

At Shaft #2, the raise feeds directly into a volumetric pocket for transfer to the skips for hoisting to surface. This volumetric system can be manually operated but will be designed to work on a timed algorithm with PLC control for automatic use.

At Shaft #3, the raise will be drawn using a rock box and vibratory feeder system that will feed a weightometer controlled loading pocket. The loading pocket will be fully automatic with PLC control.

After ore is hoisted to surface, the ore will be handled to a new surface crushing plant that will size all ore from mining operations and transfer it via conveyor to a new fine ore bin to provide surge capacity ahead of the processing facility. The PFDs for the overall operation is shown in Figure 18-13.

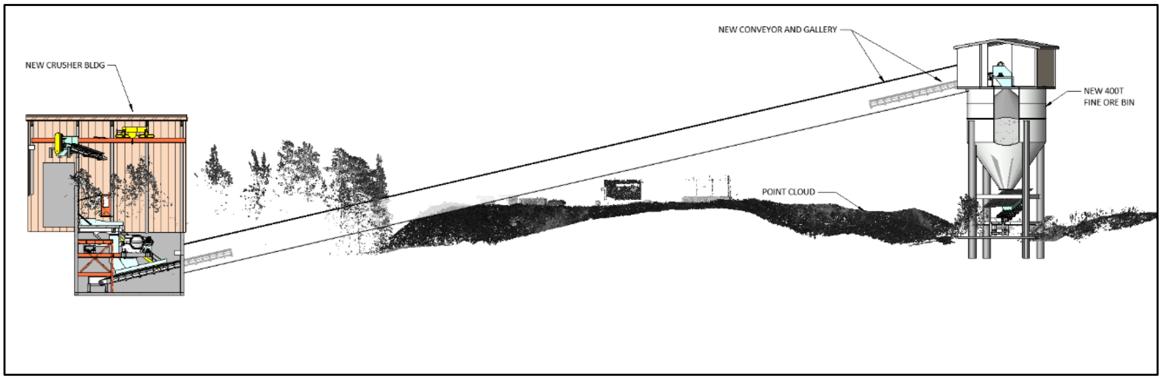
Figure 18-13 - Ore handling PFD for Madsen Project



Source: Nordmin (2025)

The new fine ore bin will have a 500-tonne capacity and will feed a short shuttle conveyor to move the sized ore into the mill reporting to the same work point as the existing transfer conveyor. A section of the new crusher to fine ore bin is shown in Figure 18-14.

Figure 18-14 - Section through new surface crusher and fine ore bin



Source: Nordmin (2025)

19 Market Studies and Contracts

19.1 Market Studies

No market studies were conducted regarding the sale of doré from the Madsen Mine as part of the PFS. Existing terms from previous sales by Pure Gold were used as the basis of the financial modelling described in Section 22 (Table 19-1).

Table 19-1: Treatment charges and refining charges

Item	Value
Refining charge ($/payable oz)	C$0.50/oz
Freight and insurance charge (% contained value)	0.06%

Source: SRK (2025)

19.2 Contracts

At this time, no contractual arrangements for shipping, port usage or refining exist, nor are there any contractual arrangements made for the doré at this time. However, doré is a highly marketable product and it is not foreseen that there will be any difficulties selling the mine’s production.

19.3 Royalties

A 1% NSR royalty exists with Sprott Resource Lending Corp.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Summary

WRLG is continuing its scientific and engineering studies at the site; consultation with regulators, First Nations and communities; monitoring programs; and detailed project design planning is underway to reopen the mine and processing facility. 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 resource use, improve water quality and bring overall benefit to local communities and First Nations.

20.2 Permitting

WRLG has maintained the permits that existed for the Madsen Mine under previous operators. As the project has advanced, operational enhancements and regulatory changes have required some permit updates. Permit status has been confirmed with both the Ministry of Mines and Ministry of the Environment, Conservation and Parks (MECP) and the following permits and authorizations are in good standing:

- **Permit to Take Water (3301-CVURLV)** for process water: This permit was updated in December 2023 and allows for the pumping of 0.45 million litres of water per day from the process pond for onsite use in the mine or mill. No pumping is currently being undertaken.
- **Permit to Take Water (5376-CVURB3)** for shaft dewatering: This permit was updated in December 2023 and allows for the pumping of 6.5 million litres of water per day from the mine workings. Shaft dewatering is ongoing.
- **Environmental Compliance Approval (ECA) Industrial Sewage Works Permit**: WRLG currently has an Amended ECA for Industrial Sewage Works (9280-C2XNTQ) that allows for the release of suitable quality water from the tailings facility. This ECA allows WRLG to manage water storage in the tailings facility and to manage water levels in the underground operations. Ongoing environmental compliance monitoring of downstream catchments is regularly conducted and annual reports are presented to MECP. The permit is in good standing.

  An amendment is expected to be applied for in 2025 for the installation of a new water treatment plant to replace the temporary/portable water treatment plant (WTP) system that has operated for the last several years.
- **ECA for Air and Noise:** Currently the site has an amended ECA 5217-BPXK2E for the operation of the fixed and mobile equipment operating on the property. This was received in January 2022 and the site completes regular onsite monitoring and submits annual reports as per requirements.
- **Mine Closure Plan, MINES:** A Mine Closure Plan was originally submitted in May 1995 with Closure Plan Amendments following in July 2011, April 2014 and September 2020. Current financial assurance is considered to be adequate for closure of the current mine infrastructure. An updated closure plan amendment is anticipated to be submitted by early 2026.

- **SAR Exemption and Benefit Program Under Clause 17(2)c of ESA for Endangered Bats:** Myotis species bats were encountered in the underground portal area in 2017. A permit allowing WRLG to continue underground operations with acceptable and manageable conditions, including funding bat disease research, reporting requirements, installation of deterrents and installation of bat houses outside of the mining area was granted in June 2017. The permit is in good standing and a final report is anticipated in 2025.
- **Registered Hazardous Waste Generation:** As required by the Ontario Environmental Protection Act Regulation 347, WRLG maintains its registration as a hazardous waste site, primarily focused on the collection of used lubricants and oils related to operations. In preparation for future operations the registration was extended in 2019 to include petroleum distillates and waste compressed gases. Future registration updates are anticipated to account for all potentially hazardous materials planned to be used at the site.
- **PCB Storage Site Closure:** WRLG has decommissioned and closed a legacy PCB storage site established by a previous operator and met all conditions set by Ministry of Environment and Climate Change (MOECC). A closure confirmation letter has been received from MOECC, which is now MECP.

The following existing permits are currently in good standing but may require updating due to process changes or regulatory changes:

- **Permit to Mine, MINES:** A notice of Project Status from “Temporary Suspension” to “Operations” will be issued for the reopening.

Other permits that may be required include:

- **ECA for Sewage:** For approval to construct and operate a domestic sewage treatment system, or Health Unit approval for smaller systems.
- **Work Permit(s):** For any road development, camp or other activities as needed.
- **Plans and Specifications Approval:** For construction of dams or berms, including those associated with tailings facilities and/or new ponds and ditches.
- **Forest Resource Licence:** Annual license for clearing of any Crown merchantable timber.
- **Aggregate Permit (Aggregate Resources Act):** For extraction of any aggregate for dam construction.
- **Notice of Construction:** Notice is required before any contractor or construction activities take place.
- **SAR Approvals:** Although the area falls within the range of several terrestrial species at risk, these have not been encountered on or near the site. Northern Bioscience has been retained to evaluate habitat suitability and develop management plans as required.

20.2.1 Environmental Assessment

Federal Environmental Assessment

There is no requirement for a Federal Environmental Assessment.

Pure Gold prepared an updated Project Definition (January 2019) for the reopening of the Madsen Mine. The Pure Gold Madsen Project as described in the Project Definition was not classed as a designated project under the Canadian Environmental Assessment (CEA) Act (2012) as it did not meet the thresholds defined in Sections 16 and 17 of the Act Designating Physical Activities (SOR/2012-147) Schedule (Sections 2 to 4) Physical Activities:

- 16 The construction, operation, decommissioning and abandonment of a new;

  (c) Rare earth element mine or gold mine, other than a placer mine, with an ore production capacity of 600 t/day or more;

- 17 The expansion of an existing;

  (c) Rare earth element mine or gold mine, other than a placer mine, that would result in an increase in the area of mine operations of 50% or more and a total ore production capacity of 600 t/d or more.

Pure Gold prepared a screening document (February 2019) for submission to the Canadian Environmental Assessment Agency (CEAA) to confirm that this is not a designated project. Due to the reuse of the existing mining and processing infrastructure and footprint, the increase in area remains well below the 50% increase threshold. The total area of the mine as described in the project definition will be expanded from 308 ha to 338 ha, a 9% increase. The expansion of the footprint is required to create a source of closure material for the TMF (rock, overburden, and soil) and will be reclaimed during progressive and final closure. All expansions occur upstream and within the catchment of the TMF in areas that have been previously disturbed by exploration and which largely fall into the historical footprint of the mine.

Provincial Environmental Assessment

There is no requirement for a Provincial Environmental Assessment.

20.3 Environmental Studies

20.3.1 Wastewater Management

The main infrastructure for the mine and TMF are located within the Coin Creek Catchment. The TMF currently receives runoff from the mine site area, pumped water from the underground workings, natural overland runoff from undisturbed areas of the catchment, and inputs from the municipal sewage facility. When the mill restarts, tailings slurry will also report to the TMF and all of these inputs are designed to be managed within the TMF water balance. Impounded water within the polishing pond is the last point of control before discharge.

Since 1977, the community of Madsen has operated a primary treatment (solids settling) septic system from which grey water and sewage are decanted to the southwest corner of the TMF. Discharges from the TMF collect in the polishing pond, from which water has been periodically pumped to Derlak Lake, which in turn drains via a 300 m tributary to Coin Creek. Downstream of this confluence, Coin Creek flows into Snib Lake and then Coin Lake before eventually entering St. Paul Bay (Red Lake). In the early period of mine operation (1930s), tailings were discharged directly to the receiving environment with no containment measures, resulting in the deposition of tailings in the current polishing pond and Derlak Lake.

As a result of historical mining on the property, several mine-related sources continue to contribute to the degradation of water quality downstream of the Project. These include:

- Surface water discharges from the polishing pond to Derlak Lake
- Seepage of tailings pore water through the TMF dams
- Remobilisation of constituents from submerged historical tailings deposits in Derlak Lake, and potentially other systems downstream

In order to manage the wastewater that is expected during mine operation, WRLG will implement several management measures to safeguard water quality downstream of the TMF, including:

- Recirculation and reuse of tailings system water
- Use of cyanide destruction technology prior to release to the TMF (managed process to limit the accumulation of sulphate in wastewater)
- Use of a water treatment plant for water to be released from the TMF (designed to manage ammonia)
- Pump back, if needed, of any seepage from the TMF
- Facilitating the Red Lake Municipality to eliminate Madsen community sewage inputs to the TMF to primarily reduce remobilisation of arsenic and other metals and reduce human health concerns related to mill water recirculation and reuse
- Progressive closure of the tailings pond to reduce water flows and the remobilisation of arsenic and to dry out the upper portions of the TMF that could contribute to seepage. Once the tailings pond is closed only the polishing pond will remain operational to the end of mine life, effectively reducing the tailings footprint by half and eliminating exposed surface area of historical tailings.

These management measures have been included in water quality prediction modelling by Lorax (2019) and an effects assessment has been undertaken. Based on the considerations outlined above, the potential for adverse effects to aquatic biota in Derlak Lake is unlikely from exposure to nutrients (Cl, NH3, NO3, P and SO4,) and metals (Ag, Al, Cu, Fe, Sb, Mn, Ni, and Zn). Adverse effects to aquatic life in Snib and Coin Lakes are not expected to occur. Integrated Sustainability (2019) predicted, and monitoring has illustrated, that the water to be released from the TMF water treatment plant during operations will meet water quality guidelines and provincial regulations.

20.3.2 Noise Management

To prevent additional noise from propagating toward the Madsen Community, acoustic barriers have been placed along the haul road and the crusher unit. These features have substantially reduced noise levels from operations to acceptable levels in the Madsen Community.

20.3.3 Domestic Sewage

Currently, the Municipality of Red Lake operates and holds the operating license for the collection and disposal of domestic sewage from the adjacent community of Madsen, as well as a portion of the domestic sewage from the Madsen Mine. This system is a simple gravity sewer design that has a sewage outfall reporting to the tailings pond after primary solids separation. Approximately 55,000 m^3 of domestic sewage is discharged into the TMF on an annual basis. In 2020 the mine permitted and installed a secondary modular treatment system to further treat the municipal effluent along with the mine sewage effluent. This system contains the following major process stages: a flow equalization, feed pump, biological treatment, secondary clarification and ultraviolet disinfection. Treated sewage is then discharged into Cell A of the TMF. After final water management, all effluent collected in the TMF goes through a final industrial wastewater treatment plant prior to discharge to the environment.

Although WRLG has no responsibility for this system it is considered undesirable that sewage water enters the tailings system as the water from the tailings pond will be recirculated to the mineral processing plant. In addition to the health risk posed by potential worker contact with sewage, sewage could affect the mineral processing circuit and could reduce recoveries or cause higher reagent consumption such as cyanide. The additional cyanide would result in higher levels of ammonia in mill discharge. Lorax (2018) also identified that the sewage is causing arsenic to be remobilised in the tailings system resulting in a higher arsenic load in the water that requires treatment prior to discharge. Additionally, if the sewage is entering the tailings system, progressive and ultimate closure as designed, will not be possible. Therefore, it is necessary to remove the sewage from the tailings system in the short-term to preserve water quality and in the long-term to allow the tailings system to be progressively and ultimately closed.

20.3.4 Infrastructure and Service Requirements

All power, provincial roads and water services are already available at or near the site and WRLG will connect locally to these services.

All significant infrastructure will be placed onto historically used mine areas therefore not increasing the mine footprint.

20.3.5 Environmental Baseline Studies

The baseline scientific studies, predictions and forward modelling required to inform permits has been completed. These are contained and outlined in the site closure plan and include:

- Air quality
- Noise
- Aquatic biology and fisheries
- Climate and hydrology
- Mine rock and water geochemistry

- Groundwater
- Surface water
- Water quality
- Aquatic effects
- Species at risk habitat suitability
- Wildlife

Permit applications will be based on the modelling outcomes. Currently no harm to fish or biological systems in the downstream water bodies is expected. Other than Myotis species bats, no SAR have been identified on the site, although they do occur in the region and a management plan is in place on the project site.

20.4 Social Considerations

WRLG has committed to engagement and consultation with local First Nations, municipal, provincial and federal governments, the public, and stakeholders throughout all stages of the redevelopment. The intent is to provide all interested parties with opportunities to learn about WRLG, identify issues, and provide input with the goal of positively enhancing mine planning and development.

WRLG recognizes the importance of timely, full and open discussion of the issues and options associated with the development and the related concerns those individuals or communities may have in relation to the activities. In light of this, WRLG will maintain open and honest communications with local communities and individual stakeholders throughout all stages of the mine life. WRLG will ensure that its operational practices, both now and into the future, reflect the values, expectations, and needs of the community in which it is operating, based upon continued mutually respectful consultation with all stakeholders.

20.4.1 First Nations Considerations

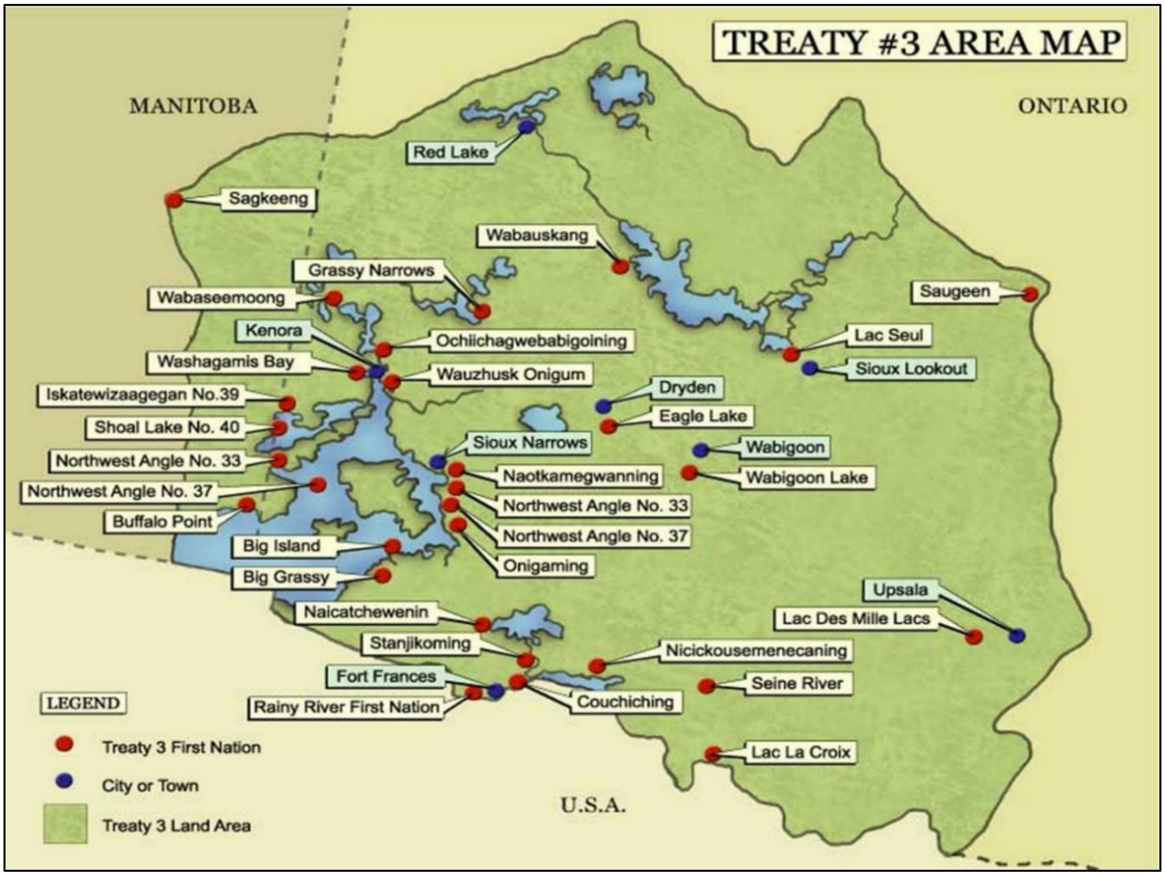
The property is located within the boundaries of Treaty # 3 (1873 and adhesions). Lac Seul First Nation and Wabauskang First Nation have identified the project area as lying within their communities’ traditional territory (Figure 20-1).

MINES has advised WRLG that Wabauskang First Nation and Lac Seul First Nation represent the comprehensive list of First Nation communities to be engaged and consulted at this stage, and further, that the MINES, where necessary, will undertake and fulfill the Crown’s Duty to Consult. Currently, the primary role of WRLG with First Nations is to ensure that appropriate information sharing occurs.

Pure Gold, Lac Seul and Wabauskang First Nations entered into a Project Agreement in June 2019, which WRLG has been assigned. The Project Agreement is broadly based on timely work and employment opportunity notification, capacity building, potential future profit sharing and a variety of other mechanisms for First Nations benefit. Lac Seul and Wabauskang, two communities that

WRLG are required to consult with, have come together on a Shared Territory Protocol for resource projects that are of mutual interest. This allows WRLG to consult openly with both parties functioning as one. There have been numerous meetings with the communities regarding the restart. Collaboration on environment, business development and recruitment is ongoing. WRLG retains a record of all communications and discussions and to date all interactions have been mutually positive and supportive.

Figure 20-1: Area map of Treaty 3 First Nations



Source: WRLG (2025)

20.4.2 Community Considerations

WRLG has had ongoing communication with Red Lake Municipality and the local business community who have an interest in the mine. WRLG held a community meeting in October 2024 to share information and present its redevelopment, recruitment and environmental plans. WRLG has presented project updates to the local Municipality as well as the Madsen Community Advisory Committee. Pure Gold had developed a Consultation Plan that forms the basis of WRLG's ongoing plans for community, regulator and First Nations consultation.

20.4.3 Regulator Considerations

WRLG has engaged with regulators and established positive working relationships. Regulators from, DFO, MINES, MOL, MECP, MNRF and Red Lake Municipality have been involved with WRLG as the reopening plans continue.

20.4.4 Social, Community and Economic Effects

In recent years, mines in the Red Lake area have been struggling to find local staffing. This is why a camp must be established. The mining growth also keeps local contracting businesses extremely busy. As a result, WRLG is working hard to plan its growth infrastructure projects to align with the schedules of local contractors, or when necessary, tendering work to out-of-town businesses.

As mining is a major contributor to the economy of the Red Lake area, reopening of the Madsen Mine will provide a welcome boost to economic activity. WRLG will need to work with other mines as well as the Red Lake Municipality to leave a positive and lasting mining heritage, achieved by building an economic base that can become independent of mining. WRLG intends to work with First Nations and the Red Lake communities to establish mechanisms that will contribute to the long-term sustainability of the communities.

20.5 Mine Closure

20.5.1 Environmental Liability – Closure Plans

WRLG inherited a mining legacy site with a history of almost a century of exploration and mining, and a closure program that was designed by previous operators. The closure plan was updated and additional funding was provided by Pure Gold in 2022. Reopening of the mine will not require an immediate update of the closure plan and closure bond reassessment, but closure plan amendments will be required as the site expands.

20.5.2 Tailings Management Facility Closure and Reclamation

Mining operations will be completed in Year 8 and tailings deposition in the TMF will cease at this time. TMF closure and reclamation activities in full at the end of economically viable mining. Closure and reclamation activities will be in line with regulatory standards and the primary objective would be to eventually return the TMF site to a self-sustaining facility with pre-mining land capability. The TMF will be required to maintain long-term geochemical and physical stability, protect the downstream environment, and manage surface water run-off.

A key objective of the tailings deposition strategy is to integrate tailings deposition planning with the final closure plan. The closure design philosophy for the TMF involves removing all surface ponds at the end of operations and covering the tailings surface with an engineered dry cover system, which naturally sheds non-contact run-off to the downstream environment. This closure plan would eliminate the need for long-term (post closure) water treatment.

Activities that will be carried out during operations and at closure to achieve the closure objectives are:

- Selective discharge of tailings around the facility prior to closure to establish a final tailing beach that will facilitate surface water management and reclamation
- Removal of the supernatant ponds in Cell A and Polishing Pond through water treatment and discharge
- Construction of final dry closure capping system on the entire area of the TMF
- Regrading the facility to re-establish a natural stream through the facility for surface drainage; closure channels will be constructed to direct the stream through natural topography and the stream bed will be lined with rip rap for erosion protection.
- Dismantling and removal of the tailings delivery and reclaim systems and all pipes, structures and equipment not required beyond mine closure
- Establishment of a permanent closure channels from the TMF
- Removal and re-grading of all access roads, ponds, ditches and borrow areas not required beyond mine operations
- Long-term stabilization and vegetation of all exposed erodible surfaces

The WTP will remain operational until water quality requirements are achieved. The WTP will then be decommissioned, and surface water will discharge from the TMF naturally via the closure channel.

20.5.3 Mine Rock Management Facility Closure

The MRMF will be reclaimed at mine closure. Mine Rock sourced from the MRMF will be used at closure to cover the TMF. It is expected that all of Mine Rock in the MRMF will be used as part of the closure of the TMF and no stockpile will remain at closure. Reclamation of the MRMF will be conducted in conjunction with on-going geotechnical and environmental monitoring. Topsoil placement and revegetation of the slopes to return the facility to pre-mining use.

20.5.4 Post-Closure Monitoring

Closure monitoring at receiving waters will be measured against water quality objectives. The following items are planned for monitoring during closure:

- Regular inspections to confirm that closure activities are being undertaken as identified in the final approved Mine Closure and Reclamation Plan
- Construction-type monitoring is undertaken during decommissioning activities
- TMF water quality monitoring until water quality guidelines are met
- Post-closure monitoring will be required after completion of closure activities. Post-closure monitoring is expected to include:
- Water quality sampling at specified discharge locations in accordance with water quality objectives.
- Final environmental effects monitoring studies in accordance with water quality objectives needed to obtain status as a recognized closed mine from Environment Canada.

21 Capital and Operating Costs

21.1 Summary

Total capital and operating costs for the project, provided on an annual basis, are summarized in Table 21-1.

21.1.1 Basis of Estimate

The cost estimate meets the requirements for a PFS, consistent with NI 43-101 reporting requirements. The estimate accuracy range of ± 25% is defined by the level of project definition, the amount of engineering inputs, the time available to prepare the estimate and the amount of project cost data available.

The capital cost estimate was compiled in Canadian dollars (C$) based on 2024 prices and is a combination of first-principles calculations, quotations from suppliers, contractors, equipment vendors, experience, and factored costs. The estimate has been compiled by SRK's QP with inputs from the following consulting companies who also contributed to this report:

- SRK Consulting (Canada) Inc.: Underground mine design, development, production, geotechnical, underground infrastructure
- Nordmin Engineering Ltd.: Madsen #2 Shaft rehabilitation, Madsen #3 Shaft, surface crusher
- Lorax Environmental Services Ltd.: hydrology and hydrogeology
- Knight Piésold Consulting.: Tailings storage, waste rock storage, surface water management and closure costs
- T Engineering: Hydraulic backfill plant and backfill distribution
- Fuse Advisors Inc.: Processing plant
- All North Consultants: surface infrastructure, mine dewatering, UG electrical reticulation
- West Red Lake Gold (WRLG): Owner's costs, existing equipment leases and quotations for line items already in procurement process
- All: Indirect costs, EPCM, contingency

The base date of all estimates is Q2 2024. No allowance has been included in the estimates for escalation beyond this date or for foreign exchange fluctuations.

Pre-production costs are all those costs before the ore is processed through the processing plant that have been assumed as upfront capital costs. Post-production costs are all those capital works after the commencement of ore processing that have been considered sustaining capital costs.

Indirect costs have been factored from the direct costs, using percentages established from industry cost data sources.

Table 21-1: Total project costs

	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033
Operating Costs										
Mining	$388.1	$38.9	$57.7	$55.7	$58.3	$56.2	$58.7	$52.1	$10.5	
Processing	$137.1	$12.5	$22.0	$21.6	$21.4	$22.0	$22.0	$14.1	$1.5	
Water/Waste Management	$1.1	$0.1	$0.1	$0.1	$0.1	$0.1	$0.1	$0.1	$0.1	
G&A	$61.8	$5.6	$9.9	$9.7	$9.7	$9.9	$9.9	$6.4	$0.7	
Total Operating Cost	**$588.1**	**$57.1**	**$89.7**	**$87.1**	**$89.5**	**$88.3**	**$90.8**	**$72.8**	**$12.7**	
Unit Costs										
Mining (C$/t)	**$212.93**	$234.72	$196.90	$194.46	$204.60	$192.00	$200.63	$277.75	$536.19	
Processing (C$/t)	**$75.25**	$75.25	$75.25	$75.25	$75.25	$75.25	$75.25	$75.25	$75.25	
G&A (C$/t)	**$33.90**	$33.90	$33.90	$33.90	$33.90	$33.90	$33.90	$33.90	$33.90	
Capital Costs										
Capital Development	$152.4	$30.3	$26.5	$27.2	$24.4	$22.2	$11.9	$9.8	$0.0	$0.0
UG Mobile Equipment	$54.0	$5.5	$10.8	$11.8	$12.6	$10.0	$1.3	$1.3	$0.7	$0.0
Allocation to UG Capital	$88.5	$15.1	$16.2	$16.2	$13.8	$12.5	$7.0	$7.6	$0.0	$0.0
UG Infrastructure	$123.1	$32.3	$15.7	$30.5	$29.3	$8.6	$5.1	$1.3	$0.2	$0.0
Surface Infrastructure	$18.4	$16.3	$0.4	$0.4	$0.4	$0.4	$0.4	$0.2	$0.0	$0.0
Water/Waste Management	$12.0	$1.5	$0.3	$0.8	$7.8	$0.4	$0.4	$0.4	$0.4	$0.0
Mill Upgrades	$0.1	$0.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Total Contingency	$45.3	$12.9	$6.0	$8.6	$9.7	$4.0	$2.5	$1.5	$0.0	$0.0
Closure Cost	$9.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$3.0	$6.1
Total Capital Cost	**$502.9**	**$114.0**	**$75.9**	**$95.6**	**$98.2**	**$58.2**	**$28.5**	**$22.1**	**$4.3**	**$6.1**

Source: SRK (2025)

21.2 Capital Cost Estimate

LOM project capital costs total C$503M, including closure, and are summarized in Table 21-2.

Table 21-2: Project capital costs

Description	Total Cost (C$M)
Capital Development	$152.4
UG Mobile Equipment	54.0
Allocations to UG Capital	88.5
UG Infrastructure	123.1
Surface Infrastructure	18.4
Processing Capital	0.1
Water/Waste Management Capital	7.2
Contingency	45.3
Closure Costs	9.1
Total	**$502.9**

Source: SRK (2025)

The following items were excluded from the capital cost estimate:

- Sunk costs incurred to date, including studies
- Geotechnical inconsistencies
- Operating costs
- Changes to design criteria
- Work stoppages
- Scope changes or an accelerated schedule
- Changes in national law
- Changes in national duties
- Environmental issues
- Hazardous waste issues

21.2.1 Mining

The cost estimate was compiled in Canadian dollars (C$). The following currency exchange rate was used to develop the costs:

- US$1.00 = C$1.34

The underground mining capital costs were estimated using a combination of first-principles calculations, quotations from the mining contractors and equipment vendors, experience, and factored costs. The costs were developed for an Owner operating scenario with contractor use for vertical development.

The mining capital cost estimate was based on the following:

- Preliminary project development plan
- First principles estimates for capital lateral and vertical development
- 2024 Budget quotes for the major mobile equipment obtained from equipment manufacturers
- Existing lease and rental agreements for existing mobile equipment
- Bid documents from mining contractors and suppliers (e.g., Madsen #2 Shaft reconditioning, surface workshop)
- SRK in-house database

The regional cost information provided by WRLG included the following:

- Labour rates including burden and bonus
- Current costs for most common materials including explosives, ground support, ventilation, pipes and fittings, cement, etc.
- Diesel fuel price of C$1.21/litre for fuel delivered to the mine site
- Electrical power price of C$0.12/kWh as per all-in electricity cost on site

Lateral and vertical development costs were estimated from first principles with vertical development costs based on contractor quotes for raiseboring and Alimak raising. Additional fixed and variable costs were added for company supplied materials, power, underground excavations and mucking.

Mobile equipment costs were developed from estimated fleet requirements and vendor budgetary quotations including lease rates. As a significant portion of the required fleet is already owned, leased or rented by WRLG, these costs were estimated based on the existing agreements.

In agreement with WRLG, the QP has assumed that all major mobile equipment will be financed either through the equipment manufacturer or equivalent lending institution. Any equipment being replaced at the end of its lifecycle was assumed to be purchased.

Equipment life-cycle operating hours were based on manufacturer recommendations. The recommended life-cycle operating hours were used to calculate equipment rebuild and replacement requirements.

21.2.2 Underground Capital Development

The total capital development cost and cost per metre are shown on Table 21-3. These represent the direct mining costs including maintenance of the mobile equipment. All development is expected to be self-performed by WRLG except for the raises requiring either an Alimak raise climber or raisebore machine.

Table 21-3: Capital development cost estimate

Type	Description	Size	Basis	Unit Cost (C$/m)	Total Cost (C$M)
Capital Lateral	Ramp and Level Access	5.0 mW x 5.0 mH	Company	5,150	91.4
	Slashing Track Drifts	5.0 mW x 5.0 mH	Company	3,960	48.9
Capital Vertical	Drop Raise Ventilation	3.0 m x 3.0 m	Company	2,320	2.1
	Raisebore	3.0 m dia.	Contractor	6,940	7.3
	Raisebore	3.65 m dia.	Contractor	7,270	1.0
	Alimak	3.0 m x 3.0 m	Contractor	10,050	1.7
Total Capital Development					**152.4**

Source: SRK (2025)

In addition to these costs, a number of costs have been allocated from the first principles operating cost estimate to the capital costs to account for the shared labour, equipment and services as shown on Table 21-4. No contingency has been applied to these allocations.

Table 21-4: Allocations to capital estimate

Description	Total Cost (C$M)
Allocation of Waste Haul and Hoist to Capital	31.3
Allocation of Maintenance and Mine Services to Capital	8.0
Allocation of Tech Services and Management to Capital	21.0
Allocation of Lease Interest to Capital	2.7
Allocation of Utilities to Capital	25.5
Total	**88.5**

Source: SRK (2025)

These costs include:

- Indirect labour and equipment costs to truck and hoisting the waste generated from capital development
- Indirect labour and equipment for fixed plant maintenance and mine services such as shaft, compressed air, process water, ventilation and mine dewatering

- Indirect labour and equipment for technical services and management such as engineering, geology and direct supervision
- A portion of the lease interest for mobile equipment is allocated back to capital
- A portion of the estimated costs for power and propane for mine air heating

Allocations are mainly based on the ratio of capital waste tonnes divided by total ore plus waste tonnes moved in period. The exception is the UG truck operating cost (excluding labour) for waste haulage which is allocated based on the ratio of capital waste tonnes divided by total waste tonnes moved in period and represent \$5.4M of the \$31.3M.

21.2.3 Underground Mobile Equipment

The total UG mobile equipment costs are shown on Table 21-5. As the mining equipment will be leased, no capital contingency was applied. Lease interest was excluded as this is considered an operating cost.

Table 21-5: Underground mobile equipment cost estimate

Description	**Total Cost (C\$M)**
Leases Existing Fleet	12.7
Leases New Equipment	16.2
Rebuilds and Replacements	25.1
Total	**54.0**

Source: SRK (2025)

21.2.4 Underground Infrastructure

The total UG infrastructure costs are shown on Table 21-6. These are installed costs including direct and indirect costs, excluding contingency. A contingency of \$24.2M was estimated for these items based on level of engineering supporting the cost estimate for each item. Almost \$15M of this is related to the Madsen #2 and #3 shaft estimates.

Table 21-6: Underground infrastructure costs

Description	Quantity	Units	Total Cost (C$M)
Madsen #2 Shaft Reconditioning & 12L Loading Pocket	1	lot	18.7
Madsen #2 Shaft Reconditioning to 24L	1	lot	2.7
Geotechnical Investigations for Madsen #3 Shaft	1	lot	0.5
Madsen #3 Shaft Electrical Upgrades	1	lot	1.4
Madsen #3 Shaft Construction	1	lot	51.2
MineArc (8 person) + EnviroLAV	8	ea	1.8
MineArc (16 person) + EnviroLAV	2	ea	0.5
Cap & Explosives Magazines	3	ea	1.2
Development Storages	12	ea	2.5
Shotcrete/Construction Storage	3	ea	0.5
Definition DDH Stations	35	ea	5.5
Ore/waste Storage	24	ea	-
UG shop w/ wash bay	1	lot	2.2
Fuel/lube Bay	1	lot	0.3
Equipping UG Shop	1	lot	0.3
Equipment Doors	3	ea	0.3
Vent doors/Regulators - simple doors/regulators	22	ea	0.3
Ventilation Fans (S &I)	1	lot	16.2
Monitoring (co, etc.) system	2	lot	0.2
Satstat mobile fuel system	4	ea	0.7
Dewatering System (Pumps & Sumps)	1	lot	2.8
Electrical Reticulation & Mine Power Centers	1	lot	13.5
Total			**$123.1**

Source: SRK (2025)

21.2.5 Surface Infrastructure

The total surface infrastructure costs are shown on Table 21-7. These are installed costs including direct and indirect costs, excluding contingency. A contingency of $4.6M was estimated for these items based on level of engineering supporting the cost estimate for each item.

Table 21-7: Project infrastructure costs

Description	Quantity	Units	Total Cost (C$M)
Hydraulic Backfill Plant Refurbishment	1	lot	0.9
Surface Piping (plant to boreholes)	1	lot	0.4
Underground Distribution System	1	lot	3.9
Surface Crusher Building & Conveyors	1	lot	8.1
Surface Mobile Equipment Shop	1	lot	5.1
Total			**$18.4**

Source: SRK (2025)

21.2.6 Processing

The Madsen plant is a fully functional mill that requires minimal capital investment to allow effective processing of the underground ore. It is expected that East-West bay doors and ADR circuit immersion heaters, costing less than C$85,000 in total, are the only pieces of additional equipment that will be needed prior to commencement of underground operations.

21.2.7 Water/Waste Management

The main capital items associated with project water/waste management are connected to the tailings management facility (TMF) (C$6.4M) and the mine rock management facility (MRMF) ($0.7M). These are direct costs including foundation preparation, road access and water control measures (culverts, underdrains, seepage recovery systems, etc.). Indirects costs are estimated at $4.8M for a total of $12.0M with a contingency on these costs of C$1.8M (17.5%).

21.2.8 Contingency

Total contingency for the project amounts to C$45.3M, equating to 15.06% of the direct costs. This is based on contingency being applied to capital development, slashing, vertical development, surface and underground infrastructure, processing and water/waste management capital (total of C$301.1M). Contingency was not applied to capital allocations or equipment purchase/rebuilds/replacements as equipment is being leased and the lease rates were based on recent (Q2 2024) quotes.

21.2.9 Closure

Closure costs, totaling C$9.1M, were factored from similar sized operations in Canada using a cost of $5.00/t ore. Any realized salvage value of equipment and/or building was assumed to offset demolition and dismantling costs.

21.3 Operating Cost Estimate

Life-of-mine project operating costs total C$588M and are summarized in Table 21-8.

Table 21-8: Project operating costs

Description	Total Cost (C$M)
Mining	$388.1
Processing	137.1
Water/Waste Management	1.1
G&A	61.8
Total	**$588.1**

Source: SRK (2025)

21.3.1 Common Cost Inputs

The regional cost information provided by WRLG included the following:

- Labour rates including burden and bonus
- Current costs for most common materials including explosives, ground support, ventilation, pipes and fittings, cement, etc.
- Diesel fuel price of C$1.21/litre for fuel delivered to the mine site
- Electrical power price of C$0.12/kWh as per all-in electricity cost to site

21.3.2 Mining

The mine operating cost was developed by SRK using a first principles model.

The total operating cost to mine the 1.823 Mt of mineral reserves of C$388.1M is shown in Table 21-9, along with the weighted average cost per tonne and the estimated operating cost per tonne by mining method.

The cost for the attack ramp, TDBs and any internal waste drives are included within the estimated operating cost for the mechanized cut and fill (MCF) and mechanized drift and fill (MDF) mining methods.

Note that the 8 Zone MDF operating cost is similar to the Normal MCF operating cost even with heavier ground support as the 8 Zone has 5.3 times the ore tonnage per cut on average and better continuity.

Table 21-9: Operating costs by cost centre/mining method

Cost Center	Total Cost (C$M)	Weighted Average Cost per Tonne (C$/t)	Cost per Tonne by Mining Method (C$/t)			
			Longhole	Normal MCF	Historic MCF	8 Zone MDF
LH Stoping	28.7	15.74	57.68			
LH Sills	14.6	8.04				
Normal MCF	61.3	33.62		136.49		
Historic MCF	109.0	59.83			206.93	
8 Zone MDF	19.5	10.71				142.21
Material Haulage and Shaft	50.8	27.88	27.88	27.88	27.88	27.88
Mine Services/ Maintenance	34.0	18.66	18.66	18.66	18.66	18.66
Technical Services/Mgmt	28.3	15.52	15.52	15.52	15.52	15.52
Equipment Leasing (Interest)	2.8	1.55	1.55	1.55	1.55	1.55
Utilities	39.0	21.39	21.39	21.39	21.39	21.39
Total Operating Cost	**388.1**	**212.93**	**142.68**	**221.49**	**291.94**	**227.22**

Source: SRK (2025)

21.3.3 Processing

Operating costs for the processing plant on site were estimated based on a review of actual cost data during Pure Gold's operation of the mill and adjusted to reflect the current cost environment and expected consumable usage. Both fixed and variable costs were evaluated, and an average variable cost of $75.25/t ore was determined. This operating cost results in total processing costs of C$137.1M over the life of the mine.

21.3.4 Water/Waste Management

Operating costs for water/waste management on site were estimated based on the expected operation of the TMF and MRMF over the life of the mine. The primary costs associated with water/waste management are pumping costs and maintenance costs. Over the life of the mine, total waste/water management costs are C$1.1M.

21.3.5 General and Administrative

General and administrative (G&A) costs for the Madsen Mine were estimated based on a review of actual cost data during Pure Gold's ownership of the project and adjusted to reflect the current cost and labour environment and expected additions to the project (e.g., camp). While G&A is comprised of both fixed and variable costs, an average variable cost of $33.90/t ore was calculated. This operating cost results in total G&A costs of C$61.8M over the life of the mine.

22 Economic Analysis

22.1 Cautionary Statement

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes Mineral Resource and Mineral Reserve estimates; commodity prices and exchange rate; smelter terms; the proposed mine production plan; projected recovery rates; use of a process method; infrastructure construction costs and schedule; mine capital and operating costs; and assumptions that Project environmental approval and permitting will be forthcoming from local, state and federal authorities.

22.2 General

Financial analysis of the Madsen Mine was carried out using a discounted cash flow (DCF) approach. This method of valuation requires projecting period cash inflows, or revenues, and subtracting period cash outflows such as operating costs, capital costs and taxes. The resulting net annual cash flows are discounted back to the date of valuation and summed to determine the net present value (NPV) of the project at selected discount rates. The DCF model used for this analysis was constructed on a monthly basis and employed an annual discount rate of 5.0% (monthly = 0.41%), which is considered appropriate for a gold project in Canada.

The internal rate of return (IRR) is expressed as the discount rate that yields a zero NPV.

The payback period is the time calculated from the start of production until all initial capital expenditures have been recovered.

This economic analysis includes sensitivities to variation in currency exchange rates, gold price, operating costs and capital costs. A fixed exchange rate between Canadian dollars to United States dollars of C$1.40:US$1.00 was used.

It should be noted that, for the sake of discounting, cash flows are assumed to occur in the middle of the month and are discounted to the start of Q2 2025.

All pricing is stated in constant (real) Q3 2024 Canadian dollars (C$).

SRK prepared the financial model based on the following contributions:

- Resource model prepared by SRK
- Mine design and schedule prepared by SRK and Mining Plus (under SRK supervision)
- Mine operating costs prepared by SRK
- Mine capital costs prepared by SRK with contributions from WRLG, Nordmin Engineering Ltd., T Engineering, Lorax Environmental Services Ltd., Knight Piésold Consulting, Fuse Advisors Inc., and All North Consultants

- Metal pricing provided by WRLG and agreed to by SRK
- Smelter terms and refining costs provided by WRLG and reviewed by SRK
- Process recoveries provided by Fuse Advisors
- Process operating costs and capital costs prepared by Fuse Advisors
- G&A operating costs provided by WRLG and reviewed by SRK
- On-site infrastructure costs prepared by Allnorth and Knight Piésold

22.3 Production Schedule

The mine design and associated production schedule are discussed in detail in Section 16. The production schedule for the Madsen Mine is summarized in Table 22-1 below.

Table 22-1: Madsen Mine production schedule

	Total	2025	2026	2027	2028	2029	2030	2031	2032
Production Rate (tpd)		460	813	796	791	813	813	522	54
Ore Recovered (kt)	1,823	166	293	286	285	293	293	188	20
Head Grade (Au g/t)	8.16	7.00	7.20	7.80	8.00	7.87	7.79	12.29	10.40
Contained Gold (koz)	478.3	37.3	67.8	71.8	73.3	74.1	73.4	74.2	6.5
Recovery (%)	95.7%	94.7%	93.8%	94.3%	95.5%	95.6%	96.5%	98.7%	98.4%
Recovered Gold (koz)	457.9	35.3	63.6	67.7	70.0	70.8	70.8	73.2	6.4

Source: SRK (2025)

22.4 Pricing Assumptions

WRLG provided their recommended gold price to SRK for review. The price forecast is based on a consensus forecast of 27 different analysts from global investment banks as of January 2025. SRK deemed the price forecast to be suitable for the purposes of the economic analysis of the Madsen Mine PFS.

The gold price starts at US$2,615/oz in 2025 and decreases to a long-term gold price of US$2,213/oz starting in 2028. The average gold price over the life of the mine is US$2,340/oz.

22.5 Process Recovery Assumptions

The process recovery is discussed in detail in Section 17 – Recovery Methods. The average recovery over the life of the mine is 95.7%.

22.6 Capital and Operating Costs

The capital and operating costs are reported in Section 21 – Cost Estimate and are summarized below in Table 22-2 and Table 22-3.

Table 22-2: Capital cost estimate

Description	Total Cost (C$M)
Capital Development	152.4
Mobile Equipment	54.0
Underground Infrastructure	123.1
Allocations from Opex to Underground Capital	88.5
Surface Infrastructure	18.4
Processing Capital	0.1
Water/Waste Management Capital	12.0
Contingency	45.3
Closure Costs	9.1
Total	**502.9**

Source: SRK (2025)

Table 22-3: Operating cost estimate

Description	Total Cost (C$M)	Average Unit Cost ($/t)
Mining	388.1	212.93
Processing	137.1	75.25
Water/Waste Management	1.1	0.59
G&A	61.8	33.90
Total	**588.1**	**322.68**

Source: SRK (2025)

22.7 Royalties

A 1% net smelter return (NSR) royalty with Sprott Resource Lending Corp. exists on the project and this has been incorporated into the revenue calculations.

22.8 Taxation

A high-level taxation calculation was made, appropriate for this level of study and specific to the province of Ontario. Depreciation pools and tax loss carryforward balances for WRLG were taken into account, resulting in no taxes being paid during operation.

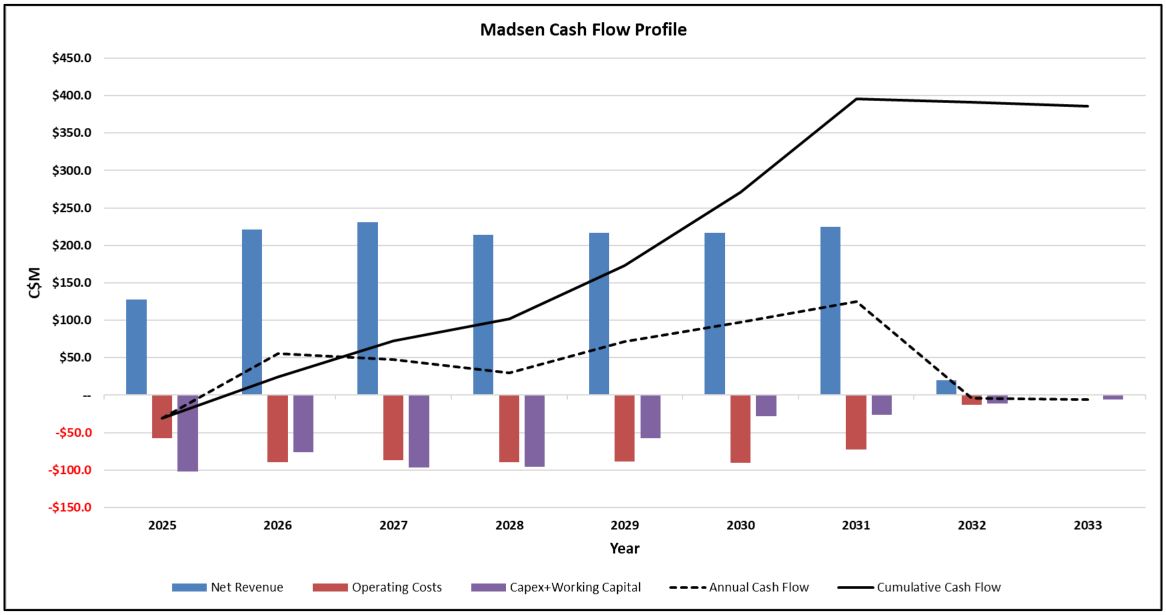
22.9 Off-site Costs

SRK was given guidance from WRLG regarding appropriate levels of refining charges and freight/insurance rates for the Madsen Mine product. These charges and rates are C$0.50/oz and 0.06% of product value, respectively.

22.10 Economic Analysis Results

Annual cashflow for the project is summarized in Figure 22-1 and Table 22-4. The project generates approximately C$71M in annual free cashflow from 2026 to 2031, resulting in an NPV at 5% of C$315M. The IRR associated with this cashflow is 170%. This is primarily due to the fact that capital expenditure projected to return the operation into production was incurred in 2024 and not included in the analysis. The project payback is 1.5 years from the start of production.

Figure 22-1: Project cash flow profile



Source: SRK (2025)

Table 22-4: Annual cash flow (C$000)

	Total	2025	2026	2027	2028	2029	2030	2031	2032	2033
Gross Revenue	$1,480,356	$129,183	$223,910	$233,630	$216,733	$219,435	$219,340	$226,780	$19,931	$0
Off-site Costs	$1,042	$89	$155	$162	$154	$156	$156	$161	$14	$0
Royalty	$14,793	$1,291	$2,238	$2,335	$2,166	$2,193	$2,192	$2,266	$199	$0
Net Revenue	**$1,464,521**	$127,803	$221,518	$231,133	$214,414	$217,087	$216,992	$224,353	$19,718	$0
Operating Costs	$588,089	$57,083	$89,746	$87,103	$89,478	$88,315	$90,843	$72,774	$12,747	$0
Mining	*$388,079*	*$38,872*	*$57,652*	*$55,702*	*$58,262*	*$56,219*	*$58,747*	*$52,146*	*$10,479*	*$0*
Processing	*$137,146*	*$12,462*	*$22,033*	*$21,555*	*$21,428*	*$22,034*	*$22,034*	*$14,128*	*$1,471*	*$0*
Waste/Water Mgmt	*$1,080*	*$135*	*$135*	*$135*	*$135*	*$135*	*$135*	*$135*	*$135*	*$0*
G&A	*$61,784*	*$5,614*	*$9,926*	*$9,711*	*$9,653*	*$9,926*	*$9,926*	*$6,365*	*$663*	*$0*
Operating Cashflow	**$876,433**	$70,720	$131,772	$144,030	$124,935	$128,772	$126,150	$151,579	$6,971	$0
Capital Costs	$499,421	$101,840	$75,907	$96,546	$95,475	$57,563	$28,339	$26,698	$10,984	$6,075
Mining	*$417,953*	*$83,203*	*$69,246*	*$85,737*	*$80,208*	*$53,388*	*$25,291*	*$20,034*	*$845*	*$0*
Plant and Infrastructure	*$18,483*	*$16,385*	*$390*	*$390*	*$390*	*$390*	*$361*	*$176*	*$0*	*$0*
Waste/Water Mgmt	*$11,993*	*$1,535*	*$307*	*$807*	*$7,819*	*$357*	*$357*	*$407*	*$407*	*$0*
Contingency	*$45,337*	*$12,878*	*$5,998*	*$8,635*	*$9,742*	*$4,020*	*$2,530*	*$1,512*	*$23*	*$0*
Closure	*$9,113*	*$0*	*$0*	*$0*	*$0*	*$0*	*$0*	*$0*	*$3,038*	*$6,075*
Change in Working Cap	*-$3,458*	*-$12,161*	*-$34*	*$977*	*-$2,683*	*-$591*	*-$200*	*$4,569*	*$6,672*	*$0*
Pre-tax Cashflow	$377,011	-$31,120	$55,865	$47,484	$29,460	$71,209	$97,811	$124,881	-$4,013	-$6,075
Tax	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Post-tax Cashflow	**$377,011**	-$31,120	$55,865	$47,484	$29,460	$71,209	$97,811	$124,881	-$4,013	-$6,075

Source: SRK (2025)

22.11 Sensitivity Analysis

A sensitivity analysis was performed on the main value drivers associated with the project, namely currency exchange rate (C$:US$), gold price, operating costs and capital costs. The results of this two-factor sensitivity analysis are summarized in Table 22-5.

Table 22-5: NPV sensitivity (C$M)

		Gold Price				
		-20%	-10%	0%	10%	20%
Exchange Rate	1.50	$133	$270	$384	$482	$579
	1.45	$97	$229	$350	$447	$540
	1.40	$61	$188	**$315**	$411	$502
	1.35	$24	$147	$270	$374	$463
	1.30	-$12	$106	$224	$336	$424

		Opex				
		-20%	-10%	0%	10%	20%
Capex	-20%	$459	$423	$386	$348	$306
	-10%	$425	$388	$351	$311	$261
	0%	$391	$353	**$315**	$265	$215
	10%	$356	$318	$270	$220	$169
	20%	$320	$274	$224	$174	$124

Source: SRK (2025)

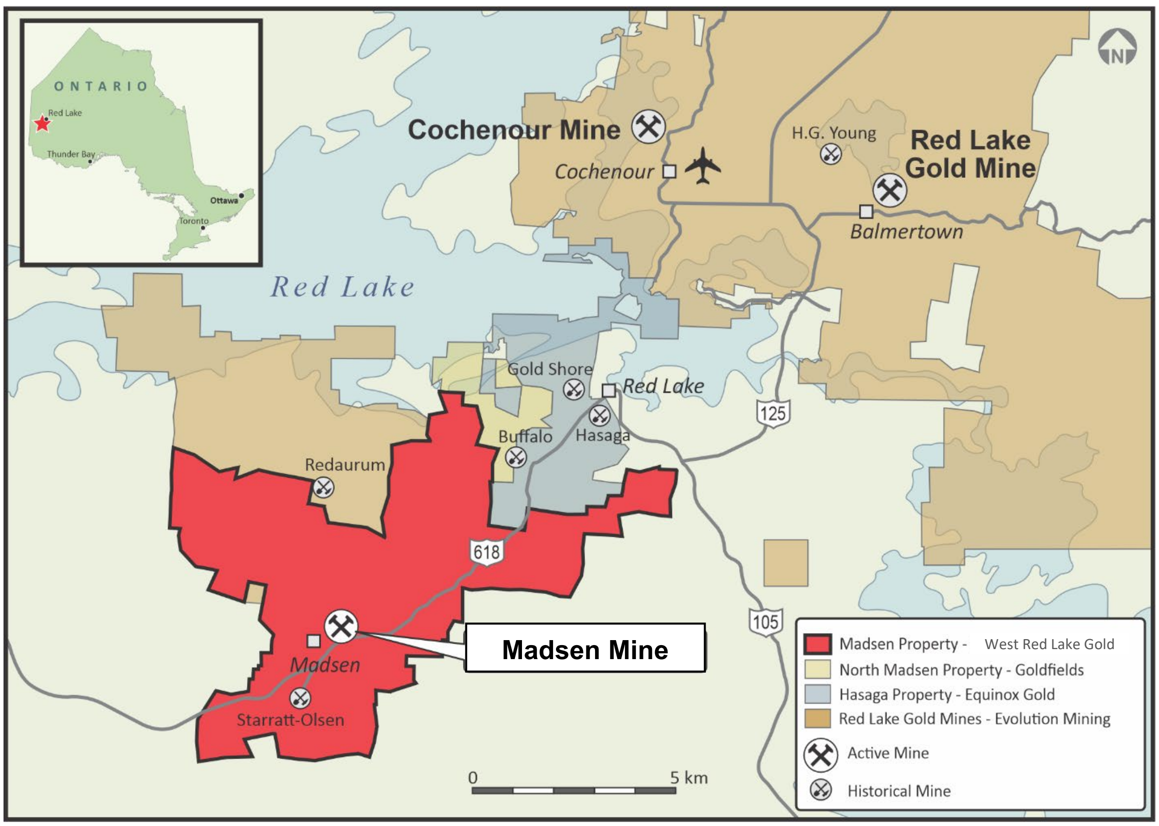
The project is most sensitive to changes in gold price, with every 1% change in gold price affecting project NPV by approximately C$11M, while the project is least sensitive to changes in capital costs, with every 1% change in capex affecting project NPV by approximately C$4M.

23 Adjacent Properties

Relevant information is provided herein for three adjacent and adjoining properties to the Madsen Mine Property – the Hasaga Property of Equinox Gold (Equinox), the North Madsen Property of Agnico Eagle Mines (Agnico), and the Red Lake Gold Mine Property of Evolution Mining (Evolution). These properties are shown on Figure 23-1.

Mr. Cliff Revering has been unable to verify the information provided with respect to the adjacent properties, which was obtained from publicly disclosed documents and such information is not necessarily indicative of the mineralization on the Madsen Mine Property. The proximity and geologic similarities between these adjacent properties and Madsen Mine does not mean that equivalent results will be obtained on the Madsen Mine Property.

Figure 23-1: Location of Madsen Mine and Adjacent Properties



Source: Pure Gold (2022)

23.1 Hasaga Property – Equinox Gold

In 2020 Equinox acquired the Hasaga Property, which is contiguous to the Madsen Mine Property on the northeast boundary (Figure 23-1). The property contains three past producing mines – the Gold Shore, Buffalo and Hasaga Mines. The combined historical gold production of these three historical operations is reported to be 240,970 ounces (Malegus et al., 2022). Active exploration on this property is ongoing. Table 23-1 summarizes the recent mineral resource statement for the Hasaga Property (Equinox, 2024).

Table 23-1: Mineral Resource Estimate, Hasaga Property

Category	Tonnage	Grade (g/t)	Gold Troy Ounces
Indicated Resources	1,470,000	8.64	408,000
Inferred Resources	2,059,000	7.31	484,000

Source: Equinox (2024)

As a result of the Pure Gold acquisition, WRLG holds a 1.0% NSR royalty on the southwestern portion of the Hasaga Property (Buffalo Claims), which was previously held by Pure Gold. The proximity and geologic similarities of Hasaga does not mean that equivalent results will be obtained on the Madsen Mine Property.

23.2 North Madsen Property – Agnico Eagle Mines

Agnico acquired the North Madsen Property from Yamana Gold in 2023. The property is contiguous along the northeast boundary of the Madsen Mine Property. The North Madsen Property has been explored since 1925, however no gold production has occurred. Table 23-2 summarizes the recent MRE of McCracken and Utiger (2014). Most of the resources in all categories are hosted in the Main (41) Zone. The Main Zone mineralization is hosted within the Dome Stock granodiorite and is associated with shear zones and overprinting quartz-tourmaline veins (McCracken and Utiger, 2014). The proximity and geologic similarities of the North Madsen Property does not mean that equivalent results will be obtained on the Madsen Mine Property.

Table 23-2: Mineral Resource Estimate, North Madsen Property

Category	Tonnage	Grade (g/t)	Gold Ounces
Measured Resources	16,728,310	1.3	685,891
Indicated Resources	6,230,600	1.0	202,862
Measured and Indicated Resources	22,958,910	1.2	888,752
Inferred Resources	10,138,000	1.2	383,936

Source: McCracken and Utiger (2014)

23.3 Red Lake Gold Mine Complex – Evolution Mining

Evolution's Red Lake Gold Mine complex is contiguous to the Madsen Mine Property on the northern boundary. The Madsen Mine and the Red Lake mine complex are approximately 16 km apart. The Red Lake Gold Mine is the largest mining operation in the Red Lake mining district and has been in continuous operation since 1948. Evolution acquired the property in 2020 and has expanded its total tenement package to 710 km^2. In FY2024, the mine had approximately 610 employees and 340 contractors. Mines on what is now the Red Lake Gold Mine Property have produced approximately 25.5 Moz of gold to 2024, including gold production of 112,700 oz in

FY2024. Active exploration is ongoing across the property. Table 23-3 provides Reserve and Resource estimates for the Red Lake operation as disclosed by Evolution in 2025.

Table 23-3: Mineral Resource Estimate (Dec 2023), Red Lake Gold Mine

Category	Tonnage	Grade (g/t)	Gold Ounces
Probable Reserves	12,400,000	6.87	2,748,000
Indicated Resources	32,400,000	6.89	7,174,000
Measured and Indicated Resources	32,400,000	6.89	7,174,000
Inferred Resources	22,700,000	6.10	4,456,000

Source: Evolution (2025)

24 Other Relevant Data

Additional conceptual design optimization was conducted to determine the sensitivity of mine design to changes in gold price, compared to the base case price of US$1,680/oz that was used for the PFS LOM Plan.

The reader is cautioned that the information contained in this section is conceptual in nature and economic viability has not been demonstrated for anything other than the PFS LOM Plan presented in the previous sections.

24.1 Gold Price Sensitivity – Excluding East Zone

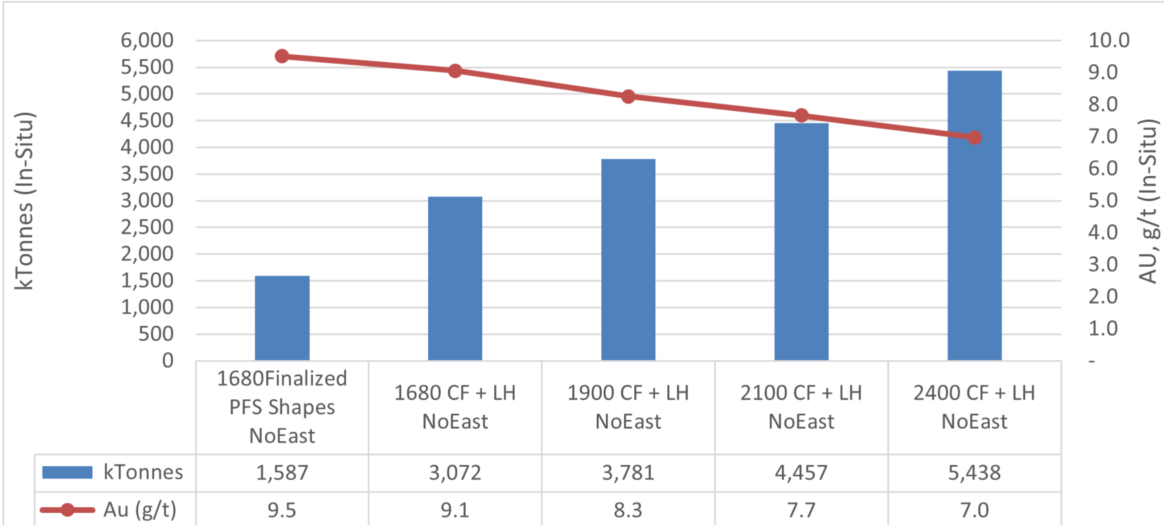
To further understand how the orebody behaved due to changing cut-off grades at various gold prices, Deswik Stope Optimizer (DSO) was used to run additional scenarios at gold prices of US$1,900/oz, US$2,100/oz, and US$2,400/oz.

It was not practical to redo all the steps to convert these DSO results into a final mine plan for each gold price, so conversion factors were developed based on the DSO stope shapes generated with the original US$1,680/oz gold price scenario that were converted to the finalized stope shapes. The finalized stopes shapes are not the same as the Mineral Reserve shapes as development is not accounted for, nor have the modifying factors been applied.

This exercise was conducted to better understand how much additional material might reasonably be expected. The conversion factors for tonnes and grade were determined to be 52% and 105% for material across the entire mine, excluding the East Zone (Derlak). The East Zone was excluded as little of the mineral resources converted to Mineral Reserves due to the lower grade and the fact that the East Zone is at least 500 m from any other mining zone making most of the zone uneconomic using the current assumptions (see Section 24.2).

Figure 24-1 shows the potential upside in terms of tonnes and grades before accounting for development and modifying factors as described in Section 15. The conversion factors indicated above would convert the 2nd column into the 1st column, which is the basis of the Mineral Reserve after accounting for development and modifying factors.

Figure 24-1: Sensitivity to gold price (excluding East Zone)



Source: Mining Plus (2025)

The resulting change in potential tonnes and grade is shown in Table 24-1, including the gold price, updated break-even cut-off grade (BECOG) and potential changes to tonnes and grade.

Table 24-1: Sensitivity to gold price (excluding East Zone)

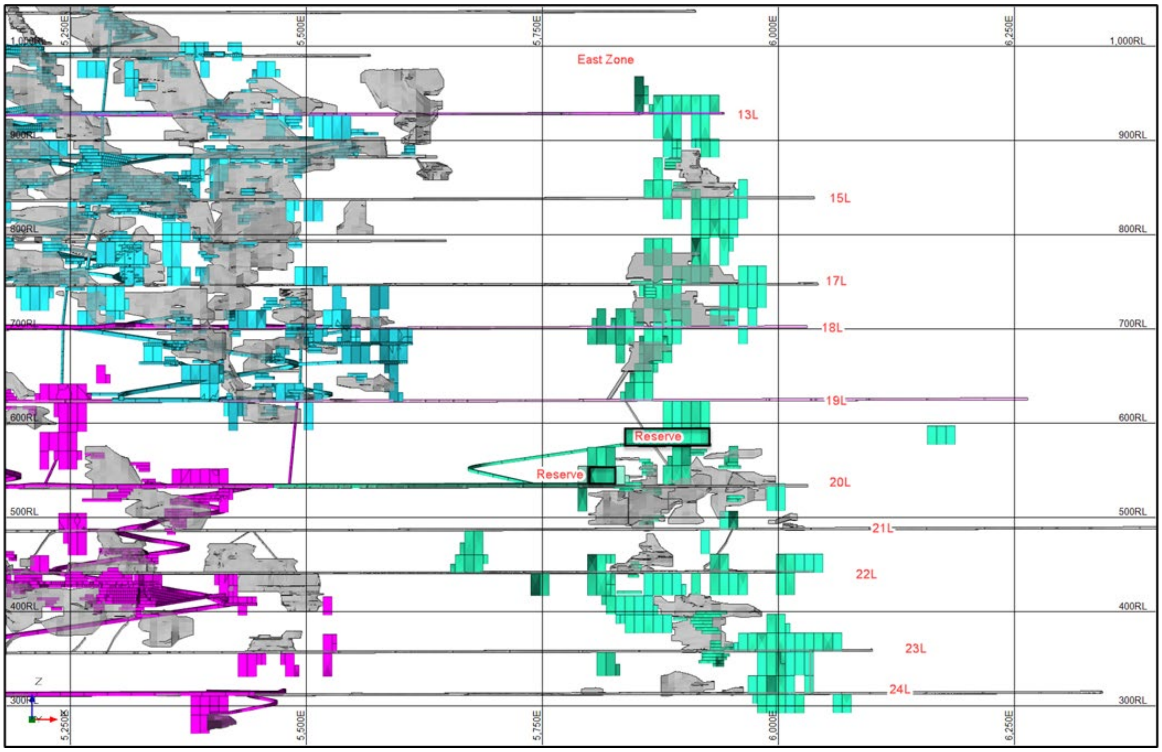
	Gold Price	BECOG (gpt Au)		Percent Change	
Description	**(US$/oz)**	**LH**	**MCF**	**Tonnes**	**Grade**
Base Case	$ 1,680	4.30	5.28	0%	0%
Upside Case #1	$ 1,900	3.80	4.67	123%	91%
Upside Case #2	$ 2,100	3.44	4.22	145%	85%
Upside Case #3	$ 2,400	3.01	3.70	177%	77%

Source: Mining Plus (2025)

24.2 Gold Price Sensitivity – East Zone

A more detailed sensitivity analysis was performed for the East Zone. Figure 24-2 shows the DSO shapes generated using the US$1,680/oz gold price. Note that only two areas above 20 Level passed the economic checks and are included in the Mineral Reserve. The zone is generally lower grade than the core mining areas and there is a 500 m gap that considerably increases capital costs to access the zone.

Figure 24-2: Long section showing the East Zone DSO results (green) at US$1,680/oz



Source: Mining Plus (2025)

The analysis was conducted by using the same economic test used for the mineral reserve process, which accounts for capital and operating development costs, mining costs, processing costs, G&A costs, modifying factors and mill recoveries. The gold price was flexed to determine when each level became economic.

Table 24-2 shows the gold price, number of economic levels and percent change relative to the base case (36 kt at 5.57 gpt). Potential tonnage increases to 414 kt at 5.0 gpt at a US$2,500/oz gold price. As no additional DSO runs were made, these values do not reflect the potential impact of changing the BECOG.

Table 24-2: Sensitivity to gold price (East Zone)

	Gold Price	Economic	Percent Change	
Description	**(US$/oz)**	**Levels**	**Tonnes**	**Grade**
Base Case	$ 1,680	1	0	0
Upside Case #1	$ 1,900	5	377%	99%
Upside Case #2	$ 2,100	10	675%	93%
Upside Case #3	$ 2,500	16	1140%	90%

Source: SRK (2025)

25 Interpretation and Conclusions

25.1 Property Description, Location and Access

The Property is centered at 50.97° North latitude and 93.91° West longitude (UTM Projection NAD83, Zone 15 North coordinates 5646000N, 435000E) within the Baird, Heyson and Dome Townships of the Red Lake Mining District in northwestern Ontario, Canada.

The Madsen Mine is located adjacent to the community of Madsen, within the Red Lake Municipality of northwestern Ontario, approximately 565 km by road (430 km direct) northwest of Thunder Bay and approximately 475 km by road (260 km direct) east-northeast of Winnipeg, Manitoba. Red Lake can be reached via Highway 105 from the Trans-Canada Highway 17. Red Lake is also serviced with daily flights from Thunder Bay and from Winnipeg by Bearskin Airlines.

The mine is accessible from Red Lake via Highway 618, a paved secondary road maintained year-round by the Ontario Ministry of Transportation. The mine is 10 km southwest of the town of Red Lake. A series of intermittently maintained logging roads and winter trails branching from Highway 618 provide further access to other portions of the Property.

25.2 Mineral Tenure and Surface Rights

The Madsen Mine property comprises a contiguous group of 241 tenures, covering an aggregate area of 4,648 hectares in northwestern Ontario. WRLG owns 100% of all mining leases, patents and unpatented claims comprising the mine property. None of the royalties described in this report apply to tenure covering the current mineral reserve.

WRLG owns surface rights in the form of mine property claims, patents and leases. Where WRLG does not hold surface rights, they 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. A single trapping tenure is held over the entire property and WRLG maintains good relations with the tenure holder. Several registered easements for highway and utility lines cross the property.

25.3 History

Gold was originally reported in the Red Lake area in 1897 by R.J. Gilbert of the North West Development Company. Intensive exploration of the district followed discovery in 1925 of the gold showings that eventually formed part of the Howey Mine (Lebourdaix, 1957).

Since 1927, a total of 28 mines have operated in the Red Lake Mining District, producing 29 Moz of gold at an average recovered grade of 15.6 g/t Au. Approximately 89% of this gold was produced from two mine complexes: Red Lake Mine and Madsen Mine (Malegus et al., 2022).

25.4 Geological Setting and Mineralization

The Madsen Mine is located within the Western portion of the Archean Superior Province of the Canadian Shield. It 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 is characterized by Mesoarchean and Neoarchean volcanic and plutonic rocks interpreted to have been emplaced within rift and arc-related environments on the continental margin of the Mesoarchean crustal rocks of the North Caribou terrane. The predominantly sedimentary rocks of the English River belt are believed to have accumulated within a synorogenic flysch basin that formed during assembly of the North Caribou terrane with the Winnipeg River terrane to the south during the Uchian Orogeny, ca. 2720-2700 Ma (Percival et al. 2006).

The mine property is underlain by Balmer, Confederation and Huston Assemblage supracrustal rocks. These older rocks are cut by a series of plutonic rocks (post-tectonic Killala-Baird batholith to the west and syn-kinematic Dome and Faulkenham Lake Stocks to the east) and associated smaller sills and dykes.

Most of the historical gold production and most of the current mineral resources at the Madsen Mine are within the Austin, South Austin and McVeigh zones which, along with the 8 Zone, comprise the Madsen deposit. At the scale of the property, these zones all lie within much broader, kilometre-scale planar alteration and deformation corridors that have been repeatedly reactivated during gold mineralization and subsequent deformation and metamorphism. The distribution of gold within these planar structures is almost exclusively within variably altered basalt, and enhanced in close proximity to major lithological contacts, such as ultramafic sills, felsic dykes and felsic volcanic strata.

Controls on mineralization at the Madsen Mine are consistent with a typical orogenic gold system. Many deposit-scale features such as control by lithological/structural contacts and association with felsic dykes are typical in these systems. Recent work indicates that, apart from its early timing of emplacement prior to the dominant regional deformation and metamorphism, the Madsen Mine shares many characteristics with typical orogenic gold deposits, including the Red Lake Mine deposit.

25.5 Exploration, Drilling and Exploration Potential

Since acquiring the Madsen Property in 2023, WRLG has conducted geological mapping, surface rock sampling and glacial till geochemical sampling, all of which were undertaken as part of the 2024 exploration program. The Madsen Mine surface (non-drilling) exploration dataset comprises systematic, property-wide, multifaceted information carefully collected using modern techniques. Combining surface geophysical (magnetic and seismic), geochemical and geological information with historical data and drilling data has allowed for a property-wide geologic map that has formed an important input for sub-surface 3D geologic interpretation supported by the drilling dataset. Delineation of several new surface targets has resulted from compilation of the surface data sets. The surface dataset continues to be refined and informed by infill geological mapping supported by mechanical stripping and by diamond drilling. In the current state it forms a valuable base for geologic interpretation and extrapolation in support of exploration.

The Property has a long history of diamond drilling, dating from initial discovery of the Madsen deposit in the 1930s through until the present day. Documentation of procedures and methods of drilling is sparse prior to the 1990s. All historical exploration and production drill testing on the mine property to date has been by diamond drill coring. Underground drilling from 1937 to 1999 at Madsen Mine employed whole core sampling and most core intervals were sampled for fire assay gold analysis at the on-site mine laboratory. Pure Gold drilled a total of 2,411 diamond drill holes for 399,661 m between 2014 and 2022. These totals include both exploration drilling outside the footprint of the Madsen Mine and definition drilling to support mining operations. Since acquiring the Madsen project in June 2023 and up to May 15, 2024, WRLG completed a total of 146 holes for 11,849 m of BQ diamond drill core (definition) and 59 holes for 8,024 m of NQ diamond drill core (expansion) from underground. Definition drilling was focused on the Austin and South Austin zones to increase geologic confidence in these areas to a level appropriate for mine development planning. Expansion drilling was focused primarily within the newly defined North Austin zone outside of the existing life-of-mine mineral resource domains, but still in close proximity to existing underground infrastructure. Underground drilling in 2023 and 2024 was completed by Boart Longyear.

Diamond (core) drilling is the most appropriate test method for the mine and this technique has been applied by all operators since early exploration and mining. Historical drilling is tightly-spaced (nominally drilled at 6 m centres) within mined-out areas but other largely non-mined areas show evidence of alteration and elevated gold and have been drilled at much broader spacing.

Exploration for gold on the mine property focuses on identifying the planar structures (or shear zones) that were active during gold deposition. Since gold is very heterogeneously distributed within these structures, assessing targets using gold assay data alone will not yield reliable results. The ground in and around the Madsen Mine has high prospectivity for gold and exploration potential exists both within the Madsen deposit and in the adjacent areas (e.g., Russet, Wedge, Fork, Starratt, Gap, Derlak exploration targets).

25.6 Sample Preparation, Analysis and Data Verification

Sampling procedures and methods have evolved significantly over the long history of exploration and mining at the Madsen Mine and specific procedures also varied among operators. The QP is of the opinion that, based on historical information available, the historical 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 undertaken up to WRLG’s acquisition of the property meet or exceed modern best practices. The historical procedures and those undertaken by WRLG are adequate for modern targeting, modelling and resource estimation.

Owing to the long history of exploration and production at the mine, there have been numerous campaigns of data verification, validation and reconciliation. The most comprehensive recorded verification effort (Cole et al., 2010) was conducted during the digitization of the mining-era hardcopy drill logs, prior to Pure Gold's acquisition of the property. This work was initiated by Claude in 1998, advanced by Placer Dome from 2002 to 2006 and completed by Claude with assistance from SRK during 2008 and 2009. The result was a modern digital database comprising 13,617 historical drill holes with lithological intervals and 550,687 gold assays. This database was the foundation for drill-targeting, geological interpretation and mining by Pure Gold and has been substantially added to and verified since 2014.

The Madsen Mine drilling database is compiled from historical and modern work that spans over 80 years. Available historical hard-copy records were collected and transferred into a modern digital database (Cole et al., 2010). Use and verification of this database shows that it is of high quality, largely free of errors and highly effective, even if assessment of the original data collection methods is not possible. Work by Pure Gold, and subsequently by WRLG, has been conducted with clear data handling protocols and an industry-standard quality control program.

25.7 Mineral Resource Estimate

The current, previously disclosed MRE for the Madsen Mine (Table 25-1) was generated by Mr. Cliff Revering, P.Eng., of SRK Consulting (Canada) Inc., with an effective date of December 31, 2021. The estimate includes Indicated mineral resources of 1,653,000 oz of gold (6.9 Mt at an average grade of 7.4 g/t) and Inferred mineral resources of 366,200 oz of gold (1.82 Mt at an average grade of 6.3 g/t). These mineral resources are reported at a cut-off grade of 3.38 g/t, using a gold price of US$1,800 per ounce, and are constrained by reasonable stope shapes within the Madsen deposit.

This MRE is based on verified historical drilling data, along with additional drilling data and underground mine development and production data collected by Pure Gold between 2014 and 2022. This MRE is also predicated on a revised geological and mineralization domain model developed in 2021 that incorporates structural controls on mineralization identified through data analysis, grade control programs and mapping of underground exposures by Pure Gold since 2018.

Since the effective date of this MRE, additional diamond drilling was conducted. A total of 688 drill holes and 54,122 m of drilling was completed in 2022. An additional 205 drill holes and 19,872 m of drilling was completed by WRLG between October 1, 2023 and May 15, 2024. Based on a review of the results of this drilling it has been determined by Mr. Cliff Revering, QP for the Madsen MRE, that the information obtained will not have a material impact on the MRE presented in this report.

Table 25-1: Mineral Resource Estimate, PureGold (Madsen) Mine, Red Lake, Ontario (effective date December 31, 2021)

Classification	Deposit – Zone	Tonnes	Gold Grade (g/t)	Total Gold (troy oz)
Indicated	Madsen – Austin	4,147,000	6.9	914,200
	Madsen – South Austin	1,696,000	8.7	474,600
	Madsen – McVeigh	388,700	6.4	79,800
	Madsen – 8 Zone	152,000	18	87,700
	Fork	123,800	5.3	20,900
	Russet	88,700	6.9	19,700
	Wedge	313,700	5.6	56,100
	Total Indicated	**6,909,900**	**7.4**	**1,653,000**
Inferred	Madsen – Austin	504,800	6.5	104,900
	Madsen – South Austin	114,100	8.7	31,800
	Madsen – McVeigh	64,600	6.9	14,300
	Madsen – 8 Zone	38,700	14.6	18,200
	Fork	298,200	5.2	49,500
	Russet	367,800	5.8	68,800
	Wedge	431,100	5.7	78,700
	Total Inferred	**1,819,300**	**6.3**	**366,200**

Notes:

1) Mineral Resources estimated in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Cliff Revering, P.Eng., Qualified Person.
2) Mineral resources are not mineral reserves and do not have demonstrated economic viability.
3) Mineral resources are reported at a cut-off grade of 3.38 g/t Au
4) Mineral resources are reported using a gold price of US$1,800/oz
5) Excludes depletion of mining activity during the period from January 1, 2022 to the mine closure on October 24, 2022 as it has been deemed immaterial and not relevant for the purpose of this report.
6) All figures have been rounded to reflect the relative accuracy of the estimate

25.8 Mineral Reserve Statement

The Madsen Mine has been mined extensively from the mid-1930s to the mid-1970s with more than 8.9 Mt of ore being extracted. Much of the higher grade material in the mineral resource model is remnants contained in sill pillars and/or immediately adjacent to the historic shrinkage stopes. The mineral reserves (Table 25-2) are contained within a mining area with a strike length of 1,250 m with a 1,200 m vertical extent with a 60° plunge to the SSE. The mineral reserves follow the trend of the historic shrinkage stopes. The strike length of the historic development is 2,000 m with a 1,300 m vertical extent. This presents unique challenges and opportunities for modern mining operations using trackless, mechanized equipment.

Table 25-2: Mineral Reserve Statement, Madsen Mine, Red Lake, Ontario (effective date June 30, 2024)

Classification	Deposit - Zone	Tonnes (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Probable	Madsen - Austin	778	7.37	184
	Madsen - South Austin	861	8.21	227
	Madsen - McVeigh	66	7.37	16
	Madsen - 8 Zone	118	13.38	51
Proven + Probable		**1,823**	**8.16**	**478**

Notes:

1) Mineral Reserves estimated in accordance with CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM, November 29, 2019 by Stephen Taylor, P.Eng., Qualified Person.
2) Longhole stope cut-off grade of 4.30 gpt Au based on an estimated operating cost of C$287.34/t including mining, plant and G&A. The mining cost component was benchmarked based on an operating mine in Ontario.
3) Mechanized Cut and Fill stope cut-off grade of 5.28 gpt Au based on an estimated operating cost of C$354.90/t including mining, plant and G&A.
4) Mineral reserve estimates based on a gold price of US$1680/oz and an exchange rate of 1.31 C$/US$.
5) Incremental development cut-off grade of 1 gpt Au.
6) A small amount of incremental longhole tonnes were included at a cut-off grade of not less than 3.4 gpt Au, these must be immediately adjacent to economic stopes that will pay for the capital to access area.

The Mineral Reserve Statement presented herein has been prepared for public disclosure according to CIM Best Practice Guidelines (November 2019) and reported as diluted tonnes delivered to the mill. As there are no Measured mineral resources included in the 2022 mineral resource model upon which the PFS mine design is based, there are no Proven mineral reserves included in the mineral reserve estimate.

25.9 Mine Plan, Development and Production

The mine plan for the Madsen Mine is based on the resource model completed by SRK. At present, the Madsen Mine has historic workings covering a 2.3 km strike length to 1,300 m depth.

A significant portion of the higher-grade mineral resources is located in close proximity to the historic workings and can be considered remnant mining targets. These include mineral resources left in place as pillars, not considered mineable at the time, below the cut-off grade at the time, or simply not recognized as ore at the time. Using modern MCF mining methods and ground support techniques, a portion of these remnants can be safely extracted today. There are also mineral resources in unmined areas, though these tend to be lower grade than the core zones extracted historically.

A number of underground mining methods were considered to deal with different challenges encountered in the various zones, but the mining methods selected for ore extraction at Madsen were narrowed down to:

- Longitudinal Retreat Longhole with ramp access (LH): 32.7% of production
- Mechanized Cut and Fill (MCF) with ramp access: 60.6% of production (normal and historic)
- Mechanized Drift and Fill (MDF) with ramp access for 8 Zone: 6.5% of production

Mining method selection was driven primarily by mineralization geometry and continuity, selectivity of method, ability to mechanize the method, proximity to historic workings and anticipated ground conditions. For design purposes, LH was the preferred method of extraction followed by MCF.

Current access to the mine is through either the East ramp or West ramp systems. The operation is currently developing a connection between these two ramp systems to improve public safety and haulage efficiency. The connection is expected to be completed in Q1 2025. This connection will eliminate the need for mine vehicles to cross the two municipal roads leading to the town of Madsen when using the West ramp portal. These ramp systems are suitable for modern 40-t class haul trucks.

A project is also ongoing to recondition the Madsen Shaft #2 and install a new loading pocket at 12 Level to facilitate hoisting operations and reduce trucking requirements as the active mining areas get deeper. This work is being undertaken by a mining contractor and is expected to be completed in Q3 2025. This shaft is a rectangular timber shaft with small skips and is also acting as the main ventilation intake.

Due to the hoisting capacity limitations of Madsen Shaft #2 to hoist the required ore and waste tonnages from below 12 Level, a new Madsen Shaft #3 has been proposed with construction expected to be completed Q4 2028. This shaft will also help support the increased ventilation flows required by modern diesel equipment. The Madsen Shaft #2 is planned to be reconditioned down to 24 Level to provide cage access to provide emergency egress from the mine, either in the cage or ladderway.

Access for personnel and materials will mainly be via the East portal and ramp systems for the life of mine with some movement via the shafts. Once commissioned, the Madsen Shaft #2 will be used to hoist ore and waste to surface until such time as Madsen Shaft #3 is completed.

Ore will be trucked from the work areas to the closest available dump point:

- Surface
- 10 Level grizzly for Madsen Shaft #2
- 18 Level grizzly for Madsen Shaft #3

Access drifts, attack ramps and TDBs will be 3.0 mW x 3.5 mH (TDB average = 1.75 mH), LH sill drifts will be 4.0 mW x 3.5 mH and MCF/MDF stoping will be variable width by 3.5 mH.

As the mine was previously in commercial production under Pure Gold, the mine is expected to ramp up to 800 tpd and achieve commercial production in Q1 2026. WRLG has been executing some pre-production work and test mining to prepare the mine for production, including developing the connection drift, continuing dewatering and beginning rehabilitation of Madsen Shaft #2.

Until the loading pocket on Shaft #2 is operational, all ore will be trucked to surface via ramp. Once the loading pocket is ready, ore will be trucked to the 10 Level grizzly or surface depending on stope location. The proposed Madsen Shaft #3 is expected to be operational by the end of 2028 as mining progresses deeper. Ore will then be trucked to either the 18 Level grizzly for Shaft #3 or the 10 Level grizzly for Shaft #2 depending on stope location. As mining progresses, more of the ore will be hoisted as the near surface stopes are depleted.

Mine sequencing is generally top down by mining area with stope sequence being bottom up within the mining areas.

The overall ventilation strategy for the Madsen Mine will provide control over the fresh air supply and routings, with uncontrolled or free exhaust routings to surface through open stopes, intermediate ore/waste passes, fringe or perimeter raises and decline accesses. The ventilation system will be developed or driven by two exhaust fans installed in the ramp accesses near the surface, and the fresh air will be provisioned by the existing Madsen Shaft #2 and the future Madsen Shaft #3.

The Madsen Mine is expected to utilize an underground workforce averaging 221 people over the LOM Plan. There are currently approximately 140 workers employed, of which approximately 60% live locally. The operation is actively recruiting with a 114-person camp and a new mine dry under construction to accommodate the additional people. Peak labour requirements occur in Q4 2025 as the mine reaches full production. Steady state is achieved in 2026 through 2028, with reductions in the work force starting in 2029 as lateral development requirements taper off.

25.10 Metallurgical Testing, Mineral Processing and Recovery Methods

Historical metallurgical data is available from mill operations dating back to the 1951 Madsen Lake Gold Mines Limited annual report. Gold recovery percentages in the mid-90s were reported at the time. The mill operated for over 40 years with mill throughput ranging from 350 t/d to 850 t/d. In later years, recoveries in the mid-90s continued to be achieved. The present mill was purchased and relocated in the 1990s from Placer Dome’s Dona Lake mine. The mill operated at a nominal rate of 600 t/d and used the carbon-in-pulp (CIP) process to recover gold. A 1998 mill report indicated average annual recovery of 90% at an average gold head grade of 4.2 g/t (Madsen Gold Corp., 1998). The most recent test program, completed in 2018 in support of the 2019 Feasibility Study completed by JDS for Pure Gold, was carried out at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC. A full breakdown of the results for the test program can be found in the BaseMet 2018 report.

Based on the results from Base Met (2018) and historical operational data, gold doré with no significant levels of deleterious elements can be produced with a primary grind size of 80% passing (P80) 75 µm followed by gravity concentration, 2-hour pre-oxidation, 24-hour cyanide leach, 5-hour carbon-in-pulp (CIP) adsorption/desorption and refining. Using the blended average recovery of the samples tested, based on the 2024 SRK mine plan, it is estimated a LOM gold recovery of 95.7% can be achieved.

After cyanide destruction, the CIP tailings will be pumped to the tailings management facility initially. Starting later in Year 1, a thickened tailings will be pumped into open stopes using a hydraulic backfill system.

25.11 Project Infrastructure

The Madsen Mine is a mature site with an existing underground mine, mineral processing facilities, a shaft, two portals (East and West), a water treatment plant, a tailings area, a rock dump and a general services area. Dewatering of Madsen Shaft #2 has been maintained since WRLG's acquisition in 2023. As part of the mine restart plan, WRLG will be adding surface facilities such as a new mine dry, a mobile workshop, and a mobile crushing unit that are required to allow the operation to restart.

The Madsen Mine is connected to the northwest Ontario power network by aerial distribution power lines. The incoming voltage to the site is from a 44 kV circuit with a 12 MW power supply. The northwest Ontario power transmission network is owned and operated by Hydro One. Red Lake is located at the end of the 115 kV transmission line coming from Ear Falls, Ontario.

The Madsen Mine has an underground leaky feeder system for communications, which will be expanded as required into the new working areas.

The Madsen Mine is being dewatered through the Shaft #2 pumping station. The pump stations house Mackley 7-stage pumps equipped with 250 kW/4160 V motors. The principal sumps are located at Shaft #2 and spaced out at six-level intervals. The average level interval is 45 m.

Approximately 2.6 Mt of mine rock material will be produced. Mine rock from underground development will be managed in the underground mine as backfill (42%) and stored (58%) in the existing MRMF, located adjacent to the TMF. Multiple void historic stopes create good opportunity to store waste rock underground, thereby reducing costs. Mine rock required for additional underground backfill and construction activities will be sourced from this stockpile.

Tailings will be managed through a combination of surface storage in the TMF and underground deposition as hydraulic backfill. The TMF has been in operation since the late 1930s and has gone through several design modifications. The TMF at the Madsen Mine is permitted to discharge tailings and will be expanded to manage a total of 1.5 Mt. The TMF is partitioned into two designated areas, Cell A and the Main TMF. Containment for the first four years of tailings deposition will be provided in Cell A, with the remainder of the tailings managed in the Main TMF. Cell A is fully constructed and ready for operation, including a 4-m dam raise that was completed in summer 2024.

The Madsen Mine will be a shaft-based operation that will utilize the existing Madsen Shaft #2 installation for production in the first three to four years, moving to a new Shaft #3 facility to support ongoing future operations. The existing Madsen Shaft #2 system was constructed in the late 1950s, with completion of the sinking effort in 1958. The finished shaft is a five-compartment rectangular shaft that was designed to accommodate both production and service hoisting with a central manway. Shaft #2 will be used for muck hoisting and as a secondary means of emergency egress. As operational efforts progress, the centroid of the mining operation will move to the northeast more than 1.5 km away from Shaft #2 and to an ultimate depth of 4,000 feet or more. As such, and with the identified need to develop a large ventilation raise to allow for improved capacity in these areas, it was decided to equip this raise for use as a combined production and service shaft. The new Shaft #3 system would be constructed in Year 3 of the project and will be developed via raisebore. The shaft will be developed in two sections, with the first from 10 Level to surface, and the second from 18 Level to 10 Level, allowing for the surface plant to be constructed and shaft equipping to be undertaken while the second leg is in development. A rock pentice would separate the two legs, then excavated once the lower section of the shaft is completed, allowing for the completion of ground control and shaft furnishing to shaft bottom.

25.12 Environmental Studies, Permitting, Social/Community Impact

WRLG is continuing its scientific and engineering studies at the site; consultation with regulators, First Nations and communities; monitoring programs; and detailed project design planning to reopen the mine and processing facility. 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 resource use, improve water quality and bring overall benefit to local communities and First Nations.

WRLG has maintained the permits that existed for the Madsen Mine under previous operators. As the project has advanced, operational enhancements and regulatory changes have required some permit updates. Permit status has been confirmed with both the Ministry of Mines and Ministry of the Environment, Conservation and Parks (MECP) and the site's permits and authorizations are in good standing.

The mine restart requires neither provincial nor federal Environmental Assessments.

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

25.13 Cost Estimate

LOM capital costs total C$502.9M, including C$45.3M in contingency and C$9.1M related to closure costs. Project operating costs total C$588.1M. The cost estimates were prepared based on pricing information obtained in 2024 and using the SRK LOM production plan.

25.14 Economic Analysis

The project generates approximately C$71M in annual free cashflow from 2026 to 2031, resulting in an NPV at 5% of C$315M. The IRR associated with this cashflow is 170%. This is primarily due to the fact that capital expenditures projected to return the operation into production were incurred in 2024 and not included in the analysis. The project payback is 1.5 years from the start of production.

The project is most sensitive to changes in gold price, with every 1% change in gold price affecting project NPV by approximately C$11M, while the project is least sensitive to changes in capital costs, with every 1% change in capex affecting project NPV by approximately C$4M.

25.15 Risks and Opportunities

25.15.1 Risks

Mining Risks

Beyond the typical risks associated with underground hardrock mining, the Madsen Mine is exposed to a number of other mining risks.

At the time of writing the mine is flooded between 14 Level and 15 Level, so the conditions of the historic excavations below 14 Level have not been inspected. This presents the following risks:

- There is a risk that sections of the historic track drifts will have collapsed and more extensive reconditioning and/or additional bypasses will be required beyond what was included in the mine plan. The mine plan assumes all historic track drifts to be utilized will be slashed to 5.0 mH x 5.0 mW and fully reconditioned and that historic stoping areas are by-passed. Any areas not utilized should be barricaded.
- There is a risk that during dewatering, the historic backfill materials will be transported out of the stopes by liquefaction and into the track drifts. The dewatering plan allows for a slow steady dewatering rate to allow the backfill materials to drain in an attempt to minimize the amount of material mobilized. The PFS mine plan also places new ramps and access drifts away from the areas that are most likely to be impacted.
- There is a risk of impounded water being trapped in various places. These will have to be identified during inspections of newly accessible areas and dealt with as they occur.

Most of the historic plans and sections showing the stoping and development details are no longer available, therefore the 3D as-built models have been built from available data and lack the finer details except in the more recently active mining areas, which present the following risks:

- There is a risk that not all of the drifts are represented in the 3D as-built models, this may lead to unexpected breakthroughs. These may sometimes be identified by inspection, diamond drill intersections and/or test holing if the existence of such is suspected and dealt with as encountered.
- It is certain that not all of the raises are represented in the 3D as-built models, this may lead to unexpected breakthroughs. These may sometimes be identified by inspection, diamond drill intersections and/or test holing if the existence of such is suspected and dealt with as encountered.

As the historic stopes have been modelled from sectional drawings, the exact stope profile between sections is uncertain. The actual location of the stope wall may be several metres from the location shown in the 3D as-built models; the one test drift developed to breakthrough in 2024 confirms this. Additional work is required to confirm the locations of stope walls through surveying of voids, diamond drilling and test holing programs prior to development.

- This presents both risks and opportunities on a stope-by-stope basis that the resource model has under/over reported the available resource in that area. This is not considered material as many small areas will contribute small losses and gains.

Similarly, the exact thickness of sill pillars above and below the historic stopes is uncertain.

- There is a risk that additional bypass will be required, though the PFS mine plan places new ramps and levels away from the historic stopes by using by-passes due to this uncertainty. Experience at other historic track mines is that the sill pillar between track drift and stope is probably insufficient to develop a stable 5.0 mH x 5.0 mW trackless drift through. Additional work is required to confirm the sill pillar thicknesses and geotechnical conditions through surveying of voids, diamond drilling and test holing programs prior to any development.

The historic MCF stopes are planned to approach and sometimes breakthrough into the historic stopes as the mineral resource model indicates areas of good grade left in the immediate walls of these historic stopes.

- There is a risk that some of these historic MCF areas will not be mineable using the methods described in Section 16 due to the following:
  - Unable to support the excavation when drifting parallel to a historic stope wall despite the specified in-cycle shotcrete and ground support systems
  - The historic backfill is saturated or otherwise unstable and will not stand up long enough for the in-cycle shotcrete to be placed

There is no visual distinction between ore and waste, which makes the grade control in the MCF stopes challenging and slows down the MCF stoping rates. The PFS assumes short rounds (2.4 or 3.0 m) are used and time allowed for grade control samples to be assayed and returned as these drifts will be under geology control.

- There is a risk that the grade control will not be as effective as assumed in the PFS, leading to higher dilution and lower head grades from the MCF stopes.

Mine Operating and Capital Costs Risks

The largest risk to operating and capital costs is labour costs. Many operations are finding it challenging to attract and retain a skilled workforce.

- There is a risk of labour costs being higher than estimated for the PFS due to competition for a limited labour pool in Red Lake.
- There is a risk of General and Administrative costs associated with labour being higher than estimated for the PFS if a higher proportion of the workforce is Fly-in/Fly-out than assumed.

The PFS assumes that all operating development, stoping and capital lateral development is executed by company employees.

- If the operation has difficulty attracting and retaining a skilled workforce, there is a risk of increasing labour costs if contractors are used to fill the gaps.
- Or risk of production shortfalls if an adequately skilled workforce cannot be maintained, which leads to higher operating costs per tonne as underground mines have a high fixed cost component.

Mine Infrastructure Risks

Risks in the construction of mine infrastructure at Madsen include:

- Labour availability for specialized trades and contractor availability in general within the mining space has been a challenge for several years and will continue to be so. This has a potential impact on both cost and schedule depending on the number of competing projects underway at the time of execution.
- Procurement of critical items and equipment, like power distribution systems (transformers, unit substations, switchgear) and hoisting systems have long lead times that have fluctuated significantly over the past several years. This can impact both cost and schedule depending on demand at the time of execution.
- The current development method proposed for the Shaft #3 installation is via the use of a raisebore system While the employment of this method has been vetted through similar projects and a review of the site geology, the final shaft location will require a pilot drill hole to be completed such that geotechnical information can confirm ground conditions If ground conditions in the local area prove challenging to a raisebored approach, a raise and slash method would be employed. This would have an impact on schedule and costs.

25.15.2 Opportunities

Mining Opportunities

There is an opportunity to extend the mine life and optimize the mine plan as the PFS mine plan is based upon a MRE with an effective date of December 31, 2021. The diamond drilling performed by Pure Gold up until closure in October 2022, plus the diamond drilling and test mining work completed by WRLG since acquiring the project, is not reflected in the PFS mine plan. A considerable amount of Indicated mineral resources that met the COG were not included in the PFS mine plan as these mining areas failed to pay for the required capital development to access them. Improving the continuity and expanding these mineral resources with additional information and/or conversion from Inferred to Indicated mineral resources can likely justify the capital development required to access some of these mining areas in future mine plans.

There is an opportunity to move towards utilizing more longhole stoping methods by bringing in new mining zones that have not been previously mined such as Fork, Russet and Wedge. The main reason for using MCF mining methods was to be able to extract the high-grade remnants left behind in and around the historic stopes. The lower grade McVeigh zone is also a good candidate for longhole mining, but much of this zone was excluded from the PFS mine plan because the zone is 500 m from the core mining areas.

There is an opportunity to optimize the short-term mine plan to take advantage of the current higher spot gold price, which is 60% higher than the cut-off grade assumption of US$1,680/oz. This would also likely bring in the lower McVeigh zone mentioned above, as well as the Fork, Russet and Wedge zones.

As not all of the raises are represented on the 3D as-built models, there may be opportunity to utilize currently unknown raises for ventilation purposes, emergency manways or as ore/waste passes, thereby reducing the number of new raises to be developed.

The existing Shaft #2 ore/waste pass system was assumed to be unusable so there may be an opportunity to utilize some sections of the existing ore/waste pass system as well as other existing raises to reduce trucking requirements as no ore/waste passes were included in the PFS mine plan.

There is an opportunity to improve the MCF productivity in the Normal MCF stopes by implementing the use of jumbo uppers. The PFS assumed a breasting method for all MCF as this is required for the historic MCF stopes and 8 Zone MCF stopes due to anticipated ground conditions.

Mine Operating and Capital Costs

There is an opportunity to reduce the capital cost per tonne ore by extending the mine life as discussed above. The PFS capital cost includes for completion of the main ramp and ventilation systems down to 25 Level. Bringing additional ore into the plan above 25 Level will generally not require a lot of additional capital ramp development or main ventilation raises in the core mining areas.

There is an opportunity to optimize the waste haulage costs as the PFS assumes that all waste rock not used for backfill is hauled/skipped to surface. There is a significant amount of empty historic stopes in the upper mine that could be used to dispose of excess waste rock in the early years of the mine life. Filling historic drifts that will not be utilized could also be considered.

26 Recommendations

Based on the technical work and analysis performed in support of the PFS for the Madsen Mine, the following sections highlight recommendations to further improve the quality of data collected on site, advance orebody knowledge and provide guidance for safe underground operations when the restart commences.

26.1 Geology, Mineralization and Mineral Resource

The following recommendations are provided to advance the understanding of the geology, mineralization controls and mineral resource for the Madsen Mine:

- Additional delineation drilling within the Fork, Russet and Wedge deposits is required to further delineate mineralization continuity, particularly higher-grade mineralization, and to support potential upgrading of Inferred resources to higher confidence levels.
- Continue to incorporate underground mapping, structural data analysis and production grade control data into updated interpretations of Madsen deposit mineralization domains to support mineral resource estimation.
- Continue with current grade control practices in areas of active mine production to support mine planning and forecasting.

Conditional simulation should be implemented to better quantify uncertainty associated with geological complexity and grade variability within the current mineral resource model. Further risk assessments associated with mine planning and forecasting activities could then be conducted to support improved LOM decisions.

26.2 Mineral Processing and Metallurgical Testing

A circuit audit is recommended once the plant is in operation to improve carbon adsorption and lower carbon concentration.

26.3 Mining

The following work is recommended to address data gaps in the geotechnical engineering:

- Targeted triple tube oriented geotechnical drilling and core logging to collect geotechnical data over the full extents of the planned underground workings. New logging data should be compared to historic logging data collected on split core to assess the validity of the older data.
- Comprehensive geomechanical testing should be performed to supplement historic data. This should involve point load testing of full diameter core as well as lab testing. Lab tests should include tests that can accommodate both weaker and more competent lithologies.
- More mapping of open workings as these become available/dewatered. Current mapping is limited to near-surface exposures.

Some of the lithologies and alteration domains have similar rock mass properties and the QP recommends simplifying domains on an operational level to two categories only, as follows:

- **Good to Fair Rock** – this includes Basalts, Veins, the Confederation unit, Peridotite, all SAFZ, and Russet Lake UMAFs
- **Poor to Very Poor Rock** – this includes all rock in 8 Zone, Venus and other weak ultramafic units (note that development in the Venus and other weak ultramafic units is very limited as these units do not host ore)

It is recommended that further hydrogeological investigation be performed to gain a better understanding of underground water sources, flows and risks.

Additional PLT and geomechanical testing is recommended to increase the confidence in rock mass strengths, specifically in the ultramafics and the 8 Zone. It is recommended that geotechnical drilling and core logging be performed to better cover the extents of the planned mining. Specifically ground conditions in the 8 Zone have not been well quantified.

It is recommended that at least every second stope along each level is backfilled, rockfilled or cemented hydraulic filled, to ensure no long open strike spans are left.

Sill pillars are recommended where mining under historic stopes, regardless of historic fill type. A sill pillar that is a minimum of 7.5 m or two times the stope span should be maintained at the narrowest pillar point.

Generally, for stopes less than 8 mW x 50 mL with a dip greater than 70º, it is recommended that a minimum crown pillar of 25 m of intact bedrock be maintained.

It is recommended that vertical development and intersections be avoided in poor ground. Four-way intersections should generally be avoided due to increased excavation spans.

Mining within the 8 Zone is anticipated to vary greatly from the rest of the mine. Squeezing ground conditions are expected in the ultramafic units here. Historic development utilized timber sets, spiling, steel sets and shotcrete. It is anticipated that development rates will be slower than in other parts of the mine, with the need for shorter rounds and greater support.

A stress sensitivity numerical modeling study should be undertaken to evaluate the impact of a more adverse principal stress direction as the current assumed stress regime has not been well understood. A stress measurement campaign should take place at the mine to confirm previous assumptions regarding the stress gradient. The production plan should also be modeled to assess changing stress conditions and seismic risk and optimize the stope sequence.

The mine should employ seismic monitoring for advance warning of increasing seismic hazards. The design of the seismic system should place emphasis on coverage of mining areas with greater anticipated seismic risk, such as areas with high historic extraction, stopes in deeper parts of the mine and pillars.

Recommendations made in regard to the mining engineering inputs and operations are as follows:

- Inspection and survey of historic workings as access becomes available to improve quality of as-builts and identify potential risks such as mobilized backfill materials, impounded water, falls of ground, excavations missing off of as-builts (lot of raises) and other hazards.
- Implementation of test hole programs to identify the thickness of sill pillars, presence of open voids and determine wall profile of stopes that cannot be surveyed directly as part of mine Standard Operating Procedures.
- Development and implementation of a good grade control program that effectively deals with the fact that there is no visual distinction between ore and waste is critical to achieving the mine plan.
- Continue to fully mechanize the operation in order to provide a safe workplace and facilitate extraction of resources from difficult locations.

26.4 Project Infrastructure

With respect to the power distribution system on site:

- WRLG should contemplate implementing a power management strategy to compensate for the diurnal variations resulting from the underground shift work.
- The process plant has based on historical data had a high-power consumption resulting from intermittent processing rates. A strategy needs to be developed to stabilize the mill production rate to reduce the power consumption per tonne processed.
- The existing substation is recent and can easily be equipped with metering of the individual feeders to manage the loads on site as well as to determine the power factor.
- The substation is not equipped with capacitor banks to compensate for power factor and to minimize power losses.
- The existing 2300 V obsolete electrical equipment need to be phased out as quickly as possible as there are no replacement parts for some of these components.

Regarding the existing shaft and muck handling infrastructure and new infrastructure to support production from the mine:

- Additional definition of the resource and geotechnical evaluations will be needed to determine the final location and overall depth of the planned Shaft #3 facility

26.5 Water/Waste Management

Recommendations for the next phase of engineering are:

- Foundation investigations and stability analyses for the Main Dam
- Confirm foundation design parameters by completing a cone penetration test (CPT) program downstream of the tailings pond and polishing pond dams

- Install vibrating wire piezometers in foundation materials so that foundation pore pressures can be monitored during construction of the tailings pond and polishing dam raises and during operation
- Continue collection of site-specific meteorological and hydrology data to refine seasonal runoff values and design storms
- Develop an Operations, Maintenance and Surveillance (OMS) Manual and Emergency Preparedness and Response Plan (EPRP) for the tailings and water management systems based on final designs and operating criteria
- Develop a full closure plan for the TMF and MRMF based on the final design configuration
- Investigate opportunities to manage underground mine dewatering flows outside the polishing pond and reduce the annual volume of water to be managed in the TMF

26.6 Future Work

The Madsen Mine is preparing to restart operations in Q2 2025. No additional studies or future work are planned prior to this date, as any expenditures in that regard will be considered operating expenses.

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28 Glossary

Units of Measure, Acronyms and Abbreviations

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
%	percent
°C	degrees Celsius
2D	two-dimensional
3D	three-dimensional
amsl	above mean sea level
ARD	acid rock drainage
Au	gold
AQTK	drill core diameter of 35.5 mm
BQ	drill core diameter of 36.5 mm
C$	dollar (Canadian)
Ca	calcium
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
d	day
DGPS	differential global positioning system
dmt	dry metric ton
E	East
EA	environmental assessment
EIS	environmental impact statement
ft	foot
g	gram
g/cm^3	grams per cubic metre

Symbol / Abbreviation	Description
g/t	grams per tonne
Ga	billion years
gpm	gallons per minute (us)
GSC	Geological Survey of Canada
ha	hectare (10,000 m^2)
ha	hectare
HG	high grade
HLEM	horizontal loop electro-magnetic
HoleID	drill hole identifier
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
kg	kilogram
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
L	litre
LOM	life of mine
m	metre
M	million
Mm	million metres
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
m^3/s	cubic metres per second

Symbol / Abbreviation	Description
Ma	million years
masl	metres above mean sea level
mg	milligram
mg/L	milligrams per litre
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations
MMI	Mobile metal ion leach
MSO	Mining Stope Optimizer
Mt	million metric tonnes
N	North
NAD	North American datum
NE	Northeast
NW	Northwest
NI 43-101	national instrument 43-101
NQ	drill core diameter of 47.6 mm
ON-105 N	Ontario Provincial Highway 105 North
ON-168 S	Ontario Provincial Highway 168 South
ON-17	Ontario Provincial Highway 17
oz	troy ounce
p80	Mesh size at which 80% of material passes
pH	quantitative measure of acidity
P.Eng.	professional engineer
P.Geo.	professional geoscientist
PEA	preliminary economic assessment
PFS	preliminary feasibility study
ppb	parts per billion

Symbol / Abbreviation	Description
PQ	drill core diameter of 85 mm
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
ROM	run of mine
RQD	rock quality designation
s	second (time)
South	South
Southeast	Southeast
Southwest	Southwest
S.G.	specific gravity
SG	specific gravity
SMG	historical Starratt Mine Grid
t	metric tonne (1,000 kg)
t/a	tonnes per year
t/d	tonnes per day
TCR	total core recovery
TFFE	target for further exploration
TMF	tailings management facility
tph	tonnes per hour
US	United States
US$	dollar (American)
UTM	Universal Transverse Mercator (projection)
V	volt
VAG	Air powered diamond drill
W	West
XRF	X-Ray Flouresence
µm	micron (micrometre)

All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

Appendix A QP Certificates

SRK Consulting (Canada) Inc.
2600 – 320 Granville Street Vancouver, BC,
V6C 1S9
T: +1.604.681.4196
F: +1.604.687.5532
vancouver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

Timothy James Coleman, P.Eng.

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report)

I, Timothy James Coleman, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Principal Consultant with the firm SRK Consulting (Canada) Inc, located at Suite 2600 – 320 Granville Street, Vancouver, British Columbia, Canada.
2. I am a graduate of Imperial College of Science, Technology and Medicine, UK where I obtained my M.Sc. in 1997. I obtained a B.Eng.(Honours) Mining Engineering in 1994, and a Diploma in Minerals Engineering (1st Class) in 1992, both from The Camborne School of Mines, UK.
3. I am a Professional Engineer (#46105) with APEGBC.
4. My relevant experience includes being actively engaged in mining since 1997 and have practised my profession continuously since then. I have been involved in mining operations, mining related rock mechanics and consulting covering a wide range of mineral commodities in the United Kingdom, Europe, North and South America, Africa, and Asia.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 16.4, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I visited the property that is the subject of the Technical Report, on 6th and 7th April 2022.
9. I have had prior involvement with the property that is the subject of the Technical Report. I provided rock mechanics review and advice for the property when under the ownership of Pure Gold.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and guidelines.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 18th day of February 2025.

"Signed and Sealed"

Timothy James Coleman, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Chris Dougherty, P.Eng., do hereby certify that:

1. I am an Engineer with Nordmin Engineering Ltd., 160 Logan Avenue, Thunder Bay, ON, P7A 6R1.
2. I graduated from the Lakehead University with a Bachelor's Degree in Engineering in 1994. Aside from the time spent studying, I have practiced my profession continuously since 1987. My relevant experience includes expertise in mine infrastructure, including mine shafts and hoisting systems, muck handling systems and other major installations.
3. I am a Professional Engineer registered as a member, in good standing, with Professional Engineers Ontario (License #90416876).
4. I have visited the Madsen site on at least eight occasions between 1995 and 2021.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 18.11, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 16th of February in Fort Lauderdale, Florida, USA.

C.A. DOUGHERTY

Chris Dougherty, P.Eng.
President
NORDMIN ENGINEERING LTD.

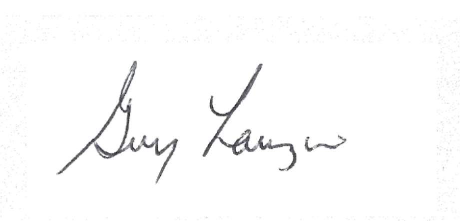
CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Guy Lauzier, do hereby certify that:

1. I am a Engineer with Allnorth Consultants, 1200–1100 Melville Street, Vancouver, BC V6E 4A6, Canada.
2. I graduated from the McGill University with a B. Eng. Mining in 1979. Aside from the time spent studying, I have practiced my profession continuously since 1979. My relevant experience includes Mining, Milling, Operations, Engineering. Construction and Infrastructure design
3. I am a Professional Engineer registered as a member, in good standing, with the EGBC (License #144172).
4. I did visit the Madsen site on February 13th and 14th, 2024.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 18.1 to 18.5, 18.7, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this February 18th, 2025 in Wellington, Ontario, Canada.



Guy Lauzier, P.Eng.

Allnorth Consultants

srk consulting

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Mark Liskowich, do hereby certify that:

1. I am an Associate Principal Consultant with SRK Consulting (Canada) Inc. with an office at 600, 350 3rd Ave North Saskatoon, Saskatchewan.
2. I graduated from the University of Regina with a BSc. Degree in Geology in 1989. Aside from the time spent studying, I have practiced my profession continuously since 1992. My relevant experience includes environmental and social management of mining projects.
3. I am a Professional geoscientist registered as a member, in good standing, with the Professional Engineers & Geoscientists of Saskatchewan - License No.: 10005.
4. I did not visit the Madsen site.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 20.1 to 20.4, 20.5.1, as well as relevant parts in the Executive Summary, Reliance on Other Experts, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this February 18, 2025 in Saskatoon, Saskatchewan, Canada.

Mark Liskowich, PGeo

SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Travis O'Farrell, do hereby certify that:

1. I am a Director of Processing and Metallurgy with Fuse Advisors at 595 Burrard, #3100.
2. I graduated from the McGill University with a Degree in Chemical Engineering in 2010. Aside from the time spent studying, I have practiced my profession continuously since 2010. My relevant experience includes process and metallurgical engineering and corporate metallurgy for various gold processing plants.
3. I am a Professional Engineer registered as a member, in good standing, with the EGBC (License # 46026).
4. I did visit the Madsen site on August 27th and 28th, 2024.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 12.5, 12.6, 13, 17, 21.2.6, 21.3.3, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this February 18th, 2025 in Vancouver, British Columbia, Canada

"original signed"

Travis O'Farrell, P. Eng

Fuse Advisors

srk consulting

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Brian Prosser, do hereby certify that:

1. I am a Practice Leader/Principal Consultant with SRK Consulting (US). 2120 N. Winery Ave., Suite 101, Fresno, California, 93703, United States
2. I graduated from the Virginia Polytechnic Institute and State University with a Bachelors of Science degree in Mining Engineering in 1994. Aside from the time spent studying, I have practiced my profession continuously since 1994. My relevant experience includes Conducting ventilation audits at metal/non-metal mines, optimizing existing ventilation systems, extrapolating existing ventilation system to function with extended mine life and updated equipment loads to meet higher production demands. Developing ventilation strategy and modeling to support PFS and FS studies.
3. I am a Professional Engineer registered as a member, in good standing, with the State of Nevada Board of Professional Engineers and Land Surveyors (License #15465).
4. I did not visit the Madsen site.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 16.6.6, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 18th day of February 2025 in Fresno, California, United States.

"original signed"

Brian Prosser, PE

SRK Consulting (US)

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Cliff Revering, do hereby certify that:

1. I am an Associate Consultant (Geological Engineering) with SRK Consulting (Canada) Inc. located at Suite 2600 – 320 Granville Street, Vancouver, British Columbia, Canada.
2. I graduated from the University of Saskatchewan with a B.E. in Geological Engineering (1995) and completed a Citation in Applied Geostatistics from the University of Alberta. My relevant experience includes more than 27 years employment in the mining industry, related to exploration, mine operations and project evaluations, with a specialization in geological modelling, mineral resource and reserve estimation, production reconciliation, grade control, exploration and production geology, and mine planning.
3. I am a Professional Engineer registered as a member, in good standing, with the Association of Professional Engineers and Geoscientists of Saskatchewan (APEGS#9764).
4. I did visit the Madsen site on July 4 to 7, 2022.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 6, 7, 8, 9, 10, 11 (not including 11.1.6, 11.3.6), 12.1 to 12.4, 12.6, 14, 23, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 18th day of February 2025 in Saskatoon, Saskatchewan, Canada.

"original signed"

Cliff Revering, PEng

SRK Consulting (Canda) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Daniel Ruane, 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 visited the Madsen site on October 1, 2024.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 18.8, 18.9, 18.10, 20.5 (not including 20.5.1), 21.2.7, 21.3.4, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 18th day of February 2025 in Vancouver B.C., Canada

"original signed"

Daniel Ruane

Knight Piésold Ltd.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Stephen Taylor, do hereby certify that:

1. I am a Principal Engineer (UG Mining) with SRK Consulting (Canada) Inc., Suite 2A – 69 Young Street, Sudbury, ON, P3E EG5, Canada.
2. I graduated from the Laurentian University with a B.Eng (Mining Engineering) in 1992, followed by a M.A.Sc. (Mining Engineering) in 1995 form the University of Nevada-Reno. Aside from the time spent studying, I have practiced my profession continuously since 1995. My relevant experience includes 15 years in mine operations, mine engineering and projects. I have been with SRK Consulting (Canada) Inc. since 2010, specializing in UG mining and Mineral Reserve Estimation.
3. I am a Professional Engineer registered as a member, in good standing, with the Professional Engineers Ontario (License # 90365834).
4. I did visit the Madsen site on October 9th to 11th, 2024.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 2, 3, 4, 5, 15, 16.1 to 16.3, 16.5 to 16.8 (not including Section 16.6.6), 19, 21.1, 21.2 (not including 21.2.6, 21.2.7), 21.3 (not including 21.3.3, 21.3.4), 22, 24, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this February 18, 2025 in Sudbury, Ontario, Canada.

"original signed"

Stephen Taylor, PEng

SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Bernie Ting, do hereby certify that:

1. I am a Lead Engineer with T Engineering located at 2857 Sherwood Heights Drive Unit 8, Oakville, Ontario L6J 7J9.
2. I graduated from the University of Toronto with a Bachelor of Applied Science and 2011 (repeat as necessary). Aside from the time spent studying, I have practiced my profession continuously since 2012. My relevant experience includes backfill operation, backfill system design, commissioning and troubleshooting.
3. I am a Professional Engineer registered as a member, in good standing, with the Professional Engineers Ontario (License #100186379).
4. I did visit the Madsen site on Feb 11th 2025.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Section 18.6, as well as relevant parts in the Executive Summary, Interpretations and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 18th day of February 2025 in Toronto, Ontario, Canada.

"original signed"

Bernie Y.J. Ting, Lead Engineer

T Engineering

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled "NI 43-101 Technical Report and Prefeasibility Study for the Madsen Mine, Ontario, Canada" prepared for West Red Lake Gold Mines Ltd. (the "Issuer") dated 18 February 2025 with an effective date of 7 January 2025 (the "Technical Report").

I, Sheila Ulansky, do hereby certify that:

1. I am a Senior Consultant with the firm SRK Consulting (Canada) Inc, located at Suite 2600 – 320 Granville Street, Vancouver, British Columbia, Canada.
2. I am a graduate of the University of Victoria, BC in 2007, I obtained a BSc in geology. In 2019 I obtained an MSc degree (geology) from Laurentian University, ON. I have practiced my profession continuously since 2007, initially in exploration geology on a variety of deposit types. Since 2012, I have worked full time as a Resource Geologist with emphasis on QA/QC, exploratory data analysis, variography, 3D geological modelling and resource estimation. I have worked on many gold deposit types, including narrow vein in Orogenic systems, Carlin-style mineralization, epithermal gold mineral systems, and VHMS systems; experience which is relevant to the Madsen Mine Project scope of work.
3. I am a Professional Geoscientist registered with the Engineers and Geoscientists BC (EGBC) with Membership Number 36085. I am also registered with the Association of Professional Geoscientists of Ontario (PGO) with registration number 4045.
4. I did not visit the Madsen site.
5. I have read the definition of "qualified person" set out in 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. As a qualified person, I am independent of the Issuer as defined in Section 1.5 of NI 43-101.
7. I am a co-author of the Technical Report, responsible for Sections 11.1.6, 11.3.6, 12.6, as well as relevant parts in the Executive Summary, Interpretation and Conclusions, Recommendations, References and Glossary of the Technical Report, and I accept professional responsibility for those sections of the Technical Report.
8. I have not had prior involvement with the subject property.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 18th day of February 2025 in West Vancouver, British Columbia, Canada.

"original signed"

Sheila Ulansky, P.Geo.